Oxide Tailings Project Prefeasibility Study for the Avino Property, Durango, Mexico

NI 43-101 Technical Report

PREPARED FOR
AVINO SILVER & GOLD MINES LTD.
VANCOUVER, BC, CANADA

EFFECTIVE DATE: FEBRUARY 5, 2024

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TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

ACRONYMS AND ABBREVIATIONS

Acronym/Abbreviation	Definition
AA	Atomic Absorption
AAS	Atomic Absorption Spectrometry
ABA	Acid-base Accounting
Acme	Acme Laboratories
Ag	Silver
AgEQ	Silver Equivalent
AHAG	Avino Historic Above-Ground
ALS	ALS Laboratory
Au	Gold
Avino	Avino Silver & Gold Mines Ltd.
Avino Mexico	Compañía Minera Mexicana de Avino, S.A. de C.V.
Blanks	Blank Reference Materials
Cannon-Hicks	Cannon-Hicks & Associates Ltd.
CCD	Countercurrent Decantation
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CITES	Convention on International Trade in Endangered Species of Wild Fauna and Flora
CMMA	Compañía Minera Mexicana de Avino, S.A. de C.V.
Coeur	Coeur Mining, Inc.
CPI	Consumer Price Index
CPT	Cone Penetration Test
CSV	Comma-Separated Values
Cu	Copper
DCS	Distributed Control System
DEM	Digital Elevation Model
dO_2	Dissolved Oxygen
DSTMF	Dry Stacked Tailings Management Facility
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EMEW	Electrometals Electrowinning
EQMP	Environmental Quality Monitoring Program
ET	Elena Tolosa
FEL	Front End Loader
FS	Feasibility Study
G&A	General and Administration

Acronym/Abbreviation	Definition
GRG	Gravity Recoverable Gold
HS	High Sulphidation
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma Mass Spectroscopy
ID	Inverse Distance
ID^2	Inverse Distance Squared
IDF	Intensity Duration Frequency
IMC	Independent Mining Consultants
Inspectorate	Inspectorate Laboratories
IP	Inductively Coupled Plasma
IP	Induced Polarization
IRR	Internal Rate of Return
IS	Intermediate Sulphidation
ISO	International Organization for Standardization
JV	Joint Venture
LAN	Local Area Network
LCI	La Cuesta International Inc.
LCT	Locked Cycle Test
LDL	Lower Detection Limit
LG	Lerchs-Grossman
LGEEPA	Ley General del Equilibrio Ecológico y la Protección al Ambiente
LGPGIR	Ley General para la Prevención y Gestión Integral de los Residuos
LOM	Life of Mine
LS	Low Sulphidation
Luismin	Compania Mineras Minas San Luis
LVC	Lower Volcanic Complex
LVS	Lower Volcanic Supergroup
M3	M3 Engineering & Technology Corp.
MDA	Mine Development Associates
MENR	Ministry of Environment and Natural Resources
MIA	Manifestación de Impacto Ambiental
MIA-P	Manifestación de Impacto Ambiental, modalidad Particular
Minerales	Minerales de Avino, Sociedad Anonima de Capital Variable
MMI	MineStart Management Inc

Acronym/Abbreviation	Definition
MRE	Mineral Resource Estimate
NaCN	Sodium Cyanide
NI 43-101	National Instrument 43-101
NN	Nearest Neighbour
NPV	Net Present Value
NSR	Net Smelter Return
NYSE	New York Stock Exchange
OIS	Operator Interface Station
OK	Ordinary Kriging
Orko	Orko Gold Corp. or Orko Silver Corp.
OTP	Oxide Tailings Project
PAS	Pan American Silver
PAX	Potassium Amyl Xanthate
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFS	Prefeasibility Study
PLS	Pregnant Leach Solution
PMLP	Proyectos Mineros La Preciosa, S.A. de C.V.
PRA	Process Research Associates Ltd.
PROFEPA	Procuraduría Federal de Proteccíon al Ambiente
Promotora	Promotora Avino S.A. de C.V.
PwC	PricewaterhouseCoopers
QA	Quality Assurance
QC	Quality Control
QP	Qualified Person
Q-Q	Quantile-Quantile
RC	Reverse Circulation
REIA	Reglamento en Materia de Evaluacion del Impactor Ambiental
ro	Rougher
RQD	Rock Quality Designation
SCPT	Seismic Cone Penetration Test
SD	Standard Deviation
Selco	Selco Mining and Development
SEMARNAT	Secretaría de Medio Ambiente y Recursos Naturales
SGS	SGS Minerals

Acronym/Abbreviation	Definition
Snowden	Snowden Engineering Inc.
SRM	Standard Reference Material
the Project	Avino Mine and La Preciosa Sites
the Property	Avino Mine and La Preciosa Sites
TMF	Tailings Management Facility
TSF	Tailings Storage Facility
TSX	Toronto Stock Exchange
UDL	Upper Detection Limit
UPS	Uninterrupted Power Supply
UVS	Upper Volcanic Supergroup
VHF	Very High Frequency
VoIP	Voice Over Internet Protocol
WAD	Weak Acid Dissociable
WBS	Work Breakdown Structure
WD	Water Displacement
WTP	Water Treatment Plant

UNITS

above mean sea level	amsl
acre	ac
ampere	A
annum (year)	a
bank cubic metres	bm^3
bags	bgs
billion	B
billion tonnes	Bt
billion years ago	Ga
British thermal unit	BTU
centimetre	cm
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic yard	yd^3
Curie	Ci
Coefficients of Variation	CVs
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBa
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	USD$
dollar (Canadian)	Cdn$
dry metric ton	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
gauge	ga
gigajoule	GJ
gigapascal	GPa
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
greater than	>
hectare (10,000 m^2)	ha
hertz	Hz

horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	"
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne (metric ton)	kWh/t
kilowatt hours per year	kWh/a
less than	<
litre	L
litres per minute	L/m
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above sea level	masl
metres Baltic sea level	mbsl
metres per minute	m/min
metres per second	m/s
metric ton (tonne)	t
Mexican peso	MXN$
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million pounds	Mlb
million tonnes	Mt
minute (plane angle)	'

minute (time)	min
month	mo
Neutron	N
ounce	oz
pascal	Pa
pico	p
centipoise	mPa·s
parts per million	ppm
parts per billion	ppb
percent	%
pound(s)	lb
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
twenty-foot equivalent unit	TEU
thousand tonnes	kt
tonne (1,000 kg)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
troy ounce	tr oz
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt
year (annum)	a

1.0 SUMMARY

1.1 Introduction

Avino Silver & Gold Mines Ltd. (Avino) is a Canadian-based mining and exploration company listed on the Toronto Stock Exchange (TSX) and the New York Stock Exchange (NYSE) American with precious metal properties in Mexico and Canada. The Avino mine site (the “Property” or the “Project”), near Durango, Mexico, is Avino’s principal asset and is the subject of this Technical Report, which includes a proposed extraction and reprocessing of historic tailings from the tailings deposit for silver and gold production within the existing Avino Property and mining operations. The information in this report consolidates current information in the light of proximity and potential synergies between the Avino tailings reprocessing facilities and the currently operating Avino Mine. This Technical Report aims to provide a NI 43-101 compliant Pre-feasibility Study, Mineral Reserves and previously disclosed Mineral Resources (Tetra Tech, 2023) based on the current economic parameters and the most recent metallurgical testwork results.

Avino holds a 99.67% interest in the Property through its subsidiary companies called Compañía Minera Mexicana de Avino, S.A. de C.V. (CMMA) and Promotora Avino, S.A. de C.V. (Promotora). Avino commenced development, including drilling and bulk sampling, on the San Gonzalo Vein in 2010, and this work is ongoing. This marks the resumption of activities on the Property since 2001, when low metal prices and the closure of a key smelter caused the mine to close after having been in operation continuously for 27 years. Between 1976 and 2001, the mine produced approximately 497 t of silver, 3 t of gold, 11,000 t of copper (Slim 2005a), and an undocumented amount of lead. In 2023, the mine produced approximately 615 kt of concentrate, containing approximately 29 t of silver, 0.23 t of gold, and 2,406 t of copper.

The majority of the Mineral Resource information presented in this Technical Report has been sourced from the data provided by Avino, Avino internal reports, Tetra Tech (2023; 2021; 2018; 2017; 2013), Slim (2005d), Gunning (2009), and a process plant review memo by Tetra Tech (2019). Most of the information was provided in English, but some were written in Spanish and translated into English.

All units of measurement used in this Technical Report and resource estimate are in metric units, and the currency is expressed in US dollars unless otherwise stated.

1.2 Property Description and Location

The Property is located in the state of Durango, Mexico, within the municipalities of Pánuco de Coronado and Canatlán, and is approximately 85 km by existing road, northeast of the city of Victoria de Durango, the state capital. The Project is situated on the eastern flank of the Sierra Madre Occidental Mountain range. It can be found on the Instituto Nacional de Estadística, Geografía e Informática General Carlos Real Topographic Map G13D72, centred on coordinates 24°25'42.4200"N Latitude and 104°27'27.2380"W Longitude (554,987.8815 mE, 2,701,771.0046 mN) in the Universal Transverse Mercator (WGS 84), Zone 13R (Northern Hemisphere) (Figure 1-1).

Figure 1-1: General Location of the Property (Avino 2024)

Avino holds 41 mineral concessions, totalling 7,943.0123 ha (see Table 1-1).

Table 1-1: Avino Property Mineral Concessions (Avino 2024)

S. No.	Concession Name	Concession No.	Area (Ha)	Expiration Date
Area Avino				
1	AMPLIACION DE LA POTOSINA	185326	84.0000	December 14, 2039
2	AMPLIACION SAN GONZALO	191837	5.8495	December 19, 2041
3	AMPLIACION LA MALINCHE	204177	6.0103	December 18, 2046
4	EL POTRERITO	185328	9.0000	December 14, 2039
5	LA MALINCHE	203256	9.0000	June 28, 2046
6	LA POTOSINA	185336	16.0000	December 14, 2039
7	SAN GONZALO	190748	12.0000	April 29, 2041
8	YOLANDA	191083	43.4577	April 29, 2041
9	AGRUP. SAN JOSE	164985	8.0000	August 13, 2029
10	AGRUP. SAN JOSE, (EL TROMPO)	184397	81.5466	October 13, 2039
11	AGRUP. SAN JOSE, (GRAN LUCERO)	189477	161.4684	December 5, 2040
12	AGRUP. SAN JOSE, (PURISIMA CHICA)	155597	136.7076	September 30, 2071
13	AGRUP. SAN JOSE, (SAN CARLOS)	117411	4.4505	December 5, 2061

table continues…

S. No.	Concession Name	Concession No.	Area (Ha)	Expiration Date
14	AGRUP. SAN JOSE, (SAN PEDRO Y SAN PABLO)	139615	12.0000	June 22, 2061
15	AGUILA MEXICANA	215733	36.7681	March 12, 2054
16	ARANJUEZ	214612	96.0000	October 2, 2051
17	AVINO GRANDE IX	216005	19.5576	April 2, 2052
18	AVINO GRANDE VIII	215224	22.8816	February 14, 2052
19	EL CARACOL	215732	102.3821	March 12, 2052
20	EL FUERTE	216103	100.3274	April 9, 2052
21	FERNANDO	205401	72.1287	August 29, 2047
22	LA ESTELA	179658	14.0000	December 11, 2036
23	LOS ANGELES	154410	23.7130	March 25, 2071
24	NEGRO JOSE	218252	58.0000	October 17, 2052
25	SAN MARTIN DE PORRES	222909	30.0000	September 15, 2054
26	SANTA ANA	195678	136.1823	September 14, 2042
--	**TOTAL**	**--**	**1,301.4314**	**--**
Area La Preciosa				
1	EL CHOQUE CUATRO	220251	629.7778	July 1, 2053
2	EL CHOQUE SEIS	220583	249.0000	September 1, 2053
3	EL CHOQUE TRES	218953	10.0000	January 28, 2053
4	FRACCION LA PRECIOSA	185128	2.5249	July 14, 2038
5	LA B	214232	28.2006	September 5, 2051
6	LA PRECIOSA	182517	143.6119	July 14, 2038
7	LUPITA	182584	27.1878	August 11, 2038
8	SAN PATRICIO	189616	29.4740	December 4, 2040
9	SANTA MONICA SUR	223097	900.0000	October 15, 2054
10	EL NIÑO	236219	10.0000	May 24, 2060
11	LA PEÑA	204828	57.3190	May 12, 2047
12	CENTINELA	244180	0.1048	June 29, 2065
13	DON MIGUEL HIDALGO Y COSTILLA	244480	0.2168	October 5, 2065
14	HURACAN 4 R1A	246910	1,768.4591	January 19, 2054
15	TIFON 3 R1A	246466	2,785.7042	February 16, 2056
--	**TOTAL**	**--**	**6,641.5809**	**--**

Notes: Figures may not add to totals shown due to rounding.

All concessions are current and up-to-date based on the information received. Mineral concessions in Mexico do not include surface rights. Avino has entered into agreements with communal landowners (Ejidos) of San Jose de Avino, Panuco de Coronado, and Zaragoza for the temporary occupation and surface rights of the concessions.

1.3 Geology and Mineralization

1.3.1 Avino Area

The Property is located within the Sierra de Gamon, on the east flank of the Sierra Madre Occidental. The area is a geological window into the Lower Volcanic series and consists of volcanic rocks of mainly Andesitic affiliation with other rock types occurring more sparsely to the north (Slim 2005d).

A large monzonitic intrusion is observed in the region in the form of dykes and small stocks, which may be related to the Avino Vein mineralization. Several younger, thin mafic sills are also found in various parts of the region.

The Avino concession is situated within a 12 km north-south by 8.5 km caldera, which hosts numerous low-sulphidation epithermal veins, breccias, stockwork, and silicified zones. These zones grade into a “near porphyry” environment in the general vicinity of the Property. The caldera has been uplifted by regional, north-trending block faulting (a graben structure), exposing a window of andesitic pyroclastic rocks of the lower volcanic sequence, which is a favourable host rock. The upper volcanic sequence consists of rhyolite to trachytes with extensive ignimbrite and is intruded by monzonite bodies.

The basal andesite-bearing conglomerate and underlying Paleozoic basement sedimentary rocks (consisting of shales, sandstones, and conglomerates) have been identified on the Avino concession in the south-central portion of the caldera, covering the Guadalupe, Santiago, San Jorge, the San Gonzalo Trend, Malinche, Porterito, and Yolanda areas. A northerly trending felsic dyke, probably a feeder to the upper volcanic sequence, transects the Property and many of the veins. The Aguila Mexicana low-temperature vein system, with significant widths but overall low precious metal values, trends north-northwest, similar to the felsic dyke and with similar continuity across the Property. The two structures may occupy deep crustal faults that control volcanism and mineralization, with the felsic dyke structure controlling the emplacement of the Avino, Nuestra Senora, and El Fuerte-Potosina volcanic centres and the Aguila Mexicana controlling the Cerro San Jose and El Fuerte-Potosina volcanic centres (Paulter 2006).

Silver- and gold-bearing veins crosscut the various lithologies and are generally oriented north-northwest to south-southeast (the Avino Vein trend) and northwest to southeast the San Gonzalo trend. In Mexico, these vein deposits may have large lateral extents but can be limited in the vertical continuity of grades due to the effects of pressure on boiling levels for mineralizing fluids. The rocks have been weathered and leached in the upper sections due to contact with atmospheric waters. The oxide tailings material is derived primarily from these shallow zones, whereas the sulphide tailings are predominantly from material sourced at depth from the underground workings.

The valuable minerals found during the period of mining of the oxide zone are reported to be argentite, bromargyrite, chalcopyrite, chalcocite, galena, sphalerite, bornite, native silver and gold, and native copper.

1.3.1.1 The Avino Vein

The Avino Vein (see Figure 1-2 for location) has been and continues to be the primary deposit mined on the Property since at least the 19th century. It is 1.6 km long and up to 60 m wide on the surface. The deepest level is at the 1,849 masl level (430 m below the surface).

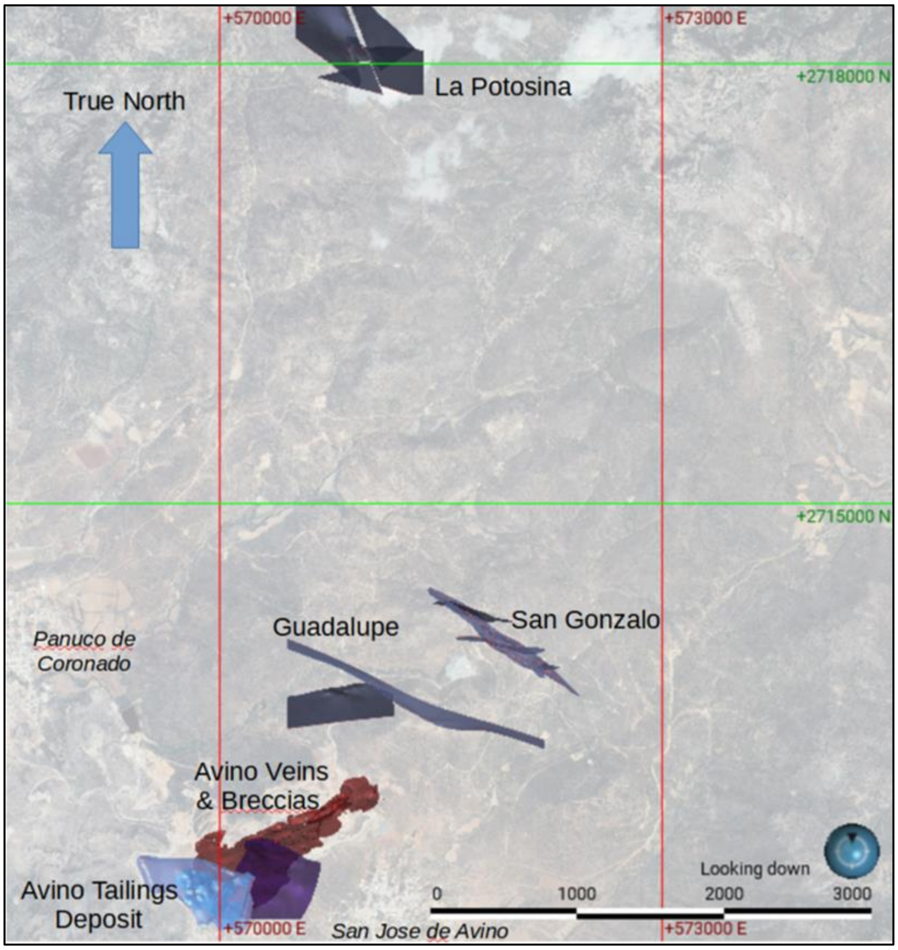


Figure 1-2: Plan View of the Avino Mine Deposits (Red Pennant 2022)

1.3.1.2 The San Gonzalo Vein

The San Gonzalo Vein system (see Figure 1-2 for location), including the crosscutting Angelica Vein, is located 2 km northeast of the Avino Vein. It constitutes a strongly developed vein system over 25 m across, trending 300° to 325°/80° northeast to 77° south. Banded textures and open-space filling are common, and individual veins have an average width of less than 2 m. The vein was mined historically, and underground workings extend approximately 1.1 km along the strike to the 1,970 masl (300 m below the surface).

1.3.1.3 Guadalupe Vein

The Guadalupe Vein (see Figure 1-2) is located approximately 0.7 km northeast of the Avino Vein. It consists of northwest-southeast and east-west striking steep-dipping vein sets. The geometry is similar to the San Gonzalo vein, but the base metal mineralization more closely resembles the Avino hanging wall breccia.

1.3.1.4 La Potosina Vein

The La Potosina Veins (see Figure 1-2) are located close to the northern margin of the caldera, approximately 7 km north of the Avino mine and processing plant. It consists of complementary northwest-southeast striking steep-dipping vein sets. The geometry is complex, with at least two ages of fault displacement.

1.3.1.5 Oxide and Sulphide Tailings

The Avino tailings deposit (see Figure 1-2) is adjacent to the processing plant, approximately 300 m west-southwest of the mine offices. The tailings have been built up over several decades of mining and processing, and several units have been defined based on the oxidation nature of the tailings and metal content.

Due to the historical processing sequence, the oxide tailings are primarily derived from weathered and oxidized rocks close to the surface of the Property, whereas the sulphide tailings are predominantly derived from material sourced at depth from the underground workings below the weathered/leached zone.

1.3.2 La Preciosa Area

La Preciosa deposit is situated on the eastern flank of the Cretaceous to mid-Tertiary Sierra Madre Occidental. The SMO is the largest silicic igneous province in North America, and it stretches from the USA-Mexico border to the latitude of Guadalajara, where the SMO is covered by the late Miocene to Quaternary Trans-Mexican Volcanic Belt.

Mineralization at the La Preciosa is hosted within multiple discrete poly-phase quartz veins, often displaying banded, smoky, drusy, and chalcedony textures. Also, in each stage, there is variably crustiform banded fracture fill/breccia cement mineralogy. Fluorite, amethyst, a substantial number of barite laths, calcite, and rhodochrosite may also be present, and sulphide mineralization in the form of sphalerite, galena, pyrite, chalcopyrite, acanthite, sparse native silver, and free gold, as well as iron and manganese oxides have been noted in drill core. The principal silver-bearing mineral at the La Preciosa is acanthite-pseudomorphic after argentite or as microcrystalline to amorphous grains.

The main vein system on the Abundancia ridge consists of dominantly southward-striking and westward-dipping veins plus east-southeast–striking, south-dipping crosscutting veins. The Abundancia ridge vein system has been traced on the surface for over 1.5 km. In the eastern part of the Project, a north- to northwest-striking, shallow west-dipping vein system with associated hanging wall veining and alteration is exposed in a series of hills. This vein system is referred to as the Martha vein or fault zone and has been traced by drilling for over 2.5 km along the strike.

The mineralization in the area occurs in veins, veinlets, and stockwork. These veins average in true width less than 15 m (Martha Vein) and consist of several stages of banded crustiform to colloform, quartz (and cryptocrystalline quartz at shallow depths), adularia, barite, and typically later carbonates (both calcite and rhodochrosite); illite commonly replaces the adularia. There are variable amounts of pyrite, sphalerite, and galena plus argentite, and variable amounts of tetrahedrite - tennantite, freibergite, and Ag sulfosalt.

The mineralization displays characteristics typical of epithermal veins in Mexico, particularly of the Ag-rich variety. Quartz veins are accompanied by adularia, barite, calcite, and rhodochrosite of variable timing, as well as acanthite, freibergite, Ag sulfosalts and minor electrum, plus variable amounts of pyrite, honey-coloured sphalerite, tennantite/tetrahedrite, chalcopyrite and galena, and supergene Fe and Mn oxides; the hypogene minerals are characteristic of intermediate-sulphidation deposits in Mexico. Mineralization is believed to be Tertiary in age both the Lower Volcanic Supergroup (LVS) and Upper Volcanic Supergroup (UVS) are mineralized, but the basalts are recent and not mineralized.

The Martha vein is the largest vein in the deposit, with at least three times the volume of the next largest vein, La Abundancia. The Martha vein dips ~20-30°, following the southwest-dipping contact of volcaniclastic rocks overlying an immature conglomeratic unit (consisting mainly of polylithic clast-supported fragmental rock with angular to sub-rounded clasts) or the underlying schist.

There are steep-dipping veins in the west on the ridge, such as the La Gloria vein. These steep veins can be considered as a mineralized zone or lode of stock work, silicification, breccias, veins, vein breccias, veinlets, and a general mix of multiple styles of mineralization. Within this broader zone, for example the Martha lode ranges from 1 to 35 m thicknesses and averages approximately 5 m.

1.4 Mineral Resource Estimates

The current mineral resources for the property (Avino Mine and La Preciosa area) are summarized in Table 1-2.

Table 1-2: Avino Property – Mineral Resources (Inclusive of Mineral Reserves, Effective Date: October 16, 2023)

Area	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
Avino Mine	MEA	8.466	142.35	71.72	0.53	0.32	38.75	19.52	144.26	60.24
	IND	27.204	142.85	59.42	0.53	0.41	124.94	51.97	465.90	243.69
	M&I	35.671	142.73	62.35	0.53	0.39	163.69	71.50	610.15	303.95
	INF	19.373	112.02	45.83	0.34	0.37	69.77	28.54	212.59	158.31
La Preciosa	MEA	-	-	-	-	-	-	-	-	-
	IND	17.441	202	176	0.34	-	113.14	98.59	189.19	-
	M&I	17.441	202	176	0.34	-	113.14	98.59	189.19	-
	INF	4.397	170	151	0.25	-	24.1	21.33	35.48	-
TOTALS	**MEA**	**8.466**	**142.35**	**71.72**	**0.53**	**0.32**	**38.75**	**19.52**	**144.26**	**60.24**
	IND	**44.645**	**165.87**	**104.89**	**0.46**	**0.25**	**238.08**	**150.56**	**655.09**	**243.69**
	M&I	**53.111**	**162.12**	**99.61**	**0.47**	**0.26**	**276.83**	**170.08**	**799.34**	**303.95**
	INF	**23.770**	**122.83**	**65.26**	**0.32**	**0.30**	**93.87**	**49.87**	**248.07**	**158.31**

Notes:
1. Figures may not add to totals shown due to rounding.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimate is classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves incorporated by reference into NI 43-101 Standards of Disclosure for Mineral Projects.
4. Mineral Resources are stated inclusive of Mineral Reserves
5. Based on recent mining costs (Section 21.0), Mineral Resources are reported at cut-off grades 60 g/t, 130 g/t, and 50 g/t AgEQ grade for ET, San Gonzalo, and oxide tailings, respectively.
6. AgEQ or silver equivalent ounces are notational, based on the combined value of metals expressed as silver ounces
7. Metal price assumptions are shown in Table 14-2
8. Metal recovery is based on operational results and column testing, shown in Table 14-2
9. The silver equivalent was back-calculated using the formulas described in Section 14.

The following table provides a synopsis of the Mineral Resources reported in this section. Table 1-3 summarizes the Mineral Resources (inclusive of Mineral Reserves) at the Avino Mine Area. The Mineral Resources exclusive of Mineral Reserves are presented in Section 14.1.1.

Table 1-3: Avino Mine – Mineral Resources (Inclusive of Mineral Reserves, Effective Date: October 16, 2023)

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
ET Avino	MEA	3.883	171	69	0.53	0.57	21.39	8.58	67.00	48.91
	IND	23.916	146	58	0.53	0.44	112.41	44.59	409.00	234.08
	M&I	27.800	150	60	0.53	0.46	133.80	53.17	476.00	283.00
	INF	17.591	106	37	0.34	0.40	59.76	20.72	191.00	154.18
San Gonzalo	MEA	0.331	332	244	1.17	0.00	3.53	2.59	12.42	0.00
	IND	0.302	293	230	0.84	0.00	2.85	2.23	8.14	0.00
	M&I	0.633	313	237	1.01	0.00	6.38	4.83	20.56	0.00
	INF	0.246	297	271	0.35	0.00	2.35	2.14	2.74	0.00
Guadalupe	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.401	169	70	0.79	0.37	2.17	0.90	10.24	3.27
	M&I	0.401	169	70	0.79	0.37	2.17	0.90	10.24	3.27
	INF	0.354	159	82	0.62	0.30	1.81	0.93	7.00	2.30
La Potosina	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.142	220	186	0.41	0.04	1.00	0.85	1.85	0.13
	M&I	0.142	220	186	0.41	0.04	1.00	0.85	1.85	0.13
	INF	0.844	176	149	0.29	0.05	4.79	4.05	7.90	1.01
Tailings Deposit	MEA	4.252	101	61	0.47	0.12	13.83	8.35	64.84	11.33
	IND	2.443	83	43	0.47	0.12	6.51	3.40	36.67	6.21
	M&I	6.695	94	55	0.47	0.12	20.34	11.75	101.50	17.55
	INF	0.338	97	65	0.36	0.11	1.06	0.70	3.95	0.82

table continues…

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
TOTALS	MEA	8.466	142.35	71.72	0.53	0.32	38.75	19.52	144.26	60.24
	IND	27.204	142.85	59.42	0.53	0.41	124.94	51.97	465.90	243.69
	M&I	35.671	142.73	62.35	0.53	0.39	163.69	71.50	610.15	303.95
	INF	19.373	112.02	45.83	0.34	0.37	69.77	28.54	212.59	158.31

Notes:

1. Figures may not add to totals shown due to rounding.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimate is classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves incorporated by reference into NI 43-101 Standards of Disclosure for Mineral Projects.
4. Based on recent mining costs (Section 21), Mineral Resources are reported at cut-off grades 60 g/t, 130 g/t, and 50 g/t AgEQ grade for ET, San Gonzalo, and oxide tailings, respectively.
5. AgEQ or silver equivalent ounces are notational, based on the combined value of metals expressed as silver ounces
6. Cut-off grades were calculated using the following consensus metal price assumptions: gold price of US$1,800/oz, silver price of US$21.00/oz, and copper price of US$3.50/lb.
7. Metal recovery is based on operational results and column testing, shown in Table 14-5.
8. The silver equivalent was back-calculated using the following formulas:
   a) ET, Guadalupe, La Potosina: AgEQ = Ag (g/t) + 71.43 * Au (g/t) + 113.04 * Cu (%)
   b) San Gonzalo: Ag Eq = Ag (g/t) + 75.39 * Au (g/t)
   c) Oxide Tailings: Ag Eq = Ag (g/t) + 81.53 * Au (g/t)

1.5 Mineral Reserve Estimate

The Mineral Reserves were estimated using both oxide and sulphide tailings and are based on Measured and Indicated Resources only. The pit design used for the estimation was at the PFS level. The ultimate pit limit was determined by the Lerchs-Grossman optimizer in Datamine™, with consideration of economic parameters and physical constraints such as pit road widths, mining bench width, and face angles for the recommended mining equipment.

The tailings material to be mined at the Avino Project was deposited upon bedrock. The visual difference between the bedrock and tailings material will allow for recovery of most of the tailings while minimizing potential dilution when mining the final bench on bedrock.

To account for this, a mining dilution of 1% and mining recovery of 99% is included within the pit optimization model.

Mineral reserves were classified based on resource categories defined during resource estimation. Measured Resources were converted to Proven Reserves, and Indicated Resources were converted to Probable Reserves. No Measured Resources were included within Probable Reserves. No Inferred Resources were included within the reserve classification.

Proven and Probable Mineral Reserves are summarized in Table 1-4.

Table 1-4: Mineral Reserve Statement of the Avino OxideTailings Project

Reserve Category	Quantity (Million tonnes)	Average Ag Grade (g/t)	Average Au Grade (g/t)	Contained Ag Metal (Million tr. Oz)	Contained Au Metal (Thousand tr. Oz)
Proven	4.27	61	0.47	8.37	65.01
Probable	2.43	43	0.47	3.38	36.53
Total	6.70	55	0.47	11.75	101.54

Notes:
1. The effective date of the Mineral Reserve estimate is January 16, 2024. The QP for the estimate is Junjie (Jay) Li, P.Eng. of Tetra Tech.
2. The Mineral Reserve estimates were prepared with reference to the 2014 CIM Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.
3. Reserves estimated assuming open pit mining methods.
4. Reserves are reported on a dry in-situ basis.
5. Reserves are based on a gold price of US $1,850/tr oz., and silver price of US $22/tr oz, mining cost of US$1.00/t mined, milling costs of US$18.00/t feed, and USG&A cost of US$3.00/t feed.
6. Mineral Reserve includes consideration for 1% mining dilution and 99% mining recovery.
7. Ore-waste cut-off was based on US$21.00/t of NSR.

1.6 Mineral Processing, Metallurgical Testing and Recovery Methods

1.6.1 Avino Mine Area

The Avino processing plant is currently processing materials from the Avino underground mine. The target metal values are gold, silver, and copper. The materials from the previous San Gonzalo Mine were processed from October 2012 to Q4 2019 with the target values of gold, silver, lead, and zinc. There are four grinding and flotation lines with a total capacity of 2,500 t/d, including two 1,000 t/d and one 250 t/d lines to recover copper, gold, and silver into a copper concentrate, and a separate 250 t/d line, which was used to produce materials from the previously operating San Gonzalo Mine, which has ceased operation since the end of 2019.

There is a potential tailings resource from previous operations; currently, there is no operation on tailings material.

1.6.1.1 Avino Vein

The Avino Vein material is currently being processed at the Avino processing plant using froth flotation to produce a marketable copper concentrate with silver and gold credits. A gravity concentration circuit was also incorporated in three of the four processing lines. The material has been successfully processed in the past.

The feed from the Avino Vein has been processed using froth flotation to produce a copper concentrate with silver and gold credits. In the 2023 operation, the average silver, gold, and copper recoveries reporting to a silver/gold/copper concentrate and a gravity concentrate were 87%, 72%, and 83%, respectively. The total material processed was 615,373 t.

Table 1-5: Total Production (Avino, 2024)

Description	2023	2022	2021*
Feed Tonnage			
Tonnes Milled (dry t)	615,373	541,823	165,304
Feed Grade			
Silver (g/t)	54	62	53
Gold (g/t)	0.51	0.42	0.84
Copper (%)	0.47	0.61	0.57
Recovery			
Silver (%)	87	92	87
Gold (%)	72	78	75
Copper (%)	83	89	88
Total Metal Produced			
Silver Produced (oz)	928,643	985,195	245,372
Gold Produced (oz)	7,335	5,778	3,386
Copper Produced (lbs)	5,304,808	6,504,177	1,869,306

Notes:
[1] After a period of operational suspension; the Avino Mine restarted production during Q3 2021.

1.6.1.2 Test Work on Tailings Materials

The test work on the tailings material is presented in Section 13 and are divided into four main sections as follows:

- Latest metallurgical test work conducted by SGS during 2022 and 2023
- Attrition and Filtration test work conducted by SGS 2023

- Historical test work conducted by Process Research Associates Inc. (PRA) under supervision of MMI between 2002 and 2003 – Oxide Tailings
- Historical test work conducted by PRA under supervision of MMI between 2002 and 2003 – Sulphide Tailings.

The test results indicate that the tailings samples responded well to cyanide leaching, including column leaching treatment. The 2022-2023 test program shows the following test results with regrinding to 80% passing approximately 75 µm:

- Early-stage Oxide Tailings Composite: silver and gold extractions were improved to approximately 90.4% and 88.3% respectively, compared to 82.7% for silver and 78.7% for gold without regrinding.
- Recent Oxide Tailings Composite, silver and gold extractions were improved to approximately 83% or slightly higher respectively, compared to 77.9% for silver and 76.6% for gold without regrinding.
- Sulphide Tailings Composite: silver and gold extractions were improved to approximately 76.1% and 82.8% respectively, compared to 69.1% for silver and 77.0% for gold without regrinding.

According to the tests results, the existing tailings will be processed at 2,250 t/d by tank cyanide leaching, followed by the Merrill-Crowe process to recover silver and gold. The residual material will be detoxified, filtered, and deposited in a lined dry stack tailings management facility.

The LOM average plant feed grade is estimated to be 54.5 g/t silver and 0.47 g/t gold. The LOM average silver and gold recovery is estimated to be 77.2% and 74.9%, respectively.

1.6.2 La Preciosa Area

Extensive metallurgical investigations were conducted to support the previous studies, including a feasibility study completed in 2014, *NI 43-101 Technical Report Feasibility Study for La Preciosa Silver-Gold Project,* prepared by M3 of Tucson, Arizona. Please refer to 2014 Feasibility Study Report (M3 Engineering & Technology Corporation 2013) for test work results on La Preciosa Area.

Further test work conducted in 2021 was focused on metallurgical response of the samples to conventional flotation concentration. The samples tested were from Abundancia, Gloria, and Martha mineralization zones. Please refer to 2023 Mineral Resource Estimate Update for the Avino Property, Durango, Mexico (Tetra Tech 2023) for 2021 test work results on La Preciosa Area.

1.7 Mining Methods

1.7.1 Avino Area

1.7.1.1 Avino Vein

Avino is currently conducting mining activity on the Avino Vein using sublevel long hole stoping and room and pillar mining methods. The last three years of production from the Avino Mine (mill feed) are summarized in Table 1-5. This data is summarized from the information listed in Avino's press releases.

1.7.1.2 San Gonzalo Vein

Avino reported that by Q4 2019, mining at the San Gonzalo Vein reached the end of its current resources, and underground mining activities at the mine were stopped. However, the mine remains open for continued exploration at different levels of the mine. No operation for the San Gonzalo Vein's mineralization has been reported since 2020.

1.7.1.3 Oxide Tailings Extraction

The Avino Oxide Tailings Project will be extracted using conventional surface mining techniques with an excavator, wheel loader, and trucks, operating 365 days per year with three 8-hour shifts per day. Based on the Measured and Indicated Resource, the mine plan includes transporting ore to the ROM feed located north of the deposit, and the waste material will be placed at a waste storage structure located east of the Project. Equipment selection and requirements are based on the existing equipment list provided by Avino and the design parameters of the pit and annual material movement, respectively. Refer to Section 15 for the geotechnical pit slope parameters.

Mining phases (pushbacks or PB) have been designed and incorporated into the mining sequence to bring higher grade material forward and to defer waste stripping. The phase designs were guided by the lower revenue factor pit shells from the pit optimization analysis. A total of five phases have been designed. The mining phases were designed to provide operational flexibility while meeting geotechnical constraints and mining equipment available on site. For the LOM, there are at least two active phases being mined at one time to reduce the operational risk from geotechnical failures. Figure 16-1 shows the pushback schedule for the LOM.

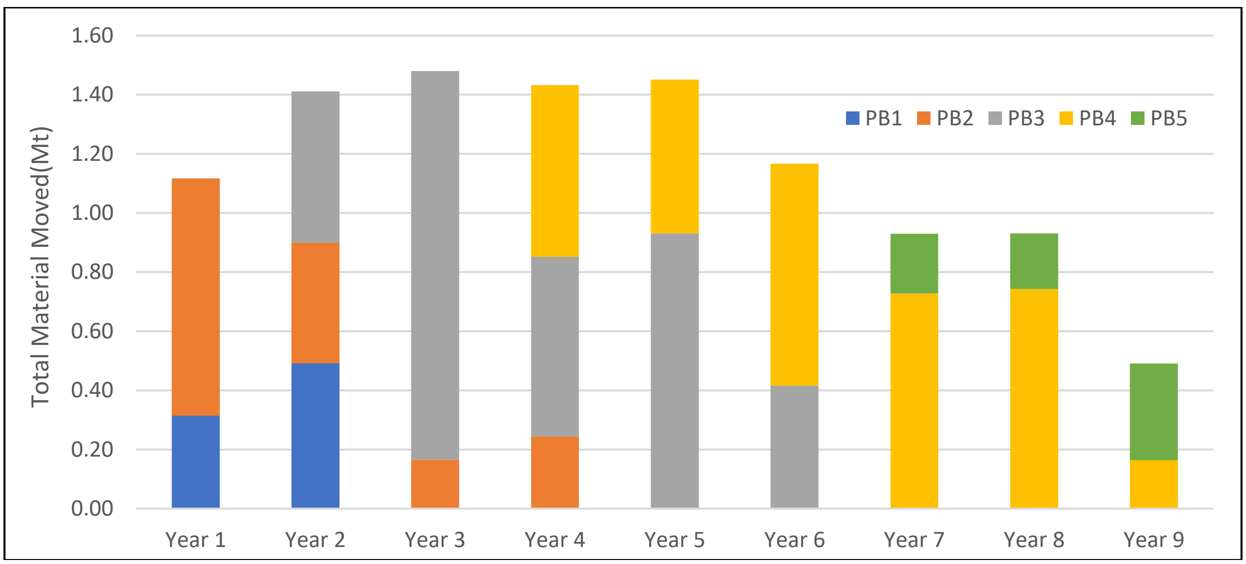


Figure 1-3: Mining Phases Schedule

Over the LOM, 3.7 Mt of waste material will be stored in the mine waste management facility located east of the mining area, sitting at a final elevation of 2220 masl. Dumps are designed to minimize haulage distances from the pits while also honoring geotechnical offsets from the ultimate pit and infrastructure. Further details of waste management are presented in Section 16.

The mine production plan has been prepared using Datamine's TM Studio OP and NPVS software. Provided with economic input parameters and operational constraints such as phase sequencing, maximum bench sink rates, mining and milling capacities, the software determines the optimal mining sequence.

The mine life of the project is expected to be approximately 9 years. The mining rate will ramp up to around 1.4 Mt in years 2 to year 5 to accommodate a high strip ratio to remove the majority of the overburden and will start to ramp down in later years as the strip ratio decreases. Table 16-1 tabulates the life of mine schedule and grade.

Mining is to be carried out using conventional surface mining techniques with an excavator, wheel loader, and trucks in a bulk mining approach with 5 m benches. Equipment requirements are based on the design parameters of the pit and production rate requirements. Equipment availability and utilization is based on Tetra Tech's experience and vendor guidance.

A total of four 125hp class submersible pumps and piping system will be used for surface dewatering in the oxide tailings extraction area.

Over the life of mine, excluding the contracted workers, the average labour complement will be 22 full-time equivalent personnel in oxide tailings extraction.

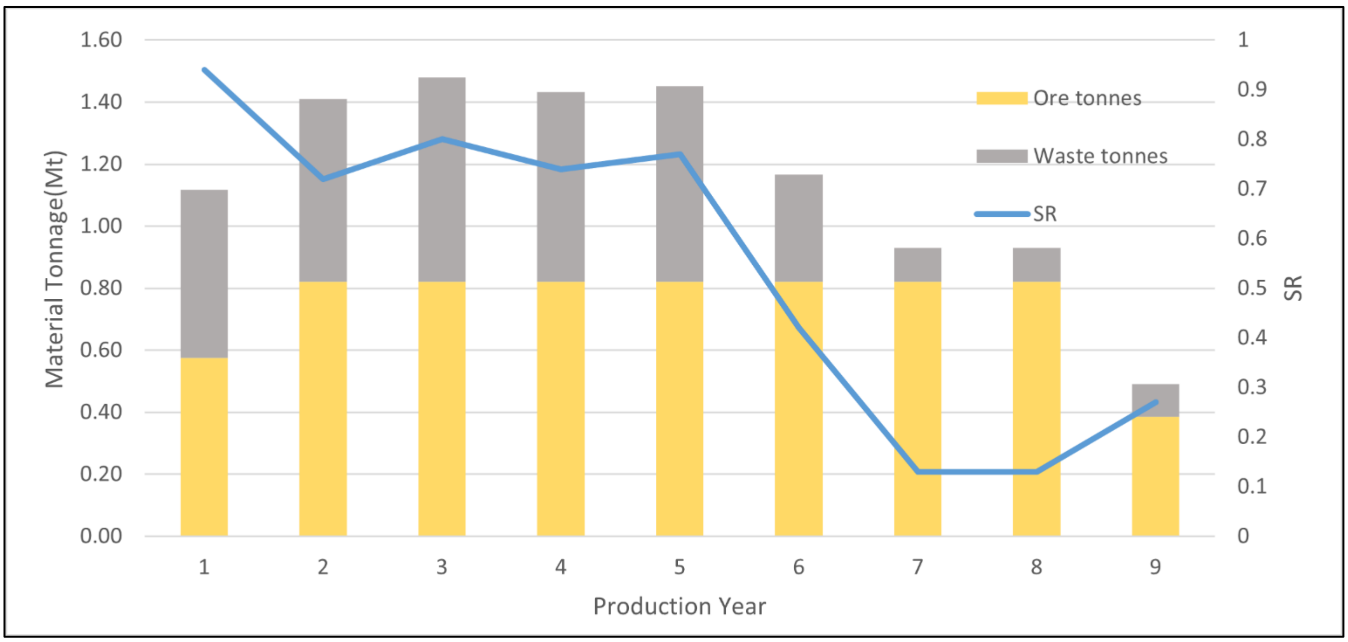


Figure 1-4: Ore and Waste Mining Schedule

1.7.2 La Preciosa Area

Currently, there are no commercial operations in the La Preciosa area.

1.8 Project Infrastructure

The Property is easily accessible by road and is an important part of the local community from which skilled workers are available. The history of operations at the Avino mine site provides ample evidence of sufficient infrastructure and services in the area. The San Gonzalo Mine entered commercial production in October 2012, followed by the reopening of the ET Mine in January 2015. Currently, only the ET Mine is in operation, and the mined materials are fed to a conventional flotation plant with four separate circuits. The processing plant, including crushing, grinding, flotation, and downstream dewatering processes, had been upgraded from 1,500 t/d to a total capacity of 2,500 t/d in 2017/18.

The offices, miner's quarters, secured explosives storage facilities, warehouse, laboratory, and other associated facilities are all in place. The proposed tailings leach facilities are planned to be located southeast of the existing tailings storage pond.

There is a water treatment plant (WTP) for treating excess water from the Avino underground mine operation before discharging it to El Caracol Dam. The effluent is sampled daily when the WTP is operational.

1.8.1 Oxide Tailings Project Infrastructure

The major areas of OTP facilities consist the following:

1. Plant Feed Stockpile area comprises a 25 m diameter (3,700 t) ROM plant feed stockpile. LHD trucks coming from the historic TMF offload the plant feed to the plant feeding dump hopper or to the plant feed stockpile, which is reclaimed into the trommel feed conveyor hopper for feeding the processing plant.

2. Processing Plant area consists of the primary thickening, primary leaching & intermediate thickening, secondary leaching & counter current decantation washing, Merrill-Crowe and tailings detox/filtration circuit to facility silver and gold extraction and production on site, support by ancillary process circuits and utilities such as reagent preparation circuit, power substation and distribution, water services and compressed air services.

3. Dry Stack TMF area comprise the Dry Stack Tailings Management Facility (DSTMF) and the material transport conveyors between tailings dewatering facilities in the process plant and dry stacking area. Contact water collected in the DSTMF and processing plant areas will be sedimented prior to being pumped to the process water tank for process use.

4. Ancillary facilities which service the existing operation will also service for the tailings reprocessing. These facilities consist of the administration building, maintenance shop, storage warehouse, assay laboratory, and diesel fuel storage.

   According to the information provided by Avino, the current total running power available to Avino operation is 7 MW, 4.5 MW of which is being consumed by the current operation, therefore, 2.5 MW excess power is available. The new tailings reprocessing operation is expected to consume 1.8 MW running power, which can be fulfilled by the existing power supply system.

A modular, prepackaged E-house will function as a power distribution center in the area. The E-house will be located close to the pre-leaching screening circuit. Step down transformers reduce the high voltage from the site power supply to 4,160V, 600V and 240/120V to power equipment and facilities. A 350kW emergency diesel generator provides critical back up power in the event of power outage. Sensitive electronic equipment will have surge protection and uninterrupted power supply (UPS).

1.8.2 Dry Stack Tailings Management Facility

The DSTMF involves dewatering of the slurry tailings using large scale pressure filters and deposition of the resulting tailings 'filter cake' in an engineered filtered DSTMF. DSTMF was selected for the Project due to the following benefits:

- A DSTMF occupies a smaller footprint compared to a conventional slurry tailings impoundment due to the higher densities achieved by dewatered tailings.
- Pressure filtration of tailings results in improved process water recycling, which reduces freshwater demand for the process plant.
- The unsaturated nature of dewatered tailings reduces infiltration through the tailings mass and into the foundation.
- Structural failures of dewatered tailings are less likely to have significant environmental impact and runout distance due to the absence of a water pond and the low pore pressures within the tailings mass.
- The dewatered tailings management option allows concurrent reclamation of the DSTMF, thus reducing potential environmental impacts, notably fugitive dust.
- The dewatered tailings management option minimizes post-closure long-term water management requirements.

The DSTMF was designed to accommodate 6.7 Mt of tailings at an assumed in-situ tailings dry density of 1.5 t/m^3 over the operational life of 9 years. The proposed geometry and key features of the DSTMF are shown in Figure 18-4.

The tailings to be stored on dry stack will be thickened, filtered, and transported by a series of conveyor and stacked at DSTMF, located west of the process plant. As shown in Figure 18-4, the proposed DSTMF is located in the valley west of the proposed OTP plant. The total footprint of the DSTMF is approximately 43 acres (174,015 m^2), which is situated in proximity to the processing plant, minimizing the distance required for tailings transport.

1.8.3 Site Water Management

A comprehensive stormwater management plan has been developed for the Avino mine site to effectively manage both contact and non-contact water. This plan includes the installation of drainage swales at strategic locations to prevent the mixing of contact water with non-contact watercourses. The proposed drainage swales are presented in Figure 18-7 as yellow channels; existing, naturally formed channels are presented as pink channels; and existing smaller watercourses are presented as magenta dotted lines.

In addition, water storage ponds are proposed at selective outfall locations throughout the mine site to store either contact or non-contact water and are presented in Figure 18-8 and Figure 18-9. Dependent on whether the water being intercepted is contact or non-contact, the water storage ponds serve the following purposes:

- Contact Water: capture for process uses or onsite treatment prior to release into the natural environment; or
- Non-Contact Water: capture and prevent from entering the mine influenced area and become contaminated. Tetra Tech recommends a pump or gravity driven system to be connected to the storage pond to safely convey and discharge into a nearby natural drainage path outside of the mine-influenced area.

Corrugated steel pipe culverts are proposed at natural-stream roadway crossings. Figure 18-10, Figure 18-11, and Figure 18-12 present the locations of these culverts along with the corresponding configuration to safely convey the 24-hour design storm event.

1.9 Environmental

Environmental settings, permits and registrations, and environmental management strategies that may be required for the Project are summarized in Section 20. Permits and authorizations required for the operation of the Project may include an operating permit, an application for surface tenures, a wastewater discharge registration, a hazardous waste generator's registration, and an Environmental Impact Assessment (EIA) or Evaluación de Impacto Ambiental. Acid-base accounting (ABA) tests have indicated that mild acid generation may already have started on the tailings dam.

1.10 Capital and Operating Costs

1.10.1 Avino Current Operation

Avino is currently conducting mining activity, including mineral processing, on the materials from the Avino Mine. There is no cost estimate applicable for the ongoing operations, and all costs below are based on actual expenditure, excluding the proposed Oxide Tailings Project.

1.10.1.1 Capital Costs

The actual capital expenditures for the last three years on the Avino Vein are summarized in Table 1-6. The San Gonzalo Mine ceased its operation at the end of 2019. Mine and mill capital costs were mainly attributed to equipment purchases, construction and site upgrading.

Table 1-6: Capital Costs for the Avino Mine (US$ in 000s) (Source: Avino, 2024)

Description	2023	2022	2021
Office Furniture	78	108	31
Computer and Communication/Automation Enhancement	1,177	136	13
Mill Machinery and Processing Equipment	3,080	4,781	1,130
Mine Machinery and Transportation Equipment	3,271	2,181	1,337
Buildings and Construction	1,042	360	445
ET Mineral Property – Avino	4,827	1,649	(113)
Total Capital Costs	13,475	9,215	2,843

1.10.1.2 Operating Costs

The mine and milling operating costs for processing materials from the Avino Mine and historical stockpiles are summarized in Table 1-7. The costs include operating and maintenance labour together with the operation-associated consumable supplies. The cost of electrical power was included in the milling costs. The geological component was mostly related to technical labour. The San Gonzalo Mine ceased its operation at the end of 2019. As part of the ramp-up of operations, 10,806 tonnes of AHAG stockpile material were processed during Q3 2021.

Table 1-7: Operating Costs for Avino Mine (US$ in 000s) (Source: Avino, 2024)

Description	2023	2022	2021
Mining Cost	15,883	13,767	2,683
Milling Cost	10,667	7,486	1,467
Geological and Other	5,280	3,989	1,152
Royalties	1,456	1,505	403
Depletion and Depreciation	2,704	2,046	1,976
Total Direct Costs	**35,990**	**28,793**	**7,681**
G&A*	7,888	7,179	5,084
Total Operating Costs	**43,878**	**35,972**	**12,765**

Note: *G&A = General & Administration

1.10.2 Oxide Tailings Project

The capital and operating costs for retreating the OTP portion of the Property, including reclaiming the historical tailings and constructing the tailings reprocessing and dry stack tailings management facility, were estimated and presented in subsequent sections.

1.10.2.1 Project Initial Capital Cost Estimate

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $49.1 million. This total includes all direct costs, indirect costs, owner's costs, and contingency. This Class 4 cost estimate has been prepared according to AACE International (2020) standards. The expected accuracy range of this initial capital cost estimate is ±25%. Details of the initial capital costs are presented in Section 21.2.

Table 1-8: Initial Capital Cost Summary

Description	Cost (Million $)
Site Preparation, Excavations & Demolition	0.2
Mining (Oxide Tailings Reclaim)	0.5
Processing Plant	26.8
TMF and Water Management	3.3
Site Services and Utilities	4.6
Total Direct Initial Capital Cost	35.3
Indirect Initial Capital Costs	7.8
Owner's Cost	0.7
Contingency	5.3
Total Initial Capital Cost	**49.1**

Note: Sums may not add due to rounding.

1.10.2.2 Project Sustaining Capital Cost Estimate

The sustaining capital costs are all required from Year 1 of operations to sustain the mining operation for the LOM and are estimated to be $5.1 million for the LOM, including the closure and reclamation costs. Details of the total sustaining cost are presented in Section 21.2.7.

Table 1-9: Sustaining Capital Costs Summary

Description	Cost (Million $)
Mining Equipment	2.0
TMF	1.5
Water Management	0.5
Total Operating Sustaining Capital Costs	**4.0**
Closure & Reclamation	1.1
Total	**5.1**

Note: Sums may not add up due to rounding.

1.10.2.3 Project Operating Cost Estimate

The project operating cost estimate consists of mining, processing, tailings management, and G&A costs, which are summarized in Table 1-10. The average LOM operating cost is estimated to be $21.34/t processed.

Table 1-10: Project Average LOM Operating Cost Summary

Description	LOM Cost (million $)	Unit Cost ($/t processed)
Mining Equipment	16.2	2.41
Processing	102.7	15.31
G&A – Onsite (including Site Services)	22.2	3.31
Tailings Management	2.2	0.32
Total Operating Cost	**143.2**	**21.34**

Note: Sums may not add up due to rounding.

1.11 Economic Analysis

1.11.1 Avino Vein – Current Operation

Avino is currently conducting mining activity, including mineral processing and concentrate production, on the materials from the Avino Vein. There is no economic analysis performed for this vein.

Avino has not based its production decisions on any FS or Mineral Reserves demonstrating economic and technical viability, and as a result, there is increased uncertainty and multiple technical and economic risks of failure that are associated with these production decisions. These risks, among others, include areas that would be analyzed in more detail in an FS, such as applying economic analysis to Mineral Resources and Mineral Reserves, more detailed metallurgy, and a number of specialized studies in areas such as mining and recovery methods, market analysis, and environmental and community impacts. Information in this section was provided by Avino.

1.11.2 Oxide Tailings Project

Tetra Tech prepared an economic evaluation of the Oxide Tailings Project PFS based on a pre-tax and a post-tax basis. For the 9-year mine life and 6.7 Mt Mineral Reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 35% IRR
- 2.9-year payback period
- $98 Million net present value (NPV) at a 5% discount rate

Taxes and depreciation for the Project were modelled based on the inputs from tax consultants engaged by Avino. The following post-tax financial results were calculated:

- 26% IRR
- 3.5-year payback period
- $61 million NPV at a 5% discount rate

The post-tax discounted annual cash flow and cumulative net cash flow are presented in Figure 1-5.

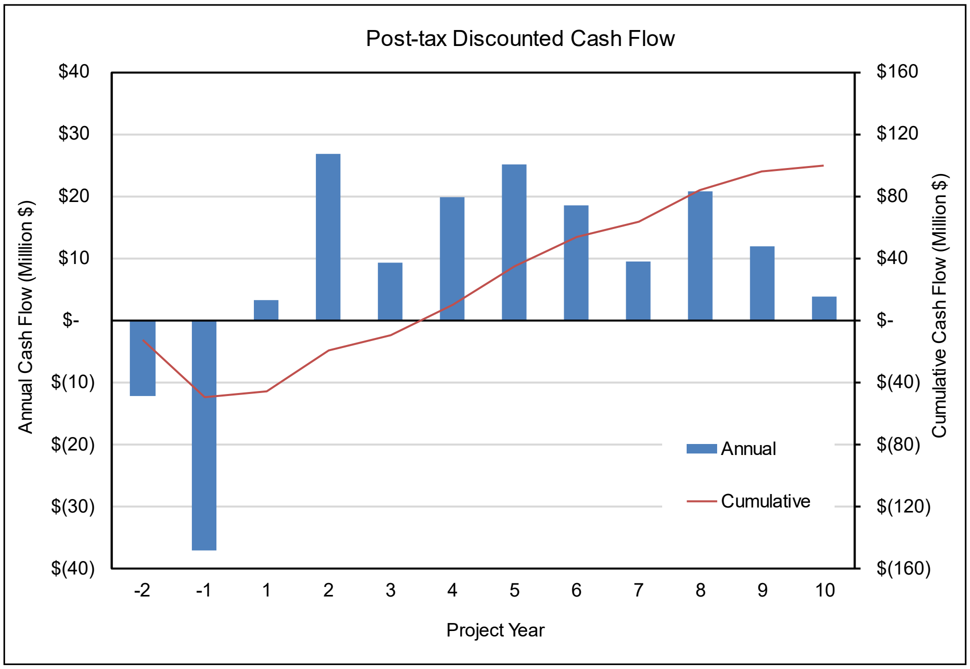


Figure 1-5: Discounted Post-Tax Annual and Cumulative Cash Flow

Sensitivity analyses and additional metal price scenarios were also developed to evaluate the 2024 PFS economics.

The analyses are presented graphically as financial outcomes regarding post-tax NPV and IRR. The NPV is most sensitive to silver price, followed by capital cost, gold price, and operating cost, while IRR is most sensitive to capital cost, followed by silver price, gold price, and operating cost. Generally, sensitivity to metal price is roughly equivalent to sensitivity to metal grade (in Figure 1-6 and Figure 1-7, respectively).

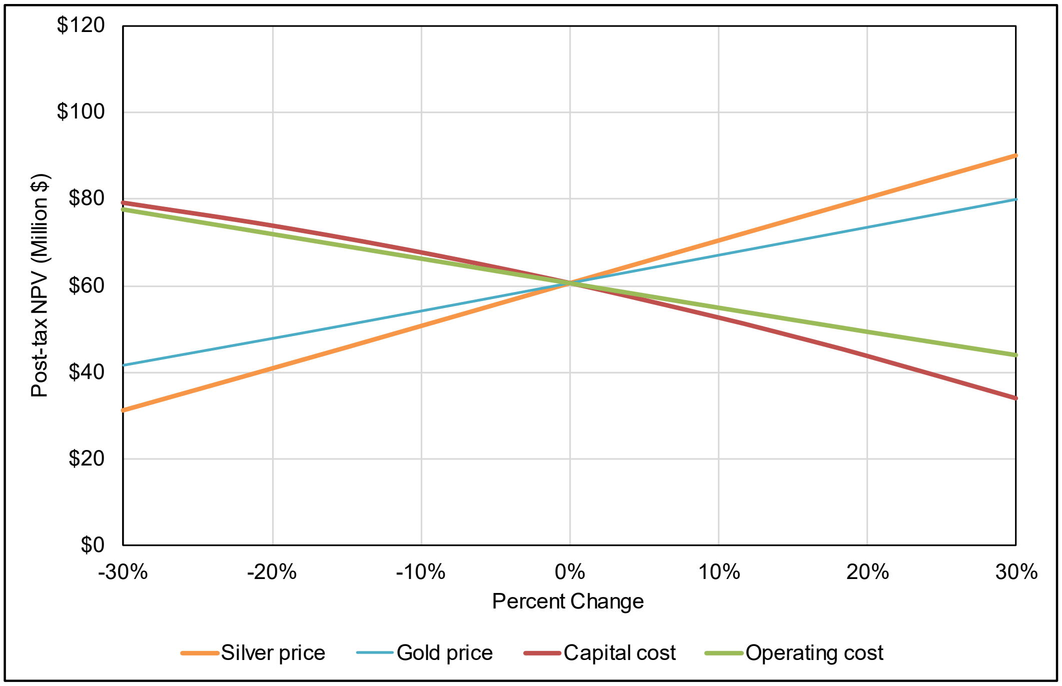


Figure 1-6: Sensitivity Analysis of Post-Tax NPV

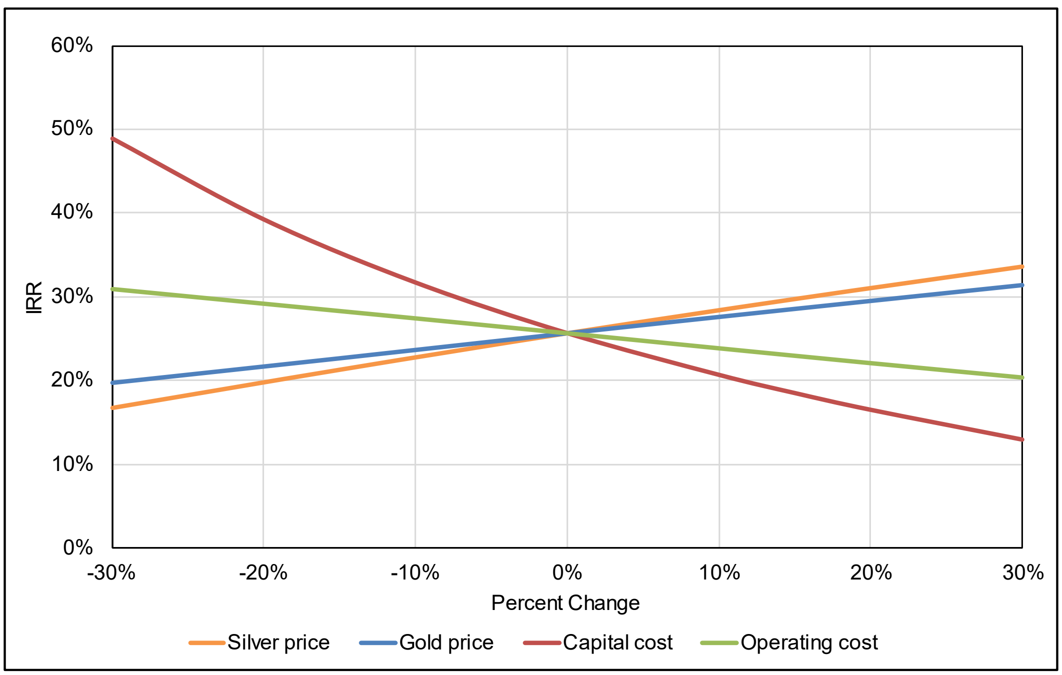


Figure 1-7: Sensitivity Analysis of Post-Tax IRR

1.11.3 Forward-looking Statements

This document contains "forward-looking information" within the meaning of Canadian securities legislation and "forward-looking statements" within the United States Private Securities Litigation Reform Act of 1995. This information and these statements, referred to herein as “forward-looking statements”, are made as of the date of this document. Forward-looking statements relate to future events or future performance and reflect current estimates, predictions, expectations, or beliefs regarding future events and include, but are not limited to, statements with respect to:

- The estimated amount and grade of Mineral Reserves and Mineral Resources.
- Estimates of the capital costs of constructing mine facilities and bringing a mine into production, operating the mine, sustaining capital, and the duration of payback periods.
- The estimated amount of future production, both material processed and metal recovered.
- Estimates of operating costs, the life of mine costs, net cash flow, NPV, and economic returns from an operating mine.
- The assumptions on which the various estimates are made are reasonable.

All forward-looking statements are based on the authors’ current beliefs, their various assumptions, and the information currently available to them. These assumptions are set forth throughout this Report, and some of the principal assumptions include:

- The presence of and continuity of metals at estimated grades.
- The geotechnical and metallurgical characteristics of rock conforming to sampled results.
- The water quantities and quality available during mining operations.
- The capacities and durability of various machinery and equipment.
- Anticipated mining losses and dilution.
- Metallurgical performance.
- Reasonable contingency amounts.

Although the QPs consider these assumptions reasonable based on currently available information, they may prove incorrect. Many forward-looking statements assume the correctness of other forward-looking statements, such as statements of net present value and internal rates of return, which are also based on most other forward-looking statements and assumptions herein.

By their very nature, forward-looking statements involve inherent risks and uncertainties, both general and specific, and risks exist that estimates, forecasts, projections, and other forward-looking statements may not be achieved or that assumptions do not reflect future experience.

1.12 Recommendations

Based on the information, data and conclusions presented in this Technical Report, it is recommended a feasibility study to be conducted for the OTP to study the Project in further details. Other recommendations are presented in Section 26. The budgetary cost summary for PFS recommendations is presented in Table 1-11.

Table 1-11: Budgetary Cost Summary for PFS Recommendations

Description	Budget (CDN$)
Geology	2,000,000
Mining	600,000
Metallurgy	330,000
Tailings Geotechnical Characterization	100,000
Environmental	80,000
Total	**3,110,000**

2.0 INTRODUCTION

Avino is a Canadian-based mining and exploration company listed on the TSX and the NYSE-American trading under the symbol ASM. Avino has precious metal properties in Mexico and Canada and has a head office located at 900-570 Granville Street, Vancouver, British Columbia V6C 3P1, Canada.

Avino retained Tetra Tech Inc. (Tetra Tech), in conjunction with Red Pennant Communications Corp. (Red Pennant), to prepare this Pre-Feasibility NI 43-101 Technical Report which aims to disclose the PFS for The Oxide Tailings Project. This Technical Report has been prepared in accordance with National Instrument 43-101 (NI 43-101) and Form 43-101F1.

2.1 Effective Dates

The effective date of this Technical Report is February 5, 2024. The effective date of the Mineral Resource estimate is October 16, 2023. The effective date of the Mineral Reserve estimate is January 16, 2024.

Latest information on mineral tenure, surface rights, and Property ownership: January 3, 2024.

2.2 Personal Inspections and Qualified Persons (QPs)

A summary of the QPs responsible for this Technical Report is provided in Table 2-1. The following QPs conducted personal inspections of the Property:

- Michael F. O'Brien, P.Geo., M.Sc., Pr.Scit.Nat., FAusIMM, FSAIMM, conducted personal inspections of the site from June 12 to 15, 2017, February 12 to 14, 2020, and July 20, 2021.
- Hassan Ghaffari, P.Eng., M.A.Sc., conducted personal inspections of the Avino property most recently on June 27, 2023 and on March 30, 2011, December 12, 2017, from August 12 to 14, 2019 and July 20, 2021.
- Jianhui (John) Huang, P.Eng., Ph.D., conducted a personal inspection of the site on June 27, 2023.
- Junjie (Jay) Li, P.Eng., conducted a personal inspection of the site on June 27, 2023.

Table 2-1: Summary of Qualified Persons

No.	Report Section	Company	Qualified Person
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
3.0	Reliance on Other Experts		
4.0	Property Description and Location	Red Pennant	Michael F. O'Brien, P.Geo., M.Sc., Pr.Sci.Nat., FAusIMM, FSAIMM
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography		
6.0	History		
7.0	Geological Setting and Mineralization		
8.0	Deposit Types		
9.0	Exploration		
10.0	Drilling		
11.0	Sample Preparation, Analyses and Security		
12.0	Data Verification		
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
14.0	Mineral Resource Estimates	Red Pennant	Michael F. O'Brien, P.Geo., M.Sc., Pr.Sci.Nat., FAusIMM, FSAIMM
15.0	Mineral Reserve Estimate	Tetra Tech	Junjie (Jay) Li, P.Eng.
16.0	Mining Methods		
17.0	Recovery Methods		Jianhui (John) Huang, Ph.D., P.Eng.
18.0	Project Infrastructure		Hassan Ghaffari, P.Eng., M.A.Sc.
19.0	Market Studies and Contracts		Jianhui (John) Huang, Ph.D., P.Eng.
20.0	Environmental Studies, Permitting and Social or Community Impact		Hassan Ghaffari, P.Eng., M.A.Sc.
21.0	Capital and Operating Costs		-
	Capital Costs (excl. Mining)		Hassan Ghaffari, P.Eng., M.A.Sc.
	Operating Costs (excl. Mining)		Jianhui (John) Huang, Ph.D., P.Eng.
	Mining Capital and Operating Costs		Junjie (Jay) Li, P.Eng.
22.0	Economic Analysis		Hassan Ghaffari, P.Eng., M.A.Sc.
23.0	Adjacent Properties	Red Pennant	Michael F. O'Brien, P.Geo., M.Sc., Pr.Sci.Nat., FAusIMM, FSAIMM.
24.0	Other Relevant Data and Information	Tetra Tech	Hassan Ghaffari, P.Eng., M.A.Sc.
25.0	Interpretation and Conclusions	All	Sign-off by Section
26.0	Recommendations	All	
27.0	References	All	
28.0	Certificates of Qualified Persons	All	

2.3 Information and Data Sources

In preparation for this Technical Report, various historical engineering, geological, and management reports compiled about the Project or site were reviewed and supplemented by direct site examinations and investigations. All the data files reviewed for this Technical Report were provided by Avino in the form of hard copy documents, electronic .pdf reports, .xls files, email correspondence, and personal communication with management and personnel from Avino. Work completed by Avino includes several decades of open pit and underground mining, drilling and sampling, trenching, metallurgical testing, and geophysical surveying.

A complete list of references is provided in Section 27.

2.4 Units of Measurement

All units of measurement used in this Technical Report are in metric units, and the currency is expressed in US dollars unless otherwise stated.

3.0 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by Hassan Ghaffari, P.Eng., M.A.Sc., Jianhui (John) Huang, Ph.D., P.Eng., Junjie (Jay) Li, P.Eng., of Tetra Tech, and Michael O'Brien, P.Geo., M.Sc., Pr.Scit.Nat., FAusIMM, FSAIMM, of Red Pennant Communications Corp. All authors are independent QPs as defined within the requirements of NI 43-101.

Michael F. O'Brien, P.Geo., Pr.Sci.Nat., FAusIMM, FSAIMM relied on the information provided by Avino regarding the claims, which comprise the Avino Property, their ownership, and their status in Section 4.

The information, conclusions, opinions, and estimates contained herein are based on:

- The information available to the authors at the time the report was prepared,
- Assumptions, conditions, and qualifications as outlined in this Technical Report,
- Production and expenditure data, reports, and other information supplied by Avino and other third-party sources.

Avino reported to the authors that, to the best of its knowledge, there is no known litigation that could potentially affect the Project.

Jianhui (John) Huang, Ph.D., P.Eng., relied on Avino for concentrate marketing related information as described in Section 19.

Hassan Ghaffari, P.Eng., M.A.Sc., relied on Avino's environmental consultant for environmental studies, permitting and social or community impact related information as described in Section 20.

Jianhui (John) Huang, Ph.D., P.Eng., relied on Avino's taxation consultant for post-tax financial analysis related information as described in Section 22.

Note: The authors of this Technical Report are not qualified to provide extensive commentary on legal or political issues associated with the Property, which are considered outside the scope of this Technical Report. For the portions of this Technical Report (Sections 1.2, 4.1, and 4.2) that deal with the types and numbers of mineral tenures and licenses, the nature and extent of title and interest in the Property, and the terms of any royalties, back-in rights, payments, or other agreements and encumbrances to which the Property is subject, we have relied upon the title opinion provided by Avino.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Property is located in the state of Durango, Mexico, within the municipalities of Pánuco de Coronado and Canatlán, and is approximately 85 km by existing road, northeast of the city of Victoria de Durango, the state capital. The Project is situated on the eastern flank of the Sierra Madre Occidental Mountain range. It can be found on the Instituto Nacional de Estadística, Geografía e Informática General Carlos Real Topographic Map G13D72, centred on coordinates 24°25'42.4200"N Latitude and 104°27'27.2380"W Longitude (554,987.8815 mE, 2,701,771.0046 mN) in the Universal Transverse Mercator (WGS 84), Zone 13R (Northern Hemisphere).

Figure 4-1: General Location of the Property (Avino 2021)

The current property consists of two parts:

- The historic Avino Mine concessions contain the Elena Tolosa and San Gonzalo mine workings, processing infrastructure, and associated vein systems.
- The newly acquired (March 21, 2022) La Preciosa Mining concessions contain the La Preciosa Veins.

Figure 4-2 shows the relative situation of these elements.

Figure 4-2: General Location of Avino and La Preciosa Concessions (Avino 2023)

4.1 Avino Mine Area

4.1.1 Property Ownership

The Avino part of the Property comprises 26 mineral concessions, totalling 1,301.4314 ha.

In 1968, Avino Mines and Resources Ltd. acquired a 49% interest in CMMA and Minera San José de Avino SA, which together held mineral claims totalling 2,626 ha (6,488 ac). Avino Mines and Resources Ltd. retained Vancouver-based Cannon-Hicks & Associates Ltd. (Cannon-Hicks), a mining consulting firm, to conduct the exploration and development of the Property. Cannon-Hicks's exploration activities included surface and underground sampling and diamond drilling (VSE 1979).

On July 17, 2006, the Company completed the acquisition of Compañía Minera Mexicana de Avino, S.A. de C.V. (Avino Mexico), a Mexican corporation, through the acquisition of an additional 39.25% interest in Avino Mexico, which, combined with the Company's pre-existing 49% share of Avino Mexico, brought the Company's ownership interest in Avino Mexico to 88.25%. The additional 39.25% interest in Avino Mexico was obtained through the acquisition of 79.09% of the common shares of Promotora, which in turn owns 49.75% of Avino Mexico's common shares, and the direct acquisition of 1% of the common shares of Avino Mexico.

July 17, 2006, the acquisition was accomplished by a share exchange by which the Company issued 3,164,702 shares as consideration, which we refer to as the “Payment Shares”, for the purchase of the additional 39.25% interest in Avino Mexico. The Payment Shares were valued based on the July 17, 2006, closing market price of the Company’s shares on the TSX.

The Company acquired a further 1.1% interest in Avino Mexico through the acquisition from an estate subject to approval and transfer of the shares to the Company by the trustee for the estate. On December 21, 2007, approval was received, and the Company obtained the 1.1% interest from the estate for no additional consideration.

On February 16, 2009, the Company converted existing loans advanced to Avino Mexico into new additional shares of Avino Mexico. As a result, the Company’s ownership interest in Avino Mexico increased to 99.28%.

On June 4, 2013, the Company converted existing loans advanced to Avino Mexico into new additional shares of Avino Mexico, resulting in the Company’s ownership increasing by 0.38% to an effective 99.67%. The issuance of shares to the Company by Avino Mexico on June 4, 2013, resulted in a reduction in the non-controlling interest from 0.72% to 0.34%.

On August 26, 2015, the Company converted existing loans advanced to Avino Mexico into new additional shares, resulting in an increase of the Company’s ownership by 0.01% to an effective 99.67%. The intercompany loans and investments are eliminated upon consolidation of the financial statements. The Company had a pre-existing effective ownership interest of 99.66% in Avino, Mexico, prior to the 0.01% increase. The issuance of shares to the Company by Avino Mexico on August 26, 2015, resulted in a reduction in the non-controlling interest from 0.34% to 0.33%.

4.1.2 Mineral Concessions and Agreements

The Avino part of the Property comprises 26 mineral concessions, totalling 1,301.4314 ha (Figure 4-3). Ownership proportions and mineral concessions are summarized in Table 4-1 and Table 4-2, respectively.

Table 4-1: Summary of Property Ownership

Company	Relationship to Avino Silver and Gold Mines Ltd.	Effective Ownership of Avino Mine Property (%)
CMMA	Subsidiary	98.45
Promotora	Subsidiary	1.22
Total Effective Ownership of Avino Mine Property	-	99.67
Estate of Ysita	Non-controlling interest	0.33
Total	**-**	**100.00**

Table 4-2: Mineral Concessions – Avino Area (Avino 2023)

S. No.	Concession Name	Concession No.	Area (Ha)	Expiration date
1	AMPLIACION DE LA POTOSINA	185326	84.0000	December 14, 2039
2	AMPLIACION SAN GONZALO	191837	5.8495	December 19, 2041
3	AMPLIACION LA MALINCHE	204177	6.0103	December 18, 2046
4	EL POTRERITO	185328	9.0000	December 14, 2039
5	LA MALINCHE	203256	9.0000	June 28, 2046
6	LA POTOSINA	185336	16.0000	December 14, 2039
7	SAN GONZALO	190748	12.0000	April 29, 2041
8	YOLANDA	191083	43.4577	April 29, 2041
9	AGRUP. SAN JOSE	164985	8.0000	August 13, 2029
10	AGRUP. SAN JOSE, (EL TROMPO)	184397	81.5466	October 13, 2039
11	AGRUP. SAN JOSE, (GRAN LUCERO)	189477	161.4684	December 5, 2040
12	AGRUP. SAN JOSE, (PURISIMA CHICA)	155597	136.7076	September 30, 2071
13	AGRUP. SAN JOSE, (SAN CARLOS)	117411	4.4505	December 5, 2061
14	AGRUP. SAN JOSE, (SAN PEDRO Y SAN PABLO)	139615	12.0000	June 22, 2061
15	AGUILA MEXICANA	215733	36.7681	March 12, 2054
16	ARANJUEZ	214612	96.0000	October 2, 2051
17	AVINO GRANDE IX	216005	19.5576	April 2, 2052
18	AVINO GRANDE VIII	215224	22.8816	February 14, 2052
19	EL CARACOL	215732	102.3821	March 12, 2052
20	EL FUERTE	216103	100.3274	April 9, 2052
21	FERNANDO	205401	72.1287	August 29, 2047
22	LA ESTELA	179658	14.0000	December 11, 2036
23	LOS ANGELES	154410	23.7130	March 25, 2071
24	NEGRO JOSE	218252	58.0000	October 17, 2052
25	SAN MARTIN DE PORRES	222909	30.0000	September 15, 2054
26	SANTA ANA	195678	136.1823	September 14, 2042
--	**TOTAL**	**--**	**1,301.4314**	**--**

Notes: Figures may not add to totals shown due to rounding.

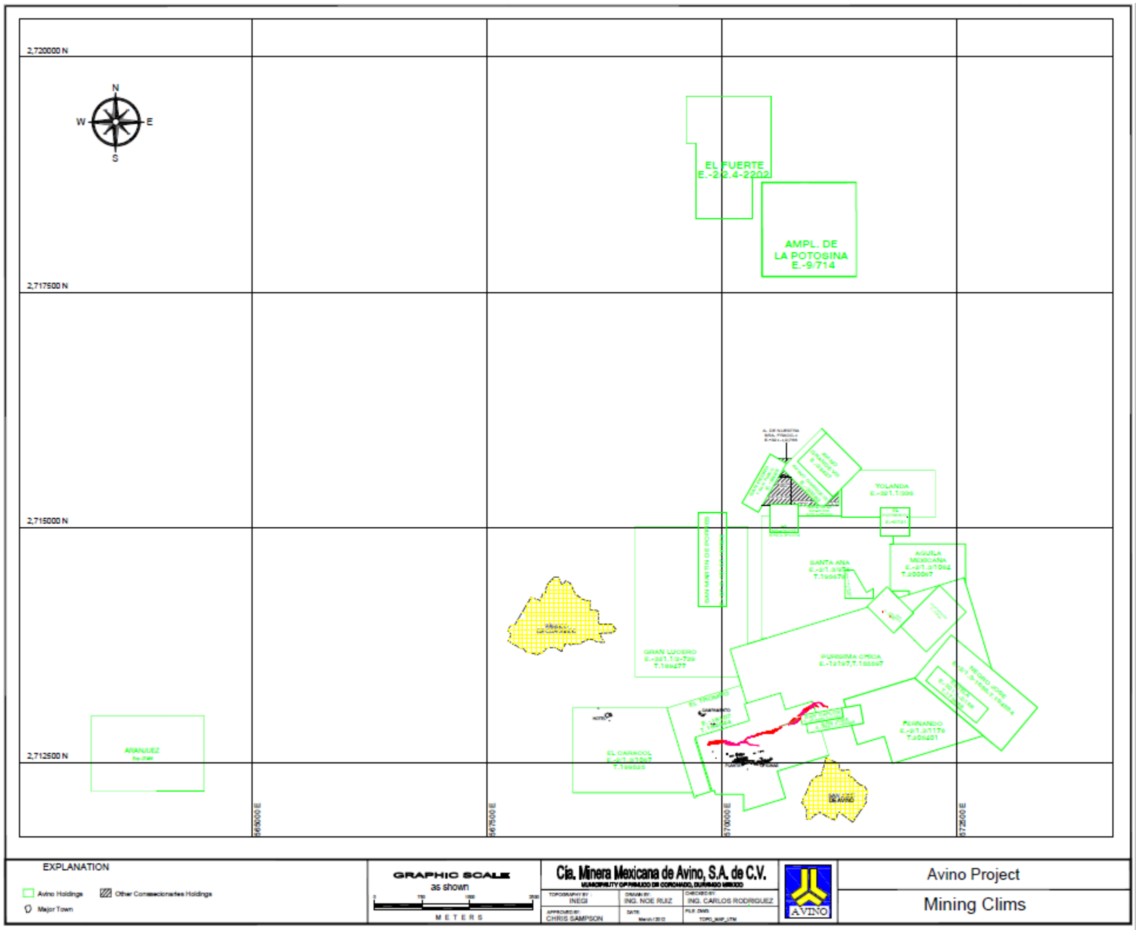


Figure 4-3: Concession Map of Avino Property (Avino 2023)

In May 1970, Avino Mines and Resources Ltd. signed a formal agreement with Selco Mining and Development (Selco), a division of Selection Trust Company. Due to other commitments, Selco abandoned its interest in the Project in 1973 (VSE 1979). On February 18, 2012, through its subsidiary company CMMA, Avino re-entered into an agreement (the Agreement) with Minerales, whereby Minerales indirectly granted Avino the exclusive mining and occupation rights to the La Platosa concession. The La Platosa concession covers 98.83 ha and hosts the Avino Vein and ET Zone.

Pursuant to the Agreement, Avino has the exclusive right to explore and mine the concession for an initial period of 15 years, with the option to extend the agreement for another 5 years. In consideration of the grant of these rights, Avino has paid Minerales the sum of US$250,000 by the issuance of 135,189 common shares of Avino. Avino will have a period of 24 months for the development of mining facilities.

Avino has agreed to pay Minerales a royalty equal to 3.5% of NSRs, at the commencement of commercial production from the concession. In addition, after the development period, if the minimum monthly processing rate of the mine facilities is less than 15,000 t, then Avino must pay Minerals, in any event, a minimum royalty equal to the applicable Net Smelter Return (NSR) royalty based on processing at a minimum monthly rate of 15,000 t. In the event of a force majeure, Avino shall pay the minimum royalty as follows:

- First quarter: payment of 100% of the minimum royalty

- Second quarter: payment of 75% of the minimum royalty
- Third quarter: payment of 50% of the minimum royalty
- Fourth quarter: payment of 25% of the minimum royalty
- In the case of force majeure still in place after one year of payments, payment shall recommence at a rate of 100% of the minimum royalty and shall continue being made as per the quarterly schedule

Minerales has also granted Avino the exclusive right to purchase a 100% interest in the concession at any time during the term of the Agreement (or any renewal thereof) upon payment of US$8 million within 15 days of Avino's notice of election to acquire the Property. The purchase would be completed under a separate purchase agreement for the legal transfer of the concession. This agreement replaces all other previous agreements.

During May of each year, Avino must file assessment work made on each concession for the immediately preceding calendar year. During January and July of each year, Avino must pay in advance the mining taxes, which are based on the surface of the concession and the number of years that have elapsed since it was issued.

Consistent with the mining regulations of Mexico, cadastral surveys have been carried out for all the listed mineral concessions as part of the field staking prior to recording (Slim 2005d). It is believed that all concessions are current and up-to-date. Mineral concessions in Mexico do not include surface rights. Avino has entered into agreements with communal landowners (Ejidos) of San José de Avino for the temporary occupation and surface rights of the concessions.

4.2 La Preciosa Area

4.2.1 Mineral Tenure

The La Preciosa part of the property consists of 15 Mining Concessions that amount to 6,641.5809 ha. Proyectos Mineros La Preciosa, S.A. de C.V. (PMLP) holds 100% of the registered, legal, and beneficial interest in and to these Mining Concessions. PMLP also holds 100% of the registered, legal, and beneficial interest in and to three (3) additional Mining Concessions that are adjacent to, and contiguous with, the Project. These three additional Mining Concessions encompass approximately 31,300 ha. Combined, the eleven (11) Mining Concessions of PMLP comprise approximately 32,400 ha and, together with approximately 1,800 ha of surface estates controlled, represent the Project consolidated property package (Property Package).

By a share purchase agreement dated October 27, 2021, with Coeur Mining, Inc. and its affiliates, Avino has agreed to indirectly acquire PMLP and the La Preciosa property. The closing of the proposed acquisition is subject to significant conditions, including that there shall not have occurred any event, change, or circumstance which has had or would reasonably be expected to have a material adverse effect, the authorization of the Mexican Federal Economic Competition Commission, approval of the transaction by the NYSE American, any other necessary third-party approvals, and the completion of all other covenants and conditions required to be performed by the parties prior to closing. There can be no assurance that the proposed transaction will be completed as proposed or at all. This report has been prepared for Avino on the basis that the proposed acquisition will be completed.

Table 4-3 provides information about the Mining Concessions in the La Preciosa area. Figure 4-4 depicts the location of the Mining Concessions and surface estates controlled.

Table 4-3: Mineral Concessions - La Preciosa Area (Avino 2023)

S. No.	Concession Name	Concession No.	Area (Ha)	Expiration date
1	EL CHOQUE CUATRO	220251	629.7778	July 1, 2053
2	EL CHOQUE SEIS	220583	249.0000	September 1, 2053
3	EL CHOQUE TRES	218953	10.0000	January 28, 2053
4	FRACCION LA PRECIOSA	185128	2.5249	July 14, 2038
5	LA B	214232	28.2006	September 5, 2051
6	LA PRECIOSA	182517	143.6119	July 14, 2038
7	LUPITA	182584	27.1878	August 11, 2038
8	SAN PATRICIO	189616	29.4740	December 4, 2040
9	SANTA MONICA SUR	223097	900.0000	October 15, 2054
10	EL NIÑO	236219	10.0000	May 24, 2060
11	LA PEÑA	204828	57.3190	May 12, 2047
12	CENTINELA	244180	0.1048	June 29, 2065
13	DON MIGUEL HIDALGO Y COSTILLA	244480	0.2168	October 5, 2065
14	HURACAN 4 R1A	246910	1,768.4591	January 19, 2054
15	TIFON 3 R1A	246466	2,785.7042	February 16, 2056
--	**TOTAL**	**--**	**6,641.5809**	**--**

Notes: Figures may not add to totals shown due to rounding.

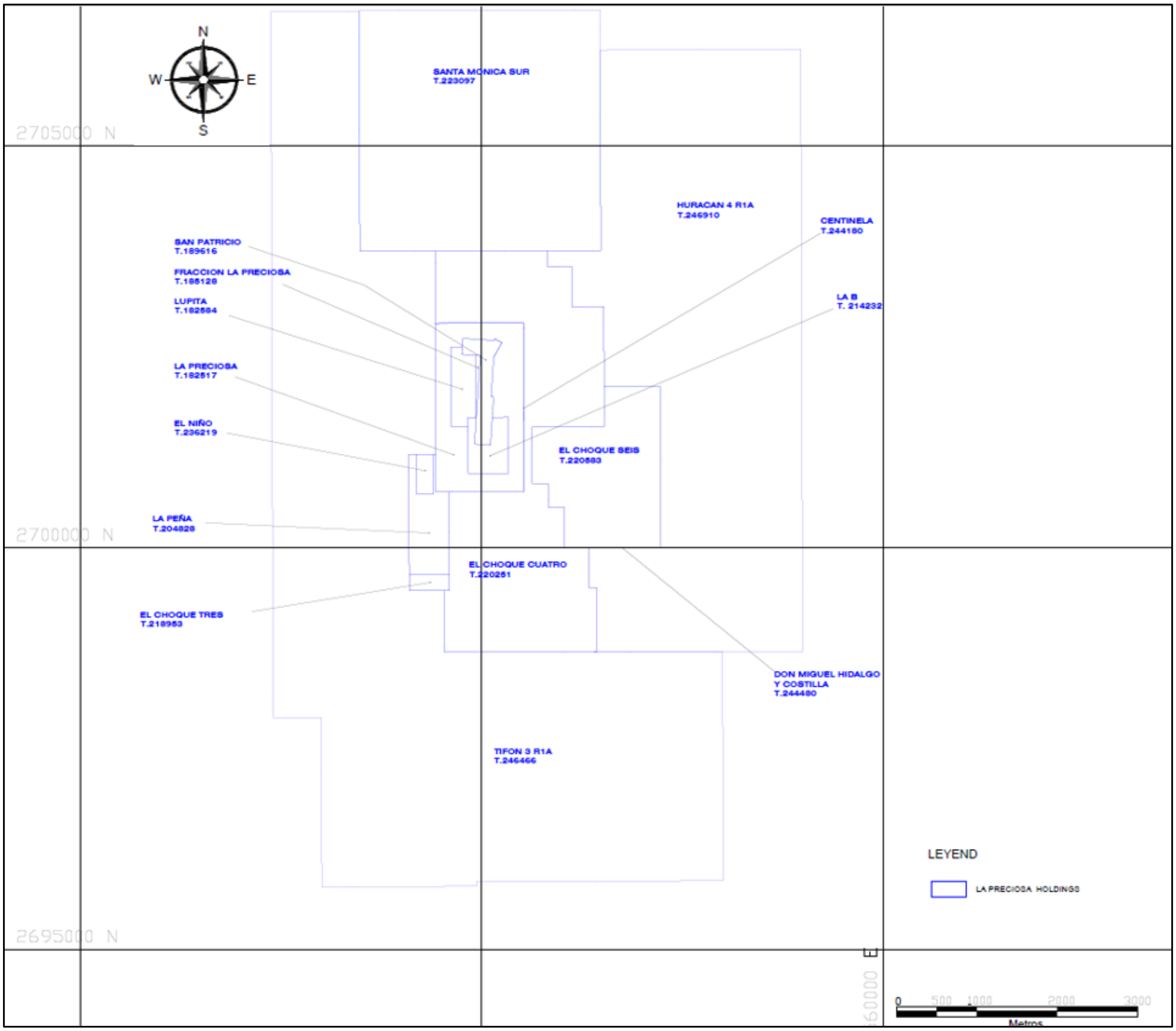


Figure 4-4: Concession Map of La Preciosa Area (Avino 2023)

The Dirección General de Minas (General Bureau of Mines) administers Mining Concessions in Mexico. A legal survey (Trabajos Periciales) of each Mining Concession was completed as a requirement of, and condition precedent to, the General Bureau of Mines granting such Concession.

Pursuant to an amendment of the Mexican Mining Law (Law), by Congressional Decree of February 22, 2005, published in the Diaro Oficial de la Federación April 28, 2005, there is no longer any distinction between an Exploration Concession and an Exploitation Concession. Consequently, all Concessions are "Mining Concessions" (Exploration and Exploitation), and as a result, all Exploration and Exploitation Concessions have been converted into Mining Concessions, expiring 50 years from the date they were originally granted.

Payment of Mining Duties are required for each Mining Concession and, each year, are payable semiannually in January and July to the Secretaría de Economía (Secretariat of Economy). The Mining Duties are calculated by determining the correct Cuota[1] (Fee), which varies based upon the age of the Mining Concession. The Fee is then multiplied by the number of hectares encompassed by the Mining Concession, the product of which equals the semiannual Mining Duty payable for the respective Mining Concession. A copy of the payment receipt of the Mining Duty must be filed with the Dirección General de Regulación Minera, a sub-directorate of the General Bureau of Mines, semiannually each February and August.

Informes Estadístico Sobre La Producción, Beneficio y Destino de Minerales o Sustancias Concesibles (Production Reports), detailing the production, beneficiation, and destination of concessionable minerals, must be submitted annually by January 30. These reports must be submitted for each Mining Concession bearing production and all Mining Concessions with over six (6) years of age, whether bearing production or not.

The surface estates overlying the Project are owned by a mixture of ejidos[2] and private parties.

4.2.2 Issuer's Interest

On February 20, 2013, Coeur announced it was entering into a definitive agreement pursuant to which Coeur would agree to acquire all of the issued and outstanding common shares of Orko, the parent company of PMLP, in a transaction with a total value of approximately CAD$350 million.

On April 16, 2013, Coeur announced the completion of its acquisition of Orko pursuant to its previously announced plan of arrangement, detailed in the news release on February 20, 2013. As a result of the completion of the arrangement, Coeur now owns all of the issued and outstanding shares of Orko.

4.2.3 Royalties, Back-in Rights, Payments, Agreements, and Encumbrances

- A Consulting and Finder's Fee Agreement dated May 1, 2002, as amended (Agreement) by and between Silver Standard Resources, Inc., the predecessor in interest of and to PMLP and La Cuesta International, Inc. (LCI). In accordance with the terms thereof, PMLP pays a Finder's Fee to LCI, comprised of: (I.) an advance royalty, every six (6) months, equal to the greater of (i.) USD$5,000 or (ii.) 2% of direct exploration costs in and to the Mining Concession San Juan (Título #226663) and (II.) a one-quarter of one percent (0.25%) NSR royalty on production derived from the Mining Concession San Juan (Título #226663), which is located adjacent to the Project, if any. The maximum amount payable under the terms of the Agreement is USD$2,000,000. PMLP has the right, at any time, to acquire from LCI all of the NSR payable in respect to LCI, including any amount remaining payable under the NSR. LCI shall not sell, transfer, or otherwise assign all or any portion of its interest in the NSR (NSR Interest) to any other party without first offering the NSR Interest to PMLP. Reciprocally, PMLP shall not sell, transfer, or otherwise assign all or any portion of its interest in and to the Mining Concession to any other party without first offering the Mining Concession to LCI;
- A Net Smelter Return Royalty Agreement dated June 19, 2002 (Sanluis Agreement #1) by and among Minas Luismin S.A. de C.V., Minas Sanluis, S.A. de C.V., collectively, the owner and the predecessor in interest of

1 The Base Rate or "Fee" is adjusted annually and published the Diario Oficial De La Federación each December for use in calculating Mining Duties payable due the following year.

2 An ejido is one of two types of social property in Mexico, granted by the government that combines communal ownership with individual use. The ejido consists of common use land, community development land, and individual parcels, which may be assigned to ejido members.

and to PMLP and Corporación Turística Sanluis, S.A. de C.V., the holder and predecessor in interest of and to SANLUIS Corporación, S.A.B. de C.V. In accordance with the terms thereof, the Owner conveyed a three percent (3%) NSR royalty (Sanluis Royalty #1) on production to the holder, derived from Mining Concessions El Choque Tres (Título #218953) and La B (Título #214232), if any. Sanluis Royalty #1 is a covenant that runs with, and binds, these two (2) Mining Concessions and the legal title thereto, the owners thereof, and their successors and/or assigns. The Sanluis Agreement #1 provides that owner has a right of first refusal to acquire the Sanluis Royalty #1 if the holder receives a bona fide proposal to acquire the Sanluis Royalty #1 from a third party;

- On June 12, 2014, SANLUIS Corporación, S.A.B. de C.V. extended to Coeur its right of first refusal pursuant to the terms, covenants, and obligations of the Sanluis Agreement #1. Coeur exercised its right of first refusal and on July 2, 2014, repurchased the Sanluis Royalty #1 encumbering the two (2) Mining Concessions El Choque Tres (Título #21895) and La B (Título #214232) for USD$12,000,000.00. The repurchase price also reflects the concurrent extension and exercise of the right of first refusal of the Sanluis Royalty #2 described immediately hereinbelow.

- A Net Smelter Return Royalty Agreement dated June 19, 2002, (Sanluis Agreement #2) by and among Minas Luismin S.A. de C.V., Minera Thesalia, S.A. de C.V., collectively, the owner and the predecessor in interest of and to PMLP and Corporación Turística Sanluis, S.A. de C.V., the holder and predecessor in interest of and to SANLUIS Corporación, S.A.B. de C.V. In accordance with the terms thereof, the Owner conveyed a three percent (3%) NSR royalty (Sanluis Royalty #2) on production to the holder, derived from Mining Concessions La Preciosa (Título #182517), Lupita (Título #182584), Fracción La Preciosa (Título #185128), and San Patricio (Título #189616), if any. Sanluis Royalty #1 is a covenant that runs with, and binds, these four (4) Mining Concessions and the title thereto, the owners thereof, and their successors and/or assigns. The Sanluis Agreement #2 provides that owner has a right of first refusal to acquire the Sanluis Royalty #2 if the holder receives a bona fide proposal to acquire the Sanluis Royalty #2 from a third party;

- On June 12, 2014, SANLUIS Corporación, S.A.B. de C.V. extended to Coeur its right of first refusal pursuant to the terms, covenants, and obligations of the Sanluis Agreement #2. Coeur exercised its right of first refusal and on July 2, 2014, repurchased the Sanluis Royalty #2 encumbering the four (4) Mining Concessions La Preciosa (Título #182517), Lupita (Título #182584), Fracción La Preciosa (Título #185128), and San Patricio (Título #189616) for USD$12,000,000.00. The repurchase price also reflects the concurrent extension and exercise of the right of first refusal of the Sanluis Royalty #2 described immediately hereinabove.

- On June 12, 2013, PMLP executed a Contrato de Ocupación Temporal Para La Extracción, Explotación, Uso y Aprovechamiento de Las Fuentes de Agua del Subsuelo (Temporary Occupancy Agreement) with Fernando Rivas Cossío, an ejidatario of the ejido Vicente Suarez (Posesionario). This Temporary Occupancy Agreement covers approx. five and nine-tenths (5.9) hectares, and has a term of thirty (30) years from June 12, 2013. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Posesionario is USD$75,000. PMLP has prepaid the annual rent through June 12, 2018;

- On June 12, 2013, PMLP executed a Temporary Occupancy Agreement with Fernando Rivas Cossío, an ejidatario of the ejido Vicente Suarez (Posesionario). This Temporary Occupancy Agreement covers approx. eight and four-tenths (8.4) hectares, and has a term of thirty (30) years from June 12, 2013. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Posesionario is USD$75,000. PMLP has prepaid the annual rent through June 12, 2018;

- On July 23, 2013, PMLP executed a Temporary Occupancy Agreement with Alejandro Hernández Jarquín (Propietario). This Temporary Occupancy Agreement covers approx. one thousand two hundred eighteen and five-tenths (1,218.5) hectares, has a term of twenty-five (25) years from July 23, 2013, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, a lump sum payment was tendered upon the execution thereof, equaling MXN$16,463,966.40, and represents the sum total consideration due from the Compañía thereunder. The land encumbered by this Temporary Occupancy Agreement overlies a portion of the Mining Concession El Choque Tres (Título #218953), El Choque Cuatro (Título #220251), La B (Título #214232), La Preciosa (Título #182517), and San Juan (Título #226663);

- On February 7, 2014, PMLP executed a Temporary Occupancy Agreement with Unidád Comercial Agricola y Ganadera Don Joaquin S. de R.L. de C.V. (Propietario). This Temporary Occupancy Agreement covers approx. twenty (20) hectares, has a term of twenty-five (25) years from February 7, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$18,000. PMLP has prepaid the annual rent through February 9, 2019. On February 10, 2019, PMLP must prepay the ensuing five (5) year's annual rent, through February 9, 2024. Thereafter, there is no longer an obligation to prepay the annual rent. The land encumbered by this Temporary Occupancy Agreement overlies a portion of the Mining Concession San Juan (Título #226663);

- On February 7, 2014, PMLP executed a Temporary Occupancy Agreement with Jorge Soto Enríquez (Propietario). This Temporary Occupancy Agreement covers approx. six (6) hectares, has a term of twenty-five (25) years from February 7, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement the annual rent payable to the Propietario is USD$5,400. PMLP has prepaid the annual rent through February 9, 2019. On February 10, 2019, PMLP must prepay the ensuing five (5) year's annual rent, through February 9, 2024. Thereafter, there is no longer an obligation to prepay the annual rent. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), San Juan (Título #226663), and Santa Monica (Título #221288);

- On February 7, 2014, PMLP executed a Temporary Occupancy Agreement with Jorge Soto Enríquez (Propietario). This Temporary Occupancy Agreement covers approx. ninety-four (94) hectares, has a term of twenty-five (25) years from February 7, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$84,600. PMLP has prepaid the annual rent through February 9, 2019. On February 10, 2019, PMLP must prepay the ensuing five (5) year's annual rent, through February 9, 2024. Thereafter, there is no longer an obligation to prepay the annual rent. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), San Juan (Título #226663), and Santa Monica (Título #221288);

- On February 13, 2014, PMLP executed a Temporary Occupancy Agreement with Petra Higareda Briceño Viuda de García (Propietario). This Temporary Occupancy Agreement covers approx. fifty-four and nine-tenths (54.9) hectares, has a term of twenty-five (25) years from February 13, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$49,422.93. PMLP has prepaid the annual rent through February 13, 2019. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251) and La Preciosa (Título #182517);

- On February 18, 2014, PMLP executed a Temporary Occupancy Agreement with ejido Lázaro Cárdenas (Ejido). This Temporary Occupancy Agreement covers approx. one hundred fifty-seven and two-tenths (157.2) hectares, has a term of thirty (30) years from February 18, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Ejido is MXN$785,944.95. The annual rent shall be adjusted annually in accordance with the changes to the Mexican CPI. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), El Choque Seis (Título #220583), and Santa Monica (Título #221288);

- On February 19, 2014, PMLP executed a Temporary Occupancy Agreement with ejido Francisco Javier Mina (Ejido). This Temporary Occupancy Agreement covers approx. eighty-nine and two-tenths (89.2) hectares, has a term of thirty (30) years from February 19, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Ejido is MXN$445,846.69. The annual rent shall be adjusted annually in accordance with the changes to the Mexican CPI. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), El Choque Seis (Título #220583), and Santa Monica (Título #221288);
- On April 11, 2014 PMLP executed a Temporary Occupancy Agreement with Candelaria Uves Solórzano (Propietario). This Temporary Occupancy Agreement covers approx. two hundred eighteen and eight-tenths (218.8) hectares, has a term of twenty-five (25) years from April 11, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$196,911.89. PMLP has prepaid the annual rent through March 28, 2019. However, beginning March 28, 2015, and continuing until March 28, 2019, PMLP must prepay, in each of those years, for the future lease periods from March 29, 2019, through March 28, 2024. On March 28, 2024, PMLP must prepay the annual rent for the ensuing five (5) years or until March 28, 2029. On March 28, 2029, PMLP must prepay the annual rent for the ensuing five (5) years or until March 28, 2034. The last annual rent payment under the terms of the Temporary Occupancy Agreement is scheduled to be made March 28, 2034, a prepayment of the annual rent for the last year of the term of the Temporary Occupancy Agreement, 2039. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), El Choque Seis (Título #220583), Fracción La Preciosa (Título #185128), La B (Título #214232), La Preciosa (Título #182517), San Patricio (Título #189616), and Santa Monica (Título #221288).

There are no other known royalties, back-in rights, payments, agreements, or encumbrances.

4.2.4 Environmental Liabilities and Permits

Please refer to Section 20 for a discussion regarding environmental and permitting factors related to the Property.

4.2.5 Significant Factors and Risks

Pursuit of the purchase or control of the necessary and convenient surface estates that overlie the La Preciosa part of the property is ongoing. There are risks that some of these surface estates, or portions thereof, may not be acquired due to unrealistic expectations of the parties, uncured or incurable defects in the legal land title, and/or survey and legal description inaccuracies.

The accuracy and completeness of ownership records maintained by the several Registros Públicos de la Propiedad y del Commercio (RPPyC) and Direcciones de Catastro within the state of Durango varies greatly. Prior to commencing negotiations for the purchase or control of a surface estate, legal land titles are thoroughly abstracted to determine legal ownership and the defects affecting validity of said ownership. Many Certificados and Constancias, issued by the several RPPyC, Direcciones de Recaudación, and Registros Agrario Nacional, are requested and obtained, in order to cross reference our own research with that of these government entities. Any disparities between the two are flagged for curing or ameliorating the title risk(s).

Well before consummating the purchase or leasehold transaction, each surface estate parcel is surveyed in the field using high-precision equipment manufactured by Trimble Navigation, LTD. Any discrepancies between the survey results, legal descriptions within the chain of title, and/or previous surveys are analyzed and curative actions are taken to formally reconcile and/or correct the legal dimensions of said surface estate. Many of the surface estates overlying the Project have been secured by long-term leasehold agreements.

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

5.1 Avino Property

5.1.1 Topography, Elevation, and Vegetation

The Property lies on the western edge of the high plains of northern Mexico, an extensive volcanic plateau characterized by narrow, northwest-trending ranges separated by wide, flat-floored filled basins. In the Durango area, the basins have elevations of between 1,900 to 2,100 metres above sea level (masl) and the higher peaks rise to 3,000 m. The Property elevation in the area of the mineralized zones at the Property is between 1,990 and 2,265 m. The highest elevations on the Property are at the northwest trending La Preciosa Ridge, which overlies the La Gloria and Abundancia veins. A broad valley forms to the east of the ridge and extends approximately 1 km toward another lower lying ridge to the northeast. Grasses, small shrubs, and cactuses comprise the typical vegetation on the steep hillsides with larger bushes and mesquite trees in the lower lying areas near springs and streams. Nearby farmers produce beans and maize with groundwater sourced from thick gravel beds in the surrounding plains or via dry farming during the rainy season. Local cattle graze on land dominated by rocky soils supporting nopal (prickly pear) and huizache (acacia) scrubland.

5.1.2 Accessibility and Local Resources

The Property is easily accessible by road, and the mine is an important employer of the local community from which skilled workers are available. Access is provided by Highway 40, a four-lane highway leading from Durango, past the airport and onto the city of Torreon in Coahuila. Successive turn-offs for the Property are at Francisco I Madero, Ignacio Zaragoza, and San José de Avino (Slim 2005d). The Avino mineral concessions are covered by a network of dirt roads, which provide easy transport access between all areas of interest on the Property and the mill at the Avino Property (Gunning 2009).

The nearest major city is Durango, with a population of 616,068. Durango is a major mining centre in Mexico where experienced labour and services can be obtained. The two towns nearest the mine are Pánuco de Coronado and San José de Avino, where the majority of the employees lived while working at the mine when it was in operation. Pánuco de Coronado has a population of 12,656, and San José de Avino is a small centre with a population of 868.

5.1.3 Climate and Length of Operating Season

The climate is temperate and semi-arid. In the region, the mean annual rainfall is 522 mm, and the average annual temperature is 16.7°C. The warmest month of the year is June, with an average temperature of 21.8°C, and January the coldest month of the year with an average temperature of 11.1°C. The driest month is April, with 6 mm of rainfall. In September, the precipitation reaches its peak, with an average of 123 mm (https://en.climate-data.org/north-america/mexico/durango/durango-3559/). In the winter, the temperature can drop below freezing, and frost and even light snowfall can occur.

Exploration, development, and mining activities may take place throughout the year without any significant seasonal impact.

5.1.4 Infrastructure

Infrastructure is disclosed in Section 18.

5.2 La Preciosa Area

Figure 5-1 shows the La Preciosa regional location.

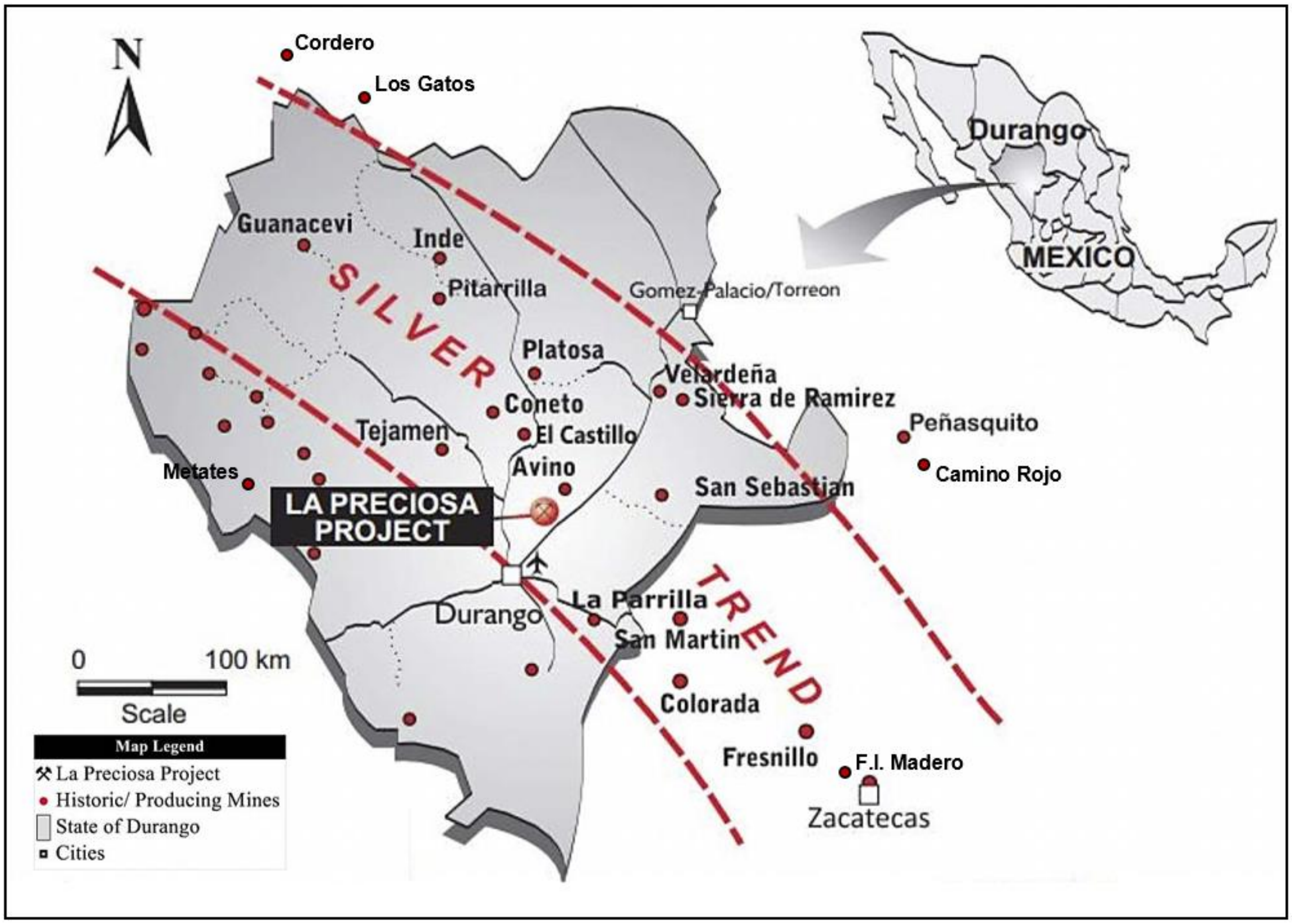


Figure 5-1: La Preciosa Project Location Map (Coeur 2013)

5.2.1 Length of Operating Season

Mining activities can take place year-round.

5.2.2 Surface Rights, Land Availability, and Mining Areas

Surface rights in the La Preciosa area are held by a combination of private landowners, ejidos, and ejidatarios, which are ejido members with rights to use specific tracts of land within the ejido. Ejidos are areas of communal land used for agriculture where community members jointly control rights to access and use the land. Ejidos are registered with Mexico's National Agrarian Registry.

5.2.3 Ownership

The issuer has verified who the registered landowners are who hold surface rights within the boundaries of the proposed mining activities.

6.0 HISTORY

6.1 Avino Mine Area

6.1.1 Avino Mine, 1555 to 1968

The Avino deposit was originally discovered around 1555 by the Spanish conquistador Don Francisco de Ibarra. In 1562, Francisco de Ibarra was appointed governor of the newly formed province of Nueva Vizcaya in the Viceroyalty of Nueva España (New Spain) and, in 1563, founded the town of Durango. Francisco de Ibarra led several expeditions in search of silver deposits in the region and is recognized as having established Minas de Avino, present-day Avino Mine; San Martín, Durango; and Pánuco, Sinaloa. Mining operations at the Avino Mine are said to have commenced in 1562–1563 and have been in production until the early 1900s. Operations at the Avino Mine continued up to the onset of the War of Independence (1810), when operations were interrupted but then restarted and continued through to the early 1900s.

In 1880, the mines were taken over by Avino Mines Ltd., a company controlled by American and British interests. The introduction of more modern industrial technology helped the Avino Mine develop into a significant mining operation at the beginning of the 20th century. By 1908, the Avino Mine was considered one of the largest open pit mines in the world and equipped with one of the largest lixification smelters (Gallegos 1960; VSE 1979; Slim 2005d).

During the early phases of the Mexican Revolution in 1910, proceeds from the mine supplied funds to the revolutionary forces. Since much of the fighting occurred in and around Durango, and the risk posed by brigands hiding in the mountains was high, the mine was abandoned in 1912.

Between 1912 and 1968, the mine was worked intermittently on a small scale (Avino Annual Report 1980). There is no documentary record of production from the Avino Mine during this period.

The Property was acquired under current ownership in 1968.

6.1.2 Avino Vein System Deposit

The Avino Vein system was the mainstay of historical exploitation and is situated adjacent to the mine offices and processing plant. The upper portion of the deposit was extensively mined in an open pit, and the lower portion is currently accessible via a ramp and has been extensively developed and mined from more than 6 km of horizontal drifts, with vertical spacings between 15 m and 25 m. The ET Mine workings extend to a maximum depth of 360 m vertically below the portal of the ramp. An old vertical shaft, no longer used for hoisting, is used for ventilation and to supply water and power for development and mining. A vertical section of Avino Mine is shown in Figure 6-1. The western portion of the Avino Vein system is referred to as the San Luis. In 2016–2017, the focus of exploration drilling was on the region between the ET Mine and San Luis. The eastern extension of the Avino Vein system is known as Chirumbo.

Figure 6-1: ET Mine: Vertical Section View Showing Development and Stoping (Red Pennant 2023)

6.1.3 San Gonzalo Vein Deposit

Shallow workings from an old mine are present in the San Gonzalo Vein and consist of small underground workings which were originally accessed by a five-level vertical shaft.

Current access to the San Gonzalo deposit (SG Mine) is via a ramp that is being actively developed. All old working levels have been dewatered. The deposit has been explored and exploited by more than 4 km of horizontal drifts with upper levels at 40 m vertical spacing and lower levels at 25 m vertical spacing. A vertical section of the San Gonzalo Mine is shown in Figure 6-2.

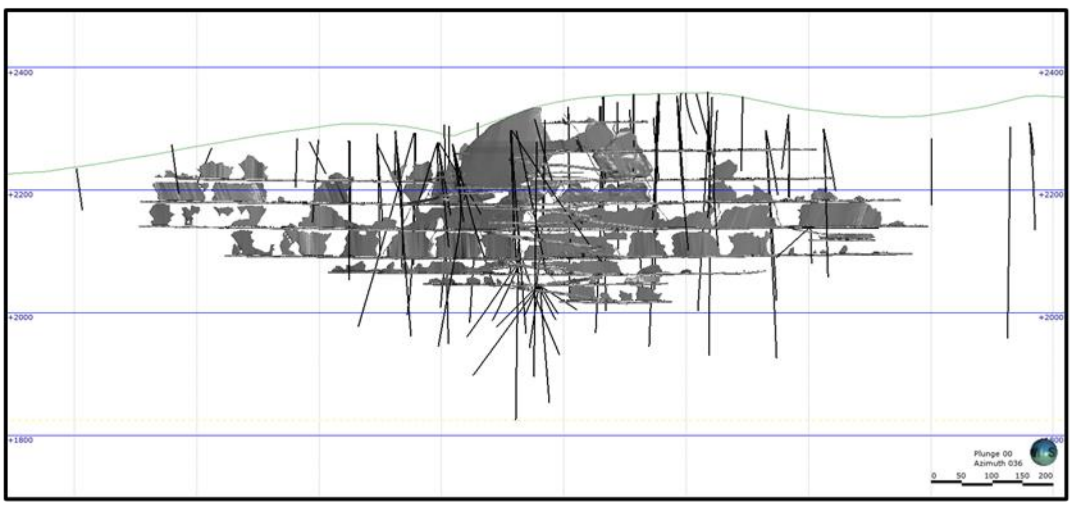


Figure 6-2: San Gonzalo Mine: Vertical Section View Showing Development and Stoping (Red Pennant 2022)

6.1.4 Guadalupe Vein Deposit

The Guadalupe Vein (see Figure 7-2) extends between the Avino Vein and the San Gonzalo Vein and is a current exploration target.

6.1.5 San Juventino Vein Deposit

The San Juventino Vein (see Figure 7-2) is adjacent to the eastern end of the Avino Vein and is a current exploration target.

6.2 La Preciosa Area

In the late 19th century, the La Preciosa part of the Property was known as Mina La Preciosa. Early work was focused on the north end of La Preciosa Ridge where the Gloria and Abundancia veins outcropped to surface. Mining ceased at the onset of the Mexican Revolution, in 1910, and further mining did not occur until the 1970s. It has been estimated by Orko personnel that a total of ~30,000 t were extracted during that time (MP 2012).

Luismin operated under the name Minera Thesalia as a joint venture with Tormex S.A. and conducted exploration on the area in 1981, 1982, 1988, and 1994. This work consisted of a surface and underground channel sampling program, a single east-west line of induced polarization (IP) resistivity across the La Preciosa area, and drilled seven diamond drill core holes totalling 1,319 m. This included five surface drillholes targeting the Gloria and Abundancia veins 50-75 m below the primary underground workings, and two holes drilled from within those older workings.

Luismin expanded the historic underground workings to a size of approximately 3 m by 3 m. This allowed for the underground drilling and also for a program of channel sampling. A reported 11,739 t of material was removed from the sides of the historic underground workings at reported grades of 0.43 g/t Au and 157 g/t Ag. That material was stockpiled outside the portal of those underground workings and is still in-place. While Luismin staff did calculate several MREs during that time, based on limited information, the channel sampling and shallow drilling were not used for the calculation of the current Mineral Resources.

Orko (subsequently Orko Silver Corp.) entered into a Joint Venture (JV) agreement with Luismin in 2003 and subsequently acquired the control of the La Preciosa area with Luismin maintaining a royalty.

Orko performed a series of exploration programs beginning in 2005 and lasting until 2008, and drilled 388 core holes for a total of 152,368 m on targets at Orito, San Juan, and La Preciosa. Additional surface sampling and mapping was also performed during that time.

Orko signed a JV agreement with Pan-American Silver (PAS) in 2009. PAS drilled 363 drillholes for a total of 91,096 m during 2009-2010. The desired result was a Measured Resource to support a feasibility study issued in 2012 by Quantitative Geoscience Pty. Ltd. and included a Technical Report by Snowden Engineering Inc. (Snowden) done in 2011 (Snowden, 2011a). PAS work included the use of some drillholes for geotechnical purposes, and four metallurgical test programs performed by SGS Mineral Services in Durango, Mexico. Problems with those metallurgical test programs were noted by Snowden and future work was recommended. However, a table of Mineral Resources was produced as a result of the MP Updated Mineral Resource Estimate (MRE) prepared for Orko in 2012 (MP 2012). A QP had not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves; and (ii) the issuer is not treating the historical estimate as current mineral resources or mineral reserves. The historical resources are presented as historical information only. It should be noted that these historical estimates were designed to support open pit/surface mining scenarios.

Table 6-1: La Preciosa Area Historical MRE (Effective Date October 25, 2012)

Mining Method	Classification	COG	Mass (Mt)	Ag (g/t)	Ag (million ounces)	Au (g/t)	Au (thousands oz.)	Ag Equivalent (g/t)	Ag Equivalent
Open Pit	**Indicated**	25	29.6	104	99	0.20	190	115	110
Open Pit	**Inferred**	25	47.7	86	132	0.16	245	95	146
Underground	**Indicated**	60	0.1	99	0	0.16	0	108	0.2
Underground	**Inferred**	60	1.9	124	8	0.21	13	136	8
Total	**Indicated**		29.7	104	99	0.20	191	115	110
	Inferred		49.6	87	140	0.16	259	97	154

Notes:

1. This statement is not current
2. Open pit resources stated are contained within a potentially economically mineable pit shell. Pit optimization is based on assumed silver and gold prices of US$25.90/oz and US$1,465/oz respectively and mill recoveries of 88% and 78% respectively, mining costs of US$1.45/t, processing costs of US$17.25/t and G&A costs of US$4.35/t. Break-even cut-off grades used were 25 g/t Ag for open pit mill material and 60 g/t Ag for underground material.
3. Silver equivalency is based on unit values calculated from the above metal prices, and assumes 100% recovery of all metals.
4. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and sums may not total due to rounding.
5. The Underground Indicated and Inferred Resources do not account for mining method and probability that mine plan design could significantly change these Resources.

In April 2013, the acquisition of Orko was completed by Coeur. Since completion of the acquisition, activities have included land and water resources acquisition plus additional efforts on geological and technical studies. All involved property owners were identified and their titles verified to be in good standing prior to acquisition of surface rights.

By a share purchase agreement dated October 27, 2021 with Coeur Mining and its affiliates, Avino agreed to indirectly acquire PMLP and the La Preciosa property. The closing of the proposed acquisition is subject to significant conditions, including that there shall not have occurred any event, change, or circumstance which has had or would reasonably be expected to have a material adverse effect, the authorization of the Mexican Federal Economic Competition Commission, approval of the transaction by the NYSE American, any other necessary third party approvals, and the completion of all other covenants and conditions required to be performed by the parties prior to closing. There can be no assurance that the proposed transaction will be completed as proposed or at all. This report has been prepared for Avino on the basis that the proposed acquisition will be completed.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Avino Mine Area

7.1.1 Regional Geology

Mineral deposits of the Sierra Madre Occidental plateau and the La Preciosa District consist mainly of silver and gold mineralization with or without significant base metal components. The known deposits form a northwest-trending belt from the state of Zacatecas and the large Fresnillo silver deposit on the southeast to the Guanaceví silver deposit near the border with the state of Chihuahua.

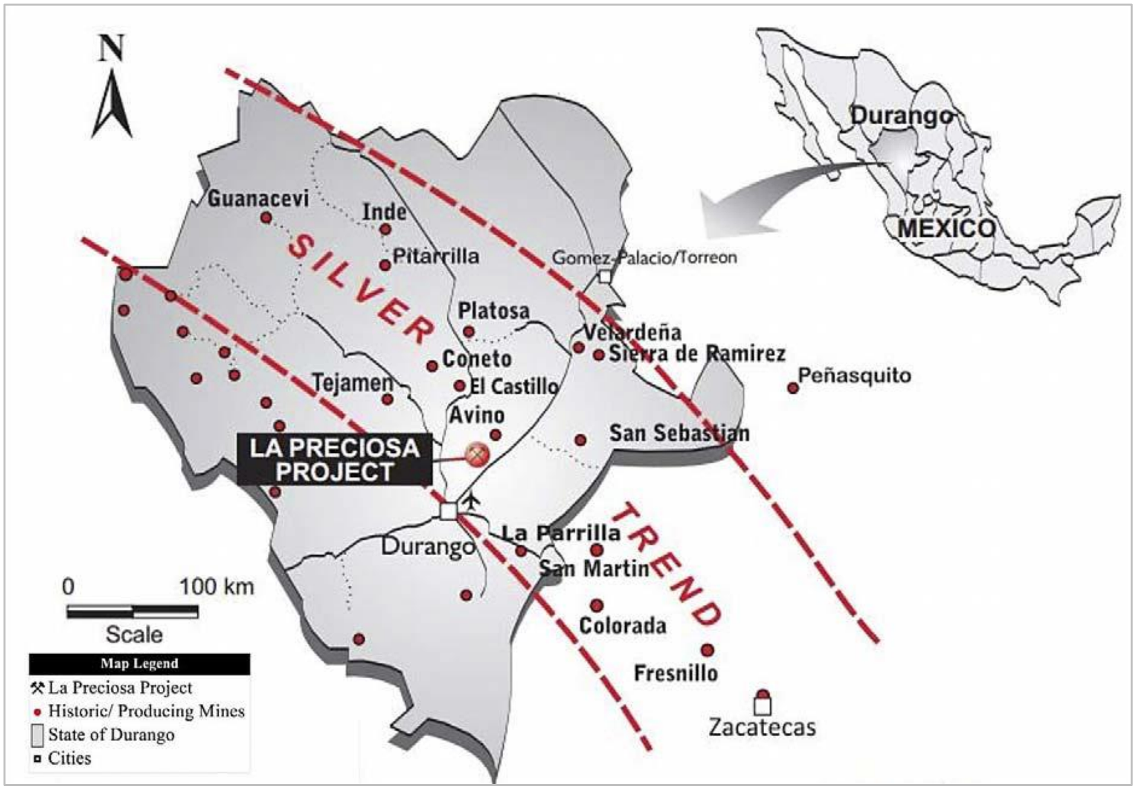


Figure 7-1: Mineral Deposits in the Project Area (M3 2013)

The historic Avino Mine and associated vein systems are located within the Sierra de Gamon, on the east flank of the Sierra Madre Occidental. The area is a geological window into the Lower Volcanic series and consists mainly of volcanic flows, sills, and tuffaceous layers of andesite, rhyolite, and trachyte. Individual rock units typically vary from 300 m to 800 m in thickness. Andesitic rocks outcrop over most of the region, with other rock types occurring more sparsely to the north (Slim 2005d).

Approximately 50 km north of the Project is the El Castillo Mine (Argonaut Gold is the operator), which is thought to be a porphyry-style gold system related to Oligocene granodiorite-diorite porphyries that intrude Cretaceous clastic and carbonate sediments in an extensional tectonic setting. Gold mineralization occurs throughout the magmatic-hydrothermal system.

San Sebastian, located 60 km to the east of the Property, contains several productive vein systems including Francine, Don Sergio, and Andrea. Production by Hecla from the Francine vein was high-grade silver, with significant gold values. Mineralization occurs in poly-phase chalcedonic quartz veins with an average width of 1.6 m. Production from the Don Sergio vein was high-grade gold, with some silver values. Several epithermal veins exist within the San Sebastian valley. The Francine, Professor, Middle, and North vein systems are hosted within a series of shale units, with interbedded fine-grained sandstones. The Don Sergio, Jessica, Andrea, and Antonella veins located in the Cerro Pedernalito area, about 6 km from the Francine vein, are hosted in the same formation, with the addition of diorite intrusion.

Directly adjacent to the Project on the west is the San Juan project of Silver Standard. Orko conducted prospecting, geological mapping, and some surface sampling. Vein targets, La Plomosa, La Plomosa Sur, El Vaquero, San Juan, Nancy Sur, and the down-dip projection of the Nancy vein are known on the San Juan property. La Plomosa vein has approximately 80 m of historical drifting and one drillhole.

Immediately south of La Plomosa and San Juan are the large Victoria and Salamandra concessions of Canasil Resources Inc. under joint venture with Blackcomb Minerals Inc. Salamandra is a skarn silver-zinc-copper prospect.

La Parrilla mine of Silver Storm Mining Ltd. is located near the Durango-Zacatecas border, approximately 65 km southeast of the city of Durango and 80 km south of the Project. La Parrilla was in production at a rate of 800 tpd. The silver-lead-zinc mineralization is hosted in vein-fault zones, breccias, and replacement bodies. These occur within the porphyritic diorite intrusive rocks and in the adjacent limestone, skarn, and hornfels rocks. While the geology is different than that at the Project, it does illustrate another example of precious metal mineral endowment in the region.

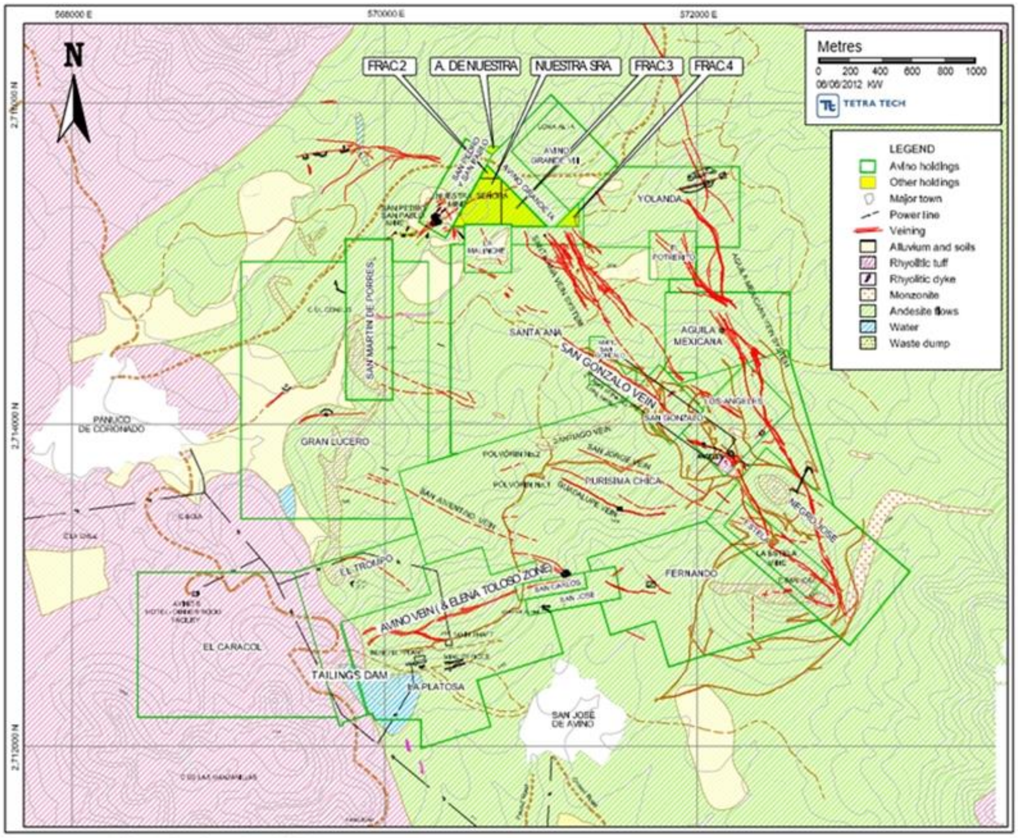
There are numerous precious metal exploration and expansion projects underway in Durango State and adjacent areas, including Metates, La Cienega, La Parrilla, Pitarrilla, Guanacevi, San Agustin, Peñasquito, Santa Cruz, San Sebastian, and Topia, as well as an expansion at the Tayoltita (San Dimas) operations. Neighboring Zacatecas state is also very active.

A large monzonitic intrusion is observed in the region in the form of dykes and small stocks, which appear to be linked to the onset of the Avino Vein mineralization. Other post-mineralization dykes of intermediate to felsic composition crop out in various areas and appear to cause minor structural displacements. Occurrences of thin mafic sills are also found in various parts of the region and are believed to be related to recent volcanism.

Higher areas of the Sierra Madre Occidental surrounding the mine are composed of rhyolites and ignimbrites of the Upper Volcanic Series, with thicknesses approaching 1,500 m.

The Laramide orogenic event is believed to have affected the Avino district. Later extrusive and intrusive igneous events appear to have caused the formation of various systems of pre-mineralization faulting. These fault systems usually produce normal displacement of the pre-existing rocks and generally strike northwest-southeast (subparallel to the Avino Vein system). Additional normal fault systems are also observed in the region, striking northeast-southwest and dipping towards the south (subparallel to the San Gonzalo Vein system).

The rugged topography is a result of erosion of the post-mineralization faulted blocks. One of the most significant regional features of the district is the Avino Fault, which strikes northwest 20° southeast, dips southeast, and appears to terminate the Avino Vein mineralization, juxtaposing the Upper and Lower Volcanic series (Figure 7-2).



Source: Modified after Gunning (2009)

Figure 7-2: General Map of Property Geology

7.1.2 Avino Mine Concessions Geology and Mineralization

The Avino concession is located within a 12 km (north-south) by 8.5 km (east-west) caldera. The area contains numerous low-sulphidation epithermal veins, breccias, stockwork, and silicified zones that grade into a "near porphyry" environment, particularly in the Avino Mine area. The caldera has been uplifted by regional north-trending block faulting (a graben structure), exposing a window of andesitic pyroclastic rocks of the lower volcanic sequence within the caldera. The Lower Volcanic Sequence is overlain by the Upper Volcanic Sequence, consisting of rhyolite to trachyte flows and extensive ignimbrites and intruded by monzonite bodies.

The basal andesite-bearing conglomerate and underlying Paleozoic basement sedimentary rocks (consisting of shales, sandstones, and conglomerates) have been identified on the Avino concession in the south-central portion of the caldera, covering the Guadalupe, Santiago, San Jorge, the San Gonzalo Trend, Malinche, Porterito, and Yolanda areas. A northerly trending felsic dyke, possibly a feeder to the upper volcanic sequence, transects the Avino area and many of the veins. The Aguila Mexicana low-temperature vein system trends north-northwest at a similar orientation to the felsic dyke and with similar continuity across the area. The two structures have been interpreted to occur along deep crustal faults that controlled volcanism and mineralization, with the felsic dyke structure controlling the emplacement of the Avino, Nuestra Senora, and El Fuerte-Potosina volcanic centres and the Aguila Mexicana structure controlling the Cerro San Jose and El Fuerte-Potosina volcanic centres (Paulter 2006).

Silver- and gold-bearing veins cross-cut the various lithologies and are generally oriented north-northwest–south-southeast and northwest–southeast (Figure 7-2). The rocks have been weathered and leached in the upper sections as a result of contact with atmospheric waters; the oxide tailings material (Section 7.1.2.5) is primarily from this source, whereas the sulphide tailings are predominantly from material sourced at depth below the leached zone. In Mexico, these types of deposits can have large lateral extents but can be limited in the vertical continuity of grades.

In the oxide zone, mineralization is primarily hosted by the minerals argentite, bromargyrite, chalcopyrite, chalcocite, galena, sphalerite, bornite, native silver, gold, and native copper. Other minerals are present in mineralized areas but not hosting the metals of interest, including hematite, chlorite, quartz, barite, pyrite, arsenopyrite, and pyrrhotite. Malachite, anglesite, and limonite are common in the quartz zones of the weathered parts of the oxide material.

7.1.2.1 Avino Vein

The geology and mineralization of the Avino Vein are summarized by Slim (2005d).

The Avino Vein (see Figure 7-3) is 1.6 km long and 60 m wide on the surface. The Avino Vein is the most striking and important example of the epithermal mineralization of the district, whose structures are normally weathered and leached in their upper section as a result of contact with atmospheric waters producing a band of oxide minerals and zones of supergene enrichment to a depth of about 70 m.

In the oxide portion of the Avino Vein, the common minerals encountered include hematite, limonite, anglesite, and copper carbonate in white or green, somewhat chloritized, quartz zones. The common primary and secondary minerals encountered are argentite, bromargyrite, chalcopyrite, chalcocite, galena sphalerite, bornite, native silver, free gold, and native copper. Other minerals present in mineralized areas include quartz, pyrite, chlorite, barite, arsenopyrite, pyrrhotite, and specularite.

Higher silver values are reported to decrease overall with depth, except at vein intersections and vein inflections, where higher values persist to depth. The same can be said for gold, although the higher values start just below the onset of silver mineralization at or near the surface. In contrast, higher copper values coincide with vein intersections and may increase with depth. Sporadic, localized copper enrichment occurs toward the footwall contact and may represent a different phase of fluid emplacement. Despite the overall decrease in precious metal grade with depth, local increases in metal grades are apparent in the mine sampling and exploration drilling, possibly reflecting changes in boiling level with pressure variations in the epithermal system.

The Avino Vein has been followed longitudinally for more than 1,300 m and vertically for more than 600 m. It strikes north at 66° east with an east-west splay and dips to the south and southeast at 60° to 70°. Steeply dipping, high-grade zones within the vein and stock-work zones are frequently found in the upper part of the vein, as well as at its intersections with a number of lateral veins. An example of a higher-grade area of mineralization encountered with a major lateral vein intersecting the Avino was the El Hundido, which exceeded 40 m in thickness. In the lower areas of the vein and mine, mineralized cross-veins, branch-veins, and stockwork zones have been found in the footwall at San Luis and El Hundido and are assumed to persist with depth.

The hanging wall of the Avino Vein is andesite, while the footwall is a monzonite intrusive with andesite sections. A post-mineralization fault parallel with the vein occurs in the hanging wall at a distance of several m in the area of San Luis; in the central part of El Hundido, this fault is located at the contact with the vein over a distance of about 300 m, up to the area of Santa Elena and San Antonio. From that point, and proceeding toward the El Chirumbo Mine, this fault cuts the vein between the face at San Carlos and the exposure at the underground ramp. The fault then enters the footwall, where it remains until a point about 30 m east of the west face of the Chirumbo area, producing a downward displacement of the vein of between 50 m to 100 m.

At Chirumbo, the fault largely replaces the vein due to strong leaching by post-mineralization circulating water in the gouge. On the east face at Chirumbo, the fault again enters the hanging wall; in this zone, the vein is composed of branches and stockwork, and to the east of this point, the fault crosses the vein numerous times.

The deposit is epithermal and made up of veins and dependent stockwork structures, mainly in the hanging wall and often associated with vein intersections. Four vein systems have been described, which, in decreasing order of importance, are:

- System striking east-west, dipping south at 60° to 70°, including the Avino Vein and its possible extension in the Cerro de San Jose.
- System striking north 60° to 70° west, dipping 60° to 80° southwest, comprising the following important veins: El Trompo, San Juventino, San Jorge, Platosa, Los Reyes, Potosina, El Fuerte, and Conejo.
- System striking north 20° to 30° west, dipping between 60° to 80° to either the southwest or northeast, comprising the following significant veins: San Gonzalo, Aguila Mexicana, and La Calcita, as well as the Stockwork La Potosina and the Stockwork El Fuerte.
- Systems striking north 60° to 80° east, dipping 60° to 80° southeast, comprising the following veins: Santiago, Retana, Nuestra Senora, and San Pedro and San Pablo.

Alteration has been reported in three main types:

- Propylitic alteration is most common in andesite, giving the andesite a greenish tint.
- Argillaceous alteration appears mainly in the upper parts of the veins and manifests itself as a whitening of the country rock due to alunite and montmorillonite clays.
- Silicification, chloritization, and pyritization alteration are observed in the hanging wall and footwall and are more prominent closer to the vein.

7.1.2.2 San Gonzalo Vein

The San Gonzalo Vein (see Figure 7-3) is located approximately 1.4 km northeast of the Avino Vein. The San Gonzalo Vein system constitutes a strongly developed vein system over 25 m wide, trending 300° to 325°/80° northeast to 77° south. It is characterized by banded textures and open-space filling. The main vein has an average width of 2 m, but the silica-pyrite or iron oxide-sericite alteration with additional stock working extends across 300 m, south of the main San Gonzalo Vein to the Los Angeles Vein.

The San Gonzalo is a typical narrow vein, precious metal deposit with some erratic values and extends approximately 2 km to the northwest to the Santa Ana-Malinche area (Gunning 2009).

The Cerro San Jose-La Estella-San Gonzalo Cerro San Jose represents a distinct hydrothermal centre with similar characteristics to the Avino system, which include the following (Paulter 2006):

- Occur on a topographic high
- Strong to intense silicification and brecciation
- Easterly trending stockwork system similar to the trend of the Avino Vein

- Similar temperatures of formation to Avino
- Presence of an intersecting northwesterly trending vein system (la Estella at San Jose and San Juventino at Avino)
- Emplacement along a northerly trending, deep crustal fault zone (defined by the Aguila Mexicana Vein at Cerro San Jose and the felsic dyke at Avino)

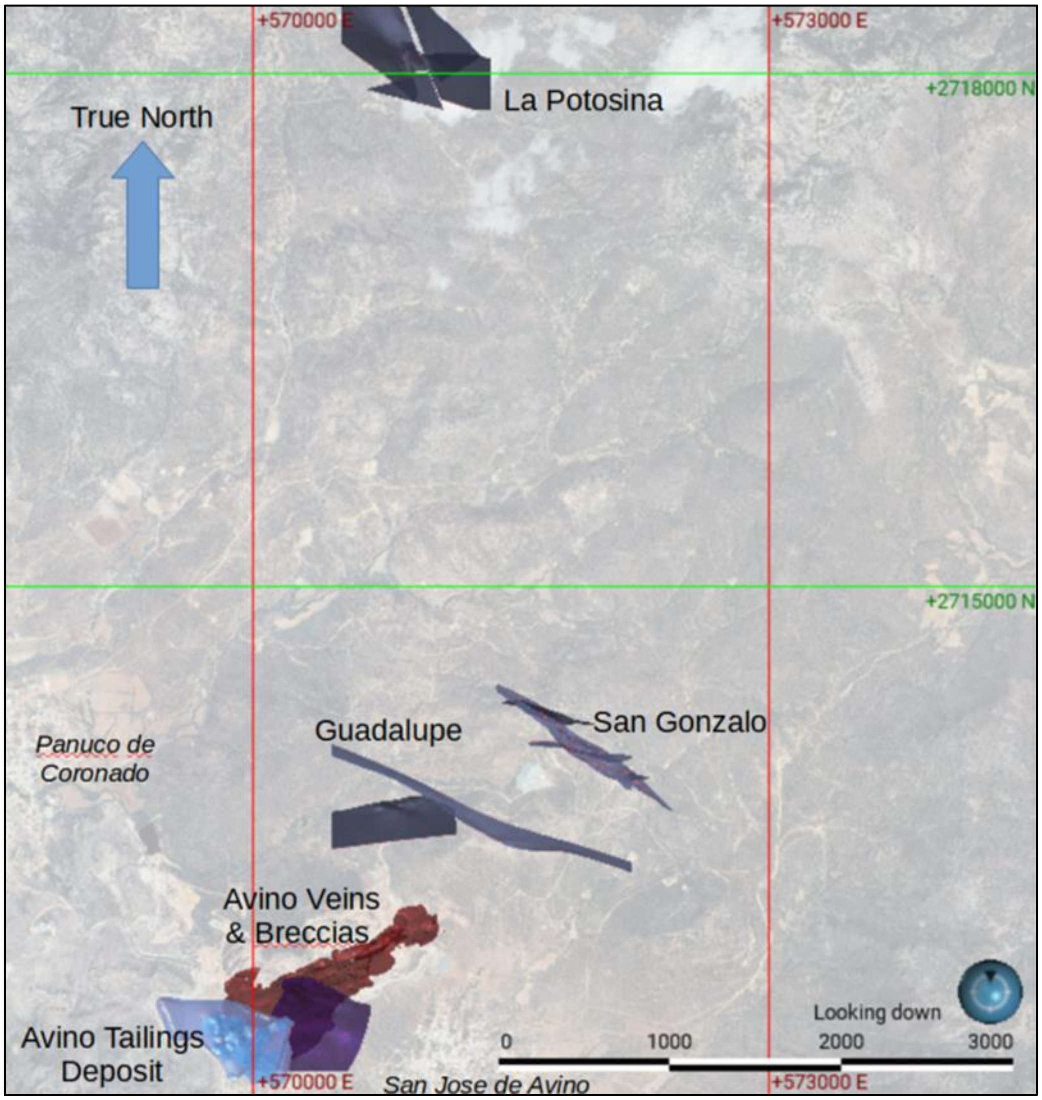


Figure 7-3: Plan View of the Avino Deposits (Red Pennant 2023)

7.1.2.3 Guadalupe Vein

The Guadalupe Vein (see Figure 7-3) is located approximately 0.7 km northeast of the Avino Vein. It consists of northwest-southeast and east-west striking steep-dipping vein sets. The geometry is similar to the San Gonzalo vein, but the base metal mineralization more closely resembles the Avino hanging wall breccia.

7.1.2.4 La Potosina Veins

The La Potosina Veins (see Figure 7-3) are located close to the northern margin of the caldera, approximately 7 km north of the Avino mine and processing plant. It consists of complementary northwest-southeast striking steep-dipping vein sets. The geometry is complex, with at least two ages of fault displacement.

7.1.2.5 Oxide and Sulphide Tailings

The Avino tailings deposit (see Figure 7-3) is adjacent to the processing plant, approximately 300 m west-southwest of the mine offices. The tailings have been built up over several decades of mining and processing, and several units have been defined based on the oxidation of the tailings and metal content.

Due to the historical processing sequence, the oxide tailings are primarily derived from weathered and oxidized rocks close to the surface on the Property, whereas the sulphide tailings are predominantly derived from material sourced at depth from the underground workings below the weathered/leached zone.

The tailings have been included in the current Mineral Resource.

7.2 La Preciosa Area

7.2.1 Regional Geology

The La Preciosa concessions is situated on the eastern flank of the Cretaceous to mid-Tertiary Sierra Madre Occidental (Figure 7-4). The SMO is the largest silicic igneous province in North America, and it stretches from the USA-Mexico border to the latitude of Guadalajara, where the SMO is covered by the late Miocene to Quaternary Trans-Mexican Volcanic Belt.

The SMO is part of the Basin and Range physiographic province where magmatism and tectonism were related to the subduction of the Farallon Plate beneath North America. Physiographically, the core of the SMO forms the boundary between the Mexican Basin and Range Province to the east and the Gulf Extensional Province to the west.

Figure 7-4 shows a simplified geological map of Northern Mexico showing the main assemblages of the Sierra Madre Occidental (from Ferrari et al. 2007). The Lower Volcanic Complex (LVC) is shown in blue, and the Upper Volcanic Supergroup (UVS) is shown in pink and orange.

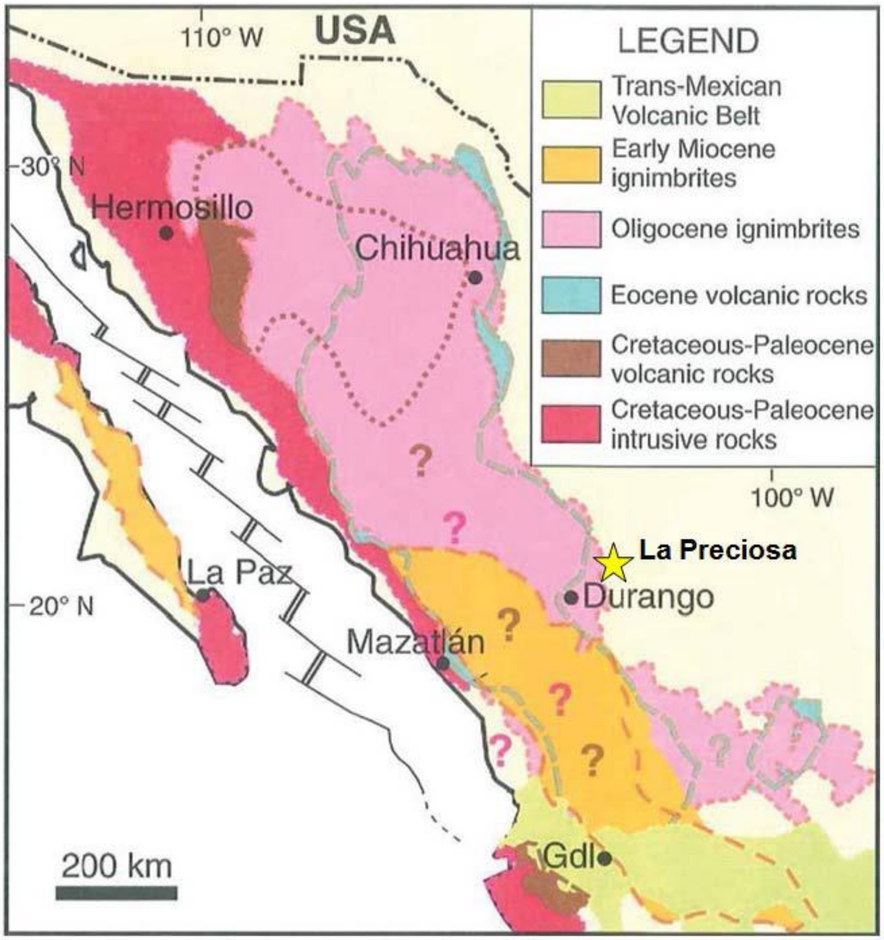


Figure 7-4: Simplified Geological Map of Northern Mexico (Ferrari et al. 2007)

The stratigraphy of the SMO comprises the following main sequences:

- Late Cretaceous to Paleocene plutonic rocks.
- Paleocene-Eocene (ca. 67-55 Ma) andesites and lesser rhyolites, traditionally grouped into the LVC (McDowell and Keizer 1977).
- Silicic ignimbrites mainly deposited during two pulses, e.g., Oligocene (ca. 32-28 Ma) and Early Miocene (ca. 24-20 Ma) and grouped into the UVS (McDowell and Keizer 1977).
- Transitional basaltic-andesitic lavas that erupted toward the end of, and after, each ignimbrite pulse.
- Post-subduction volcanism consisting of alkaline basalts and ignimbrites deposited in the Late Miocene, Pliocene, and Pleistocene.

In the area, deformed metasedimentary rocks of Cretaceous age are exposed in small windows through the Tertiary volcanic rocks of the SMO. These consist of folded and foliated clastic metasedimentary rocks that are unconformably overlain by undeformed Early Tertiary conglomerate and sandstone of the Ahuichila Formation (Aguirre-Diaz and McDowell 1993), which are in turn overlain by a sequence of intermediate tuffs, flows, and agglomerate of the Paleocene-Eocene age LVC. The LVC sequence is overlain by a thick sequence of rhyolite and intermediate to felsic ignimbrite, tuff, and volcanic breccia of Oligocene-age that are exposed along cliffs to the west.

The region is transected by the regional northwest-striking San Luis-Tepehuanes fault system (Nieto-Samaniego et al. 1999), which roughly coincides with the eastern margin of the SMO. This fault system comprises a complex network of northwest- to north-striking, west-dipping fault segments that are associated with east to northeast tilting of Tertiary stratigraphy. In the Durango region, the fault system is made up of north-northwest trending normal faults and associated (half) grabens that were active during two stages of extension between ca. 32 and 24 Ma (Nieto-Samaniego et al. 1999). The basins and parts of the lower hills in the region are covered with varying thicknesses of Pliocene to Pleistocene basalt that erupted from numerous vents now marked by small volcanic cinder cones and domes.

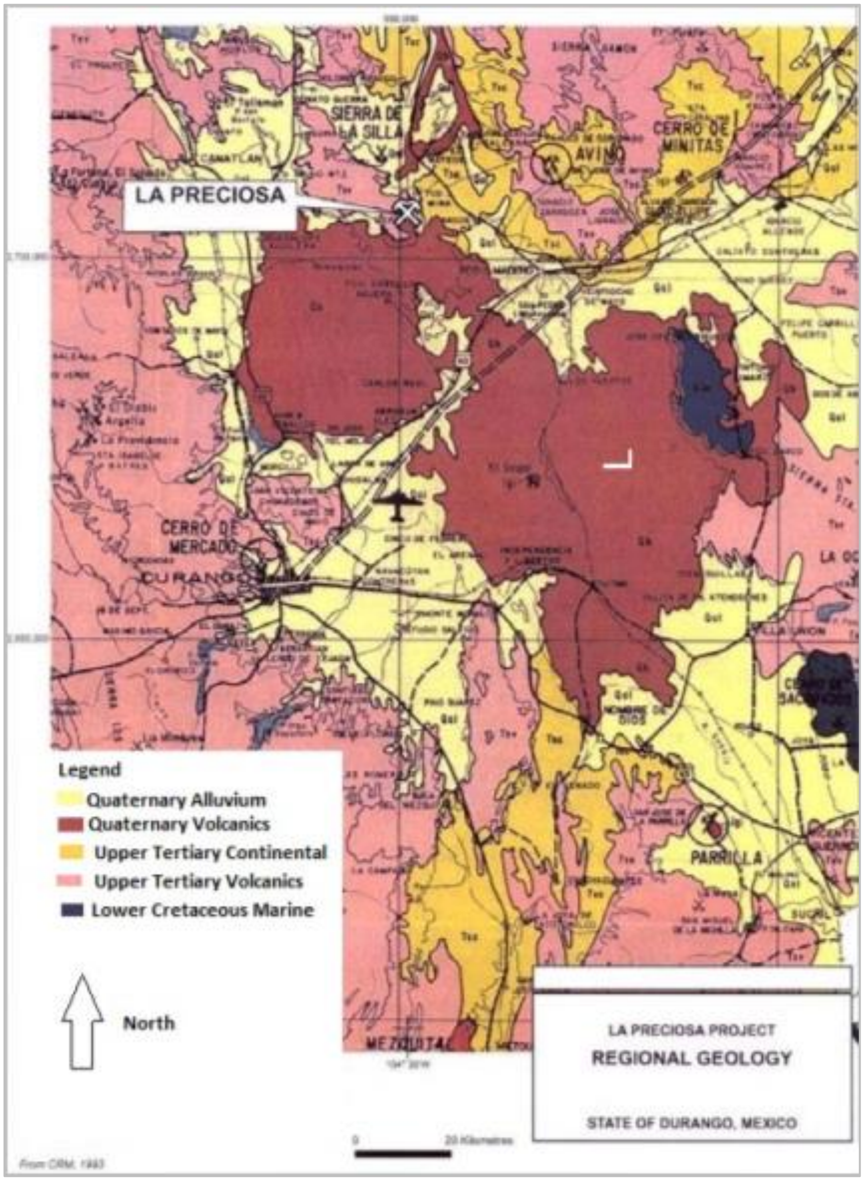


Figure 7-5: Regional Geology Map

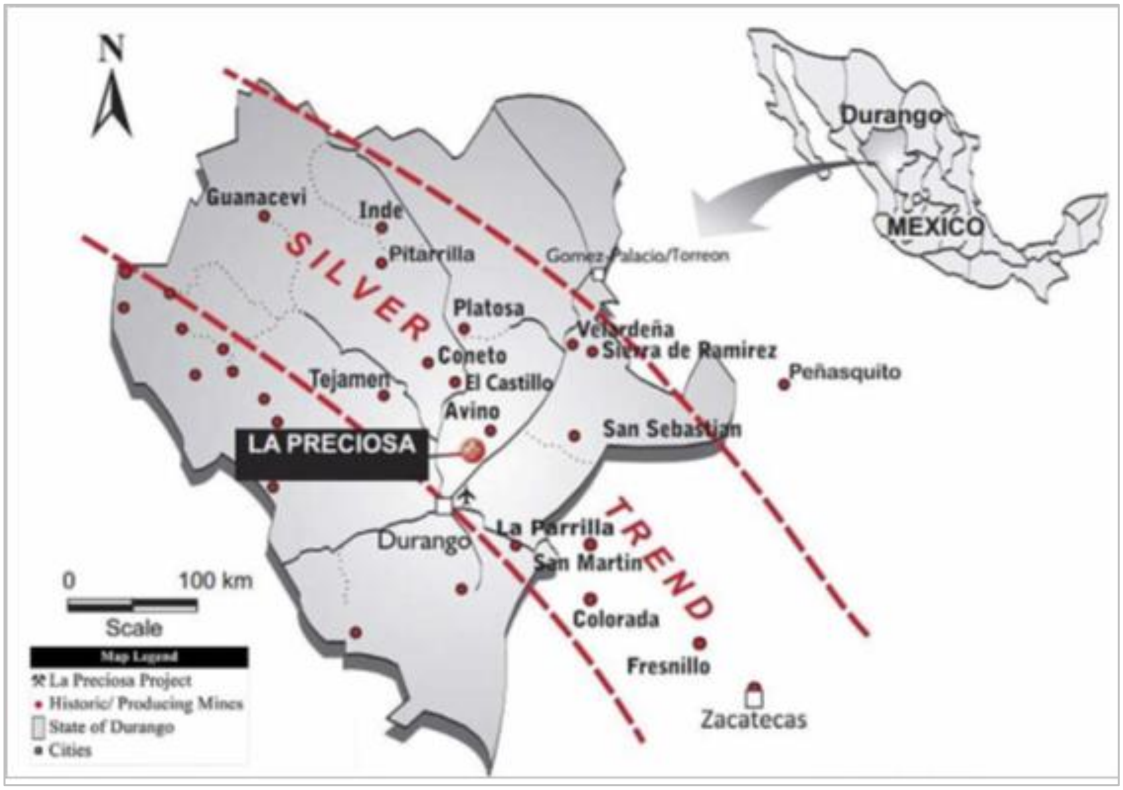


Figure 7-6: Mineral Deposits in the La Preciosa Area

7.2.2 La Preciosa Local Geology and Mineral Deposits

7.2.2.1 Local Geology

The oldest rocks in the La Preciosa area are Jurassic-Cretaceous metasedimentary graphitic schist, chlorite schist, and layers of quartzite (Figure 7-7 and Figure 7-8). These metasedimentary rocks do not outcrop at surface but are intersected in drill core. Overlying the metasedimentary sequence is a thick package of unmetamorphosed polylithic conglomerate containing lenses of arkosic sandstone of unknown age.

The sedimentary package is overlain by intermediate tuff and agglomerate of the regional Tertiary age LVC. In places, the flows are porphyritic or glomeroporphyritic, and the tuffs are partly welded. The youngest rocks within the property are basalt flows that erupted from several Pleistocene-age volcanic vents and which now fill the lower valleys. Cerro Prieto, Cerro Blanco, and Cerro La Chicharronera are prominent examples of the volcanic vents. Other nearby (9 km west) volcanic vents is the Holocene age, La Breña-El Jagüey maar complex, which is part of the Durango Volcanic Field. Sporadic mafic to felsic dikes and sills of unknown age are found in the deeper parts of the area and rarely at surface.

The area contains a series of Tertiary-age silver-bearing (±gold) epithermal quartz veins associated with barite, fluorite, and sporadic base metals, primarily zinc and lead. There are two major vein and vein-breccia systems exposed on a series of hills and ridges, which are separated by a flat-floored valley roughly 800 m wide. The conglomerate and Tertiary Lower Volcanic andesitic rocks are the main host rocks for quartz veins, although vein mineralization does extend into the basement metasedimentary rocks.

The main veins system on the Abundancia Ridge consists of dominantly south-striking and west-dipping veins plus east-southeast-striking, south dipping crosscutting veins. For example, the Abundancia Ridge vein system has been traced on surface for more than 1.5 km, and drilling has revealed that the veins continue to the north, beneath basalt cover.

Along the eastern side of the Project, a series of hills expose a north- to northwest-striking, shallow west-dipping vein system with associated hanging wall veining and alteration. This vein system is referred to as the Martha vein or fault zone and has been traced by drilling for over 2.5 km along strike.

Mineralization at the Project is hosted within multiple discrete poly-phase quartz veins, often displaying banded, smoky, drusy, and chalcedony textures. Also, in each stage there is variably crustiform banded fracture fill/breccia cement mineralogy. Fluorite, amethyst, a substantial number of barite laths, calcite, and rhodochrosite may also be present, and sulphide mineralization in the form of sphalerite, galena, pyrite, chalcopyrite, acanthite, sparse native silver, and free gold, as well as iron and manganese oxides have been noted in drill core. The principal silver bearing mineral at the Project is acanthite-pseudomorphic after argentite or as microcrystalline to amorphous grains.

Vein mineralization does extend into the basement metasedimentary rocks, but its extent and distribution is not well understood. The main vein system on the Abundancia ridge consists of dominantly southward-striking and westward-dipping veins plus east-southeast–striking, south-dipping crosscutting veins. The Abundancia ridge vein system has been traced on surface for over 1.5 km. Along the eastern part of the Project, a series of hillocks expose a north- to northwest-striking, shallow west-dipping vein system with associated hanging wall veining and alteration. This vein system is referred to as the Martha vein or fault zone and has been traced by drilling for over 2.5 km along strike.

Examination of mineralized samples identified mainly argentite, tennantite/tetrahedrite, and Ag sulphosalts in samples. The majority of gold/electrum is inter-grown with or occupying the same paragenetic position as argentite, silver sulphosalts, sphalerite, and galena, mostly transitional between quartz and carbonate/iron carbonate in formation.

Wall rocks hosting mineralization are variably silicified, with proximal patchy illite-smectite alteration and distal chlorite alteration. The presence of manganocalcite has been noted in several drillholes, but it is not uniformly distributed. In shallower drillholes, pyrolusite and limonite often appear on fracture surfaces.

The host rocks and veins have undergone intense weathering. The base of oxidation is erratically distributed as weathering is controlled by the presence of post mineralization faults which allowed the percolation of oxidized meteoric groundwater to vertical depths of 350 m below surface. Weathering minerals include iron oxides, iron carbonates, manganese oxides, and unidentified clays.

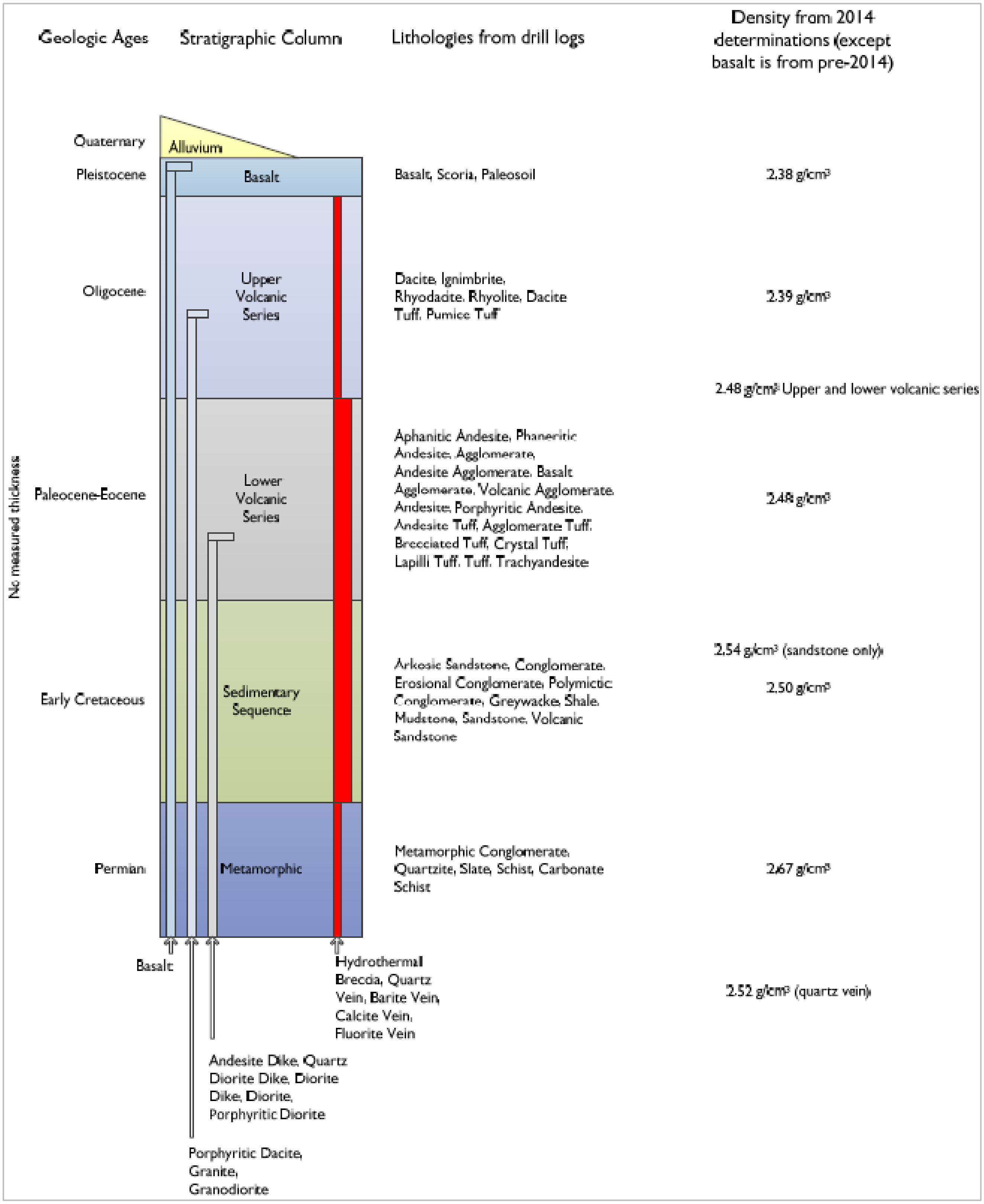


Figure 7-7: Project Local Stratigraphic Column

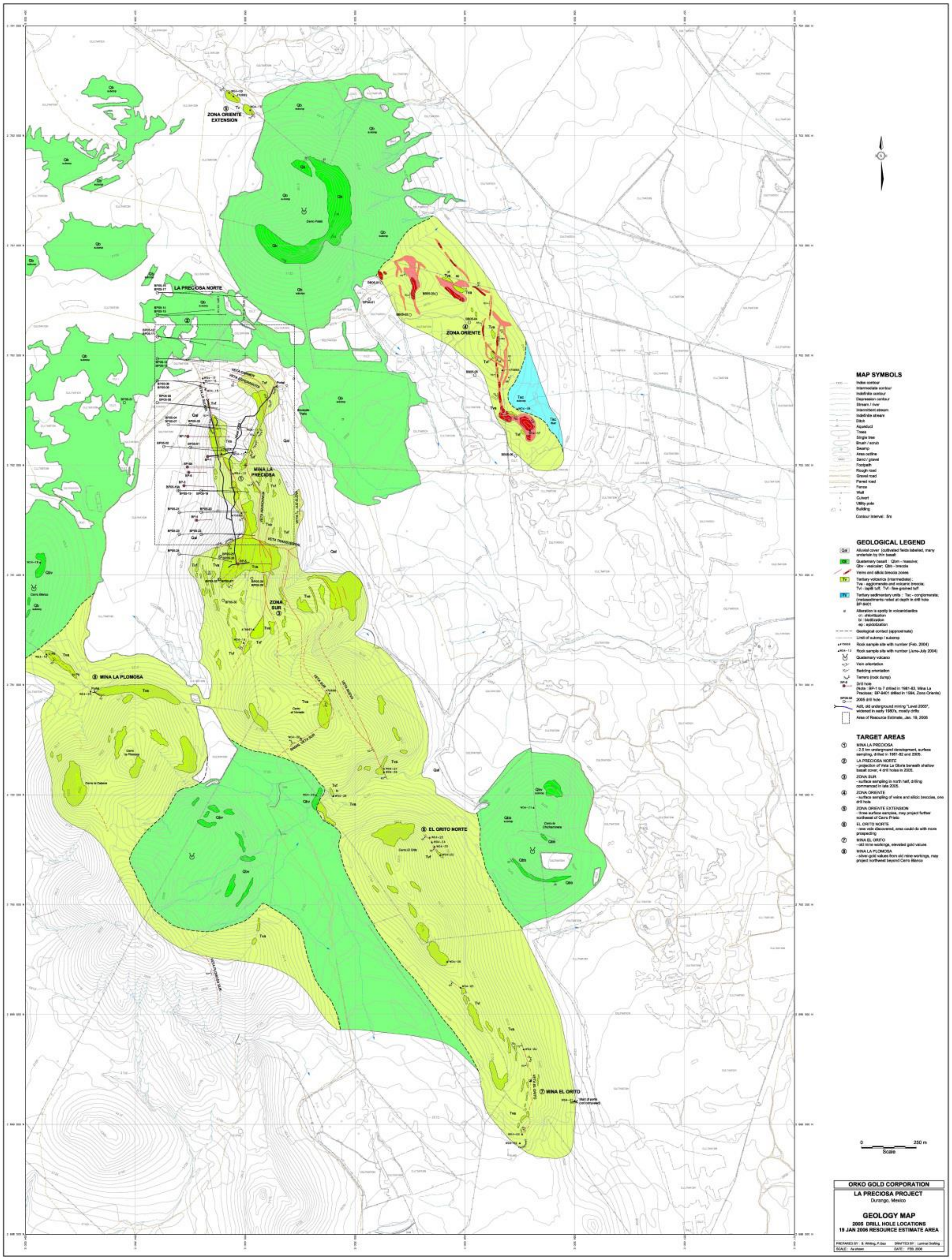


Figure 7-8: Local Geologic Map (Orko 2006)

7.2.2.2 Lithological Units

The main rock types occurring in the Project area include volcanic flows, pyroclastic rocks, sedimentary rocks (clastic and epiclastic rocks), metamorphic rocks, and subvolcanic dikes. The lithological descriptions below include the alteration products that were most commonly observed.

Volcanic Flows: Distinctive volcanic flows include vesicular basalt, basaltic andesite, and latite to andesite. The lithology logs also include rhyolite. Rhyolite flows were not observed during the alteration study. Quartz-phaneritic felsic crystal-lithic tuff is abundant in the northeastern part of the property, and it is possible that rhyolite flows occur within that unit.

Vesicular basalt: Vesicular basalt is the youngest lithology in the property. It forms sub-horizontal flows that cover the tilted volcanic, pyroclastic, and volcaniclastic sequence beneath. The vesicular basalt unit is black, fine-grained, and has abundant vesicles. It commonly overlays a paleosol horizon that varies in thickness from tens of cm to m. The vesicular basalt unit is not affected by visible hydrothermal alteration.

Basaltic andesite: Basaltic andesite is dark purple-brown, sparsely porphyritic, with phenocrysts that average less than 1 mm in length. Phenocrysts and to a lesser degree groundmass are commonly altered to carbonate.

Latite to andesite: Latite to andesite is beige colored, porphyritic, with few feldspar phenocrysts and few mafic phenocrysts. Feldspar phenocrysts are variably altered to clay, mafic phenocrysts are variably altered to chlorite, clay, pyrite, or hematite after pyrite, and groundmass is variably altered to clay.

Pyroclastic Rocks: Distinctive volcanic pyroclastic lithologies include lapilli-tuff breccia, crystal-lapilli tuff, felsic crystal-lapilli tuff-breccia, and lesser mafic lapilli tuff and mafic crystal tuff.

Lapilli-tuff-breccia: Lapilli-tuff-breccia is the most widespread pyroclastic lithology observed in the deposit. It has a dark green to brown green magmatic, porphyritic matrix enclosing angular lithic clasts and juvenile clasts of various sizes. Minor agglomerate and clastic horizons occur within the lapilli-tuff-breccia. Lapilli-tuff-breccia is consistently classified as agglomerate in lithology logs. However, according to the pyroclastic rock classification, the term agglomerate should only be applied for pyroclastic rocks that contain over 75% pyroclasts greater than 64 mm, whereas the lapilli-tuff-breccia commonly contains over 40% magmatic matrix. Intervals of autobreccia were also observed locally.

Significant differences in composition between magmatic matrix and lithic clasts makes this unit favourable for patchy alteration facies. Smectite clay is a common secondary mineral in the lapilli-tuff-breccia, and may be a product of devitrification of the magmatic matrix and of juvenile clasts, or of hydrothermal activity.

Crystal-lapilli-tuff: The lapilli-tuff-breccia (described above) grades into intervals of crystal-lapilli-tuff and lapilli-tuff that is up to several m thick. Flow banding is observed locally in the form of aligned lapilli. Feldspar crystals are variably altered to clay, which are most commonly kaolinite or illite.

Felsic crystal-lapilli-tuff-breccia: Felsic crystal-lapilli-tuff-breccia has abundant quartz crystals, subangular to angular porphyritic lithic lapilli, and numerous fiamme enclosed in a light reddish-brown to white or light green-white magmatic matrix. This unit occurs in the northeast portion of the Project area, and includes intervals of felsic crystal±lapilli-tuff up to several m wide that locally contain that contains feldspar crystals. Feldspar crystals and lithic clasts are variably altered to clays.

Sedimentary Rocks: Sedimentary rocks include conglomerate, sandstone, and minor siltstone and shale.

Sedimentary Breccia and Conglomerate: Greenish-grey to brown, matrix to clast supported, moderately to poorly, and locally well sorted polylithic sedimentary breccia to conglomerate is the most widespread sedimentary lithology. Weak carbonate and hematite cement are common. Disseminated, fine-grained euhedral pyrite is commonly observed within the cement. Locally the matrix is completely replaced by epidote.

Sandstone: Sandstone forms lenses that are typically less than 6 m thick, and occur most commonly associated with the conglomerate. Irregular, thicker sandstone horizons up to 20 m thick also occur in the upper parts of the volcanic stratigraphy in the northwestern part of the property. Locally, sandstone grades into siltstone and shale beds that are generally less than 2 m thick.

Metasedimentary Rocks: Metasedimentary rocks include muscovite-schist and quartz-muscovite-schist that form the basement to the overlying sedimentary and volcanic sequence. The upper contact of the metasedimentary rocks is commonly veined, intruded by subvolcanic dikes and sills, and faulted. The schist contains two well-developed foliations, one of which is folded by the other.

Subvolcanic Intrusions: The entire volcanic, sedimentary, and metasedimentary sequence is intruded by dikes and sills of felsic to intermediate composition. The subvolcanic intrusions are most abundant along the contacts between basement metasedimentary rocks and the overlying conglomerate. Dike and sill contacts generally show strong clay alteration.

Light to dark colored feldspar porphyry dikes and sills are sparsely porphyritic, but locally have few to moderate feldspar phenocrysts and few mafic phenocrysts that are completely converted to pyrite and clays. Feldspar phenocrysts are variably clay altered. Groundmass is commonly moderately sericitized.

7.2.2.3 Alteration

The principal visible alteration facies observed in the Property consist of:

- Patchy albite-epidote±chlorite flanks the deposit to the west, north, and southeast, and produces strengthening of the rock. Chlorite, brucite, and epidote are the most common minerals present in this facies.
- Silica-sericite-pyrite occurs along northwest and east-northeast trending corridors, and does not appear to be intense enough to affect rock strength. Illite and muscovite are the most common minerals in this facies.
- Pseudomorphic clays and carbonate after phenocrysts occurs throughout all porphyritic lithologies and does not define specific trends or affect rock strength. Illite, phengite, and montmorillonite, iron-carbonates, and lesser chlorite are the most common in this facies.
- Pervasive texture destructive silica defining northwest-trends are often defined by zones of breccia.
- Fault-fill clays are restricted to post-mineral faults. Montmorillonite is the most common mineral in this facies.

The most frequently occurring are (in order of decreasing frequency): montmorillonite, illite, phengite, iron carbonate, silica, chlorite, brucite, muscovite, kaolinite, calcite, and epidote. Among the alteration minerals, muscovite is the mineral that appears to have best spatial correlation with faults.

The Deposit lacks a distinct halo of a high illite crystallinity surrounding mineralization. This lack of an alteration halo is interpreted as being due to a combination of the strong lithological control over illite crystallinity, and to the scale of this alteration study, which was conducted along the main mineralized zones. It is possible that a broad zone of illite crystallinity high would be defined at a more regional scale.

7.2.3 Mineralization

The area has been cut by numerous structures, both northwest and northeast oriented (to north-northwest and north-northeast), as well as ~eastwest; post-Oligocene extension resulting in graben-style faults, possibly with low-angle listric-type movement. Subsequent mineralization occurred along these low and high-angle faults, and also followed the low-angle contact of the basement or conglomerate with the tuff. The Martha vein, with a dip of ~20° to the southwest, defines the unconformity at depth. The shallowly dipping Abundancia vein dips ~50° to the west-northwest, and the high-angle La Gloria vein in the west dips ~75° to the west-southwest. Internal to this main system of veins are also areas of veinlets and stockwork, which constitute most of the mineralization.

Mineralization is controlled by three types of structures:

- Type 1: structures commonly associated with faults and exhibit crustiform, cockade, and colloform textures that are representative of multiple vein opening stages. These veins generally have widths of greater than 30 cm and can form vein systems up to several m wide. Cavities are also common in these veins. Quartz stockwork comprising mm- to cm-scale quartz veinlets is common both in the hanging wall and footwall of Type 1 vein systems.
- Type 2: structures consist of veins that range in width from 1 cm to several tens of cm, and rarely include veins up to 6 m wide (e.g., Abundancia and La Gloria veins). Type 2 veins are dominated by colloform textures with sugary quartz and euhedral crystals projecting into cavities along the vein centres. Dilation (or jigsaw) breccia veins are also common, with angular clasts of wall rock (typically fine-grained volcaniclastic rock) in quartz and (or) calcite cement. Colloform textures and crystal growth into cavities are characteristics of open-space filling which commonly occurs in extensional settings.
- Type 3: structures are commonly associated with abundant hematite alteration of the host rock, breccia, minor stockwork development, and patchy or narrow quartz vein development. Type 3 structures are typically fault zones up to several m wide with variably developed quartz-carbonate-calcite veins and fault breccia.

As previously mentioned, the mineralization in the area occurs in veins, veinlets, and stockwork. These veins average in true width under 15 m (Martha Vein) and consist of several stages of banded crustiform to colloform, quartz (and cryptocrystalline quartz at shallow depths), adularia, barite, and typically later carbonates (both calcite and rhodochrosite); illite commonly replaces the adularia. There are variable amounts of pyrite, sphalerite, and galena plus argentite, and variable amounts of tetrahedrite - tennantite, freibergite, and Ag sulfosalt.

7.2.3.1 Local Mineralization

The district has many characteristics that are typical of epithermal veins in Mexico, particularly of the Ag-rich variety. Quartz veins are accompanied by adularia, barite, calcite, rhodochrosite of variable timing, as well as acanthite, freibergite, Ag sulfosalts and minor electrum, plus variable amounts of pyrite, honey-colored sphalerite, tennantite/tetrahedrite, chalcopyrite and galena, and supergene Fe and Mn oxides; the hypogene minerals are characteristic of intermediate-sulphidation deposits in Mexico. Mineralization is believed to be Tertiary in age both the LVS and UVS are mineralized, but the basalts are recent and not mineralized.

Petrographic studies of the veins in the Deposit find that multiple stages of silver and base metal mineralization are associated with repeated fluid boiling and mixing events, defined by crustiform banded fill/cement assemblages within a framework of intermittent and more significant fracturing/rupturing of wall rock and pre-existing vein/cement assemblages. There is a repetition of common hydrothermal fill/cement mineralogy, including mineralized minerals, such that correlation of vein/cement assemblages/events between drillhole intersections would be difficult.

The occurrence of adularia and style of early quartz and chalcedonic quartz replacement amongst wall rock replacement and fracture-fill/cement assemblages confirms silver and base metal mineralization associated with low sulphidation, epithermal style systems developed on the Martha and Olin structures at the Project. Significant widths of mineralized quartz and carbonate dominated fracture-fill and breccia cement assemblages have developed as a result of extended episodes of hydrothermal fluid flow and repeated rupturing of wall rock and pre-existing vein/cement assemblages. Internal crustiform banding within the different voluminous fill/cement assemblages represents incremental opening and filling of fractures/cavities between major rupturing events.

The Martha vein is the largest vein in the deposit by far, with at least three times the volume of the next largest vein, La Abundancia. Both veins are low angle, the Martha vein dips ~20-30°, following the southwest-dipping contact of volcaniclastic rocks overlying an immature conglomeratic unit (consisting mainly of polylithic clast-supported fragmental rock with angular to sub-rounded clasts) or the underlying schist.

There are also high-angle veins in the west on the ridge, such as La Gloria vein, the largest of this set of veins. These high-angle veins can be considered as a mineralized zone or lode of stock work, silicification, breccias, veins, vein breccias, veinlets, and a general mix of multiple styles of mineralization. Within this broader zone, for example the Martha lode ranges from 1 to 35 m thicknesses and averages approximately 5 m.

7.2.3.2 Structural Geology

There are three main types of syn-mineralization veins and faults in the property:

- Type 1 – Silver-gold bearing, south-southeast– and south-striking, shallow west-dipping structures (e.g., the Martha fault zone).
  - These structures are commonly associated with faults and exhibit crustiform, cockade, and colloform textures that are representative of multiple vein opening stages. Veins generally have widths greater than 30 cm and can form vein zones up to several m wide.
  - Steep down-dip (i.e., shallow west-plunging) mineral lineation and associated steps indicate that these structures developed as normal faults.
- Type 2 – Silver-gold bearing, south-southeast– to south-southwest–striking, moderate to sub vertical west-dipping structures (e.g., the Abundancia and La Gloria veins).
  - These structures contain veins that range in width from 1 cm to several tens of cm, and include rare up to 6 m wide veins. Vein textures comprise colloform banding, dilation (jig-saw) breccia, and euhedral crystals projecting into cavities along the vein centres typical of extensional veins. Few faults are associated with these veins, although vein walls are sometimes characterized by smooth and striated post-mineralization faults.
  - Type 2 veins developed as extensional veins in the hanging wall and footwall of Type 1 structures. Rare syn-mineralization faults display steep-west plunging mineral lineation and associated steps indicating normal dip-slip movement.
- Type 3 – East-southeast-striking, moderate to steep south-dipping structures (e.g., two ESE structures, La Plomosa, and Transversal veins) with sporadic silver-gold bearing quartz veins. These structures are up to several m wide, consisting of fault zones with variably developed quartz-carbonate-calcite veins and fault breccia commonly associated with hematite alteration of the host rock.
- Dominantly moderate to steeply west-plunging mineral lineation and associated steps along southwest-dipping veins indicate that these structures developed as normal-dextral oblique-slip faults.

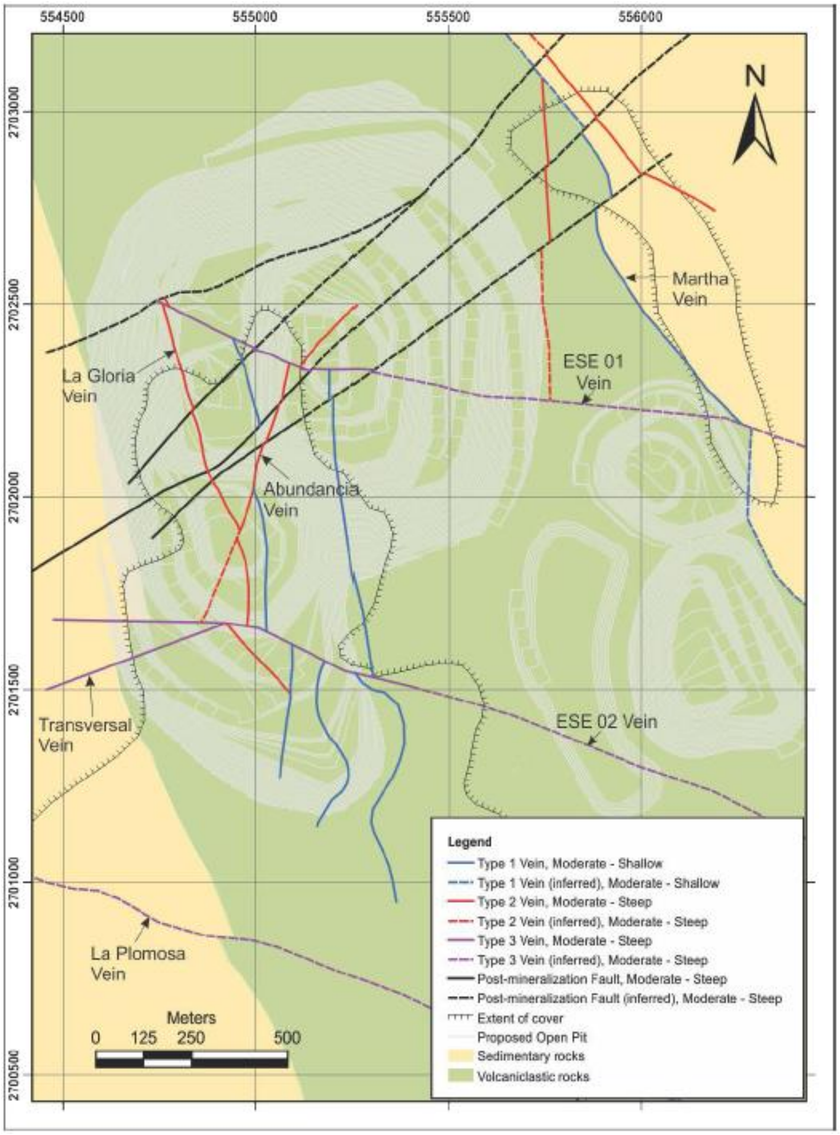


Figure 7-9: Structural Geology Map for the Project (SRK 2014)

8.0 DEPOSIT TYPES

Regionally, the Property is situated within a 12 km by 8.5 km caldera that hosts numerous low- to intermediate-sulphidation silver-gold epithermal veins, breccias, stockwork, and silicified zones, grading into a “near porphyry” environment in the Avino Property.

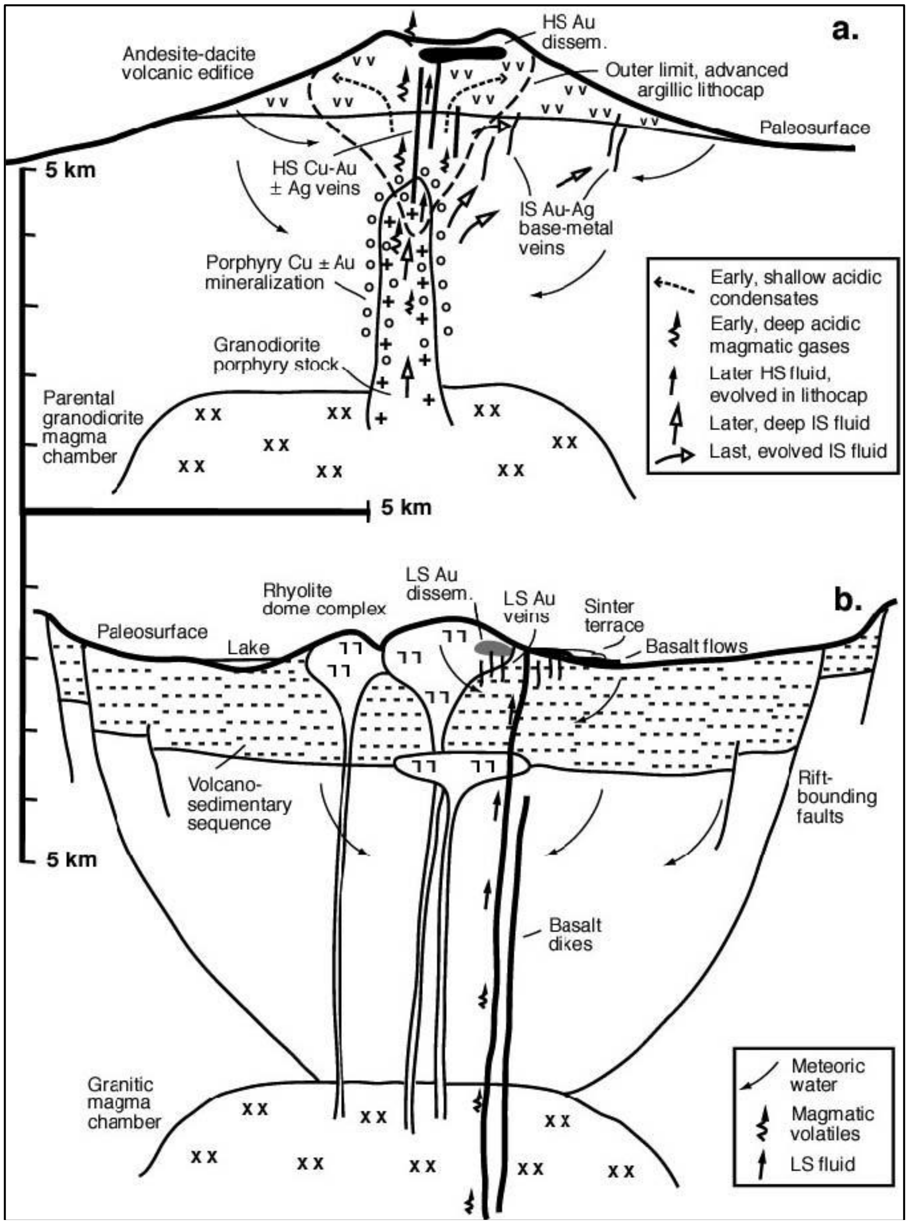
The historic mining on the Property was mainly on the Avino Vein, a steep-dipping (50 to 80 degrees), silver-gold-copper-rich epithermal vein. The steep-dipping San Gonzalo Vein, however, has a much lower copper content than the Avino Vein and is more equivalent to other silver-lead-zinc deposits of the Sierra Madres.

The numerous veins in the La Preciosa area tend to be narrower and flatter 15 to 40 degree dips). Low-sulphidation vein systems are commonly characterized by low concentrations of sulphide minerals, alteration mineralogy dominated by quartz-adularia-sericite, and a lack of extensive wall-rock alteration. Conversely, high-sulphidation vein systems are commonly characterized by sulphur saturation leading to the presence of native sulphur and sulphide minerals, quartz-alunite alteration, and extensive wall-rock alteration. The Mexican silver deposits are usually within the intermediate sulphidation range rather than either of the end member classifications.

Low sulphidation deposits occur as veins, breccias, and disseminated precious metal mineralization deposited by the circulation of neutral to weakly acidic hydrothermal fluids along regional fault structures, fracture zones, or through highly permeable lithologies such as ignimbrite and agglomerate. Because the fluids are relatively neutral, very little alteration is evident and the veins and nearby wall rock may commonly include illite, sericite, and adularia. Generally, this style of mineralization is distal from a heat source.

Sillitoe and Hedenquist (2003) subdivide epithermal deposits into High- (HS), Low- (LS) and Intermediate-sulphidation (IS) types based on mineralogy, deposit morphology, associated alteration, and geologic setting (Figure 8-1).

Type IS epithermal deposits occur in a broadly similar spectrum (to HS deposits) of andesitic-dacitic arcs, but commonly do not show such a close connection with porphyry Cu deposits as do many of the HS deposits. However, high silica igneous rocks such as rhyolite are related to only a few IS deposits. IS deposits form from fluids spanning broadly the same salinity range as those responsible for the HS type, although Au-Ag, Ag-Au, and base-metal rich Ag-(Au) subtypes reveal progressively higher mineralized material fluid salinities.



– Calc-alkaline volcanic arc with neutral to mildly extensional stress state showing relationships between HS and IS epithermal and porphyry deposits (note that the complete spectrum need not be present everywhere). Early magmatic volatiles are absorbed into ground water within the volcanic edifice (shown here as a stratovolcano, but it may also be a dome setting) to produce acidic fluid for lithocap generation, over and/or supra-adjacent to the causative intrusion. Later, less acidic IS fluid gives rise to IS mineralization, both adjacent to and distal from the advanced argillic lithocap. Where the IS fluid flows through the leached lithocap environment, it evolves to a HS fluid (Einaudi et al. 2003) to produce HS veins or disseminated mineralization, depending on the nature of the structural and lithologic permeability. The HS fluid may evolve back to IS stability during late stages, supported by paragenetic relationships and lateral transitions of high- to intermediate-sulfidation mineralogy.
– Rift with bimodal volcanism and LS deposits. Deep neutralization of magmatic volatiles, typically reduced, results in a LS fluid for shallow LS vein and/or disseminated mineralization and related sinter formation (Sillitoe and Hedenquist 2003).

Figure 8-1: Schematic Sections of End-member Volcanotectonic Settings and Associated Epithermal and Related Mineralization Types

The veins in the Project area consist of several stages of banded, crustiform (to colloform), quartz and cryptocrystalline quartz at shallow depths, adularia, barite, and typically later carbonates both calcite and rhodochrosite, "illitic clay" (illite) commonly replaces the adularia (Coote 2010). There are variable amounts of pyrite, sphalerite and galena plus argentite, and variable amounts of tetrahedrite-tennantite, freibergite and Ag sulfosalts.

The Ag:Au ratio is high, approximately 500:1 for the resource. Supergene oxidation extends to at least 300 m depth, and includes manganese oxide. There is abundant adularia, bladed calcite textures, and coexisting vapour-rich and liquid-rich inclusions, all indicating an ascending, boiling fluid consistent with the abundant evidence for brecciation which suggests that mixing caused metal deposition and carbonate formation.

In Mexico, and particularly within the Mexican Silver Belt, these types of deposits can have large lateral extents but may be limited vertically. There are many silver-gold mines in Mexico, some of which form large mining districts and others that exploit multiple veins over limited vertical horizons that are sometimes only 100 m in depth (Gunning 2009).

Adjacent to the Elena Tolosa Mine in the Avino Mine area, the oxide tailings have been predominantly sourced from legacy open pit operations, and the sulphide tailings have been predominantly sourced from later underground workings. Exposure to surface weathering and historic process activities has homogenized the tailings material to produce a deposit partly included in the Mineral Resource for the Property.

9.0 EXPLORATION

9.1 Avino Mine Area

9.1.1 Early Exploration, 1968 to 2001

Exploration in the Avino Mine area has been ongoing since before production commenced, and the majority of the recorded work has been focused on the main Avino Vein and surrounding area. The following is a summary of significant exploration work conducted either by Avino or on behalf of Avino until the mine closed in 2001.

Pre-production exploration was carried out by CMMA and others and covered 2,500 m of drifting and cross-cuts, as well as 8,000 m of surface and underground diamond drilling. Extensive rehabilitation was completed involving Selco, including connecting three of the old—possibly pre-1900—underground mine workings.

In 1970, a contract was signed with Selco, who spent more than US$1 million in exploration and FSs before returning the histroic Avino Property back to CMMA in 1972, reportedly because of low metal prices. The majority of the documentation examined covered feasibility work and was related to investigations of old underground workings that were likely developed in the late 1800s. A contract was signed in October 1973 with S.G.L. Ltd. and Sheridan Geophysics Ltd., under which a new 500 t/d processing plant was completed in May 1974.

Since 1992, exploration in/for the mine has been limited to traditional underground mine development with associated sampling and planning for production feed. In the late 1990s, it appears that development was not kept up, as company monthly reports showed decreasing historical reserve allocations for production and mill feed.

The only recorded exploration, apart from limited prospecting, is documented in the 1993 report by Servicios Administratos Luismin, SA de CV, the engineering branch of Cía Minera de San Luis Exploration. The study reported on detailed analysis and sampling of the then known showings with the emphasis on the Avino Vein and Potosina/El Fuerte area. The extensive underground sampling program carried out by Luismin provided a later direction for underground mining. The report made recommendations for follow-up for drilling and underground development for the main Avino Vein, as well as trenching and drilling recommendations for the Potosina/El Fuerte area. It is believed that these recommendations were never implemented for the prospective areas. Additionally, the report included a property-scale geological mapping and lithogeochemical sampling program, which was contoured and coloured for gold, silver, copper, lead, zinc, arsenic, antimony, and mercury.

Other notable observations from the study include the following:

- All mineralization, except for Nuestra Senora and Potosina/El Fuerte, radiate outwards in a west-to-northwest direction from the Cerro San Jose. The Cerro San Jose is a silicified and partly hornfelsed body of volcanic rock probably overlying an intrusive stock, which could have been the source of most mineralization in the historic Avino Mine area.
- Mineralization in all radiating structures is described as being strongest 2 km to 3 km from Cerro San Jose. This resembles many of the gold deposits in Nevada, where the source of mineralization is a near-surface acid-intrusive but with mineralized bodies lying 1 km to 5 km away along high-angle faults.
- The two strongest and widest structures appear to be the Avino and Aguila Mexicana veins.

- The Avino Vein has three main mineralized zones—San Luis, ET (La Gloria/Hundido) and Chirumbo areas—which rake to the west and are open at depth.
- The existence of other mineralization cutting the Cerro San Jose mineralization in the Nuestra Senora and Potosina/El Fuerte areas could offer the potential for bulk mineable stockwork zones.

Assay values from outcrop sampling of surface-mapped veins towards the San Jose hill ranged from lows of 2 g/t silver and trace gold over true thicknesses from 0.1 m to 2.3 m up to a high of 755 g/t silver with a corresponding 1.5 g/t gold over a thickness of 0.45 m.

No systematic sampling, trenching, or drilling of either the outcrops or the veins is known to have occurred during the program undertaken in 1993.

9.1.2 Recent Exploration, 2001 to Present

After the temporary mine closure in 2001, Avino intermittently conducted exploration work, with the intention of expanding and better defining known areas of mineralization. Historic near-to-surface mining activities are being relied upon for guidance, and modern techniques are being employed to integrate, manage, and interpret results. Included in the list of exploration activities is an induced polarization geophysical survey, 1,500 soil samples, satellite imagery, mapping, trenching, tailings investigations, bulk sampling, underground channel sampling, and surface drilling.

9.1.2.1 Tailings Investigations (Oxides), 2003 and 2004

Two specific mineralogical assessments were conducted in 2003 and 2004 on samples from the tailings at the Avino Mine. The purpose of the program was to provide data for an independent investigation of the 1990 drilling results on the oxide tailings (discussed in Section 10) in terms of verifying assay grades and volumes, as well as to examine the metallurgical characteristics of the material. The results and implications of these findings are discussed further in Section 13.

The following information regarding the 2004 sampling is summarized by Slim (2005d).

The 2004 tailings fieldwork was under the direction of MineStart, and excavation of the sample pits was under contract with Desarrollos Rod Construcciones of Durango. Given the hydraulic deposition of the tailings, four important factors required examination: anomaly characteristics of the samples and total population, assay comparison by the fence, examination of downstream decrease in assays, and factors arising from the downstream construction.

Comparison of the 2004 assays with those from 1990 shows consistency in assay values and provides confidence in the 1990 sampling and assaying program.

The preliminary investigations in 2003 showed the need for a sampling of the oxide tailings to validate the assay results of the 1990 drilling and to carry out metallurgical characterization, the latter requiring large samples.

The sampling exercise carried out in 2004, using shallow (4 m deep) backhoe trenches and hand-dug pits, represented a local corroboration of the previous sampling but could not be considered to constitute a representative random sampling of the oxide tailings.

The trench sampling material (Z-series) from the 1993 campaign was also considered to be non-representative.

9.1.2.2 Tailings Sampling (Sulphides), 2005

Some sampling was carried out in 2005 by means of hand-dug pits on the "upper bench" of sulphide tailings. The silver and gold values generally ranged from 40.0 g/t to 100.0 g/t and 0.3 g/t to 0.6 g/t, respectively. While these values give a general idea of the potential grade of the sulphide tailings, they have not been verified to be representative of the sulphide tailings, even at a local scale.

9.1.2.3 Bulk Sample Program of San Gonzalo Vein, 2011

Avino completed a 10,000 t bulk sample program at the San Gonzalo deposit following a comprehensive review of the data and discussions with Tetra Tech. The bulk sample feed grade was 261 g/t silver and 0.9 g/t gold. Silver and gold recoveries were stated to be 76% and 59%, respectively, and 232 dry t of flotation concentrate was produced.

9.1.2.4 Underground Channel Sampling of San Gonzalo and Avino Vein, 2010 to Present

Underground channel sampling began in 2010 and has continued to the present at ET Mine (Avino Vein system) and San Gonzalo Mine. Channel sampling data generated since 2010 are summarized in Table 9-1 and Table 9-2.

Table 9-1: Summary Underground Channel Sampling by Level for the Avino (ET) Underground Mine

Level	Elevation (m)	Total Sampled (m)	Average Channel Length (m)	Ag (g/t)	Au (g/t)	Cu (%)
6.5	2,271	373.6	3.9	113.01	0.98	0.51
7	2,241.8	1,197.6	6.1	72.50	0.42	0.44
7.5	2,212.8	230.5	7.0	66.15	0.49	0.50
8	2,199.1	486.3	6.2	141.32	1.14	0.22
8.5	2,171.9	576.9	6.6	123.19	1.37	0.48
9	2,147.1	1,343.8	7.5	121.65	1.52	0.57
9.5	2,128	768.2	9.1	122.42	2.26	0.76
10	2,115	2,905.9	7.3	72.12	0.60	0.49
10.5	2,101	1,468	7.1	106.90	0.77	0.65
11	2,083	1,214.7	9.6	87.57	0.48	0.73
11.5	2,067.2	1,289.9	7.5	89.65	0.44	0.63
12	2,051	1,092	8.2	94.87	0.41	0.76
12.5	2,034.3	1,356.5	7.8	84.94	0.51	0.69
13	2,016.1	645.8	6.1	61.64	0.24	0.63
13.5	1,995.5	386.3	5.2	57.01	0.24	0.51
14	1,975	578	5.4	54.56	0.13	0.52

table continues…

Level	Elevation (m)	Total Sampled (m)	Average Channel Length (m)	Ag (g/t)	Au (g/t)	Cu (%)
14.5	1,954.3	2,622.4	6.3	54.18	0.38	0.56
15	1,932.5	2,341.1	6.0	63.98	0.48	0.58
15.5	1,910	2,182.4	5.8	59.15	0.41	0.57
16	1,889	3,638.4	5.4	49.16	0.56	0.57
16.5	1,870	3,491.1	5.2	50.71	0.40	0.68
17	1,849	3,146.68	6.6	48.26	0.42	0.71

Table 9-2: Summary of Underground Channel Sampling by Level for the San Gonzalo Mine

Level	Elevation (m)	Number of Channels	Total Sampled (m)	Average Channel Length (m)	Ag (g/t)	Au (g/t)	Cu (%)
1	2,311.9	114	272.8	2.4	157.65	0.43	-
2	2,265.3	314	840.6	2.7	115.75	0.40	-
3	2,218.3	378	1,046.1	2.8	119.92	0.41	-
4	2,180.0	685	1,814.2	2.6	241.59	1.15	-
5	2,138.5	740	2,031.9	2.7	285.70	1.57	-
6	2,091.8	603	1,667.8	2.8	186.70	1.14	-
6.5	2,064.4	243	682.4	2.8	177.17	0.86	-
7	2,046.9	190	517.4	2.7	111.50	0.71	-
7.5	2,020.0	114	295.6	2.6	179.90	1.01	-

9.1.2.5 Underground Channel Sampling of San Gonzalo and Angelica Vein, 2010 to Present

Underground channel sampling began in 2010 and has continued to the present. Channel sampling between 2010 and 2012 was summarized by Tetra Tech (2013). Results of underground sampling since 2013 are summarized in Table 9-2.

Figure 9-1 and Figure 9-2 show the location of all channels, colour coded by grade, included in the current resource estimate (Section 14.1.2), within and adjacent to the Avino and San Gonzalo Vein systems, respectively. Drill holes are also shown for orientation.

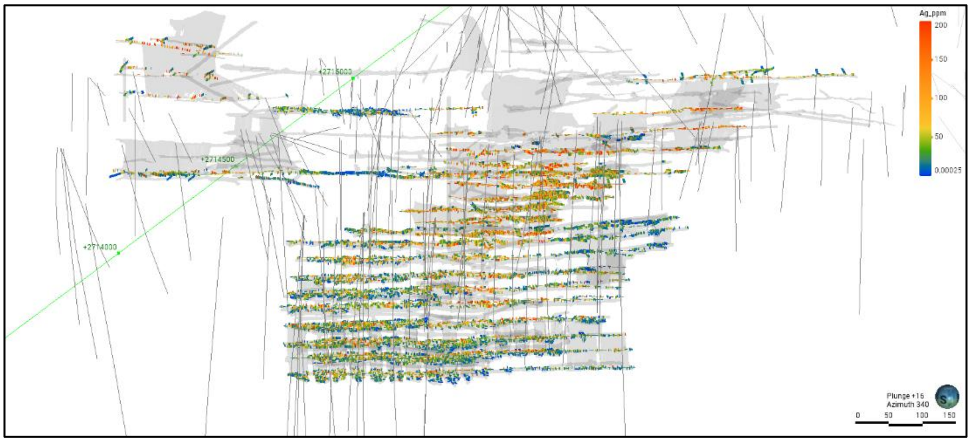


Figure 9-1: Channel and Drillhole Samples, Colour Coded by Silver Grade, within the Avino System

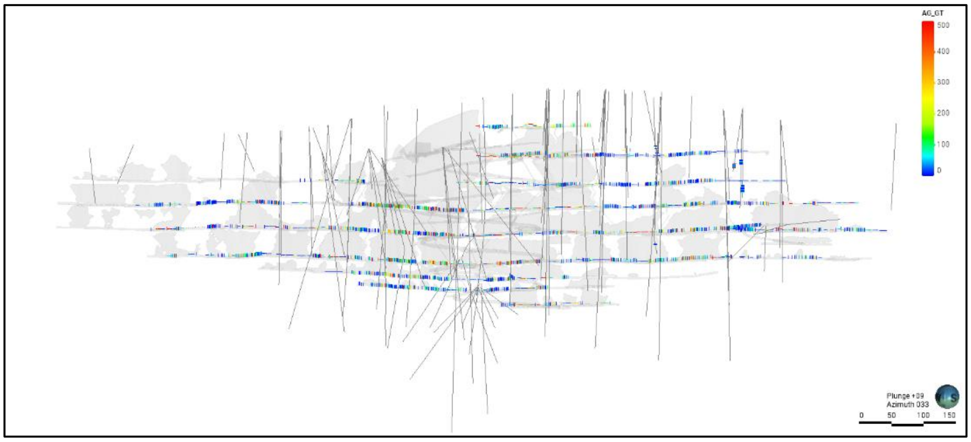


Figure 9-2: Channel Samples, Colour Coded by Silver Grade, within the San Gonzalo Vein System

9.2 La Preciosa Area

9.2.1 Summary of Past Exploration

Exploration and other work at the Project date back to mining in the late 1800s on the Abundancia and La Gloria veins, two prominent veins exposed on the surface of La Preciosa Ridge. This work, which ceased in the early 1900s, and small-scale underground mining in the 1970s, resulted in the production of a small amount of material from these two veins, estimated by MP (Head and Collins 2012) to be less than 30,000 t. This tonnage estimation was not validated by Coeur but site inspections support that a small amount of mining was previously done.

The majority of work at the Project that is material to the mineral resources is from contemporary exploration, mainly drilling, conducted by Luismin, Orko, and PAS (Table 9-3). In addition to the drilling completed by these companies, other exploration activities, consisting of:

1. Prospect sampling by Orko in 2004, followed by geologic mapping by Orko geologists.

2. Completion of three IP ground geophysical surveys in 2005 that totalled 40 line-km. The resistivity data did not appear to be a useful product of this work, but the chargeability component did identify an anomaly in the valley between La Preciosa Ridge and Zona Oriente.

3. A large geochemical soil sampling program over a grid spanning 5 km north to south and 2 km east to west. This program produced anomalous analytical results from areas near shallowly covered veins such as Veta Nueva, Orito, and Nancy.

Historic exploration (along with recognition of late 1800s/early 1900s mining) was responsible for the identification of anomalous silver and gold in soils and outcropping veins.

9.2.2 Coeur Exploration and Development

Coeur's 2013-2014 drilling program was divided into three types:

- Type I drilling: completion of 21 reverse circulation (RC) drillholes to test and condemn waste dumps and tailings impoundment areas, drilling commenced January 2014 and completed February 2014.
- Type II drilling: infill core drilling between February 2014 and mid-April 2014 completed a 75-hole drilling program totalling 11,437 m. Drilling targeted the first three years of the mine plan to convert inferred to indicated resources and reduce risk in achieving the early mine plan. All drilling was concentrated around the Abundancia Ridge area.
- Type III drilling: from December 2013 to March 2014 Major Drilling, under KP supervision, completed seven HQ3 core holes specifically to obtain geotechnical data in the area of the design pits, tailings impoundment, and process plant footprint. Subsequently, these holes also were sampled for geochemical data.

Coeur has completed development and exploration work at the Project in 2013-2014, as shown in Table 9-3.

Table 9-3: 2013 – 2014 Coeur Exploration and Development Work Summary

Quantity	Data Type	Totals	Target
75 drillholes	Core Holes	11,437 m	In-fill drilling, resource conversion
21 drillholes	RC Holes	8,543 m	Waste dumps and tailings, condemnation drilling
7 drillholes	Core Holes	2,244 m	Geotechnical information
103 drillholes	Drill Samples	12,358 samples	New assay samples from older drillholes
N/A	Geophysical	300 km^2	Magnetic survey for lithological and structural domains
N/A	Geologic Mapping (surface and underground)		Define structural geology
N/A	Drillholes (scanned 109 new and old drillholes)	35,754 m	IR measurements to define alteration
N/A	Drillholes (scanned 26 new and old drillholes)	6,166 m	Televiewer scans for structural geology and geotechnical data

In the opinion of the QP, Coeur's drilling, sampling, and logging was done to industry standards. A total of 25.908 m of RC samples, or 17 intervals, was logged as NR (no return), which is 0.3% of the total amount of RC drilled in 2014 (Table 9-3). RC drilling was specifically focused on exploring sites for waste rock and tailings facilities. Core recovery is reported as 100%, no NR intervals were reported. Because the 2014 core drilling program was designed to infill between existing drillholes, the resulting samples are representative of the mineralization as a whole and are not biased in their location, orientation, sampling method, or metal grade. Since the core drilling infilled the area designed to be mined in the first three years of the mine plan, the spatial density of sampling is good, sufficient for much of the material to be classified as indicated or measured.

In the opinion of the QP, the quantity and quality of the lithological, geotechnical, collar, and downhole survey data collected in the exploration and infill drill programs completed by Coeur, Orko, PAS, and Lusmin are sufficient to support Mineral Resource estimation as follows:

- Core logging meets industry standards for gold exploration.
- Collar surveys have been performed using industry-standard instrumentation.
- Downhole surveys were performed using industry-standard instrumentation.
- Recovery data from core drill programs are acceptable.
- Geotechnical logging of drill core meets industry standards for planned open pit operations.
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.

Core logging did not reveal any unusual geologic features that have not been observed in previous logging in the Project area. Assay results and location of mineralized intercepts are consistent in spatial location and grade of previous drilling in the Project area and no unusually high-grade intercepts or previously unknown mineralized areas were encountered, i.e., the distribution of sample grades from the 2014 drill program are similar to distributions of grades from previous drill programs.

10.0 DRILLING

10.1 Avino Mine Area

Drilling activities performed by Avino since the acquisition of the Avino Mine area are summarized in the following sections. Drillhole assay results have been previously reported (except ET 12-07 to ET-12-09) by Gunning (2009), Tetra Tech (2012), and Tetra Tech (2013) and are not disclosed here.

The most recent exploration drilling is summarized in Table 10-1. The location of this drilling is summarized in Figure 10-1 where the holes are indicated by red traces.

Table 10-1: Exploration Drilling 2019-22

Hole_ID	Easting_UTM	Northing_UTM	Elev_(m)	Depth_(m)	Zone
ET_21_01	569,943.9	2,712,246.7	2,234.6	329.0	Avino_Vein
ET_21_02	569,943.9	2,712,245.9	2,234.6	308.0	Avino_Vein
ET_21_02B	569,947.4	2,712,247.0	2,233.7	381.0	Avino_Vein
ET_21_03	570,023.8	2,712,291.3	2,235.5	336.2	Avino_Vein
ET_21_04	569,935.9	2,712,251.8	2,234.0	314.0	Avino_Vein
ET_21_05	570,532.8	2,712,261.2	2,184.1	260.2	Avino_Vein
ET_21_05B	570,533.5	2,712,261.3	2,183.2	514.2	Avino_Vein
ET_21_06	570,532.9	2,712,261.3	2,183.3	604.2	Avino_Vein
ET_21_07	570,424.0	2,712,250.3	2,183.3	503.0	Avino_Vein
ET_21_08	570,424.5	2,712,249.9	2,183.3	527.0	Avino_Vein
ET_22_01	570,341.1	2,712,214.3	2,180.2	519.4	Avino_Vein
ET_22_02	570,341.4	2,712,213.8	2,180.1	558.5	Avino_Vein
ET_22_03	570,341.6	2,712,213.7	2,180.1	661.4	Avino_Vein
ET_22_04	570,708.2	2,712,572.4	2,220.7	489.4	Avino_Vein
ET_22_05	570,230.1	2,712,169.2	2,178.1	540.1	Avino_Vein
ET_22_06	570,045.5	2,712,078.3	2,205.6	627.3	Avino_Vein
ET_22_07	570,424.1	2,712,247.2	2,182.9	600.9	Avino_Vein
ET_22_08	570,342.6	2,712,213.8	2,180.1	584.0	Avino_Vein
ET_22_09	570,391.4	2,712,220.0	2,180.6	602.7	Avino_Vein
ET_22_10	570,489.5	2,712,123.9	2,171.3	709.1	Avino_Vein
ET_22_11	570,585.1	2,712,295.5	2,186.7	545.4	Avino_Vein
ET_22_12	570,391.7	2,712,219.8	2,180.6	666.2	Avino_Vein
ET_22_13	570,282.8	2,712,193.2	2,180.1	597.4	Avino_Vein

10.1.1 Early Drilling (Prior to Mine Closure), 1968 to 2001

10.1.1.1 Avino Vein

Between 1968 and 2001, at least 25 diamond drillholes, ranging in length from 132.20 to 575.20 m, are reported to have been drilled from the surface into the Avino Vein. Included in this total are 10 holes that were drilled by Selco in 1970 when they were rehabilitating some of the old underground workings to provide access for sampling (Slim 2005d). No further information on these drillholes was available to the QP, and they are not included in the resource estimate for the Avino Vein.

10.1.1.2 Oxide Tailings, 1990 to 1991

Between November 10 and December 5, 1990, and March 8 and May 30, 1991, Avino completed 6 trenches and 28 vertical drillholes in the tailings along 7 fences at a spacing of roughly 50 m by 50 m (Benitez Sanchez 1991). Drilling was completed transversely to the drainage pattern of the tailings. Cut at 1 m vertical increments, 461 samples were assayed for silver and gold at the mine assay laboratory, and occasional moisture contents were reported. Assay results from these drillholes have been previously reported (Tetra Tech 2012). Although the Z-series trenches are included in Table 10-1 and Figure 10-1, they are not included in the oxide tailings resource estimate (Section 14.1.3) as they are not considered representative of the tailings at a local scale (see Section 9.1.2.1). During 2015 and 2016, further drilling was carried out on the oxide tailings.

10.1.2 Recent Drilling (Post Mine Closure), 2001 to Present

A total of 145 drillholes with a total length of 40,848 m have been completed on the Avino Vein system, 140 holes 26,026 m on the San Gonzalo Vein system, 44 holes 6,365 m on the Guadalupe Vein, and 26 holes 5,175 m on the La Potosina Veins, totalling 355 holes and 78,414 m of core drilling. Additional exploration holes have been drilled elsewhere on the histroic Avino Mine, but those drilling results are not considered material. Most holes were surveyed downhole using a Tropari single-shot magnetic instrument.

10.1.2.1 Avino Vein

In 2016, 5,510 m (34 holes see Figure 10-1) were drilled in an infill program in the San Luis/Avino Vein system. In 2017, 1,478 m (7 holes, see Figure 10-1) were drilled in the Chirumbo section (eastern extension) of the Avino Vein. In 2018, 1,345 m (13 holes, see Figure 10-1) were drilled north of the historic open pit and in the Chirumbo section on the Avino Vein.

A total of 25,845 m (97 holes, see Table 10-1) of documented drilling has been used for Mineral Resource estimation on the Avino Vein system.

10.1.2.2 San Gonzalo Vein

A total of 23,804 m (105 holes, Figure 10-1) of documented drilling has been used for Mineral Resource estimation on the San Gonzalo Vein system.

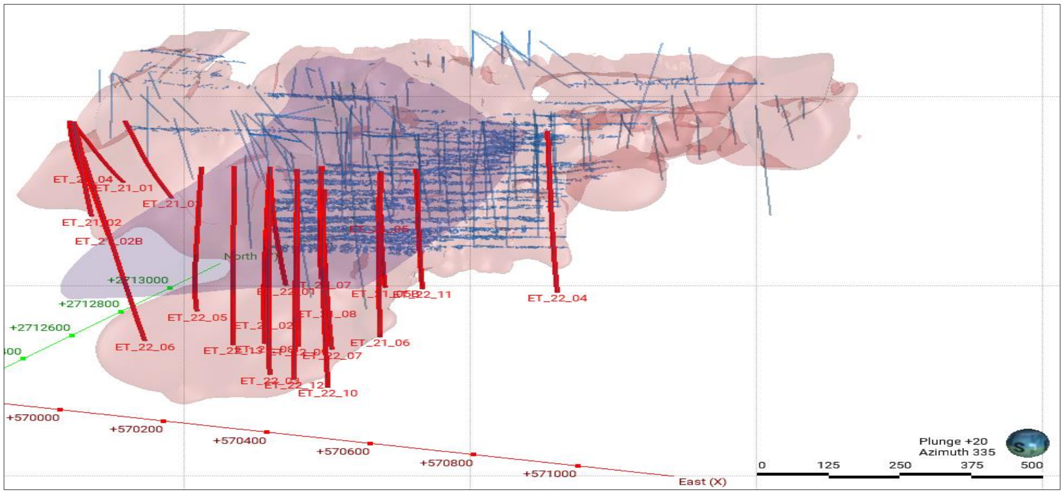


Figure 10-1: Drillholes Completed in 2021 and 2022 on the Avino Vein System, ET Mine. 2018 drill traces in red, previous drilling in blue (Red Pennant 2022)

10.1.2.3 Guadalupe Veins

During 2021 and 2022, exploration drilling was carried out on the Guadalupe veins (see Figure 10-2). These veins are positioned strategically close to and between the ET and San Gonzalo mining operations. A total of 1,106 m of documented drilling has been drilled on the Guadalupe Vein.

10.1.2.4 La Potosina Veins

During 2021 and 2022, exploration drilling was carried out on the La Potosina veins (see Figure 10-2). A total of 733 m of documented drilling has been drilled on the La Potosina Vein.

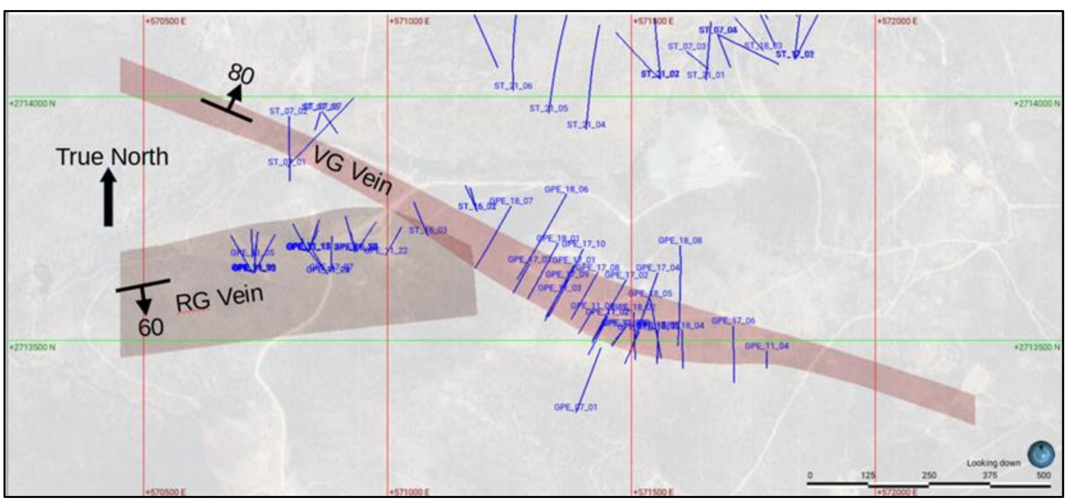


Figure 10-2: Location of 2021/22 Drillholes on the Guadalupe Veins (Red Pennant 2023)

10.1.2.5 Oxide Tailings, 2015 to 2016

During 2015 and 2016, Avino drilled 57 holes in the oxide tailings deposit. Tailings drillholes completed before 2016 on the oxide tailings have been previously reported (Tetra Tech 2022).

10.1.2.6 Oxide Tailings, 2021 to 2022

During 2021 and 2022, Avino drilled 127 new vertical holes using the sonic drilling method on the oxide tailings deposit. Collar coordinates are provided in Table 10-2 and Figure 10-3. The 2015/16 holes are indicated in red.

Table 10-2: Drillholes Drilled on Oxide Tailings 2021 to 2022

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)
PJ_21_01	569,969.5	2,712,373	2,256.284	57
PJ_21_02	569,956.6	2,712,356	2,257.396	57
PJ_21_03	570,207.5	2,712,325	2,204.616	9
PJ_21_04	570,200.8	2,712,306	2,204.44	10.5
PJ_21_05	570,196	2,712,287	2,204.191	16
PJ_21_06	570,188.3	2,712,258	2,204.067	17
PJ_21_07	570,184.9	2,712,240	2,203.789	21
PJ_21_08	570,180.4	2,712,220	2,203.299	21
PJ_21_09	570,175.2	2,712,201	2,203.12	21
PJ_21_10	570,170.6	2,712,181	2,202.989	21
PJ_21_11	570,165.4	2,712,162	2,202.619	18
PJ_21_12	570,160.7	2,712,143	2,201.759	15
PJ_21_13	570,150.6	2,712,126	2,201.393	10.5
PJ_21_14	570,135.7	2,712,111	2,201.251	7.5
PJ_21_15	570,121.2	2,712,096	2,201.22	4.5
PJ_21_16	570,056.6	2,712,408	2,246.16	31.5
PJ_21_17	569,697.1	2,712,477	2,255.046	37.5
PJ_21_18	569,738.1	2,712,496	2,254.793	36
PJ_21_19	569,724.6	2,712,462	2,255.198	40.5
PJ_21_20	569,705.7	2,712,437	2,255.774	37.5
PJ_21_21	569,711.8	2,712,396	2,256.279	33
PJ_21_22	569,730.5	2,712,421	2,256.084	39
PJ_21_23	569,748.7	2,712,444	2,255.568	42
PJ_21_24	569,766.9	2,712,469	2,255.074	45
PJ_21_25	569,784.5	2,712,493	2,254.83	36
PJ_21_26	569,802.4	2,712,516	2,254.635	30
PJ_21_27	569,826.5	2,712,499	2,254.925	33
PJ_21_28	569,807.8	2,712,475	2,255.321	40.5
PJ_21_29	569,789.9	2,712,451	2,255.44	46.5
PJ_21_30	569,771.5	2,712,427	2,255.714	46.5

table continues…

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)
PJ_21_31	569,753.8	2,712,402	2,256.283	43.5
PJ_21_32	569,735.8	2,712,378	2,256.785	34.5
PJ_21_33	569,760.6	2,712,360	2,257.194	36
PJ_21_34	569,778.2	2,712,384	2,257.151	43.5
PJ_21_35	569,796.7	2,712,408	2,256.366	48
PJ_21_36	569,814	2,712,433	2,256.093	49.5
PJ_21_37	569,832.4	2,712,456	2,255.736	49.5
PJ_21_38	569,850.8	2,712,480	2,255.299	36
PJ_21_39	569,874.2	2,712,462	2,255.746	39
PJ_21_40	569,855.6	2,712,438	2,256.17	46.5
PJ_21_41	569,838.6	2,712,414	2,256.795	46.5
PJ_21_42	569,892	2,712,486	2,255.524	36
PJ_21_43	569,916.6	2,712,469	2,255.862	36
PJ_21_44	569,898.1	2,712,445	2,256.192	43.5
PJ_21_45	569,879.9	2,712,420	2,256.584	54
PJ_21_46	569,868.7	2,712,405	2,257.29	55.5
PJ_21_47	569,904.4	2,712,401	2,256.867	55.5
PJ_21_48	569,921.9	2,712,427	2,256.276	45
PJ_21_49	569,940.1	2,712,450	2,256.091	39
PJ_21_50	569,953.7	2,712,469	2,256.547	31.5
PJ_21_51	569,981.9	2,712,457	2,256.61	37.5
PJ_21_52	569,964.3	2,712,433	2,256.447	43.5
PJ_21_53	569,946.3	2,712,409	2,256.64	54
PJ_21_54	569,928	2,712,384	2,256.958	57
PJ_21_55	570,005.8	2,712,437	2,257.24	46.5
PJ_21_56	569,988.1	2,712,414	2,256.999	40.5
PJ_21_57	570,008.3	2,712,401	2,257.586	51
PJ_21_58	569,989.7	2,712,379	2,257.361	52.5
PJ_21_59	570,047	2,712,380	2,245.685	36
PJ_21_60	570,029.4	2,712,356	2,245.583	46.5
PJ_21_61	570,011.3	2,712,332	2,245.364	48
PJ_21_62	569,992.7	2,712,310	2,245.326	48
PJ_21_63	569,970.8	2,712,290	2,245.039	48
PJ_21_64	569,789.6	2,712,338	2,249.064	30

table continues…

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)
PJ_21_65	569,807.2	2,712,363	2,248.837	37.5
PJ_21_66	569,829.7	2,712,346	2,248.839	30
PJ_21_67	569,811.6	2,712,321	2,249.029	33
PJ_21_68	569,795.1	2,712,301	2,249.143	39
PJ_21_69	569,817.6	2,712,280	2,249.11	33
PJ_21_70	569,836.6	2,712,304	2,249.298	36
PJ_21_71	569,854.6	2,712,328	2,248.96	40.5
PJ_21_72	569,872.3	2,712,353	2,248.951	45
PJ_21_73	569,907.4	2,712,351	2,249.358	46.5
PJ_21_74	569,860.4	2,712,287	2,249.521	45
PJ_21_75	569,843.2	2,712,264	2,249.378	45
PJ_21_76	569,890.2	2,712,327	2,249.347	34.5
PJ_21_77	569,878	2,712,311	2,249.382	33
PJ_21_78	569,865.7	2,712,244	2,249.788	33
PJ_21_79	569,884.6	2,712,270	2,250.61	36
PJ_21_80	569,899.5	2,712,295	2,250.129	45
PJ_21_81	569,917.3	2,712,318	2,249.721	48
PJ_21_82	569,871.8	2,712,229	2,249.96	33
PJ_21_83	570,108.1	2,712,442	2,232.687	10.5
PJ_21_84	570,113.1	2,712,414	2,233.283	16.5
PJ_21_85	570,090.3	2,712,435	2,233.067	11
PJ_21_86	570,095.1	2,712,390	2,233.369	21
PJ_21_87	570,078.1	2,712,366	2,233.964	30
PJ_21_88	570,058.7	2,712,342	2,234.621	34.5
PJ_21_89	570,040.4	2,712,319	2,235	37.5
PJ_21_90	570,021.4	2,712,295	2,234.771	40.5
PJ_21_91	570,001.5	2,712,271	2,233.875	39
PJ_21_92	569,981.3	2,712,250	2,233.09	36
PJ_21_93	569,958.2	2,712,232	2,232.993	30
PJ_21_94	569,937.2	2,712,211	2,233.44	25.5
PJ_21_95	569,928.7	2,712,183	2,233.724	21
PJ_21_96	569,908.7	2,712,162	2,233.469	19.5
PJ_21_97	569,882.6	2,712,155	2,233.002	12
PJ_21_98	569,854.4	2,712,158	2,233.569	4

table continues…

Hole ID	Easting (m)	Northing (m)	Elevation (m)	Depth (m)
PJ_21_99	569,852.5	2,712,127	2,232.007	1.5
PJ_21_100	570,098.5	2,712,467	2,230.343	4
PJ_21_101	570,123.9	2,712,450	2,227.323	4
PJ_21_102	570,152.4	2,712,460	2,225.446	2
PJ_21_103	570,171.3	2,712,437	2,220.975	4.5
PJ_21_104	570,044.4	2,712,267	2,224.492	31.5
PJ_21_105	570,025.6	2,712,242	2,223.743	30
PJ_21_106	570,004.7	2,712,221	2,223.365	25.5
PJ_21_107	569,982	2,712,201	2,222.582	21
PJ_21_108	569,962.4	2,712,181	2,222.332	13.5
PJ_21_109	569,976.5	2,712,173	2,223.239	15
PJ_21_110	569,999.4	2,712,193	2,223.081	22.5
PJ_22_01	569,843.9	2,712,186	2,236.169	7.5
PJ_22_02	569,828.5	2,712,211	2,236.981	9
PJ_22_03	569,809.2	2,712,235	2,237.677	10.5
PJ_22_04	569,788.6	2,712,259	2,238.641	10.5
PJ_22_05	569,767.1	2,712,281	2,240.464	12
PJ_22_06	569,748.1	2,712,304	2,241.355	12
PJ_22_07	569,726.6	2,712,325	2,241.748	12
PJ_22_08	569,703.9	2,712,344	2,241.919	12
PJ_22_09	569,677.7	2,712,360	2,242.424	12
PJ_22_10	569,649.5	2,712,370	2,242.744	12
PJ_22_11	569,620.2	2,712,371	2,243.229	12
PJ_22_12	569,591	2,712,365	2,244.154	10.5
PJ_22_13	569,562.3	2,712,362	2,245.195	6
PJ_22_14	570,054	2,712,432	2,245.257	21
PJ_22_14_A	570,052.9	2,712,436	2,245.224	24
PJ_22_15	570,045.8	2,712,464	2,245.205	18
PJ_22_16	570,029.6	2,712,489	2,243.345	7.5

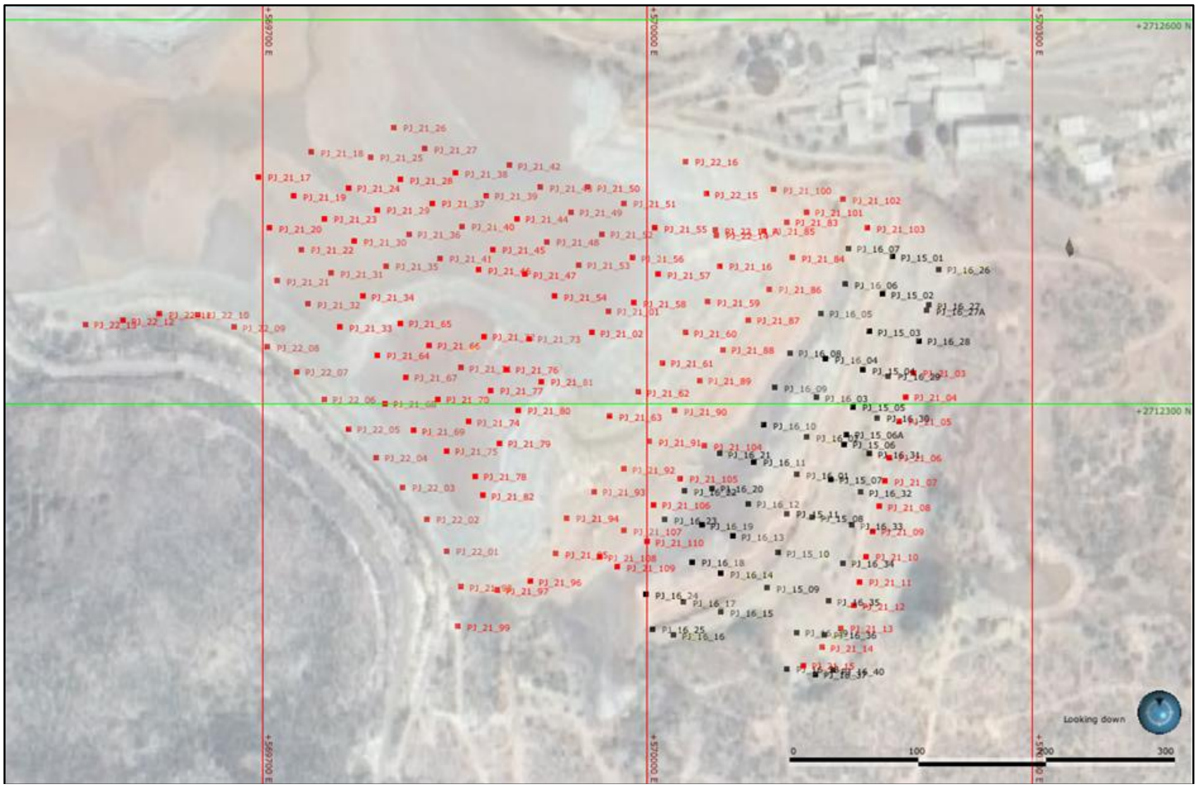


Figure 10-3: Location of Drillholes Completed from 2021 to 2022 on the Oxide Tailings (Tetra Tech 2017)

10.1.3 Specific Gravity Results

Bulk density samples were analyzed from all 2006 to 2022 drilling programs on the Avino and San Gonzalo Veins. Analytical procedures are discussed in Section 11.1.7. Table 10-3 summarizes the results of these specific gravity measurements.

Table 10-3: Avino and San Gonzalo Density Data Summary

Domain	Number	Minimum	Maximum	Mean	Variance	Coefficient of Variation
Avino Vein System						
10 (Main)	40	2.53	3.00	2.71	0.02	0.05
20	42	2.43	2.90	2.68	0.01	0.03
Wall Rock	93	2.29	3.00	2.65	0.04	0.07
Combined	175	2.29	3.00	2.67	0.03	0.06
San Gonzalo Vein System						
10	50	2.40	3.00	2.64	0.03	0.07
20	2	2.73	2.78	2.76	0.00	0.01
Wall Rock	41	2.40	3.00	2.69	0.02	0.05
Combined	93	2.40	3.00	2.67	0.03	0.06

10.2 La Preciosa Area

The issuer has not drilled on the concessions since taking ownership.

During 2014, Coeur drilled 75 HQ diamond drillholes for a total of 11,437 m, with an average depth of 150 m, and with core recoveries of 85%. Drilling was done by Layne de Mexico S.A de C.V.

In addition to infill core drilling, an additional 2,244 m of core was drilled for the geotechnical investigation by Major. These core holes were logged and on completion of geotechnical work, the core was split, sampled, and assayed; however, the assay and geology data were not available in time for use in the resource model.

All drillholes, except for RC drillholes intended by Coeur for condemnation drilling, are diamond core holes of varying diameters, mainly HQ and some NQ diameter drillholes. Exploration and development drilling to delineate mineral resources has been performed in sequential campaigns by Luismin, Orko, PAS, and Coeur as summarized in Table 10-4 (excludes RC drilling because RC was not used in resource estimation).

Table 10-4: Drilling Summary

Company	Years	Area	Number of Drillholes	Meters of Drilling	Hole Number Prefixes
Luismin	1981, 1982, 1994	La Preciosa	8	1,630	BP
Orko	2006	Orito	7	2,326	BO
	2007	San Juan	8	3,554	SJ
	2005	La Preciosa	1	451	BC
	2006		6	1,910	BB
	2005–2008		366	144,126	BP05-BP08
PAS	2009–2010	La Preciosa	363	91,095	BP09-BP-10
Orko	2011–2012	La Preciosa	5	500	BP11-BP12
Coeur	2013–2014	La Preciosa	103	22,324	CLP14, KP14, KP13, DH13
Totals			**867**	**267,916**	

10.2.1 2017 Underground Channel Sampling

Channel sampling was carried out by Coeur in drifts on the Abundancia and Gloria Veins. 426 samples (482.2 m) were captured on the Abundancia Vein and 336 samples (380.5) on the Gloria Vein. The La Preciosa area had previously been drilled intensively in the vicinity of the underground development, allowing the channel sampling data to be compared with drill hole samples. The channel samples were statistically compared with diamond drill samples where both types were present within 10 m of each other within the relevant veins. The samples were composited to 1 m lengths and nearest neighbours (Gloria: 45 pairs, Abundancia: 35 pairs) were compared by means of scatterplots and quantile-quantile (Q-Q) plots to assess whether it was reasonable or not to consider them as a single population. The channel samples and closest drill samples within 10 m proximity are shown in Figure 14-12.

10.2.2 Drilling by Luismin

Of the seven Luismin drillholes in the Project database, two were drilled from underground workings and five from the surface. The primary targets were the Abundancia and La Gloria veins, which run semi-parallel to the north-northwest-striking Abundancia Ridge, at depths of 50 to 75 m below the primary underground workings on the 2065 m level (elevation). Lusmin drilled one additional drillhole 313 m deep in 1994 in the eastern vein breccia system, but data for this drillhole are not available. There are no available details on the Luismin drilling procedures, except that the drill core was either small-diameter BQ or AX size. The remaining half-core from these holes is stored in the original core boxes on site.

10.2.3 Drilling by Orko (2005 to 2008, 2011, 2012)

Orko began drilling in March 2005, ultimately completing 388 diamond drillholes totalling 152,368 m of core, spaced on roughly 100 m centres, with all but 16 of the holes targeting various veins. Orko used Major for all of its drilling using Longyear 44, 38A, and 38B core drills. Drill core diameters started at HQ-diameter, with reductions to NQ-diameter at around 260 m downhole. Between rod changes the drillers inserted a wooden “run” block in the core boxes marked with the downhole depth in both ft. and m. Downhole surveys were taken approximately every 50 m down the hole with a Reflex survey instrument. The results of these surveys indicated only moderate deviation in downhole azimuths and inclinations.

Drill core was collected on a daily basis from the drill rig by Orko technicians, who taped the boxes shut prior to transporting the core to the site core shed. Once at the shed, technicians cleaned the boxes and core, marked the boxes with the hole number, box number, and the depth intervals, and reconciled these data with the depths marked on the driller’s core run blocks.

After completion of each hole, a PVC pipe was placed in the hole collar and a concrete cap was poured around the collar PVC pipe, and a length of PVC pipe was left protruding above the concrete cap. The concrete cap was inscribed with the drillhole number, total hole depth, and the azimuth and inclination of the hole at the collar. An independent surveyor was contracted to survey the collar coordinates on a regular basis.

10.2.4 Drilling by PAS

PAS began drilling in June 2009, under the terms of PAS’s Option Agreement to acquire a joint venture interest in the Project from Orko, and PAS completed 331 diamond drillholes. The drilling focused on in-filling the 100 m centre grid previously completed by Orko. PAS’s drilling resulted in a spacing of 50 m on every other section (100 m apart) over an area approximately 800 m by 800 m. This selective tighter spaced drilling area is located in the northern part of the deposit. Additionally, infill drillholes were drilled on selected sections as well as on two 15- to 20-m close-spaced fences to assess the short-range continuity of geology and mineralization. Major was also used by PAS to do the drilling program, which resulted in similar drilling and downhole surveying procedures as Orko, although greater capacity drill rigs were employed which resulted in fewer NQ-diameter drillholes. Beginning in early 2010, selected drillholes were surveyed using a Reflex ACT/QPQ orientation tool to obtain oriented drill core for geotechnical purposes. The drillhole collar monuments and the survey of collar coordinates followed the same procedures established earlier by Orko.

10.2.5 Drilling by Coeur

Between January and April 2014 Coeur drilled a total of 75 HQ core drill, 21 RC holes totalling 19,980 m, and 7 geotechnical core holes totalling 2,244 m. A majority of Coeur's drillholes were oriented west to east at varying dips, depending on the target vein orientation, to minimize the drillhole intersection angle with the vein. In general, the downhole length of the drill intersection approximates the true thickness of the vein, but this length can vary from hole to hole. Most of the 2013 drillholes were completed in the main deposit area, an area approximately 3,000 m north to south by 2000 m east to west.

RC drillholes were drilled on -45° to -60°, to -90° inclination with north-northeast and southwest-west azimuths. Infill drillholes were designed to increase the amount of Measured and Inferred material in the first three years of the mine plan (an area encompassing Abundancia Ridge). These drillholes were inclined from -40° to -85°, with azimuths from north-northeast to southwest-west.

Downhole surveying in core and RC drillholes was done from top to bottom approximately every 10 m downhole using a gyroscope survey instrument. Results of these surveys indicated only minor deviation in azimuth and inclination. Collars were surveyed with total station instrument using WGS 84 coordinate system.

Drill core was collected on daily from the drill rig by Coeur technicians, who taped the boxes shut prior to transporting the core to the onsite core shed. In the core shed technicians cleaned the boxes and core, marked the boxes with the drillhole number, box sequence number, depth intervals, and checked recorded depths against the depths marked on the driller's core run blocks. Colour digital photographs of each core box were taken before the core was split and sampled. Core was then laid out and logged, using paper logging forms, by project geologists, who also marked sampling intervals according to Coeur quality assurance (QA) / quality control (QC) sampling protocols. Briefly sampling criteria requires samples to be greater than 50 cm and less than 200 cm in length. Logging describes all common features, such as rock type, alteration, mineralization, faults, etc.

RC drill sampling was done on 5 ft. or 1.5 m intervals. Samples were collected after passing through a cyclone under both wet and dry conditions. Samples were placed in plastic bags when drilling dry material and in Micropore bags when the drilling was wet.

After completion of each hole, a PVC pipe was placed in the drillhole collar and a cement cap or monument was poured around the collar PVC pipe and inscribed with the drillhole number, total drillhole depth, and the azimuth and inclination of the hole at the collar. An independent surveyor was contracted to survey the coordinates of each collar on a regular basis.

10.2.6 Core Recovery and Rock Quality Designation

Only Orko drilling had drillhole core recovery and rock quality designation (RQD) values recorded in the acQuire database, these types of data were not recorded by PAS or Luismin. Although Coeur recorded core recovery and RQD measurements, at the time of this report was written those data had not been entered into the acQuire database, thus no analysis was done.

Orko's recovery values are reasonable with a mean core recovery of 94.5% and a mean RQD of 54.3%. For both silver and gold there is a decrease in Ag and Au grade with increasing core recovery where recovery is >20%, which suggests that there is a small sampling bias with loss of material in higher grade zones. Grades decrease slightly with increasing core recovery, however maximum grades increase with increasing core recovery because of the greater number of intervals with better core recovery. It is important to note that the data are not normalized for the number of recovery measurements. Because the highest Ag and Au grades are typically found in quartz veins, and core recovery in quartz veins tends to be lower because the veins are fractured, the grade-recovery relationship is expected. Given the small number of recovery measurements in the range of 0-40 percent recovery, and lower maximum grades in the 0-40% range, the impact of core recovery on the resource estimate is insignificant.

10.2.7 QP Opinion

The QP is satisfied that the amount and quality of drilling is sufficient to support the MRE.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Avino Mine

11.1.1 Drilling and Trenching of Oxide Tailings, 1990 to 1991

The oxide tailings were sampled prior to the institution of NI 43-101 and associated QA/QC requirements, and as such, no QA/QC measures were utilized during the 1990–1991 program. As a result, the resource estimate for the oxide tailings in Section 14.1.3 is all classified as Inferred. 28 holes were drilled, and six trenches were completed, from which a total of 461 samples were collected for assaying. The analyses were completed in the on-site laboratory, which is described in Section 11.1.9 and was visited during the site visit, as summarized in Section 12.1.2.

Avino's current on-site, non-certified laboratory facility consists of sample preparation, crushing and pulverizing, a fire assay, and an atomic absorption (AA) section. However, the procedures and facilities used from 1990 to 1991 may be different from the current sample analysis procedures. Because of the uncertainty associated with these analyses, two separate verification exercises have been completed. Slim (2005d) collected several samples from the oxide tailings, and the results of this verification are discussed in Section 11.1.2. In 2012, Mr. M.F. O'Brien, QP, collected numerous verification samples from the oxide tailings, and these results are discussed in Section 12.1.3.2.

11.1.2 Tailings Investigations (Test Pits in Oxide Tailings), 2004

The sampling method and approach adopted by Slim (2005d) on the test pits in the oxide tailings incorporated the following steps:

1. A backhoe was used to excavate sample pits to a depth of 4 m. Hand samples were taken at 1 m vertical increments from the sidewalls of each pit.

2. The sample mass collected from each sampling point generally amounted to between 2 kg and 5 kg.

3. The sampling program was ostensibly based on the 1990 CMMA sampling program. 14 sample pits were excavated to a depth of 4 m and generated 86 samples.

The samples were air-freighted to PRA laboratories in Vancouver, British Columbia, from Durango, Mexico. The samples had been initially bagged and sealed with identification tags attached. The samples were allotted new identification numbers and were subsequently un-bagged and dried. The dry samples were individually mixed and blended and then split into four one-quarter fractions as directed by Slim (2005d). One fraction was used to determine the head grade assay, while another quarter was used to create composite samples used for the subsequent metallurgical test work program. Instructions were followed with the compositing of the samples and the test work program.

Excess sample was archived for future test work or analyses. For analytical techniques employed during the test work program, the standard fire assay (with AA spectrophotometric finish) was initially used for the silver analyses.

However, this method is not very accurate for silver values of less than 100 g/t. Subsequently, the inductively coupled plasma mass spectroscopy (ICP-MS) method, which uses multi-acid digestion, was used for silver. This method also resulted in analyses being obtained for other elements of interest (e.g., copper, zinc, lead). The standard fire assay method was used for gold analyses. Cyanide and lime concentrations were measured using standard titrimetric methods. Total sulphur was measured using a standard Leco furnace, and sulphide sulphur assays were measured using the standard wet chemical gravimetric analysis (Slim 2005d).

The PRA laboratories (part of Inspectorate Laboratories [Inspectorate]) in Nevada and British Columbia are International Organization for Standardization (ISO) 9001:2008 certified, full-service laboratories that are independent of Avino. The QP did not independently verify nor compare the results of the sampling program.

11.1.3 Drilling Program, San Gonzalo, 2007 to Present

For the drilling programs at San Gonzalo, the core is sawed at Avino's core storage facility at the secure mine site. Samples of vein material, usually from a few cm to 1.5 m, are placed and sealed in plastic bags, which are collected by personnel from SGS Laboratories in Durango at the mine site facilities. Samples are prepared in Durango, and pulps are sent to the Inspectorate facility in Sparks, Nevada, for analysis. Since 2016, all drill core samples have been sent to SGS Durango for sample preparation and assaying. A switch was made for faster turnaround times.

Sample preparation in Durango involves the initial drying of the entire sample. Two-stage crushing is used to create a product which is at least 80% minus 10 mesh. A Jones riffle splitter is then used to separate a nominal 300 g portion of the sample. This 300 g sub-sample is then pulverized to more than 90% passing a 150-mesh screen. Inspectorate Laboratories states that they use sterile sand to clean the pulverizer between samples (Gunning 2009).

Gold analyses are by 30 g fire assay with an AA finish. Silver, zinc, and lead are analyzed as part of a multi-element inductively coupled argon plasma package using four-acid digestion with over-limit results for silver being reanalyzed with assay procedures using fire assay and gravimetric. Avino employs a rigorous QC program that includes standardized material, blank reference materials (blanks), and core duplicates. However, for the 2007 program, Avino did not perform any independent QA/QC and relied on the internal QA/QC procedures completed by the laboratories (Gunning 2009).

Inspectorate Laboratories in Nevada and British Columbia are ISO 9001:2008 certified, full-service laboratories that are independent of Avino.

Avino used a series of standard reference materials (SRMs), blanks, and duplicates as part of their QA/QC program during the analysis of assays from San Gonzalo Vein drillholes. The QP compiled and reviewed these results in Section 12.1.1.4.

11.1.4 Drilling Programs, ET Zone of the Avino Vein, 2006 to Present

Sample lengths of NQ drill core were diamond sawed into halves by mine staff and transported in sealed and labelled bags to ISGS in Durango for preparation into pulps and rejects. Gold and silver are analyzed by fire assay using aqua regia leach and AA finish. Other elements are reported from a 29-element ICP-MS package. Sample preparation and analysis and QA/QC procedures are described in Section 11.1.3.

Avino used a series of certified reference materials, blanks, and duplicates as part of their QA/QC program during the analysis of assays from Avino Vein drillholes. The QP compiled and reviewed these results in Section 12.1.1.4.

11.1.5 Avino Laboratory

The Avino laboratory has fire assay, AA, and sieving analysis equipment and has been recently upgraded with new AA equipment. A high standard of neatness and cleanliness is being maintained to reduce the risk of contamination. The laboratory was reviewed by Mr. O'Brien during site visits in 2012 and 2016.

11.1.6 SGS Laboratory, Durango

The SGS Laboratory in Durango processes the current drill core samples from Avino. It is a well-appointed modern facility which was visited by the QP in 2012.

11.1.7 Review of Drillhole Quality Assurance/Quality Control Samples

QA/QC samples continued to be submitted in the sample stream from 2021 to 2022 during the drilling programs on the Property. 143 standards, 275 duplicates and 57 blank samples were included in the 5,117 samples assayed. The QA/QC submission rate is 9.28%.

11.1.7.1 Standards for Exploration Drilling, 2021 to 2022

Nine different reference standards, bracketing the expected ranges of grades for gold, silver, and copper, were analyzed during 2021/2022. Six standard reference materials CDN-ME-1405, 1406, 1414, 1603, 1705, and 1709 were submitted during the period.

Performance graphs showing the assay results compared to the reference standards and relative to the three standard deviation acceptability limits for each reference material and gold, silver, and copper are shown in Figure 11-1 to Figure 11-18, inclusive.

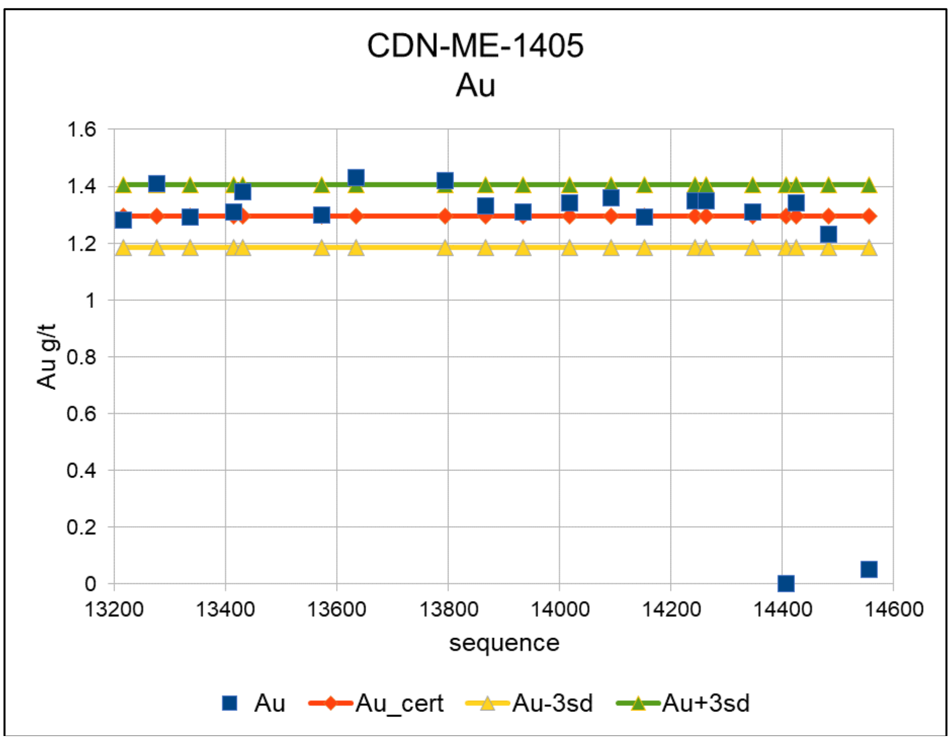


Figure 11-1: Reference Material CDN-ME-1405 Au Performance

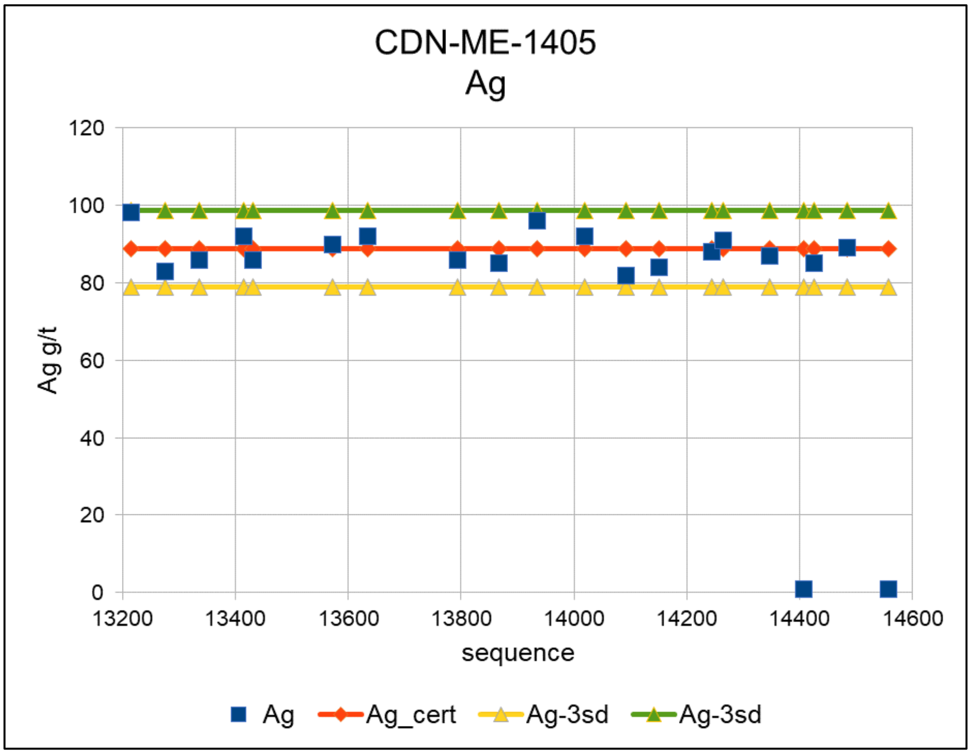


Figure 11-2: Reference Material CDN-ME-1405 Ag Performance

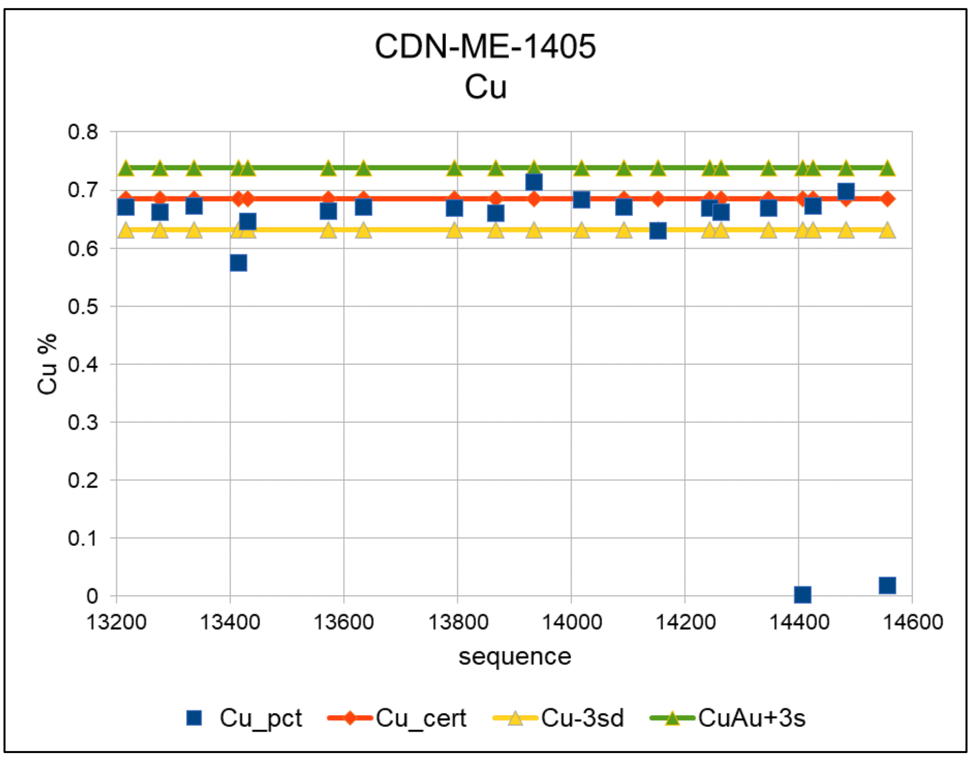


Figure 11-3: Reference Material CDN-ME-1405 Cu Performance

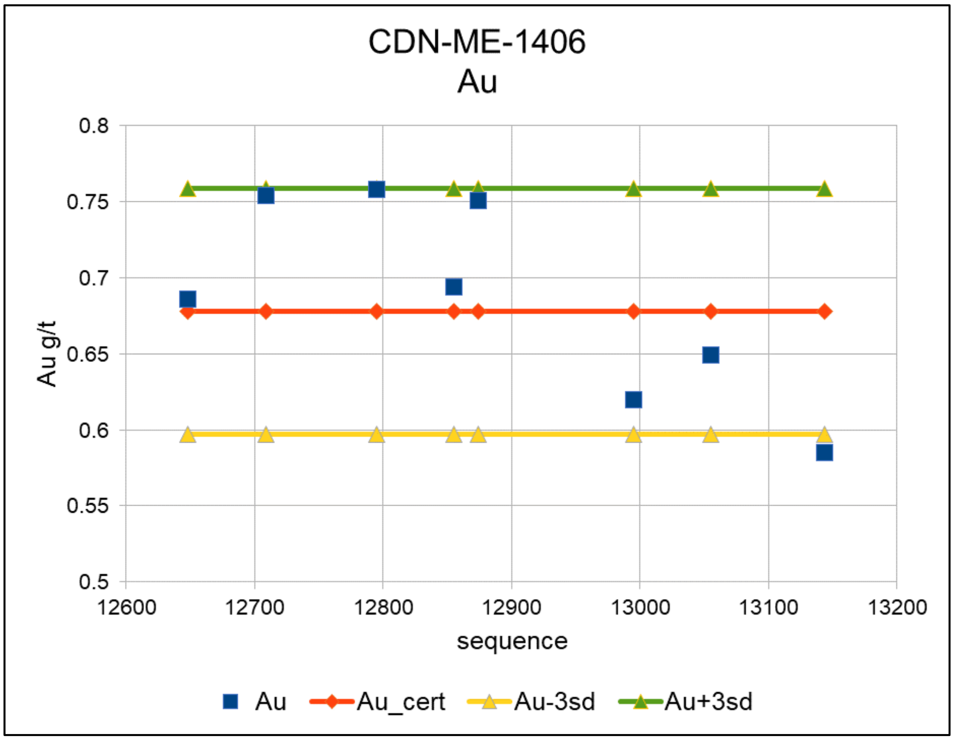


Figure 11-4: Reference Material CDN-ME-1406 Au Performance

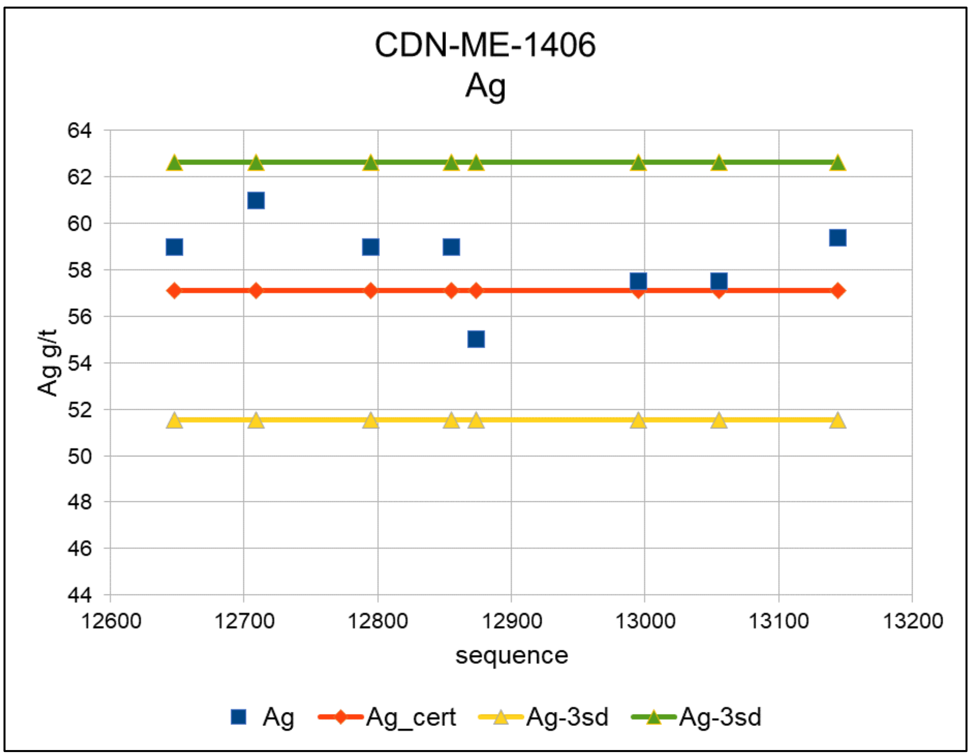


Figure 11-5: Reference Material CDN-ME-1406 Ag Performance

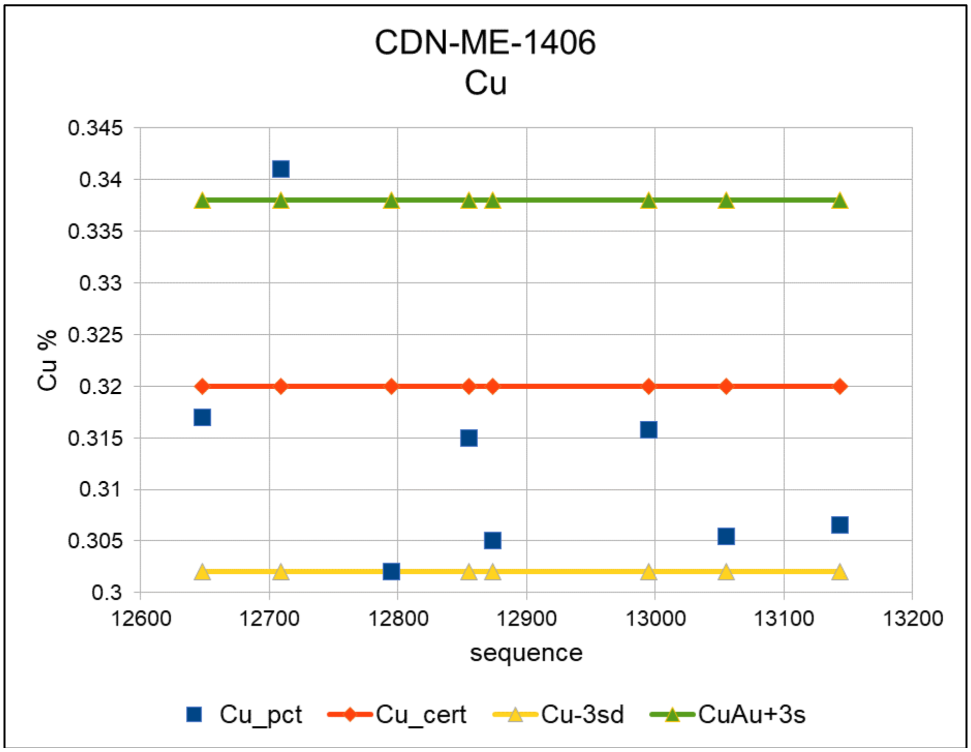


Figure 11-6: Reference Material CDN-ME-1406 Cu Performance

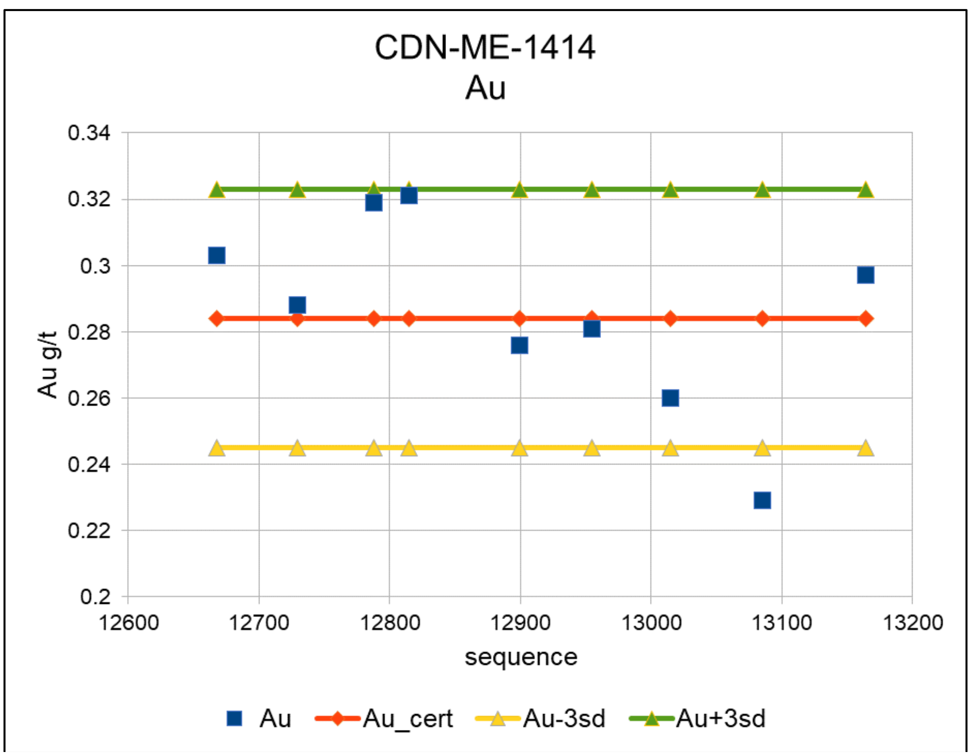


Figure 11-7: Reference Material CDN-ME-1414 Au Performance

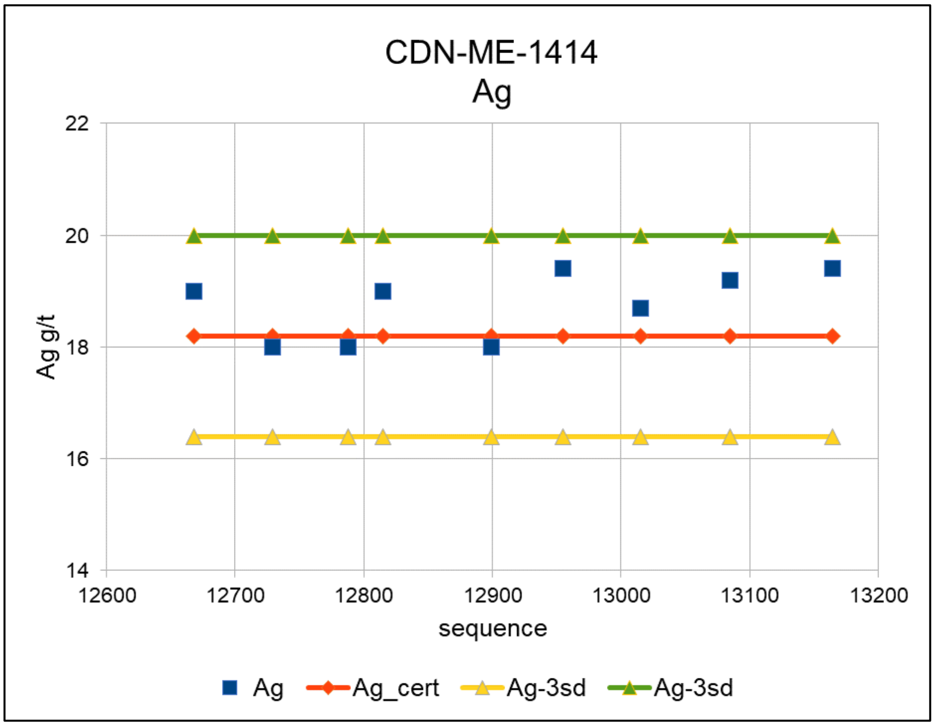


Figure 11-8: Reference Material CDN-ME-1414 Ag Performance

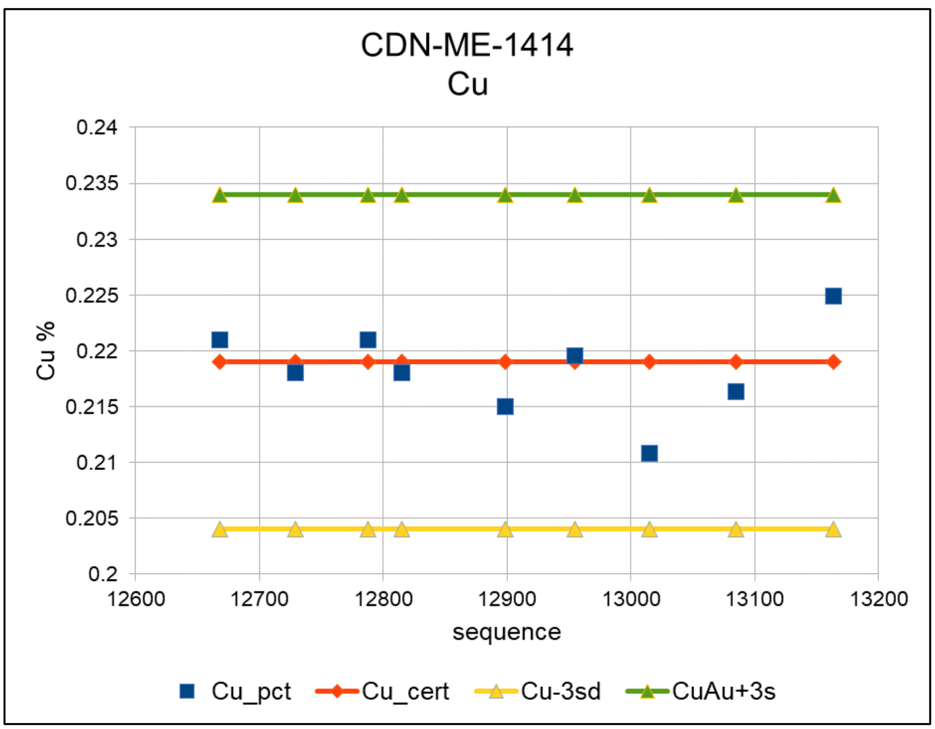


Figure 11-9: Reference Material CDN-ME-1414 Cu Performance

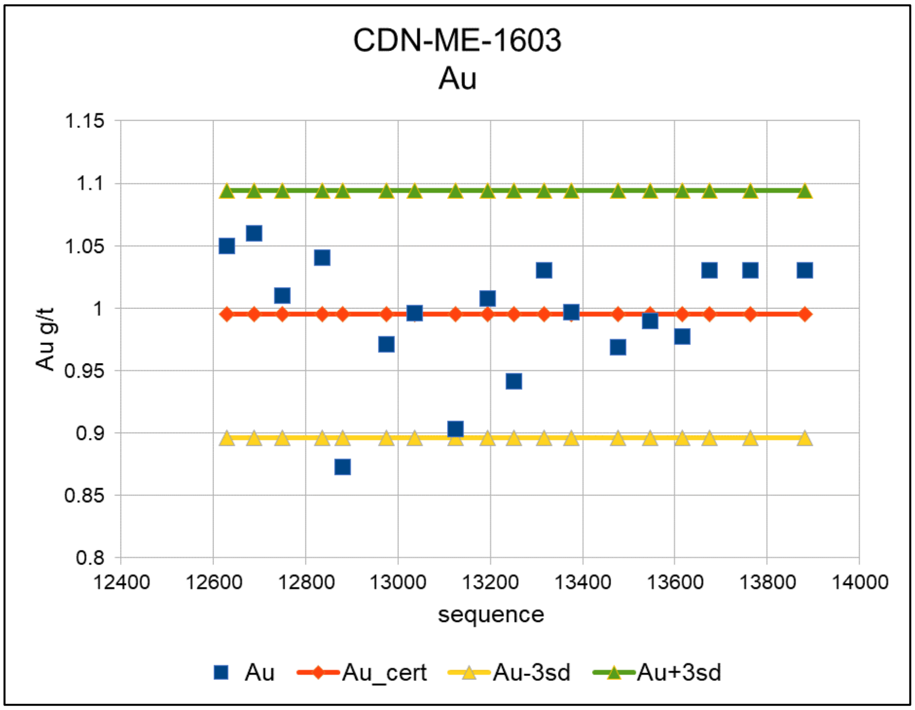


Figure 11-10: Reference Material CDN-ME-1603 Au Performance

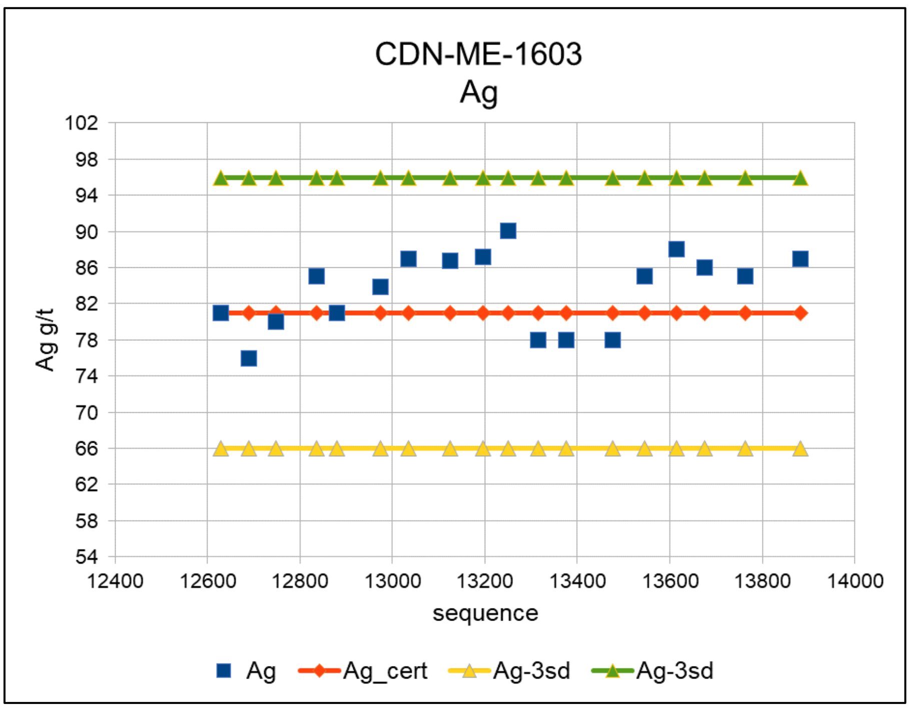


Figure 11-11: Reference Material CDN-ME-1603 Ag Performance

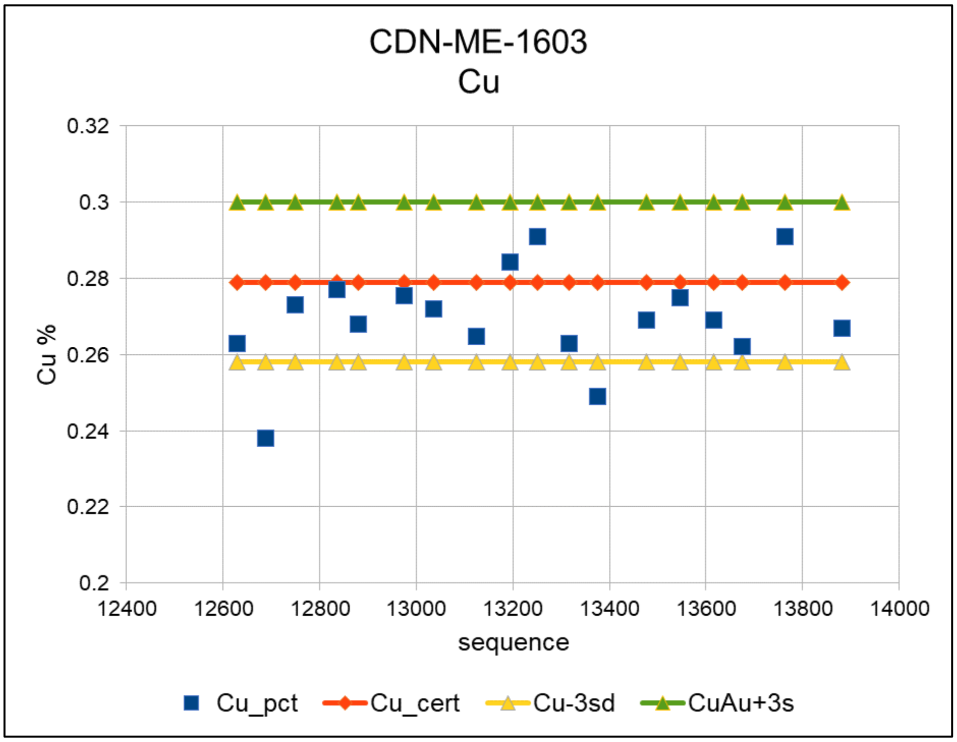


Figure 11-12: Reference Material CDN-ME-1603 Cu Performance

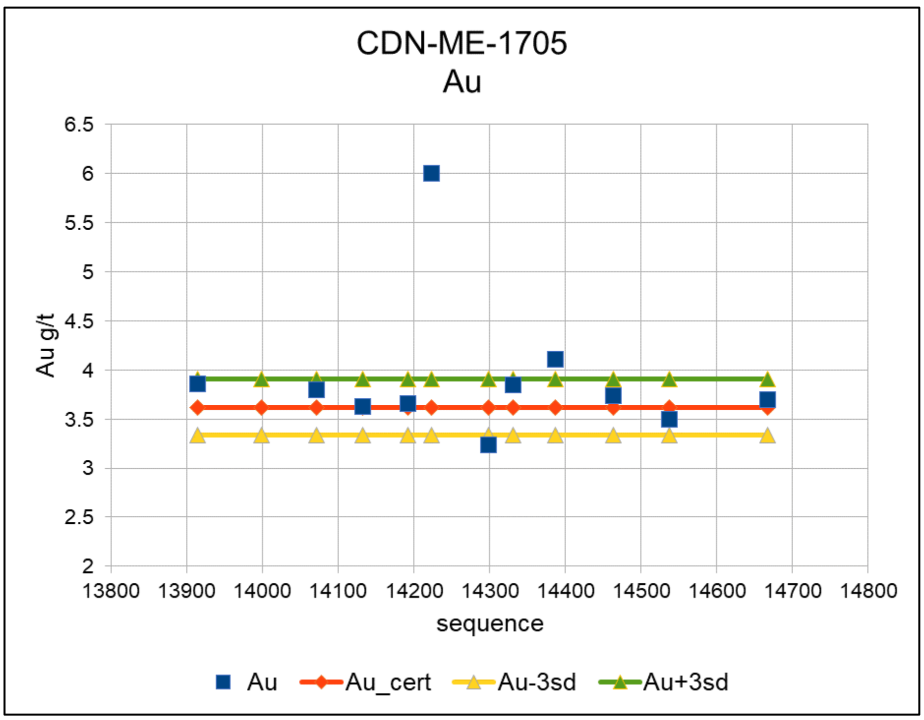


Figure 11-13: Reference Material CDN-ME-1705 Au Performance

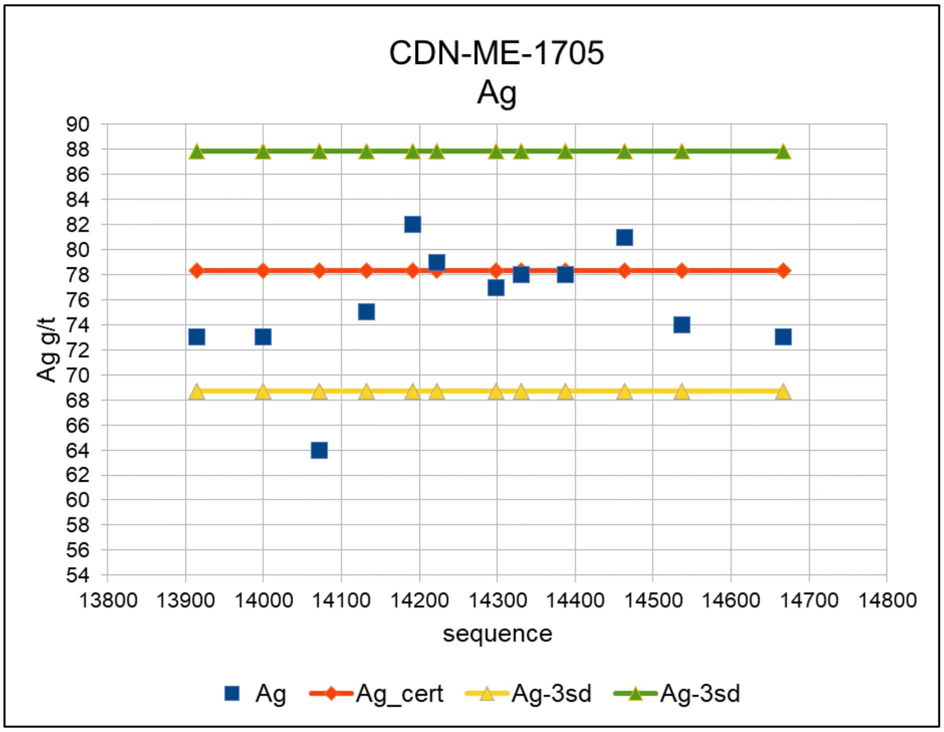


Figure 11-14: Reference Material CDN-ME-1705 Ag Performance

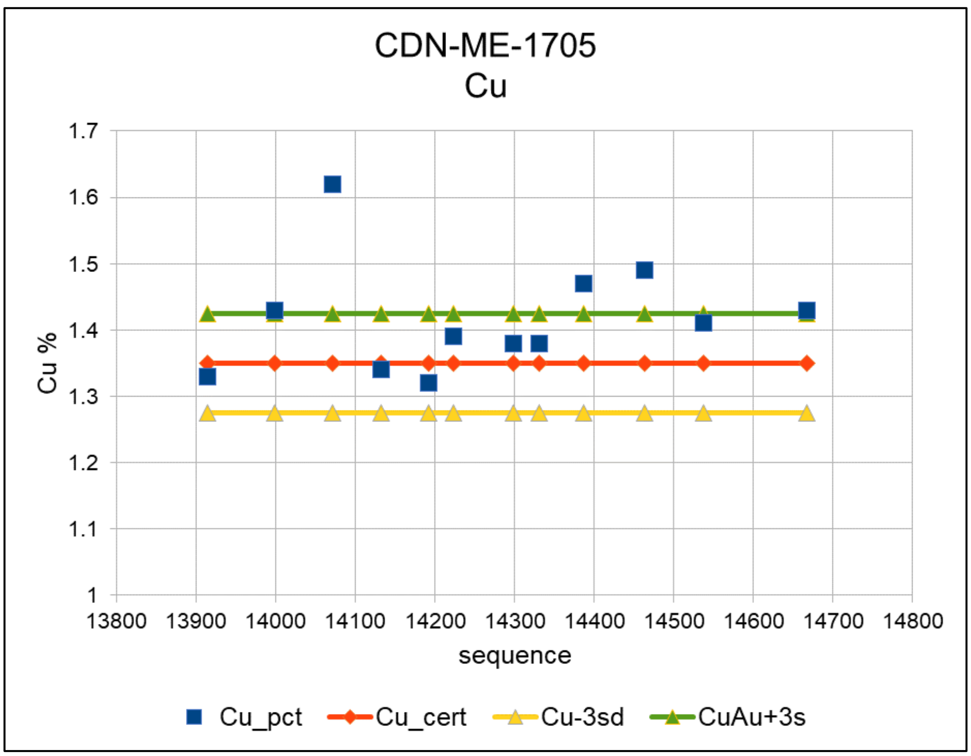


Figure 11-15: Reference Material CDN-ME-1705 Cu Performance

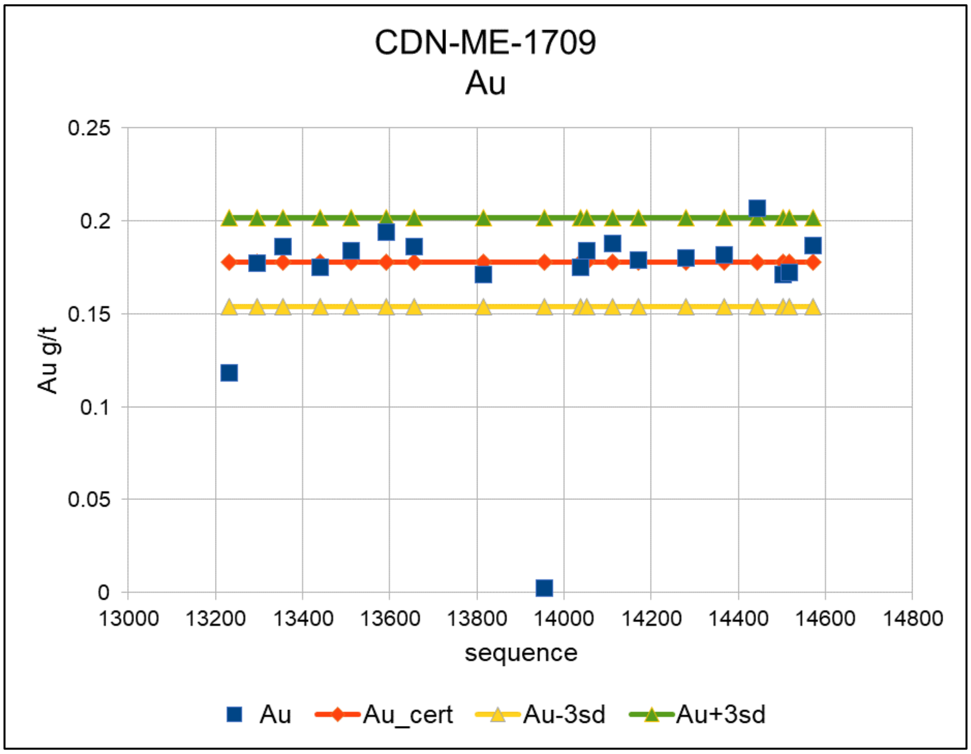


Figure 11-16: Reference Material CDN-ME-1709 Au Performance

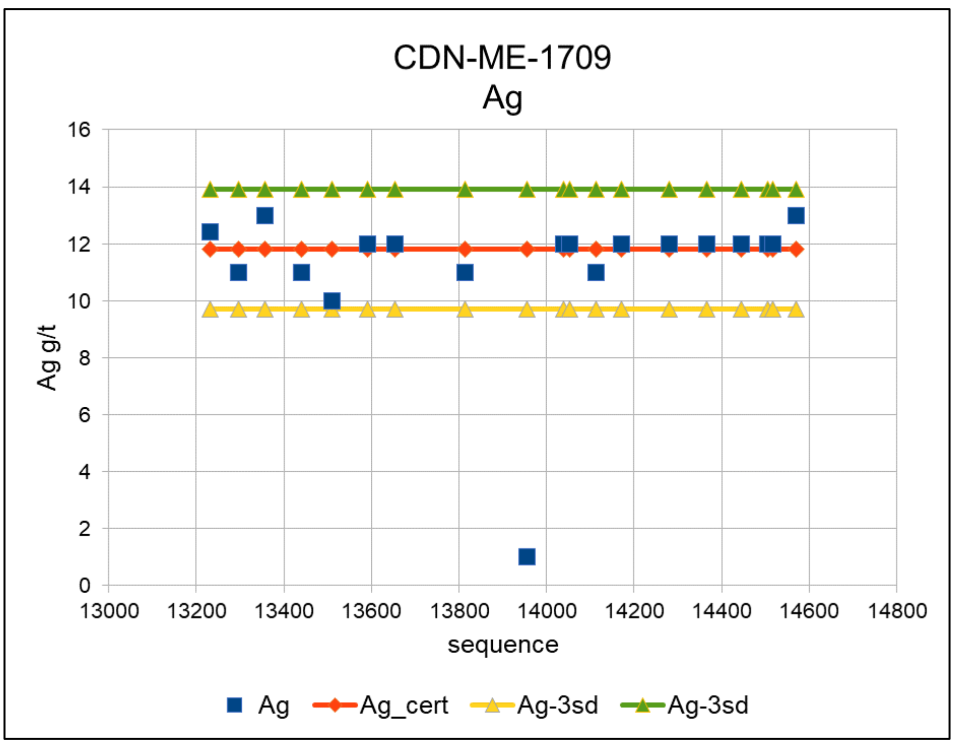


Figure 11-17: Reference Material CDN-ME-1709 Ag Performance

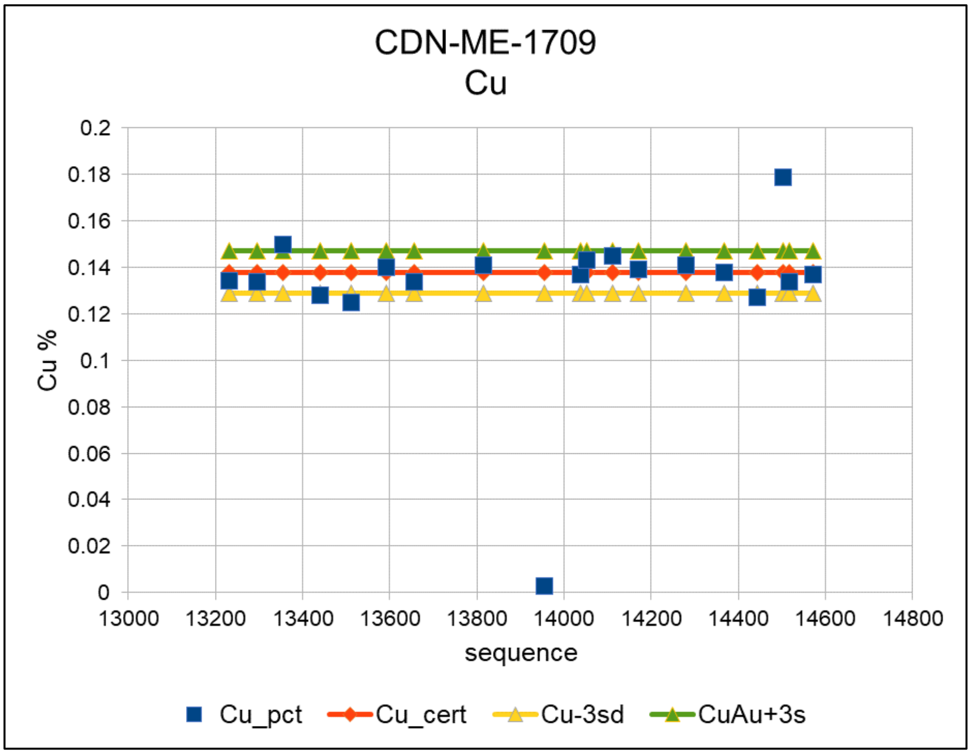


Figure 11-18: Reference Material CDN-ME-1709 Cu Performance

11.1.8 Duplicate Assays

Laboratory crushed duplicate results for gold, silver, and copper are summarized in Figure 11-19 to Figure 11-21, inclusive. Power fit curves have been generated to measure the correlation. The graphs for the three metals show no significant bias between originals and duplicates.

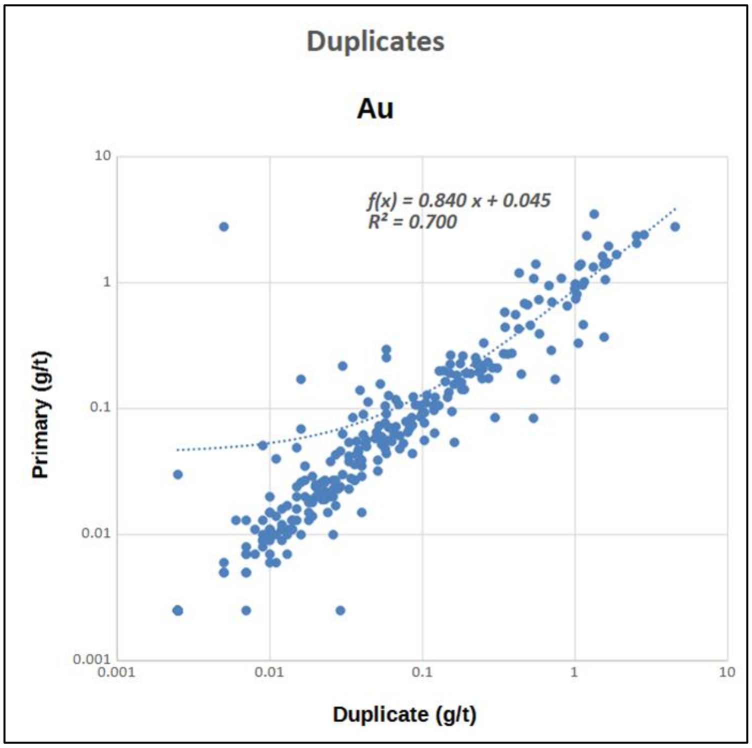


Figure 11-19: Au Duplicates

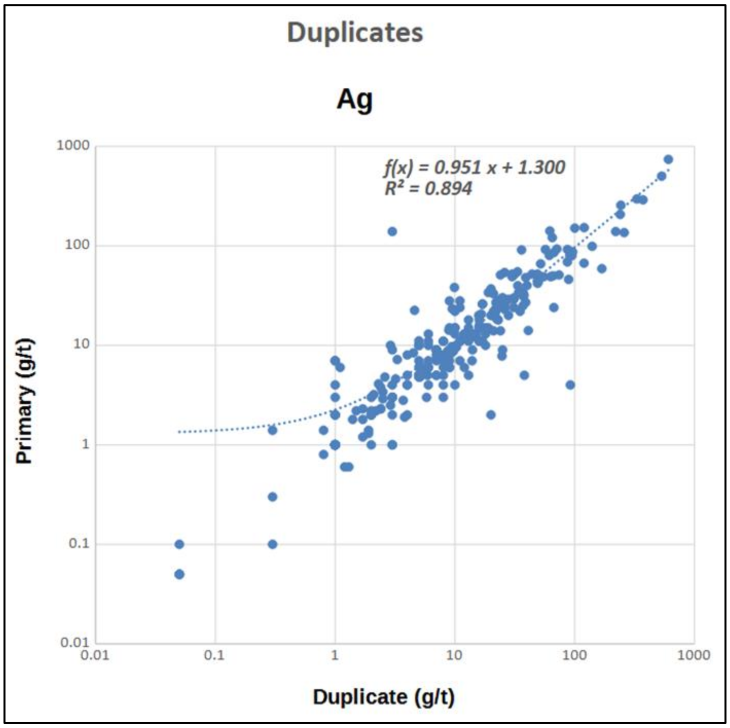


Figure 11-20: Ag Duplicates

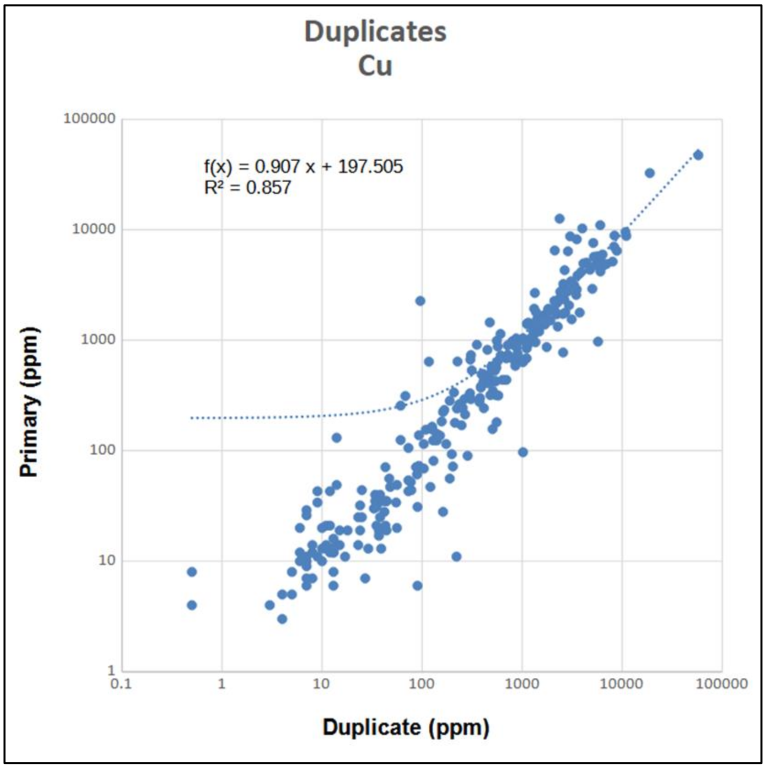


Figure 11-21: Cu Duplicates

11.1.9 Blanks

Analysis of blank submissions of barren cement-based material resulted in the expected negligible Au assays (see Figure 11-22) except for one isolated submission, which is likely to reflect a ticket swap rather than contamination.

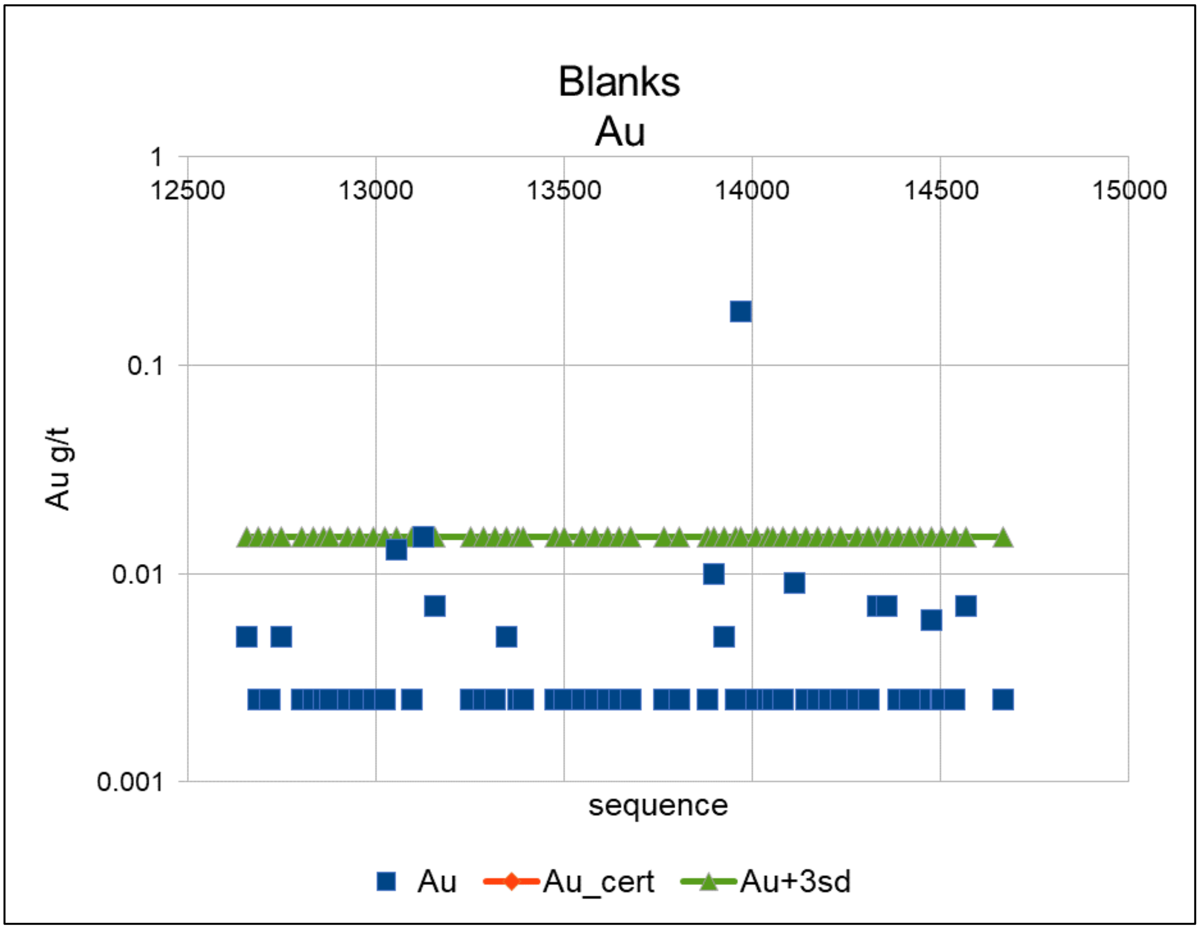


Figure 11-22: Blank Submission Assay Results

11.1.10 Bulk Density and Specific Gravity Samples

Avino completed specific gravity measurements on drillcore. All measurements were completed by Avino staff on the mine site. Two different methods were employed to obtain these specific gravity values: calliper volume calculation and water displacement (WD). The procedures followed for each method are summarized in the following sections. Density measurements are summarized in Table 11-1.

Table 11-1: Density Data Summary

Zone	Rock Code	Count	Length	Mean
Avino (ET)	AND	93	9.5	2.60
Avino (ET)	BX	87	9.0	2.68
Avino (ET)	INT	50	5.2	2.60
Avino (ET)	ROC	61	6.1	2.60
Avino (ET)	STKWK	27	2.8	2.68
Avino (ET)	VN	35	3.3	2.74

table continues…

Zone	Rock Code	Count	Length	Mean
Guadalupe	AND	31	3.1	2.59
Guadalupe	BX	4	0.4	2.90
Guadalupe	INT	1	0.1	2.58
Guadalupe	VN	13	1.3	2.83
La Potosina	AND	10	0.6	2.62
La Potosina	ROC	52	5.2	2.62
La Potosina	VN	33	3.1	2.69
San Gonzalo	AND	91	9.0	2.60
San Gonzalo	INT	30	3.0	2.56
San Gonzalo	STKWK	1	0.1	2.57
San Gonzalo	VN	18	1.8	2.48
Total	**--**	**637**	**63.7**	**2.63**

11.1.10.1 Calliper Volume Calculation Method

The calliper volume calculation method of determining the specific gravity of drillcore samples involved the following procedures, based on the methodology outlined by Lipton (2001):

- Each measurement involves pieces of a whole core with the ends neatly cut perpendicular to the core axis.
- The core diameter is determined using a pair of vernier callipers, and the diameter should be measured at several points along the core length and averaged.
- The core length is measured using a tape measure.
- The mass is determined by weighing the core; weighing should be completed once the core is dried.
- The dry bulk density is calculated by the equation below.

$$Dry\ Bulk\ Density = \frac{Mass}{Volume}\ , Volume = \pi\left(\frac{Average\ Core\ Diameter}{2}\right)^2 x\ Core\ Length$$

11.1.10.2 Water Displacement Method

The WD method of determining the specific gravity of drill core samples involved the following procedures based on Archimedes' Principle:

- The mass is determined by weighing the core; weighing should be completed once the core is dried.
- A graduated cylinder of an appropriate size to completely submerse the core is used to determine the volume. The volume of water in the graduated cylinder is measured prior to submersing the core.
- The core is then submerged in water in the graduated cylinder, and the total volume is measured.
- The difference in the volume of water before and after sample submersion is the volume of the sample.

11.1.11 QP Opinion

The QP is unaware of any drilling, sampling, or recovery factors affecting the reliability of the samples. It is the QP's opinion that the sample preparation, security, and analytical procedures followed by Avino are fit for the purpose of this Technical Report.

11.2 La Preciosa Area

11.2.1 Sample Collection Methods

11.2.1.1 Luismin (1981, 1982, 1994)

No documents/reports are available that describe the sampling methods used by Luismin. Based on descriptions provided by visual inspection of Luismin core by previous QPs, the core was split using a conventional manual splitter and properly "marked and neatly stored". Luismin reportedly collected a total of only 130 samples for assay, with variable sample lengths that ranged from 0.5 to 2.0 m. The breaks between samples reportedly respected geologic features.

11.2.1.2 Orko (2003–2008, 2011, 2012)

Orko technicians transported core from the drill rigs daily to the core logging facility, where it was cleaned and the core boxes marked with the hole number, box number, and from-to depth intervals. Each box of core was then photographed and moved to a rack for examination by a geologist who logged lithology, structure, alteration, mineralization type, intensity, and sulphide percentage and oxidation and assigned codes for rock types, structures, and veins. Logging was done manually on paper logging forms. Following geologic logging, geotechnical data including core recovery, was recorded. After logging was complete, the geologist marked the sample intervals on the core and on the core box dividers with a permanent marker, along with a cutting line along the longitudinal axis of the core and recorded the sample interval depths and corresponding sample numbers on the geological log. The core was then sawn in half along the cut line by an Orko technician using a water-cooled diamond saw, after which one half of the interval was placed in a plastic sample bag along with a sample tag. The remaining half was returned to the core boxes that then were placed on numbered racks in a large, secure, storage shed at the Project site.

To determine material density, a single piece of the sampled core was removed from each sample sack, allowed to air-dry, and then dry weighed for measurement of specific gravity. Once measured, the core piece was returned to the appropriate sample bag and the whole sample was placed in a rice sack for transport to the Inspectorate de Mexico sample preparation facility in the city of Durango. Prior to transport, each rice sack was weighed and the total weight recorded. All samples were in the possession of Orko personnel from the diamond drill rigs to the Inspectorate lab.

11.2.1.3 PAS (2008–2010)

PAS followed the same drillhole logging and sampling procedures and protocols developed by Orko, beginning with PAS drillhole BP10-458 onwards. The geologists determined the diamond core sample intervals and marked the positions of the intervals on both the core and the core box dividers. The core was then cut along the cut line marked on the core by the geologists using a water cooled diamond bladed saw, and both halves were placed back in the core boxes for transport to the core sampling area. Sample bags and sample tags were labelled with the consecutive sample numbers assigned to the sample intervals, with numbers reserved for insertion of QA/QC samples. The pieces of half core to be assayed were then placed in the appropriate labelled sample bags along with the corresponding sample tag, and then the bags containing the individual samples were inserted in groups of ten into labelled rice sacks along with the labelled standard and blank QA/QC samples. The rice bags filled with samples were stored on site until transported by a PAS employee to the SGS de Mexico laboratory in the city of Durango, Mexico.

11.2.1.4 Coeur (2013–2014)

The Coeur development program consisted of RC and core drilling. The RC drilling program was conducted by two drill rigs contracted from Layne de Mexico. One geologist was assigned to each active drilling shift. Geologists were provided with a package of sample tags which indicated the sample identification and the interval. Sample tags were included inside each sample bag and a permanent marker was used to note the sample identification and interval on each sample bag. A geologic description was recorded on a paper log at the drill rig, including any additional notes on the drilling or sample. This log was later transferred to an electronic format. Coeur's company protocol for quality control (Coeur 2012) was applied throughout the sample collection process. All RC samples were collected by Coeur technicians, accompanied by a project geologist. Samples were collected at 1.5 m intervals in two 5 g buckets. The entire sample was weighed, with typical weights ranging from 100-125 kg. The sample was initially split in half using a single Jones-type splitter with one half of this split bagged for analysis at the commercial laboratory. The remaining sample is split once more, retaining 1/8 of the original sample, and bagged for storage in the project warehouse.

Core was collected at the drill rig and transported to the core logging facility on a daily basis, where it was cleaned and the boxes were marked with hole number, box number, and the sample interval. Each box of core was photographed and moved to a rack for examination by a geologist. After geologic and geotechnical logging were complete, geologists then marked sample intervals on the core and on the core box dividers with a permanent marker, along with the cutting line along the longitudinal axis of the core. All sample intervals and corresponding sample numbers were recorded on the geologic log. The core was then sawn in half along the cut line by a Coeur technician using a water-cooled diamond saw. One half of the interval was placed in a plastic sample bag along with a sample tag. The remaining half was returned to the core box and placed on numbered racks in a large, secure, storage shed at the project site.

11.2.2 Sample Preparation and Analysis Procedures

11.2.2.1 Luismin (1981, 1982, 1994)

The drillhole samples collected by Luismin were transported to the company's in-house laboratory in Durango. No written records of the chain of custody, sample preparation, or sample analysis procedures are known to exist.

11.2.2.2 Orko (2003–2008, 2011, 2012)

11.2.2.2.1 Sample Preparation

Orko used two sample preparation labs located in the city of Durango – Inspectorate and SGS. During 2005 to 2007, SGS was the primary lab used and Inspectorate served as the secondary lab. From hole BP07-93, the primary and secondary lab designations were switched, and Inspectorate became the primary lab in order to improve assay turn-around times. Upon receipt at both the SGS and Inspectorate sample preparation laboratories, the samples were placed in order according to sample number, and then crushed, and a sub-sample split was taken for pulverization. The remaining coarse rejects were returned to the project site and stored. Neither preparation lab was ISO nor IEC certified at the time the Project samples were processed.

11.2.2.2.2 Sample Analysis

The sample pulps were sent to Inspectorate's analytical laboratory in Reno, Nevada, USA, which was ISO 9001:2008 certified, and to the SGS analytical laboratory in Toronto, Canada, which was accredited by ISO/IEC 17025. Sample pulps representing check assays also were sent to these analytical facilities, as well as to ALS Chemex in North Vancouver, Canada and ALS Chemex in Reno, Nevada, USA, each of whom is independent of Coeur. At the SGS analytical laboratory in Toronto, the pulps were analyzed by several methods. Gold content was determined by fire assay at a detection limit of 5 ppb Au. Silver was analyzed by Atomic Absorption Spectrometry (AAS), at a calibrated detection limit of 0.3 g/t Ag and an upper limit threshold of 300 g/t Ag. Samples with silver values greater than 300 g/t Ag based on this analytical method were re-run by fire assay with a gravimetric finish. All samples also were subjected to strong acid digestion followed by a 40 element Inductively Coupled Plasma (ICP) analysis, including silver.

Some of the elements in the ICP package have threshold limits for ICP analysis. Examples include silver, which due to its 10 g/t upper ICP threshold does not allow the method to be used for this Project because over half of all samples exceed this value. Similarly, the base metals Pb and Zn and the element Ba have an upper threshold of > 10,000 g/t, (or 1.0%), which also precludes the use of ICP analysis for these elements. For the minerals containing any of the 40 elements that are totally digestible by strong acids, such as oxide, sulphide, and carbonate species, the ICP analysis method works well. However, for minerals containing any of these elements that are resistant to the strong acid digestion, only partial values will result.

The laboratory procedures used at the Inspectorate lab in Reno were similar to those used by SGS and described above. However, silver was an exception where, due to more precise instrument calibration, the detection limits were a lower 0.1 g/t (g/t) Ag and the upper threshold limit was 200 g/t Ag. As a result, samples having silver contents greater than 200 g/t were subsequently re-analyzed by fire assay with a gravimetric finish.

Orko completed two drillholes in 2011 and three drillholes in 2012 for a total of 500 m. Only 29.2 m of this drilling was sampled and assayed, according to the Orko database. No record of QA/QC procedures and results exists for these drill campaigns.

11.2.2.3 PAS (2008–2010)

11.2.2.3.1 Sample Preparation

Except for the pulp duplicate samples, all PAS samples were prepared and assayed by SGS in Durango, Mexico. Upon arrival at SGS, the samples were assembled in numerical order according to the sample tag numbers, individually crushed, then riffle split to provide a sub-sample for pulverizing. The pulverized, approximately 200 g sub-sample, was placed in a small labelled paper packet. After the required assay aliquots were removed, the residual material remaining in the packet was returned to PAS for storage on site at the Project, along with the coarse reject that remained after splitting of the assay sub-sample.

Pulp duplicate samples were analyzed at Inspectorate's lab in Sparks, Nevada.

11.2.2.3.2 Sample Analysis

Sample pulps analyzed at SGS used the following procedures:

- For gold analyses at SGS, all samples were initially assayed using fire assay procedures with AAS finish. The detection limit for this procedure was 0.005 g/t and the maximum assay threshold was 10 g/t. For samples initially assaying more than 10 g/t Au, these were rerun using a fire assay with gravimetric finish procedure having a detection limit of 3 g/t Au,
- For silver analyses at SGS, all samples were initially analyzed using 3-acid digestion with an AAS finish (0.3 g/t detection limit). For samples with analyses greater than the 300 g Ag threshold limit, the samples were rerun using a fire assay with gravimetric finish procedure having a detection limit of 5 g/t Ag. In addition, 33element trace analyses using a 2-acid digestion and ICP finish having a 2 g/t detection limit and a threshold of 10 g/t for silver were completed for all samples,
- For gold analyses at Inspectorate, all samples were run by fire assay with a gravimetric finish that had a detection limit of 3 g/t Au, and
- Silver analyses for all samples run at Inspectorate were initially run using a 4-acid digestion with ICP finish (0.1 g/t Ag detection limit) that had a 200 g/t Ag upper threshold limit. For samples with analyses greater than 200 g/t, the samples were rerun using fire assay with a gravimetric finish that had a detection limit of 5 g/t and an upper threshold limit of 5,000 grams per tonne Au.

11.2.2.4 Coeur (2013–2014)

11.2.2.4.1 Sample Preparation

All Coeur samples in 2013 and 2014 were submitted to an accredited commercial laboratory. Coeur contracted ALS Laboratory (ALS) in Zacatecas, ZAC, MX to complete all sample preparation on RC cuttings and split HQ drill core. The sample is logged in the tracking system, weighed, dried, and finely crushed to better than 70% passing a 2 mm screen. A riffle split of up to 250 g is taken and pulverized to better than 85% passing a 75 micron screen. The method is appropriate for both RC cuttings and drill core.

11.2.2.4.2 Sample Analysis

Sample pulps were created in Zacatecas and sent to ALS's analytical laboratory in Vancouver, BC, CA which is ISO 9001:2008 certified. Orko era pulps representing re-assays were sent to ALS Vancouver, as well as to SGS in Lakefield, ON, CA which is ISO 17025 certified. Both labs are independent of Coeur.

Silver Detection

At ALS, silver content was determined by Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES). A 0.25 g sample is digested with perchloric, nitric, hydrofluoric, and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed. The lower and upper detection limits (LDL and UDL) for this method are 0.5 ppm and 100 ppm, respectively. At 100 ppm the sample triggers an additional 4-Acid Digestion ICP-AES analysis that is optimized for accuracy and precision at high metal concentrations. This method utilizes the same acids as the prior method, but includes additional stages of heating and drying, along with the addition of de-ionized water to aid in further digestion. The LDL and UDL for this method are 1 ppm and 1500 ppm, respectively.

At SGS, silver content was determined by Inductively Coupled Plasma-Atomic Absorption Finish. This is a 4-Acid digestion. A 2 g sample is digested with perchloric, nitric, hydrofluoric, and hydrochloric acids. The LDL and UDL are 0.3 g/t and 300 g/t respectively.

Gold Detection

Gold content was determined by ICP-AES, following an initial Fire Assay Fusion of a precious metal bead. The sample bead is digested in 0.5 ml dilute nitric acid in the microwave oven. 0.5 ml of concentrated hydrochloric acid is then added for further digestion. The LDL and UDL of this method are 0.001 ppm and 10 ppm, respectively. At 10 ppm the sample triggers an additional Gravimetric analysis. This includes the creation of a lead button containing the 30 g sample, which is then cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid and weighed as gold. The LDL and UDL for this method are 0.05 ppm and 1000 ppm, respectively.

At SGS, gold content was determined by exploration grade fire assay. This is a 30 g fire assay with an ICP finish. The LDL and UDL are 1 ppb and 10000 ppb, respectively.

Multi-Element Detection

At ALS, an additional suite of 40 elements were also analyzed by ICP for all new samples created in 2013 and 2014. The drilling also encountered base metal values of zinc and lead which triggered the multi-element over limit method. Table 11-2 lists the elements and associated units, and detection limits.

Table 11-2: Multi-element ICP Package Analyzed by Coeur (Coeuer 2014)

Element	Symbol	Units	Lower Limit	Upper Limit
Aluminum	Al	%	0.01	50
Arsenic	As	ppm	5	10,000
Barium	Ba	ppm	10	10,000
Beryllium	Be	ppm	0.5	1,000
Bismuth	Bi	ppm	2	10,000
Calcium	Ca	%	0.01	50
Cadmium	Cd	ppm	0.5	500
Cobalt	Co	ppm	1	10,000
Chromium	Cr	ppm	1	10,000
Copper	Cu	ppm	1	10,000
Iron	Fe	%	0.01	50
Gallium	Ga	ppm	10	10,000
Potassium	K	%	0.01	10
Lanthanum	La	ppm	10	10,000
Magnesium	Mg	%	0.01	50
Manganese	Mn	ppm	5	100,000
Molybdenum	Mo	ppm	1	10,000
Sodium	Na	%	0.01	10
Nickel	Ni	ppm	1	10,000

table continues…

Element	Symbol	Units	Lower Limit	Upper Limit
Phosphorus	P	ppm	10	10,000
Lead	Pb	ppm	2	10,000
Sulphur	S	%	0.01	10
Antimony	Sb	ppm	5	10,000
Scandium	Sc	ppm	1	10,000
Strontium	Sr	ppm	1	10,000
Thorium	THz	ppm	20	10,000
Titanium	Ti	%	0.01	10
Thallium	TI	ppm	10	10,000
Uranium	U	ppm	10	10,000
Vanadium	V	ppm	1	10,000
Tungsten	W	ppm	10	10,000
Zinc	Zn	ppm	2	10,000

11.2.3 Sample Security

RC samples were collected and bagged at the drill rig by Coeur technicians, supervised by a project geologist. The bags are labelled with a unique sample ID and the interval meterage. Core samples were bagged by Coeur technicians at the project logging and storage facility. Samples were delivered daily by Coeur technicians and a project geologist to ALS in Zacatecas, ZAC, MX. Sample prep was completed Zacatecas and pulps samples were shipped to ALS Vancouver, BC, CA for analytical test work.

Chain of custody for delivery is established by transmittal sheets and sample receipt documents from the lab. Final chain of custody is ensured through electronic delivery of work orders and PDF assay certificates.

Hard copies of assay certificates and the geologic logs were stored at the Coeur project office in Durango, MX. The geologic logs include the sample sequence list, including inserted QA/QC. Electronic copiers of all data were stored by Coeur on a corporate server in Chicago. Ultimately all data are stored in an acQuire database located on an independent, backed-up server.

Coarse reject and sample pulps were returned to the Project site by laboratory staff and stored onsite in multiple secure storage facilities (the current core storage and logging facility).

11.2.4 Analytical Results

11.2.4.1 Assay Methods

Table 11-3 contains a listing of the assay methods and associated metadata used by Coeur. The sample preparation, security and analytical procedures are adequate and within industry accepted norms.

Based on a review of the database and procedures, a visit to the core storage and logging facilities, the QP is satisfied that the documented sample preparation, security, and analytical procedures are adequate and within industry accepted norms and suitable to support an MRE.

Table 11-3: Assay Methods

Analytical Laboratory	Element	Analytical Method	Units	Lower Limit	Upper Limit
ALS	Ag	ME-ICP41	ppm	0.2	100
ALS	Ag	ME-ICP61	ppm	0.5	100
ALS	Ag	ME-OG46	ppm	1	1,500
ALS	Ag	ME-OG62	ppm	1	1,500
ALS	Ag	GRA21	ppm	5	10,000
SGS	Ag	GE-AAS42E	g/t	0.3	300
ALS	Au	ICP21	ppm	0.001	10
ALS	Au	GRA21	ppm	0.005	1,000
SGS	Au	GE-FAI313	ppb	1	10,000

11.2.4.2 Data Delivery and Storage

Following the completion of analyses at the commercial laboratory, electronic results are delivered via email to a distribution list of Coeur recipients, approved by the project manager. ALS also provides secure online access to review the status of work orders and offers the ability to download data files and certificates.

Data are loaded into the acQuire database by a database manager or geologist with sufficient acQuire permissions. AcQuire is designed to securely store all original data. Acquire uses calculated and derived fields to produce data in a consistent format that can be uploaded into a 3D modelling package which allows for a further visual review of the data.

11.2.5 Quality Assurance and Quality Control, Check Samples and Check Assays

11.2.5.1 Luismin QA/QC (1981, 1982, 1994)

There are no records of any QA/QC programs or protocols prior to 2003.

11.2.5.2 Orko QA/QC (2003–2008, 2011, 2012)

11.2.5.2.1 Orko QA/QC Procedures

Orko maintained a QA/QC program during its tenure that consisted primarily of inserting standards and blanks into the sample sequence. Although no duplicates were included in the regularly submitted sample batches, duplicates (check samples) were submitted to a secondary laboratory in separate batches to check for systematic bias by the primary assay laboratory.

According to earlier Technical Reports (MDA, 2009, Snowden 2011a, MP 2012), alternating standards and blanks were inserted every tenth sample in the sample sequence, equivalent to a 5% insertion rate for each sample type. MP noted that based on the 88,235 core samples submitted by Orko to the primary laboratories, a 5% insertion rate is roughly equivalent to 4,400 blanks and an equal number of standards. However, MP stated that in the QA/QC files in the database provided for its MRE, there were data for a slightly smaller number of standards (3,994 standards, 4.3% of the total samples), but a significantly lesser number of blanks (1,127 blanks, or 1.3% of the total samples) and 1,103 duplicates. Similar quantities of Orko QA/QC data were reported by Mine Development Associates (MDA) and Snowden. The data verification process performed by the QPs responsible for this Report also detected a shortfall in the amount of expected QA/QC data, which is discussed in the following Section 12 of this Technical Report.

MP and Snowden stated in their Technical Reports that Orko's blank samples consisted of a combination of basalt core drilled during its exploration program and material collected from basalt boulders found in the La Preciosa area. The basalt blanks reportedly used were designated as Orko-2, Orko-4, Orko-5, Orko-7, and Orko-9. MDA noted in its Technical Report that SGS provided certificates for the basalt blanks based on approximately 150 analyses by aqua regia digestion and ICP-AES finish, but that no round-robin multi-laboratory analyses were done to substantiate the SGS values for the blanks.

Early on in Orko's exploration program, two commercial standards were used in a small number of sample batches, one of which was certified for gold and silver, and the other for gold only. The accepted values for these standards are unknown. For most of Orko's tenure, custom standards were used that were prepared from a stockpile of mineralized material situated near the Luismin portal. An unspecified amount of this material was sent to SGS's metallurgical division for certification prior to preparation of the standards. MDA reported that this certification was based on approximately 150 analyses. As with the SGS-certified basalt blanks, MDA noted that values established for the standards were not supported by round-robin testing at multiple labs. Over the course of Orko's exploration activities, four such custom standards were compiled, numbered as follows with accepted values in parenthesis: Orko-1 (0.210 g/t Au, 293.40 g/t Ag), Orko-3 (0.068 g/t Au, 112.00 g/t Ag), Orko-6 (0.072 g/t Au, 146.10 g/t Ag), and Orko-8 (0.134 g/t Au, 237.90 g/t Ag). In addition to these standards, in accordance with recommendations by MDA, a fifth standard was compiled that was subjected to round-robin multi-lab analyses. This standard, Orko-10, did not have a final certified value at the time MDA issued its Technical Report (results from one of the five round-robin labs had not been received). This standard subsequently saw limited use by PAS, and is discussed in Section 11.2.5.3.1 of this Technical Report.

11.2.5.2.2 Orko QA/QC Results

The results of Orko's QA/QC results are not easily interpreted. The standards were problematic because they were not certified, and record keeping was reported to have been inadequate such that the identity of the standards in the sample batches and assay certificates was not always certain (MDA 2009). The standard deviation (SD) of some of the standards was unusually high, particularly silver for Orko-1 and gold for all standards. The relative closeness of the accepted silver values for the high grade standards (Orko-1: 293.40 g/t Ag, and (Orko-8: 237.90 g/t Ag) and the moderate grade standards (Orko-3: 112.00 g/t Ag and Orko-6: 146.10 g/t Ag) result in overlapping of the two-SD ranges for the standard pairs, making it unclear whether some observed "failures" falling outside of these ranges were due to inconsistencies in the standards themselves, mislabeling of the standards during sample submission (as noted by MDA), or actual errors in the assay analyses. Snowden's graphical plots of the standard results suggest that there may have been some switching of standard labels for standards Orko–3 and Orko-8.

MDA (whose personnel were the only independent QPs involved one-on-one with Orko personnel during exploration drilling) reported that although Orko had examined the standard sample assay results on a batch-by-batch basis, these results were not systematically charted over time, and as a consequence, analytical failures were not investigated in a timely manner. This was compounded by the uncertainty as to the identities of some standards (as described above), which made the identification of actual failures more difficult. As a result, trends in the results of standards during drilling were not identified. Also, in some cases the standards were analyzed by an analytical method that differed from the method used to analyze certain drill core samples in a batch. This occurred at the Inspectorate lab, where all initial silver assays that fell below 200 g/t Ag were analyzed using an

ICP method (which occurred for two out of the four standards), whereas those samples with initial ICP assays greater than 200 g/t Ag were analyzed by fire assay with gravimetric finish. However, after reviewing the results of the Orko standards analyses, MDA concluded that the data neither revealed any systematic analytical biases, nor did it provide assurance that no such problems existed.

Four percent of the samples representing the Orko-2 blank and 10% of the basalt drill core samples exceeded the allowable value for silver, while the basalt drill core samples experienced no failures for gold. However, due to the lack of true round-robin certification of the blanks, in Snowden's opinion the blank samples could not be considered to be void of mineralization, making it impossible to determine whether blank sample failures were a result of contamination or background silver grade. Thus, Snowden judged the blanks to be unreliable for determining whether contamination was an issue in the sample preparation and analytical laboratories. However, Snowden did recognize an important point – if the blank failures were totally due to contamination, their magnitude did not indicate "significant concern with sample contamination".

A discussion of Orko's QA/QC duplicate samples can be found in Section 12 of this Technical Report.

Orko completed two drillholes in 2011 and three drillholes in 2012 for a total of 500 m. Only 29.2 m of this drilling was sampled and assayed, according to the Orko database. Therefore, no record of QA/QC procedures and results exists for these drill campaigns.

11.2.5.3 PAS QA/QC (2008–2010)

The descriptions and discussions of results presented in this section rely on those provided by Snowden in its 2011 Technical Report prepared for Orko and PAS (Snowden, 2011a).

11.2.5.3.1 PAS QA/QC Procedures

Relative to the Orko QA/QC program, the PAS QA/QC procedures provided for a much more systematic insertion of blanks, standards, and pulp duplicates into the batches of drill core samples. PAS batches consisted of 50 samples submitted to SGS, within which sample numbers ending with 10 or 60 were silver standards, sample numbers ending in 30 or 80 were gold standards, and sample numbers ending in 20, 40, 70, or 90 were blanks. Duplicate samples were collected by the laboratory by taking a pulp duplicate split of every sample represented by sample numbers in the sequence ending with 49 or 99, and these pulp duplicates were subsequently assigned the next sample numbers (ending in 50 and 00, respectively), which had been reserved in the sample numbering sequence by the geologists. These duplicates were subsequently submitted in separate batches of pulps to an umpire (secondary) laboratory.

The blank samples used by PAS consisted of half-core basalt. A total of 652 blank samples were inserted as described in the previous paragraph, resulting in a 4% insertion rate. A total of 662 standards were inserted as described, resulting in a similar 4% insertion rate. PAS used three different standards summarized as follows, with expected values shown in parenthesis - Orko-10 (145.47 g/t Ag, 0.057 g/t Au), GBM908-13 (151.4 g/t Ag), and G308-7 (0.27 g/t Au). Standards GBM908-13 and G308-7 were commercial standards purchased from Geostats Pty. Ltd. Standard GBM908-13 was a base metal standard that was also certified for silver and sulphur, while standard G308-7 was certified for gold only. Standard Orko-10 was the custom standard described earlier in this Report. Snowden confirmed that Standard Orko-10 was prepared by SGS in Durango from material obtained from stockpiles at the Project site and noted that although this standard was round-robin tested in five independent laboratories from which expected values for gold and silver were derived, it was never officially certified. A total of only 21 samples from this standard were inserted in sample batches, all of which were from drillholes BP09-355 to BP09-364. PAS commissioned SGS Peru to evaluate the Orko-10 standard material, and SGS Peru concluded that this standard had unacceptably high variances, probably due to the presence of native silver in the sample. As a result, PAS discontinued its use of standard Orko-10 beginning with hole BP09-365 and thereafter.

Snowden reported that PAS geologists regularly monitored the performance of standard and blank assays received from SGS by plotting values as line graphs in Excel as soon as each batch of assays was reported by the laboratory. Whenever any of the standard results exceeded three SDs from the expected value, the entire batch of assays was re-submitted for analysis.

11.2.5.3.2 PAS QA/QC Results

Snowden reported failures for only two gold standards and five silver standards which represent failure rates of 0.3% for gold, and 0.8% for silver. These results indicate that laboratory contamination was not a material concern for the samples from the PAS drilling campaigns.

The results for both silver and gold from certified commercial standards GBM908-13 and G308-7 displayed a bias on the high side of the expected values. Although the results for GBM908-13 fell within acceptable limits, a consistent high bias relative to the expected mean was present. Results for standard G308-7 displayed a very slight bias towards the under-reporting of the gold grades. However, all results were within two SDs of the certified expected values, indicating acceptable levels of laboratory accuracy.

11.2.5.4 Coeur QA/QC Program (2013–2014)

Coeur maintained a QA/QC program that was structured on guidelines set forth in the written company QA/QC policy. QA/QC consisted of routine insertion of standards, blanks, and duplicates into the primary sample stream for both RC and core samples. Umpire check assays have been commissioned in 2014. Table 11-4 defines the suggested Test % for QA/QC sample insertion based upon the total count of primary samples.

Table 11-4: Coeur Development Program QA/QC Recommendations

	Primary Lab Control					External Control Samples				
Sample Type	**Duplicates**			**Standards**	**Blanks**	**Pulps**	**Rejects**	**Standards**	**Blanks**	**Duplicates**
Duplicate Type	**Sample**	**Prep**	**Analytical**							
Suggested Test %	2.5%	2.5%	2.5%	5.0%	5.0%	10.0%	1.0%	1.0%	1.0%	0.5%

11.2.5.4.1 Certified Standards and Blanks

The 2013 and 2014 assay campaigns utilized five certified commercial standards and one round robin tested standard. The campaigns used one certified blank and one round robin tested blank. The certified standards were purchased from CDN Resource Laboratories Ltd., in Langley, B.C., Canada and SGS de Mexico, in Durango, Mexico. The blank was purchased from Rocklabs, in Auckland, New Zealand. Table 11-5 lists the standards and blank and their certified silver and gold values.

Table 11-5: Coeur Certified Standards and Blanks

Standard ID	Certifying Lab	Element	Standard Value (ppm)	1 SD (ppm)
CDN-ME-1101	CDN	Ag	68	2.3
CDN-ME-1101	CDN	Au	0.564	0.28
CDN-ME-1205	CDN	Ag	25.6	1.2
CDN-ME-1205	CDN	Au	2.20	0.14
CDN-ME-1303	CDN	Ag	152	10
CDN-ME-1303	CDN	Au	0.924	0.1
CDN-ME-1304	CDN	Ag	34.0	1.6
CDN-ME-1304	CDN	Au	1.8	0.06
HGRS-02	SGS	Ag	98.7	4.0
HGRS-02	SGS	Au	0.01	0.003
ORKO-10	SGS (round robin)	Ag	145.466	4.227
ORKO-10	SGS (round robin)	Au	0.057	0.005
AuBlank58	Rocklabs	Ag	< 0.002	N/A
AuBlank58	Rocklabs	Au	N/A	N/A
BLANK-5	Acme, ALS, SGS (round robin)	Ag	< 5	N/A
BLANK-5	Acme, ALS, SGS (round robin)	Au	< 0.004	N/A

QA/QC comparison analyses are completed in the acQuire database using separate tools for blanks, standards, and duplicates. Performance of the standards was tracked over time and against lower and upper cut-off limits. Run plots were generated, with multiple user controlled options. Plots generated for this report include the assay value plotted against the certificate number, which depicts the standard's performance over time. The plots also contain error lines indicating the acceptable minimum and maximum values for the given standard and assay method. Coeur policy recognizes QA/QC failures as ± 3 SDs for standards and ± 5 times the lower detection of the assay method for blanks.

When a blank or standard fails QA/QC it is moved to a “rejected” status in acQuire, along with all primary, duplicate, and lab QA/QC samples above and below the failure, and up to the next or previous passing blank or standard. This partial batch of samples must be re-run at the original laboratory, and with the same analysis method as the failed method. In the case of the Project, the QA/QC focuses on silver and gold, although the primary analysis includes multi-element ICP. Coeur does not re-run the multi-element ICP on failed sample batches. If the blank or standard fails a second re-assay the entire batch must be analyzed a third time at a second commercial laboratory, using an analysis method similar to that of the original test work.

11.2.5.4.2 Coeur QA/QC Results

Blanks and Standards

In 2013-2014, Coeur submitted 21,991 primary samples for assay at two commercial laboratories. 1,018 blanks and 1,369 standards where inserted into the sample streams, representing insertion rates of 4.63% and 6.23%, respectively. The combined insertion rates exceed the total of 10% standards and blanks suggested by Coeur protocol and included in Table 11-4. All standards and blanks were analyzed at ALS in Vancouver, BC, CA and/or SGS in Lakefield, ON, CA. Table 11-6 is an actual example of a QA/QC report exported directly from the acQuire database. The report tables the statistical performance of all standards and blanks, defined the assaying laboratory and by the analytical method utilized. The column # Outside Limit is the count of failed standard or blanks.

Table 11-6: acQuire Standards QA/QC Report, ALS

Assay Field	Std ID	# of Analyses	# Outside Limit	% Outside Limit	Mean	Median	Min	Max	Standard Deviation	% Rel Std Dev	Standard Error	% Rel Std Err	Total Bias
Ag_Ag_GRA21_ppm	ORKO-10	6	0	0	134.33	136	120	141	6.97	5.19	2.84	2.12	0
Ag_Ag_OG46_ppm	ORKO-10	6	0	0	137	135.5	120	161	12.18	8.89	4.97	3.63	0
Ag_Ag_OG62_ppm	ORKO-10	137	38	27.74	145.75	143	124	185	11.47	7.87	0.98	0.67	0
Ag_Ag_OG62_ppm	HGRS-02	238	1	0.42	101.95	102	96	141	3.53	3.46	0.23	0.22	0.03
Ag_ME_ICP61_ppm	HGRS-02	282	13	4.61	95.13	97.55	36.1	100	10.69	11.24	0.64	0.67	-0.04
Ag_ME_ICP61_ppm	CDN-ME-1101	285	0	0	68.74	68.9	59.5	78.7	2.75	4	0.16	0.24	0.01
Ag_ME_ICP61_ppm	CDN-ME-1205	172	0	0	26.39	26.3	23.8	29.8	1.15	4.36	0.09	0.33	0.03
Ag_ME_ICP61_ppm	CDN-ME-1304	225	3	1.33	34.98	34.8	20.8	65.1	3.04	8.7	0.2	0.58	0.03
Au_Au_GRA21_ppm	CDN-ME-1205	157	0	0	2.31	2.3	1.91	2.88	0.18	7.82	0.01	0.62	0.05
Au_Au_GRA21_ppm	CDN-ME-1304	2	2	100	2.5	2.5	2.24	2.75	0.26	10.22	0.18	7.23	0.39
Au_Au_ICP21_ppm	ORKO-10	143	0	0	0.06	0.06	0.05	0.07	0	5.16	0	0.43	0.02
Au_Au_ICP21_ppm	HGRS-02	508	18	3.54	0.01	0	0	1.81	0.08	822.12	0	36.48	-0.01
Au_Au_ICP21_ppm	CDN-ME-1101	285	0	0	0.58	0.58	0.49	0.68	0.03	5.77	0	0.34	0.03
Au_Au_ICP21_ppm	CDN-ME-1205	172	0	0	2.22	2.21	1.84	2.63	0.17	7.48	0.01	0.57	0.01
Au_Au_ICP21_ppm	CDN-ME-1304	225	3	1.33	1.78	1.81	0.01	2.1	0.21	11.75	0.01	0.78	-0.01

The manual insertion of standards and blanks includes an inherent risk for mislabeled or incorrectly inserted samples. Original standards and blanks failed QA/QC and the sample batches were re-assayed. Further review was conducted on the failed standards and blanks by Coeur and Acme Laboratories. The result of the review identified 189 standards and seven blanks that were likely assigned an incorrect standard ID or blank ID. These samples were reclassified based on the original assay results which produced silver and gold values that were associated with another active project standard or blank. The result of the reclassification was the approval of the Round 1 QA/QC for these samples.

Further review of the results of standard Orko-10 indicated a resulting bias to the overall QA/QC performance. The PEA indicated that the use of Orko-10 had been discontinued by PAS after a re-evaluation of the standard resulted in unacceptably high variances (M3 2013). The Orko-10 standard was inserted 143 times into the 2013-2014 campaign. The decision to use the standard was the result of recent performance indicating low SDs, based on 161 sample results; and its immediate availability to the Project. The standard failed 38 silver analyses, a 26.6% failure rate. The run plots in Figure 11-23 illustrates that the standard performed consistently when analyzed by Ag_OG62_ppm, by failing both above and below the standard limits, with locally larger magnitude failures above the maximum acceptable limit. In order to quantify these findings, Table 11-7 and Table 11-8 contain values in parentheses that indicate QA/QC results which exclude the entire Orko-10 sample set. Coeur has moved to discontinue the used of standard Orko-10, and dispose of any remaining sample material.

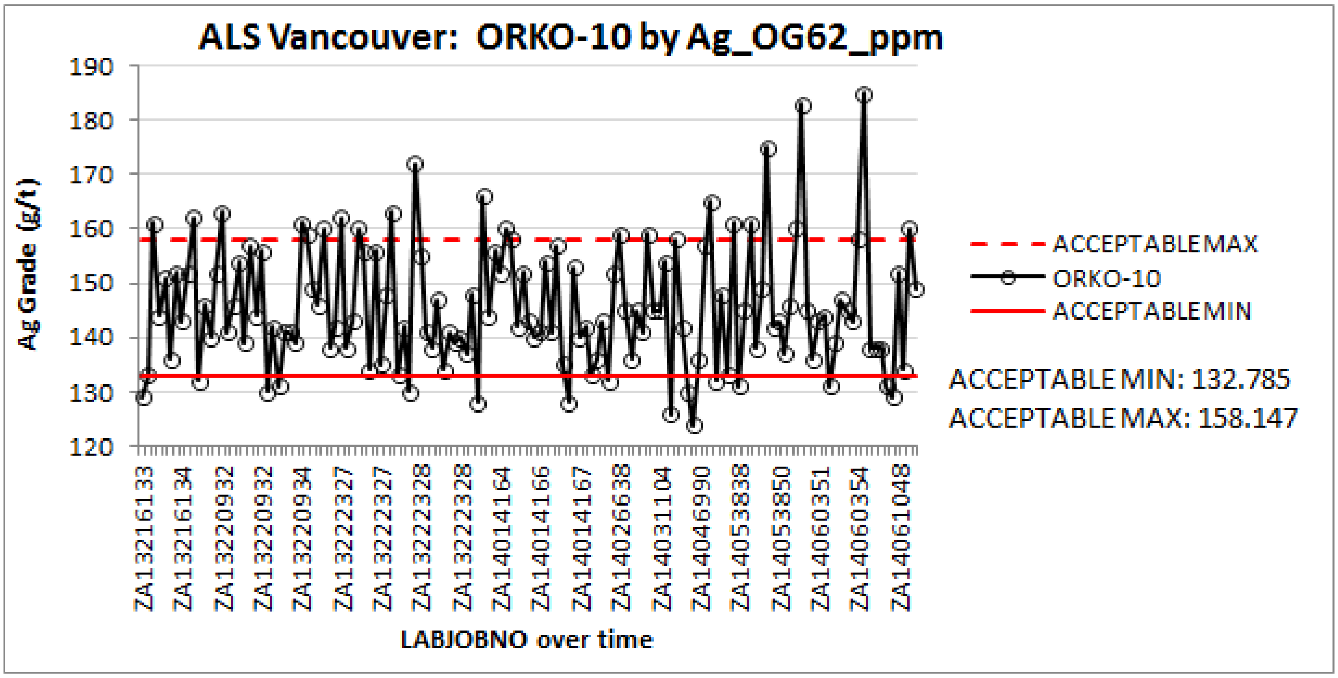


Figure 11-23: Orko-10 Silver Standard. Medium Population, with Multiple Failures Outside the Acceptable Minimum and Maximum.

Table 11-7 contains a Round 1 summary of Coeur QA/QC results for company inserted standards and blanks. Round 1 results are those submitted with the original sample batches for the 2013-2014 campaign. The Total Failure is the combined failure rate of standards and blanks with respect to the total submitted primary samples for the assay campaign. The values in parentheses in Table 11-7 indicate the resulting standard and failure count after all ORKO-10 standards are removed from the Round 1 QA/QC statistics. The resulting recalculated failure rate is in parentheses. The Round 1 sample statistics indicate a low (<2%) failure rate for the Coeur campaign after removal of the ORKO-10 subset.

Table 11-7: Summary of Round 1 QA/QC Results

Group	Count/Rate
Primary Samples	21,991
Blanks	1,018
Blank Failures	16
Standards	1,369 (1,226)
Standard Failures	60 (22)
Failure Rate	3.27% (1.78%)

The run plot examples in this report (Figure 11-24 through Figure 11-27) illustrate various graphical representations of the performance of actual blanks and standards. The run plot demonstrates assay value on the y-axis, plotted against the LABJOBNO on the x-axis. The LABJOBNO on the x-axis is also a depiction of the standard performance over time.

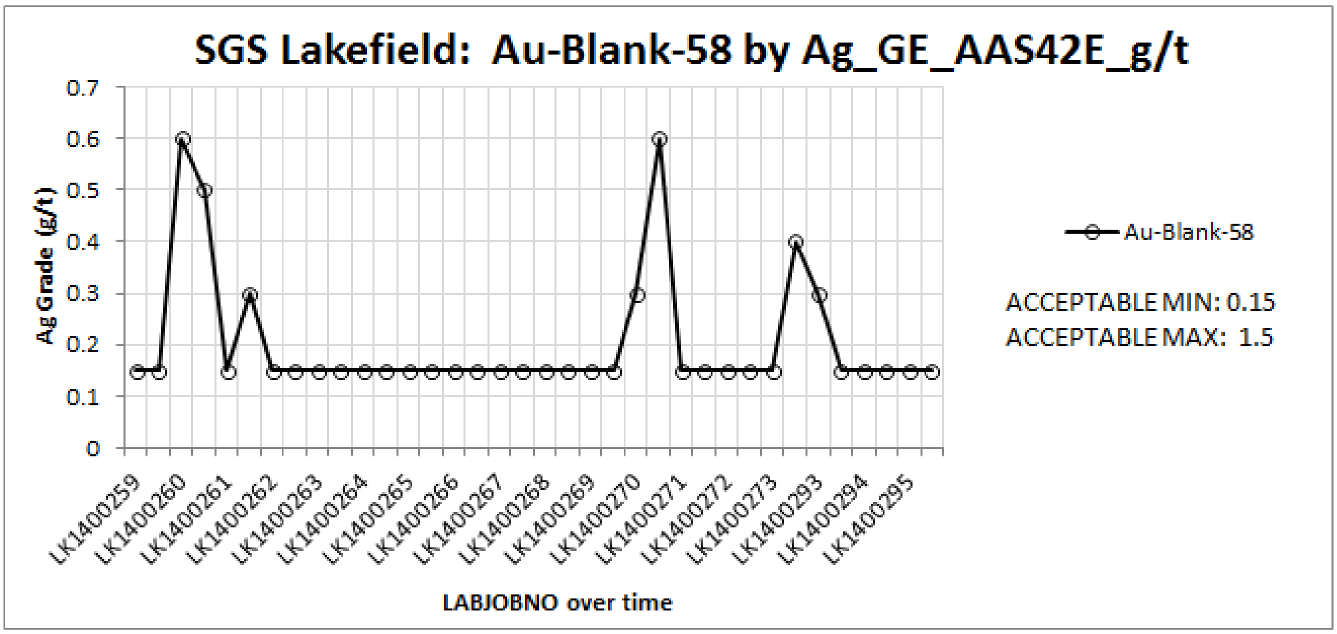


Figure 11-24: Silver Blanks. Small Population, Zero Failures. Baseline or ACCEPTABLEMIN is ½ the LOWERDETECTION of the Assay Method.

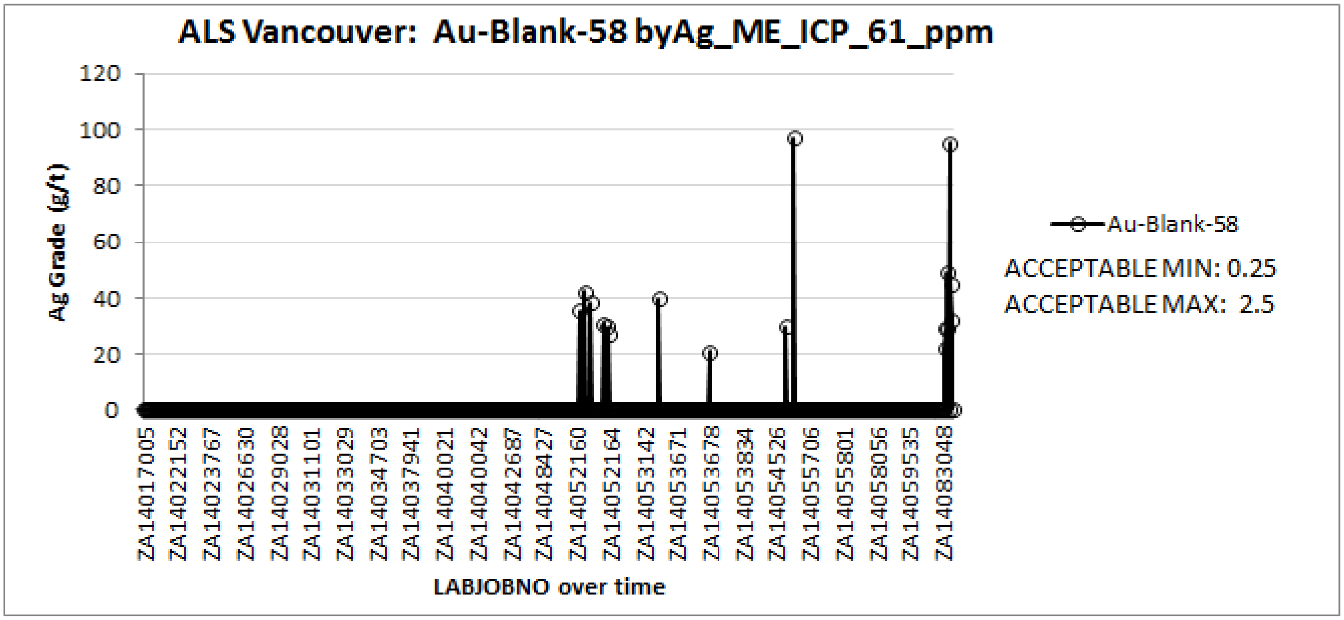


Figure 11-25: Silver Blanks. Large Population with Multiple Failures of Large Magnitude. Baseline, or ACCEPTABLEMIN is ½ the LOWERDETECTION of the Assay Method.

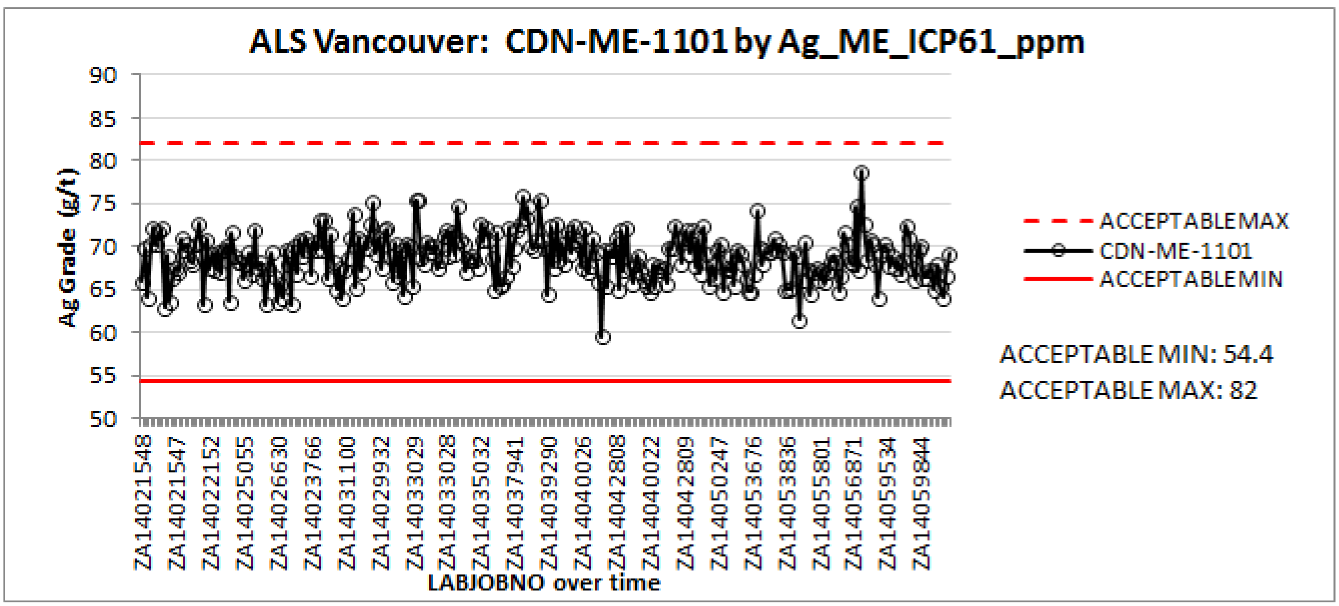


Figure 11-26: Silver Standard. Large Population with Zero Failures. Results Trend Along the True Standard Value.

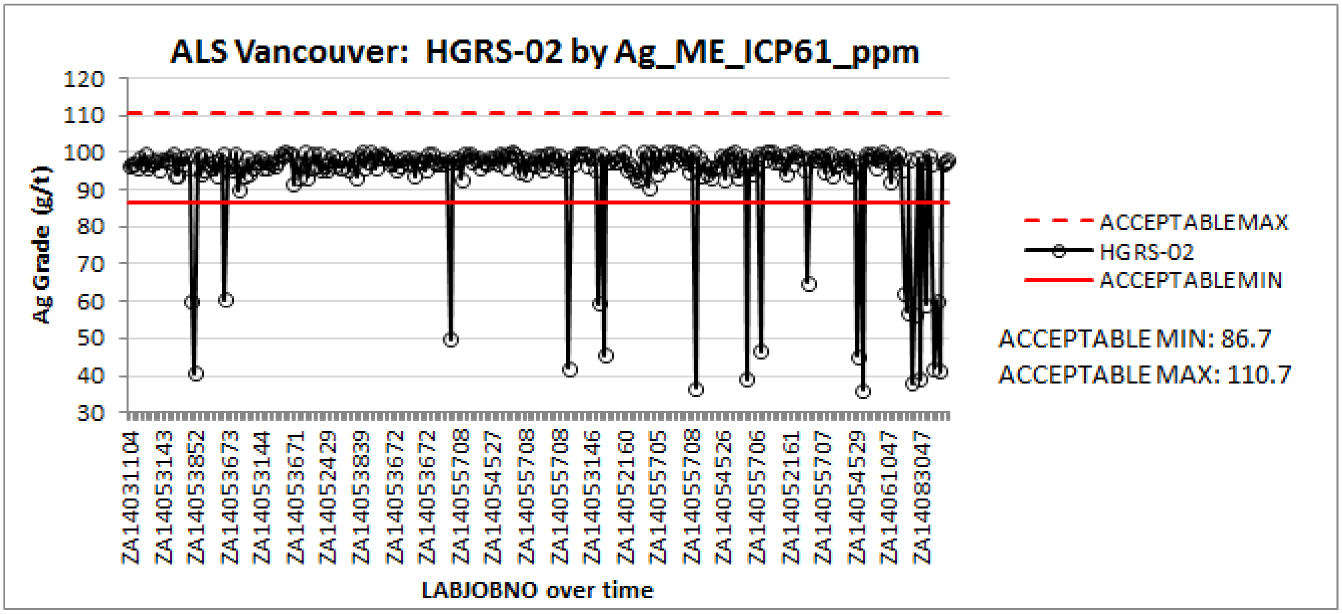


Figure 11-27: Silver Standard. Large Population with Multiple Failures of Large Magnitude Below the ACCEPTABLEMIN.

A percentage of failures of Round 1 QA/QC were subjected to reassay at the same laboratory using the same assay method as the original sample. Table 11-8 summarizes the result and status of standards and blanks that have received a Round 2 analysis. Samples pending results of a Round 2 analysis are also tabled. At the time of this report, 11 blanks and 30 standards are pending results of the Round 2 analyses. As in Table 11-7, values in parentheses represent sample counts that exclude Orko-10 standards. All samples that failed Round 2 QA/QC, and their associated sample batches, will be analyzed at a second commercial laboratory with an analysis method equivalent to that used by ALS.

Table 11-8: Summary of Round 2 QA/QC Results

		Development Drilling
Blanks	Reassayed Round 2	6
	Passed Round 2	0
	Failed Round 2	5
	Pending Round 2 Results	11
Standards	Reassayed Round 2	30 (15)
	Passed Round 2	16 (3)
	Failed Round 2	14 (2)
	Pending Round 2 Results	28 (5)
	Total Percent Passing Round 2	**44% (14%)**

All QA/QC samples that have not successfully passed Coeur's QA/QC procedures remain in a "rejected" status in the acQuire database. Additionally, all primary samples associated with these failed control samples remain in a "rejected" status and are unavailable for use in the resource evaluation. When a control sample passes QA/QC, the new primary assay value of the associated primary samples will be moved to an "accepted" status in the acQuire database.

Duplicates

In 2013-2014 Coeur submitted 15,107 new primary samples to ALS. These samples are subjected to the company duplicate QA/QC protocol, included in Table 11-4. The protocol suggests an equal distribution of duplicate samples at various check stages. These include the sample (S), prep (C), and analytical (P) stages. Coeur submitted 1,347 duplicates resulting in an insertion rate of 8.92% which exceeded the total suggested insertion rate of 7.5%. Check stage totals and resulting failures are included in Table 11-9. The Threshold is a dynamic value that eliminates sample pairs from the population based on their assay value's proximity to the lower detection level of the method used. For this exercise, the Threshold is equal to 10 times the lower detection of the method. Any primary sample with an assay value less than this is considered below the Threshold and is removed from the analysis. Failed duplicates will be reassayed in 2014 with the primary sample and all associated duplicates per Coeur QA/QC protocol.

Table 11-9: Duplicate Sample Summary

Check Stage	Sample Type	Count	Count Above Threshold	% Acceptable Difference	Failed Duplicates	% Failure
Sample	(S) Sample	661	236	N/A	N/A	N/A
Prep	(C) Crush	343	173	25%	14	8.1%
Analytical	(P) Pulp	343	172	20%	18	10.5%
Eligible Primary Samples		**Duplicate Samples**			**Insertion Rate**	
15,107		1,347			8.92%	

Figure 11-28 is a scatter plot of the primary sample grade on the x-axis and the duplicate sample grade on the y-axis. The plot is segregated by check stage. The plot illustrates R2 values for each check stage that indicate moderate correlation between the sample duplicates (S) and excellent correlation between the analytical duplicates (P).

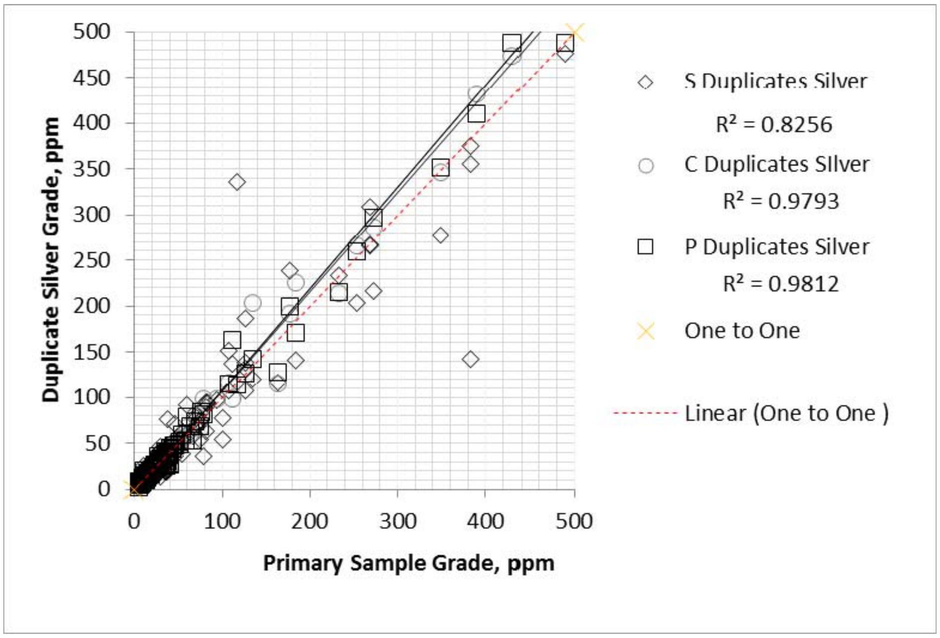


Figure 11-28: Scatter Plot of Primary vs. Duplicate Values by Check Stage

Umpire Assays

Umpire assays for the 2013-2014 sample campaigns are currently in progress. Per Coeur policy, a random selection of pulps chosen throughout the range of grades was selected from each assay certificate from the primary lab and sent to another laboratory for check analysis; using the same analytical digestion method and instrumental finish.

11.2.5.5 Opinions and Recommendations of the Qualified Person

Based on a review of the core shack and sampling facilities while the Coeuer team were still working at La Preciosa and periodic reviews of the Avino area database and procedures, the QP is satisfied that the QA/QC, check samples, and check assays are adequate and within industry accepted norms and suitable to support an MRE.

12.0 DATA VERIFICATION

According to NI 43-101 standards for mineral project disclosure, "data verification" means the process of confirming that data has been generated with proper procedures, has been accurately transcribed from the original source and is suitable to be used. Data verification carried out by the current QPs and applied to the property is summarized in Table 12-1.

Table 12-1: QPs Opinion on Data Verification

Item	Data Type	Source	Verification Method	QP	Remarks
Topography Data	Digital contours	Avino	Quantitative 3D comparison with public domain SRTM data	MO'B	No material discrepancies
Underground excavations	Wireframe files	Avino	Sectional checks with drill trace data 3D comparison using field GPS spot checks of adit entrances	MO'B	Reviewed and modified by QP to close open shapes, no material discrepancies
Drill Collar Positions	Digital point data	Avino	Quantitative 3D comparison with topography and field GPS spot checks	MO'B	No material discrepancies
Drill downhole surveys	Digital point data	Avino	3D model sectional reviews to detect extreme deviations of desurveyed hole traces	MO'B	No material discrepancies
Grade data	Digital Database	Avino	5% random check against laboratory assay certificates, 3D review of sample grades and positions	MO'B	No material discrepancies, direct digital uploads since 2019 have reduced transcription risk
Lithology logs	Digital database	Avino	random comparisons with paper logs and vein cores during site visits	MO'B	No material discrepancies
Geology Model	Wireframe files	Avino	wireframes backtagged to litho codes in drill database and sectional reviews	MO'B	Modified by QP to close open shapes and invert negative volumes, no material discrepancies
Metallurgy & Process	Test results & operation data	Avino & Test Reports	QP reviewed operation and laboratory data	JH	Data present in the report are the summary of the test results and operation data. Tailings reprocessing design was based on the test results
Capital and Operating Costs	Operation data	Avino	QPs reviewed operation data	HG/ JH/ MM	Operation data reviewed is summarized in the report.
Infrastructure	Operation data	Avino & Site Visit	QP visited the site and reviewed operation data	HG	QP visited the site and verified the infrastructure.
Mining	Operation data	Avino	QP reviewed operation data	MM	QP reviewed operation data
Environmental	Report	Avino	QP reviewed the report and data	HG	QP reviewed the report
Market Studies	Report	Avino	QP reviewed the report and data	HG	QP reviewed the report and summary is provided in the report

More detailed descriptions of processes and results are described below.

12.1 Avino Mine Area

12.1.1 Drillhole Database Verification

The QP reviewed the drillhole data provided by Avino in the form of a Microsoft Access database before and after loading the data into Leapfrog Geo™, on a hole-by-hole basis, including drillhole collar, survey, lithology, and assay data.

During several site visits between 2012 and February 2020, the QP verified several drill hole logs against the physical cores for ET06_02, ET07_01, ET07_03, SG15_03, and SG15_02. No significant discrepancies were found.

Collars for exploration drillholes are marked by concrete monuments, and the collars have been surveyed. Subsequent collars have been similarly marked and were observed during a site visit in June 2016.

A check of collar coordinates at the La Preciosa area with a handheld global positioning system during a site visit in 2020 Confirmed the positions of drill collars previously reported by Coeur.

The QP opinion on the reliability of the Avino drillhole data is discussed in Section 12.1.3.3, and detailed recommendations are provided in Section 26.1.

12.1.1.1 Collar and Assay Data

In early 2020, Ausenco (under supervision of the QP) validated the drillhole database for the Avino Mine area, and it was consolidated into a consistent and comprehensive Microsoft Access database that is maintained by mine personnel with offsite backup in the Avino office in Durango and on the Cloud using Egnyte. The QP subsequently reviewed the data in December 2022. Table 12-2 summarizes the database validation results.

The Avino Vein lithology data for 13 historic drillholes at the Avino ET Mine and all channel samples are very sparse owing to the age of the records. The upper part of the deposit model has consequently been modelled using assay data and development mapping information. As the deficient lithology information pertains mainly to parts of the deposit that have been mined out, the QP considers the database to be adequate to support Mineral Resource estimation.

Table 12-2: Number of Records and Discrepancies for the Avino (ET and San Gonzalo) Drillhole and Channel Sampling Data

Deposit	Number of Records	Discrepancies	Discrepancy Rate (%)
Avino Vein			
Survey Data	11,415	-	-
No Surveys for Collar	-	0	-
Duplicate Collar and Surveys	-	0	-
Duplicate Survey Depths	-	0	-
Assays	42,564		0

table continues…

Deposit	Number of Records	Discrepancies	Discrepancy Rate (%)
No Sample for Collars		0	
Overlapping Segments	-	0	-
Collar Max Depth Exceeded	-	0	-
Lithology	4,074	-	-
From to Depth Overlap	-	26	0.6
No Samples for Collar	13	13 historic holes	-
Collar Max Depth Exceeded	-	0	-
San Gonzalo Vein			
Survey Data	3,636	-	0
No Survey for Collar	-	0	-
Duplicate Collar and Surveys	-	0	-
Duplicate Survey Depths	-	0	-
Incomplete Survey Data	-	0	-
Assays	17,514		0
No Samples for Collars	-	0	-
Overlapping Segments	-	1	-
Collar Max Depth Exceeded	-	2	-
Lithology	140 holes	-	0
From to Depth Overlap	-	0	-
No Samples for Collar	-	0	-
Collar Max Depth Exceeded	-	0	-
Guadalupe Vein			
Survey Data	3,636	-	0
No Survey for Collar	-	0	-
Duplicate Collar and Surveys	-	0	-
Duplicate Survey Depths	-	0	-
Incomplete Survey Data	-	0	-
Assays	17,514	-	0
No Samples for Collars	-	0	-
Overlapping Segments	-	3	-
Collar Max Depth Exceeded	-	0	-
Lithology	140 holes	-	0
From to Depth Overlap	-	0	-
No Samples for Collar	-	0	-
Collar Max Depth Exceeded	-	0	-

table continues…

Deposit	Number of Records	Discrepancies	Discrepancy Rate (%)
La Potosina Vein			
Survey Data	3,636	-	0
No Survey for Collar	-	0	-
Duplicate Collar and Surveys	-	0	-
Duplicate Survey Depths	-	0	-
Incomplete Survey Data	-	0	-
Assays	17,514	-	0
No Samples for Collars	-	0	-
Overlapping Segments	-	13 historic holes	-
Collar Max Depth Exceeded	-	0	-
Lithology	622	-	-
From to Depth Overlap	-	3	0.5
No Samples for Collar	-	0	-
Collar Max Depth Exceeded	-	0	-

The QP believes that the sample assay integrity is adequate for Mineral Resource estimation.

12.1.1.2 Downhole Survey Data

Downhole survey data exists for 650 drillholes completed on the Avino Property. The QP believes that the downhole survey data is adequate for 3D geological modelling and to support Mineral Resource estimation. Routine photography of drillcore and underground drifts is being carried out. A digital photographic record is kept of all drillcores and underground drifts for future reference and to facilitate consistent core logging and geology interpretation.

The QP believes that logging and record keeping are adequate to support Mineral Resource estimation.

12.1.1.3 Assay Verification of 1990/1991 Drillholes in Oxide Tailings

Tetra Tech (2013) verified 54% of drillholes in this database (15 of 28 drillholes) and 58% of both silver and gold assays (444 of 766 values) used for this estimation. QG Consulting verified the oxide tailings data in 2016, and the QP reviewed the work again in 2017 and is unaware of any significant errors.

The QP opinion of the reliability of the 1990 to 1991 oxide tailings assays is discussed in Section 12.1.3.2.

12.1.1.4 Oxide Tailings Verification Samples

As was reported by Tetra Tech (2013), during a previous site visit conducted on June 7 and 8, 2012, Michael F. O’Brien visited the tailings heaps and supervised the collection of eight samples from the oxide tailings (3 kg to 4 kg each). The samples were collected from gulleys that had eroded into the tailings pile and provided a vertical section through the tailings. It is believed that while such samples cannot provide a statistically representative reflection of the overall grade, they do provide some insight into the grade of the tailings near the surface. The eight samples were each split into three separate sub-samples, which were submitted in turn to the Avino Mine laboratory together with SGS laboratories in Durango and Vancouver.

Statistical analysis of the three sets of results demonstrated that there is a good correlation between the three laboratories, and this conclusion remains valid.

The sampling exercise in 2012 provided the opportunity to review the artificial sedimentary deposit that comprises the Avino oxide tailings and supported the previous assumptions of the tailings, such as regarding the oxide tailings as two superimposed units with slightly different chemical and particle size characteristics and pronounced horizontal continuity. The source data and plans prepared more than 20 years ago, after the initial drilling campaign, were examined at the mine and found to be of professional standard and provide support for their use in the estimation of the oxide tailings. The overall homogeneity of the material, horizontal continuity, and relatively high confidence in the volume and tonnage mitigate any uncertainty in the historical dataset. The pattern of sample grades from the 2015/2016 drill campaigns and the earlier drilling forms a coherent pattern with no obvious discontinuity between campaigns.

12.1.2 Site Visits

Michael F. O'Brien conducted site visits on June 7 and 8, 2012, June 6 and 7, 2016, June 12 to 15, 2017, February 11 to 13, 2020, and July 20, 2021. During the latest visit, the QP visited the Project on July 20, 2021, and reviewed the tailings deposit and deep drilling for the Avino Vein System.

12.1.3 The QP Conclusion and Opinion

The drill dataset has been produced over a long period of time within a brownfield property. All data used for this study is obtained from work carried out by the staff of the current issuer, which has owned the Property continuously since the start of this work.

The QP believes that the geological and sampling data are adequate and that appropriate and current verification work has been carried out to support Mineral Resource estimation.

12.1.3.1 Avino and San Gonzalo Veins

12.1.3.1.1 Drillhole Database

The drillhole data for the Property has been consolidated into a single Microsoft Access database, with the exception of the shallow surface drilling for the oxide tailings deposit. The consolidated database with an offsite and cloud-based backup has materially improved the consistency and security of the exploration data. The QP recommends that the exploration data pertaining to the oxide tailings deposit and the QA/QC information be integrated into the database.

12.1.3.1.2 Downhole Survey Data

Downhole survey data and the location of the Avino and San Gonzalo Vein intersections observed in drillholes have been verified by both surface and underground mapping, providing confidence in the location, orientation, and true width of both veins.

12.1.3.1.3 Geology Data and Interpretation

The legacy data from the Avino Vein is understandably deficient in recorded lithology data. Modelling of the Avino Vein and San Gonzalo Vein systems made use of grade as well as lithology data. Consequently, the QP regards the lithology database as adequate and fit for the purposes of resource estimation. The recent mining history supports that the potential economic units persistently demonstrate continuity as new exposures become available.

12.1.3.1.4 Density Samples

Based on a review of density data from drillholes in the Avino and San Gonzalo Veins, the QP concludes that future bulk density measurements should be completed using a water displacement method (see Section 11.1.7).

12.1.3.1.5 QA/QC Samples

The rate of QA/QC SRMs, blanks, and duplicates insertions meets recommended industry standards. HARD charts indicate less assay precision than would be expected for pulp duplicates.

The QP has found no evidence of systematic laboratory bias, indicating that the assay results are adequate.

12.1.3.2 Tailings

The identified grade pattern is similar in character to other tailings deposits, such as overall homogeneity and a pronounced horizontal continuity.

Verification samples taken by the QP have confirmed the presence of gold and silver mineralization at grades similar to those obtained in the original tailings drilling campaign, with a low silver bias consistent with the superficial position of the samples in the zone most likely to have suffered surface leaching. The verification samples also confirm that the mine laboratory assays are not materially different from those of external laboratories.

12.1.3.3 QP Opinion

There were no limitations on or failure to conduct data verification.

The QP believes the assay, sample location, vein lithology, and bulk density data from the Avino and San Gonzalo Veins are adequate to support the purpose of this Technical Report and a current Mineral Resource on both vein and tailings deposits.

12.2 La Preciosa Area

12.2.1 Current Verifications

The QP visited the project on July 20, 2021, and reviewed drill cores, the logging facilities, vein outcrops, and drill collar positions.

Five well-marked collars were located, and positions were checked with a hand-held GPS unit (GPSMAP 66i). Plan positional differences were well within the expected limits of error (see Table 12-3).

Table 12-3: Summary of Collar Locations and Positions

	Field measured		Database		Plan Difference
DHID	Easting	Northing	Easting	Northing	(m)
BP07-126	555161.0	2702386.0	555161.0	2702385.3	0.7
BP09-355	555159.0	2702383.0	555159.7	2702382.3	1.0
BP09-357	555159.0	2702383.0	555159.9	2702382.1	1.2
BP09-361	555203.0	2702431.0	555203.0	2702431.0	0.1
BP09-360	555204.0	2702431.0	555203.6	2702431.0	0.4

The core shack and core processing facilities were visited, and several representative cores were reviewed and compared with logging sheets. Logging matched the cores reviewed (BP07-102, CLP14-077) and assay grades were supported by signs of mineralization. The core storage and logging facilities are clean and weather-proof, and the core stacks and boxes are clearly labelled (Figure 12-1 and Figure 12-2).

Underground drift development on the Gloria and Abundancia Vein and outcrops were visited (Figure 12-3 and Figure 12-4), and the veins were examined.

Figure 12-1: Core Logging Facilities La Preciosa (Red Pennant 2021)

Figure 12-2: Core Storage (Red Pennant 2021)

Figure 12-3: Adit level Drift on Abundancia Vein (Red Pennant 2021)

Figure 12-4: Gloria Vein Outcrop Looking South. Width of View ~200 m in Foreground (Red Pennant 2021)

12.2.2 Historic Verifications

The Project drillhole database was validated by the Coeur technical services group. It has been verified and deemed appropriate for resource modelling. A review and validation of the 2013 - 2014 assay, collar coordinate, downhole survey, and assay data has been performed by Coeur.

The historic drillhole database has been verified by Orko (pre-2009), MDA (2009), PAS (2008-2010), Snowden (2011), MP (2012), and Independent Mining Consultants (IMC) (2013).

12.2.2.1 Orko – Pre-2009

Prior to 2009, Orko reportedly sent 331 duplicate sample pulps from five of its drillholes to ALS Global (formerly ALSChemex) in North Vancouver, B.C., Canada, as a check against Inspectorate's primary assay results for these holes. Although the analytical methods used for silver by ALS for some of the check samples reportedly differed slightly from those methods used by Inspectorate, the results for these check samples indicated an only slight high bias on the part of Inspectorate for silver grades less than 30 g/t Ag, and a corresponding very slight high bias across the board for Inspectorate gold assays, as determined using fire assay with gravimetric finish methods. The QP responsible for this section of the Report reviewed scatterplots of 317 of these 331 gold and silver check analyses provided in earlier Technical Reports (which did not provide reasons for the 14 check assays missing from scatterplot comparisons). Based on these reviews, in the opinion of the QP, these check assay data fall within acceptable limits (M3 2013).

In addition to the 331 duplicate samples, Orko submitted coarse rejects to SGS for 134 samples from drillhole BP07-102 that were originally prepared and assayed by Inspectorate. SGS in turn prepared pulp duplicates for these samples that were subsequently submitted to ALS. MP, consultants to Orko and the QPs for the November 5, 2012 Technical Report on the Project, created Q-Q plots of assays for 120 of the 134 samples from the three labs that were available in the MP database. These plots indicated reasonable correlation with no biases between Inspectorate and SGS for gold or silver. The Q-Q plots for Inspectorate versus ALS showed similar correlations, but with an apparent slight high bias for silver in the Inspectorate assays. In the opinion of the QP responsible for this section of the Report, these comparisons are acceptable for both gold and silver between all three laboratories (M3 2013).

12.2.2.2 Mine Development Associates – 2009

As a follow-up to the pre-2009 check assay programs conducted by Orko, MDA in 2009 conducted a comprehensive check assay program that included pulp and coarse reject samples reportedly representing each of the mineralized vein intercepts. Submitted by MDA to ALS in Reno, Nevada, these checks samples consisted of 240 pulp rejects, of which 61 original pulps were assayed by SGS and 179 original pulps, which were assayed by Inspectorate. Q-Q plots of the results revealed acceptable correlation between the two primary laboratories (SGS and Inspectorate) and the secondary laboratory (ALS), with no indication of biases. In conjunction with this check assay effort, MDA inserted QA/QC blanks and gold and silver standard samples into the check assay batches at select but unequal intervals. The QA/QC results for the blank samples indicated no failures for silver and 2 failures (out of 10) for gold. All standard assays fell within the acceptable ranges (M3 2013).

For determination of material density, Orko had routinely conducted one density measurement from each sample sent for assay, using a single piece of half-core removed from each sample and a water immersion method that resulted in a set of recorded densities that exceeded 88,000 in number. After each density analysis, the samples used were returned to the appropriate sample sacks for shipment to the laboratory for assay. Concerned that Orko’s density determinations were possibly biased high because the method used did not account for the presence of vugs in the vein samples, MDA had Orko complete an additional 92 density determinations using a dry analysis technique on whole core representing the Martha vein and other lesser veins from the deposit. In the opinion of the QP responsible for this section of the Technical Report, this density validation testing generated specific gravity values that are not significantly different than the average specific gravities obtained by Orko’s analyses (M3 2013).

12.2.2.3 PAS – 2008-2010

The Snowden (September 2011) and MP (November 2012) Technical Reports both make reference to a suite of pulp duplicate samples representing the Martha vein from both earlier Orko and PAS drillholes. According to the Snowden Technical Report, “To eliminate any concerns about the quality of Orko data, PAS undertook a specific testing program of original data by reassaying drillhole samples and by comparing recent PAS drillhole sample grades with earlier Orko sample grades, which also showed grade biases”. The Snowden Technical Report further states in Table 11.3 that the duplicate samples, which totaled 146 in number, were submitted “because of problems in correlating mineralization over short distances between Orko and Pan American holes”. The MP Technical Report added that of the 146 duplicate pulps, 43 of the pulps were originally assayed by Inspectorate, while 103 were originally assayed by SGS. The duplicate assays for all 146 pulps were generated by SGS. However, neither report provides any details of the results of comparisons between the duplicate sample pairs (M3 2013).

To follow up on MDA’s validations of material density, PAS applied four different testing methods to the same individual rock samples removed from 252 different sample intervals in the remaining half core. These included

133 samples from veins and adjacent silicified material and 119 samples from un-silicified andesite, the most common host rock to the Project mineralization. The selection of each of the 252 samples considered the variable degree of oxidation in the deposit by taking approximately 40 samples of vein/silicified material and approximately 40 samples of andesite from shallow (highly oxidized), middle (moderately oxidized), and deep (weakly oxidized to unoxidized) portions of the deposit. One of the four measurement techniques employed included data for determination of a “void index” that could be used to derive bulk density. The selected samples (most of which were previously measured for specific gravity by Orko) weighed between 400 grams and 600 grams in order to reduce measurement error. Prior to testing, each sample was geologically described. The resulting specific gravity measurements were reportedly made by a technician in the metallurgical laboratory at PAS’s La Colorado mine operations in Zacatecas, Mexico. The results of these measurements indicated that the Orko-specific gravity data were suitable for use in Mineral Resource estimation, provided that a bulk density conversion factor of 0.99 was applied to the average specific gravity for each of the material types (vein, vein silicification, and various host rocks). In the opinion of the QP responsible for this section of the Report, this fine tuning of the measured specific gravities for the various material types is acceptable, but not material to estimation of Mineral Resources in the Project deposit (M3 2013).

12.2.2.4 Snowden – 2011

The Snowden Technical Report (Snowden 2011a; 2011b) does not mention the collection by Snowden of any independent drill core samples or existing coarse reject or pulp duplicates for check assay. Snowden reportedly reviewed original assay certificates from SGS in Durango, Mexico and from Inspectorate’s laboratory in Sparks, Nevada, which included 441 assays from PAS’s drilling and 3,188 assays from Orko drillholes. In total, 44 errors were noted in the PAS database assay files (an error rate of 1.4%), 41 of which were determined to be related to a single assay batch that apparently was subsequently reassayed. Two of the three remaining errors reportedly involved the entry of incorrect assay detection limits. Nine errors were noted in the Orko assay database, seven of which were reportedly due to cases where the average values of acceptable duplicate assay pairs were entered rather than the primary sample assays. Snowden did not perform or recommend additional material density (specific gravity) testing (M3 2013).

12.2.2.5 Mining Plus – 2012

MP independently collected 74 samples consisting of 23 samples of half (sawn) core and 46 existing coarse rejects. Although the MP Technical Report states that, “All results from this program returned values well within acceptable limits”, no actual data for the duplicate sample analyses were provided. MP also reportedly compared the results of 3,285 assays representing the major zones of mineralization reported on laboratory certificates with entries in the assay database and found six errors (an error rate of only 0.2%), and concluded that none of these errors were materially significant with respect to mineral resource estimation (M3 2013). MP checked collar coordinates for 17 drillholes using a hand-held GPS unit and noted no significant discrepancies (considering the accuracy of the GPS unit) with the values in the project database. Downhole survey data also were reviewed by MP, which found errors believed to have been caused either by inaccurate data transcription or by errors in the actual survey measurements. Where original data could be located, these downhole survey data were verified or corrected. MP also noted variances between drillhole azimuths and inclinations at the drillhole collars and the initial downhole survey data points. In MP’s view, the reason for these discrepancies probably was due to the drill set-ups not corresponding to the planned azimuths and inclinations. To address these differences, MP reported that hole collar markers were removed and new collar measurements made of drillhole azimuths and inclinations, except where prevented by deterioration of the drillhole collars due to caving and/or the presence of the magnetic basalts on the eastern portion of the project drill pattern. Also, MP reported removing the downhole survey data for several drillholes for which it determined that the trace of these holes as defined by the downhole surveys was physically impossible. In summary, outside of the errors described above, MP noted that with the collection of downhole survey data on 50 m intervals, the potential impact of individual survey errors is limited. MP did not perform or recommend additional material density (specific gravity) testing (M3 2013).

12.2.2.6 IMC – 2013

Only five additional drillholes that were not included in the MP MRE were considered for inclusion in the IMC MRE summarized in Section 14, (holes 1, 2, 3, 4, and 5) and only one of these has assay data (BP12-718). In the opinion of the QPs responsible for this Report, no additional independent validations of these data were warranted. This opinion is based on the lack of any significant amount of additional data that post-dates the MP study and Technical Report, and in light of the documented acceptable agreement between duplicate sampling and assaying exercises and material density validations conducted by previous QPs (M3, MDA, Snowden, and MP). The QA/QC information that has been described in this section was loaded into the IMC system and analyzed to confirm previous work.

In summary, the primary basis for confirmation of the drillhole database are the inserted standards that have been completed during the assay programs. There were also a number of inserted blank samples. As described earlier, there are a total of 1,103 duplicate assays in the QA/QC database. These are a mixture of:

- Duplicate Pulps to the same lab: 43 samples
- Check Pulps to a second lab: 793 samples
- Coarse Rejects to the same lab: 192 samples
- Coarse Rejects to a second lab: 75 samples

The analysis by previous QPs and IMC do not indicate any particular issues with bias in the above data sets. However, none of the above programs are sufficiently consistent in procedure, data distribution, or purpose to be considered as a major component of the QA/QC data. The above duplicate samples represent 1.1% of the total assay database. The reliability of the entire drill program is therefore relying on the inserted standards and blank samples.

Standards

There are 4,580 standards inserted into 104,720 assays (4.4%) of which 547 standards were inserted into the 12,955 assays above 25 g/t (4.2%).

Blanks

There are 1,765 blanks out of 104,720 assays (1.7%) of which 348 blanks were inserted into the 12,955 assays above 25 g/t (2.7%).

Blanks were not inserted on a regular basis, although it appears that blanks were likely inserted after high-grade intercepts on a visual judgment basis rather than on a consistent insertion basis.

Standards were generally inserted throughout the database. The percentage inclusion is similar in both mineralized material and waste components of the deposit. IMC prepared maps and sections of the holes that contain standards and found them to cover the area of the mineral resource on a relatively consistent basis. Analysis of tested standards versus the published Standard Value in the MP Report dated November 5, 2012 do not show any immediate issues regarding assay lab bias, regardless of the lab used for the primary assay. There are some confusing data points in the QA/QC database that present different certified values for some standards compared to the MP Report. IMC has chosen to use the certified values as published in the MP Report.

The statistical analysis of the QA/QC database indicates that the project database can be accepted for estimation of mineral resources, based almost entirely on the reliable results from inserted standards. However, the QA/QC database in general does not meet industry "best practices" in the opinion of IMC.

12.2.2.7 Coeur Validation of Drill Data

Assay data were imported into an acQuire database using assay import object. The commercial laboratories provide assay data in a pre-constructed Comma-Separated Values (CSV) template that imports seamlessly into the acQuire database when no errors are present. AcQuire imports adhere to very strict rules relating to sample IDs, assay data, and lab job numbers. When these rules are violated, an error report is generated. There is no manual data entry related to the assay import process. In addition, continuous comparison of the database values against the original PDF certificates is a valuable check on the database integrity.

12.2.2.8 Coeur Geologic Data Validation

In 2014 Coeur initiated and completed a comprehensive review and data entry campaign for all drillhole geologic logs from Orko, PAS, and Coeur. The review included 843 drillholes for a total of 259,919 m. The Project was contracted to HRC and included review and data entry of hardcopy and scanned geologic drill logs. HRC inspected four database tables; lithology, alteration, mineralization, and structure. A list of accepted logging codes was provided by Coeur with edits applied as needed as the Project proceeded. HRC supplemented the data process with an internal validation process that reviewed the logs for legibility, completeness, and consistency with regards to geologic interpretation. Select drillholes were reviewed in 3D to identify potential error in interpretation. Mechanical audits were completed to identify overlapping intervals, gaps in geology, and inconsistencies in drill depths.

12.2.2.9 Coeur Collar Survey Checks

Drillhole collars are checked visually in MineSight Comprehensive Modeling and Mine Planning Platform and on topographic based maps to confirm that holes are on the correct drill pads and map coordinates.

Downhole survey data are imported into an acQuire database using an import object. The 2013-2104 downhole surveys were completed by IDS Mexico. IDS provided CSV data files that, in most cases, imported seamlessly into acQuire. On several occasions the CSV file was constructed from a different template, and additional manual formatting was required and completed by the geologic database manager. On import, the data are checked for overlapping intervals and intervals below the recorded length of the drillhole.

12.2.2.10 Coeur Validation of Historical Drillhole Data

Coeur continued to verify Orko era data in 2013 and 2014. A 2013 report includes a review of the original Orko database and reviews subsets of the data loaded into the acQuire Database. This review included drillhole collar, downhole survey, density, RQD, and sample ID and assays. The review also outlines the general acQuire database structure. Recommendations are proposed and have been addressed by Coeur, or are included in this report as recommendations for further work. Coeur identified the need to verify assay data against original hardcopy assay certificates. Coeur initiated this project in 2014. Verification of samples against hard copy assay certificates was conducted manually at both the Chihuahua Exploration office and the Chicago Corporate office. The initial comparisons checked 298 standards and 497 primary samples that were analyzed at Inspectorate from 2006 through 2008. No discrepancies between the two datasets were identified. This amounts to verification of 2.9% of QA/QC samples and 0.7% of primary samples. The Orko and PAS assay results required verification for use in resource estimation. Further evaluation of the Orko master database was completed in 2014. Inspectorate provided official data files and certificates for 100% of the lab jobs completed from 2007 - 2009. The Inspectorate results were loaded to acQuire directly from the data files and are presumed validated, but will benefit from an additional check against the hardcopy certificate.

SGS data files provided by Orko had been modified. Coeur was able to obtain 426 of 753 (57%) original data files and 634 of the original PDF certificates from the SGS. The data files received were loaded directly into acQuire and the resulting data are considered validated. QA/QC procedures were completed on the imported SGS data files, per Coeur's company protocol. A combined silver and gold total of 3,180 identifiable QA/QC values were analyzed in acQuire, resulting in 30 failures. Seven pairs of data were excluded from the analysis due to either the standard value exceeding the UDL of the assay method, or because the standard was not certified for a gravimetric analysis. Coeur policy mandates that a standard fails when the value exceeds ± three SDs of the standard value and a blank fails if it exceeds ± five times the LDL of the analysis method. Failure rates for silver and gold for from the SGS data files were 0.7% and 1.3%, respectively. Assay pairs from the Orko master database and the SGS data files were compared in acQuire to validate the primary assay database. A combination of 46,551 sample pairs of silver and gold were analyzed using x-y scatter plots. (Figure 12-1 and Figure 12-2) The silver and gold values should be exact matches and therefore a failure is defined as any deviation greater than zero. The comparison resulted in 510 failures, or 1.1% of the pairs. Figure 12-1 and Figure 12-2 visually indicate that the failures are of relatively low magnitude. Coeur is confident 54 of the silver failures are the result of Orko gravimetric results being merged into a fire assay field. These are not true failures, but the result of manual data manipulation. The remaining silver and gold failures are assumed to be attributed to manual error or discrepancies with reporting of assays at or near the detection level of the assay method.

The data and statistics presented in this section illustrate that Coeur was able to verify 60% of the primary SGS sample count and was able to verify a 43% increase in the QA/QC sample count. The QA/QC insertion rate of 12%, from the SGS data files indicated that the validated data contained an acceptable quantity of QA/QC samples which exceeded Coeur's current internal requirement. The QA/QC completed on the SGS data performed well, with low failure rates. The primary assay comparison produced very low failure rates and overall low magnitude failures. In summary, these comparisons represent 60% of the primary assay database and 57% of the total lab jobs stored in acQuire. The remainders of the outstanding original data files are considered to be unobtainable. Therefore, Coeur considered the performance of the Orko data that was verified by the SGS data files to be indicative of the performance of the entire Orko master database, and subsequently acceptable for use in resource evaluation.

12.2.2.11 Coeur Data Collection Campaigns

12.2.2.11.1 Density

The Orko master density database contained 89,226 records of calculated density, with an average density of 2.51 g/cm^3. In 2014, Coeur developed a standardized procedure to improve the accuracy of density measurements. In March 2014 Coeur employees completed 1,667 density measurements using the new procedure (see Table 12-4). The average density of these measurements was 2.52 g/cm^3.

Table 12-4: Density Data 2014 Summary

Lithology	Density (g/cm^3)
Basalt	2.38
Metamorphic	2.67
Sedimentary	2.50
Volcanic	2.48

12.2.2.11.2 Geomechanical

In 2013, KP initiated a geomechanical drill program. The drilling consisted of two holes completed in 2013 and four holes completed in 2014. The drilling was completed by Major Drilling using HQ3 diameter core using triple tube techniques. The geomechanical core was oriented using the Reflex ACTIII device. The drill core was split and assayed using Coeur's procedures discussed in this report. The assay results for these drillholes are not included in the resource estimation data set.

12.2.2.11.3 Reassay of Pulps

Initial silver assaying by Orko at SGS used a 3-acid digestion with ICP-AES finish for all samples, followed by fire assays for samples exceeding 300 g/t Ag. In 2013, 313 pulp samples were reassayed and demonstrated that using a 4-acid digestion with an ICP finish resulted in a more complete sample digestion and, on average, higher Ag grades. In 2014, Coeur selected all Orko era samples within the 25 - 100 ppm silver grade range for reassay by the 4-Acid ICP method. Coeur submitted 6,059 and 825 pulp samples to ALS and SGS, respectively. Reassay results from ALS showed an average increase in Ag grade of 11% when comparing Coeur's 4-acid digestion ICP results to Orko's 3-acid digestion results. Reassay results from showed an average increase in Ag grade of 2.2% when comparing Coeur's 4-acid digestion ICP results to Orko's 3-acid digestion results.

12.2.2.12 Coeur Review of Orko Era Drill Core Sampling

Core drilled during the Orko and PAS campaigns was selectively sampled based on geologic observation during the core logging process. In 2013, Coeur initiated a campaign to re-log, sample, and assay portions of the previously unsplit core. During the review of the core and logs, Coeur geologists identified samples that ended in mineralized material as well as structures and alteration that were unsampled. The geologists collected these new samples from 35 drillholes along three parallel sections. The program resulted in 3,520 new primary samples added to the acQuire database.

12.2.2.13 Coeur Review of Orko Era Quartz Vein Sampling

In addition to the legacy drill core sampling, Coeur geologists reviewed the database to identify un-sampled intervals logged as quartz vein. A total of 95 samples, representing 120 meters of core length, were identified in the drillhole database. If the material was available and un-sampled, the vein interval and several meters of core above and below it would be sampled and assayed.

12.2.2.14 Spectroscopy Study

In 2014, Coeur contracted SRK Consulting of Toronto, Ontario to conduct an Infrared Absorption scan for alteration on the Project drill core. SRK scanned 76 drillholes for a combined 8,568 intervals totalling 24,987 m. The spectral data identifies the presence or absence of clay species and is useful for geotechnical purposes such as identifying fault zones, zones of high cohesive strength, and depth to water. The spectral dataset is stored in the project acQuire database.

12.2.3 Data Verification Conclusion

The QP is satisfied that the sampling and assay data, topographic information, and drill core management for this project have been comprehensively verified and are suitable to be used for mineral resource estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Avino Mine Area

The Avino processing plant is currently processing materials from the Avino underground mine. The target metal values are gold, silver, and copper. The materials from the previous San Gonzalo Mine were processed from October 2012 to Q4 2019 with the target values of gold, silver, lead, and zinc. There are four grinding and flotation lines with a total capacity of 2,500 t/d, including two 1,000 t/d and one 250 t/d lines to recover copper, gold, and silver into a copper concentrate, and a separate 250 t/d line, which was used to produce materials from the previously operating San Gonzalo Mine, which has ceased operation since the end of 2019.

There is a potential tailings resource from previous operations; currently, there is no operation on tailings material.

13.1.1 Avino Vein

The Avino Vein material is currently being processed at the Avino processing plant using froth flotation to produce a marketable copper concentrate with silver and gold credits. A gravity concentration circuit was also incorporated in three of the four processing lines. The material has been successfully processed in the past.

The Avino Vein was mined during the 27 years of open pit and underground production prior to 2001. From 1997 to 2001, the mine and processing plant production averaged 1,000 t/d and achieved up to 1,300 t/d. The mine and plant operations were then suspended. Following several years of redevelopment, in Q4 2014, Avino completed its Avino mining facility and plant expansion. Full-scale operations commenced on January 1, 2015, and commercial production was declared effective on April 1, 2016, following a 19-month advancement and test period.

The feed from the Avino Vein has been processed using froth flotation to produce a copper concentrate with silver and gold credits. In the 2023 operation, the average silver, gold, and copper recoveries reporting to a silver/gold/copper concentrate and a gravity concentrate were 87%, 72%, and 83%, respectively. The total material processed was 615,373 t.

Bismuth was identified as a deleterious material in the concentrate. SGS completed six batch flotation tests to explore the possibility of bismuth and copper separation from a copper concentrate produced from the Avino processing plant. The separation tests included:

- Floating copper minerals with suppressing bismuth-bearing minerals
- Floating bismuth-bearing minerals with suppressing copper minerals on the copper concentrate reground to 80% passing approximately 25 µm

The test results show that approximately 30% of the bismuth in the bulk copper concentrate could be floated into a concentrate containing 18.7% Bi from the copper concentrate containing 1.8% Bi using flotation separation. The copper and silver reporting to the bismuth concentrate were 1.3% and 14.7%, respectively. SGS indicated that compared to the floating copper and suppressing bismuth process, and it appears that the process with floating bismuth and suppressing copper is feasible and promising. SGS recommended that further copper and bismuth separation test work using the flotation procedure, including a mineralogical study, be conducted.

13.2 Test Work on Tailings Materials

The test work on the tailings material is presented into four main sections as follows:

- Latest metallurgical test work conducted by SGS during 2022 and 2023
- Attrition and Filtration test work conducted by SGS 2023
- Historical test work conducted by Process Research Associates Inc. (PRA) under supervision of MMI between 2002 and 2003 – Oxide Tailings
- Historical test work conducted by PRA under supervision of MMI between 2002 and 2003 – Sulphide Tailings.

13.2.1 SGS Metallurgy Test Work 2022–2023

For the 2022-2023 SGS test work, three composite samples named ancient oxides, recent oxides, and sulphides was evaluated using flotation, column test, and bottle roll leaching test to identify most suitable process options to recover gold and silver. Additional test work for cyanide detoxification was also conducted for safe management of the tailings.

13.2.1.1 Test Program

Table 13-1 lists the details of the test work conducted for composite samples for recent oxides, ancient oxides and sulphides during the 2022-2023 test campaign.

The test work includes the sample characterization procedure such as head assay, bulk leach extractable silver and gold, and particle size distribution. The test work also involved flotation for preconcentration of valuable minerals for subsequent downstream processes. Also, the effectiveness of tank leaching and heap leaching also re-evaluated using bottle-roll and column test, respectively.

Table 13-1: Test Procedures for Ancient Oxides – SGS 2022-2023 Test Program (SGS 2023)

Process/Procedure	Details of Test	Sample Identify
Sample Preparation	Composite sample – Homogenized and Split in Rotary Splitter	Ancient Oxides, Recent Oxides and Sulphides
Head Assays	Fire assays, AA finish for Au and gravimetric finish for Ag; ICP multi-acid; C and S using Leco; CN soluble Au and Ag.	Ancient Oxides, Recent Oxides and Sulphides
Particle Size Distribution	As-received sample sieve analysis	Ancient Oxides, Recent Oxides and Sulphides
Bulk Leach Extractable Gold (BLEG)	25g, Mini Cyanide Leach, AAS or ICP Finish.	Ancient Oxides, Recent Oxides and Sulphides
Grind Calibration Test	To achieve target P80 of 105 and 75 µm	Ancient Oxides, Recent Oxides and Sulphides
Gravity Recoverable Gold (GRG) Test	Two stages: Stage 1 (As is) & Stage 2 (Regrinding), 10 kg	Ancient Oxides, Recent Oxides and Sulphides
Bulk Flotation	Effect of particle size and flotation kinetics	Ancient Oxides, Recent Oxides and Sulphides

table continues…

Process/Procedure	Details of Test	Sample Identify
Bottle Roll Test	Effect of particle size, cyanide concentration, addition of oxidizing agents and viscosity reducer.	Ancient Oxides, Recent Oxides and Sulphides
Column Leach Test	As received feed; 100 days duration; 0.5 g/L NaCN; 10 L/m^2/h Irrigation rate; size by size fraction analysis on feed and leach residue;	Ancient Oxides, Recent Oxides and Sulphides
Merrill Crowe Precipitation Test	Effect of Zinc and Lead Nitrate Dosage	Composites, Ancient Oxides, Recent Oxides and Sulphides
Cyanide Destruction Test	Cyanide detox on pulp and filtered cake	Ancient Oxides, Recent Oxides and Sulphides

13.2.1.2 Tetra Tech reviewed the metallurgical tests conducted for the 2022-2023 test program. Sample Preparation

Approximately 150 kg of each composite sample named ancient oxides, recent oxides, and sulphides were sent for the metallurgical study. Each of the samples was homogenized and rotary split to obtain samples for chemical analysis (~5 kg), bottle roll test (~15 kg), column test (~70 kg), flotation tests (~25 kg) and the remaining sample (~35 kg) was stored for future use.

13.2.1.3 Chemical Analysis

Each of the sub-sample composite was analysed for Au (fire assay with AA finish), Ag (fire assay with gravimetric finish), multi-element analysis (four-acid digestion with ICP finish), sulphur and carbon analysis using LECO. Table 13-2 and Table 13-3 list chemical assay results for the three composite samples.

For all the composites, gold and silver does not show significant variability in grades and samples contains only a minor amount of deleterious elements such as arsenic, copper, bismuth, antimony and zinc.

Table 13-2: Gold, Silver, Sulphur and Total Carbon Analysis Results for the three composite samples

Element	Unit	Sulphides	Ancient Oxides	Recent Oxides	Method
Au	g/t	0.24	0.45	0.48	Fire Assay, AA
Au, Duplicate	g/t	0.25	0.43	0.49	Fire Assay, AA
Ag	g/t	21	101	37	Fire Assay, Gravimetric
Ag, Duplicate	g/t	19	92	49	Fire Assay, Gravimetric
S	%	1	0.21	0.9	LECO
C, Total	%	0.1	0.09	0.06	LECO

Table 13-3: Multi-Element Analysis Results for the three composite samples

Element	Unit	Sulphides	Ancient Oxides	Recent Oxides	Method
Al	%	2.9	3.7	2.9	Four-Acid ICP
As	ppm	319	141	246	Four-Acid ICP
Ba	ppm	645	663	519	Four-Acid ICP
Be	ppm	0.7	1	1	Four-Acid ICP
Bi	ppm	123	335	263	Four-Acid ICP
Ca	%	0.7	1	0.4	Four-Acid ICP
Cd	ppm	12	7	11	Four-Acid ICP
Co	ppm	5	6	5	Four-Acid ICP
Cr	ppm	19	77	81	Four-Acid ICP
Cu	ppm	592	1341	1335	Four-Acid ICP
Fe	%	5.8	7.2	7.8	Four-Acid ICP
K	%	1.7	1.9	1.2	Four-Acid ICP
La	ppm	10	13	8	Four-Acid ICP
Li	ppm	69	50	70	Four-Acid ICP
Mg	%	0.5	0.3	0.5	Four-Acid ICP
Mn	ppm	1855	1907	1621	Four-Acid ICP
Mo	ppm	23	14	18	Four-Acid ICP
Na	%	0.1	0.03	0.03	Four-Acid ICP
Ni	ppm	6	6	6	Four-Acid ICP
P	ppm	300	300	200	Four-Acid ICP
Pb	ppm	1155	9311	2189	Four-Acid ICP
Sb	ppm	85	185	128	Four-Acid ICP
Sc	ppm	3	4	3	Four-Acid ICP
Sn	ppm	<10	<10	<10	Four-Acid ICP
Sr	ppm	39	45	33	Four-Acid ICP
Ti	%	0.1	0.1	0.1	Four-Acid ICP
V	ppm	25	33	21	Four-Acid ICP
W	ppm	33	30	53	Four-Acid ICP
Y	ppm	8	11	7	Four-Acid ICP
Zn	ppm	1160	1572	1127	Four-Acid ICP
Zr	ppm	29	40	30	Four-Acid ICP

13.2.1.4 Particle Size Distribution

Sieve analysis was performed on the as-received sample for each of the composite. Results are shown in Figure 13-1 and it can be noted that ancient oxides are slightly coarser than the sulphides and recent oxides. The percentage of particles passing 80% (P80) for sulphides, recent oxides and ancient oxides are 156 µm, 167 µm, and 212 µm, respectively.

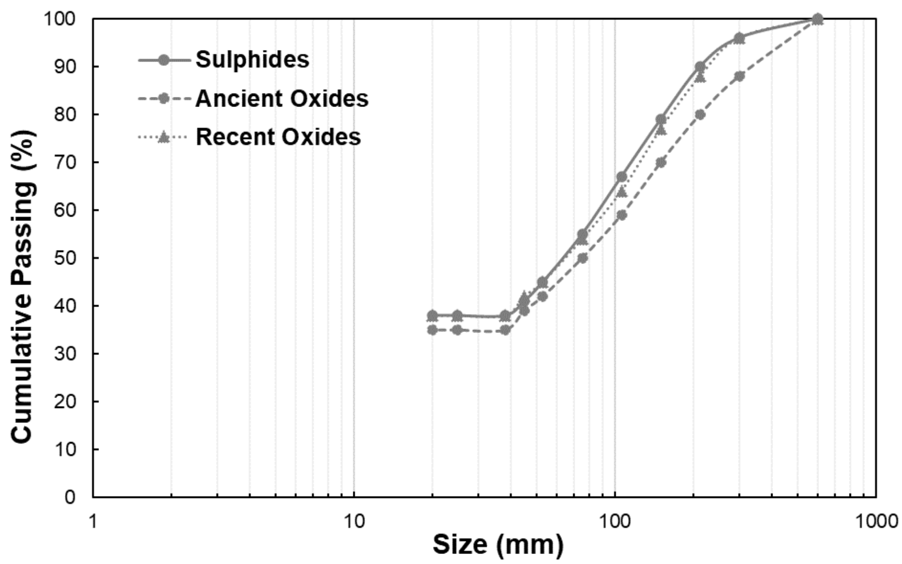


Figure 13-1: Particle Size Distribution of the three composite samples

13.2.1.5 Bulk Leach Extractable Gold and Silver (BLEG)

Bulk Leach Extractable Gold and Silver (BLEG) is a cyanide-based partial leach procedure. This procedure is performed in a mini leach with hot cyanide (25 grams sample), digestion was performed for a time of one hour. The resultant cyanide leach solution is measured by AAS or ICP.

This procedure was performed on the original sample and on each of the fractions of the size distribution, using the original size sample to know its extractions fraction by fraction. Table 13-4 provides the BLEG extraction results for head sample and size by size fraction for three composite samples.

Table 13-4: BLEG Extraction Results for Three Composite Samples

Size (µm)	Au (%)	Ag (%)	Cu (%)	Fe (%)	Zn (%)
Sulphides					
Head	83.3	85.3	37.8	0	6.5
150	100	65	36.8	0.1	5.1
75	95.2	30.9	30.7	0	1.8
45	74.1	43	31.1	0	1.6
38	66.7	43	29.6	0	1.9
20	82.5	50.7	27.1	0	2.1

table continues…

Size (µm)	Au (%)	Ag (%)	Cu (%)	Fe (%)	Zn (%)
-20	83.3	69.4	30.5	0	5.3
Ancient Oxides					
Head	84	76.1	18.1	0	3.1
150	95.6	94.1	20.6	0	3.8
75	97.3	74.8	20.9	0	4
45	70.4	54.7	15.3	0	0.9
38	86.5	62.8	13.7	0	0.9
20	87.2	70.3	14.9	0	1
-20	99.6	90.3	20.5	0	3.7
Recent Oxides					
Head	100	68.8	42.5	0	8.1
150	100	61.9	52.1	0	6.3
75	93.3	69.1	54.4	0	5.9
45	63.8	56.4	42.9	0	2
38	97.3	49.8	34.1	0	1.9
20	76.5	58.3	38	0	2.3
-20	100	75.1	38.9	0	5

For gold, the best extractions were observed for particle size fractions of +150, -150 to +75, -38 to +20, and -20 microns, applicable to both the ancient oxides and sulphides. The extractions for these particle size fractions were greater than 95.6% and 82.5% for ancient oxides and sulphides, respectively. Conversely, recent oxides showed gold extraction higher than 93.3% for the particle size fractions of +150, -150 to +75, -75 to +38, and -20 microns.

For silver, extractions tend to be lower than gold, this could be attributed to a short leaching residence time. The silver extractions exhibit variability, ranging from 50.7% to 69.4% for sulphides, 62.8% to 94.1% for ancient oxides, and 49.8% to 75.1% for recent oxides.

13.2.1.6 Gravity Recoverable Gold (GRG) Test

The GRG content provides a theoretical quantitative assessment of gravity-recoverable gold in a sample. As the feed sample comprises tailings material, the GRG test was conducted using two stages of centrifugal gravity concentration on a 10-kg sample. For the first stage, the GRG test was conducted on as received tailings sample without grinding. For the second stage, the sample was ground to 106 µm for sulphides and ancient oxides, while the sample was ground to 75 µm for recent oxides.

The gold recovery in concentrates was 5.6%, 3.7% and 3.5% for sulphides, ancient oxides, and recent oxides, respectively. For silver, the recovery was 1.7%, 1.7% and 1.2%. The average gravity concentrate grades were low, only 6.2 g/t Au and 169 g/t Ag for the sulphides sample, 6.1 g/t Au and 639 g/t Ag for the Ancient Oxide sample and 7.0 g/t Au and 185 g/t Ag for the Recent Oxides sample.

Due to the low recoveries and concentrate grades for both gold and silver, it appears that that the samples tested were not amenable to gravity concentration.

13.2.1.7 Bulk Flotation

Preliminary bulk flotation testing was conducted at three different particle sizes (as-received, 80% passing 105 µm and 75 µm) and different flotation time to identify optimum particle size and flotation time. Based on the results of four preliminary tests, the particle size of 80% passing 75 µm and flotation time of 4 to 5 minutes were identified to provide the highest recovery for gold and silver.

Using the preliminary test results, second series of flotation test was carried out on reagent type and dosage to maximize the gold and silver recovery. Table 13-5 lists the bulk flotation conditions used in the test work for the three composite samples.

Table 13-5: Bulk Flotation Test Conditions Adopted for the three composite samples

Stage	A31	PAX	A-3418	1065	pH	Air Flow	Time
	g/t	g/t	g/t	g/t		L/min	min
Sulphides							
Grinding	10				6.6		
Conditioning 1		30	20		6.8		3
Rougher #1				10		6	5
Conditioning 2							2
Rougher #2	7				7.0	5	1
Ancient Oxides							
Grinding	10				6.6		
Conditioning 1		30	20		6.8		3
Rougher #1				10		4	5
Conditioning 2							2
Rougher #2	7				7.0	4	1
Recent Oxides							
Grinding	10				6.6		
Conditioning 1		30	20		6.8		3
Rougher #1				10		4	5
Conditioning 2							2
Rougher #2	7				7.0	4	1

Note: Reagent A-3418 was not added in Test #5

The bulk flotation test results for Test #5 and Test #6 are illustrated in Figure 13-2: Bulk Flotation Results for Three Composite Samples. Table 13-6 lists the stage wise yield and recovery for gold, silver, copper, lead, zinc and iron. The gold recovery ranging from 43.7% to 45.8% for sulphides, 41.4% to 46.3% for ancient oxides, and 45.7% to 47.6% for recent oxides. While for silver, it varied from 46.5% to 46.9% for sulphides, 39.7% to 45.5% for ancient oxides, and 40% to 45.5% for recent oxides.

Table 13-6: Stage Wise Bulk Flotation Recovery for Test #5 and Test #6

Test ID	Product	Weight %	R/C	Au (%)	Ag (%)	Cu (%)	Fe (%)	Pb (%)	Zn (%)
Sulphides									
Test #5	Rougher #1	5.1	20	40.2	40.7	30.9	9	8.6	30.2
	Rougher #2	1.9	53	5.6	5.8	1.5	2.1	0.8	0.8
	Rougher 1+2	6.9	14	45.8	46.5	32.5	11.1	9.4	31.0
	Tailings	93.1		54.2	53.5	67.5	88.9	90.6	69.0
Test #6	Rougher #1	6.3	16	40.7	43.4	65.7	15.3	11.9	31.2
	Rougher #2	1.5	67	3.0	3.4	4.0	3.1	2.0	5.7
	Rougher 1+2	7.8	13	43.7	46.9	39.8	18.4	14.0	36.8
	Tailings	92.2		56.3	53.1	60.2	81.6	86.0	63.2
Ancient Oxides									
Test #5	Rougher #1	6.7	15	37.5	33.2	15.7	8.8	9	12.5
	Rougher #2	2.7	38	3.9	6.5	4.2	1.5	3.1	1.7
	Rougher 1+2	9.4	11	41.4	39.7	19.9	10.3	12.1	14.2
	Tailings	90.6		58.6	60.3	80.1	89.7	87.9	85.8
Test #6	Rougher #1	12.2	8	42.4	40.8	22.4	14.5	14.8	18.5
	Rougher #2	2.9	35	3.9	4.7	3.2	3	3.1	2.8
	Rougher 1+2	15.1	7	46.3	45.5	25.6	17.5	17.9	21.3
	Tailings	84.9		53.7	54.5	74.4	82.5	82.1	78.7
Recent Oxides									
Test #5	Rougher #1	7.9	13	45.7	40	36.1	15.2	12.2	23.6
	Tailings	92.1		54.3	60	63.9	84.8	87.8	76.4
Test #6	Rougher #1	7.8	13	44.8	40.4	39.2	17.9	12.3	23.9
	Rougher #2	1.8	56	2.8	5.1	4.6	3	2.1	1.8
	Rougher 1+2	9.6	10	47.6	45.5	43.8	20.9	14.4	25.8
	Tailings	90.4		52.4	54.5	56.2	79.1	85.6	74.2

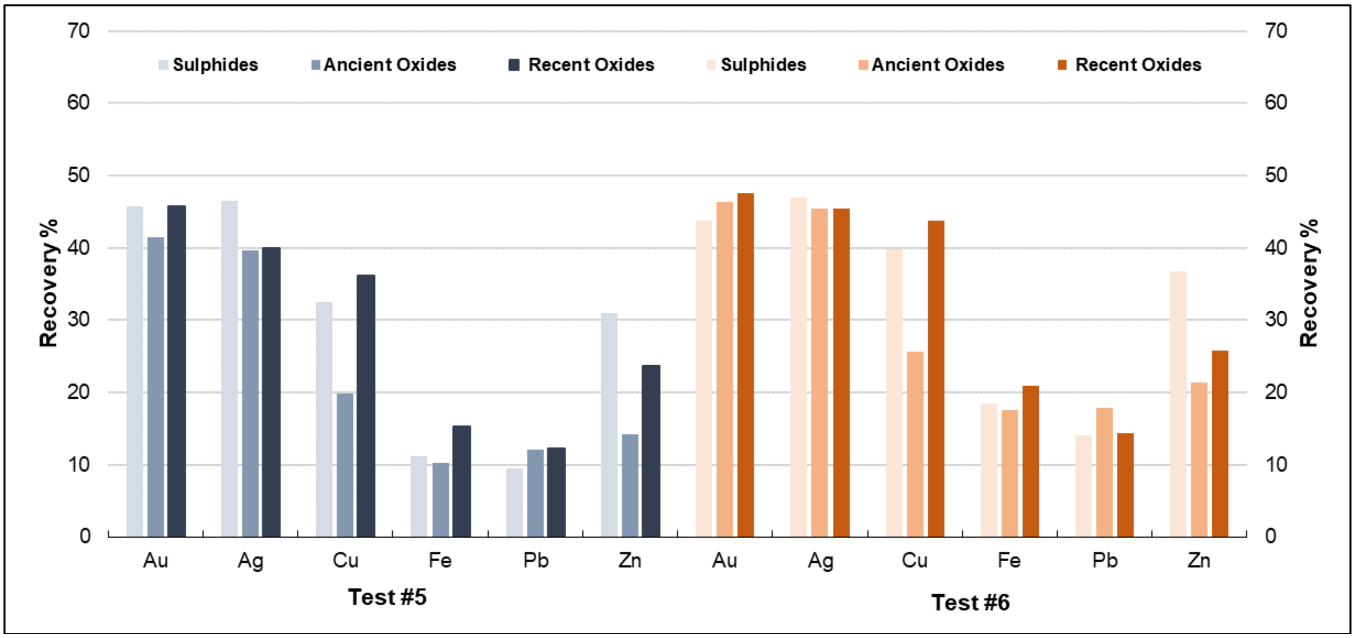


Figure 13-2: Bulk Flotation Results for Three Composite Samples

13.2.1.8 Bottle Roll Tests

Preliminary bottle-roll testing was conducted at various different test conditions, including particle size (as-received, 80% passing 105 µm and 80% passing 75 µm), cyanide concentration, addition of oxidizing agents to identify the best condition for extraction kinetics test. Based on the results obtained from the preliminary test, extraction kinetics test was carried out for 96 hours.

Preliminary bottle roll test was carried out using approximately 1 kg sample in a 2-gallon Nalgene bottle for 96 hours at pH of 10.5-11.5 with 33% w/w solids. At a regular interval, solution sample was collected to assess the metal extraction rate, sodium cyanide consumption and calcium oxide use.

Figure 13-3: Effect of Particle Size on Gold and Silver Extraction for Three Composite Samples shows the effect of particle size on gold and silver extraction for the three different composite samples using cyanide concentration of 1000 ppm. The gold extractions ranging from 77.0% to 82.8% for sulphides, 78.7% to 88.3% for ancient oxides, and 76.6% to 83.0% for recent oxides were determined. For silver, it varied from 69.1% to 76.1% for sulphides, 82.7% to 90.4% for ancient oxides, and 77.9% to 83.4% for recent oxides.

The highest extraction for all the three composite sample was achieved at a particle size of 75 µm with low extractions of other gangue elements such as iron and zinc.

The effect of cyanide concentration on gold and silver extraction was also investigated using the bottle roll test procedure at three different cyanide concentrations of 500 ppm, 1000 ppm and 1500 ppm at a particle size of 75 µm. Based on the results, it was concluded that the best cyanide concentrations of 500 ppm would be used for further test work for all three different composite samples.

Additional bottle roll test performed with adding leaching aid agents, such as lead nitrate and oxygen, did not yield any better performance in terms of extraction for any given sample. Hence, it was not further included in the test work.

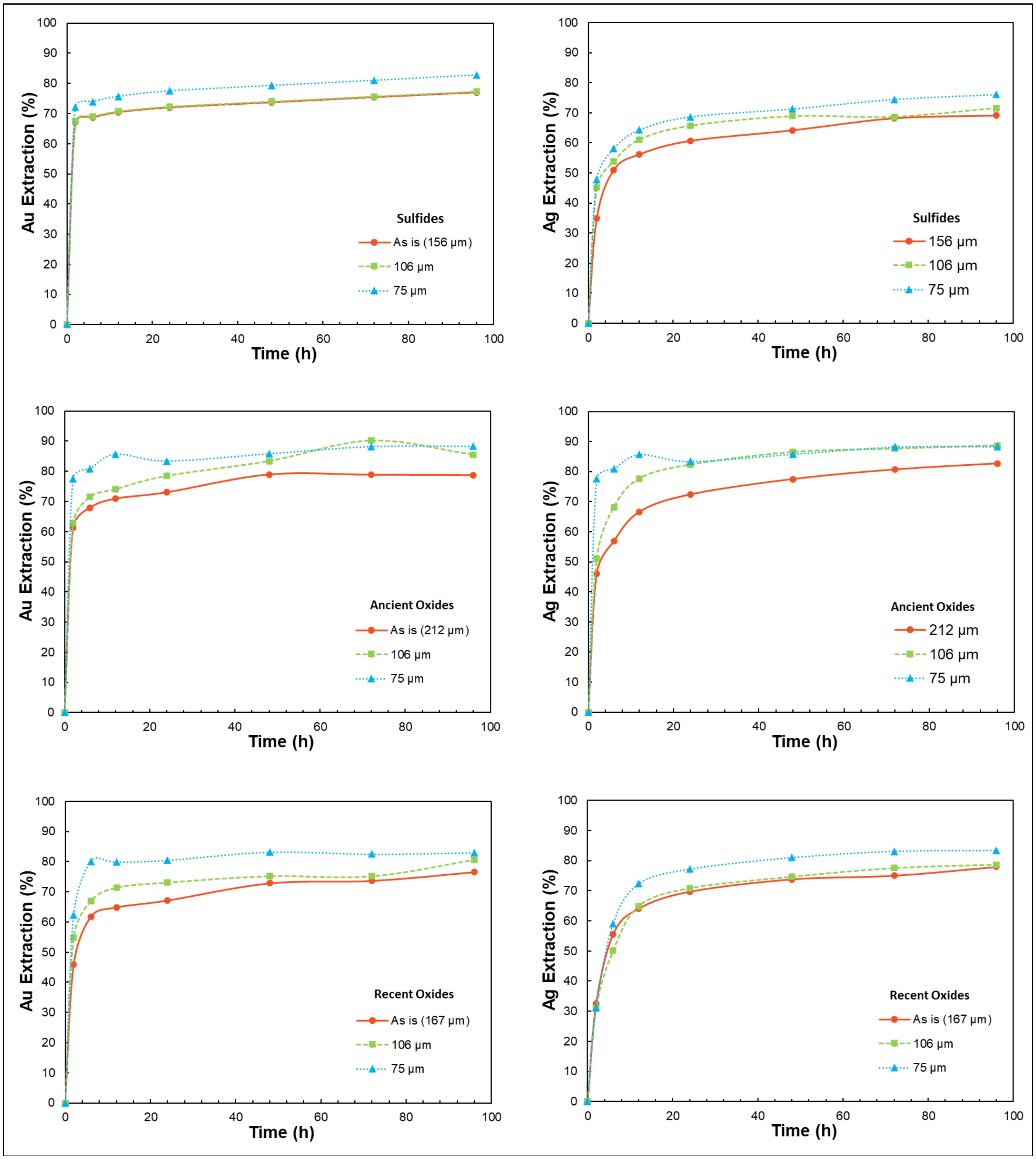


Figure 13-3: Effect of Particle Size on Gold and Silver Extraction for Three Composite Samples

The extraction kinetics of gold and silver for the three different composite samples at cyanide concentration of 500 ppm and an 80% passing 75 µm particle size is shown in Figure 13-4. The highest extraction for gold is achieved within 48 hours of leaching retention time, whereas for silver up to 60 hours of leaching retention time is required.

The optimal gold extraction rates obtained in the kinetics test were 78.5%, 84.6% and 82.8% for sulphides, ancient oxides and recent oxides, respectively. While Silver exhibited extraction rates of 68.6%, 85.4%, and 73.6%.

Additionally, a comparative test was also performed to improve the dynamic leaching results by using a viscosity reducing agent (P-625) with the established test conditions for kinetics test. The results indicated that viscosity reducer did not improve the gold or silver extraction. Hence, it was not further tested.

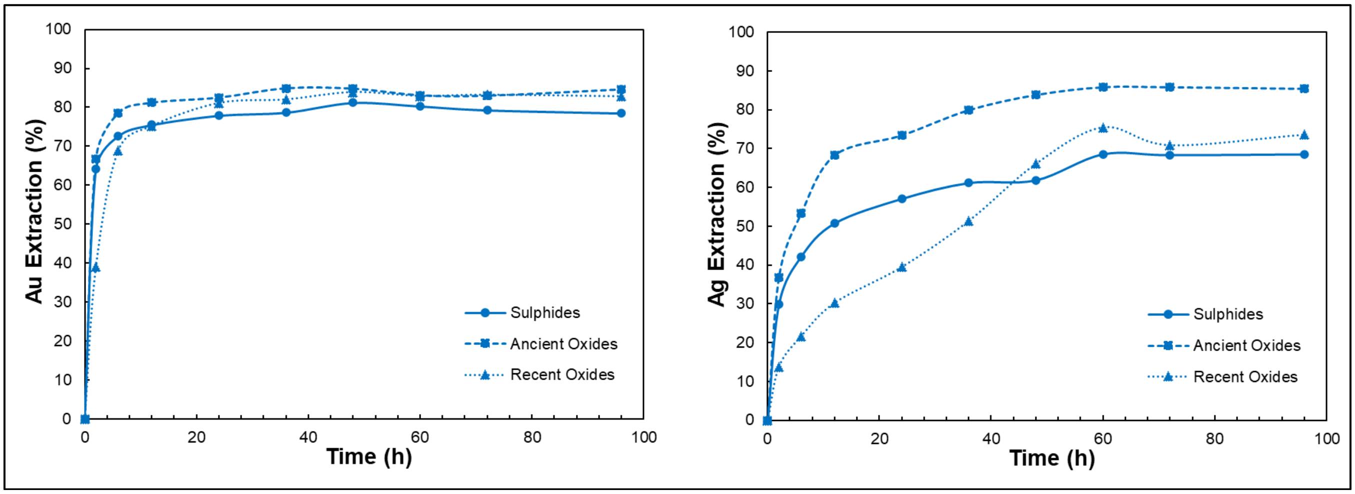


Figure 13-4: Extraction Kinetics of Gold and Silver for Three Composite Samples

13.2.1.9 Column Tests

The evaluation of gold and silver extraction using column test was performed with as-received particle size for three different composites. The column was operated in an open circuit with fresh solution being replenished daily in the column. Table 13-7 provides the test conditions used for the column test for all the three composite samples. The solution sample was assayed every day until day 14 to record extraction data. After day 14, a composite solution, derived from a 7-day period, was used to gather extraction data.

Figure 13-5: Extraction Kinetics of gold and silver for three composite samples in the column test depicts the column extraction of gold and silver for 100 days of column operation. For gold, extraction reached the maximum of 74 to 76% for the three composite samples. Whereas, for silver, extractions tend to be varied for different sample type. The maximum extraction achieved for silver were 88.8% for sulphides, 77.3% for ancient oxides, and 63.7% for recent oxides.

For gold, the maximum extraction was achieved by day 28, 49 and 42 for sulphides, ancient oxides and recent oxides, respectively. Whereas silver exhibited the maximum extraction around 90 to 100 days.

Table 13-7: Column Test Conditions Adopted for Three Composite Samples

Parameter	Conditions
Particle Size	As-received
Sample Weight	50 kg
Column Diameter	6 inches
NaCN Concentration	500 ppm
Irrigation Rate	10 L/m^2/h
Days of Operation	100

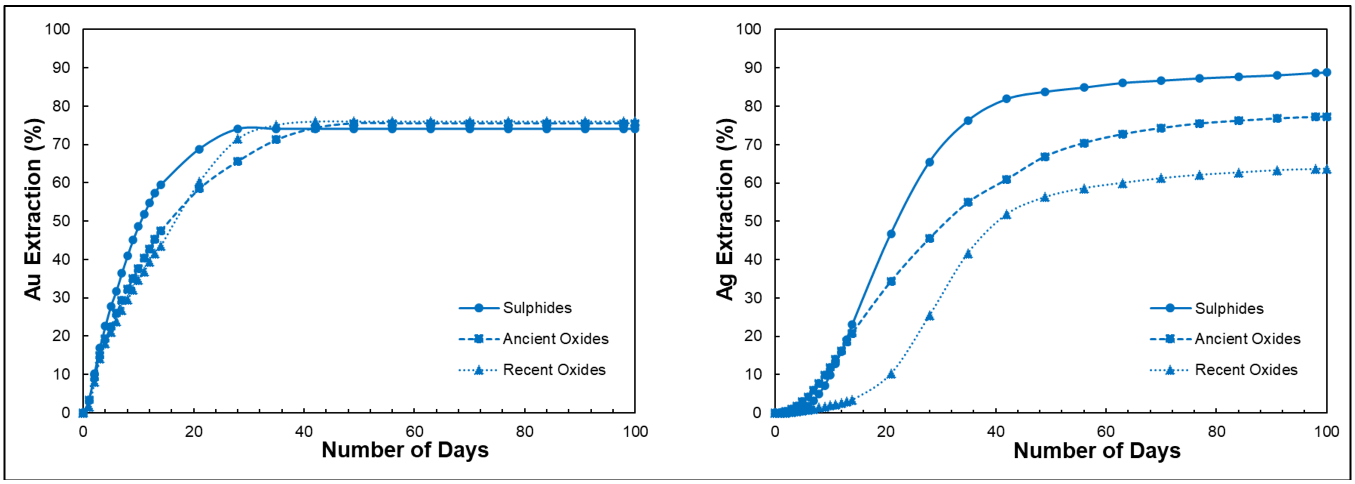


Figure 13-5: Extraction Kinetics of Gold and Silver for Three Composite Samples in the Column Test

13.2.1.10 Merrill Crowe Precipitation Test

Preliminary batch gold and silver precipitation testing was conducted on a composite solution sample generated from representative samples of sulphides, ancient oxides, and recent oxides. The effect of zinc concentration on the precipitation efficiency of gold and silver was evaluated with and without the addition of lead nitrate, in the ratio of 4:1 (Zn:$Pb(NO_3)$), in a series of test.

Based on the preliminary test results, using a stoichiometric ratio of 15 for zinc with the addition of lead nitrate in the ratio of 4:1 (Zn:$Pb(NO_3)$), the precipitation potential of gold and silver from the solutions produced from the individual composite samples (sulphides, ancient oxides, and recent oxides) was determined. The precipitation efficiency ranged between 79% and 94% for silver and between 70% and 88% for gold. It was observed that increasing cyanide or lead nitrate did not enhance the precipitation efficiency.

13.2.1.11 Cyanide Destruction

Cyanide destruction tests were performed in a slurry and on the filtered cake to determine the performance and required sodium hypochlorite consumption.

Table 13-8 lists cyanide destruction results on the slurry sample. The free cyanide of less than 25 ppm can be achieved within 30 minutes. Approximately 2 m^3/t of 5% sodium hypochlorite would be required for processing 100 t/d. The cyanide destruction behaviour was similar for all the three different composite samples.

Table 13-8: Cyanide Destruction on Slurry Sample

Sample	5% NaOCl (mL)	Agitation Time (min)	RPM	Feed CNfree (ppm)	% Solids (%)	Sample Weight (g)	Solution (mL)	Final CNfree (ppm)
Sulphides	3	30	200	350	45	500	611	175
	5	30	200	350	45	500	611	100
	10	30	200	350	45	500	611	<25
Ancient Oxides	3	3	200	350	45	500	611	175
	5	30	200	350	45	500	611	100
	10	30	200	350	45	500	611	<25
Recent Oxides	3	30	200	350	45	500	611	200
	5	30	200	350	45	500	611	150
	10	30	200	350	45	500	611	<25

Table 13-9 lists cyanide destruction results on the filtered cake sample. The free cyanide of less than 25 ppm can be achieved within 30 minutes. Approximately 0.5 m^3/t of 5% sodium hypochlorite would be required for processing 100 t/d. These test works are preliminary in nature and recommended to carry out further test work to optimize cyanide destruction and tailings treatment, including using SO2 procedure.

Table 13-9: Cyanide Destruction on Filtered Cake

Sample	Filtered Cake				CN Destruction		Cake after Cyanide Destruction
	Sample Weight	Moisture	Solution	CNfree	5% NaOCl	Time	Final CNfree
	(g)	(%)	(mL)	(ppm)	(mL)	(h)	(ppm)
Sulphides	1000	24	316	350	10	0.5	<25
	1000	24	316	350	10	1	<25
	1000	24	316	350	10	3	<25
Ancient Oxides	1000	24	316	350	10	3	<25
	1000	24	316	350	5	0.5	<25
	1000	24	316	350	3	0.5	100
	1000	24	316	350	1	0.5	250
Recent Oxides	1000	24	316	350	10	3	<25

13.2.2 SGS Attrition and Filtration Test Work 2023

In 2023, SGS also has conducted test work on Attrition and solid-liquid separation for five samples named ancient oxides, recent oxides, sulphides, agglomerates ancient oxides, and fresh flotation tailings from the current operation. These tests are performed to understand the settling rate, filtration rate and attrition performance of the samples.

13.2.2.1 Sample

Approximately 15-20 kg of each composite sample named ancient oxides, recent oxides, sulphides, agglomerates ancient oxides and current flotation tailings were sent to SGS for this study.

Table 13-10: Sample Description and Weight Shipped to SGS for the Test Work

Sample	Weight (kg)
Sulphides	18.0
Ancient Oxides	18.4
Recent Oxides	18.0
Agglomerates Ancient Oxides	15.9
Current Flotation Tailings	21.4
Total	**91.7**

13.2.2.2 Attrition Test

Six attrition tests were conducted with varying conditions as listed in Table 13-11. A sample 1.5 kg was used for each attrition test.

Table 13-11: Test Conditions used for Attrition Test

Test ID	Test Conditions		
	Percent Solids (%)	Time (min)	RPM
1	65	1.5	500
2	65	1.0	600
3	65	1.0	600
4	60	1.0	500
5	60	1.0	500
6	70	1.0	600

The particle size distribution of the sample before and after attrition test are presented in Figure 13-7 and Figure 13-8, respectively.

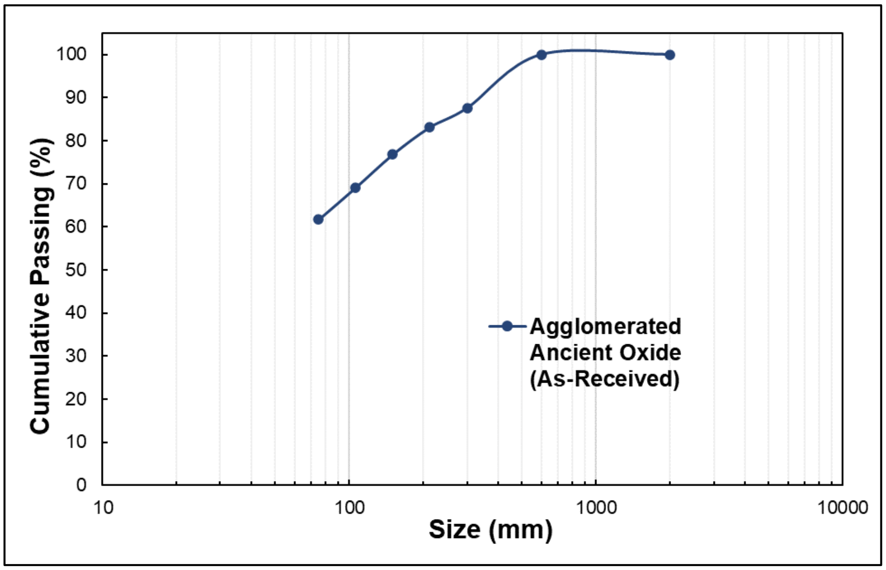


Figure 13-6: Particle Size Distribution of As-Received Agglomerated Ancient Oxide

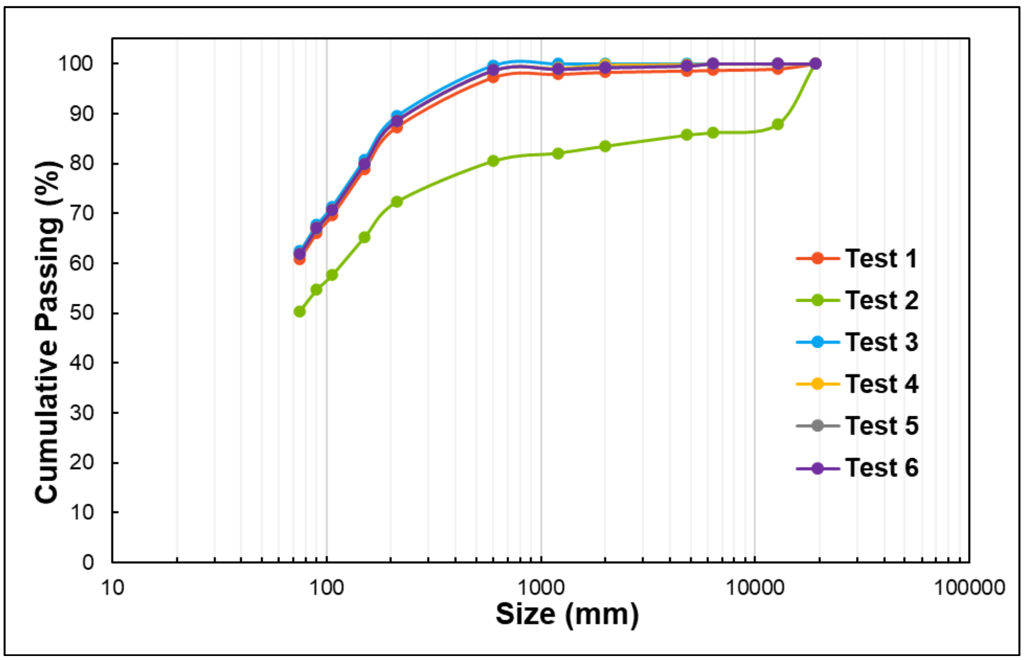


Figure 13-7: Particle Size Distribution of Attrition Test Products

The test results indicated that agglomerates were successfully destroyed with one minute of attritional time and particle size of agglomerates after attrition test is shown in Figure 13-8. It was found that solid content in range of 60-65% found to be the optimal condition and increasing solid content further leads to higher viscosity and was detrimental to attrition effect.

13.2.2.3 Settling Test

The natural settling rate of the four samples (sulphides, ancient oxides, recent oxides, and composite) are determined using settling test on different particle sizes (As received, 105 µm, 75 µm). The effect of flocculant addition on settling rate also performed on the select few samples.

The test conditions and results of settling tests are presented Table 13-12. The settling rate of the as-received sample was approximately 3 ft/h. whereas for the finer particle size of 106 and 75 µm, the settling rate decreased, lying within the range of 1.8 to 2.6 ft/h.

Table 13-12: Settling Test Conditions and Results for Different Samples

Test ID	Sample	Particle Size	pH	Flocculant Addition	Rate			Percent Solids	
		(µm)		(g/t)	mm/seg	m/h	ft/h	Initial (%)	Final (%)
1	Sulphides	156 (As Received)	10.7		0.29	1.04	3.4	20	63.2
2		106	10.7		0.19	0.69	2.28	20	58.3
3		75	10.7		0.21	0.75	2.47	20	60.5
4		75	10.7	5	0.40	1.44	4.72	20	52.4
5	Ancient Oxides	212 (As Received)	10.7		0.28	1.00	3.28	20	60.4
6		106	10.7		0.19	0.68	2.21	20	55.4
7		75	10.7		0.17	0.62	2.04	20	57.2
8		75	10.7	5	0.20	0.72	2.36	20	50.6
9		75	10.7	10	0.25	0.90	2.95	20	53.8
10	Recent Oxides	167 (As Received)	10.7		0.32	1.14	3.73	20	60.4
11		106	10.7		0.22	0.80	2.63	20	57.9
12		75	10.7		0.19	0.69	2.25	20	65.0
13		75	10.7	5	0.30	1.08	3.54	20	49.8
14	Composite (Sulphides, Ancient Oxides & Recent Oxides)	178 (As Received)	10.7		0.27	0.98	3.22	20	59.7
15		106	10.7		0.20	0.74	2.42	20	57.9
16		75	10.7		0.15	0.55	1.81	20	57.2
17		75	10.7	5	0.20	0.72	2.36	20	54.1

13.2.2.4 Filtration Test

The standard pressure filtration test was carried out on five samples (sulphides, ancient oxides, recent oxides, composite and current flotation tailings) to understand the dewatering performance by filtration.

The test conditions and results of filtration tests are presented Table 13-13.

The filtration rates of the ancient oxides sample range from 0.6 to 1.0 t/m²/hr, and it is noteworthy that this sample likely contains a substantial amount of clay, resulting in a low filtration rate with a coarse size. However, regrinding the clays has a mitigating effect on filtration. The recent oxide sample exhibits slightly higher filtration rates, ranging from 1.5 to 1.9 t/m²/hr.

Comparatively, the sulphide sample demonstrates higher filtration rates than the oxide samples but lower rates than the current tailings sample, with values ranging from 1.8 to 2.5 t/m²/hr. The current tailings exhibit higher filtration rates, ranging from 3.6 to 3.8 t/m²/hr.

Table 13-13: Pressure Filtration Test Conditions and Results for Different Samples

Test ID	Sample	Particle Size	Percent Solid	Pressure	Cake Thickness	Moisture	Filtration Rate
		(µm)	(%)	(psi)	(in)	(%)	t/m²/h
1	Ancient Oxides	212 (As Received)	60	40	1.8	18.2	0.6
2		106	60	40	1.8	20.0	0.9
3		75	60	40	1.7	20.2	0.6
4		75	60	80	1.7	18.1	1.0
5	Recent Oxides	167 (As Received)	60	40	1.7	18.5	1.9
6		106	60	40	1.8	19.0	1.9
7		75	60	40	1.8	21.2	1.5
8	Sulphides	156 (As Received)	60	40	1.7	18.6	2.5
9		106	60	40	1.7	18.9	2.5
10		75	60	40	1.8	21.6	1.8
11	Composite (Sulphides, Ancient Oxides & Recent Oxides)	175 (As Received)	60	40	1.7	16.9	1.4
12		106	60	40	1.8	17.3	1.7
13		75	60	40	1.9	19.2	1.3
14	Current Flotation Tailings	149 (As Received)	60	40	1.7	18.9	3.8
15		106	60	40	1.7	18.7	3.7
16		75	60	40	1.9	19.3	3.6
17		75	60	80	1.9	17.5	7.4

13.2.3 Historical Metallurgical Test Results – Oxide Tailings

The potential for processing the oxide tailings resource from previous operations was also studied by previous test programs. The subsections summarize the metallurgical characterization obtained from the previous test work and the study conducted by MMI in 2005 (Slim 2005c). MMI's report used the metallurgical results and conclusions drawn by PRA (Huang, 2003; Huang and Tan 2005). The revised and final report by MMI was dated October 2005 (Slim 2005d).

This Section is extracted from the 2012 and 2017 Technical Reports (Tetra Tech 2012, 2017), with some minor modifications to summarize the findings of the metallurgical test programs conducted so far.

Several metallurgical evaluations have been completed on various samples from the oxide TMF, according to the MMI report produced in 2003 (Slim 2003). The first cyanidation tests were conducted in 1982, followed by further tests performed over the years. The summarized cyanidation test results are shown in Table 13-14, taken from the 2003 MMI report (Slim 2003), while the reported flotation test results are given in Table 13-15 (Slim 2003). The results obtained from the test work program initiated by MMI in 2003 and 2004 were reported in the MMI 2005 technical report (Slim 2005d) and are included in Table 13-14 for comparison purposes. The results will be discussed in greater detail later in this section.

Table 13-14: Cyanidation Test Results (Slim 2003)

Author	Date of Test	Extraction (%)		Leaching Time (h)	Particle Size (µm)
		Ag	Au		
Denver Equipment	1982	69.3	66.7	24	66.6% < 149
Penoles	1987	78.3	88.9	24	87% < 74
Maja	1990	85.9	80.9	24	100% < 105
Chryssoulis	1990	85.9	80.9	24	no data
Rosales	1996	83.9	76.9	23	75% < 74
MMI	2003	77.1	71.4	24	86% < 74
MMI	2003	88.8	88.4	48	86% < 74

Table 13-15: Flotation Test Results (Slim 2003)

Author	Date of Test	Recovery (%)		Particle Size (µm)
		Ag	Au	
Penoles	1987	60.2	47.1	87% < 74
Rosales	1996	69.4	66.9	75% < 74

For the tests outlined in Table 13-14 and Table 13-15, no details have been provided regarding:

- The location or the manner in which the samples were taken
- Why these particular samples were taken
- The test parameters employed
- The assay techniques used, etc.

The first set of results for tests conducted on MMI samples from the 2003 sampling campaign indicates a silver extraction of 77.1% and gold extraction of 71.4%. However, these results cannot be verified since the origin of this set of numbers, as quoted in the MMI technical report (Slim 2005d), is not known. The second set of results was reported in the 2003 PRA report (Huang 2003). Considered in general terms, it would appear that the cyanidation test results were reasonably consistent over the indicated period. However, no specific conclusions should be drawn, since nothing is known about the head grades of the samples, the samples used, or the test and assay procedures used at the time that these tests were conducted.

The flotation results vary widely for similar particle sizes, with recoveries ranging from 60% to 69% for silver and 47% to 67% for gold. However, the test details of these reported cyanidation and flotation tests are unknown.

13.2.3.1 The MMI Technical Reports

Avino commissioned MMI to produce a document that was NI 43-101 compliant with respect to detailing the indicated oxide tailings resource (subsequently referred to as an Inferred Resource) and to define the metallurgical characterization and assay results for this material.

The first report prepared by MMI was titled "Tailings Valuation" and was dated November 2003 (Slim 2003). Two further reports by MMI titled "Preliminary Feasibility" (Slim 2005a) and "Tailings Valuation" (Slim 2005b) were produced in May 2005. The “Tailings Valuation” report (Slim 2005b) was subsequently revised and re-titled "A Tailings Resource" in July 2005 (Slim 2005c). This July 2005 MMI report (Slim 2005c) was reviewed by the Canadian Securities Administrators and returned to MMI for revision. The revised MMI report was re-issued as "A Tailings Resource" dated October 2005 (Slim 2005d) and was resubmitted to the CSRA for review. The October 2005 report (Slim 2005d) was produced for Avino Mines, Cia Minera Mexicana, Durango, Mexico, by Bryan Slim of MMI, North Vancouver, British Columbia, Canada. The document was submitted as a technical report to the CSRA.

PRA conducted two sets of test programs under the direction of MMI. One was conducted in 2003, for which no sample origin can be determined (Huang 2003), and the other, a more detailed test program, was conducted in 2004 (Huang and Tan 2005). The 2004 test work and the assaying program were designed and supervised by MMI. It was conducted on the samples collected from the tailings dam by MMI in 2004 while also using the results from the preliminary metallurgical scoping tests completed during 2003 as a guide. PRA staff at their facilities in Vancouver, British Columbia, conducted all the test work from both MMI test programs.

13.2.3.2 Introduction to the MMI 2003 Metallurgical Test Program

The 2003 test program consisted of the following tests, as summarized in Table 13-16. The cyanidation extraction results obtained were used in a preliminary report by MMI (Slim 2003). MMI considered using a 2,000 t/d vat leaching process to recover silver and gold from the oxide tailings; however, this treatment process option was revised when the results of the 2004 test program became available.

Table 13-16: Test Procedures – MMI 2003 Test Program (Slim 2003)

Process/Procedure	Details of Test	Sample Identify
Sample Preparation	No details documented	Sample L and Sample U
Head Assays	Fire assays, AA, and ICP multi-acid	Composite of L and U
Specific Gravity	Standard pycnometer test	Composite of L and U
Cyanidation Leach	P80 = 68 µm; 40% solids; pH 10.5; 1.0 g/L NaCN; 48 h; dO_2 > 7.9 mg/L 0.4 kg sample	Composite of L and U
Flotation	Rougher and two scavenger stages; P80 = 85 µm; 35% solids; pH 5.5; PAX & A208 with MIBC; 1 kg sample	Composite of L and U
Mineralogical	Examination of flotation tailings	Composite of L and U
Sample Preparation	No details documented	Sample L and Sample U

Notes: dO_2 = dissolved oxygen; PAX = potassium amyl xanthate; NaCN = sodium cyanide

The exact origin of Sample L and Sample U is not known and does not appear to have been documented. The manner in which each of the samples was collected by MMI has also not been documented. The size of both samples, 0.8 kg for Sample L and 0.9 kg for Sample U, is small, and their representation is questioned. Also, there appears to be no documentation relating to the arrival and receipt of these samples at PRA. There is no receiving log in PRA Report No. 0302303 (Huang 2003). Also, no assay certificates have been recovered to date. Even though these tests were considered to be scoping tests only, the results cannot be validated. Considering all the above factors, it is apparent that these results cannot be used with any degree of validity in reviewing process options for recovering silver and gold.

13.2.3.3 Introduction to the MMI 2004 Metallurgical Test Program

The 2004 test program was a better-structured program, which included the pre-concentration processes such as gravity concentration and flotation, both with and without regrinding, in an attempt to upgrade the material into a smaller mass for the subsequent treatment for the recovery of silver and gold. Also, cyanidation leach tests were conducted on as-received samples and reground samples to attempt to improve the liberation of silver and gold from the associated minerals. One column leach test was also performed.

Additional work completed included establishing the specific gravity and bulk density of the material, determining the Bond Mill Work Index on an oxide sample from the open pit, settling and filtration tests following cyanidation tests, and electrowinning tests using Electrometals electrowinning (EMEW) technology. All the different test procedures are summarized in Table 13-17.

Table 13-17: Test Procedures – MMI 2004 Test Program (Slim 2003)

Process/Procedure	Details of Test	Sample Identify
Sample Preparation	Individually numbered; dried; weighed; subsequently composited	Composites A, B, and C
Head Assays	Fire assays, AA and ICP multi-acid	Individual samples and Composites A, B, and C
Specific Gravity	Standard pycnometer test	Composites A, B, and C
Bulk Density	Standard volume displacement test	Composites A, B, and C
Mineralogical Examination	Examination of as-received samples	Selected Samples
Test Product Assays	Fire assays, AA and ICP multi-acid	All test Products
Bond Mill Work Index	Six cycles; closing screen size 150 µm	Oxide Sample
Size-assay Distribution	Screened and assayed the size fractions	Selected Samples
Gravity Concentration	Various test conditions	Composites A, B, and C
Cyanidation Leach	Various test conditions	Composites A, B, and C
Flotation	Various test conditions	Composites A, B, and C
Column Leach Test	Agglomerated feed; 81 d duration; 0.5 to 1.0 g/L NaCN; pH 10.5; 0.05 mL/s	Composite of A and B
EMEW	Various test conditions	PLS from Leach Test
ABA	Acid generation tests	Composites A, B, and C

The results obtained from this test program led MMI to include the heap leach process as the recommended treatment option in their report dated May 2005 (Slim 2005a).

13.2.3.4 Evaluation and Review of Metallurgical Tests (2004 Test Program)

Tetra Tech reviewed the metallurgical tests conducted during the MMI 2004 test program. The most promising process option should be selected as the recommended process treatment route based on the evaluation of the results obtained from the test program. This process option should then be evaluated with respect to capital and operating cost estimates. The process implications of the procedures and processes investigated and the results obtained are discussed in this section.

13.2.3.4.1 Sample Preparation and Characteristics

Bagged samples carrying the MMI identification tags were prepared at the Avino mine site under the direct supervision of MMI personnel. These samples were then transported from the mine site to Durango, Mexico, and shipped via airfreight to Vancouver, British Columbia. The samples were delivered to the PRA facility and unpacked in the presence of MMI personnel to ensure that no tampering had occurred to the samples en route. The samples were subsequently renumbered by MMI prior to PRA staff un-bagging and drying the samples. These details are shown on the PRA sample receiving log (Huang and Tan 2005). The individual samples were initially air-dried, followed by a low temperature of less than 50°C of oven drying.

The individual samples were subsequently homogenized, riffled, and split into four one-quarter fractions. One of these fractions was used for head assay determinations. A second fraction was used for compositing selected individual samples to create the sample Composite A, representing the oxide material of the lower bench of the tailings dam. Similarly, Composite B, representing the oxide material of the middle bench of the tailings dam, was prepared by compositing selected individual samples, as was Composite C, representing the sulphide tailings of the upper bench.

Although the samples had arrived at PRA from the Avino mine site without any indication of tampering, the sampling regime is considered deficient. First, the sampling of the oxide section of the tailings dam was incomplete. The sampling did not replicate the 1990 CMMA program, and certain parts of the tailings dam were not sampled. Second, the samples taken by MMI only represented the first 4 m of the depth of the tailings dam. Indications are, however, that the overall depth of the oxide section of the tailings dam varies between 7 m and 27 m. These two major deficiencies were also recognized by the Canadian Securities Administrators as deficiencies during their review. Both these items were addressed in the final MMI report dated October 2005 (Slim 2005d). The October 2005 report recommended a more detailed program of sampling of the whole tailings dam up to bedrock or ground soil level, as well as conducting metallurgical characterization tests using representative material from this more detailed sampling process whenever this is to be performed.

13.2.3.4.2 Moisture Content

The moisture contents of the samples, as received from the Avino mine site tailings dam, varied widely. A frequency distribution for moisture content of all the oxide tailings samples as received by PRA is given in Table 13-18. The bi-nodal distribution is apparent.

Table 13-18: Moisture Content of Samples

Frequency Distribution	
Moisture Content (%)	**Number**
5.00 - 7.50	9
7.51 - 10.00	14
10.01 - 12.50	19
12.51 - 15.00	16
15.01 - 17.50	5
17.51 - 20.00	5

These high moisture content values in the tailings dam confirm the high moisture content values found during the 1990 sampling program conducted by CMMA. Although the precise sampling procedure and drying conditions are unrecorded, a datasheet provided by Avino, as ostensibly related to this sampling program, provides assay values and moisture contents obtained during the program. The moisture values obtained varied from a low moisture value of 13.89% to a high value of 29.4% and a calculated average of 22.87% moisture. A possible reason for the high moisture content of the tailings material is that the mine was operational during this period when the sampling program was undertaken, i.e., 1990, and that routine tailings deposition was still in progress.

The specific reason for the relatively high moisture content found during the 2004 MMI sampling program is not apparent. The MMI technical report (Slim 2005d) has referred to the possibility of the original manner of tailings deposition, which has resulted in localized areas of high moisture content. Also, the presence of artesian springs under the tailings dam has been mentioned as a possible reason. It was also observed that any rainwater run-off from the higher levels above the tailings dam would collect at the head of the tailings dam and subsequently seep through the dam, exiting at the foot of the dam. Whatever the reason(s) may be, areas of high moisture content do exist and will influence the method of recovery of the tailings and the subsequent agglomeration process.

13.2.3.4.3 Head Assays and Test Products Assays

Gold assaying was completed using the standard fire assay procedure. Initially, the silver was also analyzed by the fire assay procedure followed by an AA spectrophotometric finish. However, this fire assay-based method for silver is not very accurate in the low concentration range of less than 100 g/t for silver. Assaying for silver was then done using ICP-MS, preceded by the total digestion of the sample in a suite of mineral acids. A further method was also investigated, namely that of total acid digestion followed by an AA finish. The results obtained with this acid digestion and AA method were similar to the ICP-MS. Therefore, the assay method selected for all the silver assays was the ICP-MS method, preceded by the total digestion of the sample in a suite of mineral acids (ICP-MS). All the other analyses for the various products arising from the metallurgical tests were done by the standard and universal methods using titration, ICP-MS, or AA methods.

All the various head sample analyses conducted during the test program are listed in Table 13-19. The reference to the test number relates to the stage of the test work that the sample was submitted for analysis. The average values for the four different composite samples tested, namely Composite A, Composite B, Composite C, and the Composite A + B blended sample, have all been calculated and are given in the table together with the respective standard deviation values. The standard deviation of the head samples representing Composite A and Composite B are shown to be within 10% of the deviation from the average value, which is considered reasonable.

However, the average silver value of all the head assay analyses assayed as head samples representing Composite A and Composite B is only 86.8 g/t silver. This average silver grade is less than the 95.5 g/t silver as given in the MMI technical report as being the overall silver grade of the material of the whole oxide tailings dam (Slim 2005d). Similarly, the average gold value of all the head assay analyses assayed as head samples representing both Composite A and Composite B (i.e., representing the oxide tailings dam) taken during the test work program is 0.44 g/t gold, which also is less than the 0.53 g/t gold, as quoted in the MMI technical report (Slim 2005d). For silver, this amounts to a difference of about 9% based on the MMI quoted head grade of 95.5 g/t silver, while for gold, the difference is larger at 17% based on the MMI quoted gold value of 0.53 g/t gold. It is of interest that the average head assay for the Composite A + B sample is closer to the calculated average from Composite A and for Composite B, namely 89.6 g/t compared with 86.8 g/t for silver and 0.41 g/t compared with 0.44 g/t for gold. The above discussion assumes that the tonnages of the tailings dam labelled Composite A (lower bench) will be mixed in equal proportion to the area of the tailings dam designated as Composite B (middle bench). In the absence of specific tailings dam volumes or tonnages, this assumption may be an oversimplification and may, therefore, not be entirely valid. However, the assay values for Composite B are lower than the overall average head grade of the tailings samples collected.

A further comment regarding the assay results above relates to the methods employed for the assaying techniques for silver from these samples. The MMI technical report (Slim 2005d) states that for the CMMA 1990 tailings drilling program, the silver assaying was completed using the standard mine practice of fire assay followed by acid digestion and AA finish. The PRA metallurgical test work program used multi-acid digestion followed by the ICP assay method for silver analyses. It is anticipated that there will not be a significant difference between the silver assays as reported in 1990 and those from the MMI test program as conducted by PRA, but the extent of this difference cannot be quantified in this review. Similarly, no comment can be given on the accuracy of the assays conducted by CMMA, since the standards of precision of sampling, sample preparation, and detailed methodology of the assaying methods are unknown. However, a summary sheet containing assay values has been provided by Avino as being the silver and gold grades obtained from the 1990 CMMA sampling program. No calculations have been performed using these assay values and are only included in this Technical Report since it is part of the CMMA sampling program. The MMI report (Slim 2005d) provides a grid map identifying the various sample holes.

Table 13-19: Head Assays

Test No.	Assays (g/t)		Test No.	Assays (g/t)	
	Ag	Au		Ag	Au
Composite A			**Composite B**		
SA9	99.8	0.37	SA10	88.3	0.55
Ave. 1	103.4	0.34	Ave. 1	82.6	0.68
Ave. 2	105.3	0.36	Ave. 2	88.4	0.51
C1	95.2	0.35	C4	76.3	0.52
C2	94.3	0.35	C5	70.6	0.49
C3	94.1	0.36	C6	71.4	0.50
C7	88.7	0.36	C9	70.3	0.52
C8	88.7	0.36	C10	70.3	0.52
C13	95.9	0.28	C15	77.2	0.49
C14	98.9	0.37	C16	78.3	0.52
C17	95.2	0.35	C18	77.2	0.49
Average Value	**96.3**	**0.35**	**Average Value**	**77.3**	**0.53**
Standard Deviation	**5.27**	**0.025**	**Standard Deviation**	**6.72**	**0.054**
Composite C			**Column Composite A + B**		
C11	39.8	0.34	C4	87.4	0.42
C12	39.8	0.34	C5	90.1	0.40
Ave. 1	31.7	0.29	C6	91.4	0.42
Ave. 2	39.8	0.39	C9	-	-
Average Value	**37.8**	**0.34**	**Average Value**	**89.6**	**0.41**
Standard Deviation	**4.05**	**0.041**	**Standard Deviation**	**2.04**	**0.012**

13.2.3.4.4 Mineralogical Evaluation

At the start of the 2004 metallurgical test program, MMI requested that a sample from some of the individual samples be submitted for mineralogical analysis. The mineralogical findings have not been reported in PRA Report No. 0406407 (Huang and Tan 2005) and were not alluded to in the MMI technical report (Slim 2005d) or any preceding reports. The reason(s) why these results have not been communicated to Avino or the test program investigators at PRA is unknown.

13.2.3.4.5 Bond Ball Mill Work Index

Although this information was not required to treat the oxide tailings dam material, a Bond Ball Mill Work Index determination test was done on an oxide material sample. The work index was determined to be 12.3 kWh/t using a closing screen size of 74 µm (200 mesh) with the convergence of the specific energy input (grams of product per revolution) found after five testing cycles. This makes the sample tested a moderately hard rock type. The details regarding the origin of this sample have not been documented, and its relevance as data is therefore questioned.

13.2.3.4.6 Bulk Density and Specific Gravity

Bulk density and specific gravity determinations were conducted on samples specifically identified by MMI. The specific gravity measurements were done using the standard pycnometric method, while the bulk density values were obtained by measuring the volume of dry solids in a measuring cylinder. The values obtained are reproduced in Table 13-20.

Table 13-20: Bulk Density and Specific Gravity

Location/Bench	Sample Identity	P80 Size (µm)	Bulk Density (g/cm3)	Specific Gravity
Upper Bench	S2	226	1.66	2.74
Lower Bench	S10	326	1.73	2.62
Lower Bench	S22	367	1.73	2.76
Middle Bench	S45	254	1.60	2.76
Middle Bench	S50	201	1.63	2.74
Upper Bench	S74	301	1.57	2.72
Average	-	-	1.65	2.72

The bulk density values determined for the oxide tailings material were found to vary between 1.57 g/cm^3 and 1.73 g/cm^3, with an average of 1.65 g/cm^3. The specific gravity values obtained were generally consistent, with an average value of 2.72.

13.2.3.4.7 Particle Size – Assay Analysis

A particle size–fraction analysis was done on the same samples used for the bulk density and specific gravity determinations. These tests were conducted to determine whether the silver and gold predominantly occurred in a particular particle size range. The size-assay analyses indicated that the metal distributions varied according to the location but that all displayed varying degrees of the bi-nodal distribution for silver, gold, and mass.

Sample S10 from Composite A from the Lower Bench of the tailings dam indicated one maximum metal distribution occurring in size range of 149 µm to 210 µm and another in the minus 37 µm size range. The maximum mass distributions are generally similar, although it occurs over a wider range in coarse size, namely 105 µm to 210 µm. The second sample from this bench, Sample S22, was similar but with a shifted maximum metal and mass distribution in the 210 µm to 297 µm size range and a secondary maximum metal and mass distribution in the minus 37 µm size range.

Sample S45 from the Middle Bench of the tailings dam, and part of Composite B, indicated maximum metal distribution in the 149 µm to 210 µm size range with maximum mass distribution in the 105 µm to 149 µm size range. The secondary maximum metal and mass distribution was found in the minus 37 µm size range. The second sample from the Middle Bench, Sample S50, had the maximum metal and mass distributions in the 105 µm to 149 µm size range and the minus 37 µm size range.

The two samples from the Upper Bench of the tailings dam of Composite C displayed totally different particle size distributions. Sample S2 was bi-nodal with one maximum for metal and mass distribution in the size range of 105 µm to 149 µm and the second maximum occurring for the size range of minus 37 µm. Sample S74 displayed only one maximum metal and mass distribution over the relatively wide coarse particle size range of 105 µm to 297 µm. This sample was almost entirely devoid of slimes or minus 37 µm material.

These samples reflect the operating discharge conditions and history during plant operations and tailings deposition. The results typify the use of a tailings cyclone situated on the tailings dam wall discharging the coarse undersize material onto the wall area with the finer cyclone overflow material flowing downstream and settling within the tailings dam. Changes in the size distribution would be anticipated with downstream distance from the point of discharge by the cyclones at the tailings dam wall. This is typified by the size distribution of Sample S74, which purports to be a cyclone underflow sample taken at the point of discharge and which was found to be almost totally devoid of fines or minus 37 µm material.

13.2.3.4.8 Gravity Concentration Tests

Pre-concentration tests using the centrifugal gravity concentration method were conducted to evaluate the potential upgrading of silver and gold. The laboratory-size concentrator used was the Falcon Model SB40 centrifugal concentrator. The tests were conducted on samples from Composites A, B, and C. MMI dictated the test parameters used for these tests, including a set of tests where the samples were reground prior to conducting the gravity concentration test. The results from the gravity concentration tests are summarized in Table 13-21.

The mass recoveries varied between 20% and 25%, indicating that the tests were performed in a uniform and consistent manner. The highest silver recovery obtained was 40% (after regrinding) for Composite C and decreased to 31% for Composite B (after regrinding) and about 27% for Composite A, also after regrinding. The gold recoveries were higher than the equivalent silver recoveries, particularly after regrinding, indicating that the liberation of the precious metals could be incomplete.

However, the upgrading factor for both silver and gold is very low, namely about 1.4 for silver and up to 2.3 for gold. No further upgrading or silver and gold recovery tests were conducted on the gravity concentrates, possibly due to the relatively low grades and recoveries obtained. Also of interest is that no historical test work was documented by MMI where gravity concentration was used to produce a saleable high-grade concentrate.

Table 13-21: Summary of Results of Gravity Concentration Tests

Sample Identity	Head Grade		Concentrate Grade		Recovery (%)			P80 (µm)	Remarks (Note: All tests are 3-pass tests)
	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Mass	Ag	Au		
Composite A	93.8	0.35	124.7	0.52	24.1	32.1	36.5	269	Pressure 1.5 psi; no regrind
Composite B	70.3	0.50	96.9	0.71	23.6	32.5	33.3	180	-
Composite C	39.7	0.33	58.0	0.65	24.1	35.2	47.0	254	-
Composite A	92.1	0.33	126.1	0.71	19.7	27.2	42.1	76	Pressure 1.0 psi; reground
Composite B	70.5	0.56	96.5	1.29	22.4	30.7	51.5	77	-
Composite C	40.7	0.38	65.5	0.98	24.8	39.9	64.3	79	-

13.2.3.4.9 Flotation

Different scoping flotation tests were conducted on samples from Composite A and Composite B using various reagent schemes and conditions as dictated by MMI. The results of the flotation tests are summarized in Table 13-22. The test results reported led to the following conclusions.

For Composite A, a regrind from a P80 size of 238 µm (as received particle size) to a P80 of 72 µm improved the flotation recovery of silver from 18% to 23% and that of gold from 18% to 39%. The standard reagents were used for these tests (Tests F1, F3, and F4). For Composite B, a regrind from a P80 size of 173 µm (as received particle size) to a P80 of 74 µm improved the flotation recovery of silver from 22% to 33% and that of gold from 12% to 32% (Tests F2, F5, and F6). A particle size fraction analysis conducted on the tailings of Test F4 (Composite A) indicated that the major proportion of the mass and the silver and gold is present in the slimes or minus 37 µm size fraction. However, significant losses of silver and particularly gold occurred in the coarser sizes, namely the size range 53 µm to 105 µm, indicating that the degree of liberation could be improved and that some metal appears to be occluded in the coarser particle sizes. Some silver may also occur within secondary oxide minerals and be unrecoverable by flotation. A similar mass and metal distribution were obtained in the case of Test F9 (also Composite A), which was a flotation test performed using a sulphidization reagent.

Variable mass, metal recoveries, and concentrate grades were obtained in testing the various flotation reagent suites. However, the maximum silver grade obtained for a rougher concentrate was 909 g/t silver, while the overall recoveries for silver could not be improved beyond approximately 40%. This indicated that mineral surface alteration or oxidation, or occlusion of precious metals in gangue, was inhibiting the concentration by the flotation process. Since the silver recoveries obtained were deemed low and unsatisfactory, no further flotation tests were conducted, and no extraction tests were performed on flotation concentrates.

The head assays obtained during the flotation testing stage gave inconsistent results. Table 13-22 shows the actual head assays obtained for each flotation test compared with the head assay obtained for silver for the composite samples. For Composite A, the individual silver head values for each flotation test conducted are all higher than the assay for the composite sample, except in the case of Test F11. The gold (and silver) values obtained for Tests F7, F8, and F9 are known to have resulted from the poor sampling technique adopted for these three tests. The composite head assay gold value of 0.36 g/t gold is probably a reasonably representative assay value for Composite A. For Composite B, the silver head value for the composite sample is slightly lower than the assays for the individual flotation tests. For gold, the composite sample value is higher at 0.52 g/t gold than the assays for the individual tests.

The historical results of the flotation tests reported in Table 13-15 are significantly higher at 60% to 69% recovery for silver and 47% to 67% for gold. However, in the absence of information regarding the origins of these samples, the lack of head grade data and the absence of sampling and flotation procedures involved, these results will not be considered when selecting the processing options for the oxide tailings dam material.

Table 13-22: Summary of Results of Flotation Tests

Sample Identity & Test No.	Head Grade		Concentrate Grade		Recovery (%)			P80 (µm)	Remarks (Note: All tests are 3-pass tests)
	Ag (g/t)	Au (g/t)	Ag (g/t)	Au (g/t)	Mass	Ag	Au		
Composite A/F1	112.2	0.35	908.7	3.17	2.1	17.8	18.4	238	3-stage ro., pH 8;
Composite A/F3	119.2	0.39	734.6	3.88	2.6	21.0	30.4	103	Conditioning NaCN
Composite A/F4	104.6	0.40	630.9	3.36	3.8	22.6	38.6	72	Na2CO3; A404, PAX
Composite A/F7	111.9	1.39	654.6	5.56	2.3	16.3	34.9	~75	2-stage ro., nil NaCN
Composite A/F8	108.5	2.38	887.2	11.91	0.9	7.8	30.7	~75	2-stage ro., nil NaCN
Composite A/F9	114.5	1.67	723.9	5.86	2.7	20.8	45	~75	2-stage ro., NaS2, PAX
Composite A/F10	103.5	0.58	401.3	1.62	8.9	34.6	39.8	~75	With Na2CO3, CuSO4
Composite A/F11	99.6	0.34	484.8	1.83	8.8	42.2	48.3	~75	With CuSO4, A208
Composite B/F2	88.4	0.42	695.4	2.65	2.6	22.0	12.2	173	3-stage ro., pH 8
Composite B/F5	89.7	0.47	806.1	4.18	2.9	27.0	24.6	92	Conditioning NaCN
Composite B/F6	89.9	0.51	867.1	5.45	2.9	32.5	32.1	74	Na2CO3; A404, PAX
Composite A: Head	99.8	0.36	-	-	-	-	-	-	-
Composite B: Head	88.3	0.52	-	-	-	-	-	-	-

Notes: $CuSO_4$ = copper sulphate; $NaCO_3$ = sodium carbonate, ro = Rougher

13.2.3.4.10 Cyanidation Tests

Cyanide leaching tests were conducted on samples from Composite A, Composite B, and Composite C using different leaching conditions. The first set of tests was to determine the effect of regrinding the tailings samples prior to leaching, while subsequent tests determined the effect of cyanide concentration in the leach solution.

For Composite A, the silver extractions varied from 66% for the unground (as received) sample to 80% for the reground samples, while the gold extractions varied from 82% to 89%, respectively. For Composite B, the silver extractions ranged between 69% for as-received material to 77% for samples that were reground. The corresponding gold extractions varied between 82% and 87%. Although the cyanide consumption increased with the regrinding of samples tested for both Composite A and Composite B, the increase in extraction may compensate for the additional cost of cyanide reagent and regrinding, provided that the filtration characteristics are not detrimentally affected. Higher cyanide concentrations in the leach solution improved the extractions of silver and gold but increased the cyanide consumption significantly. The results from the sulphide tailings, namely Composite C, indicate that between 73% and 87% of the silver can be extracted, with between 77% and 85% of the gold. However, the cyanide consumption values were higher than the results from the oxide tailings. Only two leach tests were conducted on reground samples from Composite C, each having a P80 of about 69 µm. A summary of the cyanide leach test results is given in Table 13-23.

Table 13-23: Summary of Results of PRA Cyanidation Tests

Sample Identity and Test No.	Extraction (%)		Reagent Usage (kg/t)		NaCN Concentration (g/L)	P80 (µm)
	Ag	Au	NaCN	Lime		
Composite A+/C1	66.4	81.5	1.8	1.4	1.0	269
Composite A+/C2	79.3	85.7	1.6	1.8	1.0	103
Composite A+/C3	80.4	89.1	2.6	1.6	1.0	78
Composite A+/C7	78.6	82.7	2.2	1.8	0.5	74
Composite A+/C8	89.7	85.5	5.1	0.8	2.0	74
Composite A*/C13	79.7	86.8	1.5	1.3	0.5	74
Composite A*/C14	83.1	82.1	3.7	0.8	2.0	74
Composite A*/C17	79.4	90.9	1.0	1.2	1.0	74
Composite B+/C4	69.1	82.0	2.6	1.8	1.0	180
Composite B+/C5	77.1	88.3	1.7	1.8	1.0	100
Composite B+/C6	77.3	86.9	1.7	1.9	1.0	84
Composite B+/C9	73.2	86.0	2.6	1.2	0.5	84
Composite B+/C10	79.5	86.4	4.5	1.0	2.0	84
Composite B*/C15	72.9	82.6	1.6	2.0	0.5	84
Composite B*/C16	75.4	83.4	3.8	1.0	2.0	84
Composite B*/C18	67.7	78.6	0.9	1.3	1.0	84
Composite C+/C11	73.8	77.3	4.0	2.8	1.0	69
Composite C+/C12	86.6	85.0	7.3	2.6	2.0	67

Notes: "+" indicates Original Composite Sample, "*" indicates New Composite Sample, Tests C17 and C18 = 24 h leach duration; other tests + 72 h leach duration.

During the cyanide leach test program, a new Composite A and Composite B sample had to be prepared since the original composite samples had been exhausted. A comparison of results from the two composite samples indicated similar behaviour patterns, although there are some noticeable differences in the extractions. Also, the cyanide and lime consumption values, as recorded, are inconsistent. This indicates that absolute numbers cannot be assigned to a single test, although any observed trends would be valid. The averages of similar tests would more likely predict the overall responses more accurately. It is also apparent that non-systematic variations in the assay results could have arisen from subtle variations in mineralogy, sample preparation, the sample regrinding process, and possibly daily variations in temperature.

The cyanide leach extraction results quoted by MMI in Table 13-23 and the averaged results from the present test program are summarized below in Table 13-24 and will be discussed in the following section.

The average extraction results obtained from samples from Composite A and Composite B in the present study are generally lower than those from the historical test work, as detailed in Table 13-24. However, in the absence of details, these historical results cannot be used in the overall evaluation of this process. The MMI claim of a 77% silver extraction, based on the MMI (2003) test program, cannot be considered an acceptable result since only one test was done. The sample origin is purported to be four holes dug at approximately 25 m intervals with samples scraped into a bag, one for the lower bench and one for the upper bench of the oxide tailings dam. Clearly, a sample collected in this manner cannot be considered representative. Also, the other MMI (2003) claim for an extraction result of 89% silver and 88% gold cannot be validated. Therefore, these test results cannot be considered valid and will not be used in further discussions or evaluations.

Table 13-24: Summary of Cyanidation Test Results Used by the MMI Reports

Sample Identity and Test No.	Extraction (%)		Remarks
	Ag	Au	
Composite A/C1	66	82	As received; 1.0 g/L NaCN
Composite A/C7 & C13	80	85	Average; reground; 0.5 g/L NaCN
Composite B/C4	69	82	As received; 1.0 g/L NaCN
Composite B/C9 & C15	73	84	Average; reground; 0.5 g/L NaCN
MMI 2003	77	71	Results from the 2003 test program
MMI 2003	88	88	Origin of results unrecorded
MMI 2004/C8 & C10	85	86	Average; reground; 2.0 g/L NaCN

The MMI (2004) results, as claimed in the technical report and listed in Table 13-24 above, are also considered unusable. The reasons for this statement are that these results were obtained with a reground sample and leached at a high cyanide concentration of 2.0 g/L sodium cyanide, whereas the other tests were done using 1.0 g/L sodium cyanide. Both these conditions, the regrinding of the tailings material and a high cyanide concentration leach condition, will not be implemented in a recovery process, and these results are considered unrealistic.

The extraction results from the cyanidation tests obtained using as-received samples from Composite A and Composite B, namely 66% to 69% for silver and 82% for gold, were encouraging.

13.2.3.4.11 Column Leach Test

One column leach test was conducted on a 30.9 kg sample of an equal mix of material from Composite A and Composite B. The sample was mixed with water, Portland Cement, and lime and then agglomerated to a P80 size of 2,614 µm. After curing, the sample was put into a column with a diameter of 102 mm and a height of 3 m. The column test was run for 81 d after the solution flow rate and pH had stabilized. The silver extraction obtained was 73.0%, while the gold extraction was 78.9%. These results compare very well to the average extraction values calculated from the cyanidation tests of the individual composite samples leached in the as-received condition, namely 67.8% for silver and 81.8% for gold. The cyanide consumption values are also comparable. The results obtained from the column test and the calculated average extraction values obtained from the tests conducted on the as-received samples of Composite A and Composite B have been summarized in Table 13-25.

The kinetics of leaching had slowed down significantly by Day 81 when the test was terminated, although there was evidence that some leaching was still in progress.

Table 13-25: Summary of Results of Column Leach Tests

Sample Identity and Test No.	Extraction (%)		Reagent Usage (kg/t)			NaCN Concentration (g/L)	P80 (µm)	Remarks
	Ag	Au	NaCN	Lime	Cement			
Column Test, Composites A and B	73.0	78.9	2.32	13.73	21.8	0.5 and 2.0	2,614	pH 11;
Composites A and B Average, Tests C1 and C4	67.8	81.8	2.18	1.59	-	1.0	225	pH 10.5/11; bottle roll

A particle size assay analysis of the leach residue of the column test found that the highest unleached (undissolved) silver grade was in the coarsest size range of plus 210 µm, while the highest gold value was found in the minus 37 µm size range. This suggests both inadequate liberation of the silver grains and/or minerals, occlusion of gold possibly by clay minerals, or the presence of tarnished/coated mineral surfaces, or the presence of refractory minerals. The subsequent leaching of de-agglomerated column leach test residue resulted in a negligible extraction of silver and gold. This indicates that the column leach test had virtually reached its maximum potential extraction, confirming that the leaching rate had slowed down.

Only one column leach test was conducted. Also, the material tested was a mixture of samples from Composite A and Composite B, that is, a mixture of material from the lower and the middle benches of the oxide tailings dam. Flow problems were encountered during the test, which resulted in the column having to be unloaded and the material having to be re-agglomerated. The test was re-started after filling the column. Generally, the results from one test cannot be regarded as representative of the whole oxide tailings dam. However, despite these limitations and problems encountered, the encouraging results obtained and the close comparison with the bottle-roll tests implies that the results are relatively reliable. The extraction values obtained from the column test, namely 73.0% for silver and 78.9% for gold, will therefore be used in evaluating this treatment process. The reagent consumption values also appear to be very high, namely 13.73 kg/t for lime, 21.8 kg/t for cement, and 2.32 kg/t for cyanide. However, lime and cement consumption values obtained in laboratory tests generally approximate commercial operations, although in this case, they seem to be unrealistically high. The cyanide consumption of a commercial operation would typically only be 30% to 50% of that measured in a laboratory test.

13.2.3.4.12 Acid-base Accounting

The ABA results predict the overall acid-generating potential of selected samples. A net acid general potential was found for the sulphide tailings but not the oxide tailings. The processing of the sulphide tailings for silver and gold recovery could modify the ABA and increase the stability of the ultimate residues. Alternatively, the sulphide tailings would require the addition of lime during the process of relocating this material. This would ensure that the sulphide tailings would not cause acid-generating environmental problems.

13.2.3.4.13 Electrowinning

Electrowinning metal recovery tests were conducted using EMEW technology (from the Electrometals Electrowinning company), specifically designed for the electrodeposition of metals from dilute solution tenors. The tests were carried out using filtered cyanide leach pregnant solutions. Although the test results were favourable, it appears unlikely at this stage that this technology could be applied in this situation, given the high solution volumes generated and the very low silver concentrations anticipated in the pregnant solution from the heap. However, further test work using the EMEW metal recovery system should be undertaken if the Project advances to the FS level because the potential for savings in capital and operating costs need to be investigated.

Historical Metallurgical Test Results - Sulphide Tailings

Limited test work has been completed on material from the sulphide tailings before 2012. The two sets of results on the reground sulphide samples indicate that 73% and 87% of the silver and 77% and 85% of the gold can be extracted using 1 g/L and 2 g/L cyanide solutions, respectively. However, the cyanide consumptions were higher than the results from the oxide tailings.

13.2.3.5 Historical Test Results Review

13.2.3.6 Gravity Concentration

Review of Results

As indicated in Table 13-21, the upgrading for silver from the as-received oxide tailings was poor, with a maximum concentrate grade of 125 g/t silver at a mass recovery of 20%. The upgrading of gold is similarly poor. The re-grinding of the samples prior to gravity concentration led to an almost negligible improvement in upgrading silver to 126 g/t silver, while for gold, a maximum concentrate grade of 1.29 g/t gold was obtained. The sulphide tailings response to gravity concentration is equally poor, with even lower-grade gravity concentrates being obtained despite slightly improved recoveries observed for both silver and gold.

13.2.3.7 Flotation

Review of Results

The flotation results have been summarized in Table 13-22. The results indicate that the overall recoveries for silver and gold are low, namely between 8% and 42% for silver and 12% to 48% for gold. The re-grinding of both the tailings samples (Composite A and Composite B) is seen to improve the recoveries, while the testing of various reagent regimes also resulted in improvements to the overall recoveries of silver and gold in some cases. However, the overall recoveries are generally considered low at less than 40% for silver and less than 48% for gold, coupled with producing a very low-grade concentrate. This poor flotation response is probably the result of surface alterations and/or inadequate liberation of the silver- and gold-bearing minerals. No extraction tests were conducted on any flotation concentrates produced, so the total extent of extraction is unknown. No tests were conducted on the sulphide tailings material (Composite C), and its response to flotation as a pre-concentration process is therefore not known.

13.2.3.8 Cyanide Leaching

Review of Results

Cyanidation leach tests were done on samples from Composite A and Composite B under different conditions of particle size and solution cyanide concentration. The results have been summarized in Table 13-23. The results generally indicated that cyanidation was still occurring after 72 h of the leaching time used for the laboratory tests, but at a much-reduced rate. The base metals copper and zinc also dissolved during the cyanide leach and contributed to the overall consumption of cyanide. Increasing the cyanide concentration in the leach solution generally improved the extraction of silver and gold but also increased the overall cyanide consumption. The extraction of silver and gold from Composite A increased with the fineness of grind size, while Composite B did not improve the extraction for finer grinds than P80 of 100 µm. The cyanide consumption figures are inconsistent in some cases, although trends are apparent. Although limited test work was done on material from Composite C, namely the sulphide tailings, a set of results has been included in Table 13-26 below for comparison purposes.

Table 13-26: Cyanide Leaching Parameters

Sample Identity	Head Grade (g/t)		Extraction (%)		Reagent Usage (kg/t)		NaCN Concentration (g/L)	P80 (µm)	Remarks
	Ag	Au	Ag	Au	NaCN	Lime			
Composite A	94.7	0.35	66.4	81.5	1.8	1.4	1.0	269	As-received sample
Composite B	95.9	0.28	69.1	82.0	2.6	1.8	1.0	180	
Average of A and B	95.3	0.32	67.8	81.8	2.2	1.6	1.0	225	
Composite A	94.7	0.35	79.3	85.7	1.6	1.8	1.0	103	Reground sample
Composite B	70.3	0.52	77.1	88.3	1.7	1.8	1.0	100	
Average of A & B	82.5	0.44	78.2	87.0	1.7	1.8	1.0	102	
Composite C	39.8	0.34	73.8	77.3	4.0	2.8	1.0	69	

13.2.3.9 Column Leach Test.

Review of Results

One column leach test was conducted using a blend of equal proportions of as-received (unground) Composite A and Composite B oxide tailings material. Despite interruptions in the leaching cycle due to the de-agglomeration of material in the column and the resultant percolation of fines, the overall extraction of silver was 73% and 79% for gold (see Table 13-25 for the results). Although the test was terminated after a total leaching time of 81 d, indications were that the leaching process was nearing completion but had not been finalized at that stage. The above extraction results compare very well with the average extraction results obtained from the bottle roll leach tests, namely 68% extraction for silver and 82% for gold. The cyanide consumption of 2.3 kg/t for the column test was also comparable with that obtained for the bottle roll leach tests, namely 2.2 kg/t. The lime consumption for the column test was significantly higher, probably due to the two repeated agglomeration exercises.

13.2.3.10 Precious Metal Recovery

Review of Results

Only one technology was tested for recovering precious metals from cyanide leach solutions. The pregnant solution from leach tests performed on oxide tailings material was used to conduct electrowinning tests. Three tests were conducted using the EMEW technology. These tests indicated that silver could be electrowon from solutions with a starting concentration of about 58 mg/L silver to a depleted electrolyte with about 3 mg/L silver. The deposition was also shown to be very selective regarding the co-deposition of base metals. However, the pregnant solution from a leaching heap is expected to be significantly less than 58 mg/L silver, possibly as low as 16 mg/L silver. It is unclear whether the EMEW technology could operate efficiently under such low silver tenors.

The alternative process options for recovering precious metals would likely be activated carbon or the zinc precipitation method. No tests were conducted on these two process options. The use of an activated carbon circuit to recover silver is not recommended because of the added operational complexity. Also, the relatively high grade of the silver in the solution will result in the treatment of relatively large amounts of carbon, which will add to the cost of the Project.

13.3 La Preciosa Area

Extensive metallurgical investigations were conducted to support the previous studies, including a feasibility study completed in 2014, NI 43-101 Technical Report Feasibility Study for La Preciosa Silver-Gold Project, prepared by M3 of Tucson, Arizona. Please refer to 2014 Feasibility Study Report (M3 Engineering & Technology Corporation 2013) for test work results on La Preciosa Area.

Further test work conducted in 2021 was focused on metallurgical response of the samples to conventional flotation concentration. The samples tested were from Abundancia, Gloria, and Martha mineralization zones. Please refer to 2023 Mineral Resource Estimate Update for the Avino Property, Durango, Mexico (Tetra Tech 2023) for 2021 test work results on La Preciosa Area.

14.0 MINERAL RESOURCES ESTIMATE

The current mineral resources for the property (Avino Mine and La Preciosa area) are summarized in Table 14-1.

Table 14-1: Avino Property – Mineral Resources (Inclusive of Mineral Reserves, Effective Date: October 16, 2023)

Area	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
Avino Mine	MEA	8.466	142.35	71.72	0.53	0.32	38.75	19.52	144.26	60.24
	IND	27.204	142.85	59.42	0.53	0.41	124.94	51.97	465.90	243.69
	M&I	35.671	142.73	62.35	0.53	0.39	163.69	71.50	610.15	303.95
	INF	19.373	112.02	45.83	0.34	0.37	69.77	28.54	212.59	158.31
La Preciosa	MEA	-	-	-	-	-	-	-	-	-
	IND	17.441	202	176	0.34	-	113.14	98.59	189.19	-
	M&I	17.441	202	176	0.34	-	113.14	98.59	189.19	-
	INF	4.397	170	151	0.25	-	24.1	21.33	35.48	-
TOTALS	**MEA**	**8.466**	**142.35**	**71.72**	**0.53**	**0.32**	**38.75**	**19.52**	**144.26**	**60.24**
	IND	**44.645**	**165.87**	**104.89**	**0.46**	**0.25**	**238.08**	**150.56**	**655.09**	**243.69**
	M&I	**53.111**	**162.12**	**99.61**	**0.47**	**0.26**	**276.83**	**170.08**	**799.34**	**303.95**
	INF	**23.770**	**122.83**	**65.26**	**0.32**	**0.30**	**93.87**	**49.87**	**248.07**	**158.31**

Notes:

1. Figures may not add to totals shown due to rounding.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimate is classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves incorporated by reference into NI 43-101 Standards of Disclosure for Mineral Projects.
4. Mineral Resources are stated inclusive of Mineral Reserves
5. Based on recent mining costs (Section 21.0), Mineral Resources are reported at cut-off grades 60 g/t, 130 g/t, and 50 g/t AgEQ grade for ET, San Gonzalo, and oxide tailings, respectively.
6. AgEQ or silver equivalent ounces are notational, based on the combined value of metals expressed as silver ounces
7. Metal price assumptions are shown in Table 14-2
8. Metal recovery is based on operational results and column testing, shown in Table 14-2
9. The silver equivalent was back-calculated using the formulas described in Section 14.

Table 14-2: Silver Equivalent-Based Metal Prices and Operational Recovery Parameters

Metal	Price	Unit	Recovery (%)
San Gonzalo Vein System			
Ag	21.00	US$/oz	83
Au	1,800.00	US$/oz	73
ET, Guadalupe, and La Potosina Deposits			
Ag	21.00	US$/oz	90
Au	1,800.00	US$/oz	75
Cu	3.50	US$/lb	89
Avino Tailings			
Ag	21.00	US$/oz	82
Au	1,250.00	US$/oz	78
La Preciosa Veins			
Ag	19.00	US$/oz	90
Au	1,750.00	US$/oz	75

14.1 Avino Property

14.1.1 Resource Summary

The following table provides a synopsis of the Mineral Resources reported in this section. Table 14-3 and Table 14-4 summarize the Mineral Resources at the Avino Mine Area.

Table 14-3: Avino Mine – Mineral Resources (Inclusive of Mineral Reserves, Effective Date: October 16, 2023)

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
ET Avino	MEA	3.883	171	69	0.53	0.57	21.39	8.58	67.00	48.91
	IND	23.916	146	58	0.53	0.44	112.41	44.59	409.00	234.08
	M&I	27.800	150	60	0.53	0.46	133.80	53.17	476.00	283.00
	INF	17.591	106	37	0.34	0.40	59.76	20.72	191.00	154.18
San Gonzalo	MEA	0.331	332	244	1.17	0.00	3.53	2.59	12.42	0.00
	IND	0.302	293	230	0.84	0.00	2.85	2.23	8.14	0.00
	M&I	0.633	313	237	1.01	0.00	6.38	4.83	20.56	0.00
	INF	0.246	297	271	0.35	0.00	2.35	2.14	2.74	0.00
Guadalupe	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.401	169	70	0.79	0.37	2.17	0.90	10.24	3.27
	M&I	0.401	169	70	0.79	0.37	2.17	0.90	10.24	3.27
	INF	0.354	159	82	0.62	0.30	1.81	0.93	7.00	2.30
La Potosina	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.142	220	186	0.41	0.04	1.00	0.85	1.85	0.13
	M&I	0.142	220	186	0.41	0.04	1.00	0.85	1.85	0.13
	INF	0.844	176	149	0.29	0.05	4.79	4.05	7.90	1.01
Tailings Deposit	MEA	4.252	101	61	0.47	0.12	13.83	8.35	64.84	11.33
	IND	2.443	83	43	0.47	0.12	6.51	3.40	36.67	6.21
	M&I	6.695	94	55	0.47	0.12	20.34	11.75	101.50	17.55
	INF	0.338	97	65	0.36	0.11	1.06	0.70	3.95	0.82

table continues…

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
TOTALS	MEA	8.466	142.35	71.72	0.53	0.32	38.75	19.52	144.26	60.24
	IND	27.204	142.85	59.42	0.53	0.41	124.94	51.97	465.90	243.69
	M&I	35.671	142.73	62.35	0.53	0.39	163.69	71.50	610.15	303.95
	INF	19.373	112.02	45.83	0.34	0.37	69.77	28.54	212.59	158.31

Notes:

1. Figures may not add to totals shown due to rounding.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimate is classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves incorporated by reference into NI 43-101 Standards of Disclosure for Mineral Projects.
4. Based on recent mining costs (Section 21.0), Mineral Resources are reported at cut-off grades 60 g/t, 130 g/t, and 50 g/t AgEQ grade for ET, San Gonzalo, and oxide tailings, respectively.
5. AgEQ or silver equivalent ounces are notational, based on the combined value of metals expressed as silver ounces
6. Cut-off grades were calculated using the following consensus metal price assumptions: gold price of US$1,800/oz, silver price of US$21.00/oz, and copper price of US$3.50/lb.
7. Metal recovery is based on operational results and column testing, shown in Table 14-5.
8. The silver equivalent was back-calculated using the following formulas:
   a. ET, Guadalupe, La Potosina: AgEQ = Ag (g/t) + 71.43 * Au (g/t) + 113.04 * Cu (%)
   b. San Gonzalo: Ag Eq = Ag (g/t) + 75.39 * Au (g/t)
   c. Oxide Tailings: Ag Eq = Ag (g/t) + 81.53 * Au (g/t)

Table 14-4: Avino Mine – Mineral Resources (Exclusive of Mineral Reserves, Effective Date: October 16, 2023)

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
ET Avino	MEA	3.883	171	69	0.53	0.57	21.39	8.58	67.00	48.91
	IND	23.916	146	58	0.53	0.44	112.41	44.59	409.00	234.08
	M&I	27.800	150	60	0.53	0.46	133.80	53.17	476.00	283.00
	INF	17.591	106	37	0.34	0.40	59.76	20.72	191.00	154.18
San Gonzalo	MEA	0.331	332	244	1.17	0.00	3.53	2.59	12.42	0.00
	IND	0.302	293	230	0.84	0.00	2.85	2.23	8.14	0.00
	M&I	0.633	313	237	1.01	0.00	6.38	4.83	20.56	0.00
	INF	0.246	297	271	0.35	0.00	2.35	2.14	2.74	0.00
Guadalupe	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.401	169	70	0.79	0.37	2.17	0.90	10.24	3.27
	M&I	0.401	169	70	0.79	0.37	2.17	0.90	10.24	3.27
	INF	0.354	159	82	0.62	0.30	1.81	0.93	7.00	2.30
La Potosina	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.142	220	186	0.41	0.04	1.00	0.85	1.85	0.13
	M&I	0.142	220	186	0.41	0.04	1.00	0.85	1.85	0.13
	INF	0.844	176	149	0.29	0.05	4.79	4.05	7.90	1.01
Tailings Deposit	MEA	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	M&I	0.000	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	INF	0.338	97	65	0.36	0.11	1.06	0.70	3.95	0.82

table continues…

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
TOTALS	MEA	4.214	91.56	41.04	0.29	0.26	24.92	11.17	79.42	48.91
	IND	24.761	135.40	55.53	0.49	0.40	118.43	48.57	429.23	237.48
	M&I	28.976	124.99	52.10	0.44	0.36	143.35	59.75	508.65	286.40
	INF	19.373	112.02	45.83	0.34	0.37	69.77	28.54	212.59	158.31

Notes:

1. Figures may not add to totals shown due to rounding.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimate is classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves incorporated by reference into NI 43-101 Standards of Disclosure for Mineral Projects.
4. Based on recent mining costs (Section 21.0), Mineral Resources are reported at cut-off grades 60 g/t, 130 g/t, and 50 g/t AgEQ grade for ET, San Gonzalo, and oxide tailings, respectively.
5. AgEQ or silver equivalent ounces are notational, based on the combined value of metals expressed as silver ounces
6. Cut-off grades were calculated using the following consensus metal price assumptions: gold price of US$1,800/oz, silver price of US$21.00/oz, and copper price of US$3.50/lb.
7. Metal recovery is based on operational results and column testing, shown in Table 14-5.
8. The silver equivalent was back-calculated using the following formulas:
   d. ET, Guadalupe, La Potosina: AgEQ = Ag (g/t) + 71.43 * Au (g/t) + 113.04 * Cu (%)
   e. San Gonzalo: Ag Eq = Ag (g/t) + 75.39 * Au (g/t)
   f. Oxide Tailings: Ag Eq = Ag (g/t) + 81.53 * Au (g/t)

The cut-off grades are summarized in Table 14-5 and are based on current operational and economic conditions at the Avino Mine.

Table 14-5: Avino Silver Equivalent Cut-off Grades with Metallurgical Recovery for Deposits based on Operational Performance and Column Tests

Deposit	Cut-off AgEQ (g/t)
Avino Vein (ET)	60
San Gonzalo Vein (SG)	120
Guadalupe	100
La Potosina	100
Tailings Deposit	50

Metal	ET, Guadalupe, La Potosina	San Gonzalo	Oxide Tailings
Ag	90	83	82
Au	75	73	78
Cu	89	-	-

14.1.2 Data

Avino supplied Drillhole data to the QP in the form of database export text files and Leapfrog Geo projects for the Avino, San Gonzalo, Tailings, Guadalupe and La Potosina deposits.

Wireframe meshes (.dxf files) of the topography, underground development and previous 3D models of the San Gonzalo and Avino Veins and cross-section and plan view images were supplied by Avino.

Data includes both underground channel sampling and diamond drill data.

14.1.3 Avino Vein

14.1.3.1 Geological Interpretation

The Avino Vein and the surrounding system are interpreted as part of a low- to intermediate-sulphidation system of silver-gold epithermal veins, breccias, stockworks, and silicified zones. The Avino system is relatively thick (up to 60 m thick in places) and exhibits lower silver but higher copper grades than the San Gonzalo Vein system.

The Avino Veins system at ET Mine is a broad zone (up to 60 m thick) of anastomosing veins, breccias, and stockworks, with more persistent downdip continuity. The average dip of the veins is 60° to 65° (see Figure 14-1 and Figure 14-2). The mineralized system has been exposed to a down-plunge extent of 770 m (to 1,620 m amsl) and is still open to depth. Figure 14-2). The Avino system can be summarized as two persistent vein structures with a breccia containing mineralized veinlets between the veins and similar breccias in the hanging wall.

Horizontal and vertical sections highlighting interpretation are shown in Figure 14-1 and Figure 14-2. The QP believes that it is reasonable to extend the geometric interpretation of the Avino Vein system to accommodate the mineralization physically sampled in the underground development (see Figure 14-2).

In 2022, the Avino Vein system was remodelled by Avino using Leapfrog Geo to account for the broader extents of the breccias, stockworks, deeper vein extension, and listric geometry.

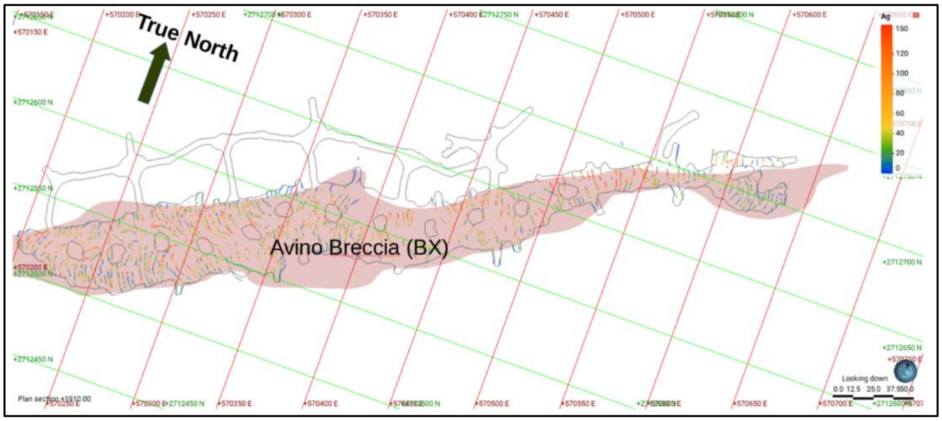


Figure 14-1: Horizontal Section Elevation 1,910 m amsl, 2022 Interpretation, Avino Breccia Shown in Red (Red Pennant 2020)

Note high-grade underground sampling extending into the hanging wall of the vein as a stockwork, and veins are exposed by underground development. Two veins modelled with lower grade stockwork exposed by underground development.

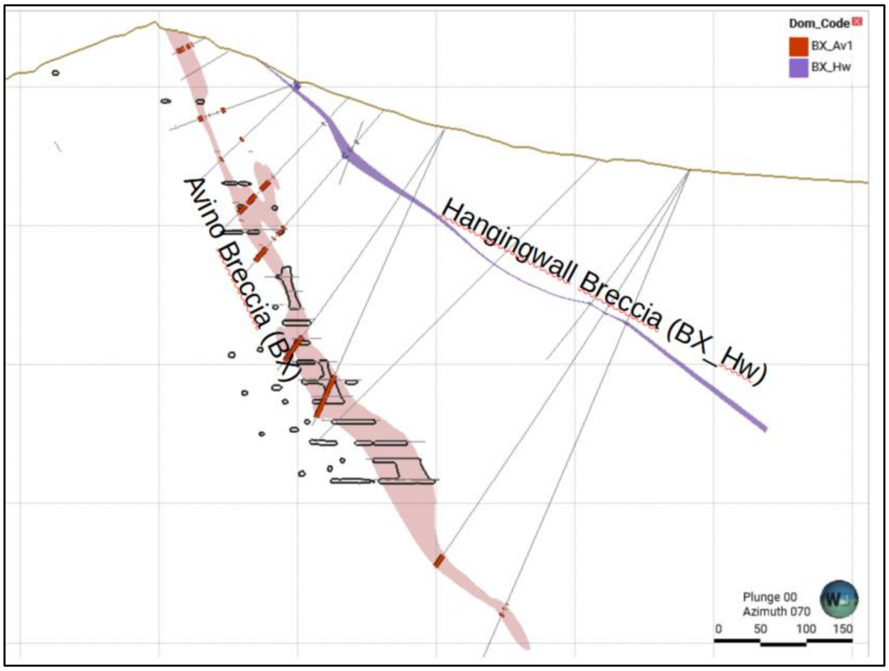


Figure 14-2: Vertical Section (reference +2712600.00, Y+570400.00, Azimuth 070) Showing 2017 Avino Vein System (BX) and Hanging-wall Breccia (BX_Hw) Models (Red Pennant 2023)

Mineralized zones were modeled by Avino in Leapfrog Geo™ software, utilizing the drillhole data, topography, and underground development information. The model was verified by the QP using Leapfrog Geo™ software. The QP believes the domain models are adequate for Mineral Resource estimation. The Avino mineralized vein (Figure 14-3) was modelled as a broad breccia zone. The breccia zone is aligned east-northeast–west-southwest and dips steeply (40° to 80°) south. The Hanging-wall Breccia (coded BX_Hw) is aligned east-west with increasing separation from the main Avino Vein towards the east.

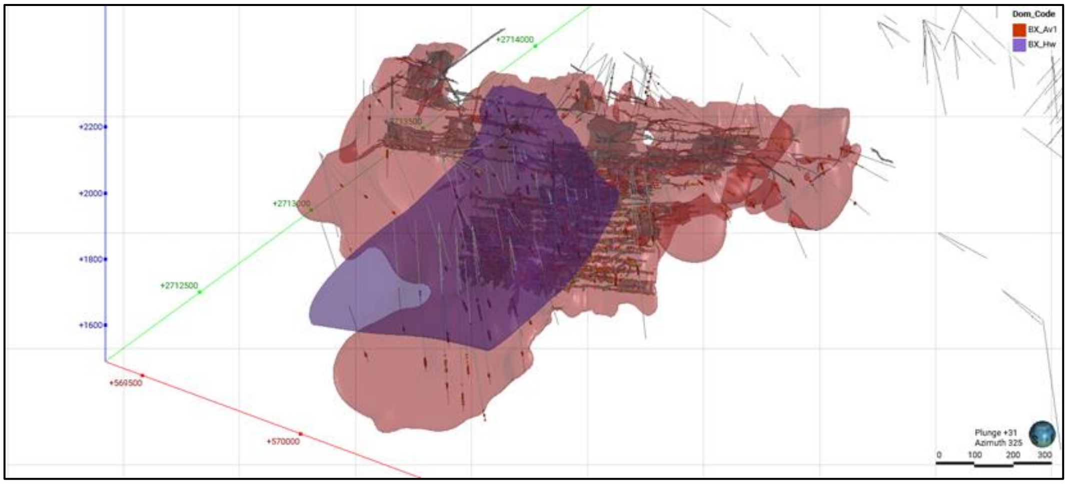


Figure 14-3: Oblique View, Looking Northwest, of the Avino Vein System Model (Breccia – red, HW Breccia - purple)

The QP believes that the 3D geological model of the ET breccia and vein deposit is adequate to support the Mineral Resource estimates.

14.1.4 San Gonzalo Vein

14.1.4.1 Geological Interpretation

The San Gonzalo Vein is interpreted as part of a low- to intermediate-sulphidation system of silver-gold epithermal veins and silicified zones. The individual veins in the San Gonzalo system are relatively narrow (mostly less than 3 m thick in places) and exhibit higher silver but lower copper grades than the Avino Vein system.

14.1.4.2 Wireframing

The San Gonzalo Vein system is narrow with high silver grades and is vertically restricted to a vertical interval less extending from the surface down to 350 m below the surface. The system was modelled using Leapfrog Geo™ software as six sub-parallel but cross-cutting veins consisting of a main vein (SG1) and five subsidiary units (SG2 through SG6). Lithology information and grade information were used to interpret the extent of the veins and to flag sampled and logged intervals as country rock or as one of the units of interest. Figure 14-4 displays the modeling results graphically.

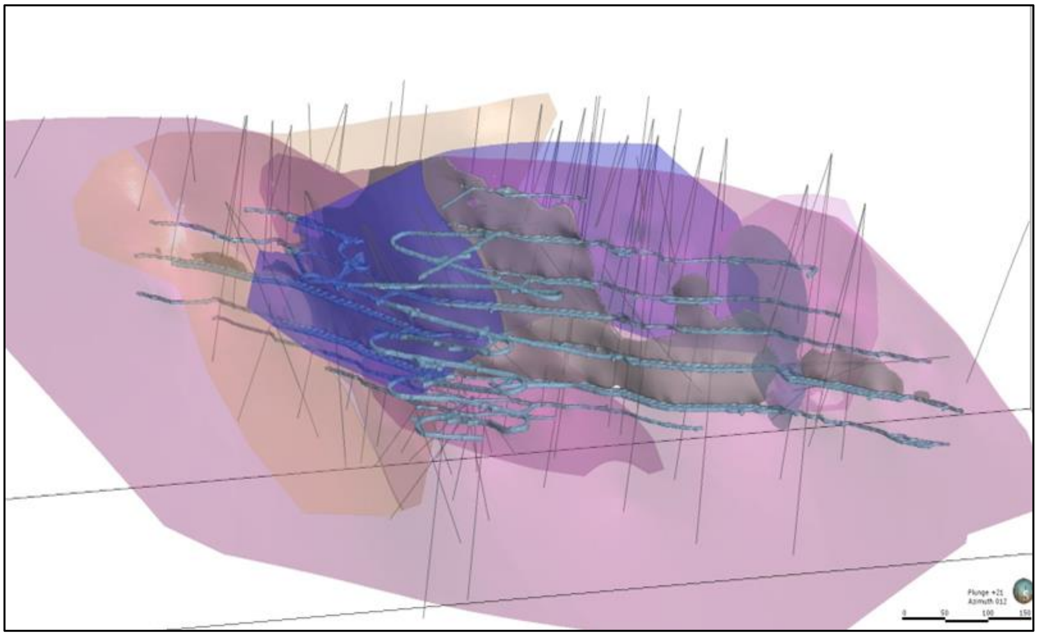


Figure 14-4: Oblique View, Looking North, of the San Gonzalo Vein System Model (Red Pennant 2020)

To assess the level to which the interpreted San Gonzalo Veins have honoured the selection, contact profiles were generated to examine how well the wireframe meshes segregate the metal based on the assayed samples. The San Gonzalo Veins are more compact than the Avino Veins, and the metal grades at San Gonzalo are confined to the vein material. The contact profiles were generated by determining the average grades for several metals within successive 5 m wide slices inside and outside the San Gonzalo Vein system models for a range of distances from -1.5 m (inside the system) to +1.7 m (outside) from the contact. The profiles are shown in Figure 14-5. There is a rapid decrease with increasing distance from the vein contacts in silver, gold, and copper profiles.

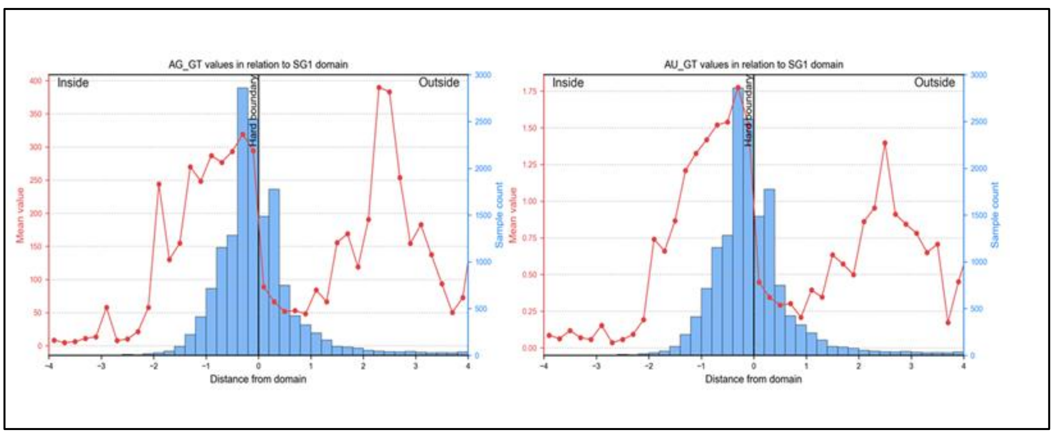


Figure 14-5: Silver and Gold Grade Profiles Across the Main San Gonzalo Vein Contacts (Red Pennant 2020)

For example, the silver grade is generally in excess of 200 g/t within the vein but decreases to an average of less than 50 g/t at a distance of 1 m from the vein contact.

The contacts were less conspicuous for gold, but average grade data still showed that the populations were statistically distinct. All contacts between mineralized and wall rock populations were treated as hard boundaries in estimation.

The San Gonzalo Vein system model is more robust when compared to the data and displays more abrupt metal profiles than the equivalent for the Avino Vein. This may reflect real differences in the mineralization styles (thickness and metal grade differences also support a different process) but may also, to some degree, be a result of the sparse legacy lithology data in the upper part of the Avino mine site.

The QP believes that the 3D geological model of the San Gonzalo vein deposit is adequate to support the Mineral Resource estimates.

14.1.5 Guadalupe Vein

14.1.5.1 Geological Interpretation

The Guadalupe Vein System was modelled in 2022 by Avino using Leapfrog Geo software as a series of intersecting veins (Figure 14-6). The RG vein has an average thickness of 3.1 m, and the VG Vein has an average thickness of 1.0 m.

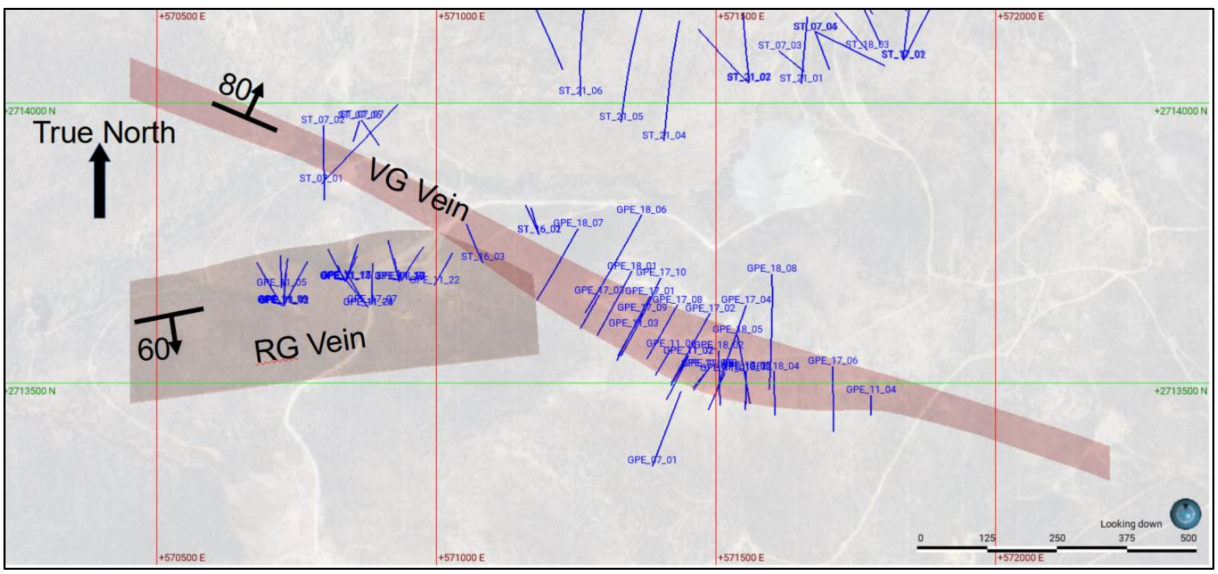


Figure 14-6: Plan View of Guadalupe Veins (Red Pennant 2020)

The QP believes that the 3D geological model of the Guadalupe vein deposit is adequate to support the Mineral Resource estimates.

14.1.6 La Potosina Vein

14.1.6.1 Geological Interpretation

The La Potosina Vein System was modelled in 2022 by Avino using Leapfrog Geo software as a series of intersecting veins (Figure 14-7).

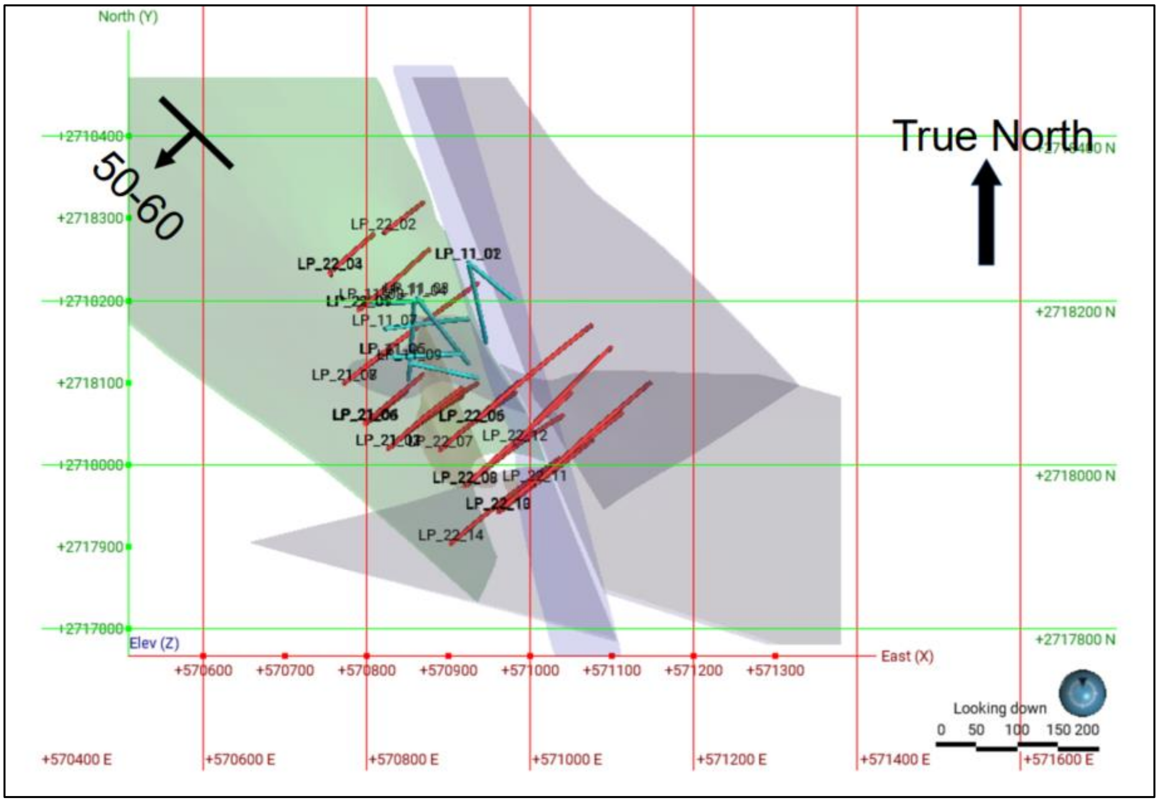


Figure 14-7: Plan View of La Potosina Veins (Red Pennant 2020)

The QP believes that the 3D geological model of the La Potosina vein deposit is adequate to support the Mineral Resource estimates.

14.1.7 Tailings

14.1.7.1 Geological Interpretation

In the Avino tailings deposit, a prominent bench separates the lower portion of the deposit (referred to as the “oxide lower bench” in various documents) from the upper portion of the oxide tailings (the “middle bench”). Overlying the oxide tailings are a volume of sulphide tailings material (the “upper bench” or “sulphide tailings”). The sulphide tailings material lacks representative sampling data.

Figure 14-8 is a perspective view looking north, showing the five units composing the tailings and the positions of drillholes.

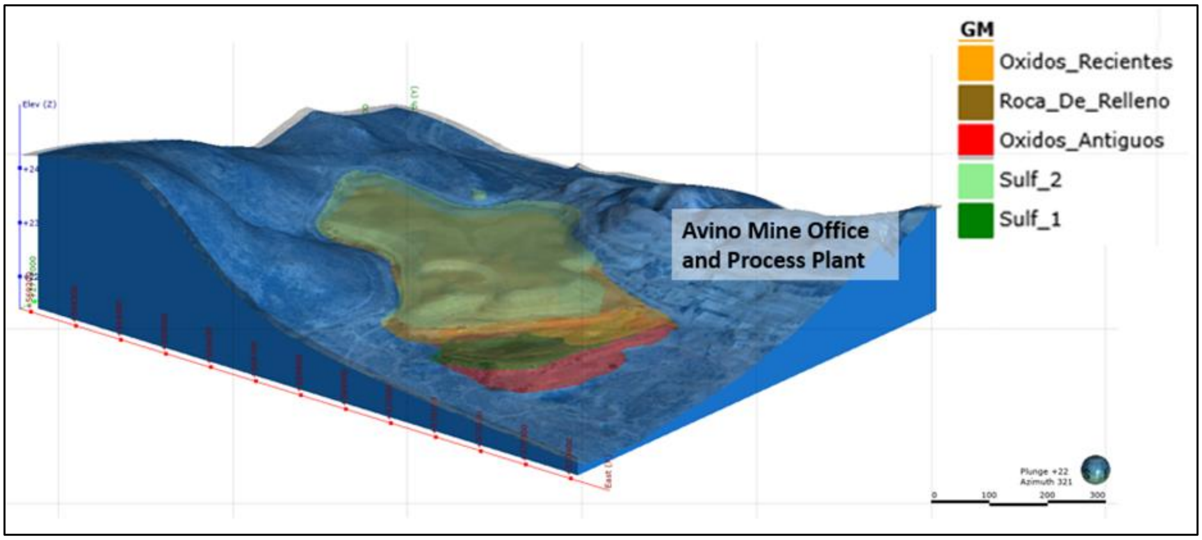


Figure 14-8: Perspective View of Tailings Deposit looking Northwest (Red Pennant 2020)

14.1.7.2 Wireframing

The tailings deposit was modelled using topography information supplied by Avino and drill logs and bedrock contact information.

The grade pattern of the upper, lower and middle benches has been defined by the addition of the recent drilling information. Previously, the middle and lower oxide benches were considered separately from an estimation perspective based on the colour difference, with the middle bench appearing to be more reddish than the lower material. More detailed data from sonic drilling has allowed for a more detailed consisting of six units. The sampling data (see Figure 14-9) shows a pattern of silver depletion at the top of the middle bench with enrichment immediately below.

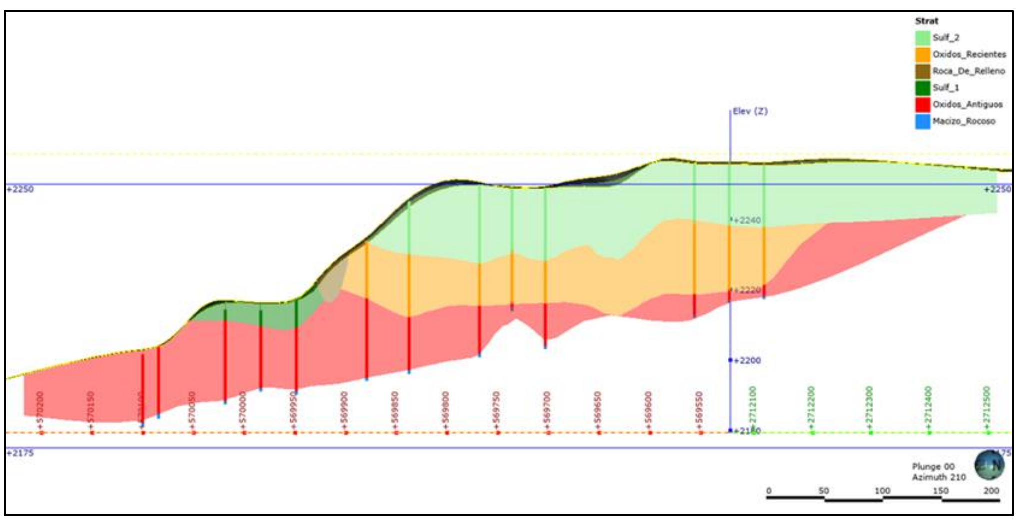


Figure 14-9: Section View, Looking Northwest, Showing the Geometry of the Five Units (vertical exaggeration x3) (Red Pennant 2020)

Spatial analysis indicates gradational changes between the middle and lower oxide benches, except for silver, which appears to have been leached downwards from the top of the middle bench. While the colour difference between the two units may be significant for iron-bearing species, there is not a great statistical difference between the silver and gold grades in the middle and lower benches, and the leaching effect (see Figure 14-8) appears to affect the upper portion of the middle bench. Consequently, it was decided to estimate the middle and lower bench as a single domain. The use of a variogram and search ellipse, flattened in the horizontal direction, also reduces the risk of smearing grade vertically. In future, it may be preferable to evaluate the leached cap of the tailings separately from the remainder. The risk of not doing so at this time is ameliorated by the horizontal variogram continuity, which reduces the risk of mixing.

The QP believes that the 3D geological model of the tailings deposit is adequate to support the Mineral Resource estimates.

14.1.8 Exploratory Data Analysis

14.1.8.1 Raw Data Assay and Statistics

14.1.8.1.1 Avino Elena Tolosa

Table 14-6 shows the length-weighted metal statistics for the sample data for the Avino (ET) Mine. Assayed metals include silver, gold, and copper. Metals considered in the Avino ET resource estimate include silver, gold, and copper.

Table 14-6: Length-weighted Metal Statistics for the Sample Data for the Avino (ET) Mine

Description	Name	Count	Length	Mean	Standard deviation	Coefficient of variation	Minimum	Median	Maximum
Drill Data	Ag_ppm	14,549	13,282.0	29.68	115.76	3.90	0.0050	7.00	5373
	Au_ppm	14,549	13,282.0	0.23	1.13	4.83	0.0003	0.04	94
	Cu_ppm	14,548	13,281.6	1595.24	4241.29	2.66	0.5000	345.00	219,000
	Pb_ppm	14,549	13,282.0	848.60	3732.04	4.40	1.0000	118.00	282,000
	Zn_ppm	14,549	13,282.0	1445.15	5111.55	3.54	0.5000	565.00	309,700
Channel Data	Ag_ppm	51,349	52,345.1	88.84	265.50	2.99	0.0003	39.00	14768
	Au_ppm	51,349	52,345.1	0.62	2.30	3.69	0.0003	0.22	277
	Cu_ppm	51,349	52,345.1	4783.70	7113.78	1.49	0.0100	3,000.00	660,000
	Pb_ppm	40,671	39,177.9	1523.08	7282.17	4.78	1.0000	400.00	627,900
	Zn_ppm	40,652	39,156.9	2120.06	8237.02	3.89	1.0000	800.00	270,000

14.1.8.1.2 San Gonzalo

The length-weighted metal statistics for the sample data for the San Gonzalo mineralization are summarized in Table 14-7. Metals considered in the San Gonzalo Vein resource estimate include silver and gold.

Table 14-7: Metal Grade Statistics for 2 m Composites for the San Gonzalo Vein Systems

Metal	Domain	Number of Composites	Mean	CV	Variance	Minimum	Maximum	Capping Value	Number Capped
Ag (g/t)	SG1	6,669	283.66	2.08	349,328.43	0	14,768.40	5,000	25
Ag (g/t)	SG2	117	72.32	1.11	6,500.57	0.9	890.24	0	-
Ag (g/t)	SG3	38	95.94	1.39	17,852.42	1.69	604.45	0	-
Ag (g/t)	SG4	168	200.05	2.81	315,058.50	3.72	5,265.20	3,000	2
Ag (g/t)	SG5	40	111.98	1.71	36,807.33	1.5	708.1	600	2
Ag (g/t)	SG6	54	40.25	1.99	6,418.32	0.7	331.2	0	-
Au (g/t)	SG1	6,669	1.48	2.8	17.22	0	204.17	0	-
Au (g/t)	SG2	117	0.56	1.04	0.35	0.01	3.79	0	-
Au (g/t)	SG3	38	0.58	1.07	0.38	0.02	3.05	0	-
Au (g/t)	SG4	168	0.69	2.24	2.41	0.01	13.96	10	2
Au (g/t)	SG5	40	0.74	2.26	2.78	0.01	9.84	4	2
Au (g/t)	SG6	54	0.3	2.08	0.39	0	2.77	0	-

14.1.8.1.3 Guadalupe

The length-weighted metal statistics for the sample data for the Guadalupe deposit are summarized in Table 14-8.

Table 14-8: Length-weighted Metal Statistics for the Sample Data for the Guadalupe Deposit

Metal Grade	Count	Length	Mean	Standard deviation	Coefficient of variation	Minimum	Median	Maximum
Ag_ppm	1,390	1,105.765	16.02	60.03	3.75	0.050	4.00	1724.60
Au_ppm	1,390	1,105.765	0.14	0.80	5.57	0.003	0.03	43.05
Cu_ppm	1,390	1,105.765	586.60	1,931.15	3.29	0.500	55.00	28,400.00
Pb_ppm	1,390	1,105.765	881.39	5,169.56	5.87	2.000	161.00	147,100.00
Zn_ppm	1,390	1,105.765	1,730.27	6,342.74	3.67	10.000	674.00	178,600.00

14.1.8.1.4 La Potosina

The length-weighted metal statistics for the sample data for the La Potosina deposit are summarized in Table 14-9.

Table 14-9: Length-weighted Metal Statistics for the Sample Data for the La Potosina Deposit

Metal Grade	Count	Length	Mean	Standard deviation	Coefficient of variation	Minimum	Median	Maximum
Ag_ppm	990	732.51	52.25	184.66	3.53	0.400	6.00	3,305.18
Au_ppm	990	732.51	0.12	0.25	2.10	0.003	0.04	2.57

table continues…

Metal Grade	Count	Length	Mean	Standard deviation	Coefficient of variation	Minimum	Median	Maximum
Cu_ppm	990	732.51	170.57	663.22	3.89	0.500	17.00	17,500.00
Pb_ppm	990	732.51	964.37	3513.44	3.64	1.000	112.00	75,900.00
Zn_ppm	988	731.56	2298.44	8603.66	3.74	2.500	225.00	254,000.00

14.1.8.1.5 Oxide Tailings

The oxide tailings drillhole dataset included 91 drillholes with a total metreage of 1,396 m that was completed in the tailings from 1990 to 2016. The data are presented in Table 14-10.

Table 14-10: Oxide Tailings Samples Summary

Name	Count	Length	Mean	Standard deviation	Coefficient of variation	Minimum	Median	Maximum
Ag (ppm)	3,021	4,483.45	50.19	34.94	0.70	0.150	37.40	233.00
Au (ppm)	3,022	4,484.95	0.41	0.19	0.46	0.006	0.40	1.73
Co (ppm)	3,239	4,828.33	4.80	3.74	0.78	0.005	5.00	120.00
Cu (ppm)	3,022	4,484.95	1,064.19	658.99	0.62	0.080	980.00	6,980.00
Pb (ppm)	3,018	4,478.95	4,273.70	4,165.13	0.97	0.570	2,150.00	24,900.00
Zn (ppm)	3,022	4,484.95	1,170.19	970.35	0.83	0.090	982.00	32,500.00

The tailings data have been subdivided by domain unit, and the metal grade statistics are summarized in Table 14-11.

Table 14-11: Metal Grade Statistics in Tailings Deposit by Domain

Metal	Name	Count	Length	Mean	Standard deviation	Variance	Minimum	Maximum
Ag_ppm		3,176	4,483.45	50.19	0.70	1,221.02	0.150	233.00
	Sulf_2	954	1,327.5	19.90	0.43	74.62	3.800	75.00
	Oxidos_Recientes	1,005	1,454.75	36.39	0.33	145.72	0.150	127.83
	Roca_De_Relleno	37	41.5	53.40	0.61	1,045.96	16.000	152.75
	Sulf_1	107	141.35	69.53	0.17	142.19	47.000	130.00
	Oxidos_Antiguos	991	1,419.4	91.98	0.29	716.02	1.000	233.00
	Macizo_Rocoso	9	7.45	43.09	0.83	1,266.94	4.000	112.00

table continues…

Metal	Name	Count	Length	Mean	Standard deviation	Variance	Minimum	Maximum
Au_ppm		3,177	4,484.95	0.41	0.46	0.04	0.006	1.73
	Sulf_2	954	1,327.5	0.24	0.48	0.01	0.084	0.90
	Oxidos_Recientes	1,005	1,454.75	0.51	0.35	0.03	0.006	1.73
	Roca_De_Relleno	37	41.5	0.12	0.86	0.01	0.029	0.40
	Sulf_1	107	141.35	0.45	0.16	0.01	0.273	0.66
	Oxidos_Antiguos	992	1420.9	0.47	0.30	0.02	0.010	1.08
	Macizo_Rocoso	9	7.45	0.17	0.57	0.01	0.030	0.31
Cu_ppm		3,177	4,484.95	1,064.19	0.62	434,271.09	0.080	6,980.00
	Sulf_2	954	1,327.5	567.34	0.64	130,474.25	0.500	3,260.00
	Oxidos_Recientes	1,005	1,454.75	1,245.56	0.62	588,218.91	5.000	6,160.00
	Roca_De_Relleno	37	41.5	1,014.48	0.46	216,239.28	259.000	2,060.00
	Sulf_1	107	141.35	1,057.74	0.23	57,144.55	598.000	1,690.00
	Oxidos_Antiguos	992	1420.9	1,361.58	0.35	223,029.27	0.080	3,590.00
	Macizo_Rocoso	9	7.45	902.72	0.75	456,280.66	178.000	2,890.00
Pb_ppm		3,173	4,478.95	4,273.70	0.97	17,348,205.17	0.570	24,900.00
	Sulf_2	954	1,327.5	1,049.40	0.56	343,842.82	9.000	5,510.00
	Oxidos_Recientes	1,005	1,454.75	2,153.80	0.44	898,457.53	10.000	9,496.00
	Roca_De_Relleno	37	41.5	2,969.64	0.35	1,083,494.73	1530.000	5,790.00
	Sulf_1	107	141.35	6,741.50	0.25	2,770,466.54	2580.000	11,100.00
	Oxidos_Antiguos	988	1414.9	9,488.00	0.34	10,283,948.86	0.570	24,900.00
	Macizo_Rocoso	9	7.45	3,845.13	1.22	21,891,772.03	135.000	17,800.00
Zn_ppm		3,177	4,484.95	1,170.19	0.83	941,573.16	0.090	32,500.00
	Sulf_2	954	1,327.5	1,030.25	1.08	1,228,825.83	2.500	22,500.00
	Oxidos_Recientes	1,005	1,454.75	980.30	0.48	224,959.57	34.000	3,931.00
	Roca_De_Relleno	37	41.5	2,082.80	0.28	348,703.18	924.000	3,300.00
	Sulf_1	107	141.35	932.31	0.31	81,052.91	460.000	1,810.00
	Oxidos_Antiguos	992	1,420.9	1,463.54	0.40	347,036.76	0.090	5,120.00
	Macizo_Rocoso	9	7.45	735.97	0.91	444,724.14	161.000	2,670.00

14.1.8.2 Outlier Management and Capping Strategy

It is common practice in the mineral industry to restrict the influence of high assays through “top-cutting” or “capping”. Capping was implemented for each element and each domain after compositing. Capping limits were chosen based on a review of sample histograms (including a review of histograms of logs and cumulative probabilities) using Seequent’s Leapfrog Edge software and an examination of the coefficient of variation statistic for each domain. The coefficient of variation, also known as relative standard deviation, is a standardized measure of the dispersion of a probability distribution and provides an indication of the presence of significant outliers. The coefficient of variation statistics greater than two and the visual detection of irregular behaviour in the upper portion of the log histogram of the data were used as an indicator that capping should be applied. Capping for all deposits is summarized in Table 14-12.

Table 14-12: Capping Values Tabulated by Deposit, Element and Domain

Deposit	Element	Domain	Capping Value
ET	Ag (ppm)	Ag in BX_Av1_N	650.0
ET	Ag (ppm)	Ag in BX_Av1_S	500.0
ET	Ag (ppm)	Ag in BX_Hw	150.0
ET	Au (ppm)	Au in BX_Av1_N	8.0
ET	Au (ppm)	Au in BX_Av1_S	8.0
ET	Au (ppm)	Au in BX_Hw	2.5
ET	Bi (ppm)	Bi in BX_Av1_N	1,600.0
ET	Bi (ppm)	Bi in BX_Av1_S	1,600.0
ET	Bi (ppm)	Bi in BX_Hw	600.0
ET	Cu (ppm)	Cu in BX_Av1_N	6.0
ET	Cu (ppm)	Cu in BX_Av1_S	5.0
ET	Cu (ppm)	Cu in BX_Hw	1.0
ET	Pb (ppm)	Pb in BX_Av1_N	2.5
ET	Pb (ppm)	Pb in BX_Av1_S	1.5
ET	Pb (ppm)	Pb in BX_Hw	0.3
ET	Zn (ppm)	Zn in BX_Av1_N	1.3
ET	Zn (ppm)	Zn in BX_Av1_S	1.0
ET	Zn (ppm)	Zn in BX_Hw	0.3
Tailings	Ag (ppm)	GM: Oxidos_Antiguos	150.0
Tailings	Ag (ppm)	GM: Oxidos_Recientes	150.0
Tailings	Ag (ppm)	GM: Roca_De_Relleno	150.0
Tailings	Ag (ppm)	GM: Sulf_1	150.0
Tailings	Ag (ppm)	GM: Sulf_2	60.0
Tailings	Au (ppm)	GM: Oxidos_Antiguos	0.9
Tailings	Au (ppm)	GM: Oxidos_Recientes	1.0
Tailings	Au (ppm)	GM: Roca_De_Relleno	0.6
Tailings	Au (ppm)	GM: Sulf_1	0.8
Tailings	Au (ppm)	GM: Sulf_2	0.8
Tailings	Cu (ppm)	GM: Oxidos_Antiguos	2,900.0
Tailings	Cu (ppm)	GM: Oxidos_Recientes	3,500.0

table continues…

Deposit	Element	Domain	Capping Value
Tailings	Cu (ppm)	GM: Roca_De_Relleno	2,000.0
Tailings	Cu (ppm)	GM: Sulf_1	1,950.0
Tailings	Cu (ppm)	GM: Sulf_2	1,800.0
Tailings	Pb (ppm)	GM: Oxidos_Antiguos	16,000.0
Tailings	Pb (ppm)	GM: Oxidos_Recientes	6,200.0
Tailings	Pb (ppm)	GM: Roca_De_Relleno	0.5
Tailings	Pb (ppm)	GM: Sulf_1	0.8
Tailings	Pb (ppm)	GM: Sulf_2	3,000.0
Tailings	Zn (ppm)	GM: Oxidos_Antiguos	3,000.0
Tailings	Zn (ppm)	GM: Oxidos_Recientes	3,200.0
Tailings	Zn (ppm)	GM: Roca_De_Relleno	2,700.0
Tailings	Zn (ppm)	GM: Sulf_1	1,850.0
Tailings	Zn (ppm)	GM: Sulf_2	3,500.0
San Gonzalo	Ag (ppm)	AG_GT_SG1	5,000.0
San Gonzalo	Ag (ppm)	AG_GT_SG2	3,000.0
San Gonzalo	Ag (ppm)	AG_GT_SG3	3,000.0
San Gonzalo	Ag (ppm)	AG_GT_SG4	3,000.0
San Gonzalo	Ag (ppm)	AG_GT_SG5	600.0
San Gonzalo	Ag (ppm)	AG_GT_SG6	331.2
San Gonzalo	Au (ppm)	Au_GT_SG1	50.0
San Gonzalo	Au (ppm)	Au_GT_SG2	10.0
San Gonzalo	Au (ppm)	Au_GT_SG3	10.0
San Gonzalo	Au (ppm)	Au_GT_SG4	10.0
San Gonzalo	Au (ppm)	Au_GT_SG5	4.0
San Gonzalo	Au (ppm)	Au_GT_SG6	2.8
Guadalupe	Ag (ppm)	Ag in GM Guadalupe: RG	250.0
Guadalupe	Ag (ppm)	Ag in GM Guadalupe: VG	250.0
Guadalupe	Au (ppm)	Au in GM Guadalupe: RG	2.0
Guadalupe	Au (ppm)	Au in GM Guadalupe: VG	5.0
Guadalupe	Cu (ppm)	Cu in GM Guadalupe: RG	10,000.0
Guadalupe	Cu (ppm)	Cu in GM Guadalupe: VG	16,000.0
Guadalupe	Pb (ppm)	Pb in GM Guadalupe: RG	35,000.0
Guadalupe	Pb (ppm)	Pb in GM Guadalupe: VG	10,000.0
Guadalupe	Zn (ppm)	Zn in GM Guadalupe: RG	10,000.0

table continues…

Deposit	Element	Domain	Capping Value
Guadalupe	Zn (ppm)	Zn in GM Guadalupe: VG	10,000.0
La Potosina	Ag (ppm)	Ag_V1	800.0
La Potosina	Ag (ppm)	Ag_V2	800.0
La Potosina	Ag (ppm)	Ag_V3	800.0
La Potosina	Ag (ppm)	Ag_V5	350.0
La Potosina	Au (ppm)	Au_V1	1.3
La Potosina	Au (ppm)	Au_V2	1.0
La Potosina	Au (ppm)	Au_V3	1.0
La Potosina	Au (ppm)	Au_V5	1.0
La Potosina	Cu (ppm)	Cu_V1	3,000.0
La Potosina	Cu (ppm)	Cu_V2	250.0
La Potosina	Cu (ppm)	Cu_V3	250.0
La Potosina	Cu (ppm)	Cu_V5	1,600.0
La Potosina	Pb (ppm)	Pb_V1	15,000.0
La Potosina	Pb (ppm)	Pb_V2	4,000.0
La Potosina	Pb (ppm)	Pb_V3	4,000.0
La Potosina	Pb (ppm)	Pb_V5	20,000.0
La Potosina	Zn (ppm)	Zn_V1	31,600.0
La Potosina	Zn (ppm)	Zn_V2	6,000.0
La Potosina	Zn (ppm)	Zn_V3	6,000.0
La Potosina	Zn (ppm)	Zn_V5	40,000.0

The QP is satisfied that the capping is appropriate to ameliorate the impact of extreme grade values in the deposits and is appropriate for Mineral Resource estimation.

14.1.8.3 Drillhole Compositing

Compositing is carried out to ensure a common 'change of support' length to avoid bias due to variable sampling lengths. Some variation in sampling lengths is inevitable as samplers endeavour to sample homogeneous geological units. If samples are not composited, small-length samples with a high grade (and the converse) might bias the estimation process.

Composite lengths for each deposit are summarized in Table 14-13. The QP is satisfied that the compositing is appropriate to manage potential length-weighting bias and is appropriate for Mineral Resource estimation.

Table 14-13: Composite Lengths

Deposit	Composite Length (m)
Avino (ET)	3
Tailings	1.6
San Gonzalo	1
Guadalupe	0.75
La Potosina	1

14.1.9 Bulk Density

14.1.9.1 Density Data

Density data was supplied by Avino in the form of a set of measurements made on a wax-coated core at the site using the Archimedes immersion method. Summary statistics for these measurements are provided in Table 14-14.

Table 14-14: Avino Vein System Density Data Summary

Zone	Rock Code	Count	Length	Mean
Avino (ET)	AND	93	9.5	2.60
Avino (ET)	BX	87	9.0	2.68
Avino (ET)	INT	50	5.2	2.60
Avino (ET)	ROC	61	6.1	2.60
Avino (ET)	STKWK	27	2.8	2.68
Avino (ET)	VN	35	3.3	2.74
Guadalupe	AND	31	3.1	2.59
Guadalupe	BX	4	0.4	2.90
Guadalupe	INT	1	0.1	2.58
Guadalupe	VN	13	1.3	2.83
La Potosina	AND	10	0.6	2.62
La Potosina	ROC	52	5.2	2.62
La Potosina	VN	33	3.1	2.69
San Gonzalo	AND	91	9.0	2.60
San Gonzalo	INT	30	3.0	2.56
San Gonzalo	STKWK	1	0.1	2.57
San Gonzalo	VN	18	1.8	2.48
Total		**637**	**63.7**	**2.63**

The QP estimated bulk density from the density measurements at all the deposits. Simple kriging using the global density means for each of the domains was employed.

Before 2015, Avino conducted bulk density measurements on 432 samples from 20 drillholes in the oxide tailings. Based on these data, Slim (2005d) determined a global average bulk density value of 1.605 for the oxide tailings. No new specific gravity data that can be considered representative of the tailings pile has been collected, so the QP used the specific gravity value of 1.605 for the current oxide tailings estimate, except for the waste rock layer in the tailings, which was assigned a bulk density of 2.5 t/m^3.

The QP believes that the quality and quality of bulk density data is sufficient to support Mineral Resource estimation.

14.1.10 Variography and Spatial Analysis

Variography was conducted utilizing Snowden Supervisor™ software for the San Gonzalo and oxide tailings deposits. Leapfrog EDGE software was used to model the Avino Vein domain variograms using normal scores transform to reduce the effect of extreme values. The experimental variograms were modelled parallel to the orientations of the veins in the case of the Avino and San Gonzalo Veins and horizontal for the tailings deposit.

Experimental variograms were modelled for the Avino Vein system and the tailings for all domains and the relevant potentially useful metals, silver, gold, and copper. Experimental variograms were modelled for the San Gonzalo Vein system for all domains and the relevant, potentially useful metals, silver and gold. The variogram models for San Gonzalo have been revised since 2016 to incorporate the observed sub-horizontal trends in grade that appear to reflect the telescoping of the mineralization into a vertical window some two to three hundred metres in vertical extent.

Figure 14-10, through Figure 14-16, show representative variograms. Variograms were modelled in normal scores transform, then back transformed and used for estimation on the untransformed grade data in Leapfrog Edge™. NN and inverse distance (ID) estimates used the same search ellipses applied for the corresponding OK estimators.

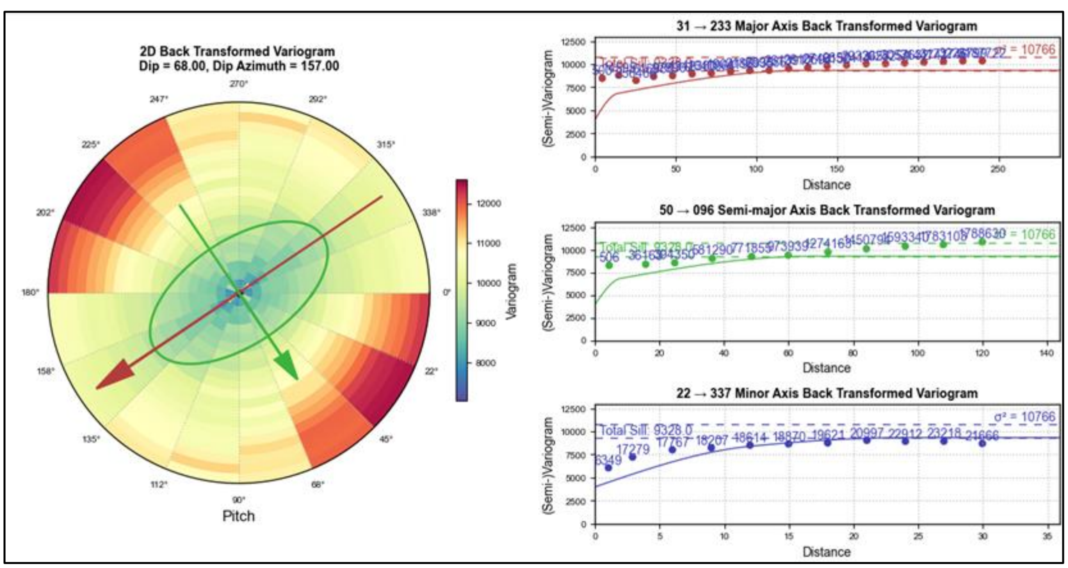


Figure 14-10: Avino Vein: Main Zone Experimental and Modelled Silver Variograms (Red Pennant 2023)

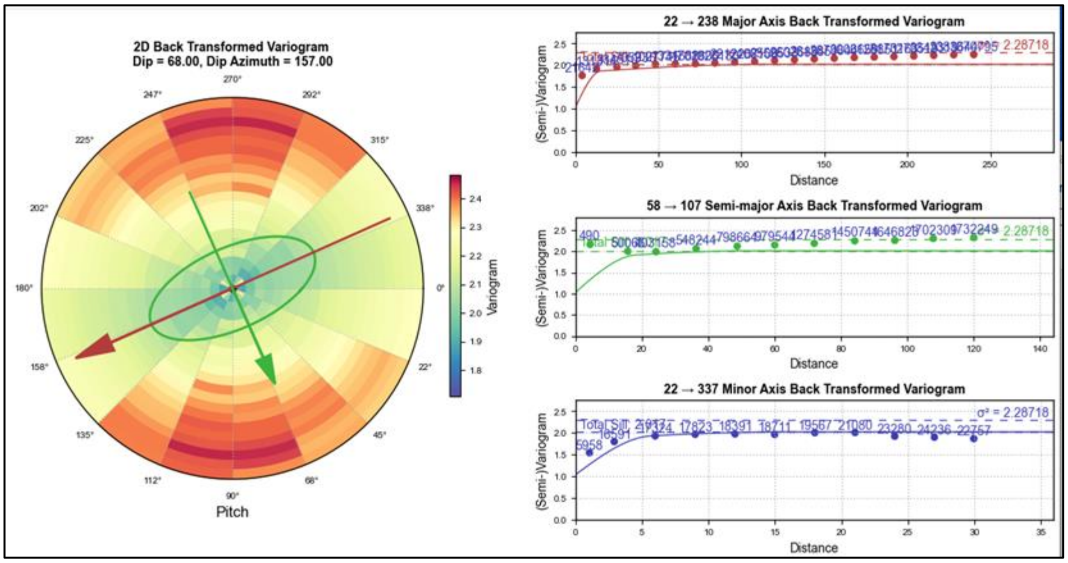


Figure 14-11: Avino Vein: Main Zone Experimental and Modelled Gold Variograms (Red Pennant 2023)

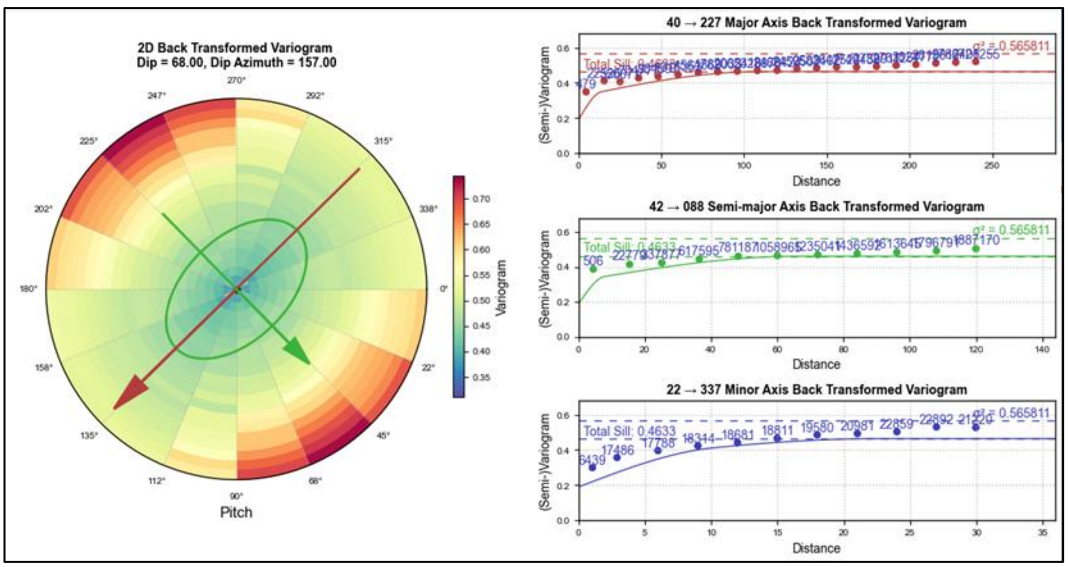


Figure 14-12: Avino Vein: Main Zone Experimental and Modelled Copper Variograms (Red Pennant 2023)

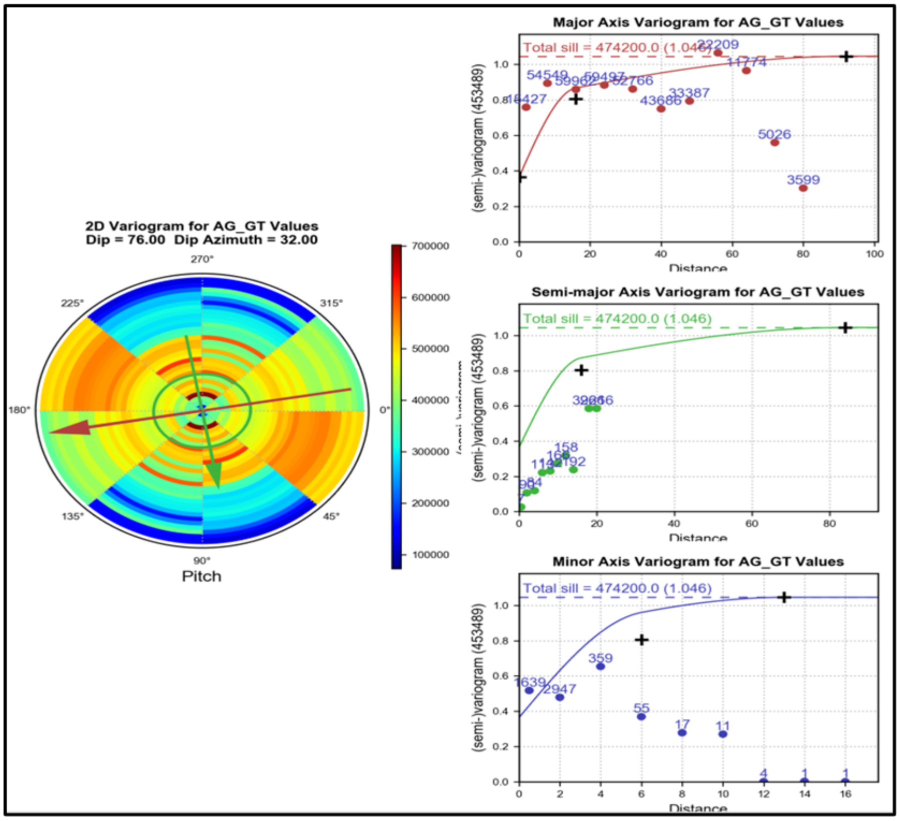


Figure 14-13: San Gonzalo Vein: SG1 Experimental and Modelled Silver Variograms (Red Pennant 2020)

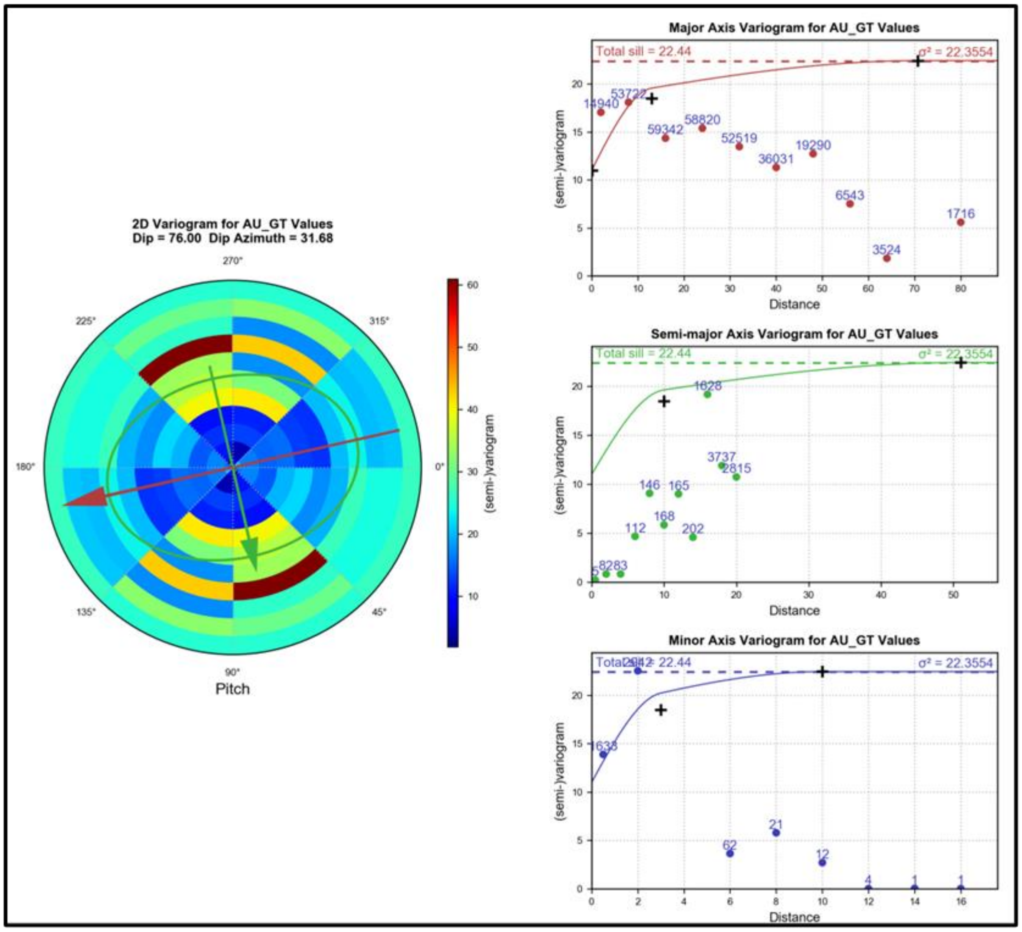


Figure 14-14: San Gonzalo Vein: SG1 Experimental and Modelled Gold Variograms (Red Pennant 2020)

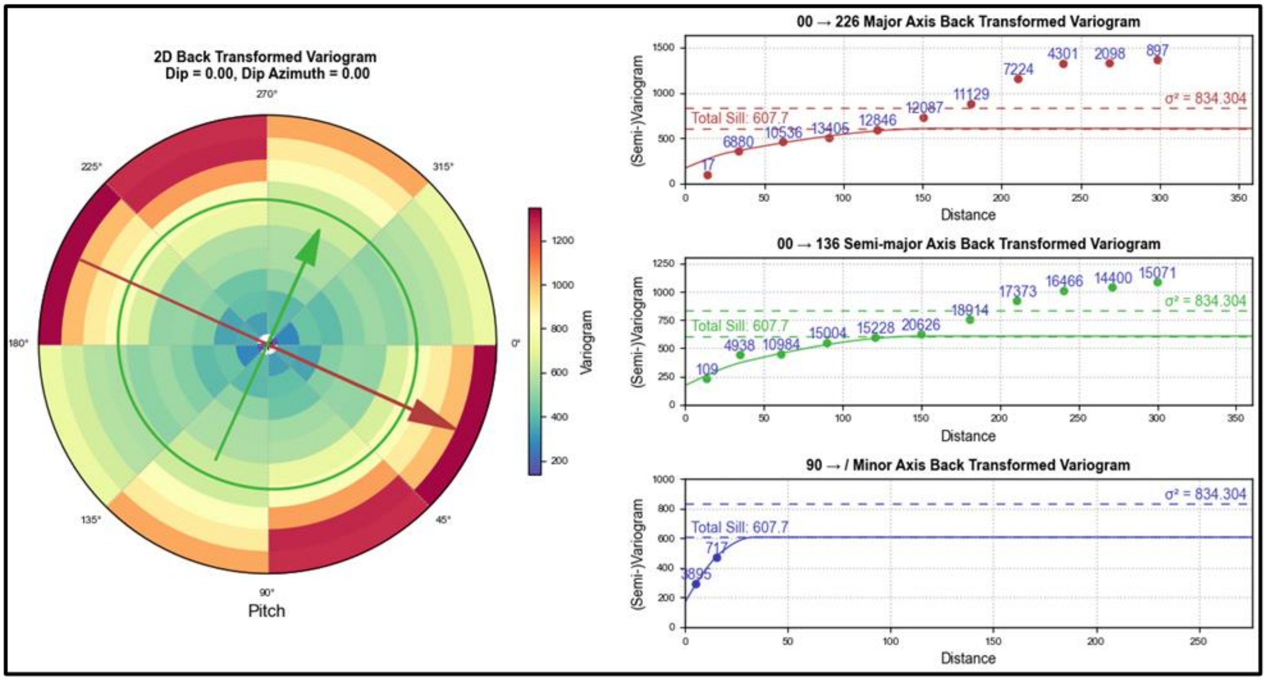


Figure 14-15: Older Oxide Tailings Domain Experimental Silver Variograms (Red Pennant 2022)

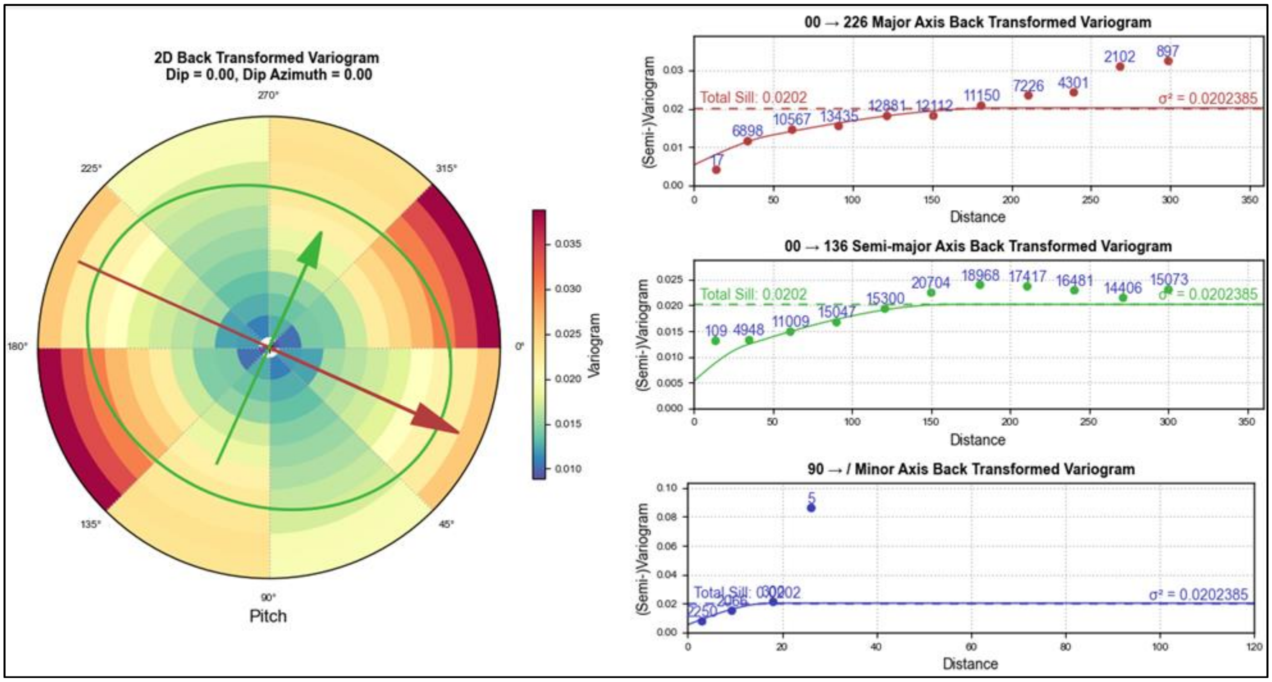


Figure 14-16: Older Oxide Tailings Domain Experimental Gold Variograms (Red Pennant 2020)

14.1.11 Interpolation Plan and Kriging Parameters

Estimation for the Avino and San Gonzalo Vein systems was carried out using Leapfrog Edge™ software, and parameters were optimized by minimizing the number of negative kriging weights and maximizing the Kriging slope of the regression of the estimates.

14.1.11.1 Avino

A sub-blocked block model was created to cover the Avino system. The parent block size of 16m by 16 m by 16 m was used for the block model and resource estimate. Sub-blocks of 2 m by 2 m by 2 m were used to better fill the vein system model shapes. Sub-blocks were populated with grades corresponding to those estimated in the parent blocks. The interpolation method used for populating the block model was OK with locally variable anisotropy, following the orientation of the veins.

Estimation parameters for the Avino Vein System are summarized in Table 14-15 and Table 14-16.

14.1.11.2 San Gonzalo

A single-block model was created to cover the San Gonzalo system. A block size of 10 m by 10 m by 10 m was used for the block model and resource estimate. The interpolation method used for populating the block model was OK. Estimation parameters used for the San Gonzalo system are summarized in Table 14-17.

14.1.11.3 Guadalupe

A single-block model was created to cover the Guadalupe veins. A block size of 5 m by 5 m by 5 m with sub-blocks of a minimum size of 0.625 m x 0.625 m x 0.625 m. was used for the block model and resource estimate. The interpolation method used for populating the block model was OK. A minimum of 16 and a maximum of 48 composites were used per block, with a maximum of 40 samples per drillhole.

14.1.11.4 La Potosina

A single-block model was created to cover the San Gonzalo system. A block size of 10 m by 10 m by 10 m was used for the block model and resource estimate. The interpolation method used for populating the block model was OK following a kriging neighbourhood specification tested in Snowden Supervisor™ software. A minimum of 16 and a maximum of 48 composites were used per block, with a maximum of 40 samples per drillhole.

14.1.11.5 Oxide Tailings

A single-block model was created to cover the Avino oxide tailings deposit. A block size of 30 m by 30 m by 4 m was used for the block model and resource estimate, as the average distance between sample drillholes approximates 30 m and the composite length is 2 m. To conserve volume, sub-blocks were created to a minimum lateral dimension of 3.75 m and vertical dimension of 0.5 m. The primary interpolation method was OK, with ID and NN estimates used as checks. A minimum of 3 and a maximum of 20 composites were used per block, with a maximum of 2 samples per drillhole. The data used for tailings deposit estimation was restricted to the more modern data from the 2015/16 and 2021/22 drilling campaigns. Pre-2015 data did not sample the sulphide tailings material and was not supported by a systematic QA/QC program.

Estimation parameters used for the oxide tailings deposit are summarized in Table 14-18.

Table 14-15: Avino ET Deposit Variogram Parameters

General	Direction			Variance	Nugget	Normalized Nugget	Structure 1							Structure 2						
NS Transformed Variogram Model	Dip	Dip Azimuth	Pitch				Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
Cu_ppm in ET_H02	64.17	157.21	133.9936	4621.559	936.7901	0.2027	2208.181	0.4778	Spherical		21.75	14.62	6.227	1473.353	0.3188	Spherical		204.4	139.2	26.79
Cu_ppm in ET_H01	64.17	157.21	133.9936	15475680	3136920	0.2027	7549037	0.4878	Spherical		21.75	14.62	6.227	4768057	0.3081	Spherical		74.5	45.82	16.47
Cu_ppm in ET_F03	64.17	157.21	133.9936	7915.139	1601.233	0.2023	3806.391	0.4809	Spherical		21.75	14.62	6.227	2528.887	0.3195	Spherical		128.6	139.2	26.79
Cu_ppm in ET_F02	64.17	157.21	133.9936	22146700	4489136	0.2027	7625109	0.3443	Spherical		17.37	11.63	6.227	9950512	0.4493	Spherical		159.4	111	26.79
Cu_ppm in ET_F01	64.17	157.21	133.9936	28254257	5732789	0.2029	13773950	0.4875	Spherical		14.15	19.95	6.227	8741867	0.3094	Spherical		127.9	148.9	28.4
Cu_ppm in ET04	64.17	157.21	133.9936	13359779	2710699	0.2029	5049996	0.378	Spherical		21.43	14.62	6.227	2941823	0.2202	Spherical		110.4	71.62	26.79
Cu_ppm in ET03	64.17	157.21	133.9936	7074.763	1435.469	0.2029	3383.152	0.4782	Spherical		21.75	14.62	6.227	2256.142	0.3189	Spherical		204.4	139.2	26.79
Cu_ppm in ET02	64.17	157.21	133.9936	24359931	4942630	0.2029	8121601	0.3334	Spherical		13.4	9.861	6.227	7271439	0.2985	Spherical		107.8	81.49	25.44
Cu_ppm in ET01	64.17	157.21	133.9936	31817929	6455858	0.2029	11740816	0.369	Spherical		14.21	14.62	6.227	9294017	0.2921	Spherical		202.9	100.9	26.79
Bi_ppm in ET_H02	64.17	157.21	133.9936	4621.559	936.7901	0.2027	2208.181	0.4778	Spherical		21.75	14.62	6.227	1473.353	0.3188	Spherical		204.4	139.2	26.79
Bi_ppm in ET_H01	64.17	157.21	133.9936	2163.58	438.7741	0.2028	1057.342	0.4887	Spherical		21.75	14.62	6.227	654.2667	0.3024	Spherical		69.21	100.3	26.79
Bi_ppm in ET_F03	64.17	157.21	133.9936	7915.139	1601.233	0.2023	3806.391	0.4809	Spherical		21.75	14.62	6.227	2528.887	0.3195	Spherical		128.6	139.2	26.79
Bi_ppm in ET_F02	64.17	157.21	133.9936	2369.105	480.2177	0.2027	1131.959	0.4778	Spherical		21.75	14.62	6.227	755.2708	0.3188	Spherical		204.4	139.2	26.79
Bi_ppm in ET_F01	64.17	157.21	133.9936	7736.629	1569.762	0.2029	3693.467	0.4774	Spherical		21.75	14.62	6.227	2454.059	0.3172	Spherical		225.2	113	38.91
Bi_ppm in ET04	64.17	157.21	133.9936	7046.654	1429.766	0.2029	3369.71	0.4782	Spherical		21.75	14.62	6.227	2247.178	0.3189	Spherical		204.4	139.2	26.79
Bi_ppm in ET03	64.17	157.21	133.9936	7074.763	1435.469	0.2029	3383.152	0.4782	Spherical		21.75	14.62	6.227	2256.142	0.3189	Spherical		204.4	139.2	26.79
Bi_ppm in ET02	64.17	157.21	133.9936	136696	27735.63	0.2029	41541.93	0.3039	Spherical		18.08	13.74	4.651	66981.06	0.49	Spherical		156.8	140	18.35
Bi_ppm in ET01	64.17	157.21	133.9936	136696	27735.63	0.2029	41541.93	0.3039	Spherical		18.08	13.74	4.651	66981.06	0.49	Spherical		156.8	140	18.35
Au_ppm in ET_H02	64.17	157.21	133.9936	4621.559	936.7901	0.2027	2208.181	0.4778	Spherical		21.75	14.62	6.227	1473.353	0.3188	Spherical		204.4	139.2	26.79
Au_ppm in ET_H01	64.17	157.21	150.2386	0.013356	0.002707	0.2027	0.0066	0.4947	Spherical		26.63	14.62	6.227	0.004	0.3013	Spherical		76.85	137.5	26.79
Au_ppm in ET_F03	64.17	157.21	133.9936	0.527164	0.106645	0.2023	0.2557	0.485	Spherical		21.75	14.62	6.227	0.1632	0.3096	Spherical		87	87	27
Au_ppm in ET_F02	64.17	157.21	133.9936	0.744035	0.150816	0.2027	0.2414	0.3245	Spherical		24.41	26.52	9.662	0.3503	0.4708	Spherical		110.6	139.2	33.51
Au_ppm in ET_F01	64.17	157.21	133.9936	5.199326	1.054943	0.2029	1.5109	0.2906	Spherical		17.92	14.62	6.227	2.6366	0.5071	Spherical		250	185.4	39.41
Au_ppm in ET04	64.17	157.21	133.9936	1.822118	0.369708	0.2029	0.8737	0.4795	Spherical		21.75	14.62	6.227	0.5798	0.3182	Spherical		167.7	110.6	32.48
Au_ppm in ET03	64.17	157.21	133.9936	7074.763	1435.469	0.2029	3383.152	0.4782	Spherical		21.75	14.62	6.227	2256.142	0.3189	Spherical		204.4	139.2	26.79
Au_ppm in ET02	64.17	157.21	133.9936	8.443554	1.54517	0.183	4.0267	0.4769	Spherical		17.18	17.52	9.776	2.8776	0.3408	Spherical		172.9	122.6	60.77
Au_ppm in ET01	64.17	157.21	133.9936	1.305068	0.264798	0.2029	0.5455	0.418	Spherical		17.05	15.16	10	0.4928	0.3776	Spherical		258.8	182.1	26.79
Ag_ppm in ET_H02	64.17	157.21	133.9936	4621.559	936.7901	0.2027	2208.181	0.4778	Spherical		21.75	14.62	6.227	1473.353	0.3188	Spherical		204.4	139.2	26.79
Ag_ppm in ET_H01	64.17	157.21	133.9936	2163.58	438.7741	0.2028	1057.342	0.4887	Spherical		21.75	14.62	6.227	654.2667	0.3024	Spherical		69.21	100.3	26.79
Ag_ppm in ET_F03	64.17	157.21	133.9936	7915.139	1601.233	0.2023	3806.391	0.4809	Spherical		21.75	14.62	6.227	2528.887	0.3195	Spherical		128.6	139.2	26.79
Ag_ppm in ET_F02	64.17	157.21	133.9936	2369.105	480.2177	0.2027	1131.959	0.4778	Spherical		21.75	14.62	6.227	755.2708	0.3188	Spherical		204.4	139.2	26.79

table continues…

General	Direction			Variance	Nugget	Normalized Nugget	Structure 1							Structure 2						
NS Transformed Variogram Model	Dip	Dip Azimuth	Pitch				Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
Ag_ppm in ET_F01	64.17	157.21	133.9936	7736.629	1569.762	0.2029	3693.467	0.4774	Spherical		21.75	14.62	6.227	2454.059	0.3172	Spherical		225.2	113	38.91
Ag_ppm in ET04	64.17	157.21	133.9936	7046.654	1429.766	0.2029	3369.71	0.4782	Spherical		21.75	14.62	6.227	2247.178	0.3189	Spherical		204.4	139.2	26.79
Ag_ppm in ET03	64.17	157.21	133.9936	7074.763	1435.469	0.2029	3383.152	0.4782	Spherical		21.75	14.62	6.227	2256.142	0.3189	Spherical		204.4	139.2	26.79
Ag_ppm in ET02	64.17	157.21	133.9936	6544.979	1327.976	0.2029	3381.136	0.5166	Spherical		21.75	17.52	6.227	1831.285	0.2798	Spherical		177.4	144.2	60.77
Ag_ppm in ET01	64.17	157.21	133.9936	7039.369	1428.288	0.2029	3366.226	0.4782	Spherical		21.75	14.62	6.227	2244.855	0.3189	Spherical		204.4	139.2	26.79

Table 14-16: Avino Vein System Search Parameters

General			Ellipsoid Ranges			Variable Orientation	Number of Samples		Drillhole Limit
Name	Domain	Values	Maximum	Intermediate	Minimum		Minimum	Maximum	Max Samples per Hole
Kr, Ag_ppm in ET01	GM_2020: ET01	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET02	GM_2020: ET02	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET03	GM_2020: ET03	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET04	GM_2020: ET04	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET_F01	GM_2020: ET_F01	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET_F02	GM_2020: ET_F02	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET_F03	GM_2020: ET_F03	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET_H01	GM_2020: ET_H01	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Ag_ppm in ET_H02	GM_2020: ET_H02	Ag_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET01	GM_2020: ET01	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET02	GM_2020: ET02	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET03	GM_2020: ET03	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET04	GM_2020: ET04	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET_F01	GM_2020: ET_F01	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET_F02	GM_2020: ET_F02	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET_F03	GM_2020: ET_F03	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET_H01	GM_2020: ET_H01	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Au_ppm in ET_H02	GM_2020: ET_H02	Au_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET01	GM_2020: ET01	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET02	GM_2020: ET02	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET03	GM_2020: ET03	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET04	GM_2020: ET04	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET_F01	GM_2020: ET_F01	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3

table continues…

General			Ellipsoid Ranges			Variable Orientation	Number of Samples		Drillhole Limit
Name	Domain	Values	Maximum	Intermediate	Minimum		Minimum	Maximum	Max Samples per Hole
Kr, Cu_ppm in ET_F02	GM_2020: ET_F02	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET_F03	GM_2020: ET_F03	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET_H01	GM_2020: ET_H01	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
Kr, Cu_ppm in ET_H02	GM_2020: ET_H02	Cu_ppm	204.4	139.2	26.79	Variable Orientation	7	20	3
SKr, Bi_ppm in ET02	GM_2020: ET02	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET03	GM_2020: ET03	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET04	GM_2020: ET04	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET_F01	GM_2020: ET_F01	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET_F02	GM_2020: ET_F02	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET_F03	GM_2020: ET_F03	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET_H01	GM_2020: ET_H01	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2
SKr, Bi_ppm in ET_H02	GM_2020: ET_H02	Bi_ppm	204.4	139.2	26.79	Variable Orientation	3	20	2

Table 14-17: San Gonzalo Vein System: Variogram and Search Parameters

Metal	Domain	Nugget C0	C1	C2	R11	R12	R13	R21	R22	R23	Ellipsoid Rotation		Pitch	Min Comps	Max Comps
											Dip	Azimuth			
Ag	SG1	165000	200000	109160	16	16	6	92	84	13	76	32	170	8	32
Ag	SG2	4060	6156	900	16	16	6	92	84	16	84	358	168	8	32
Ag	SG3	5000	9000	4013	16	16	6	92	84	16	73	42	166	8	32
Ag	SG4	160000	180000	72635	16	16	6	92	84	16	86	9	117	8	32
Ag	SG5	10000	15000	6487	16	16	6	92	84	16	76	198	14	8	32
Ag	SG6	2000	2000	1400	16	16	6	92	84	16	87	34	168	8	32
Au	SG1	11	7.5	3.94	13	10	3	71	51	10	76	32	170	8	32
Au	SG2	0.17	0.1	0.071	13	10	3	71	51	10	82	358	12	8	32
Au	SG3	0.27	0.2	0.069	13	10	3	71	51	10	73	42	166	8	32
Au	SG4	2.03	0.7	0.33	13	10	3	71	51	10	86	9	168	8	32
Au	SG5	1.6	0.9	0.66	13	10	3	71	51	10	76	198	14	8	32
Au	SG6	0.16	0.1	0.06	13	10	3	71	51	10	87	34	89	8	32

Table 14-18: Oxide Tailings Deposit: Variogram and Search Parameters

Domain 10		C1	R11	R12	R13	C2	R21	R22	R23	Angle1	Angle2	Angle3	Min Comps	Max Comps
Metal	Nugget													
Ag	0.37	0.49	30	7	68	0.14	130	20	170	0	-90	135	8	32
Au	0.39	0.51	55	8	16	0.1	100	20	92	0	-90	135	8	32
Cu	0.2	0.59	68	8	24	0.21	195	19	112	0	-90	135	8	32
Pb	0.45	0.35	63	13	70	0.2	269	14	170	0	-90	135	8	32
Zn	0.23	0.42	190	11	51	0.35	191	12	165	0	-90	135	8	32

Table 14-19: Guadalupe Variogram Parameters

	General	Direction			Model Space	Variance	Nugget	Normalized Nugget	Structure 1						
	Variogram Name	Dip	Dip Azimuth	Pitch					Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
0	Ag in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Data	13537.56488	3996.289153	0.2952	9485.771712	0.7007	Spherical		120	70	8
1	Ag in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Normal score	0.993758923	0.200268487		0.7935165		Spherical		120	70	8
2	Ag in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Normal score	0.99781427	0.281676333		0.716131301		Spherical		120	70	8
3	Ag in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Data	4284.033634	1693.050092	0.3952	2585.414298	0.6035	Spherical		120	70	8
4	Au in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	59.04263724	Data	0.399657818	0.13520424	0.3383	0.263294571	0.6588	Spherical		120	70	8
5	Au in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	59.04263724	Normal score	0.993758923	0.200268487		0.7935165		Spherical		120	70	8
6	Au in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Data	2.518726395	1.154332307	0.4583	1.362379107	0.5409	Spherical		120	70	8
7	Au in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Normal score	0.99781427	0.286155285		0.711740918		Spherical		120	70	8
8	Bi in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Data	13537.56488	3996.289153	0.2952	9485.771712	0.7007	Spherical		120	70	8
9	Bi in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Normal score	0.993758923	0.200268487		0.7935165		Spherical		120	70	8
10	Bi in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Data	14061.66786	5905.900501	0.42	8154.361191	0.5799	Spherical		70.84	51.26	7.997
11	Bi in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Normal score	0.99781427	0.281676333		0.71812693		Spherical		70.84	51.26	7.997
12	Cu in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Data	13537.56488	3996.289153	0.2952	9485.771712	0.7007	Spherical		120	70	8
13	Cu in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Normal score	0.993758923	0.200268487		0.7935165		Spherical		120	70	8

table continues…

	General	Direction			Model Space	Variance	Nugget	Normalized Nugget	Structure 1						
	Variogram Name	Dip	Dip Azimuth	Pitch					Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
14	Cu in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Data	15478913.55	5495014.309	0.355	9959132.976	0.6434	Spherical		120	70	8
15	Cu in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Normal score	0.99781427	0.281676333		0.716131301		Spherical		120	70	8
16	Pb in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Data	13537.56488	3996.289153	0.2952	9485.771712	0.7007	Spherical		120	70	8
17	Pb in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Normal score	0.993758923	0.200268487		0.7935165		Spherical		120	70	8
18	Pb in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Data	15478913.55	5495014.309	0.355	9959132.976	0.6434	Spherical		120	70	8
19	Pb in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Normal score	0.99781427	0.281676333		0.716131301		Spherical		120	70	8
20	SG_Guadalupe: Variogram Model	0	0	57.31172605	Data	0.027738775	0	0	0.048154513	1.736	Spherical		200	200	112.5
21	Zn in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Data	13537.56488	3996.289153	0.2952	9485.771712	0.7007	Spherical		120	70	8
22	Zn in GM Guadalupe: RG: Transformed Variogram Model	61.67136536	174.8535386	47.39313454	Normal score	0.993758923	0.200268487		0.7935165		Spherical		120	70	8
23	Zn in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Data	15478913.55	5495014.309	0.355	9959132.976	0.6434	Spherical		120	70	8
24	Zn in GM Guadalupe: VG: Transformed Variogram Model	77.33	22.14869774	112.2649222	Normal score	0.99781427	0.281676333		0.716131301		Spherical		120	70	8

Table 14-20: Guadalupe Search Parameters

General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of samples		Outlier Restrictions			Sector Search			Drillhole Limit	
Interpolant Name	Domain	Numeric Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum	Method	Distance	Threshold	Method	Max Samples	Max Empty Sectors	Max Samples per Hole	Apply Drillhole Limit per Sector
ID_Ag_RG	GM Guadalupe: RG	Ag	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Ag_VG	GM Guadalupe: VG	Ag	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Au_RG	GM Guadalupe: RG	Au	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Au_VG	GM Guadalupe: VG	Au	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Cu_RG	GM Guadalupe: RG	Cu_ppm	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Cu_VG	GM Guadalupe: VG	Cu_ppm	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Pb_RG	GM Guadalupe: RG	Pb_ppm	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Pb_VG	GM Guadalupe: VG	Pb_ppm	150	150	12				Variable Orientation	1	20	None			None			2	1

table continues…

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General			Ellipsoid Ranges			Ellipsoid Directions				Number of samples		Outlier Restrictions			Sector Search			Drillhole Limit	
Interpolant Name	Domain	Numeric Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch	Variable Orientation	Minimum	Maximum	Method	Distance	Threshold	Method	Max Samples	Max Empty Sectors	Max Samples per Hole	Apply Drillhole Limit per Sector
ID_Zn_RG	GM Guadalupe: RG	Zn_ppm	150	150	12				Variable Orientation	1	20	None			None			2	1
ID_Zn_VG	GM Guadalupe: VG	Zn_ppm	150	150	12				Variable Orientation	1	20	None			None			2	1
Kr_Ag_RG	GM Guadalupe: RG	Ag	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Ag_VG	GM Guadalupe: VG	Ag	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Au_RG	GM Guadalupe: RG	Au	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Au_VG	GM Guadalupe: VG	Au	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Cu_RG	GM Guadalupe: RG	Cu_ppm	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Cu_VG	GM Guadalupe: VG	Cu_ppm	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Pb_RG	GM Guadalupe: RG	Pb_ppm	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Pb_VG	GM Guadalupe: VG	Pb_ppm	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Zn_RG	GM Guadalupe: RG	Zn_ppm	150	150	12				Variable Orientation	3	20	None			None			2	1
Kr_Zn_VG	GM Guadalupe: VG	Zn_ppm	150	150	12				Variable Orientation	3	20	None			None			2	1
NN_ Ag_RG	GM Guadalupe: RG	Ag	150	150	12	77.3	22.1	112.3											
NN_ Ag_VG	GM Guadalupe: VG	Ag	150	150	12	77.3	22.1	112.3											
NN_ Au_RG	GM Guadalupe: RG	Au	150	150	12	77.3	22.1	112.3											
NN_ Au_VG	GM Guadalupe: VG	Au	150	150	12	77.3	22.1	112.3											
NN_ Cu_RG	GM Guadalupe: RG	Cu_ppm	150	150	12	77.3	22.1	112.3											
NN_ Cu_VG	GM Guadalupe: VG	Cu_ppm	150	150	12	77.3	22.1	112.3											
NN_ Pb_RG	GM Guadalupe: RG	Pb_ppm	150	150	12	77.3	22.1	112.3											
NN_ Pb_VG	GM Guadalupe: VG	Pb_ppm	150	150	12	77.3	22.1	112.3											
NN_ Zn_RG	GM Guadalupe: RG	Zn_ppm	150	150	12	77.3	22.1	112.3											
NN_ Zn_VG	GM Guadalupe: VG	Zn_ppm	150	150	12	77.3	22.1	112.3											

Table 14-21: La Potosina Variogram Parameters

	General	Direction			Model Space	Variance	Nugget	Normalized Nugget	Structure 1									Structure 2				
	Variogram Model	Dip	Dip Azimuth	Pitch					Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
0	Ag_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Normal score	0.998158056	0.281375741		0.388882379		Spherical		15	15	8	0.328893079		Spherical		120	70	25
1	Ag_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Data	91752.33371	41224.32354	0.4493	33801.55974	0.3684	Spherical		15	15	8	16698.92474	0.182	Spherical		120	70	25
2	Ag_V2: Transformed Variogram Model	70.65	240.23	159.6950395	Data	12449.84951	5268.776312	0.4232	4468.250989	0.3589	Spherical		15	15	8	2614.468397	0.21	Spherical		120	70	25
3	Ag_V2: Transformed Variogram Model	70.65	240.23	159.6950395	Normal score	0.985259486	0.277739699		0.383857096		Spherical		15	15	8	0.324643001		Spherical		120	70	25
4	Ag_V3: Transformed Variogram Model	53.46	229.71	91.46	Data	135.2200781	43.60847518	0.3225	51.19432156	0.3786	Spherical		15	15	8	38.48363422	0.2846	Spherical		120	70	25
5	Ag_V3: Transformed Variogram Model	53.46	229.71	91.46	Normal score	0.969211873	0.273215957		0.377604946		Spherical		15	15	8	0.319355312		Spherical		120	70	25
6	Ag_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Data	11247.49938	3717.298544	0.3305	4338.16051	0.3857	Spherical		15	15	8	3152.674075	0.2803	Spherical		120	70	25
7	Ag_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Normal score	0.99470828	0.280403266		0.387538346		Spherical		15	15	8	0.327756378		Spherical		120	70	25
8	Au_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Data	0.099158718	0.029628625	0.2988	0.043996723	0.4437	Spherical		15	15	8	0.025473875	0.2569	Spherical		120	70	25
9	Au_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Normal score	0.998158056	0.237607733		0.432601701		Spherical		15	15	8	0.328893079		Spherical		120	70	25
10	Au_V2: Transformed Variogram Model	70.65	240.23	159.6950395	Data	12449.84951	5268.776312	0.4232	4468.250989	0.3589	Spherical		15	15	8	2614.468397	0.21	Spherical		120	70	25

table continues…

	General	Direction			Model Space	Variance	Nugget	Normalized Nugget	Structure 1									Structure 2				
	Variogram Model	Dip	Dip Azimuth	Pitch					Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
11	Au_V2: Transformed Variogram Model	70.65	240.23	159.6950395	Normal score	0.985259486	0.277739699		0.383857096		Spherical		15	15	8	0.324643001		Spherical		120	70	25
12	Au_V3: Transformed Variogram Model	53.46	229.71	91.46	Data	135.2200781	43.60847518	0.3225	51.19432156	0.3786	Spherical		15	15	8	38.48363422	0.2846	Spherical		120	70	25
13	Au_V3: Transformed Variogram Model	53.46	229.71	91.46	Normal score	0.969211873	0.273215957		0.377604946		Spherical		15	15	8	0.319355312		Spherical		120	70	25
14	Au_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Data	11247.49938	3717.298544	0.3305	4338.16051	0.3857	Spherical		15	15	8	3152.674075	0.2803	Spherical		120	70	25
15	Au_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Normal score	0.99470828	0.280403266		0.387538346		Spherical		15	15	8	0.327756378		Spherical		120	70	25
16	Cu_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Data	612955.0189	209814.503	0.3423	266819.3197	0.4353	Spherical		15	15	8	133746.7851	0.2182	Spherical		120	108.8	25
17	Cu_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Normal score	0.998158056	0.237607733		0.432601701		Spherical		15	15	8	0.322604684		Spherical		120	108.8	25
18	Cu_V2: Transformed Variogram Model	70.65	240.23	159.6950395	Data	12449.84951	5268.776312	0.4232	4468.250989	0.3589	Spherical		15	15	8	2614.468397	0.21	Spherical		120	70	25
19	Cu_V2: Transformed Variogram Model	70.65	240.23	159.6950395	Normal score	0.985259486	0.277739699		0.383857096		Spherical		15	15	8	0.324643001		Spherical		120	70	25
20	Cu_V3: Transformed Variogram Model	53.46	229.71	31.93709148	Data	1766.034162	615.9927156	0.3488	660.3201731	0.3739	Spherical		15	15	8	466.762829	0.2643	Spherical		120	70	25
21	Cu_V3: Transformed Variogram Model	53.46	229.71	31.93709148	Normal score	0.969211873	0.273215957		0.377604946		Spherical		15	15	8	0.319355312		Spherical		120	70	25

table continues…

	General	Direction							Structure 1							Structure 2						
	Variogram Model	Dip	Dip Azimuth	Pitch	Model Space	Variance	Nugget	Normalized Nugget	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
22	Cu_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Data	262084.0666	96892.47941	0.3697	101243.0749	0.3863	Spherical		15	15	8	62873.96757	0.2399	Spherical		120	47.42	25
23	Cu_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Normal score	0.99470828	0.280403266		0.387538346		Spherical		15	15	8	0.325866432		Spherical		120	47.42	25
24	Pb_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Data	9823804.02	3015907.834	0.307	4459024.645	0.4539	Spherical		13.31	8.304	8	2311541.086	0.2353	Spherical		140.5	108.8	25.31
25	Pb_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Normal score	0.998158056	0.237607733		0.442882729		Spherical		13.31	8.304	8	0.313820893		Spherical		140.5	108.8	25.31
26	Pb_V2: Transformed Variogram Model	70.65	240.23	126.0421913	Data	4132078.927	1671839.134	0.4046	1487961.622	0.3601	Spherical		15	15	8	936329.0849	0.2266	Spherical		120	70	25
27	Pb_V2: Transformed Variogram Model	70.65	240.23	126.0421913	Normal score	0.985259486	0.277739699		0.383857096		Spherical		15	15	8	0.324643001		Spherical		120	70	25
28	Pb_V3: Transformed Variogram Model	53.46	229.71	31.93709148	Data	1766.034162	615.9927156	0.3488	660.3201731	0.3739	Spherical		15	15	8	466.762829	0.2643	Spherical		120	70	25
29	Pb_V3: Transformed Variogram Model	53.46	229.71	31.93709148	Normal score	0.969211873	0.273215957		0.377604946		Spherical		15	15	8	0.319355312		Spherical		120	70	25
30	Pb_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Data	262084.0666	96892.47941	0.3697	101243.0749	0.3863	Spherical		15	15	8	62873.96757	0.2399	Spherical		120	47.42	25
31	Pb_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Normal score	0.99470828	0.280403266		0.387538346		Spherical		15	15	8	0.325866432		Spherical		120	47.42	25
32	SG: Variogram Model	51.37	216.17	158.5941841	Data	0.041409017	0	0	0.041160563	0.994	Spherical		150	140	40							

table continues…

	General	Direction			Model Space	Variance	Nugget	Normalized Nugget	Structure 1							Structure 2						
	Variogram Model	Dip	Dip Azimuth	Pitch					Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized Sill	Structure	Alpha	Major	Semi-major	Minor
33	Zn_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Data	89373310.08	29546816.31	0.3306	39860496.29	0.446	Spherical		13.31	8.304	8	19983872.13	0.2236	Spherical		104.7	93.26	25.35
34	Zn_V1: Transformed Variogram Model	55.72	213.85	157.8142458	Normal score	0.998146432	0.237604966		0.442877572		Spherical		13.31	8.304	8	0.319706302		Spherical		104.7	93.26	25.35
35	Zn_V2: Transformed Variogram Model	70.65	240.23	126.0421913	Data	4132078.927	1671839.134	0.4046	1487961.622	0.3601	Spherical		15	15	8	936329.0849	0.2266	Spherical		120	70	25
36	Zn_V2: Transformed Variogram Model	70.65	240.23	126.0421913	Normal score	0.985259486	0.277739699		0.383857096		Spherical		15	15	8	0.324643001		Spherical		120	70	25
37	Zn_V3: Transformed Variogram Model	53.46	229.71	31.93709148	Data	1766.034162	615.9927156	0.3488	660.3201731	0.3739	Spherical		15	15	8	466.762829	0.2643	Spherical		120	70	25
38	Zn_V3: Transformed Variogram Model	53.46	229.71	31.93709148	Normal score	0.969211873	0.273215957		0.377604946		Spherical		15	15	8	0.319355312		Spherical		120	70	25
39	Zn_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Data	262084.0666	96892.47941	0.3697	101243.0749	0.3863	Spherical		15	15	8	62873.96757	0.2399	Spherical		120	47.42	25
40	Zn_V5: Transformed Variogram Model	56.97	192.12	20.66299841	Normal score	0.99470828	0.280403266		0.387538346		Spherical		15	15	8	0.325866432		Spherical		120	47.42	25

Table 14-22: La Potosina Search Parameters

	General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of samples		Outlier Restrictions			Sector Search			Drillhole Limit	
	Interpolant Name	Domain	Numeric Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum	Method	Distance	Threshold	Method	Max Samples	Max Empty Sectors	Max Samples per Hole	Apply Drillhole Limit per Sector
0	ID, Ag_V1	Domains_MOB: V1	Ag	200	150	40				Variable Orientation	3	20	None			None			2	1
1	ID, Ag_V2	Domains_MOB: V2	Ag	200	150	40				Variable Orientation	3	20	None			None			2	1
2	ID, Ag_V3	Domains_MOB: V3	Ag	200	150	40				Variable Orientation	3	20	None			None			2	1
3	ID, Ag_V5	Domains_MOB: V5	Ag	200	150	20				Variable Orientation	3	20	None			None			2	1
4	ID, Au_V1	Domains_MOB: V1	Au	200	150	40				Variable Orientation	3	20	None			None			2	1
5	ID, Au_V2	Domains_MOB: V2	Au	200	150	40				Variable Orientation	3	20	None			None			2	1
6	ID, Au_V3	Domains_MOB: V3	Au	200	150	40				Variable Orientation	3	20	None			None			2	1
7	ID, Au_V5	Domains_MOB: V5	Au	200	150	20				Variable Orientation	3	20	None			None			2	1
8	ID, Cu_V1	Domains_MOB: V1	Cu_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
9	ID, Cu_V2	Domains_MOB: V2	Cu_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
10	ID, Cu_V3	Domains_MOB: V3	Cu_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
11	ID, Cu_V5	Domains_MOB: V5	Cu_ppm	200	150	20				Variable Orientation	3	20	None			None			2	1
12	ID, Pb_V1	Domains_MOB: V1	Pb_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
13	ID, Pb_V2	Domains_MOB: V2	Pb_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
14	ID, Pb_V3	Domains_MOB: V3	Pb_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
15	ID, Pb_V5	Domains_MOB: V5	Pb_ppm	200	150	20				Variable Orientation	3	20	None			None			2	1
16	ID, Zn_V1	Domains_MOB: V1	Zn_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
17	ID, Zn_V2	Domains_MOB: V2	Zn_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1

table continues…

	General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of samples		Outlier Restrictions			Sector Search			Drillhole Limit	
	Interpolant Name	Domain	Numeric Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum	Method	Distance	Threshold	Method	Max Samples	Max Empty Sectors	Max Samples per Hole	Apply Drillhole Limit per Sector
18	ID, Zn_V3	Domains_MOB: V3	Zn_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
19	ID, Zn_V5	Domains_MOB: V5	Zn_ppm	200	150	20				Variable Orientation	3	20	None			None			2	1
20	Kr, Ag_V1	Domains_MOB: V1	Ag	200	150	40				Variable Orientation	3	20	None			None			2	1
21	Kr, Ag_V2	Domains_MOB: V2	Ag	200	150	40				Variable Orientation	3	20	None			None			2	1
22	Kr, Ag_V3	Domains_MOB: V3	Ag	200	150	40				Variable Orientation	3	20	None			None			2	1
23	Kr, Ag_V5	Domains_MOB: V5	Ag	200	150	20				Variable Orientation	3	20	None			None			2	1
24	Kr, Au_V1	Domains_MOB: V1	Au	200	150	40				Variable Orientation	3	20	None			None			2	1
25	Kr, Au_V2	Domains_MOB: V2	Au	200	150	40				Variable Orientation	3	20	None			None			2	1
26	Kr, Au_V3	Domains_MOB: V3	Au	200	150	40				Variable Orientation	3	20	None			None			2	1
27	Kr, Au_V5	Domains_MOB: V5	Au	200	150	20				Variable Orientation	3	20	None			None			2	1
28	Kr, Cu_V1	Domains_MOB: V1	Cu_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
29	Kr, Cu_V2	Domains_MOB: V2	Cu_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
30	Kr, Cu_V3	Domains_MOB: V3	Cu_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
31	Kr, Cu_V5	Domains_MOB: V5	Cu_ppm	200	150	20				Variable Orientation	3	20	None			None			2	1
32	Kr, Pb_V1	Domains_MOB: V1	Pb_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
33	Kr, Pb_V2	Domains_MOB: V2	Pb_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
34	Kr, Pb_V3	Domains_MOB: V3	Pb_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
35	Kr, Pb_V5	Domains_MOB: V5	Pb_ppm	200	150	20				Variable Orientation	3	20	None			None			2	1
36	Kr, Zn_V1	Domains_MOB: V1	Zn_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1

table continues…

	General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of samples		Outlier Restrictions			Sector Search			Drillhole Limit	
	Interpolant Name	Domain	Numeric Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum	Method	Distance	Threshold	Method	Max Samples	Max Empty Sectors	Max Samples per Hole	Apply Drillhole Limit per Sector
37	Kr, Zn_V2	Domains_MOB: V2	Zn_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
38	Kr, Zn_V3	Domains_MOB: V3	Zn_ppm	200	150	40				Variable Orientation	3	20	None			None			2	1
39	Kr, Zn_V5	Domains_MOB: V5	Zn_ppm	200	150	20				Variable Orientation	3	20	None			None			2	1
40	NN, Ag_V1	Domains_MOB: V1	Ag	200	150	40	55.72	213.85	157.8142458											
41	NN, Ag_V2	Domains_MOB: V2	Ag	200	150	40	55.72	213.85	157.8142458											
42	NN, Ag_V3	Domains_MOB: V3	Ag	200	150	40	53.46	229.71	91.46											
43	NN, Ag_V5	Domains_MOB: V5	Ag	200	150	20	53.46	229.71	91.46											
44	NN, Au_V1	Domains_MOB: V1	Au	200	150	40	55.72	213.85	157.8142458											
45	NN, Au_V2	Domains_MOB: V2	Au	200	150	40	55.72	213.85	157.8142458											
46	NN, Au_V3	Domains_MOB: V3	Au	200	150	40	53.46	229.71	91.46											
47	NN, Au_V5	Domains_MOB: V5	Au	200	150	20	53.46	229.71	91.46											
48	NN, Cu_V1	Domains_MOB: V1	Cu_ppm	200	150	40	55.72	213.85	157.8142458											
49	NN, Cu_V2	Domains_MOB: V2	Cu_ppm	200	150	40	55.72	213.85	157.8142458											
50	NN, Cu_V3	Domains_MOB: V3	Cu_ppm	200	150	40	53.46	229.71	91.46											
51	NN, Cu_V5	Domains_MOB: V5	Cu_ppm	200	150	20	53.46	229.71	91.46											
52	NN, Pb_V1	Domains_MOB: V1	Pb_ppm	200	150	40	55.72	213.85	157.8142458											
53	NN, Pb_V2	Domains_MOB: V2	Pb_ppm	200	150	40	55.72	213.85	157.8142458											
54	NN, Pb_V3	Domains_MOB: V3	Pb_ppm	200	150	40	53.46	229.71	91.46											
55	NN, Pb_V5	Domains_MOB: V5	Pb_ppm	200	150	20	53.46	229.71	91.46											

table continues…

	General			Ellipsoid Ranges			Ellipsoid Directions			Variable Orientation	Number of samples		Outlier Restrictions			Sector Search			Drillhole Limit	
	Interpolant Name	Domain	Numeric Values	Maximum	Intermediate	Minimum	Dip	Dip Azimuth	Pitch		Minimum	Maximum	Method	Distance	Threshold	Method	Max Samples	Max Empty Sectors	Max Samples per Hole	Apply Drillhole Limit per Sector
56	NN, Zn_V1	Domains_MOB: V1	Zn_ppm	200	150	40	55.72	213.85	157.8142458											
57	NN, Zn_V2	Domains_MOB: V2	Zn_ppm	200	150	40	55.72	213.85	157.8142458											
58	NN, Zn_V3	Domains_MOB: V3	Zn_ppm	200	150	40	53.46	229.71	91.46											
59	NN, Zn_V5	Domains_MOB: V5	Zn_ppm	200	150	20	53.46	229.71	91.46											

14.1.12 Resource Block Models

14.1.12.1 Block Model Configurations

The specifications for the estimation block models are summarized in Table 14-23 to Table 14-27.

Table 14-23: Avino (ET) Deposit: Estimation Block Model Specifications

Blocks	X	Y	Z
Parent block size:	16	16	16
Sub-block count:	8	8	8
Minimum size:	2	2	2
Extents			
Base point:	569720.00	2712165.00	2383.00
Boundary size:	1440.00	976.00	960.00
Azimuth:	0.00	degrees	Enclose Object
Dip:	0.00	degrees	Set Angles From
Pitch:	0.00	degrees	
Size in blocks:	90 × 61 × 60 = 329,400		

Table 14-24: San Gonzalo (SG) Deposit: Estimation Block Model Specifications

Parent blocks:	X	Y	Z
Parent block size:	10	5	10
Sub-blocks			
☐ Variable height		Minimum height:	0.000
Sub-block count:	4	20	10
Extents			
Base point:	571065.00	2714432.50	2355.00
Boundary size:	1630.00	360.00	560.00
Dip:	0.00 degrees		Enclose Object
Azimuth:	35.00 degrees		Set Angles From
Size in blocks:	163 × 72 × 56 = 657,216		

Table 14-25: Tailings Deposit: Estimation Block Model Specifications

Blocks	X	Y	Z
Parent block size:	30	30	4
Sub-block count:	8	8	8
Minimum size:	3.75	3.75	0.5
Extents			
Base point:	569330.00	2712030.00	2500.00
Boundary size:	990.00	870.00	360.00
Azimuth:	0.00	degrees	Enclose Object
Dip:	0.00	degrees	Set Angles From
Pitch:	0.00	degrees	
Size in blocks:	33 × 29 × 90 = 86,130		

Table 14-26: Guadalupe Deposit: Estimation Block Model Specifications

Blocks	X	Y	Z
Parent block size:	5	5	5
Sub-block count:	8	8	8
Minimum size:	0.625	0.625	0.625
Extents			
Base point:	570500.00	2713250.00	2370.00
Boundary size:	1490.00	750.00	500.00
Azimuth:	0.00	degrees	Enclose Object
Dip:	0.00	degrees	Set Angles From
Pitch:	0.00	degrees	
Size in blocks:	298 × 150 × 100 = 4,470,000		

Table 14-27: La Potosina Deposit: Estimation Block Model Specifications

Blocks	X	Y	Z
Parent block size:	12	12	12
Sub-block count:	8	8	16
Minimum size:	1.5	1.5	0.75
Extents			
Base point:	570170.00	2718190.00	1820.00
Boundary size:	1020.00	984.00	444.00
Azimuth:	44.90	degrees	Enclose Object
Dip:	-52.74	degrees	Set Angles From
Pitch:	0.00	degrees	
Size in blocks:	85 × 82 × 37 = 257,890		

14.1.12.2 Interpolation

The reported resource relies on OK as the best unbiased linear estimator of grade. Other interpolation methods, including ID2 and NN, were employed for model validation and are retained in the block models.

14.1.13 Model Validation

14.1.13.1 Statistics

Mean metal grade values (silver, gold, copper, lead and zinc) for estimated blocks using OK and ID2 for the Avino ET domains are shown in Table 14-28. The two types of estimates are similar, but the OK estimates are preferred as the spatial weighting applied to individual informing composites is based on the empirical results of experimental variography and is applied in a kriging matrix which provides the best linear unbiased estimator.

Table 14-28: Avino ET: Block Grade Estimates

Name		Block Count	Volume	Mean	Std. Dev.	CV	Variance	Minimum	Median	Maximum
BX		1,025,214	23,274,728							
	AgID	878,444	20,351,352	51.29	46.21	0.90	0.62	37.62	510.04	AgID
	AgOK	860,420	20,024,936	56.57	40.80	0.72	1.34	45.65	293.04	AgOK
	AuID	878,444	20,351,352	0.42	0.48	1.14	0.00	0.27	5.06	AuID
	AuOK	860,420	20,024,936	0.47	0.42	0.91	0.01	0.34	3.78	AuOK
	CuID	878,444	20,351,352	0.42	0.35	0.84	0.01	0.31	4.26	CuID
	CuOK	860,420	20,024,936	0.43	0.29	0.68	0.01	0.35	1.86	CuOK
	PbID	808,374	18,833,360	0.13	0.21	1.58	0.00	0.06	1.95	PbID
	PbOK	808,374	18,833,360	0.13	0.17	1.29	0.00	0.07	1.63	PbOK
	ZnID	808,374	18,833,360	0.16	0.18	1.09	0.01	0.11	1.25	ZnID
	ZnOK	808,374	18,833,360	0.15	0.12	0.81	0.01	0.12	1.03	ZnOK
BX_Hw		161,139	1,357,376							
	AgID	147,680	1,249,648	33.40	25.43	0.76	1.53	25.32	132.31	AgID
	AgOK	131,262	1,117,520	36.22	21.33	0.59	5.59	32.57	104.74	AgOK
	AuID	147,680	1,249,648	0.51	0.36	0.70	0.01	0.48	1.86	AuID
	AuOK	131,262	1,117,520	0.53	0.27	0.51	0.07	0.54	1.38	AuOK
	CuID	147,680	1,249,648	0.20	0.13	0.62	0.03	0.15	0.97	CuID
	CuOK	131,262	1,117,520	0.20	0.09	0.46	0.07	0.16	0.56	CuOK
	PbID	147,680	1,249,648	0.04	0.05	1.22	0.00	0.02	0.27	PbID
	PbOK	131,262	1,117,520	0.05	0.05	0.95	0.00	0.04	0.20	PbOK
	ZnID	147,680	1,249,648	0.05	0.05	0.93	0.01	0.03	0.22	ZnID
	ZnOK	131,262	1,117,520	0.06	0.05	0.75	0.01	0.05	0.18	ZnOK

Mean metal grade values (silver and gold) for estimated blocks using OK and ID2 for the San Gonzalo Vein system domains are shown in Table 14-29. The two types of estimates are comparable, particularly in the Main Zone, which is the largest domain.

Table 14-29: San Gonzalo Vein: Block Estimates and Composite Sample Grades

Name		Block Count	Volume	Mean	Std. Dev.	CV	Variance	Minimum	L. Quartile	Median	U. Quartile	Maximum
SG1		356842	568531.25									
	Ag_ID	318859	509182.81	159.86	195.31	1.22	38144.29	2.30	37.33	94.93	211.95	3946.41
	Ag_OK	318859	509182.81	156.18	181.65	1.16	32995.31	3.04	42.33	100.38	203.94	2301.70
	Au_ID	271130	434606.25	0.81	1.19	1.47	1.41	0.01	0.19	0.41	0.99	28.14
	Au_OK	271130	434606.25	0.81	1.23	1.51	1.50	0.02	0.19	0.41	1.01	23.15
SG2		47936	74900.00									
	Ag_ID	36088	56387.50	38.81	30.67	0.79	940.36	5.98	21.94	31.47	44.11	624.41
	Ag_OK	36088	56387.50	47.95	33.71	0.70	1136.40	10.92	20.70	37.65	69.21	257.38
	Au_ID	24189	37795.31	0.31	0.27	0.85	0.07	0.05	0.09	0.16	0.50	1.16
	Au_OK	24189	37795.31	0.33	0.24	0.73	0.06	0.05	0.11	0.24	0.50	1.06
SG3		10025	15664.06									
	Ag_ID	5840	9125.00	78.37	23.01	0.29	529.47	22.23	65.38	79.68	92.21	200.60
	Ag_OK	9693	15145.31	80.66	24.89	0.31	619.35	29.95	70.31	75.23	82.29	249.26
	Au_ID	4535	7085.94	0.40	0.18	0.44	0.03	0.18	0.35	0.37	0.39	1.67
	Au_OK	4535	7085.94	0.40	0.17	0.43	0.03	0.22	0.32	0.34	0.38	1.24
SG4		122062	190721.88									
	Ag_ID	27942	43659.38	255.74	179.66	0.70	32277.38	27.03	105.08	179.65	416.42	1466.25
	Ag_OK	27942	43659.38	263.03	174.03	0.66	30285.34	33.52	100.17	239.88	402.12	821.71
	Au_ID	13551	21173.44	0.84	0.56	0.67	0.32	0.12	0.46	0.68	1.06	5.07
	Au_OK	13551	21173.44	0.91	0.61	0.67	0.37	0.21	0.47	0.64	1.49	2.75
SG5		47581	74345.31									
	Ag_ID	24108	37668.75	98.25	80.16	0.82	6426.00	14.83	19.64	80.86	170.84	326.19

table continues…

Name		Block Count	Volume	Mean	Std. Dev.	CV	Variance	Minimum	L. Quartile	Median	U. Quartile	Maximum
	Ag_OK	24108	37668.75	94.40	64.34	0.68	4139.40	13.57	22.95	93.00	156.26	259.26
	Au_ID	14339	22404.69	0.68	0.63	0.93	0.40	0.06	0.12	0.43	1.23	2.42
	Au_OK	14339	22404.69	0.67	0.57	0.85	0.33	0.09	0.14	0.48	1.29	2.02
SG6		100563	157129.69									
	Ag_ID	45332	70831.25	47.82	44.86	0.94	2012.74	4.21	10.78	29.16	77.66	197.34
	Ag_OK	45332	70831.25	48.69	28.05	0.58	786.74	3.76	22.14	48.89	68.46	123.33
	Au_ID	26683	41692.19	0.30	0.31	1.04	0.10	0.01	0.05	0.16	0.49	1.57
	Au_OK	26683	41692.19	0.39	0.28	0.73	0.08	0.01	0.11	0.37	0.66	0.94

Mean values for estimated blocks and composites used for the estimation in the oxide tailings model are shown in Table 14-30. The block estimates show lower silver grades than the composites due to the declustering effect of kriging and the large numbers of relatively high-grade composites in the development sampling.

Table 14-30: Oxide Tailings: Block Estimates and Composite Sample Grades

Estimator	Ag (g/t)	Au (g/t)	Cu (%)	Zn (%)	Pb (%)
Ordinary Kriging	93.24	0.48	0.130	0.950	0.150
Nearest Neighbour	92.56	0.48	0.129	0.930	0.150
Inverse Distance	93.94	0.48	0.128	0.949	0.151
Composites	96.48	0.46	0.125	0.958	0.153
Number of Blocks	45,607	-	-	-	-

14.1.13.2 Sections

The spatial pattern of metal grade distributions for the Avino Vein is shown in Figure 14-17 to Figure 14-20, inclusive. Figure 14-17 shows a typical transverse section illustrating the interaction between the secondary (NE1) zone developed on the footwall side of the main Avino Vein system. Figure 14-18 to Figure 14-20 are longitudinal sections viewed from the south, showing silver, gold, and copper grades with high-grade zones tending to plunge at about 40° to the west. The spatial pattern conforms to the geometric pattern of historic stoping volumes and the experimental variography.

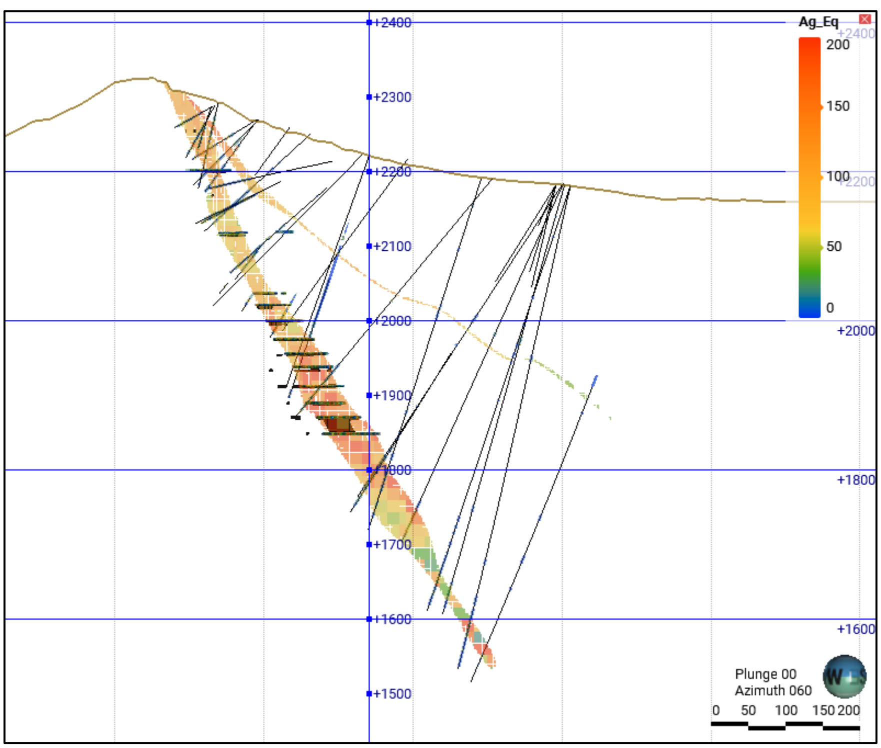

Figure 14-17: Avino Vein: Typical Transverse Section, Looking Northeast Showing the Block Model Colour Coded by Silver Equivalent Grade (Red Pennant 2023)

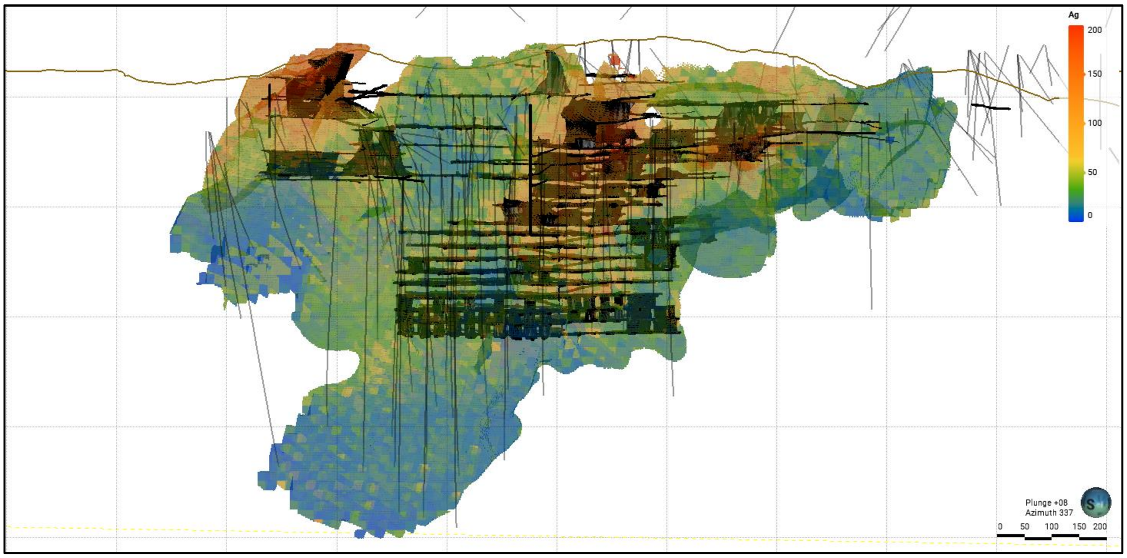

Figure 14-18: Avino Vein: Longitudinal Section Showing the Block Model Colour Coded by Silver Grade (Red Pennant 2022)

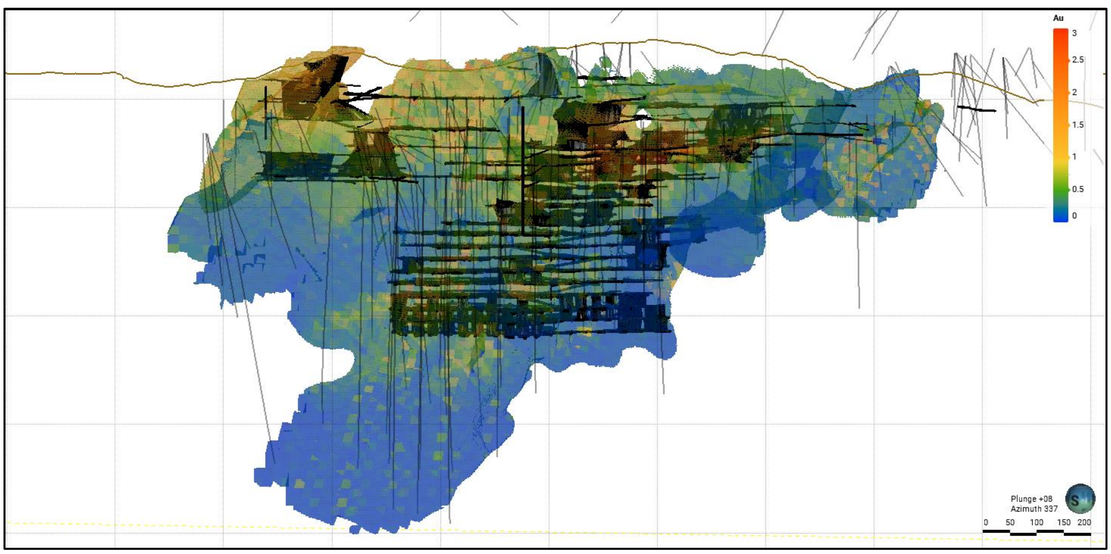

Figure 14-19: Avino Vein: Longitudinal Section Showing the Block Model Colour Coded by Gold Grade (Red Pennant 2022)

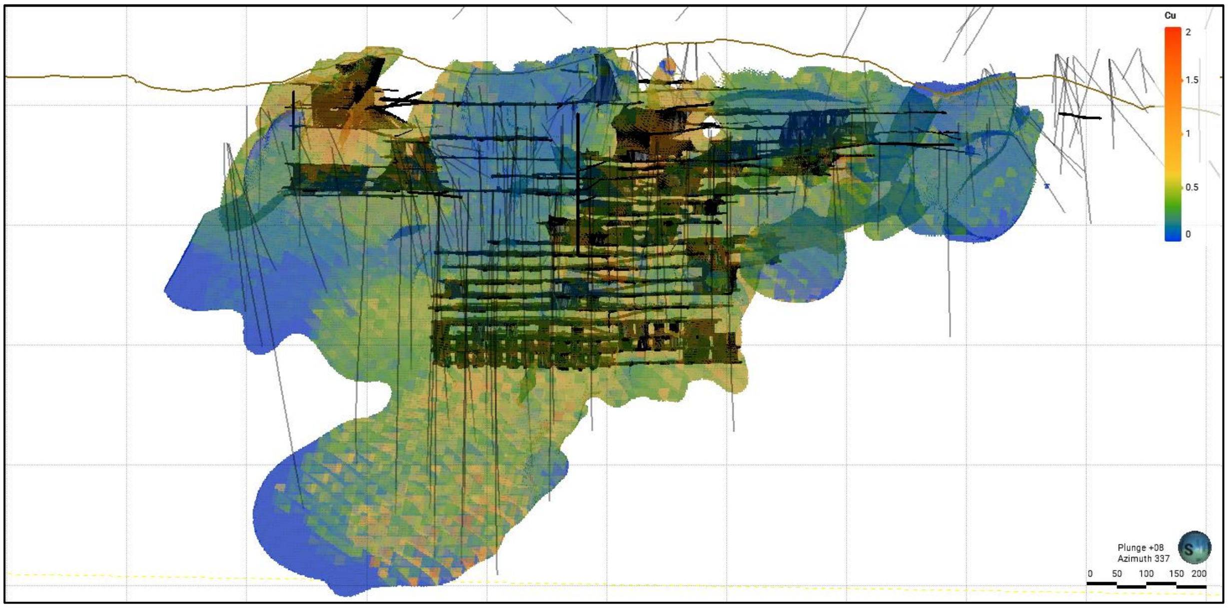

Figure 14-20: Avino Vein: Longitudinal Section Showing the Block Model Colour Coded by Copper Grade (Red Pennant 2020)

The spatial pattern of metal grade distributions for the San Gonzalo Vein is shown in Figure 14-21 to Figure 14-24, inclusive. Figure 14-21 shows a typical transverse section illustrating the relatively narrow San Gonzalo Vein and the Anjelica Vein (SG4). Figure 14-22 to Figure 14-24 are longitudinal sections viewed from the south showing silver, gold, and copper grades with high-grade zones with a subhorizontal tendency despite some local steepening. It has become clear by continued underground mapping and sampling that the San Gonzalo mineralization is depth-constrained within a 250 m interval, probably due to pressure constraining boiling levels during the residence time of the mineralizing fluids.

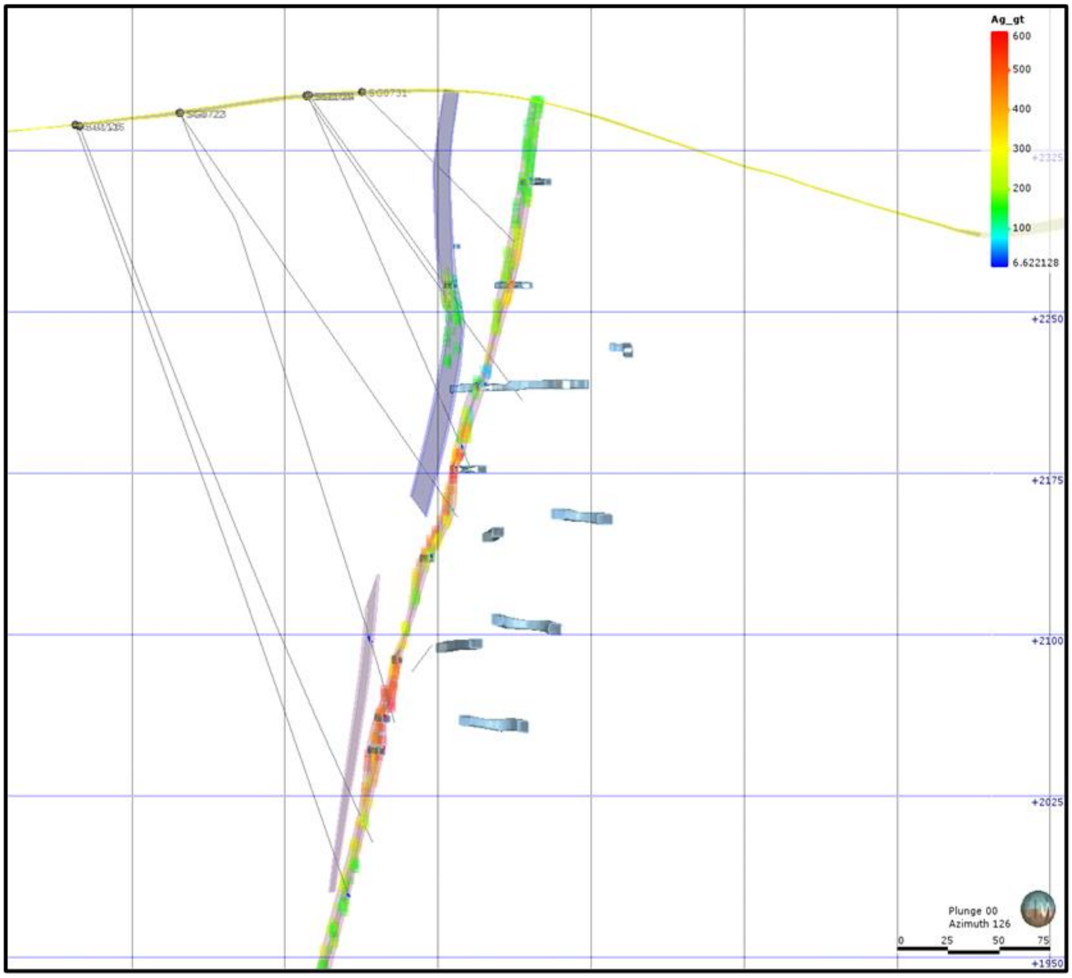

Figure 14-21: San Gonzalo Vein: Typical Transverse Section, Looking East Aligned Along Drillhole SG1115 Showing the Block Model Centroids Colour Coded by Silver Grade (Red Pennant 2020)

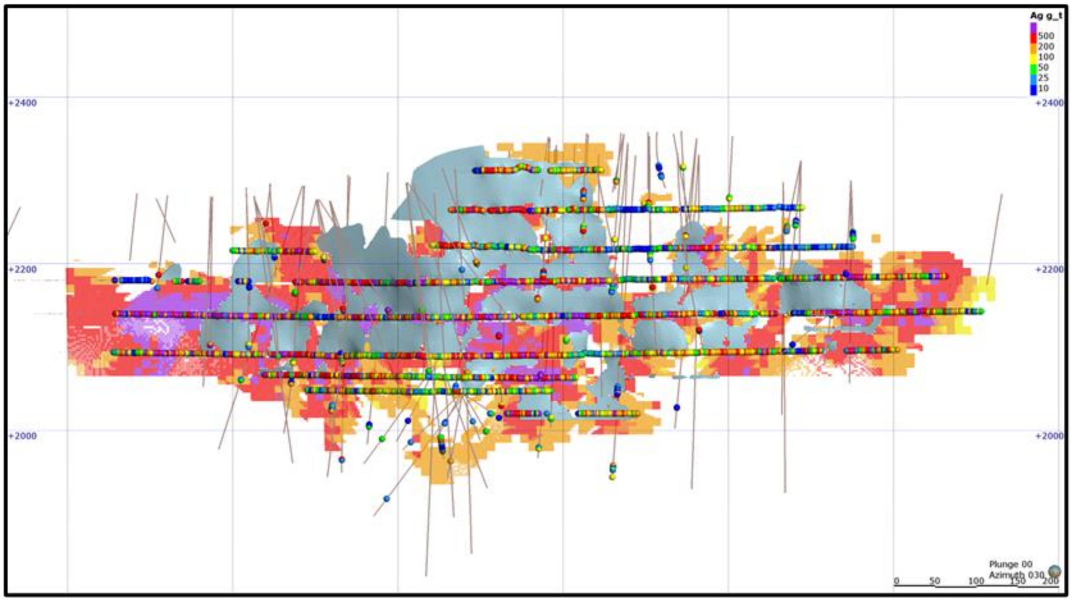

Figure 14-22: San Gonzalo Vein: Longitudinal Section Showing the Block Model Centroids Colour Coded by Silver Grade (Red Pennant 2020)

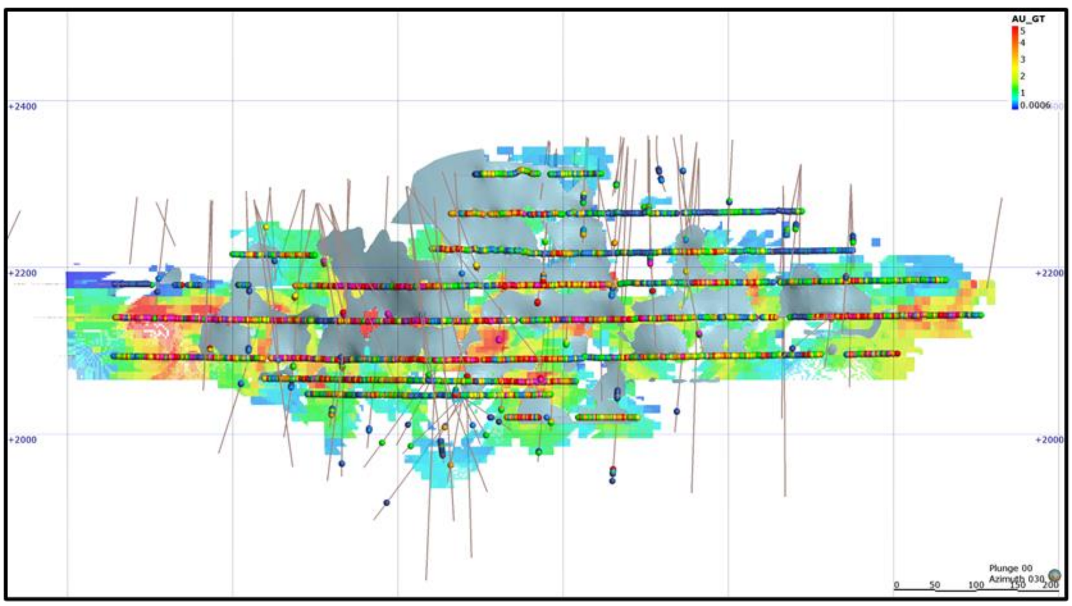

Figure 14-23: San Gonzalo Vein: Longitudinal Section Showing the Block Model Centroids Color Coded by Gold Grade (Red Pennant 2020)

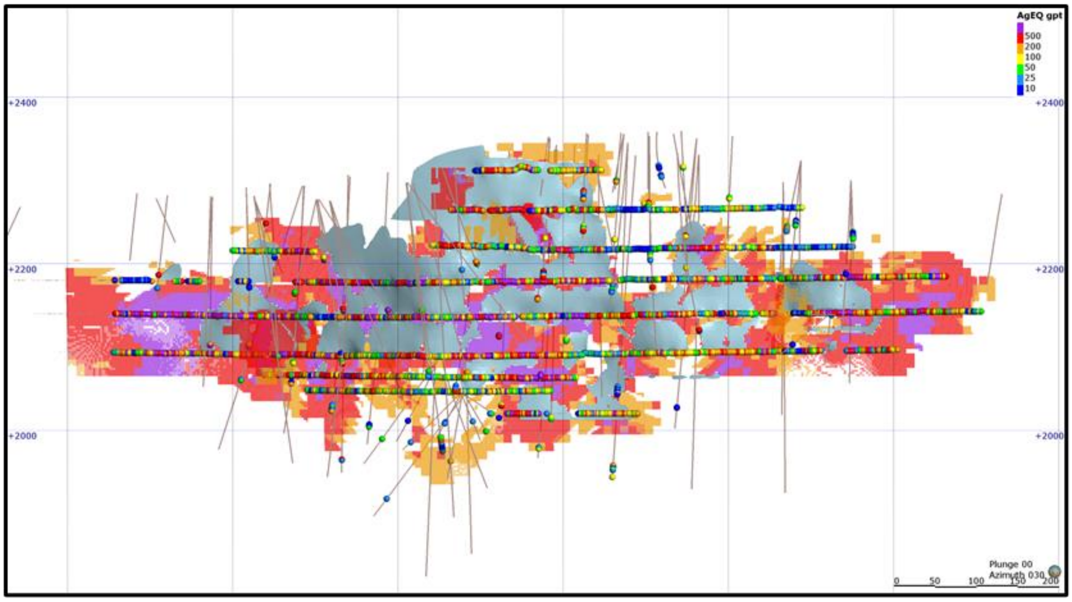

Figure 14-24: San Gonzalo Vein: Longitudinal Section Showing the Block Model Centroids Colour Coded by Silver Equivalent (Red Pennant 2020)

14.1.13.3 Swath Plots

Swath plots were generated for the underground vein deposits to compare trends in the estimated grades for the three estimation methods (OK, ID, and NN) in the block models to the source sampling data. The estimation methods for comparison are OK (green), NN (red), and ID2 (blue) block estimates for silver, gold, and copper, and averages were generated for slices oriented parallel to the eastings, northings, and elevations. The widths of the swaths (or slices) are 20 m for eastings and 10 m for elevations, and the number of blocks is plotted as histograms. The average of assay composites is shown as a black line.

Figure 14-25 through Figure 14-29 display the swath plots for the ET (Avino main) deposit, comparing block model estimates (lines) and the number of estimated blocks (bars).

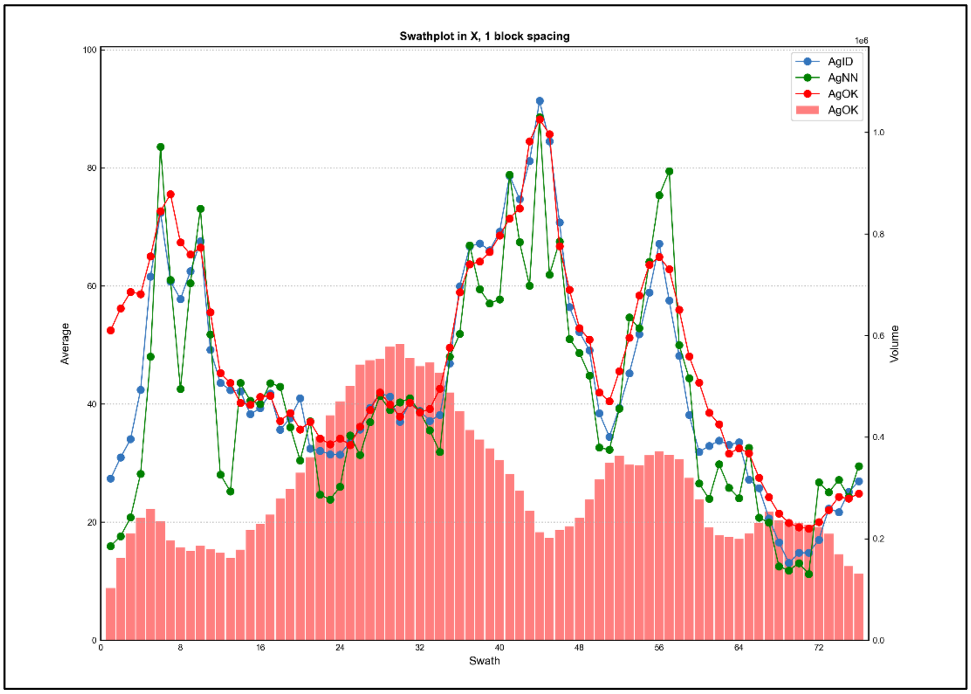

Figure 14-25: Avino Vein, Swath Plot for Silver, Eastings (Red Pennant 2022)

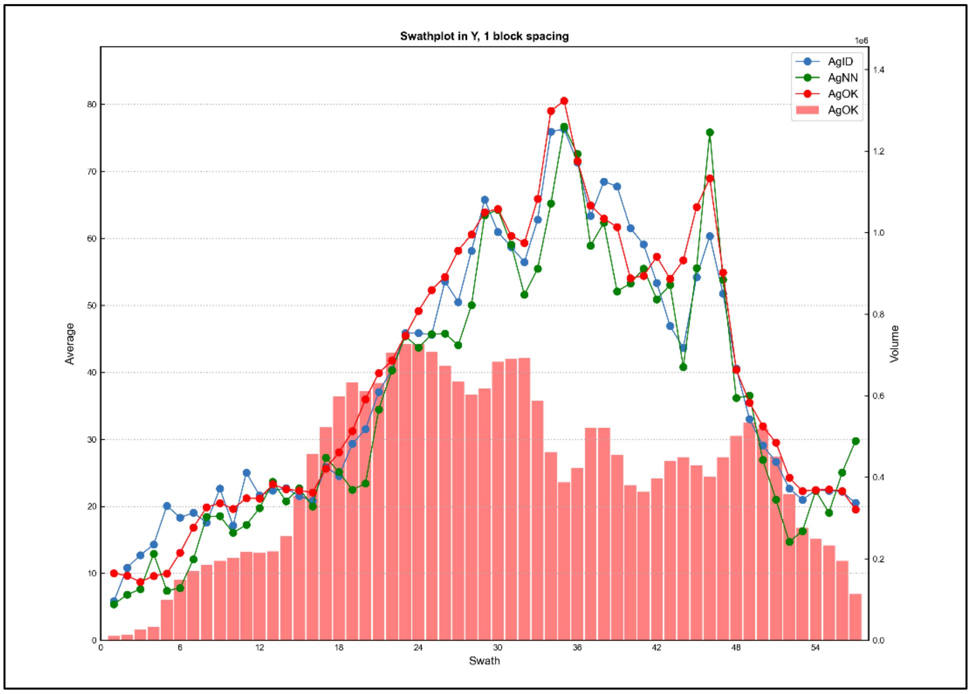

Figure 14-26: Avino Vein, Swath Plot for Silver, Northings (Red Pennant 2022)

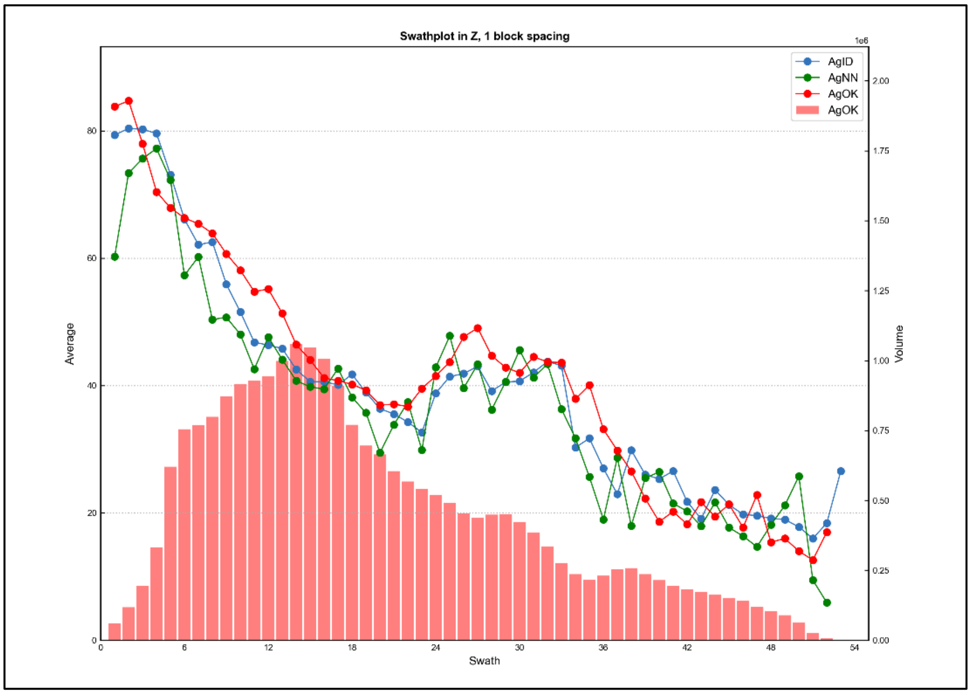

Figure 14-27: Avino Vein, Swath Plot for Silver, Elevation (Red Pennant 2020)

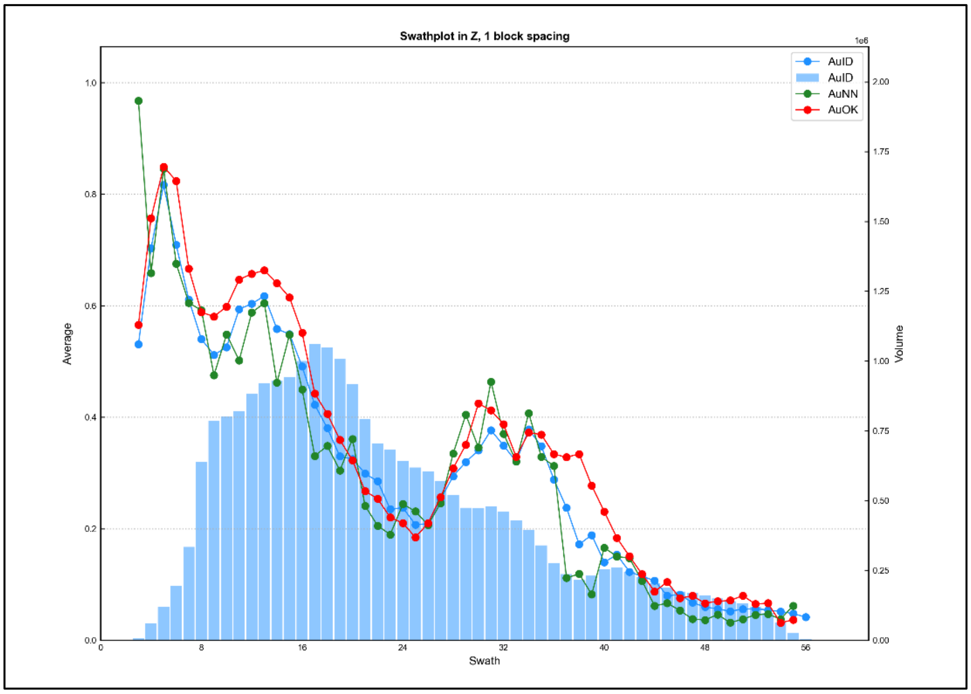

Figure 14-28: Avino Vein, Swath Plot for Gold, Elevation (Red Pennant 2022)

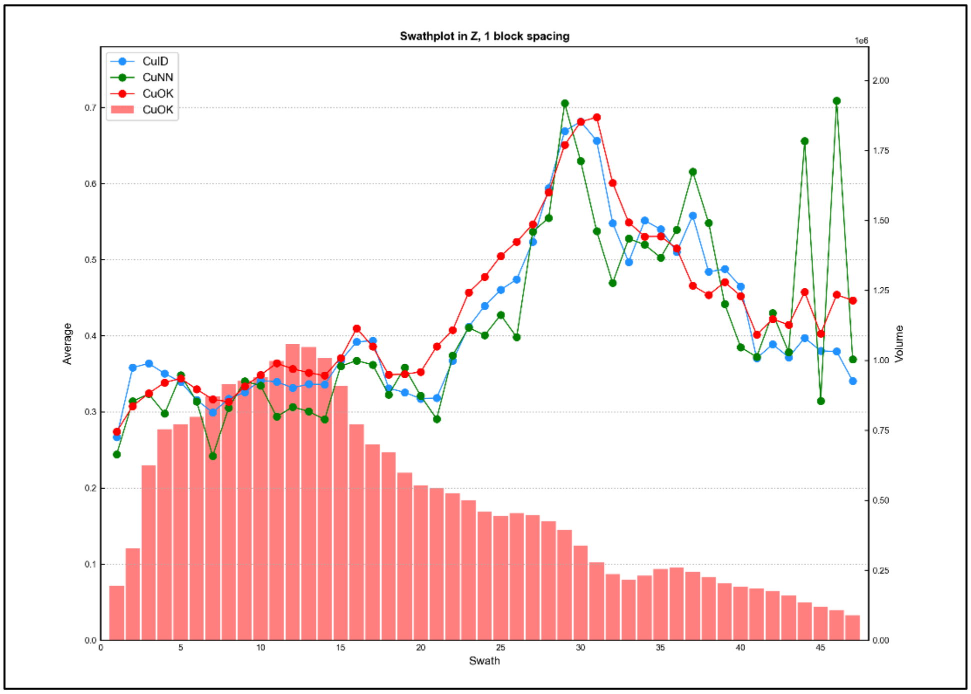

Figure 14-29: Avino Vein, Swath Plot for Copper, Elevation (Red Pennant 2022)

Figure 14-30 through Figure 14-35 display the swath plots for the San Gonzalo deposit, comparing block model estimates and sample grades.

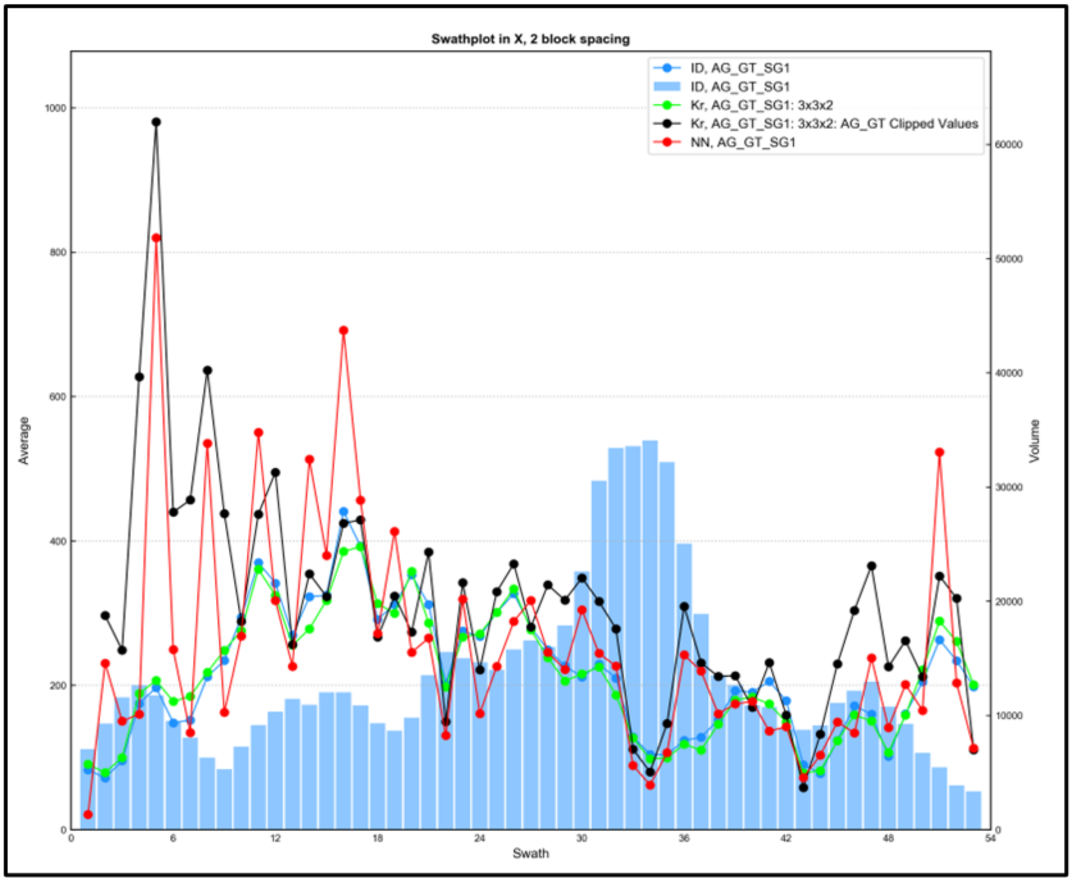

Figure 14-30: San Gonzalo Vein, Swath Plot for Silver, Eastings (Red Pennant 2020)

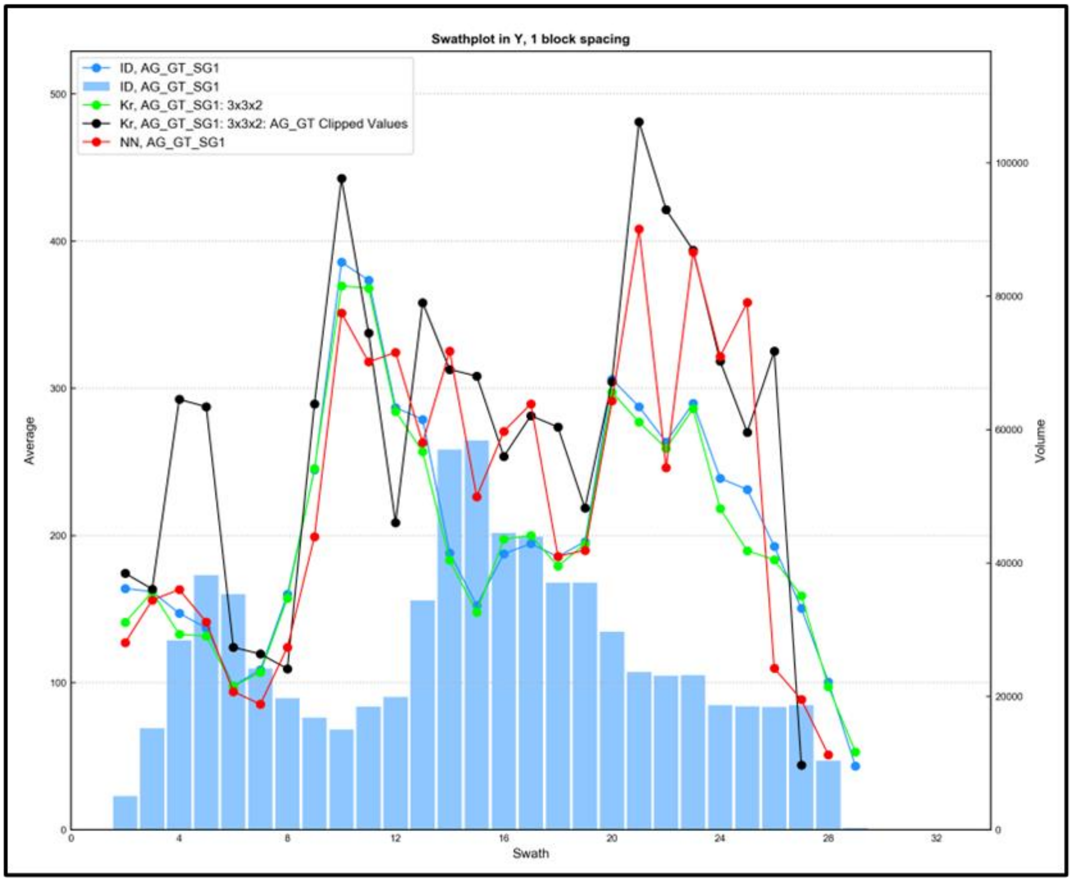

Figure 14-31: San Gonzalo Vein, Swath Plot for Silver, Northings (Red Pennant 2020)

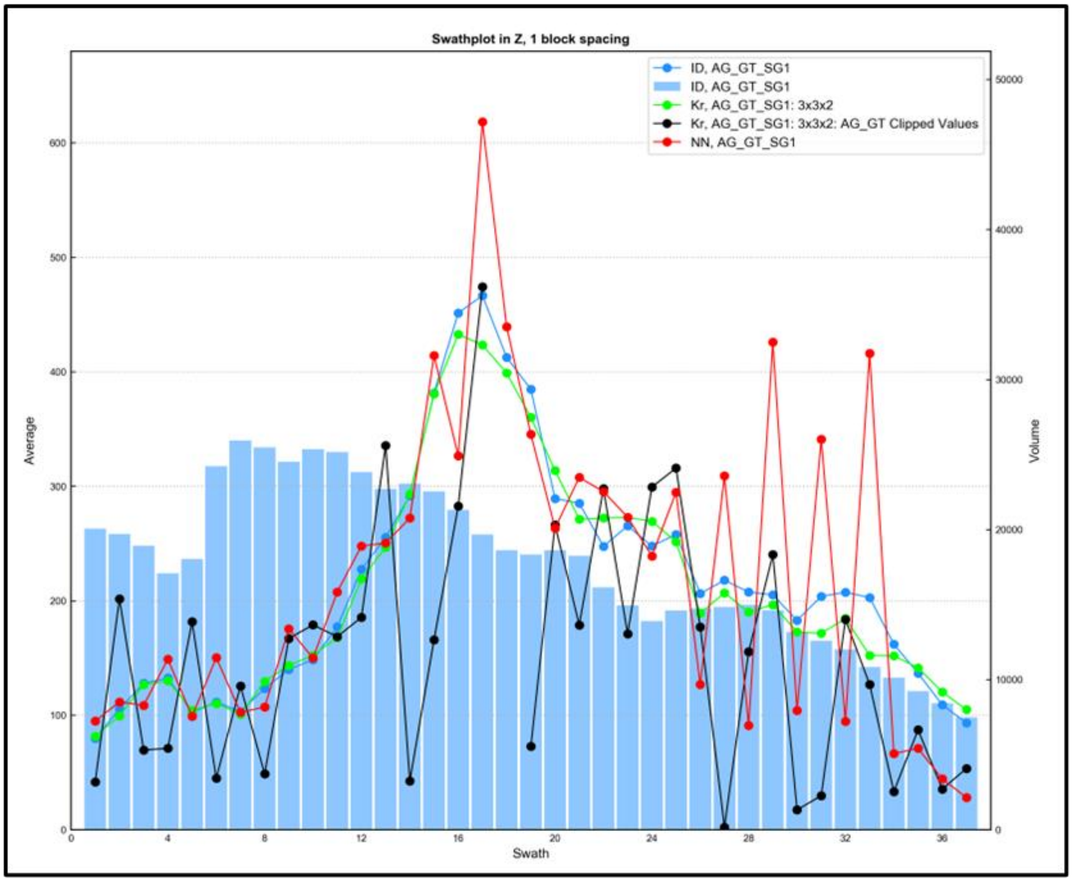

Figure 14-32: San Gonzalo Vein, Swath Plot for Silver, Elevation (Red Pennant 2020)

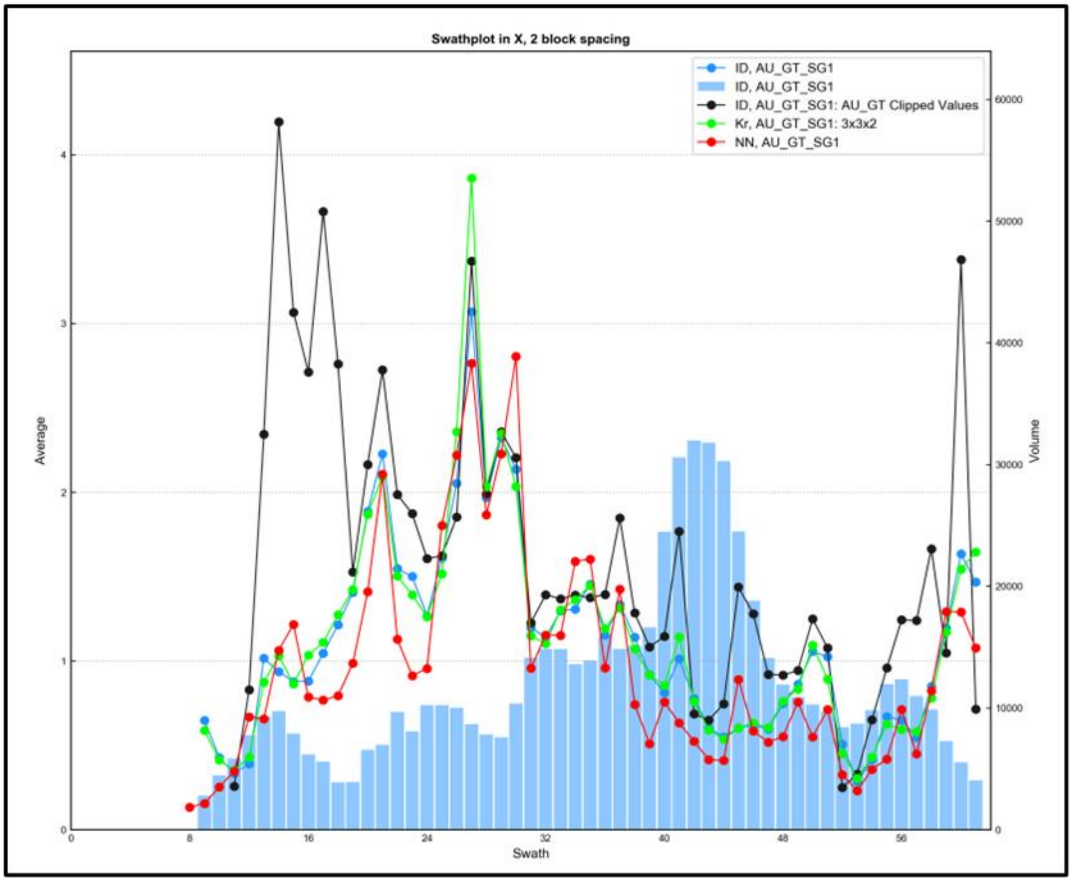

Figure 14-33: San Gonzalo Vein, Swath Plot for Gold, Eastings (Red Pennant 2020)

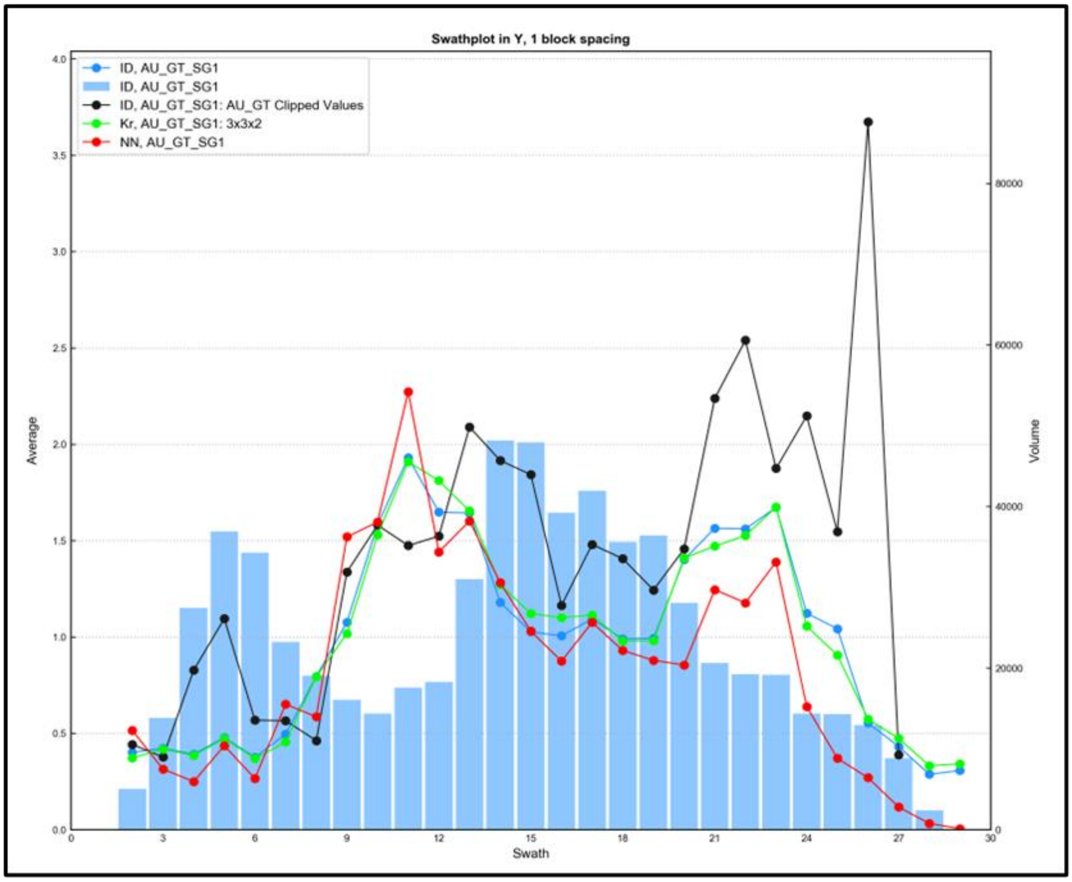

Figure 14-34: San Gonzalo Vein, Swath Plot for Gold, Northings (Red Pennant 2020)

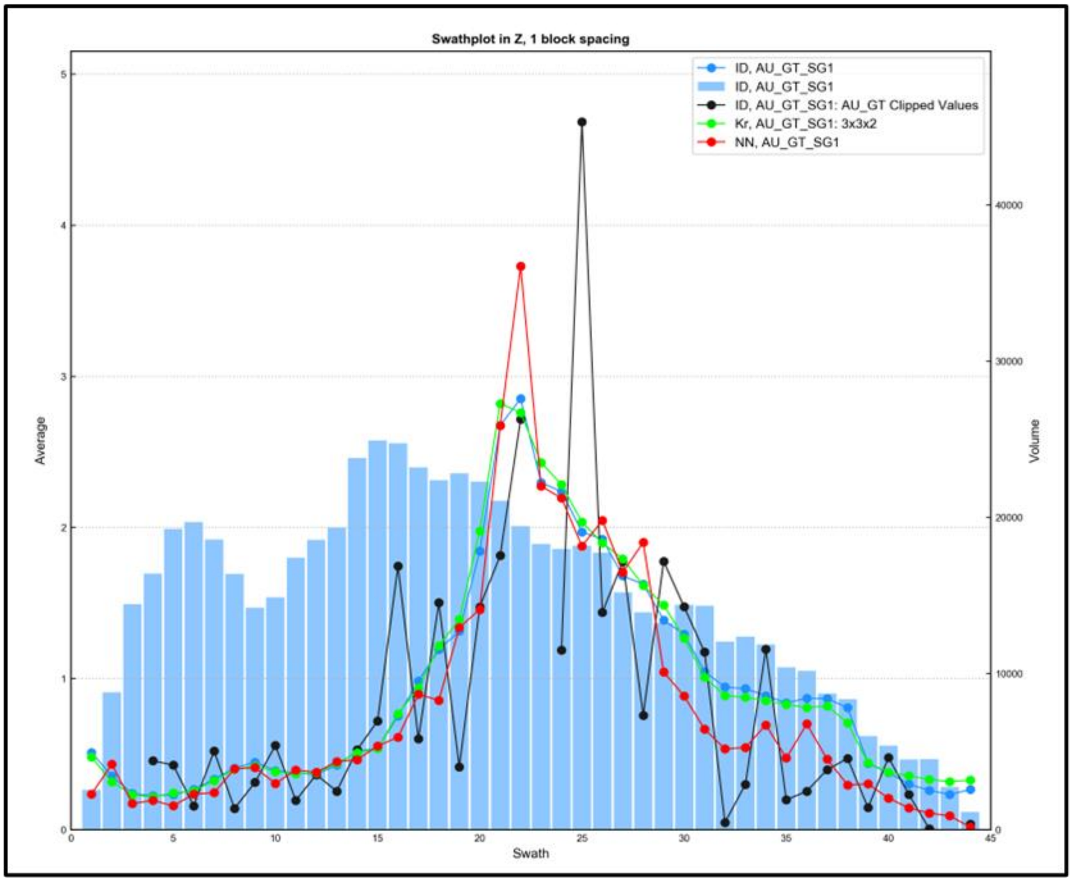

Figure 14-35: San Gonzalo Vein, Swath Plot for Gold, Elevation (Red Pennant 2020)

Figure 14-32 and Figure 14-35 illustrate the restricted vertical interval within which the pressure conditions for silver and gold grade mineralization were operative in the San Gonzalo mineralization event. Figure 14-28 shows a similar pattern for gold in the Avino Vein. However, Figure 14-27 and Figure 14-29 indicate that the silver and copper mineralization window for the Avino Vein system is more persistent with depth.

Figure 14-36 through Figure 14-41 display the swath plots for the oxide tailings deposit, comparing block model estimates and sample grades.

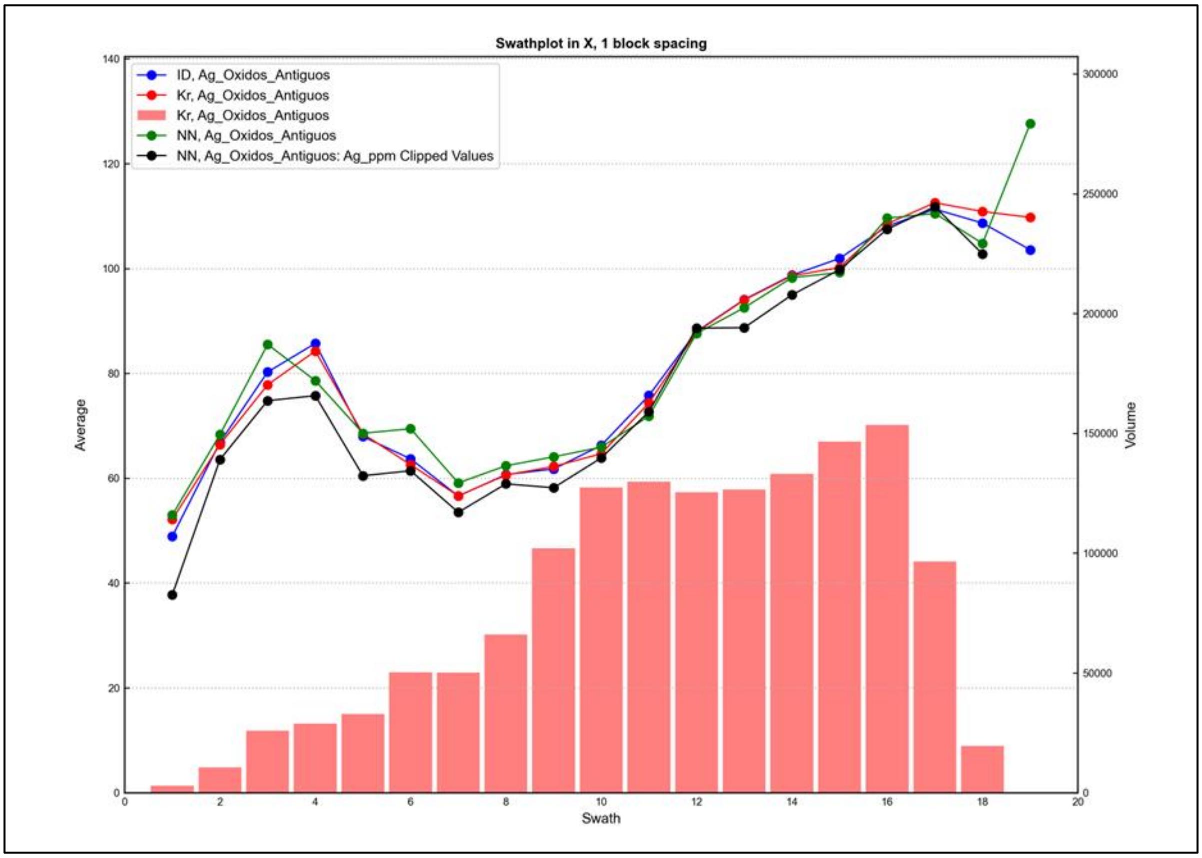

Figure 14-36: Older Oxide Tailings Domain, Swath Plot for Silver, Easting (Red Pennant 2022)

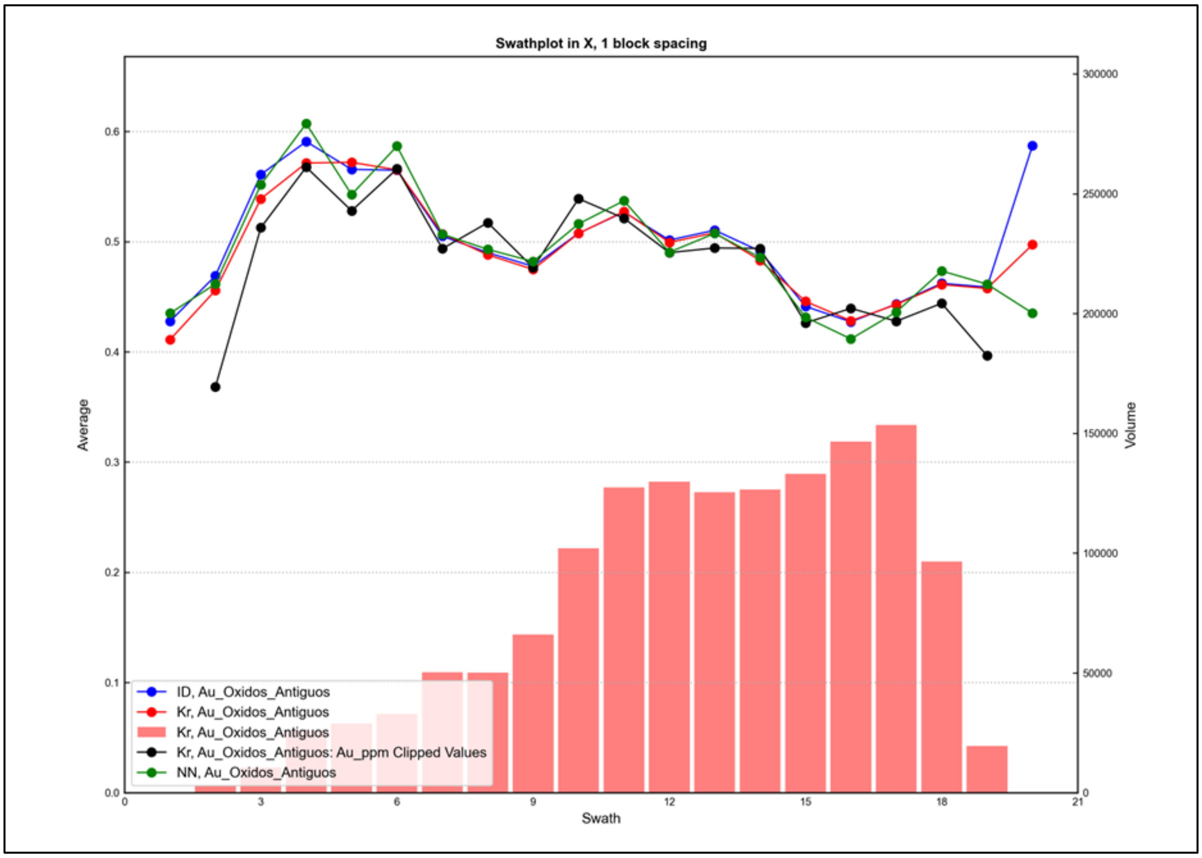

Figure 14-37: Older Oxide Tailings Domain, Swath Plot for Gold, Easting (Red Pennant 2022)

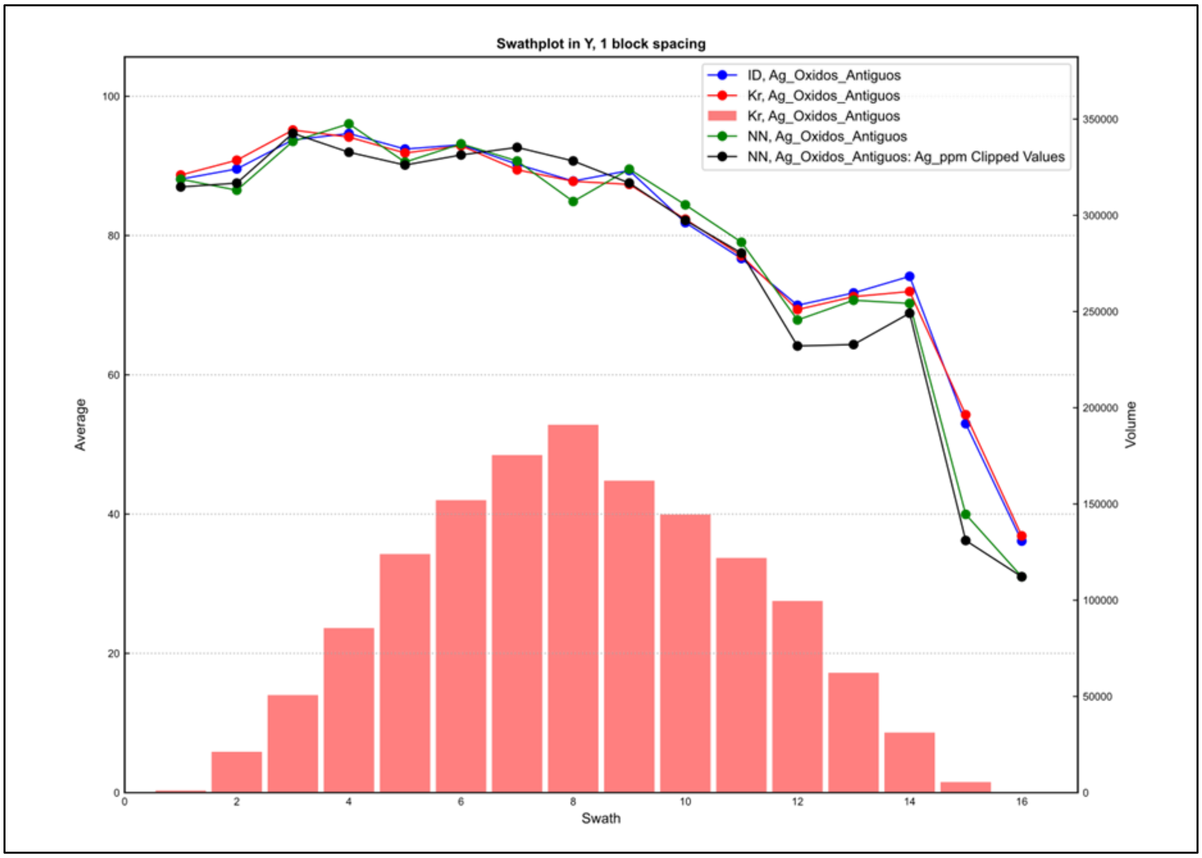

Figure 14-38: Older Oxide Tailings Domain, Swath Plot for Silver, Northing (Red Pennant 2022)

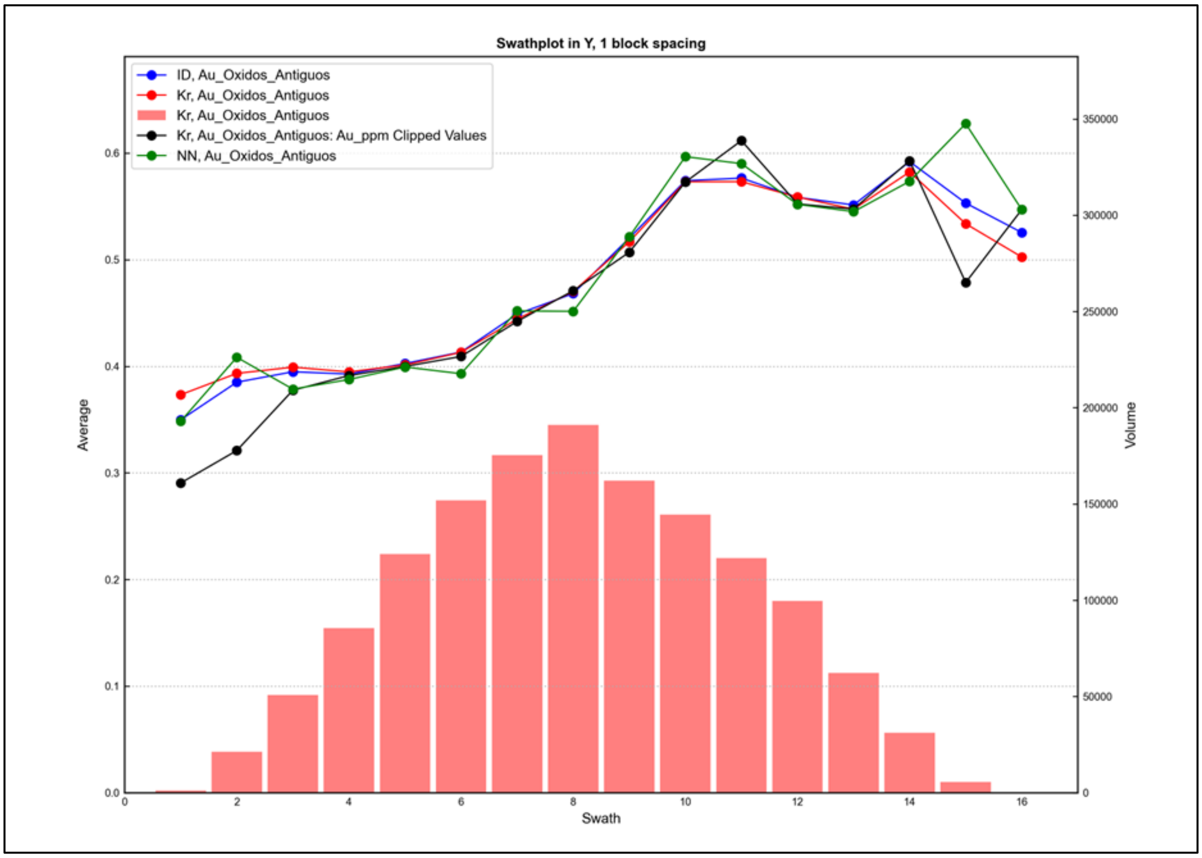

Figure 14-39: Older Oxide Tailings Domain, Swath Plot for Gold, Northing (Red Pennant 2022)

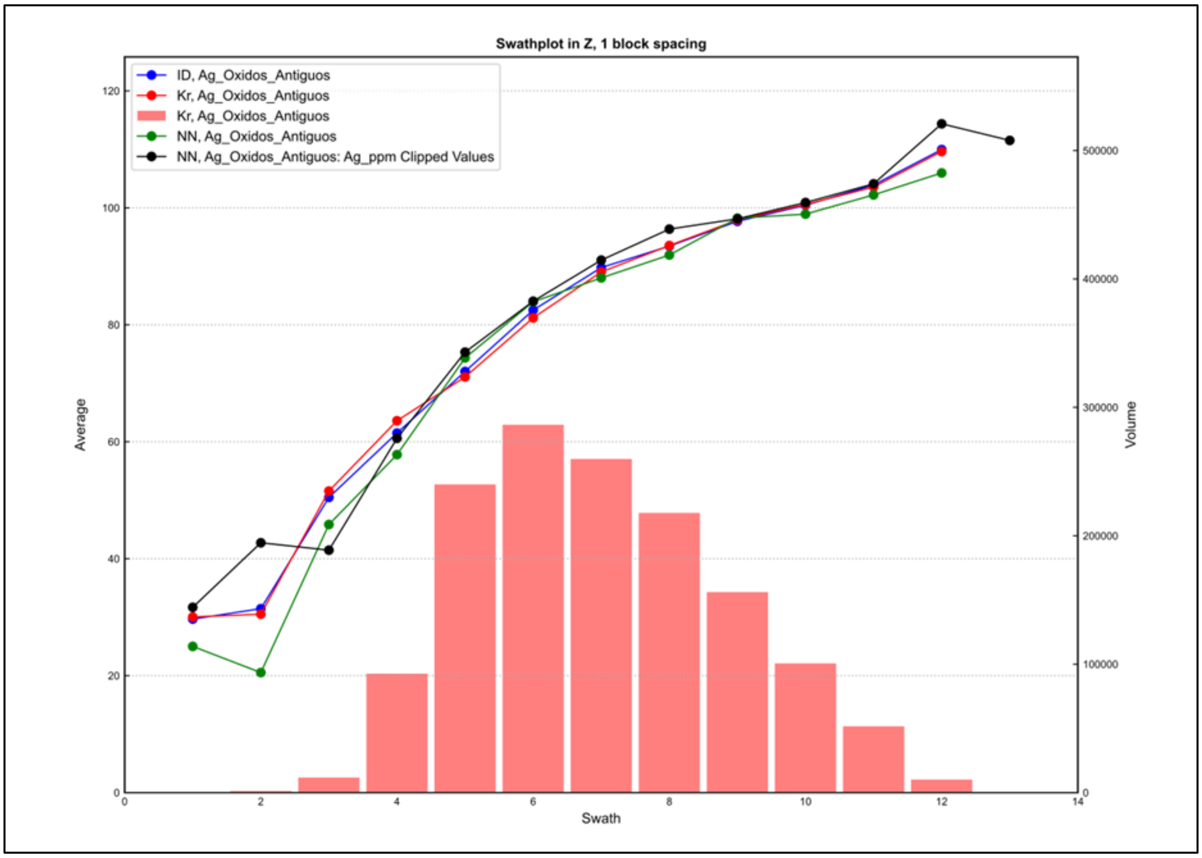

Figure 14-40: Older Oxide Tailings Domain, Swath Plot for Silver, Elevation (Red Pennant 2022)

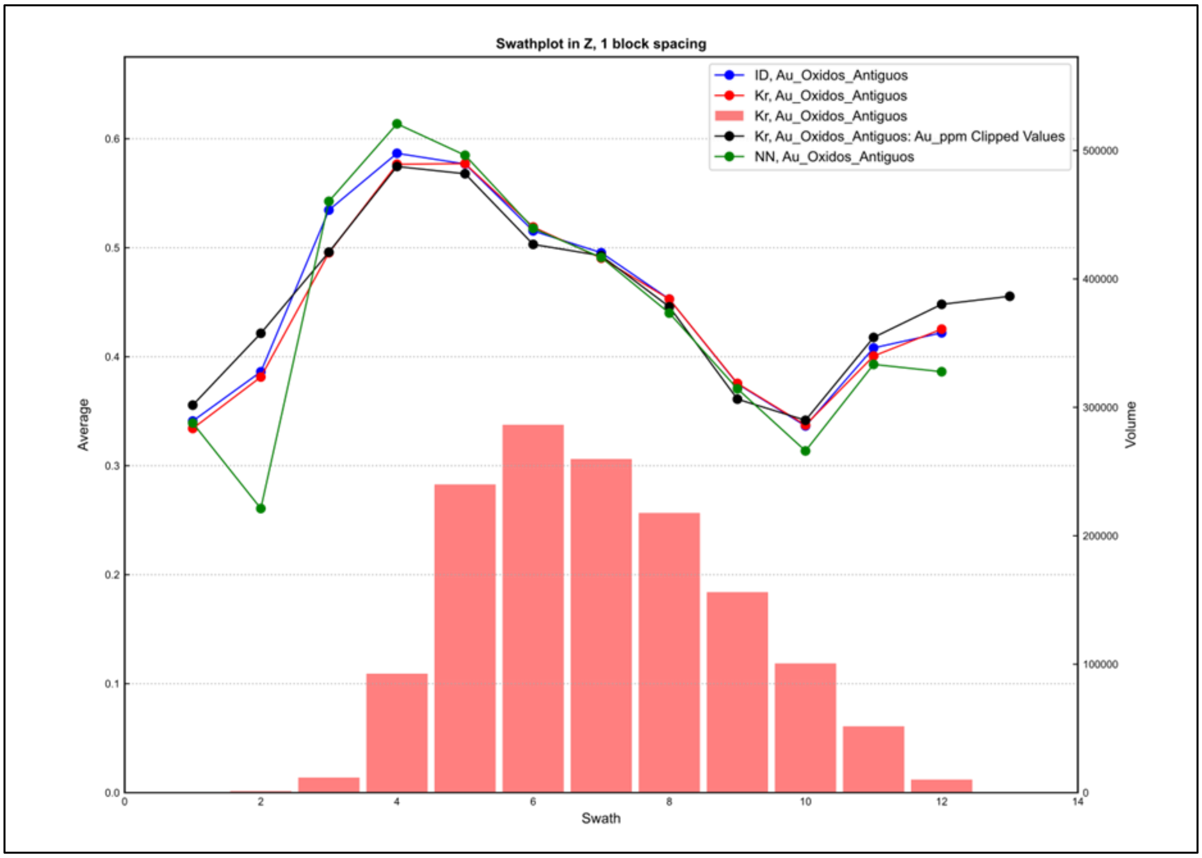

Figure 14-41: Older Oxide Tailings Domain, Swath Plot for Gold, Elevation (Red Pennant 2022)

The swath plot comparisons show reasonable correspondence between block estimates and sampling data. As expected, the OK and ID estimates show fewer extreme values than the NN estimates, particularly near the edges of the models.

14.1.14 Mineral Resource Classification

14.1.14.1 Introduction

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (adopted by the CIM Council on May 10, 2014) for reporting on Mineral Resources are stated below:

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

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

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

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase ‘reasonable prospects for eventual economic extraction’ implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word ‘eventual’ in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage ‘eventual economic extraction’ as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the Project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

The Avino resource has estimates for silver, gold, and copper. Silver, gold, and copper are recovered from the Avino material, and gold and copper are included in the silver equivalent calculation. The bismuth grade has been estimated into the block model for guidance as a potential smelter penalty but is not included in the Mineral Resource statement.

The San Gonzalo resource estimates are only for silver and gold.

Resource classification for Avino and San Gonzalo is based on Kriging variance, Kriging Slope of Regression, and geological considerations mapped to the average distance of estimated blocks from estimating data.

The 2014 CIM Definition Standards of indicated Mineral Resources includes the phrase that “quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit".

Quantitative criteria used for the classification of the Mineral Resources are summarized in Table 14-31 for the vein deposits and the tailings deposit.

In the opinion of the QP, the current information available for estimating in situ density is insufficient to support localized (block) estimates to the same level of detail as the metal grades. However, the current data shows that the wall-rock and vein material of the Avino and San Gonzalo deposits have small differences (density difference of vein to wall-rock less than 2%) and low variability within the veins as measured by the coefficient of variation (less than 0.08, see Table 14-14). The variability of the metal grades shows levels of variability orders of magnitude higher as measured by the coefficient of variation (between 0.4 and 4.4). It would be ideal to have density measurements sufficient for local block estimation. However, the potential error resulting from the use of a global density mean is likely to be less than 2%. No significant density anomalies have been reported during the current phase of production at the Avino Property. Based on the data from the Avino and San Gonzalo Veins, the grade is a much more material risk factor than the density information. The QP considers the restricted amount of density information to be less material and significant than the metal grade variability. The QP used the kriging variance of the silver grade estimates as the main factor for resource classification.

The QP has also noted that, despite the lack of metallurgical bulk sampling, there have been several years of metal production from the mineralized material of all operating levels on the Avino and San Gonzalo Veins using current processing facilities and that there has been no report of unforeseen metallurgical recovery issues. The QP considers that this production history mitigates the lack of a formal bulk sampling program or density data and has allowed Mineral Resources to be defined with sufficient confidence to support detailed mine planning and evaluation of the economic viability of the deposit.

Table 14-31: Criteria for Classification of Underground Mineral Resources

Deposit / Category	Criteria
Avino ET	
Measured:	Kriging Slope of Regression > 0.6 and less than 35 m from sampled development
Indicated:	Kriging Slope of Regression <=0.2 from 35 to 80 m from sampling, contiguous with development and measured
Inferred:	Up to 200 m from sampling data with demonstrated vein continuity
San Gonzalo	
Measured:	Kriging variance <=0.18 and distance to from sampled development < 15 m
Indicated:	Kriging variance <=0.24 and 15 to 30 m from development development samples)
Inferred:	Up to 200 m from at least 3 holes data (with demonstrated vein continuity)
Guadalupe	
Indicated:	less than 30 m from drilling
Inferred:	30 to 150 m from drilling
La Potosina	
Indicated:	less than 30 m from drilling
Inferred:	30 to 160 m from drilling
Tailings Deposit	
Measured:	Kriging Slope of Regression >=0.90 and distance from drill data < 35 m
Indicated:	Kriging Slope of Regression >=0.50 and 35 m to 60 m from drill data
Inferred:	Up to 150 m from sampling data with assumed geometric continuity

Reasonable Prospects for Eventual Economic Extraction are a necessary condition for Mineral Resources. The Avino and Sand Gonzalo underground Mineral Resources are contiguous with or close to existing underground development and meet the cut-off criteria shown in Table 14-32. The tailings Mineral Resources have been constrained by an ultimate pit shell generated using the Lerch-Grossman (LG) algorithm using the parameters shown in Table 14-32. Metal recovery parameters are based on the values obtained in 2012 for heap leach testing on the oxide tailings material.

Table 14-32: Avino Tailings Deposit Ultimate Pit Parameters

Parameter	Value
Overall Pit Slope Angle	25 degrees
Silver Price	$21/tr.oz
Gold Price	$1,800/tr.oz
Silver Recovery	73%
Gold Recovery	78.9%
Mining Excavation Cost	$1.00/t
Treatment Cost	$15.14/t
G&A Cost	$1.51/t

14.1.15 Mineral Resource Tabulation

The following table provides a synopsis of the Mineral Resources reported in this section. Table 14-33 summarizes the Mineral Resources on the Avino Mine area.

Table 14-33: Avino Mine area – Inclusive Mineral Resources (Effective Date: October 16, 2023)

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
ET Avino	MEA	3.88	171	69	0.53	0.57	21.39	8.58	67	48.91
	IND	23.92	146	58	0.53	0.44	112.41	44.59	409	234.08
	M&I	27.80	150	60	0.53	0.46	133.8	53.17	476	283
	INF	17.59	106	37	0.34	0.4	59.76	20.72	191	154.18
San Gonzalo	MEA	0.33	332	244	1.17	0	3.53	2.59	12.42	0
	IND	0.30	293	230	0.84	0	2.85	2.23	8.14	0
	M&I	0.63	313	237	1.01	0	6.38	4.83	20.56	0
	INF	0.25	297	271	0.35	0	2.35	2.14	2.74	0
Guadalupe	MEA	0.00	0	0	0	0	0	0	0	0
	IND	0.40	169	70	0.79	0.37	2.17	0.9	10.24	3.27
	M&I	0.40	169	70	0.79	0.37	2.17	0.9	10.24	3.27
	INF	0.35	159	82	0.62	0.3	1.81	0.93	7	2.3
La Potosina	MEA	0.00	0	0	0	0	0	0	0	0
	IND	0.14	220	186	0.41	0.04	1	0.85	1.85	0.13
	M&I	0.14	220	186	0.41	0.04	1	0.85	1.85	0.13
	INF	0.84	176	149	0.29	0.05	4.79	4.05	7.9	1.01

table continues…

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
Tailings Deposit	MEA	4.25	101	61	0.47	0.12	13.83	8.35	64.84	11.33
	IND	2.44	83	43	0.47	0.12	6.51	3.40	36.67	6.21
	M&I	6.70	94	55	0.47	0.12	20.34	11.75	101.50	17.55
	INF	0.34	97	65	0.36	0.11	1.06	0.70	3.95	0.82
TOTALS	**MEA**	**8.47**	**142.35**	**71.72**	**0.53**	**0.32**	**38.75**	**19.52**	**144.26**	**60.24**
	IND	**27.20**	**142.85**	**59.42**	**0.53**	**0.41**	**124.94**	**51.97**	**465.90**	**243.69**
	M&I	**35.67**	**142.73**	**62.35**	**0.53**	**0.39**	**163.69**	**71.50**	**610.15**	**303.95**
	INF	**19.37**	**112.02**	**45.83**	**0.34**	**0.37**	**69.77**	**28.54**	**212.59**	**158.31**

Notes:

1. Figures may not add to totals shown due to rounding.
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimate is classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves incorporated by reference into NI 43-101 Standards of Disclosure for Mineral Projects.
4. Based on recent mining costs (Section 21.0), Mineral Resources are reported at cut-off grades 60 g/t, 130 g/t, and 50 g/t AgEQ grade for ET, San Gonzalo, and oxide tailings, respectively.
5. AgEQ or silver equivalent ounces are notational, based on the combined value of metals expressed as silver ounces
6. Mineral Resources are disclosed on an inclusive basis, inclusive of material included in the mineral reserve
7. Cut-off grades were calculated using the following consensus metal price assumptions: gold price of US$1,800/oz, silver price of US$21.00/oz, and copper price of US$3.50/lb.
8. Metal recovery is based on operational results and column testing, shown in 5.
9. The silver equivalent was back-calculated using the following formulas:
   a. ET, Guadalupe, La Potosina: AgEQ = Ag (g/t) + 71.43 * Au (g/t) + 113.04 * Cu (%)
   b. San Gonzalo: Ag Eq = Ag (g/t) + 75.39 * Au (g/t)
   c. Oxide Tailings: Ag Eq = Ag (g/t) + 81.53 * Au (g/t)

For information, the exclusive mineral resources (exclusive of material included in the mineral reserve, see item 15) are summarized in Table 14-34.

TETRA TECH

Table 14-34: Avino Mine area – Exclusive Mineral Resources

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
ET Avino	MEA	3.883	171	69	0.53	0.57	21.39	8.58	67	48.91
	IND	23.916	146	58	0.53	0.44	112.41	44.59	409	234.08
	M&I	27.8	150	60	0.53	0.46	133.8	53.17	476	283
	INF	17.591	106	37	0.34	0.4	59.76	20.72	191	154.18
San Gonzalo	MEA	0.331	332	244	1.17	0	3.53	2.59	12.42	0
	IND	0.302	293	230	0.84	0	2.85	2.23	8.14	0
	M&I	0.633	313	237	1.01	0	6.38	4.83	20.56	0
	INF	0.246	297	271	0.35	0	2.35	2.14	2.74	0
Guadalupe	MEA	0	0	0	0	0	0	0	0	0
	IND	0.401	169	70	0.79	0.37	2.17	0.9	10.24	3.27
	M&I	0.401	169	70	0.79	0.37	2.17	0.9	10.24	3.27
	INF	0.354	159	82	0.62	0.3	1.81	0.93	7	2.3

table continues…

Area/Zone	Category	Mass (Mt)	Average Grade				Metal Content			
			AgEQ (g/t)	Ag (g/t)	Au (g/t)	Cu (%)	AgEQ (million tr oz)	Ag (million tr oz)	Au (thousand tr oz)	Cu (million lb)
La Potosina	MEA	0	0	0	0	0	0	0	0	0
	IND	0.142	220	186	0.41	0.04	1	0.85	1.85	0.13
	M&I	0.142	220	186	0.41	0.04	1	0.85	1.85	0.13
	INF	0.844	176	149	0.29	0.05	4.79	4.05	7.9	1.01
Tailings Deposit	MEA	0.00	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	IND	0.00	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	M&I	0.00	0	0	0.00	0.00	0.00	0.00	0.00	0.00
	INF	0.34	97	65	0.36	0.11	1.06	0.70	3.95	0.82
TOTALS	MEA	4.21	91.56	41.04	0.29	0.26	24.92	11.17	79.42	48.91
	IND	24.76	135.40	55.53	0.49	0.40	118.43	48.57	429.23	237.48
	M&I	28.98	124.99	52.10	0.44	0.36	143.35	59.75	508.65	286.40
	INF	19.37	112.02	45.83	0.34	0.37	69.77	28.54	212.59	158.31

14.1.15.1 Cut-offs and Silver Equivalent Calculations

The San Gonzalo and Avino reported Mineral Resources are tabulated based on AgEQ cut-offs (Table 14-35).

Table 14-35: Silver Equivalent-Based Metal Prices and Operational Recovery Parameters

Metal	Price	Unit	Recovery (%)
San Gonzalo Vein System			
Ag	21.00	US$/oz	83
Au	1,800.00	US$/oz	73
ET, Guadalupe, and La Potosina Deposits			
Ag	21.00	US$/oz	90
Au	1800	US$/oz	75
Cu	3.50	US$/lb	89
Avino Tailings			
Ag	21.00	US$/oz	82
Au	1,250.00	US$/oz	78

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.1.15.2 Grade-Tonnage Graphs

Figure 14-42 to Figure 14-46 summarize the grade and tonnage response to changes to the cut-off grade for the five deposits. This provides an indication of the sensitivity of the mineralized material in the deposits to cut-off grade expressed as silver equivalent.

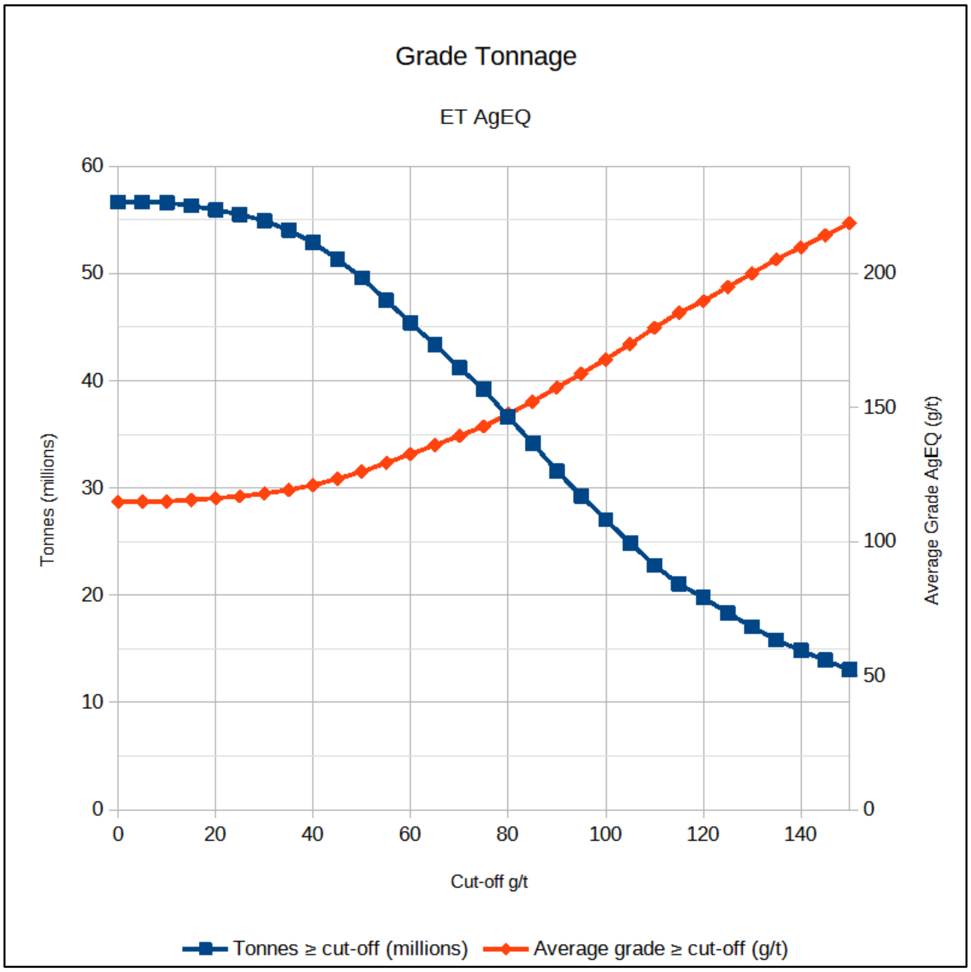

Figure 14-42: Avino (ET) Grade Tonnage Graph

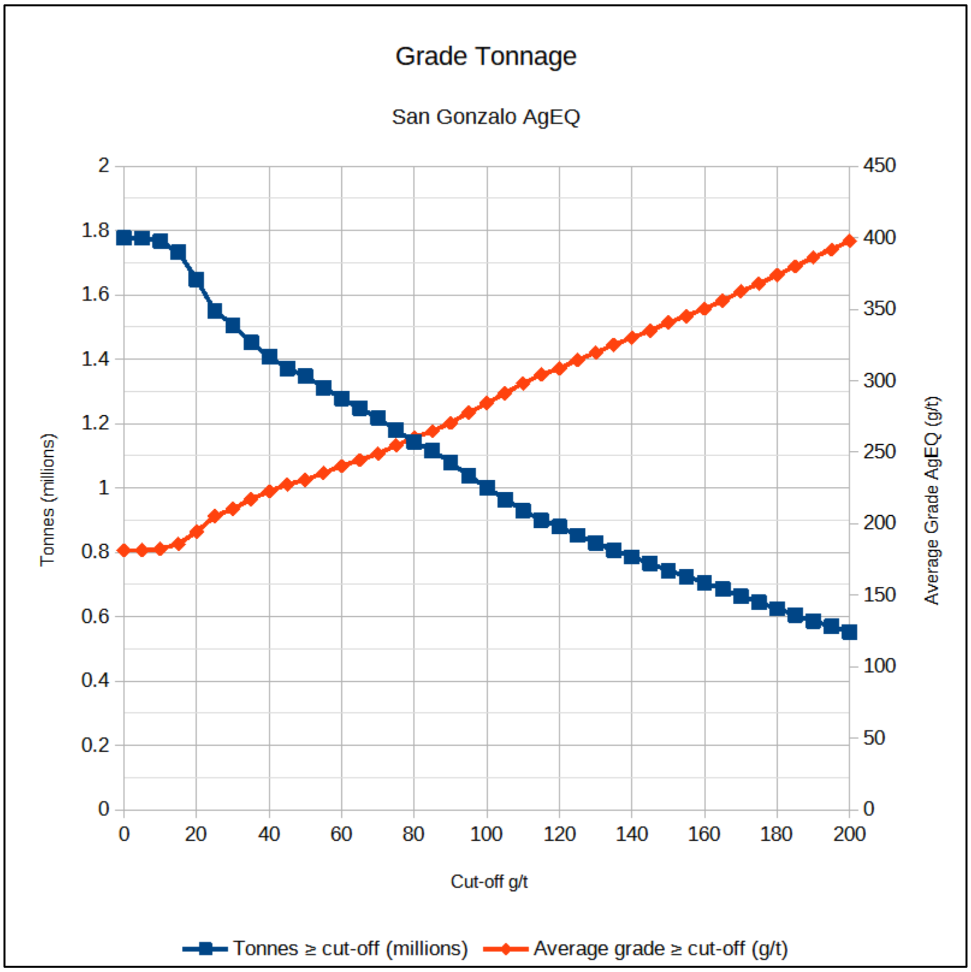

Figure 14-43: San Gonzalo (SG) Grade Tonnage Graph

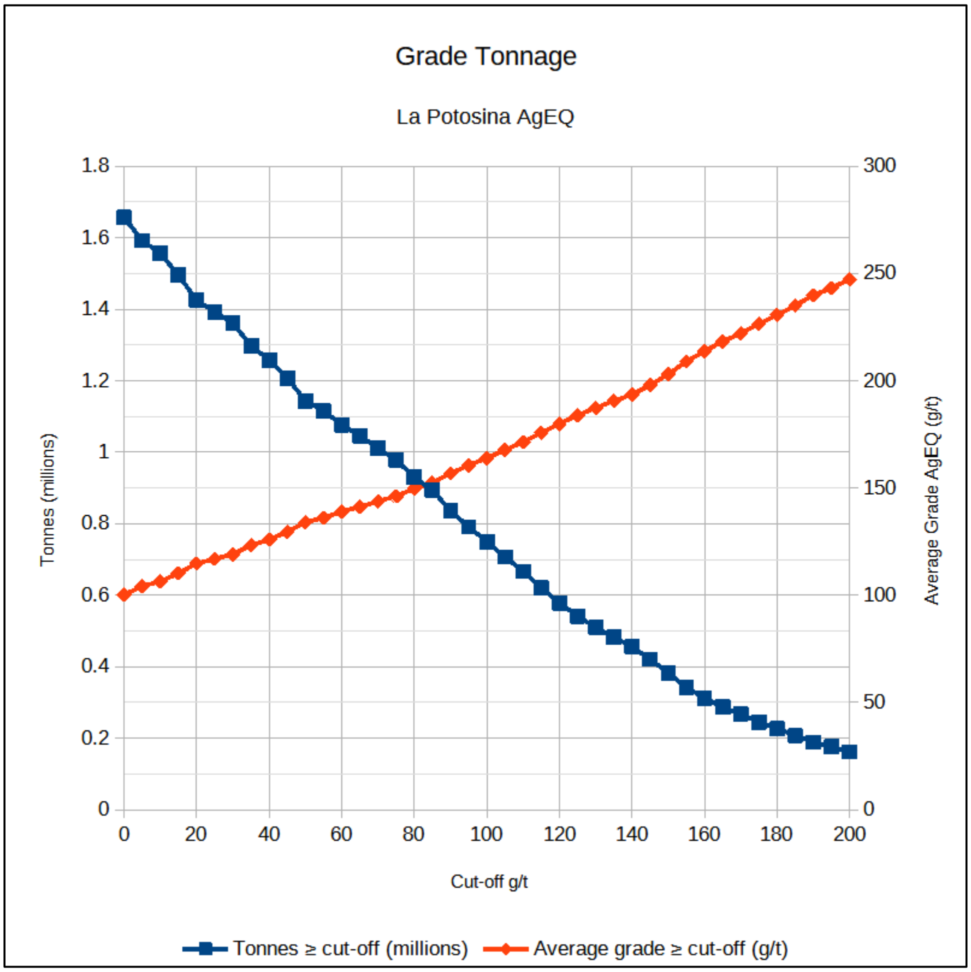

Figure 14-44: Guadalupe Grade Tonnage Graph

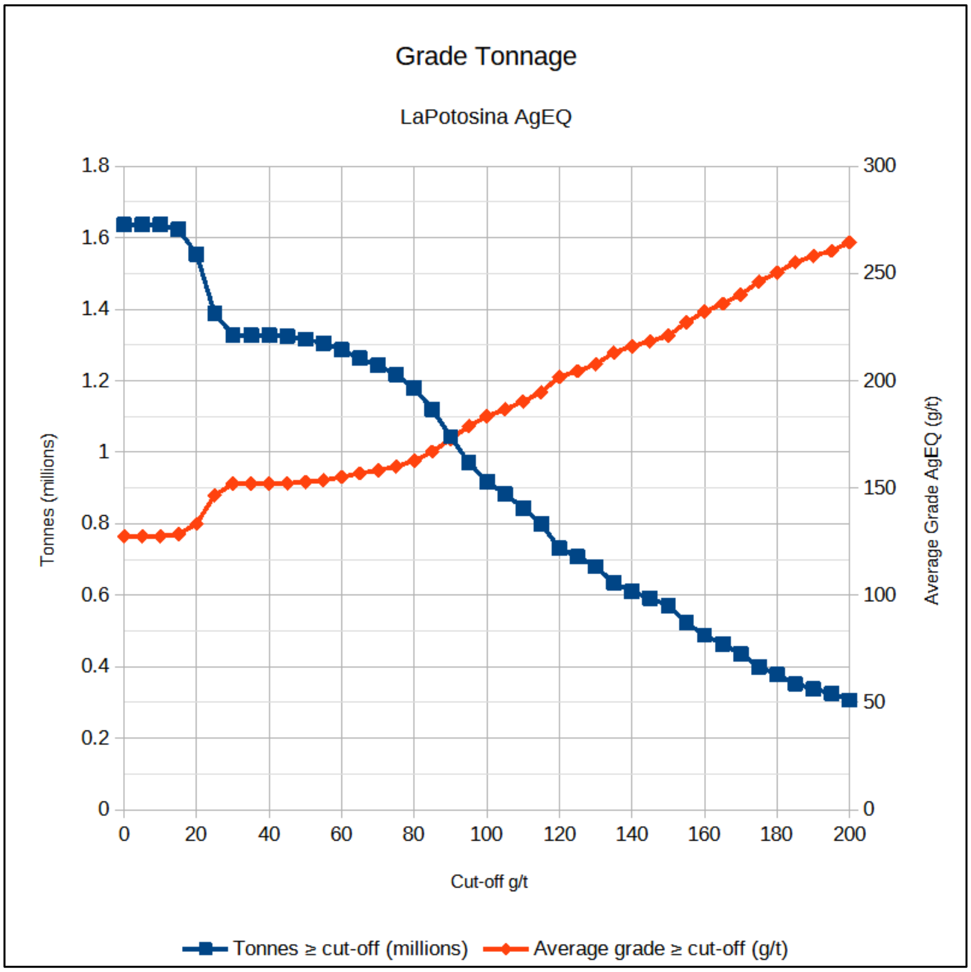

Figure 14-45: La Potosina Grade Tonnage Graph

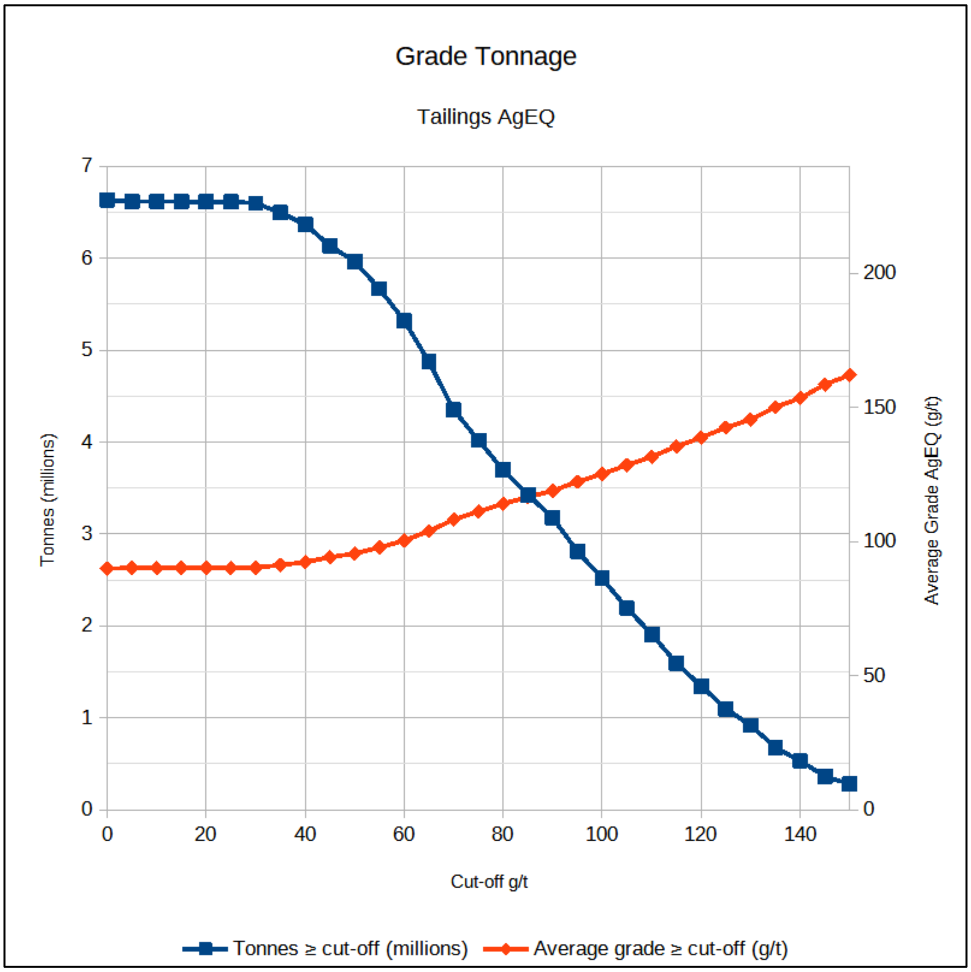

Figure 14-46: Tailings Material Grade Tonnage Graph

14.2 La Preciosa Area

14.2.1 Introduction

The La Preciosa veins are narrow tabular units. A two-dimensional approach was first used for estimation on the La Preciosa Deposit by Marcelo Zangrandi (AMBA 2020), as an internal resource estimation exercise for Coeur. The QP believes that the two-dimensional approach is appropriate to this deposit.

When undertaking resource estimation on any deposit with narrow vein- or layer-like geometry, the variable of ultimate interest (e.g., grade) may not be the best variable for direct kriging (Bertoli et al. 2003). This is because the grade of mineralized intercepts is clearly defined, but on varying length or thickness supports. Grade can also be defined as the ratio of two other variables (thickness and accumulation – the product of grade by thickness potentially weighted by bulk density), which are amenable to direct kriging (Chiles and Delfiner 1999). The variables of economic interest, i.e., those upon which economic decisions and optimizations will be made, are actually the projected horizontal or true geometric thickness (tonnage) and the accumulation (metal content) and not the grade. The grade over the thickness of the mineralized unit is easily obtained as the quotient of accumulation and the thickness. For bodies that have thickness less than or equal to the equipment size or selective mining unit cross-section, the grade of short intervals within the unit is of little importance, as high grades cannot be practically selectively mined. Two-dimensional estimation, using thickness and metal accumulations, has been successfully applied for many decades on the South African Witwatersrand Gold Mines.

Two-dimensional metal accumulations were estimated by the QP for the silver and gold grades of the veins, and the projected true thickness was estimated based on the attitude of the veins and intercept length. OK was applied as the primary method, with inverse distance estimates generated as a check. The QP has modified the earlier Zangrandi approach to use projected true thickness for each vein (rather than a generalized horizontal thickness), independently modelled variograms, search parameters and block model parameters, and using a dynamic workflow in Leapfrog Geo and Edge software.

14.2.2 Topographic Information

The topography provides an important limit to the extent of outcropping veins. In 2011, PAS commissioned PhotoSat™ to estimate the site topography using high-definition satellite photos (colour ortho photos with 50 cm resolution acquired on October 14, 2011) from which a digital elevation model (DEM) was built, and the DEM then reduced to a topographic map with 1, 5, 10, and 50 m contour intervals. PhotoSat reported an accuracy of ±30 cm on a 1 m grid. An unknown number of control points were established on the ground to register the satellite image to the project datum (datum used is WGS84).

14.2.3 Geological Models

During 2013 and 2014, Coeur reviewed the cores, interpreted the geological model, and revised the lithology classifications for an NI 43-101 Feasibility Study (Neff et al. 2014). The MRE used for this historic feasibility study was informed by an open pit mining method and the estimation method was Multiple Indicator Kriging applied independently to silver and gold.

In 2020, geological modelling of the veins was conducted by Hugo Zúñiga, Coeur staff geologist, using Leapfrog Geo software. The interpretation defined numerous vein bodies. A two-dimensional approach to mineral resource estimation was pioneered on the La Preciosa deposit during 2020, by Zangrandi (2020) as an unpublished internal mineral resource estimation exercise for Coeur. The estimation was carried out using Vulcan software.

In 2021, the QP remodelled the veins from the previous vein interpretations, added channel sampling sections, and estimated the mineral resource using a two-dimensional OK approach utilizing Leapfrog Geo 6.0.3 software and leapfrog EDGE.

During 2021, the domains and drillhole assay data were modified to include channel sampling data, depleted by historic mining voids and verified in Leapfrog Geo by the QP. Independent grade estimates were made using Leapfrog Edge by the QP.

The estimation workflow is as follows:

Vein composites were created for each drill hole or channel sample intercept of each mineralized vein. The composite length is the aggregated sample lengths comprising the samples of a single vein intercept, corrected to true width using the average inclination of the vein. The accumulations for Ag and Au are the aggregated products of grade and true thickness over the full vein thickness. The three variables, thickness (m), Ag accumulation (m.g/t), and Au accumulation (m.g/t) were estimated for 15 m x 15 m blocks over the thickness of the veins. Estimation was carried out using OK and inverse distance to the second power weighting.

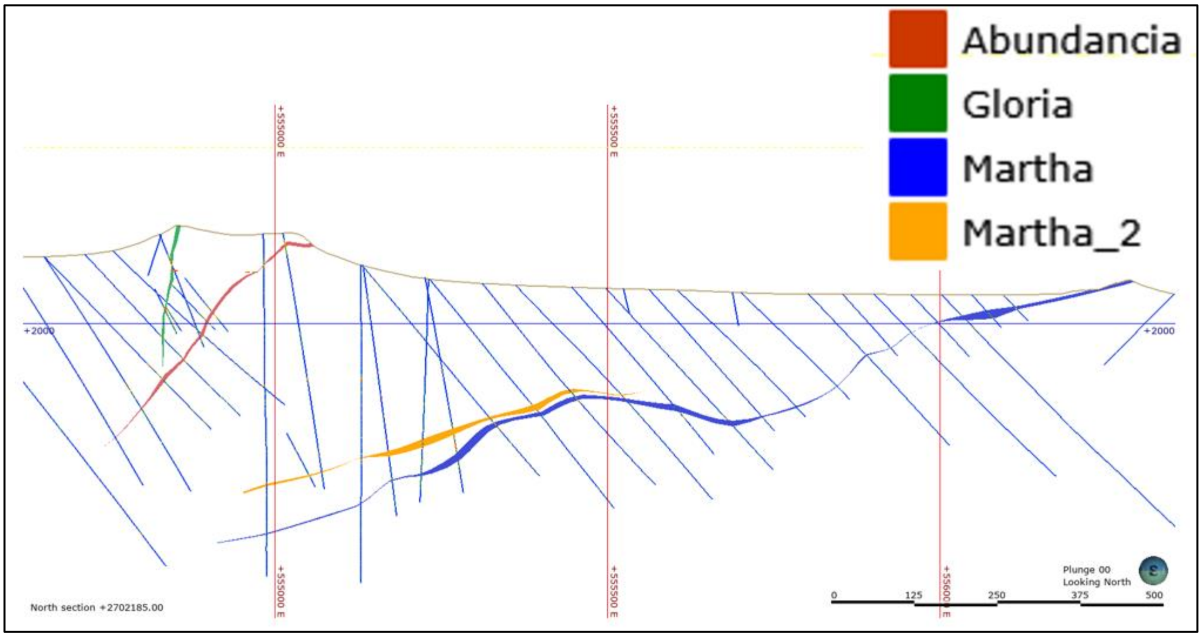

Figure 14-47: West-East Section Across La Preciosa Showing Significant Veins (Northing Y+2702185)

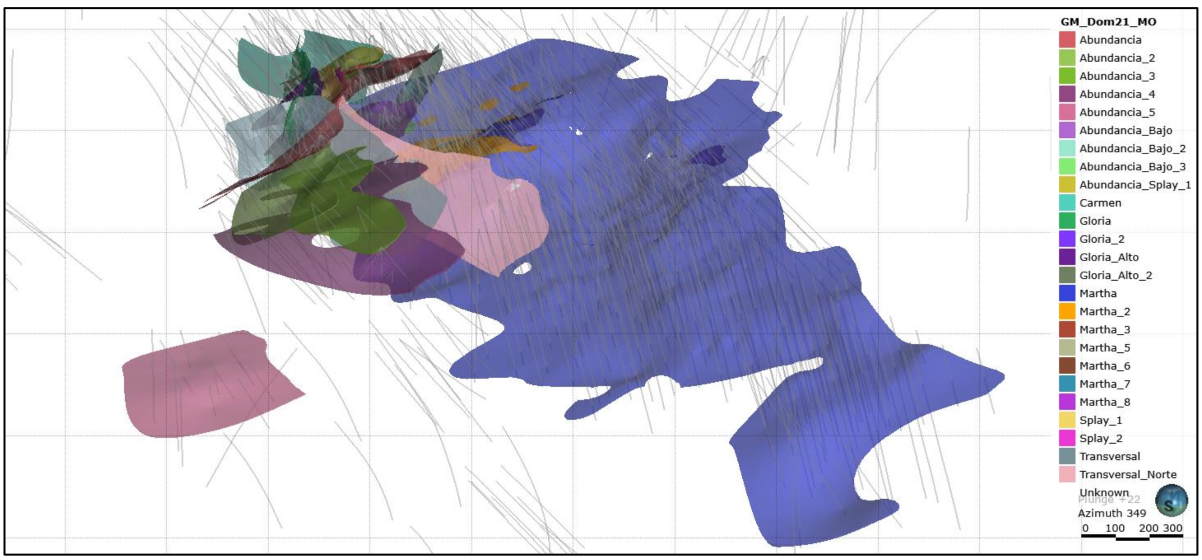

Figure 14-48: Orthographic View of the Veins (Source: Red Pennant, 2021)

The resource estimation domains for the veins are shown in Figure 14-48.

14.2.4 Exploratory Data Analysis

The data for estimation consists of sampling and logging from 1,004 drillholes.

Silver and gold sample grade and sample length statistics for the veins constituting the Resource Estimation Domains are listed in Table 14-36.

Table 14-36: Length-Weighted Silver and Gold Sample Grade and Sample Length Statistics for Vein Domains

Vein Grade weighting: Statistics Length-weighted																										
Name																										
Ag (g/t)	Total	Abundancia	Abundancia_2	Abundancia_3	Abundancia_4	Abundancia_5	Abundancia_Bajo	Abundancia_Bajo_2	Abundancia_Bajo_3	Abundancia_Splay_1	Carmen	Gloria	Gloria_2	Gloria_Alto	Gloria_Alto_2	Martha	Martha_2	Martha_3	Martha_5	Martha_6	Martha_7	Martha_8	Splay_1	Splay_2	Transversal	Transversal_Norte
Count	133406	1362	44	76	68	32	152	96	64	100	153	886	76	54	16	4919	464	76	16	249	67	46	97	37	169	164
Length	297462.8	1245.9	35.0	59.7	58.8	22.7	99.7	75.2	49.5	77.2	106.3	799.5	79.8	56.6	15.3	4107.9	374.9	50.3	14.5	170.9	57.6	54.2	100.9	26.3	139.1	113.1
Mean	7.1	186.5	64.0	113.2	72.8	68.2	131.3	185.5	201.1	115.9	130.1	215.3	117.9	68.8	35.6	118.7	233.9	93.9	123.6	121.7	158.0	69.3	137.1	137.7	121.3	71.8
SD	48.1	212.4	104.6	136.1	180.0	67.7	162.9	417.0	299.6	112.1	212.2	207.3	127.2	71.1	39.3	208.9	347.2	98.4	243.9	175.6	271.0	56.1	113.5	137.4	148.5	158.7
Coefficient of variation	6.8	1.1	1.6	1.2	2.5	1.0	1.2	2.2	1.5	1.0	1.6	1.0	1.1	1.0	1.1	1.8	1.5	1.0	2.0	1.4	1.7	0.8	0.8	1.0	1.2	2.2
Variance	2315.0	45105.0	10935.8	18531.8	32397.9	4585.5	26527.7	173895.0	89787.9	12569.6	45048.5	42966.0	16192.0	5052.2	1544.6	43651.9	120523.1	9691.2	59509.1	30823.6	73431.8	3143.7	12879.3	18875.0	22044.4	25200.5
Minimum	0.0	0.0	9.1	0.0	0.0	0.8	7.9	4.5	0.0	0.0	0.0	0.0	15.9	0.0	0.0	0.0	0.0	13.2	9.1	1.2	4.3	13.5	15.1	11.4	0.0	0.0
Median	0.0	136.9	27.2	66.8	16.5	53.0	69.8	108.0	103.8	77.2	79.2	160.0	78.0	51.0	18.2	58.0	127.0	60.2	57.7	65.6	88.0	55.4	117.0	104.0	78.2	37.0
Maximum	5897.4	5897.4	614.0	670.0	968.0	319.0	708.0	3761.7	1361.6	722.0	1700.0	1805.0	850.0	373.0	133.0	4036.6	2818.0	666.0	1030.0	1828.3	1770.0	194.0	619.0	743.0	1314.5	1480.0
Au (g/t)																										
Count	133406	1362	44	76	68	32	152	96	64	100	153	886	76	54	16	4919	464	76	16	249	67	46	97	37	169	164
Length	297462.768	1245.852	35.030	59.721	58.841	22.700	99.710	75.230	49.525	77.225	106.349	799.473	79.804	56.581	15.291	4107.939	374.909	50.250	14.500	170.920	57.600	54.190	100.858	26.310	139.143	113.054
Mean	0.017	0.366	0.076	0.152	0.056	0.282	0.155	0.111	0.266	0.289	0.166	0.432	0.457	0.091	0.080	0.237	0.506	0.157	0.447	0.261	0.207	0.079	0.244	0.157	0.144	0.108
SD	0.164	0.754	0.086	0.216	0.143	0.539	0.225	0.145	0.278	0.449	0.187	0.951	2.016	0.104	0.133	0.535	0.796	0.186	0.307	0.171	0.169	0.050	0.447	0.141	0.149	0.140
Coefficient of variation	9.904	2.060	1.137	1.421	2.535	1.911	1.449	1.305	1.045	1.554	1.123	2.201	4.410	1.136	1.670	2.260	1.573	1.178	0.685	0.656	0.815	0.632	1.831	0.898	1.033	1.297
Variance	0.027	0.569	0.007	0.047	0.020	0.290	0.050	0.021	0.077	0.202	0.035	0.905	4.064	0.011	0.018	0.287	0.634	0.034	0.094	0.029	0.029	0.002	0.200	0.020	0.022	0.020
Minimum	0.000	0.000	0.006	0.001	0.001	0.007	0.003	0.005	0.000	0.001	0.000	0.000	0.021	0.000	0.001	0.000	0.001	0.005	0.073	0.007	0.010	0.010	0.006	0.018	0.001	0.000

table continues…

Vein Grade Statistics weighting: Length-weighted																										
Name																										
Ag (g/t)	**Total**	**Abundancia**	**Abundancia_2**	**Abundancia_3**	**Abundancia_4**	**Abundancia_5**	**Abundancia_Bajo**	**Abundancia_Bajo_2**	**Abundancia_Bajo_3**	**Abundancia_Splay_1**	**Carmen**	**Gloria**	**Gloria_2**	**Gloria_Alto**	**Gloria_Alto_2**	**Martha**	**Martha_2**	**Martha_3**	**Martha_5**	**Martha_6**	**Martha_7**	**Martha_8**	**Splay_1**	**Splay_2**	**Transversal**	**Transversal_Norte**
Median	0.001	0.200	0.047	0.075	0.010	0.096	0.084	0.072	0.170	0.135	0.115	0.227	0.137	0.061	0.011	0.106	0.279	0.089	0.359	0.226	0.151	0.065	0.142	0.128	0.097	0.060
Maximum	70.900	16.850	0.429	1.195	0.852	2.387	2.160	1.610	1.199	3.720	1.030	23.000	18.000	0.479	0.409	33.737	7.671	1.175	1.187	0.934	0.958	0.235	3.650	0.907	0.840	0.942
Sample length (m)																										
Count	133406	1362	44	76	68	32	152	96	64	100	153	886	76	54	16	4919	464	76	16	249	67	46	97	37	169	164
Mean	2.23	0.91	0.80	0.79	0.87	0.71	0.66	0.78	0.77	0.77	0.70	0.90	1.05	1.05	0.96	0.84	0.81	0.66	0.91	0.69	0.86	1.18	1.04	0.71	0.82	0.69
SD	12.59	0.48	0.52	0.44	0.54	0.31	0.46	0.35	0.63	0.53	0.45	0.49	0.48	0.50	0.55	0.52	0.42	0.34	0.32	0.35	0.42	0.49	0.45	0.41	0.41	0.50
Coefficient of variation	5.65	0.53	0.66	0.56	0.62	0.43	0.71	0.45	0.82	0.69	0.65	0.55	0.46	0.48	0.58	0.63	0.52	0.52	0.36	0.51	0.49	0.42	0.43	0.58	0.50	0.73
Variance	158.63	0.23	0.28	0.19	0.29	0.09	0.21	0.12	0.40	0.28	0.20	0.24	0.23	0.25	0.30	0.27	0.17	0.12	0.10	0.12	0.18	0.24	0.20	0.17	0.17	0.25
Minimum	0.00	0.00	0.12	0.08	0.00	0.30	0.14	0.20	0.17	0.05	0.17	0.03	0.35	0.17	0.20	0.03	0.10	0.20	0.25	0.17	0.20	0.50	0.04	0.20	0.15	0.02
Median	0.80	0.90	0.64	0.75	0.80	0.65	0.50	0.85	0.60	0.65	0.60	0.85	1.00	1.00	0.93	0.80	0.85	0.63	0.80	0.70	0.85	1.05	1.00	0.60	0.75	0.52
Maximum	577.29	6.13	2.00	2.00	2.10	1.50	2.00	1.65	4.45	4.37	2.00	6.87	2.00	2.05	2.00	9.00	3.35	1.40	1.65	2.00	3.00	3.10	2.30	2.00	2.00	2.95

Histograms (logarithmic) of the silver sample grade distributions of Martha, Martha 2, Abundancia, and Gloria Veins are shown in Figure 14-49 to Figure 14-52, inclusive.

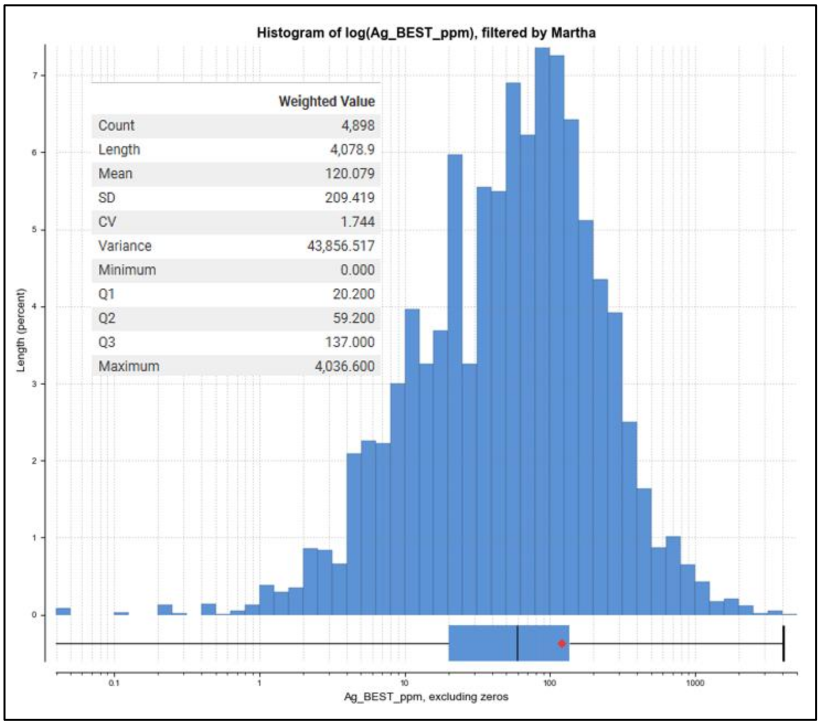

Figure 14-49: Log Histogram of Sample Silver Grades for Martha Vein

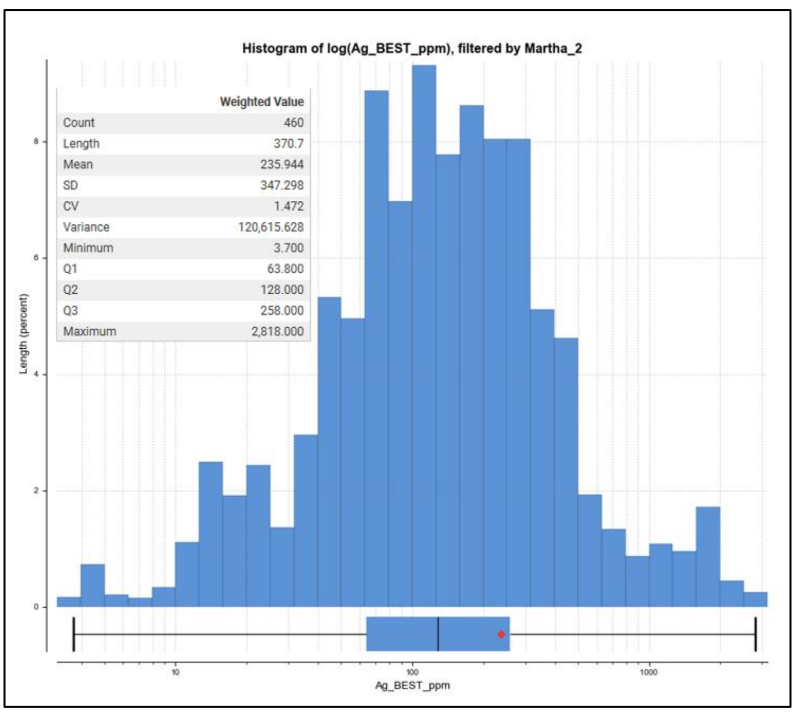

Figure 14-50: Log Histogram of Sample Silver Grades for Martha 2 Vein

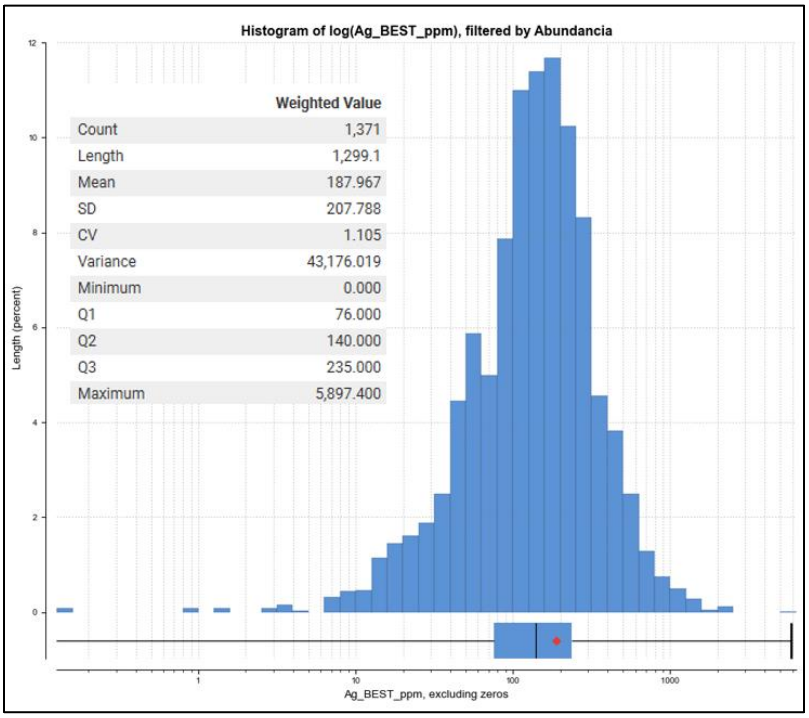

Figure 14-51: Log Histogram of Sample Silver Grades for Abundancia Vein

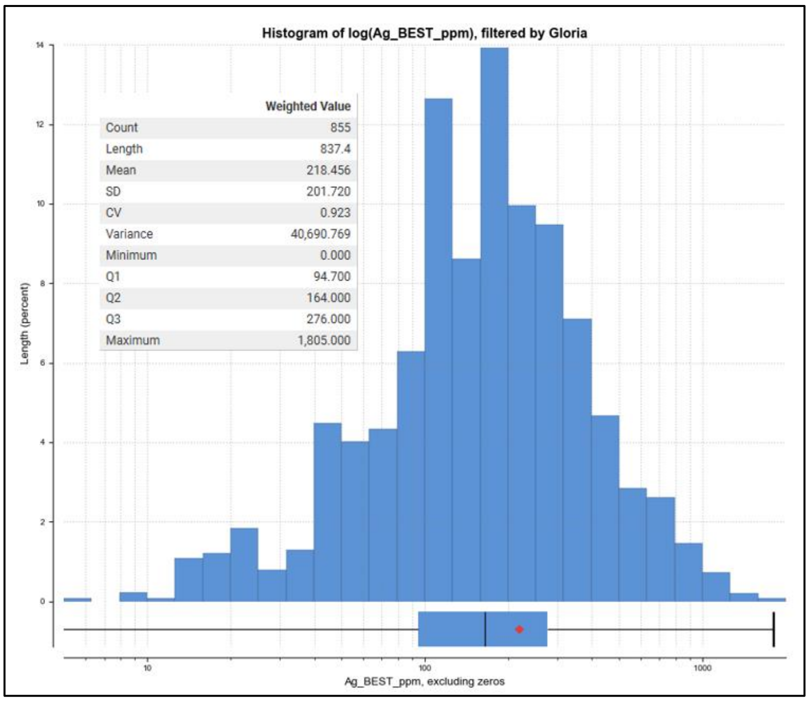

Figure 14-52: Log Histogram of Sample Silver Grades for Gloria Vein

Histograms (logarithmic) of the silver sample grade distributions of Martha, Martha 2, Abundancia, and Gloria Veins are shown in Figure 14-53 to Figure 14-56, inclusive.

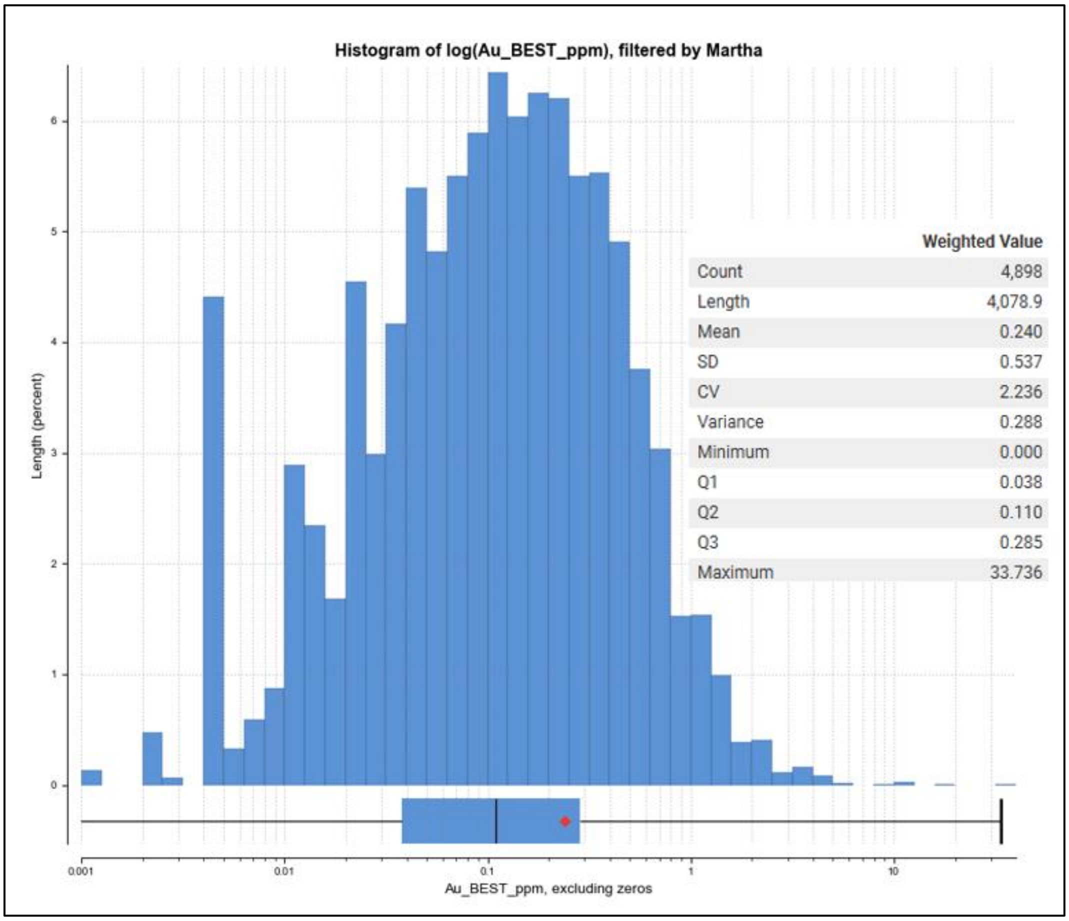

Figure 14-53: Log Histogram of Sample Gold Grades for Martha Vein

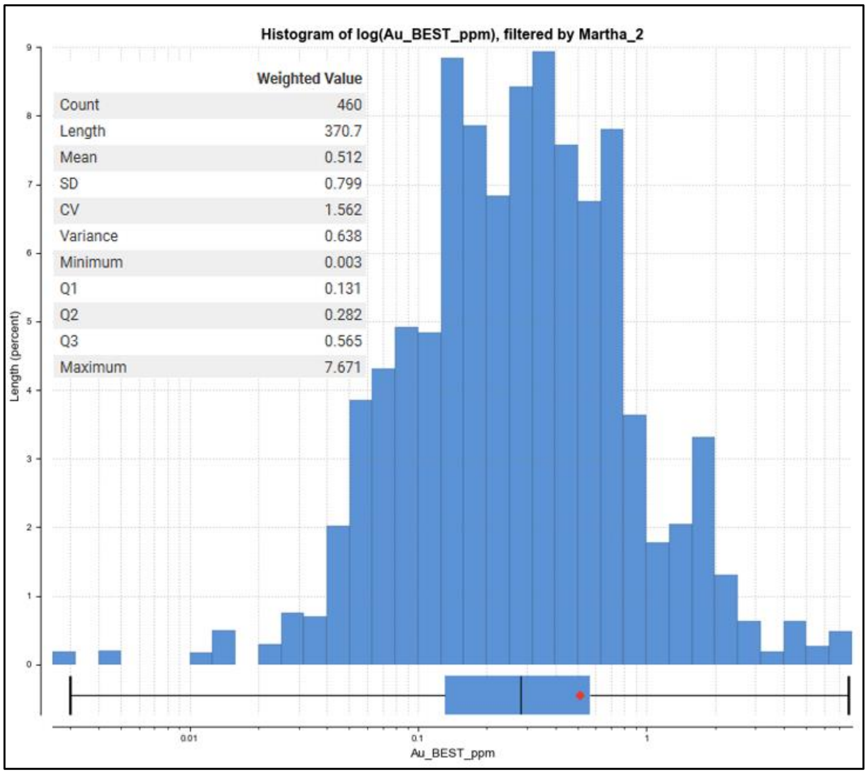

Figure 14-54: Log Histogram of Sample Gold Grades for Martha 2 Vein

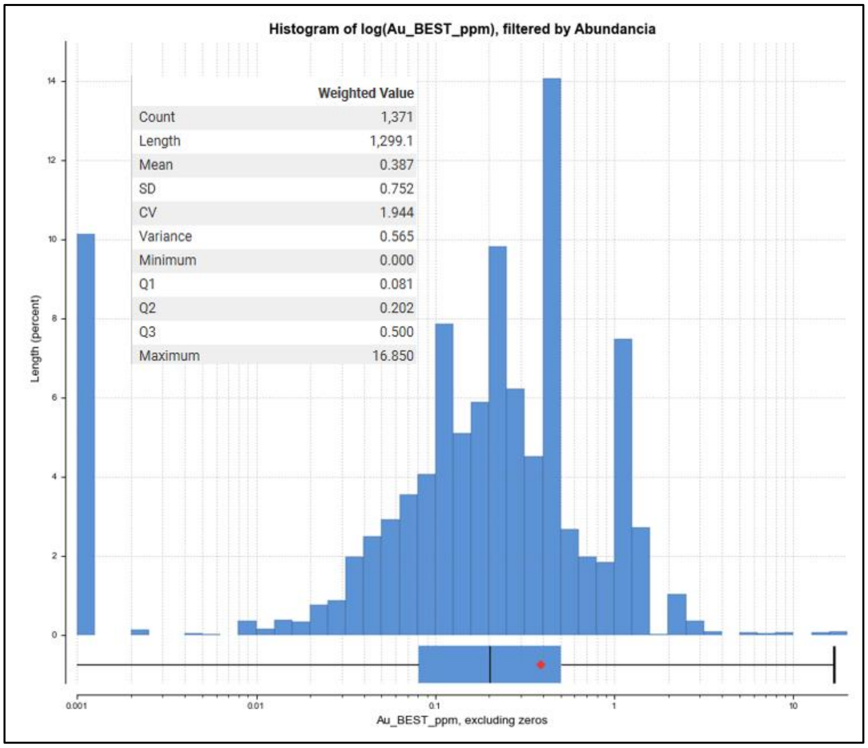

Figure 14-55: Log Histogram of Sample Gold Grades for Abundancia Vein

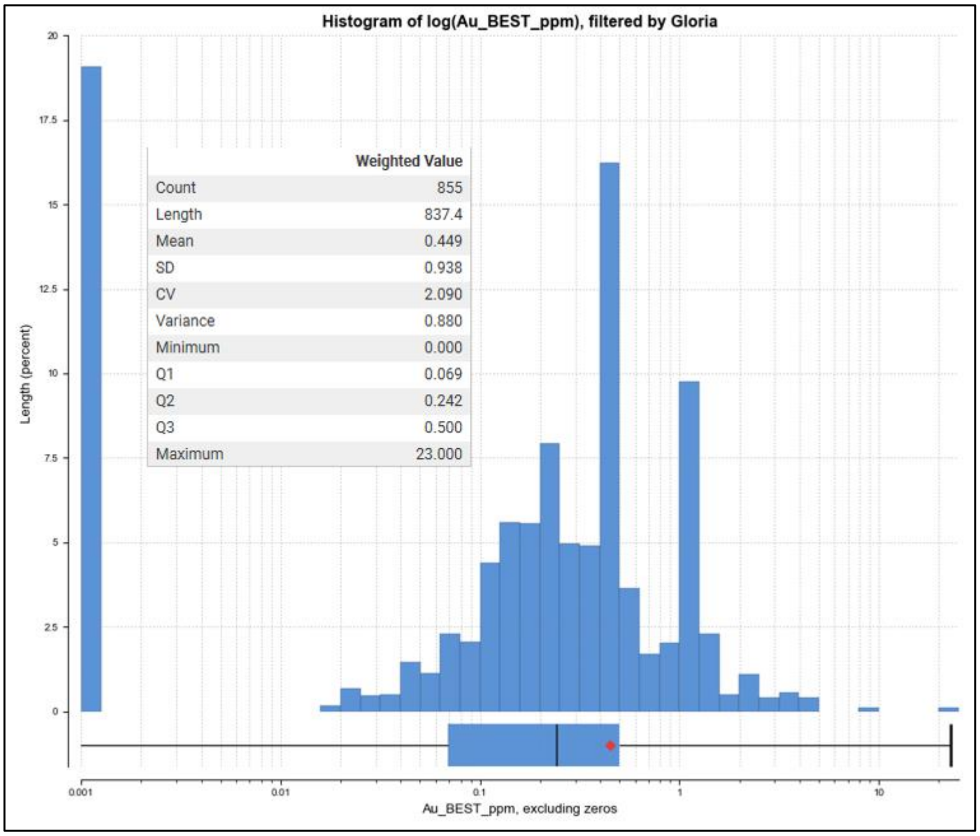

Figure 14-56: Log Histogram of Sample Gold Grades for Gloria Vein

The gold and silver grade distributions all show similar approximations to lognormality.

The gold grades are positively and significantly correlated with the silver grades (e.g., for Martha, Martha 2, Abundancia, and Gloria Veins, see Figure 14-57).

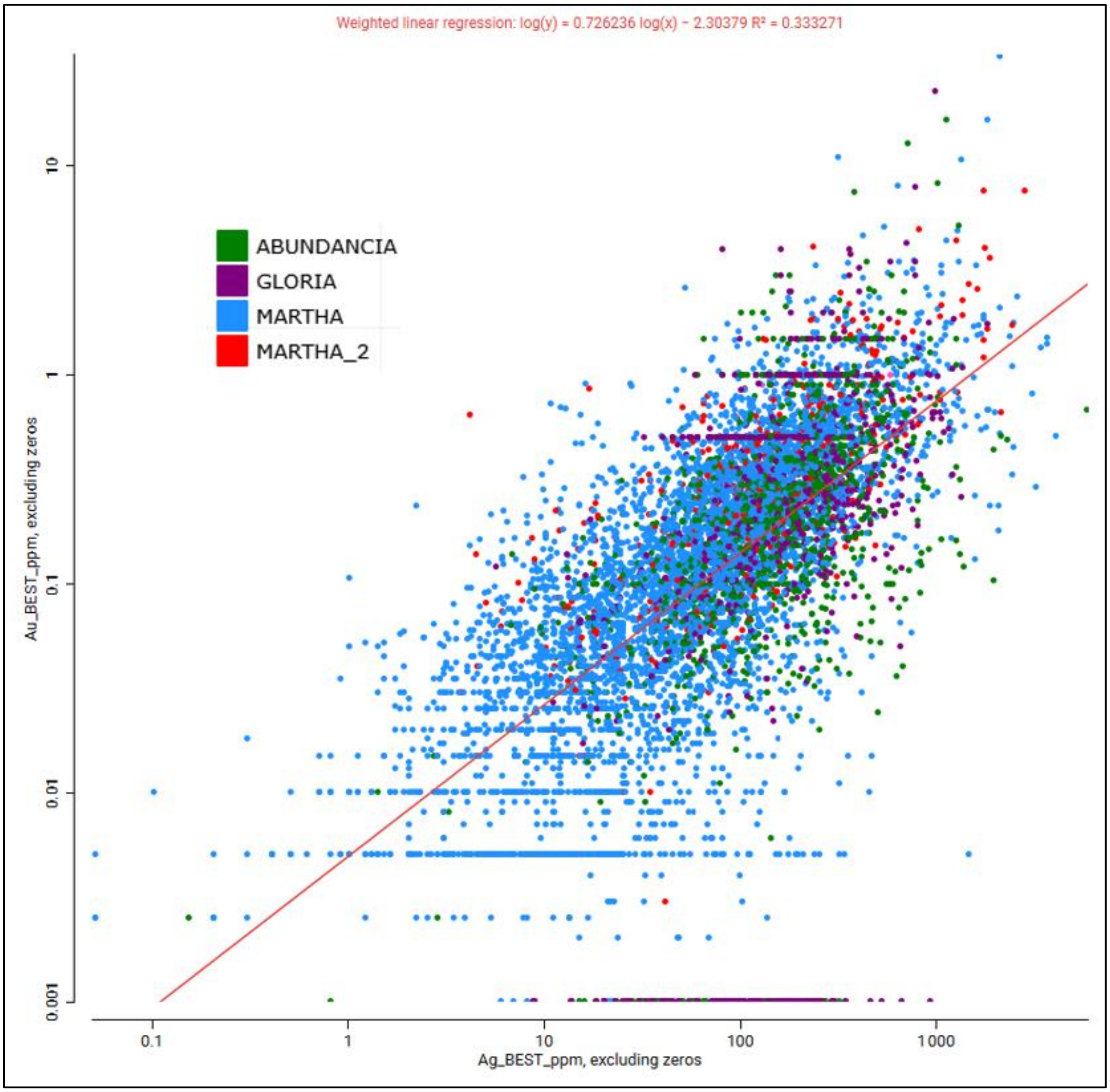

Figure 14-57: Scatterplot (Logarithmic) of Gold vs. Silver for the Four Main Veins at La Preciosa

Channel sampling was carried out by Coeur in drifts on the Abundancia and Gloria Veins. 426 samples (482.2 m) were captured on the Abundancia Vein and 336 samples (380.5 m) on the Gloria Vein. The property has been drilled intensively in the vicinity of the underground development, allowing the two types of sampling data to be compared. The channel samples were statistically compared with diamond drill samples where both types were present within 10 m of each other within the veins. The samples were composited to 1 m lengths, and nearest neighbours (Gloria: 45 pairs, Abundancia: 35 pairs) were compared by means of scatterplots and Q-Q plots to assess whether it was reasonable or not to consider them as a single population. The channel samples and closest drill samples within 10 m proximity are shown in Figure 14-48.

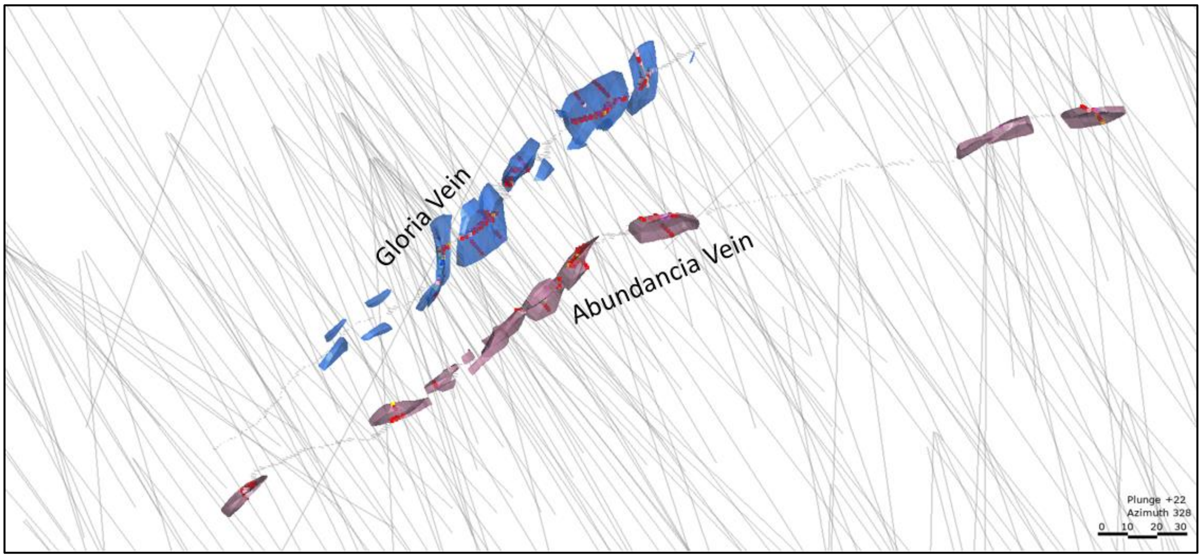

Figure 14-58: Oblique View of Proximity Volumes for Channel Samples and Drill Samples in Gloria and Abundancia Veins

Despite the different methods of sampling, the Q-Q plots (see Figure 14-59) approached the ideal 45° line, thus indicating that the populations of channel and drill samples for the Abundancia and Gloria veins are similar. Consequently, the QP decided to use the limited channel sampling and drill sampling in the mineral resource estimation process.

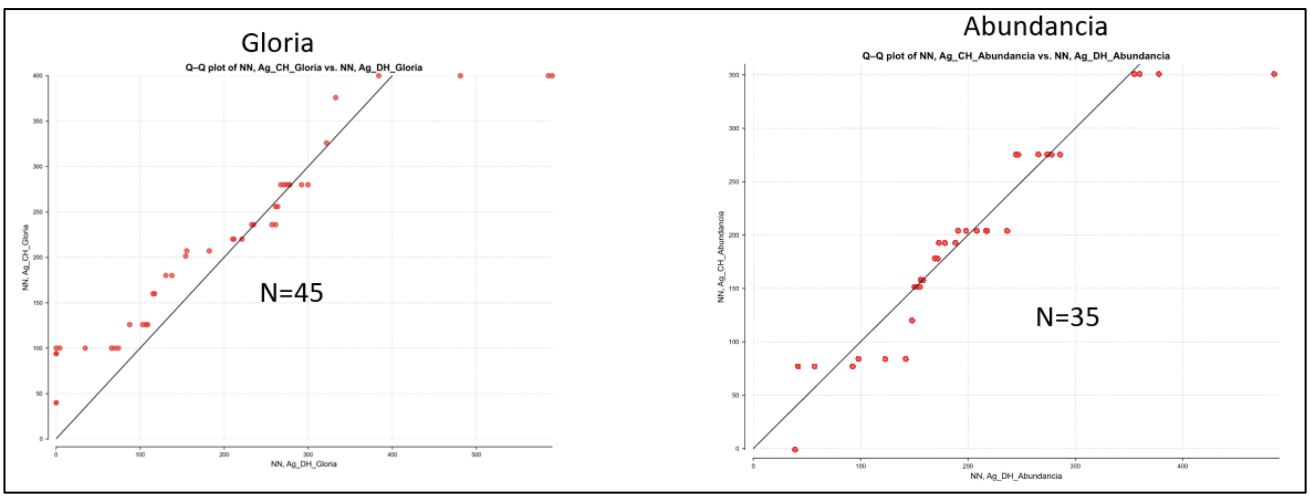

Figure 14-59: Q-Q Plots Illustrating the Similarity Between Channel and Drill Samples Within 10 m Proximity.

14.2.5 Estimation Process

Following the reasoning described in Section 14.2.1, and due to the variable length sample support, silver (and gold) accumulation was selected to estimate the silver and gold grades for the 23 mineralized veins.

Sample accumulations were composited across the full width of the vein. The accumulated products of metal grade and sample thickness (true width) and accumulated true thickness were stored for each intercept. The intercept length and the length weighted accumulations average grades and thickness across the vein for silver and gold are summarized by estimation domain in Table 14-37.

Table 14-37: Vein Composite Statistical Summary

Variable	Vein	Count	Total Length (m)	Mean	SD	Coefficient of Variation	Variance	Minimum	Median	Maximum
Ag acc (m.g/t)	Abundancia	428	1245.9	667.53	696.64	1.04	485309.79	0.00	430.16	4536.17
Ag acc (m.g/t)	Abundancia_2	23	35.0	117.35	168.82	1.44	28501.47	2.31	69.27	645.84
Ag acc (m.g/t)	Abundancia_3	29	59.7	237.25	211.13	0.89	44575.85	0.01	167.21	914.45
Ag acc (m.g/t)	Abundancia_4	34	58.8	149.24	249.00	1.67	61998.77	0.00	29.00	709.26
Ag acc (m.g/t)	Abundancia_5	12	22.7	151.26	143.31	0.95	20537.77	6.96	101.60	386.94
Ag acc (m.g/t)	Abundancia_Bajo	36	99.7	407.05	328.03	0.81	107601.54	4.08	253.07	1254.77
Ag acc (m.g/t)	Abundancia_Bajo_2	30	75.2	525.21	397.51	0.76	158010.51	1.93	500.60	3109.74
Ag acc (m.g/t)	Abundancia_Bajo_3	24	49.5	489.47	504.36	1.03	254382.41	0.00	342.93	1678.65
Ag acc (m.g/t)	Abundancia_Splay_1	42	77.2	344.21	421.03	1.22	177264.12	0.00	167.00	1454.06
Ag acc (m.g/t)	Carmen	12	106.3	157.59	128.75	0.82	16576.77	0.00	128.59	323.04
Ag acc (m.g/t)	Gloria	280	799.5	891.92	799.28	0.90	638843.00	0.00	639.07	4350.09
Ag acc (m.g/t)	Gloria_2	36	79.8	246.78	160.55	0.65	25775.04	20.49	210.27	556.96
Ag acc (m.g/t)	Gloria_Alto	30	56.6	195.59	148.62	0.76	22088.51	0.00	166.75	554.77

table continues…

Variable	Vein	Count	Total Length (m)	Mean	SD	Coefficient of Variation	Variance	Minimum	Median	Maximum
Ag acc (m.g/t)	Gloria_Alto_2	16	15.3	38.50	46.40	1.21	2153.27	0.00	18.35	127.44
Ag acc (m.g/t)	Gloria_Alto_2	16	15.3	0.10	0.21	2.04	0.04	0.00	0.02	0.66
Ag acc (m.g/t)	Martha	574	4107.9	1538.18	1827.94	1.19	3341355.02	0.00	832.78	9065.98
Ag acc (m.g/t)	Martha_2	75	374.9	1722.04	1708.34	0.99	2918413.17	0.00	1225.96	7159.84
Ag acc (m.g/t)	Martha_3	16	50.3	421.60	341.24	0.81	116445.07	39.32	256.37	954.98
Ag acc (m.g/t)	Martha_5	6	14.5	404.86	400.93	0.99	160747.83	10.57	366.52	1025.45
Ag acc (m.g/t)	Martha_6	54	170.9	907.68	1010.10	1.11	1020299.09	1.03	360.84	2854.20
Ag acc (m.g/t)	Martha_7	16	57.6	1214.17	1225.03	1.01	1500686.68	68.16	515.22	2826.92
Ag acc (m.g/t)	Martha_8	9	54.2	547.01	326.68	0.60	106721.33	68.56	696.94	999.83
Ag acc (m.g/t)	Splay_1	38	100.9	489.09	346.47	0.71	120044.01	4.06	379.09	1034.57
Ag acc (m.g/t)	Splay_2	14	26.3	377.91	310.92	0.82	96670.46	34.13	206.49	816.44
Ag acc (m.g/t)	Transversal	42	139.1	470.68	431.07	0.92	185822.08	0.00	333.19	1760.29
Ag acc (m.g/t)	Transversal_Norte	35	113.1	274.66	376.79	1.37	141970.76	0.00	148.93	2110.07
Au acc (m.g/t)	Abundancia	428	1245.9	1.14	1.52	1.34	2.30	0.00	0.83	15.04

table continues…

Variable	Vein	Count	Total Length (m)	Mean	SD	Coefficient of Variation	Variance	Minimum	Median	Maximum
Au acc (m.g/t)	Abundancia_2	23	35.0	0.14	0.12	0.84	0.01	0.00	0.13	0.47
Au acc (m.g/t)	Abundancia_3	29	59.7	0.33	0.31	0.93	0.09	0.00	0.22	1.16
Au acc (m.g/t)	Abundancia_4	36	58.8	0.08	0.12	1.55	0.02	0.00	0.02	0.49
Au acc (m.g/t)	Abundancia_5	12	22.7	0.65	0.82	1.26	0.68	0.04	0.43	2.61
Au acc (m.g/t)	Abundancia_Bajo	36	99.7	0.53	0.40	0.75	0.16	0.01	0.36	1.42
Au acc (m.g/t)	Abundancia_Bajo_2	30	75.2	0.34	0.17	0.50	0.03	0.00	0.34	0.67
Au acc (m.g/t)	Abundancia_Bajo_3	25	49.5	0.64	0.50	0.78	0.25	0.00	0.59	1.59
Au acc (m.g/t)	Abundancia_Splay_1	42	77.2	0.84	0.94	1.11	0.87	0.00	0.40	2.78
Au acc (m.g/t)	Carmen	12	106.3	0.20	0.16	0.80	0.03	0.00	0.17	0.42
Au acc (m.g/t)	Gloria	280	799.5	1.46	1.77	1.21	3.12	0.00	1.23	21.13
Au acc (m.g/t)	Gloria_2	36	79.8	0.75	1.41	1.89	2.00	0.03	0.43	8.64
Au acc (m.g/t)	Gloria_Alto	30	56.6	0.25	0.23	0.94	0.05	0.00	0.21	1.10
Au acc (m.g/t)	Martha	574	4107.9	2.93	3.27	1.12	10.68	0.00	1.51	14.54
Au acc (m.g/t)	Martha_2	75	374.9	3.55	3.18	0.90	10.10	0.00	2.30	12.31

table continues…

Variable	Vein	Count	Total Length (m)	Mean	SD	Coefficient of Variation	Variance	Minimum	Median	Maximum
Au acc (m.g/t)	Martha_3	16	50.3	0.73	0.73	1.00	0.54	0.03	0.32	2.11
Au acc (m.g/t)	Martha_5	6	14.5	1.65	0.86	0.52	0.75	0.02	1.84	2.34
Au acc (m.g/t)	Martha_6	54	170.9	1.67	1.37	0.82	1.87	0.01	1.23	4.06
Au acc (m.g/t)	Martha_7	16	57.6	1.54	1.34	0.87	1.81	0.06	0.89	3.29
Au acc (m.g/t)	Martha_8	9	54.2	0.66	0.41	0.61	0.16	0.05	0.66	1.10
Au acc (m.g/t)	Splay_1	38	100.9	0.85	0.87	1.02	0.75	0.01	0.55	4.00
Au acc (m.g/t)	Splay_2	14	26.3	0.50	0.43	0.86	0.18	0.03	0.34	1.07
Au acc (m.g/t)	Transversal	42	139.1	0.57	0.42	0.73	0.18	0.00	0.60	1.43
Au acc (m.g/t)	Transversal_Norte	35	113.1	0.45	0.40	0.88	0.16	0.00	0.38	1.90
True thickness (m)	Abundancia	428	1245.9	3.64	2.41	0.66	5.82	0.00	3.10	11.84
True thickness (m)	Abundancia_2	23	35.0	2.17	1.10	0.51	1.22	0.19	2.36	3.91
True thickness (m)	Abundancia_3	29	59.7	2.22	1.02	0.46	1.05	0.21	1.87	3.86
True thickness (m)	Abundancia_4	36	58.8	1.97	0.94	0.48	0.88	0.17	1.83	3.85
True thickness (m)	Abundancia_5	12	22.7	2.06	0.72	0.35	0.52	0.95	1.68	3.06

table continues…

Variable	Vein	Count	Total Length (m)	Mean	SD	Coefficient of Variation	Variance	Minimum	Median	Maximum
True thickness (m)	Abundancia_Bajo	36	99.7	3.51	1.50	0.43	2.25	0.31	3.84	6.47
True thickness (m)	Abundancia_Bajo_2	30	75.2	3.56	1.62	0.45	2.62	0.28	3.83	5.71
True thickness (m)	Abundancia_Bajo_3	25	49.5	2.60	1.36	0.52	1.86	0.43	2.46	5.06
True thickness (m)	Abundancia_Splay_1	42	77.2	2.75	1.88	0.68	3.54	0.40	2.28	7.26
True thickness (m)	Carmen	12	106.3	1.13	0.96	0.85	0.93	0.01	0.66	3.09
True thickness (m)	Gloria	280	799.5	3.96	2.51	0.63	6.29	0.03	3.20	10.34
True thickness (m)	Gloria_2	36	79.8	2.30	0.92	0.40	0.85	0.46	2.31	4.12
True thickness (m)	Gloria_Alto	30	56.6	2.77	1.20	0.43	1.44	0.17	3.24	4.52
True thickness (m)	Gloria_Alto_2	16	15.3	1.19	0.51	0.43	0.26	0.19	1.12	1.99
True thickness (m)	Martha	574	4107.9	11.88	7.97	0.67	63.47	0.14	10.46	46.72
True thickness (m)	Martha_2	75	374.9	7.02	3.34	0.48	11.16	0.19	6.40	12.69
True thickness (m)	Martha_3	16	50.3	4.21	1.89	0.45	3.57	0.62	4.06	6.50
True thickness (m)	Martha_5	6	14.5	3.37	1.42	0.42	2.01	0.24	3.66	4.60
True thickness (m)	Martha_6	54	170.9	6.13	4.46	0.73	19.93	0.21	4.69	13.64

table continues…

Variable	Vein	Count	Total Length (m)	Mean	SD	Coefficient of Variation	Variance	Minimum	Median	Maximum
True thickness (m)	Martha_7	16	57.6	6.54	4.04	0.62	16.32	0.60	5.78	11.79
True thickness (m)	Martha_8	9	54.2	8.69	6.01	0.69	36.09	1.83	8.22	15.81
True thickness (m)	Splay_1	38	100.9	3.44	1.72	0.50	2.97	0.03	2.63	7.18
True thickness (m)	Splay_2	14	26.3	2.96	1.84	0.62	3.39	0.40	3.63	5.60
True thickness (m)	Transversal	42	139.1	3.72	1.94	0.52	3.76	0.44	3.26	6.62
True thickness (m)	Transversal_Norte	35	113.1	4.28	2.54	0.59	6.47	0.18	3.65	8.93

The composited accumulation values and the thickness of the veins were independently estimated using OK and inverse distance weighting (ID). The grades were estimated as the quotient of the accumulation estimates and the thickness estimates.

The QP considers that two-dimensional (accumulation) estimation of metal accumulations (metal sample grade multiplied by thickness, accumulated over total vein thickness) and vein thickness are appropriate for estimation of relatively narrow veins (Bertoli et al. 2003).

14.2.6 Domain Estimation Boundaries

Grade estimates in each of the vein domains were treated as discrete ("hard") boundaries and only the composites falling within each vein volume were used for the estimation of that specific vein.

14.2.7 Density Assignment

Zangrandi (2020) studied on the available density information (89,208 SG/density measures), dividing the samples by domain. Samples with values outside of ±2 SDs were considered outliers and were separated from each population. The summary of density by lithology is shown in Table 14-38.

Table 14-38: Summary of Density by Unit

Unit	SG
Vein	2.55
Basalt	2.40
Abundancia Wall Rock	2.49
Gloria Wall Rock	2.47
Other wall rock	2.52

A value of 2.55 t/m^3 was applied to the MREs for the veins.

14.2.8 Grade Capping/Outlier Restrictions

Capping values were applied to the metal accumulation variables based on analysis of histograms and log histograms. Capping values are shown in Table 14-39.

Table 14-39: Capping Values for Metal Accumulations

Domained Estimation Name	Capping Value
Ag_m_gpt_Abundancia	2500
Ag_m_gpt_Gloria	2000
Ag_m_gpt_Martha	4000
Ag_m_gpt_Martha_2	6000
Au_m_gpt_Abundancia	5
Au_m_gpt_Gloria	10
Au_m_gpt_Martha	10
Au_m_gpt_Martha_2	12

14.2.9 Variography

Metal accumulation variography was modelled based on normal scores of metal accumulations. Thickness variography was modelled using raw thickness data. In all cases, variograms were modelled with two spherical structures and within a best fit plane for each vein. The variogram model parameters used for Kriging estimates are summarized in Table 14-40.

Table 14-40: Variogram Parameters

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalized Nugget	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor
Ag_m_gpt_Abundancia: Transformed Variogram Model	34.5	286.4	160.4	Data	297,213	64,644	0.218	80,158	0.270	Spherical		59	32	500	152,025	0.512	Spherical		225	106	500
Ag_m_gpt_Abundancia_2: Transformed Variogram Model	34.5	260.1	32.7	Data	19,263	5,977	0.310	5,868	0.305	Spherical		48	27	500	7,422	0.385	Spherical		227	150	500
Ag_m_gpt_Abundancia_3: Transformed Variogram Model	20.7	268.1	7.0	Data	37,977	9,407	0.248	10,896	0.287	Spherical		48	27	500	17,682	0.466	Spherical		227	150	500
Ag_m_gpt_Abundancia_4: Transformed Variogram Model	14.9	286.0	9.6	Data	31,390	9,006	0.287	9,781	0.312	Spherical		48	27	500	12,594	0.401	Spherical		227	150	500
Ag_m_gpt_Abundancia_5: Transformed Variogram Model	10.1	211.4	173.2	Data	16,599	4,427	0.267	4,482	0.270	Spherical		48	27	500	7,683	0.463	Spherical		227	150	500
Ag_m_gpt_Abundancia_Bajo: Transformed Variogram Model	30.8	264.1	150.2	Data	103,273	23,205	0.225	28,607	0.277	Spherical		48	27	500	51,451	0.498	Spherical		227	150	500
Ag_m_gpt_Abundancia_Bajo_2: Transformed Variogram Model	58.6	255.3	156.3	Data	345,151	101,336	0.294	101,716	0.295	Spherical		48	27	500	141,995	0.411	Spherical		227	150	500
Ag_m_gpt_Abundancia_Bajo_3: Transformed Variogram Model	19.9	252.3	11.9	Data	212,255	54,231	0.256	60,450	0.285	Spherical		48	27	500	97,510	0.459	Spherical		227	150	500
Ag_m_gpt_Abundancia_Splay_1: Transformed Variogram Model	42.5	266.0	174.2	Data	79,731	20,810	0.261	23,632	0.296	Spherical		48	27	500	35,265	0.442	Spherical		227	150	500
Ag_m_gpt_Carmen: Transformed Variogram Model	77.8	16.0	37.3	Data	14,406	3,254	0.226	3,810	0.265	Spherical		48	27	500	7,334	0.509	Spherical		227	150	500
Ag_m_gpt_Gloria: Transformed Variogram Model	82.9	259.0	170.4	Data	351,447	98,300	0.280	148,240	0.422	Spherical		68	23	500	106,453	0.303	Spherical		272	150	500
Ag_m_gpt_Gloria_2: Transformed Variogram Model	31.6	274.9	174.5	Data	23,048	5,107	0.222	6,191	0.269	Spherical		48	27	500	11,747	0.510	Spherical		227	150	500
Ag_m_gpt_Gloria_Alto: Transformed Variogram Model	52.6	278.9	154.5	Data	19,850	4,375	0.220	5,548	0.280	Spherical		48	27	500	9,921	0.500	Spherical		227	150	500
Ag_m_gpt_Gloria_Alto_2: Transformed Variogram Model	31.9	278.1	131.5	Data	1,631	468	0.287	457	0.280	Spherical		48	27	500	706	0.433	Spherical		227	150	500

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalized Nugget	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor
Ag_m_gpt_Martha: Transformed Variogram Model	21.7	233.9	91.0	Data	1,455,576	280,344	0.193	260,111	0.179	Spherical		29	29	500	916,576	0.630	Spherical		255	230	500
Ag_m_gpt_Martha_2: Transformed Variogram Model	19.4	271.6	44.1	Data	2,165,255	296,423	0.137	312,446	0.144	Spherical		34	23	500	1,547,725	0.715	Spherical		156	174	500
Ag_m_gpt_Martha_3: Transformed Variogram Model	24.7	258.7	167.5	Data	81,930	21,244	0.259	23,006	0.281	Spherical		48	27	500	37,688	0.460	Spherical		227	150	500
Ag_m_gpt_Martha_5: Transformed Variogram Model	28.5	254.2	164.2	Data	146,856	40,620	0.277	40,723	0.277	Spherical		48	27	500	65,571	0.447	Spherical		227	150	500
Ag_m_gpt_Martha_6: Transformed Variogram Model	29.4	272.3	164.1	Data	388,711	113,309	0.292	121,472	0.313	Spherical		48	27	500	153,968	0.396	Spherical		227	150	500
Ag_m_gpt_Martha_7: Transformed Variogram Model	22.9	267.8	155.5	Data	693,785	218,820	0.315	205,569	0.296	Spherical		48	27	500	269,397	0.388	Spherical		227	150	500
Ag_m_gpt_Martha_8: Transformed Variogram Model	18.1	276.3	122.7	Data	102,679	23,226	0.226	27,805	0.271	Spherical		48	27	500	51,617	0.503	Spherical		227	150	500
Ag_m_gpt_Splay_1: Transformed Variogram Model	52.4	261.9	146.9	Data	102,030	23,783	0.233	27,671	0.271	Spherical		48	27	500	50,597	0.496	Spherical		227	150	500
Ag_m_gpt_Splay_2: Transformed Variogram Model	39.7	266.0	165.0	Data	70,152	18,436	0.263	19,341	0.276	Spherical		48	27	500	32,382	0.462	Spherical		227	150	500
Ag_m_gpt_Transversal: Transformed Variogram Model	66.6	182.0	13.2	Data	149,376	36,029	0.241	42,946	0.288	Spherical		48	27	500	70,386	0.471	Spherical		227	150	500
Ag_m_gpt_Transversal_Norte: Transformed Variogram Model	45.0	170.8	26.4	Data	144,309	45,183	0.313	46,568	0.323	Spherical		48	27	500	52,543	0.364	Spherical		227	150	500
Au_m_gpt_Abundancia: Transformed Variogram Model	34.6	285.5	161.8	Data	1	1	0.682	0	0.210	Spherical		42	30	500	0	0.109	Spherical		106	93	500
Au_m_gpt_Abundancia_2: Transformed Variogram Model	34.5	260.1	32.7	Data	0	0	0.431	0	0.206	Spherical		43	27	500	0	0.363	Spherical		175	131	500
Au_m_gpt_Abundancia_3: Transformed Variogram Model	20.7	268.1	7.0	Data	0	0	0.428	0	0.208	Spherical		43	27	500	0	0.365	Spherical		175	131	500

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalized Nugget	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor
Au_m_gpt_Abundancia_4: Transformed Variogram Model	14.9	286.0	9.6	Data	0	0	0.503	0	0.212	Spherical		43	27	500	0	0.285	Spherical		175	131	500
Au_m_gpt_Abundancia_5: Transformed Variogram Model	10.1	211.4	173.2	Data	1	0	0.498	0	0.203	Spherical		43	27	500	0	0.300	Spherical		175	131	500
Au_m_gpt_Abundancia_Bajo: Transformed Variogram Model	30.8	264.1	150.2	Data	0	0	0.430	0	0.203	Spherical		43	27	500	0	0.366	Spherical		175	131	500
Au_m_gpt_Abundancia_Bajo_2: Transformed Variogram Model	58.6	255.3	156.3	Data	0	0	0.409	0	0.201	Spherical		43	27	500	0	0.390	Spherical		175	131	500
Au_m_gpt_Abundancia_Bajo_3: Transformed Variogram Model	19.9	252.3	11.9	Data	0	0	0.417	0	0.203	Spherical		43	27	500	0	0.379	Spherical		175	131	500
Au_m_gpt_Abundancia_Splay_1: Transformed Variogram Model	42.5	266.0	174.2	Data	0	0	0.455	0	0.208	Spherical		43	27	500	0	0.337	Spherical		175	131	500
Au_m_gpt_Carmen: Transformed Variogram Model	77.8	16.0	37.3	Data	0	0	0.436	0	0.190	Spherical		43	27	500	0	0.373	Spherical		175	131	500
Au_m_gpt_Gloria: Transformed Variogram Model	80.4	259.9	168.1	Data	3	2	0.604	0	0.134	Spherical		44	31	9	1	0.259	Spherical		180	140	64
Au_m_gpt_Gloria_2: Transformed Variogram Model	31.6	274.9	174.5	Data	23,048	9,689	0.420	4,644	0.202	Spherical		43	27	500	8,712	0.378	Spherical		175	131	500
Au_m_gpt_Gloria_Alto: Transformed Variogram Model	52.6	278.9	154.5	Data	0	0	0.463	0	0.207	Spherical		43	27	500	0	0.330	Spherical		175	131	500
Au_m_gpt_Gloria_Alto_2: Transformed Variogram Model	31.9	278.1	131.5	Data	0	0	0.574	0	0.197	Spherical		43	27	500	0	0.229	Spherical		175	131	500
Au_m_gpt_Martha: Transformed Variogram Model	21.7	233.9	91.0	Data	6	0	0.043	1	0.226	Spherical		59	31	500	4	0.730	Spherical		209	144	500
Au_m_gpt_Martha_2: Transformed Variogram Model	19.4	271.6	31.1	Data	8	1	0.174	2	0.231	Spherical		27	17	500	5	0.592	Spherical		203	149	500
Au_m_gpt_Martha_3: Transformed Variogram Model	24.7	258.7	167.5	Data	0	0	0.482	0	0.202	Spherical		43	27	500	0	0.317	Spherical		175	131	500

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalized Nugget	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor
Au_m_gpt_Martha_5: Transformed Variogram Model	28.5	254.2	164.2	Data	1	0	0.453	0	0.191	Spherical		43	27	500	0	0.356	Spherical		175	131	500
Au_m_gpt_Martha_6: Transformed Variogram Model	29.4	272.3	164.1	Data	1	0	0.458	0	0.207	Spherical		43	27	500	0	0.335	Spherical		175	131	500
Au_m_gpt_Martha_7: Transformed Variogram Model	22.9	267.8	155.5	Data	1	0	0.499	0	0.198	Spherical		43	27	500	0	0.303	Spherical		175	131	500
Au_m_gpt_Martha_8: Transformed Variogram Model	18.1	276.3	122.7	Data	0	0	0.416	0	0.201	Spherical		43	27	500	0	0.383	Spherical		175	131	500
Au_m_gpt_Splay_1: Transformed Variogram Model	52.4	261.9	146.9	Data	1	0	0.459	0	0.215	Spherical		43	27	500	0	0.327	Spherical		175	131	500
Au_m_gpt_Splay_2: Transformed Variogram Model	39.7	266.0	165.0	Data	0	0	0.470	0	0.197	Spherical		43	27	500	0	0.333	Spherical		175	131	500
Au_m_gpt_Transversal: Transformed Variogram Model	66.6	182.0	13.2	Data	0	0	0.415	0	0.205	Spherical		43	27	500	0	0.381	Spherical		175	131	500
Au_m_gpt_Transversal_Norte: Transformed Variogram Model	45.0	170.8	26.4	Data	0	0	0.447	0	0.209	Spherical		43	27	500	0	0.343	Spherical		175	131	500
True_thick_Abundancia: Variogram Model	19.1	275.5	142.9	Data	4	0	0.013	1	0.219	Spherical		32	21	500	6	1.537	Spherical		162	83	500
True_thick_Abundancia_2: Variogram Model	20.0	262.1	153.2	Data	1	0	0.060	0	0.290	Spherical		39	30	500	1	0.840	Spherical		195	96	500
True_thick_Abundancia_3: Variogram Model	20.7	268.1	7.0	Data	1	0	0.060	0	0.290	Spherical		39	30	500	1	0.840	Spherical		195	96	500
True_thick_Abundancia_4: Variogram Model	14.9	286.0	9.6	Data	1	0	0.060	0	0.290	Spherical		39	30	500	1	0.840	Spherical		195	96	500
True_thick_Abundancia_5: Variogram Model	84.0	258.6	158.2	Data	0	0	0.060	0	0.290	Spherical		39	30	500	0	0.840	Spherical		195	96	500
True_thick_Abundancia_Bajo: Variogram Model	30.8	264.1	150.2	Data	2	0	0.060	1	0.290	Spherical		39	30	500	2	0.840	Spherical		195	96	500
True_thick_Abundancia_Bajo_2: Variogram Model	58.6	255.3	156.3	Data	3	0	0.060	1	0.290	Spherical		30	39	500	3	0.840	Spherical		195	96	500

table continues…

General	Direction							Structure 1							Structure 2						
Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalized Nugget	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor	Sill	Normalized sill	Structure	Alpha	Major	Semi-major	Minor
True_thick_Abundancia_Bajo_3: Variogram Model	19.9	252.3	11.9	Data	1	0	0.060	0	0.290	Spherical		39	30	500	1	0.840	Spherical		195	96	500
True_thick_Abundancia_Splay_1: Variogram Model	42.5	266.0	174.2	Data	2	0	0.060	1	0.290	Spherical		39	30	500	2	0.840	Spherical		195	96	500
True_thick_Carmen: Variogram Model	77.8	16.0	37.3	Data	1	0	0.060	0	0.290	Spherical		39	30	500	1	0.840	Spherical		195	96	500
True_thick_Gloria: Variogram Model	84.0	258.6	158.2	Data	3	0	0.141	0	0.136	Spherical		22	21	500	2	0.724	Spherical		226	42	500
True_thick_Gloria_2: Variogram Model	31.6	274.9	174.5	Data	1	0	0.060	0	0.290	Spherical		39	30	500	1	0.840	Spherical		195	96	500
True_thick_Gloria_Alto: Variogram Model	52.6	278.9	154.5	Data	2	0	0.060	1	0.290	Spherical		39	30	500	2	0.840	Spherical		195	96	500
True_thick_Gloria_Alto_2: Variogram Model	31.9	278.1	131.5	Data	0	0	0.060	0	0.290	Spherical		39	30	500	0	0.840	Spherical		195	96	500
True_thick_Martha: Variogram Model	21.4	228.6	4.7	Data	35	4	0.105	13	0.382	Spherical		54	47	500	18	0.517	Spherical		243	168	500
True_thick_Martha_2: Variogram Model	17.1	271.5	145.5	Data	11	0	0.000	5	0.420	Spherical		47	31	500	6	0.576	Spherical		148	91	500
True_thick_Martha_3: Variogram Model	24.7	258.7	167.5	Data	4	0	0.060	1	0.290	Spherical		39	30	500	3	0.840	Spherical		195	96	500
True_thick_Martha_5: Variogram Model	28.5	254.2	164.2	Data	3	0	0.060	1	0.290	Spherical		39	30	500	2	0.840	Spherical		195	96	500
True_thick_Martha_6: Variogram Model	29.4	272.3	164.1	Data	10	1	0.060	3	0.290	Spherical		39	30	500	8	0.840	Spherical		195	96	500
True_thick_Martha_7: Variogram Model	22.9	267.8	155.5	Data	11	1	0.060	3	0.290	Spherical		39	30	500	9	0.840	Spherical		195	96	500
True_thick_Martha_8: Variogram Model	21.5	267.3	6.5	Data	19	1	0.060	6	0.290	Spherical		39	30	500	16	0.840	Spherical		195	96	500
True_thick_Splay_1: Variogram Model	52.4	261.9	146.9	Data	3	0	0.060	1	0.290	Spherical		39	30	500	2	0.840	Spherical		195	96	500
True_thick_Splay_2: Variogram Model	48.1	260.8	169.3	Data	2	0	0.060	1	0.290	Spherical		39	30	500	2	0.840	Spherical		195	96	500
True_thick_Transversal: Variogram Model	39.7	266.0	165.0	Data	3	0	0.060	1	0.290	Spherical		39	30	500	3	0.840	Spherical		195	96	500
True_thick_Transversal_Norte: Variogram Model	45.0	170.8	26.4	Data	5	0	0.060	1	0.290	Spherical		39	30	500	4	0.840	Spherical		195	96	500

14.2.10 Estimation/Interpolation Methods

Block estimates require at least five relevant vein intercepts within the search ellipsoid. In addition, to reduce the marginal effects of extrapolation, an octant search was also applied. Variable orientations, with the variograms rotated parallel to the mid-vein reference surface of the relevant domain. Search parameters are detailed in Table 14-41.

Table 14-41: Grade Interpolant and Search Parameters

General		Ellipsoid Ranges				Number of Vein Intercepts		Sector Search		
Interpolant Name	Numeric Values	Maximum	Intermediate	Minimum	Variable Orientation	Minimum	Maximum	Method	Max Samples	Max Empty Sectors
ID, Ag_m_gpt_Abundancia	linear_grade	520	210	500	Variable Orientation	5	20	Octant	4	6
ID, Ag_m_gpt_Gloria	linear_grade	540	230	500	Variable Orientation	5	20	Octant	4	6
ID, Ag_m_gpt_Martha	linear_grade	380	580	500	Variable Orientation	5	20	Octant	4	6
ID, Ag_m_gpt_Martha_2	linear_grade	126.4	139.4	500	Variable Orientation	5	20	Octant	4	6
ID, Au_m_gpt_Abundancia	linear_grade	340	300	500	Variable Orientation	5	20	Octant	4	6
ID, Au_m_gpt_Gloria	linear_grade	126.4	139.4	63.71	Variable Orientation	5	20	Octant	4	6
ID, Au_m_gpt_Martha	linear_grade	250	280	500	Variable Orientation	5	20	Octant	4	6
ID, Au_m_gpt_Martha_2	linear_grade	300	500	500	Variable Orientation	5	20	Octant	4	6
ID, True_thick_Abundancia	true_length	320	160	500	Variable Orientation	5	20	Octant	4	6
ID, True_thick_Gloria	true_length	450	140	500	Variable Orientation	5	20	Octant	4	6
ID, True_thick_Martha	true_length	500	300	500	Variable Orientation	5	20	Octant	4	6
ID, True_thick_Martha_2	true_length	300	170	500	Variable Orientation	5	20	Octant	4	6
Kr, Ag_m_gpt_Abundancia	linear_grade	520	210	500	Variable Orientation	5	20	Octant	4	6
Kr, Ag_m_gpt_Gloria	linear_grade	540	230	500	Variable Orientation	5	20	Octant	4	6
Kr, Ag_m_gpt_Martha	linear_grade	380	580	500	Variable Orientation	5	20	Octant	4	6
Kr, Ag_m_gpt_Martha_2	linear_grade	126.4	139.4	500	Variable Orientation	5	20	Octant	4	6
Kr, Au_m_gpt_Abundancia	linear_grade	340	300	500	Variable Orientation	5	20	Octant	4	6
Kr, Au_m_gpt_Gloria	linear_grade	126.4	139.4	63.71	Variable Orientation	5	20	Octant	4	6
Kr, Au_m_gpt_Martha	linear_grade	260	280	500	Variable Orientation	5	20	Octant	4	6
Kr, Au_m_gpt_Martha_2	linear_grade	300	500	500	Variable Orientation	5	20	Octant	4	6
Kr, True_thick_Abundancia	true_length	320	160	500	Variable Orientation	5	20	Octant	4	6
Kr, True_thick_Gloria	true_length	450	140	500	Variable Orientation	5	20	Octant	4	6
Kr, True_thick_Martha	true_length	500	300	500	Variable Orientation	5	20	Octant	4	6
Kr, True_thick_Martha_2	true_length	300	170	500	Variable Orientation	5	20	Octant	4	6

14.2.11 Block Model Parameters

Block Models were generated for each of the 23 veins that were estimated. Estimates were made into blocks with dimensions 15 m x 15 m x vein thickness oriented in the best fit plane of each vein. Sub-blocking was applied to a minimum block size of 5 m x 5 m x vein thickness. Unique Block Models parameters are summarized in Table 14-42.

Table 14-42: Block Model Parameters

Vein	Base point			Block Size		Orientation		Block Count		
	X	Y	Z	dx	dy	Dip	Azimuth	nx	ny	nz
Abundancia_2	555,271.63	2,700,844.59	2,289.05	15	15	33.35	260.52	48	37	1
Abundancia_3	555,502.00	2,700,584.73	2,317.58	15	15	20.03	262.08	75	40	1
Abundancia_4	555,647.44	2,700,470.98	2,364.39	15	15	20.03	262.08	85	60	1
Abundancia_5	555,630.95	2,699,823.61	2,208.91	15	15	9.94	206.94	58	60	1
Abundancia_Bajo_2	555,201.96	2,702,285.54	2,209.54	15	15	60.65	257.81	37	34	1
Abundancia_Bajo_3	555,378.93	2,701,949.44	2,142.58	15	15	17.71	259.03	41	49	1
Abundancia_Bajo	555,148.66	2,701,667.00	2,318.06	15	15	35.40	270.39	36	31	1
Abundancia	555,054.24	2,701,007.04	2,369.73	15	15	35.40	270.39	166	52	1
Abundancia_Splay_1	555,054.38	2,702,040.75	2,285.77	15	15	36.60	269.21	36	34	1
Carmen	554,539.57	2,702,643.01	2,215.39	15	15	80.82	16.51	46	29	1
Gloria_2	554,889.56	2,701,582.63	2,264.01	15	15	83.35	258.67	40	22	1
Gloria_Alto_2	554,882.84	2,701,887.31	2,197.20	15	15	27.82	278.20	14	10	1
Gloria_Alto	554,929.68	2,701,833.83	2,225.69	15	15	49.63	268.87	17	22	1
Gloria	554,917.09	2,701,445.23	2,264.01	15	15	83.35	258.67	98	32	1
Martha_2	555,652.22	2,702,045.98	2,090.43	15	15	19.83	252.22	44	58	1
Martha_3	555,594.45	2,702,147.37	2,027.39	15	15	25.92	266.55	20	23	1
Martha_5	555,297.74	2,702,229.18	1,868.14	15	15	28.11	254.32	19	17	1
Martha_6	555,457.16	2,701,958.70	1,943.06	15	15	29.66	269.32	40	47	1
Martha_7	555,375.00	2,702,040.00	1,910.00	15	15	21.53	267.34	28	22	1
Martha_8	556,191.85	2,701,899.06	2,054.51	15	15	16.64	274.78	18	20	1
Martha	557,779.25	2,700,595.47	2,383.22	15	15	25.00	226.36	264	114	1

table continues…

Vein	Base point			Block Size		Orientation		Block Count		
	X	Y	Z	dx	dy	Dip	Azimuth	nx	ny	nz
Splay_1	555,024.89	2,701,626.57	2,307.21	15	15	48.09	260.84	35	20	1
Splay_2	555,005.01	2,701,869.36	2,307.21	15	15	40.99	265.32	19	20	1
Transversal_Norte	555,800.94	2,701,870.69	2,269.61	15	15	44.99	170.95	66	44	1
Transversal	555,524.68	2,701,615.21	2,341.73	15	15	66.64	181.97	59	41	1

14.2.12 Block Model Validation

Several validation techniques have been utilized to ensure that the estimates are reasonable.

Swath plots comparing composite grade to the kriged estimate in corridors in the rotated X and Y directions. Comparison was also made with ID2 estimates (examples are shown in Figure 14-60 to Figure 14-63.

Visual comparisons of block estimates and vein composites are shown in Figure 14-64 to Figure 14-71.

Comparison of grade–tonnage curves for the kriged estimates and the previous MREs and the inverse distance estimates (see Figure 14-58 to Figure 14-61).

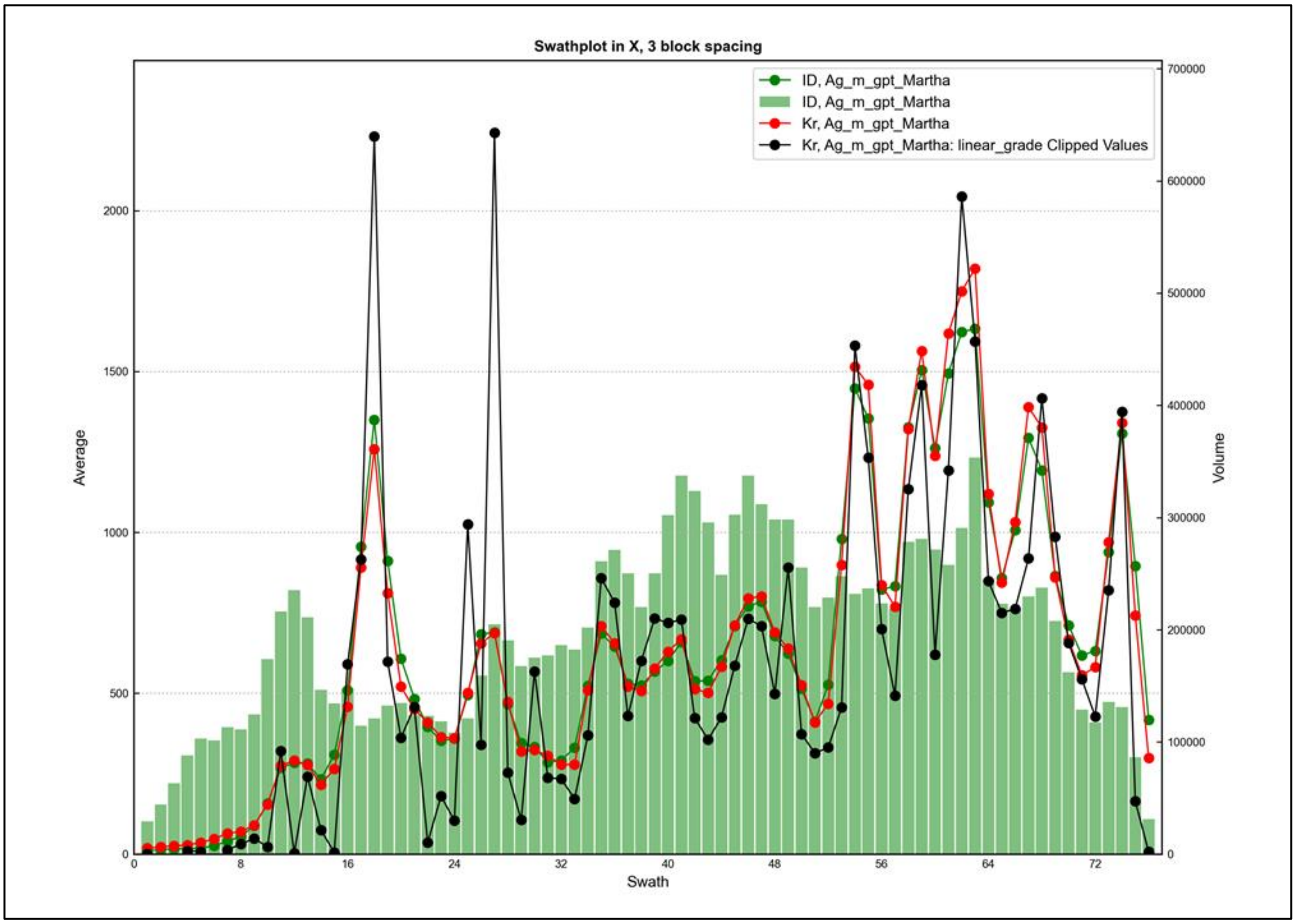

Figure 14-60: Silver Rotated X Swathplot Example (all estimated blocks, bars represent number of blocks)

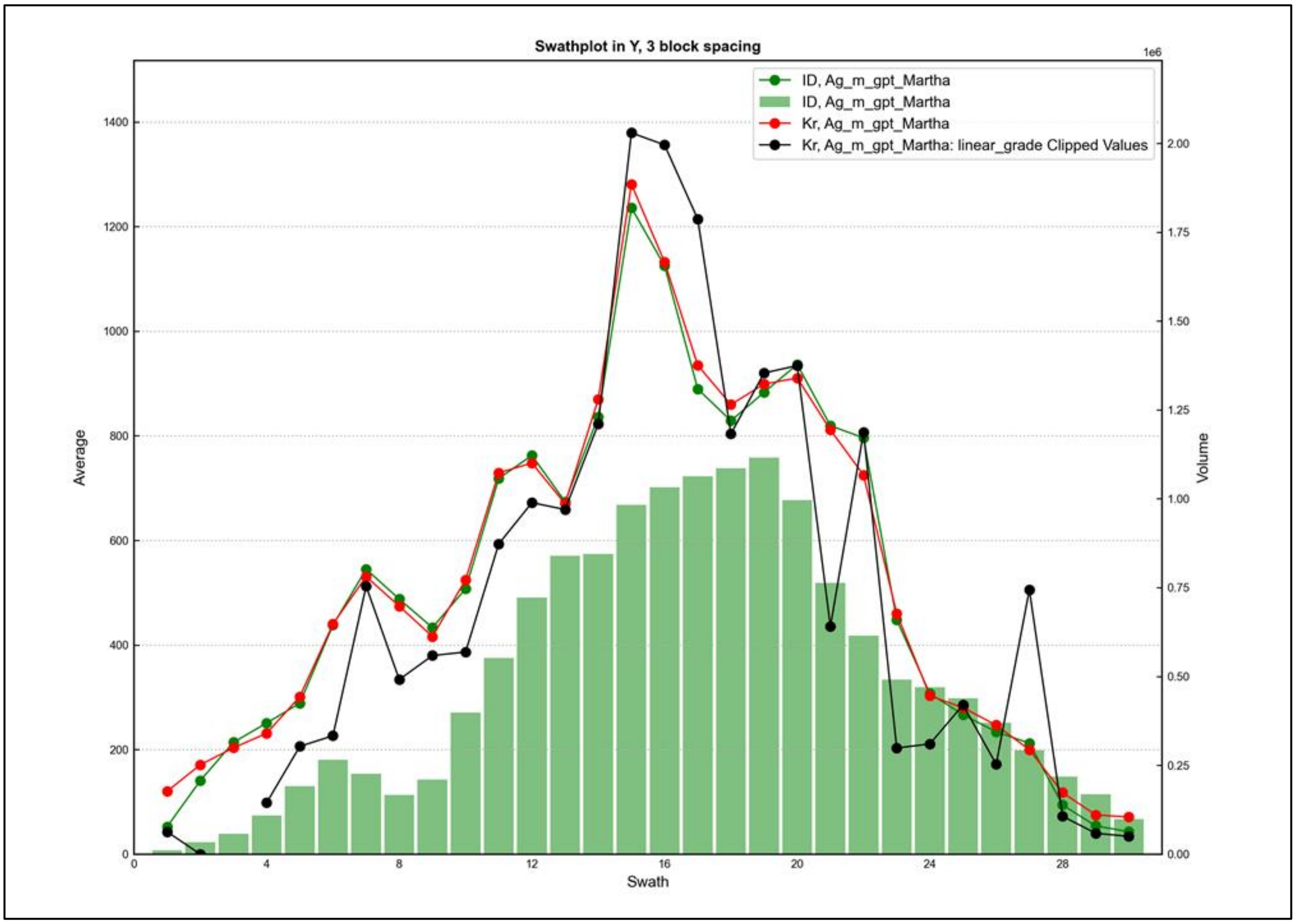

Figure 14-61: Silver Rotated Y Swathplot Example (Red Pennant 2021)

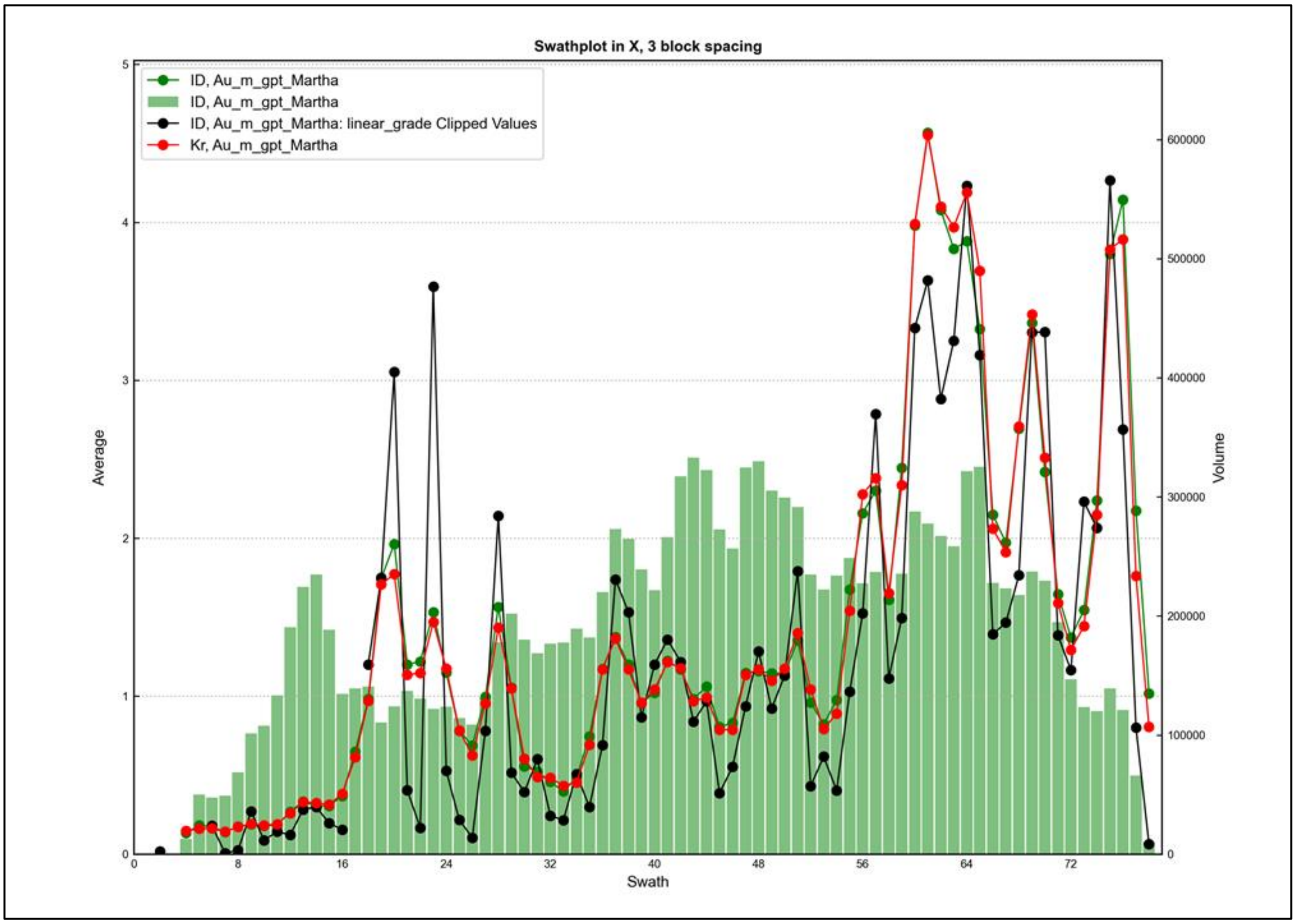

Figure 14-62: Gold Rotated X Swathplot Example (Red Pennant 2021)

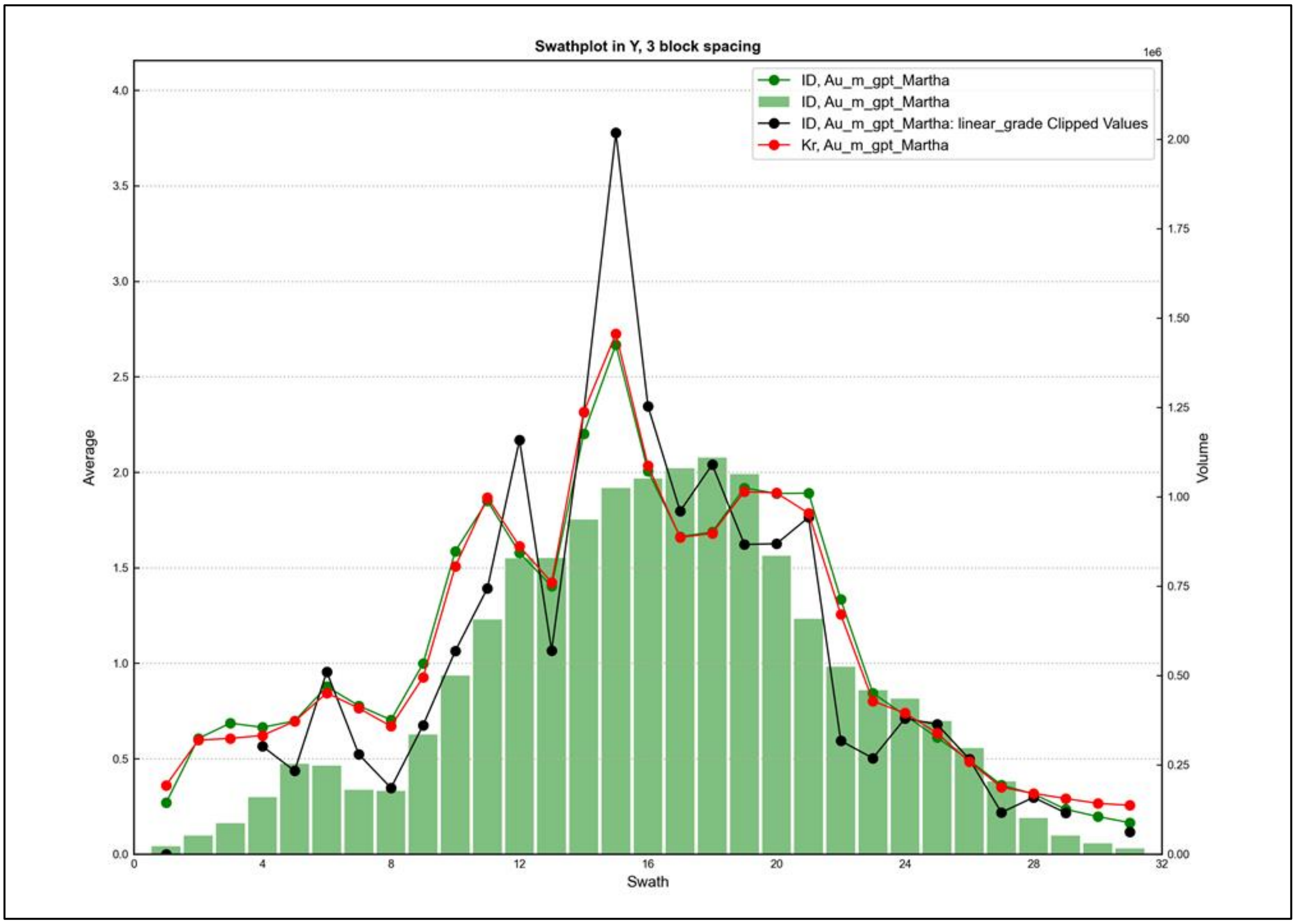

Figure 14-63: Gold Rotated Y Swathplot Example (Red Pennant 2021)

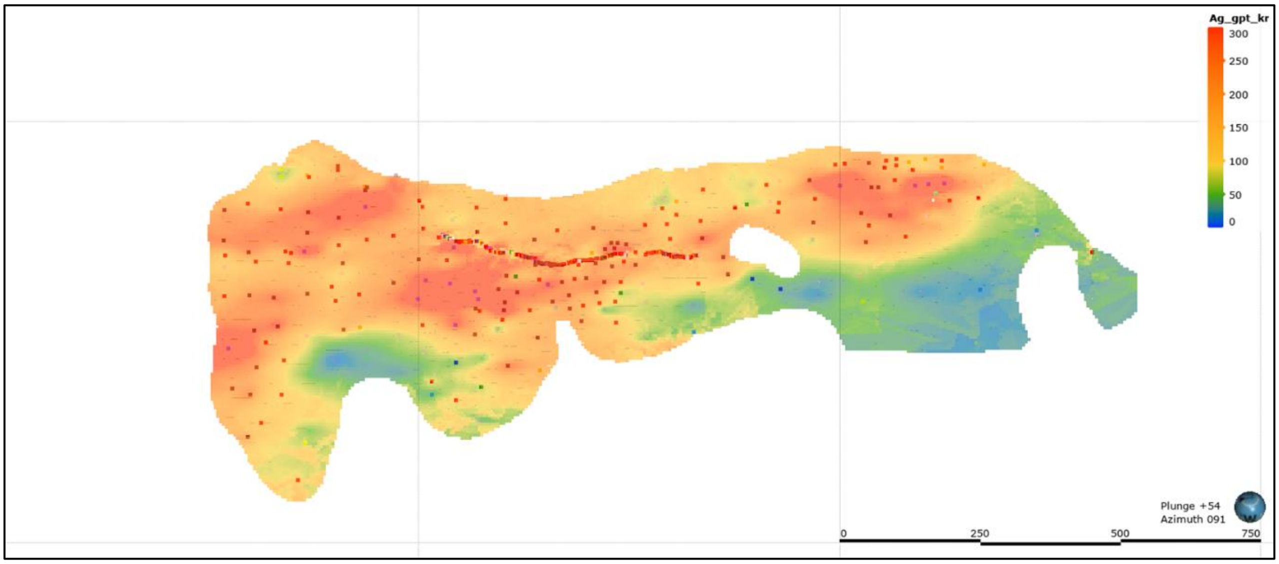

Figure 14-64: Silver Grade Estimates – Abundancia Vein (Red Pennant 2021)

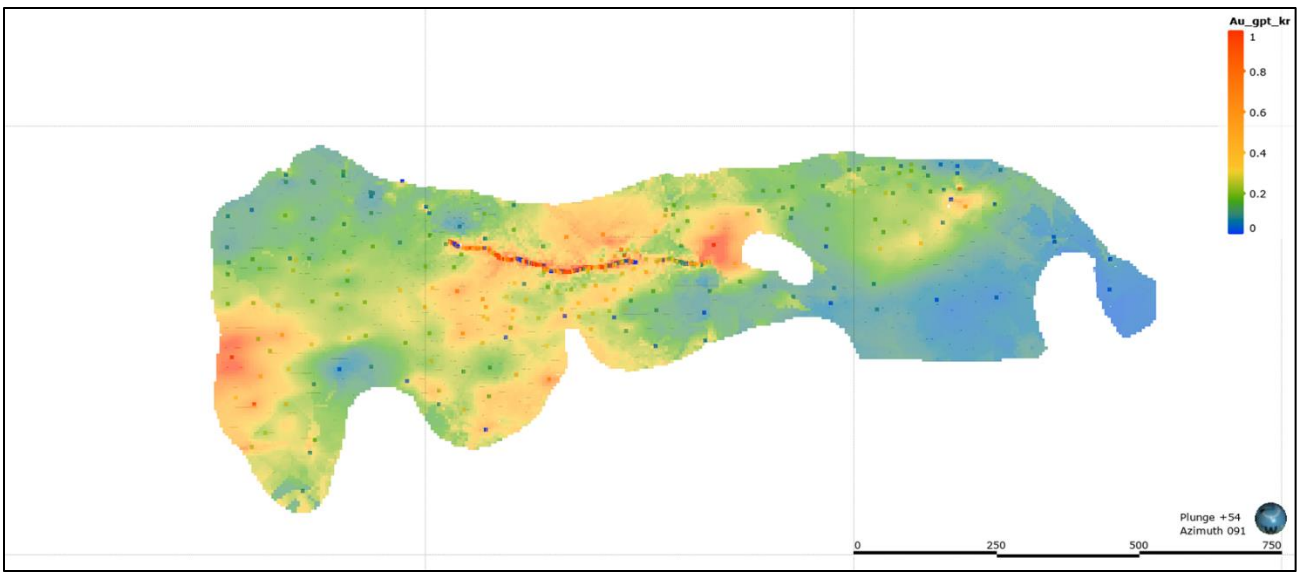

Figure 14-65: Gold Grade Estimates – Abundancia Vein (Red Pennant 2021)

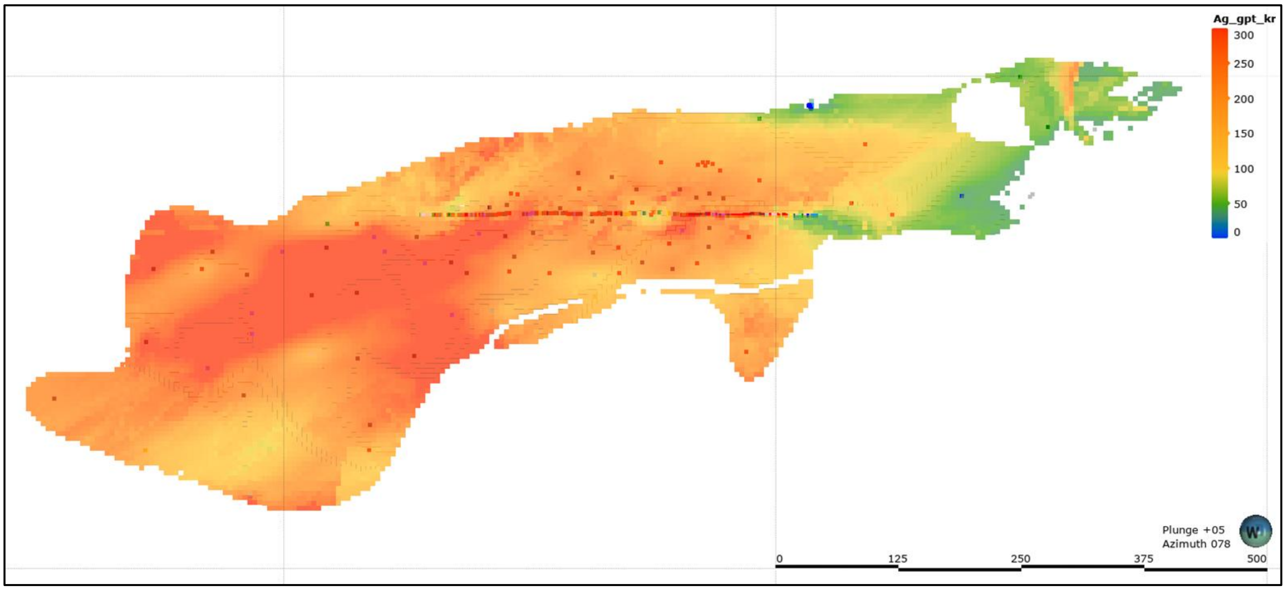

Figure 14-66: Silver Grade Estimates – Gloria Vein (Red Pennant 2021)

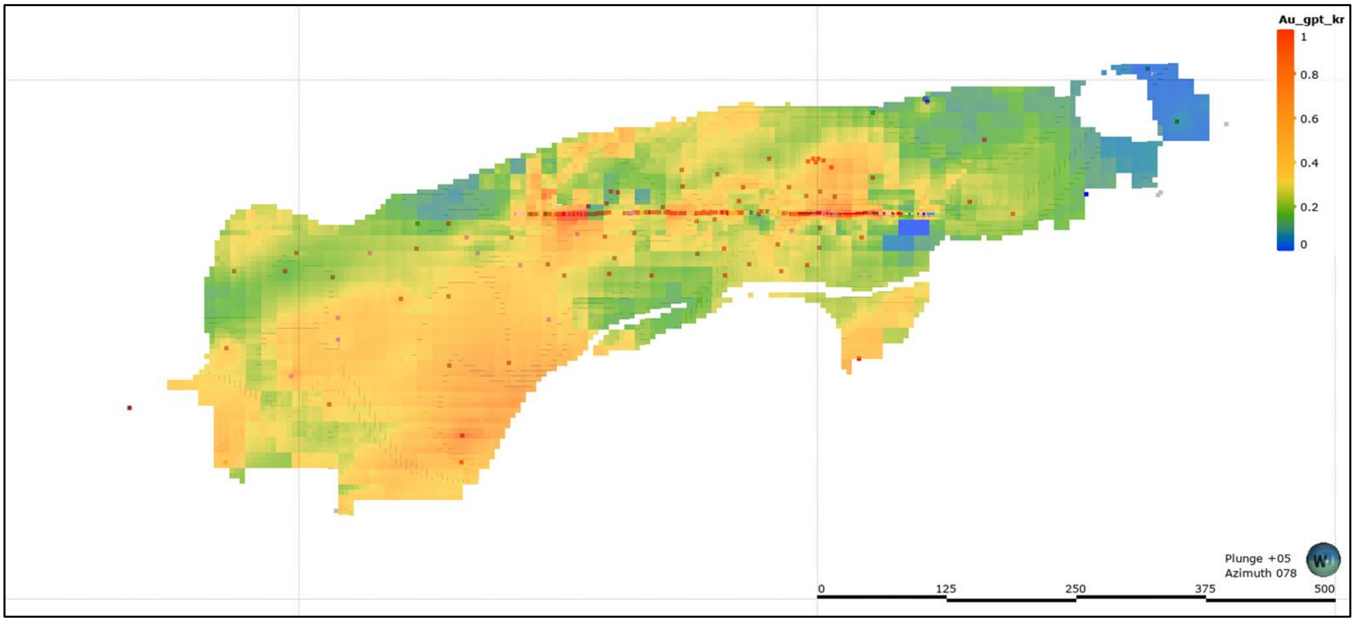

Figure 14-67: Gold Grade Estimates – Gloria Vein (Red Pennant 2021)

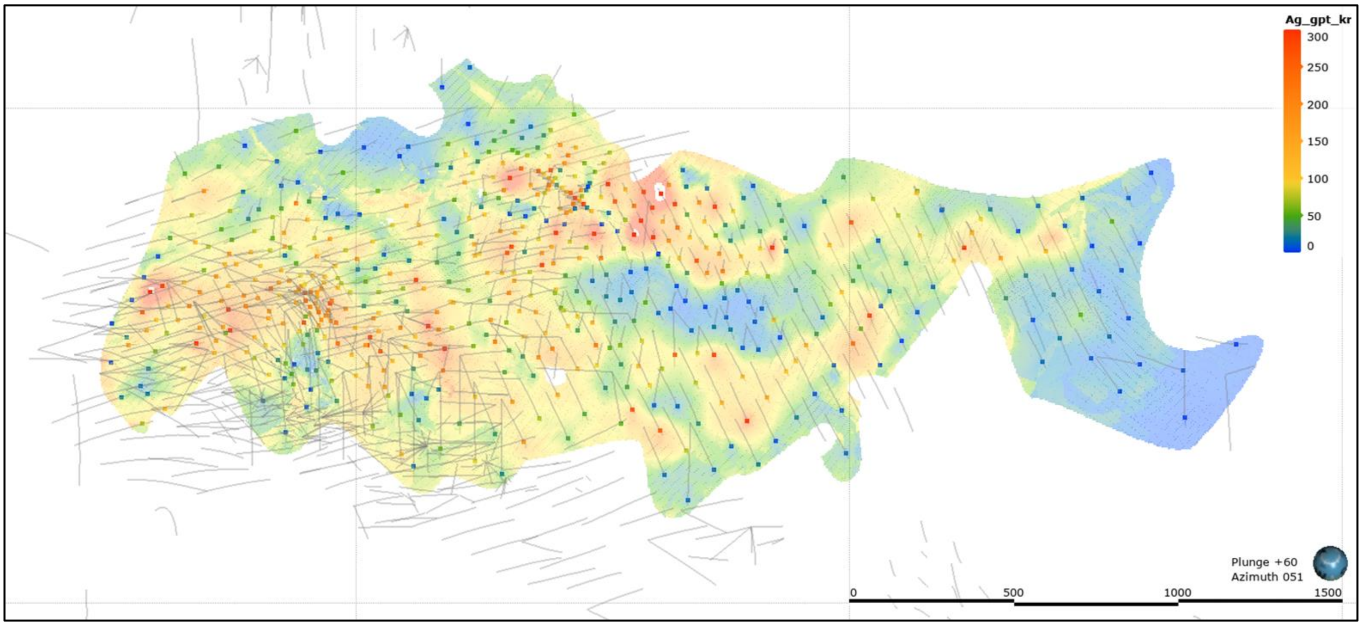

Figure 14-68: Silver Grade Estimates – Martha Vein (Red Pennant 2021)

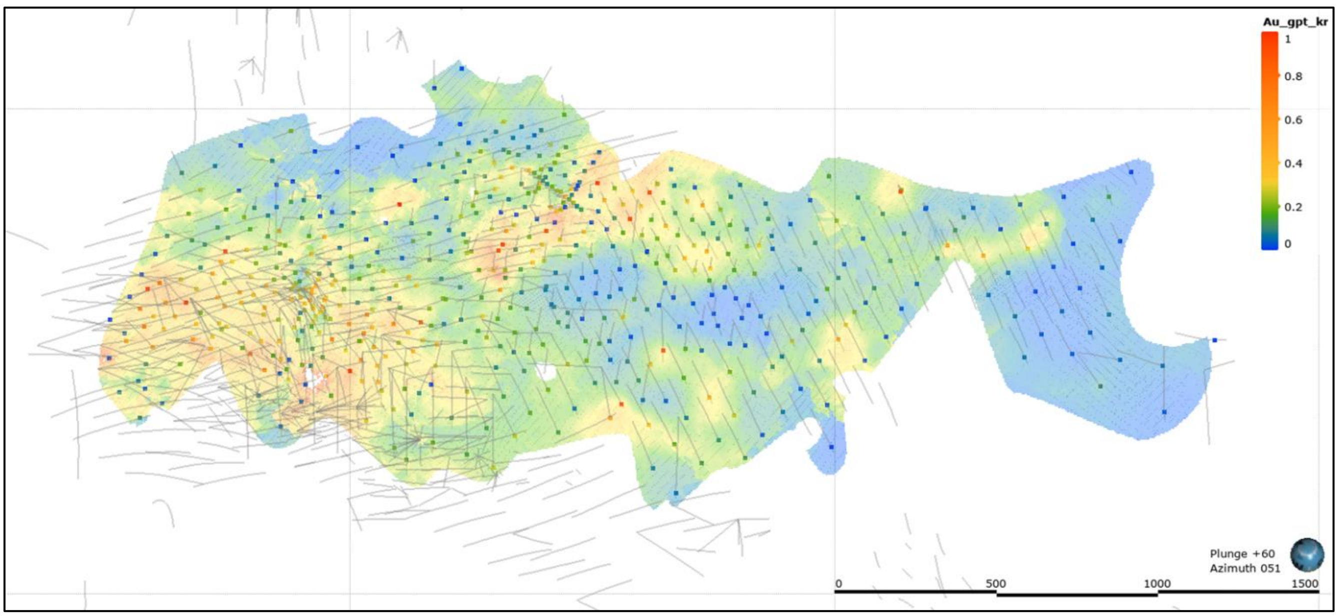

Figure 14-69: Gold Grade Estimates – Martha Vein (Red Pennant 2021)

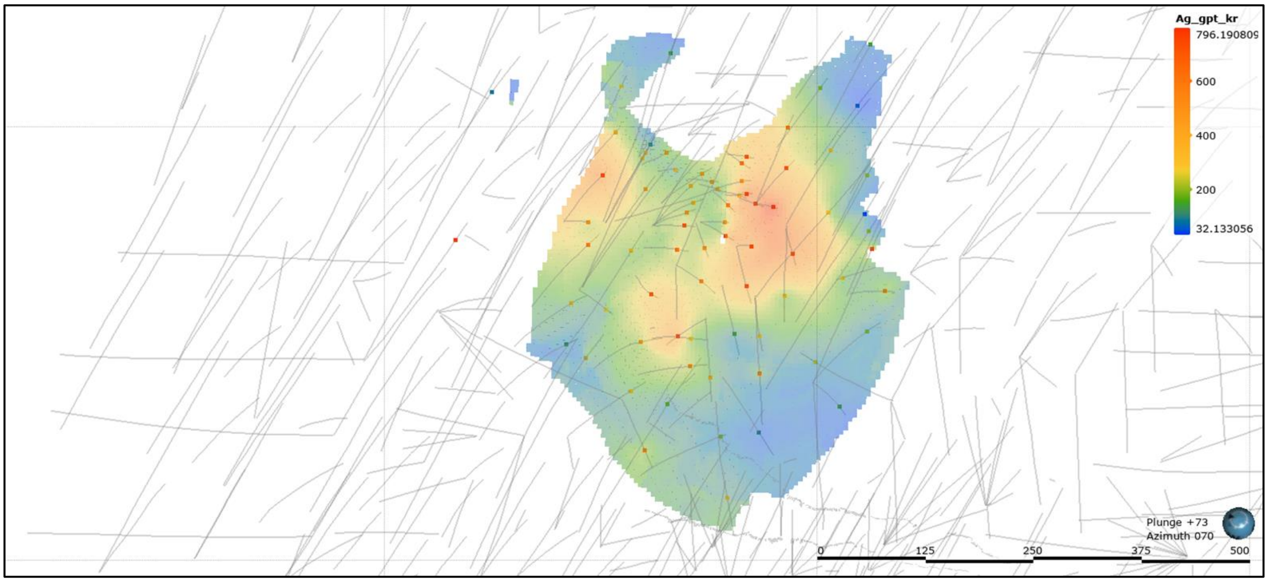

Figure 14-70: Silver Grade Estimates – Martha 2 Vein (Red Pennant 2021)

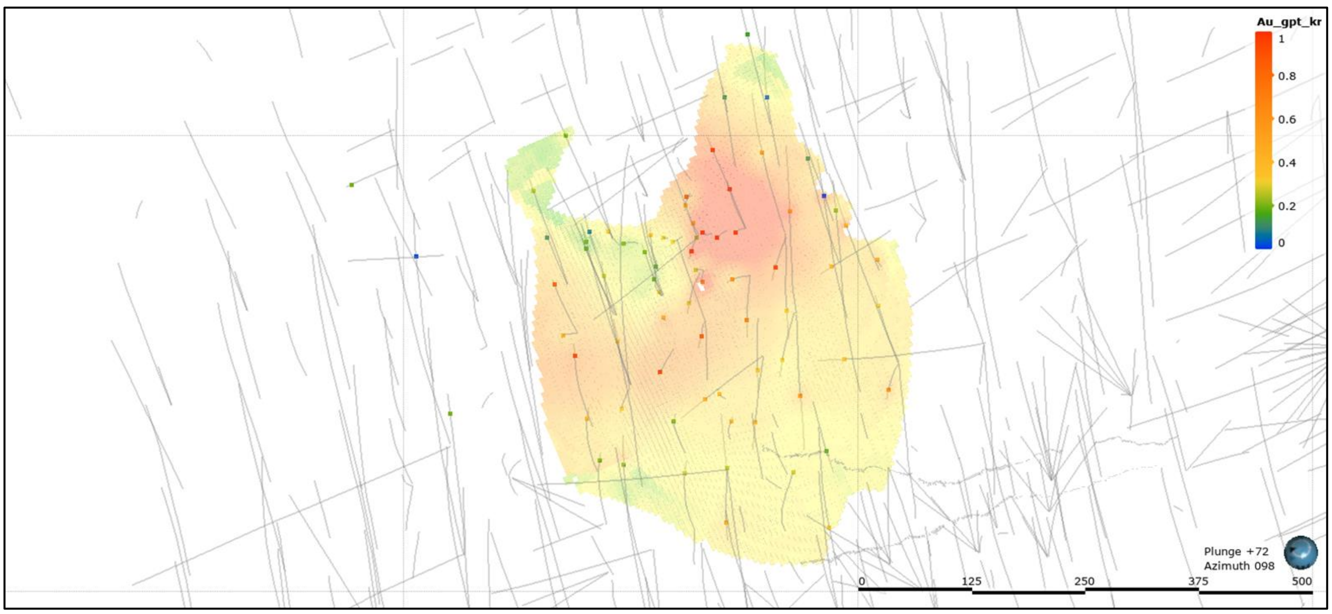

Figure 14-71: Gold Grade Estimates – Martha 2 Vein (Red Pennant 2021)

14.2.13 Classification of Mineral Resources

The Kriging Slope of Regression was generated during the estimation and is a measure of confidence related to distance from data in the context of the spatial uncertainty (corresponding with the variogram models). The Silver Kriging Slope of Regression was used as the main driver of uncertainty.

Measured, Indicated, and Inferred Mineral Resources were classified using the parameters outlined in Table 14-43.

Table 14-43: Confidence Classification Criteria

Categories	Average Distance to Estimation Composites	Slope of Regression
Indicated	< 125 m	> 0.9
Inferred	< 220 m	--

The extents of measured, indicated, and inferred mineral resources within the deposit are illustrated in Figure 14-72 to Figure 14-76, inclusive.

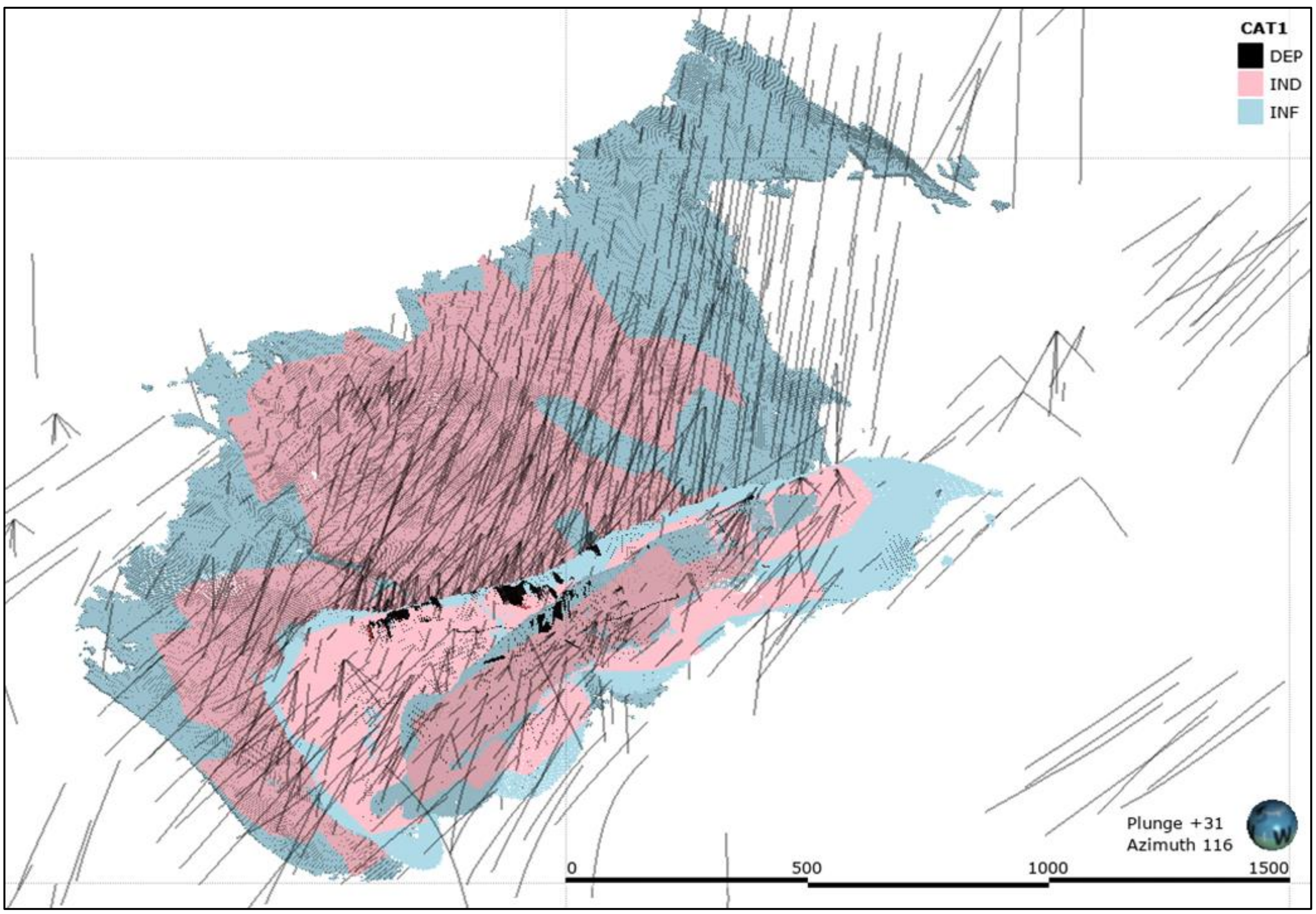

Figure 14-72: View of La Preciosa Mineral Resource Categories (Red Pennant 2021)

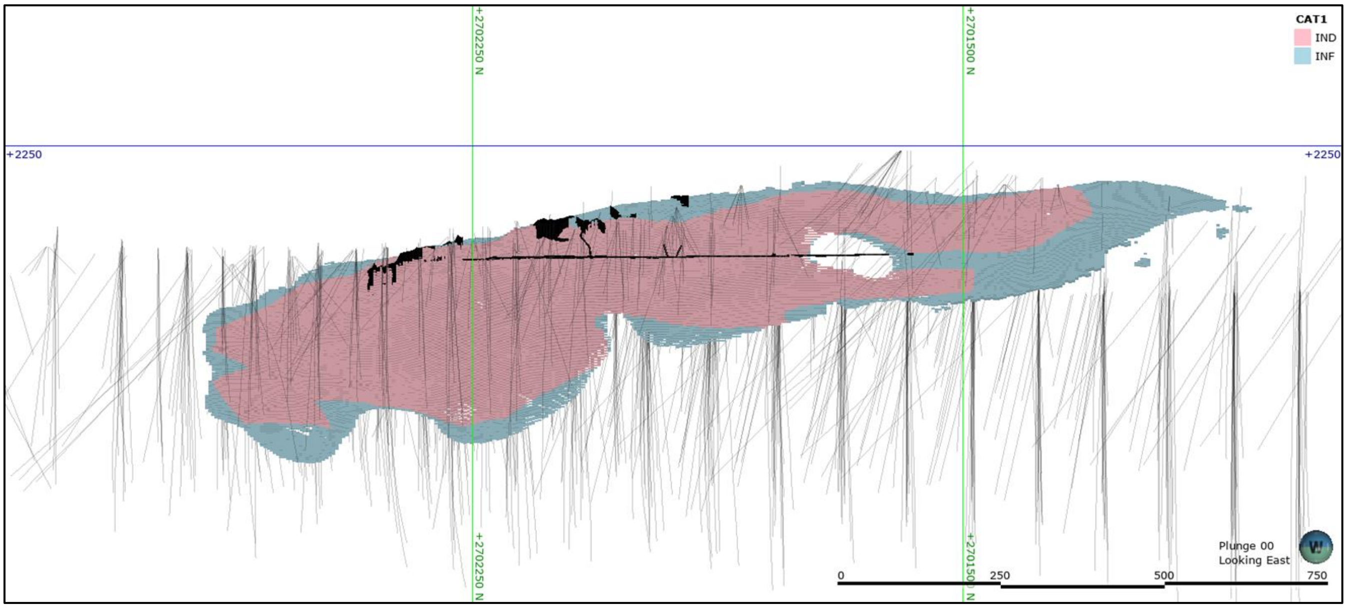

Figure 14-73: Normal View of Categorized Mineral Resource Blocks, Abundancia Vein (Red Pennant 2021)

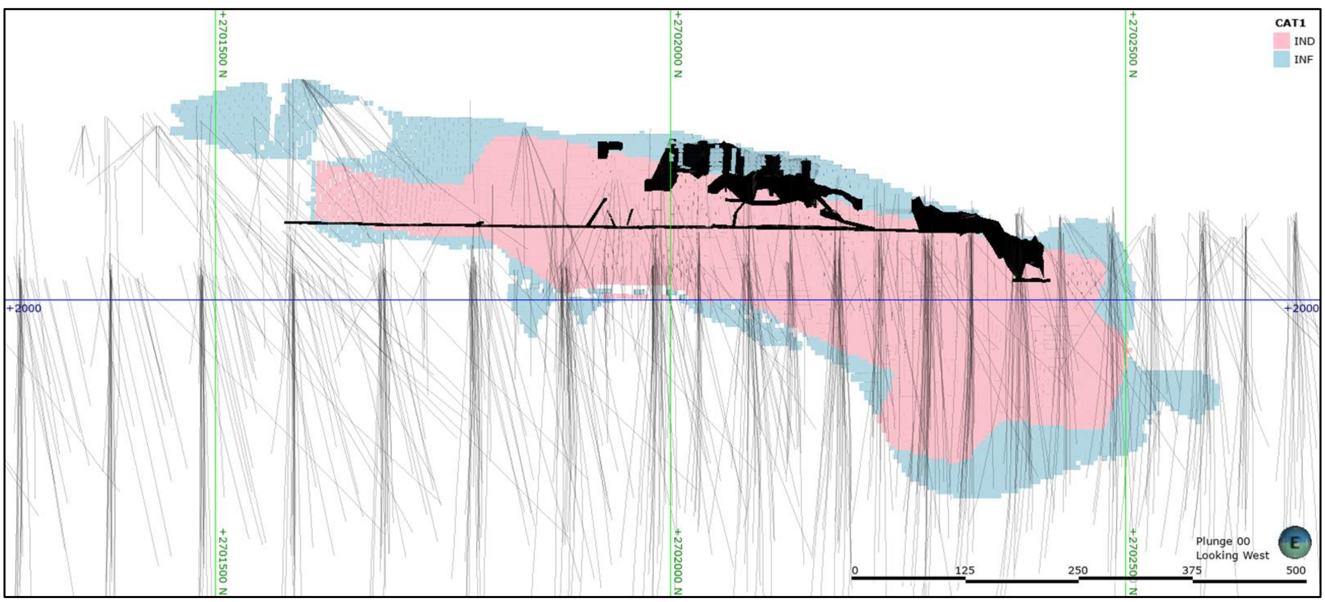

Figure 14-74: Normal View of Categorized Mineral Resource Blocks, Gloria Vein (Red Pennant 2021)

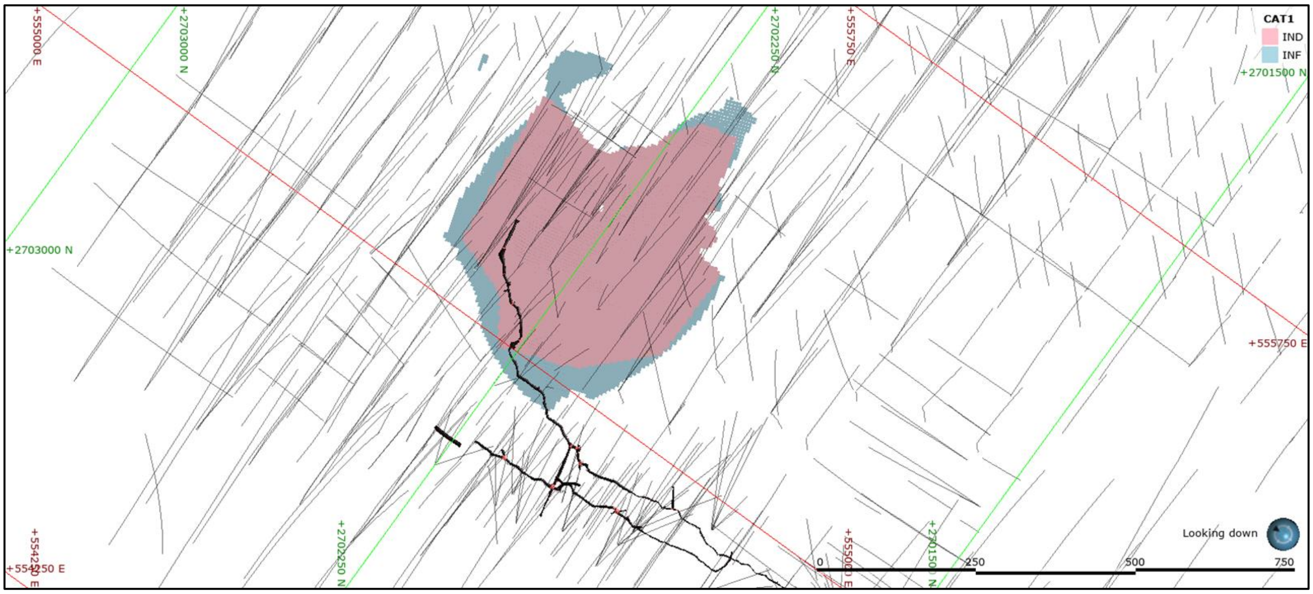

Figure 14-75: Normal View of Categorized Mineral Resource Blocks, Martha 2 Vein (Red Pennant 2021)

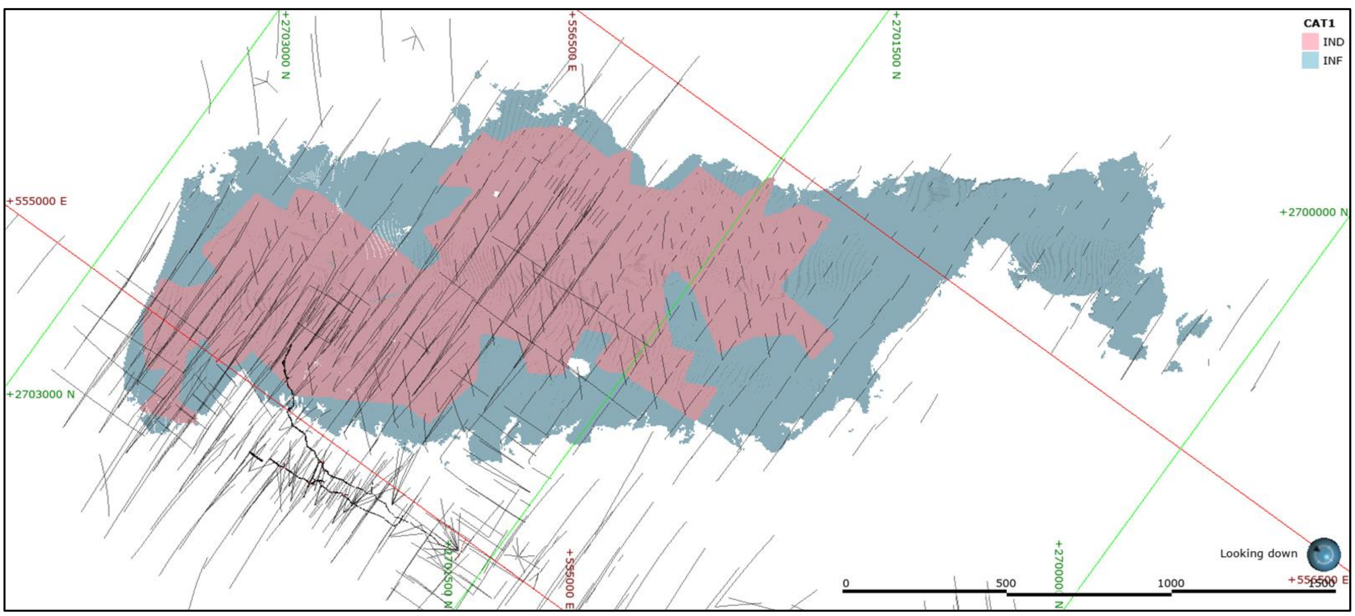

Figure 14-76: Normal View of Categorized Mineral Resource Blocks, Martha Vein (Red Pennant 2021)

14.2.14 Reasonable Prospects of Eventual Economic Extraction

MREs were constrained within the four highest-grade veins and no more 500 m from surface. The cut-off grade applied to the mineral resource is conceptual and based on selective narrow-vein cut-and-fill mining for the steeper veins (e.g., Gloria and Abundancia) and room and pillar for the flat-lying veins (e.g., Martha).

14.2.15 Resource Sensitivity to Cut-off

The tonnage and grade sensitivities for all mineralized vein material over a range of cut-off (AgEQ) values are illustrated in Figure 14-77, for all estimated vein material.

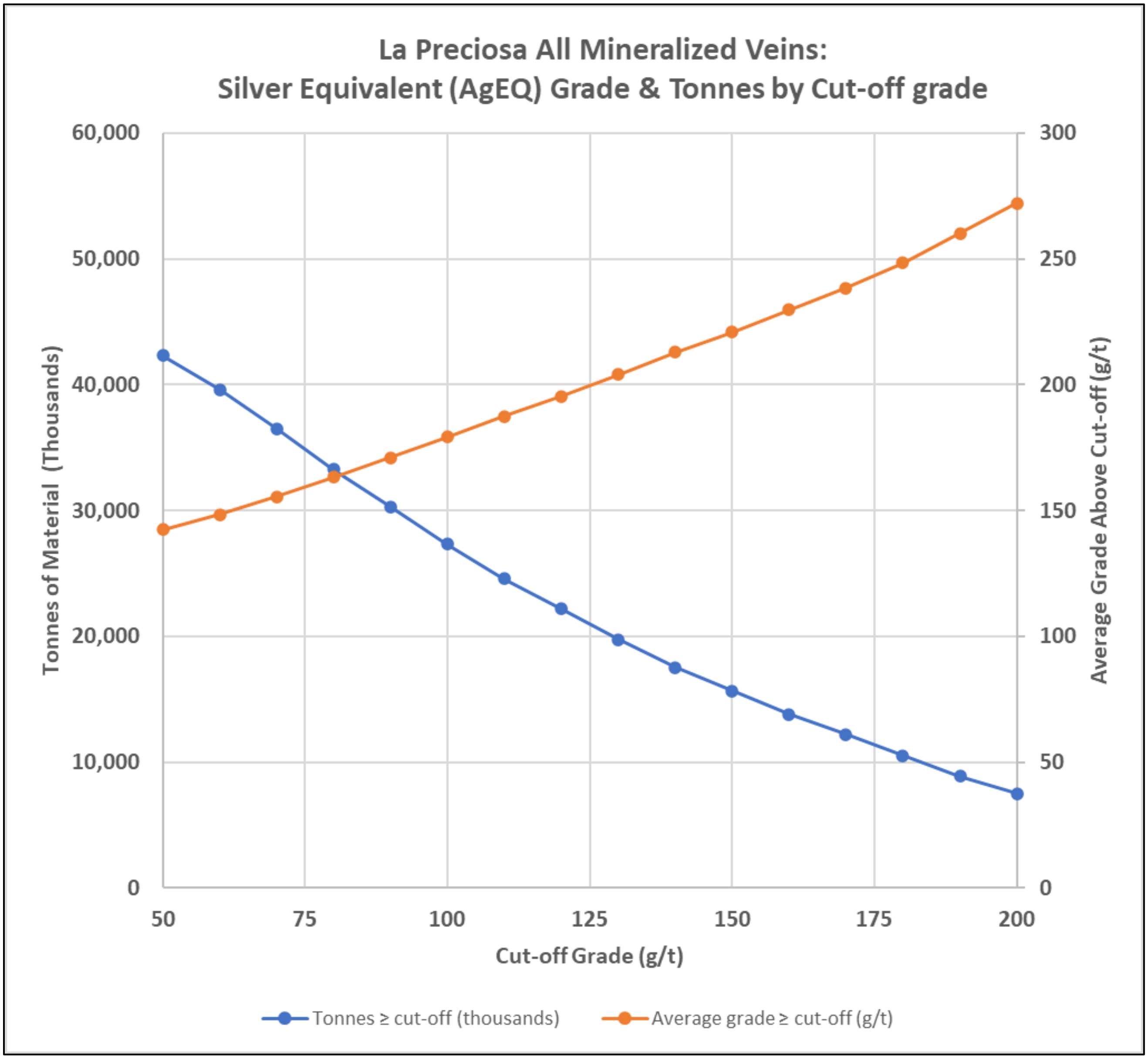

Figure 14-77: Mineralized Vein Material Grade and Tonnage (Red Pennant 2021)

The tonnages and grades at a series of cut-off (AgEQ) values are summarized in Table 14-44.

Table 14-44: Tonnages and Silver Equivalent Grades of Mineralized Vein Material at La Preciosa

AgEQ Cut-off grade g/t	Material Thousand Tonnes	average AgEQ grade g/t	AgEQ Million Tr. Oz
50	42,300	142	194
60	39,600	148	189
70	36,500	156	183
80	33,300	163	175
90	30,300	171	167
100	27,300	179	158
110	24,600	188	148
120	21,800	195	137
130	19,800	204	130
140	17,500	213	120
150	15,700	221	111
160	13,800	230	102
170	12,200	239	94
180	10,500	249	84
190	8,900	260	74
200	7,500	272	66

14.2.16 Mineral Resource Statement

The Mineral Resource was estimated in October 2021 and restricted to continuous regions amenable to underground exploitation using costs derived from the nearby Avino Mine to provide reasonable prospects for eventual extraction. The Mineral Resources are summarized in Table 14-45.

Table 14-45: Mineral Resource Summary

La Preciosa Mineral Resource Summary October 2021									
Cut-off: AgEQ_kr ≥ 120 g/t									
Density: 2.55 g/cm³									
			Average Value				*Material Content*		
Vein	CATEGORY	Mass	AgEQ	Ag	Au	True_Thickness	AgEQ	Ag	Au
		000 Tonnes	g/t	g/t	g/t	m	thousand t. oz	thousand t. oz	thousand t. oz
Abundancia_2	IND	2.6	148	139	0.13	0.7	12.5	11.7	0.0
	INF	0.3	214	203	0.14	0.8	1.9	1.8	0.0
Abundancia_3	IND	75.4	174	161	0.18	2.0	422.0	389.0	0.0
	INF	202.4	161	147	0.18	1.8	1,046.0	957.0	1.0
Abundancia_4	IND	2.2	274	270	0.05	0.7	19.0	19.0	0.0
	INF	202.6	165	162	0.03	1.2	1,073.0	1,058.0	0.2
Abundancia_5	IND	4.0	124	104	0.26	2.1	16.0	13.0	0.0
	INF	1.3	123	103	0.25	1.9	5.0	4.0	0.0
Abundancia_Bajo_2	IND	263.7	166	158	0.10	3.2	1,409.5	1,341.5	0.9
	INF	99.7	197	188	0.12	2.2	631.4	601.3	0.4
Abundancia_Bajo_3	IND	111.4	211	193	0.23	2.3	755.2	692.0	1.0
	INF	52.9	242	222	0.25	2.0	411.7	378.6	0.0
Abundancia_Bajo	IND	167.6	186	172	0.18	3.1	1,003.4	928.2	1.0
	INF	157.8	222	206	0.21	2.9	1,124.5	1,043.7	1.1
Abundancia	IND	2553.5	234	213	0.27	3.8	19,211.9	17,527.2	21.9
	INF	360.4	179	157	0.28	3.1	2,072.3	1,822.0	3.3
Abundancia_Splay_1	IND	116.3	170	149	0.28	2.0	636.6	557.5	1.0
	INF	13.0	152	132	0.27	1.6	63.6	55.0	0.1
Carmen	IND	36.0	159	146	0.17	1.2	184.4	169.0	0.2
	INF	50.4	149	136	0.17	1.2	241.2	220.0	0.3
Gloria_2	IND	72.2	151	130	0.27	1.8	350.7	302.2	0.6
	INF	48.8	190	160	0.39	1.5	297.6	250.9	0.6
Gloria_Alto	IND	1.3	159	145	0.18	1.8	6.7	6.1	0.0
	INF	0.4	145	135	0.14	0.7	1.9	1.8	0.0
Gloria	IND	1093.6	269	243	0.33	4.0	9,455.9	8,555.7	11.7
	INF	178.5	255	229	0.35	3.5	1,463.8	1,311.9	2.0

table continues…

La Preciosa Mineral Resource Summary October 2021									
Cut-off: AgEQ_kr ≥ 120 g/t									
Density: 2.55 g/cm³									
				Average Value				*Material Content*	
Vein	CATEGORY	Mass	AgEQ	Ag	Au	True_Thickness	AgEQ	Ag	Au
		000 Tonnes	g/t	g/t	g/t	m	thousand t. oz	thousand t. oz	thousand t. oz
Martha2	IND	1391.0	291	248	0.56	6.0	13,012.7	11,076.6	25.2
	INF	72.8	185	153	0.42	3.1	433.4	357.2	1.0
Martha_3	IND	40.8	129	118	0.14	3.7	169.5	155.0	0.2
	INF	14.3	131	118	0.17	2.7	60.1	54.2	0.1
Martha_5	INF	34.3	160	124	0.47	2.3	176.5	136.7	0.5
Martha_6	IND	388.7	155	136	0.25	6.4	1,931.2	1,695.8	3.1
	INF	109.4	177	154	0.30	2.9	623.1	541.7	1.1
Martha_7	IND	55.9	157	146	0.14	6.3	281.5	262.7	0.2
	INF	193.9	141	130	0.14	6.5	879.1	813.5	0.9
Martha	IND	10579.1	182	155	0.35	9.7	61,790.3	52,627.5	119.4
	INF	2252.8	162	139	0.29	5.7	11,701.7	10,084.6	21.1
Splay_1	IND	172.8	159	142	0.22	3.5	883.6	789.6	1.2
	INF	95.9	147	134	0.17	3.5	451.6	411.9	0.5
Splay_2	IND	24.8	192	181	0.14	1.7	153.1	144.4	0.1
	INF	6.0	141	132	0.11	2.0	27.1	25.5	0.0
Transversal_Norte	IND	8.9	173	158	0.20	2.5	49.5	45.1	0.1
	INF	67.3	146	130	0.22	1.7	316.5	280.8	0.5
Transversal	IND	279.0	154	142	0.15	3.7	1,382.5	1,277.2	1.4
	INF	181.2	170	158	0.17	2.4	992.2	917.8	1.0
			Average Value				*Material Content*		
All Veins	**CATEGORY**	**Mass**	**AgEQ**	**Ag**	**Au**	**True_Thickness**	**AgEQ_kr**	**Ag_gpt_kr**	**Au_gpt_kr**
		kt	g/t	g/t	g/t	m	thousand t. oz	thousand t. oz	thousand t. oz
	IND	17440.6	202	176	0.34	2.3	113,137.7	98,586.0	189.2
	INF	4396.5	170	151	0.25	1.3	24,095.2	21,329.8	35.5
Differences may occur in totals due to rounding.									
Notes:	Ag price		19	$/oz					
	Au Price		1750	$/oz					
	Ag recovery		90	%					
	Au recovery		75	%					

Notes on Table 14-45:

- kt=thousands of tonnes, Ag=Silver, Au=gold, t. ozs.= troy ounces.
- Mineral Resources are reported using CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) and CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019).
- The QP for the estimate is Mr. Michael F O'Brien, P.Geo., Red Pennant Resources. Geoscience.
- The Mineral Resources have an effective date of October 16, 2023.
- Mineral Resources are reported using a silver equivalent cut-off of 120 g//t, Metal prices are in $US.
- Silver equivalent (AgEQ) has been calculated thus AgEQ = Ag (g/t) + 76.75438596 * Au (g/t)
- Totals may not sum due to rounding.
- Areas of uncertainty that may materially impact the MREs include:
  - Changes to long-term metal price assumptions;
  - Changes in local interpretations of mineralization geometry, fault geometry and continuity of mineralized zones;
  - Changes to metallurgical recovery assumptions;
  - Changes to resource classification approach
  - Variations in geotechnical, hydrogeological, and mining assumptions;
  - Changes to environmental, permitting, and social license assumptions.

14.2.16.1 QP Comments

The QP believes that the Mineral Resources have been estimated using good industry practice and conform to the 2014 CIM Definition Standards. Mineral Resources are constrained by reasonable open pit mining assumptions.

There are no other known environmental, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

15.0 MINERAL RESERVES ESTIMATE

15.1 Introduction

The Mineral Reserves were estimated using both oxide and sulphide tailings and are based on Measured and Indicated Resources presented in Chapter 14, and an updated mine design presented in Chapter 16.

The following inputs and constraints were utilized for pit optimization and are further defined in the following sections:

- Resource model with associated assay grades and densities for the Avino Oxide Tailings Project (Section 14);
- Topographic surface;
- Bedrock surface;
- Metallurgical Recoveries;
- Pit geotechnical slope parameters;
- Local infrastructure constraints, including the Carr. A Panuco Road bordering the West limits of the oxide tailings deposit;
- Commodities prices for pit optimization;
- Operating cost estimates including mining, milling, and other costs;
- Dilution and mining losses;
- Processing rate of 821,250 kt/year.

15.2 Reserve Estimation Considerations

15.2.1 Mining Area Constraints

The Avino Oxide Tailings Project includes both oxide tailings and sulphide tailings, separated by a prominent bench. As described in Section 14, the oxide tailings are within the units “Oxidos_Antiguos” and “Oxidos_Recientes” overlaid by Sulphide tailings within the units “Sulf_1” and “Sulf_2”.

The mining extents used for reserve estimation include the original bedrock surface upon which tailings have been deposited. The layered domains of tailings material are shown in Figure 15-1 below. The economic tailings are overlaid with overburden material (uneconomical sulphide tailings) that will need to be mined for access to economic material. Approximately 3.7 million tonnes of overburden lie within domain “Sulphide 2” as seen in the cross section of Figure 15-2 below.

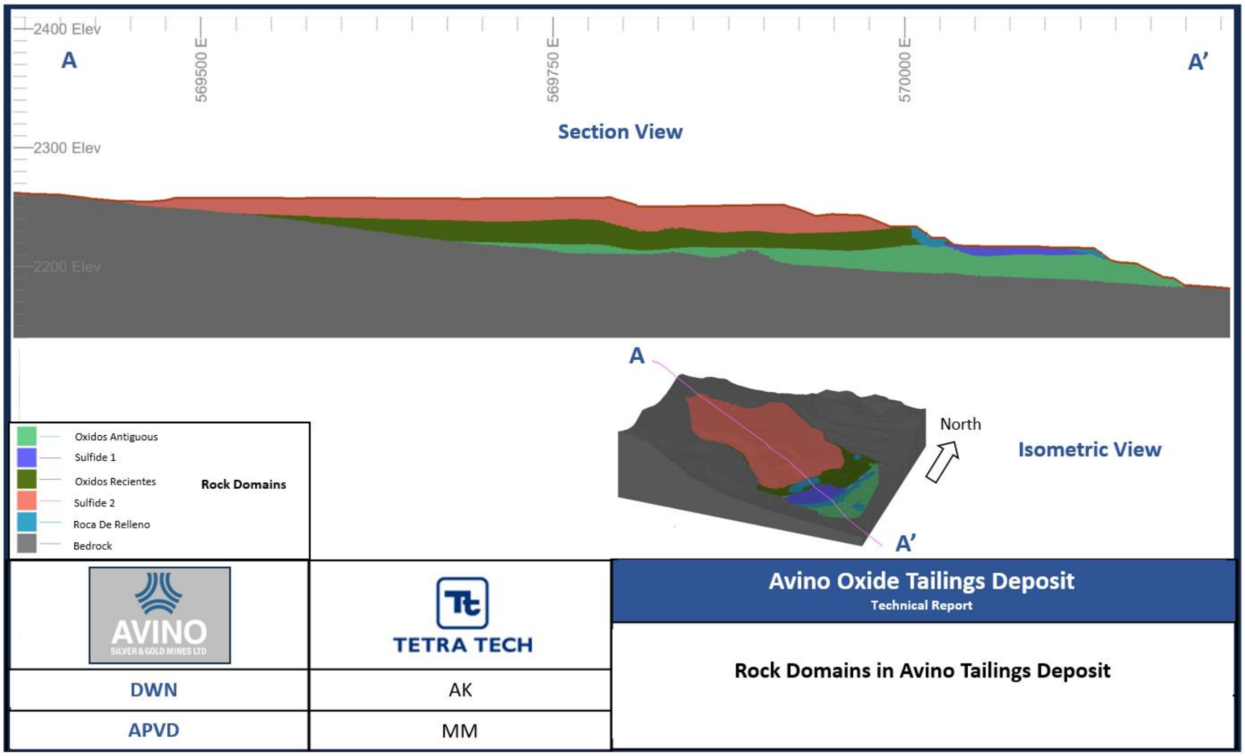

Figure 15-1: Cross Section AA' Viewing Rock Domains

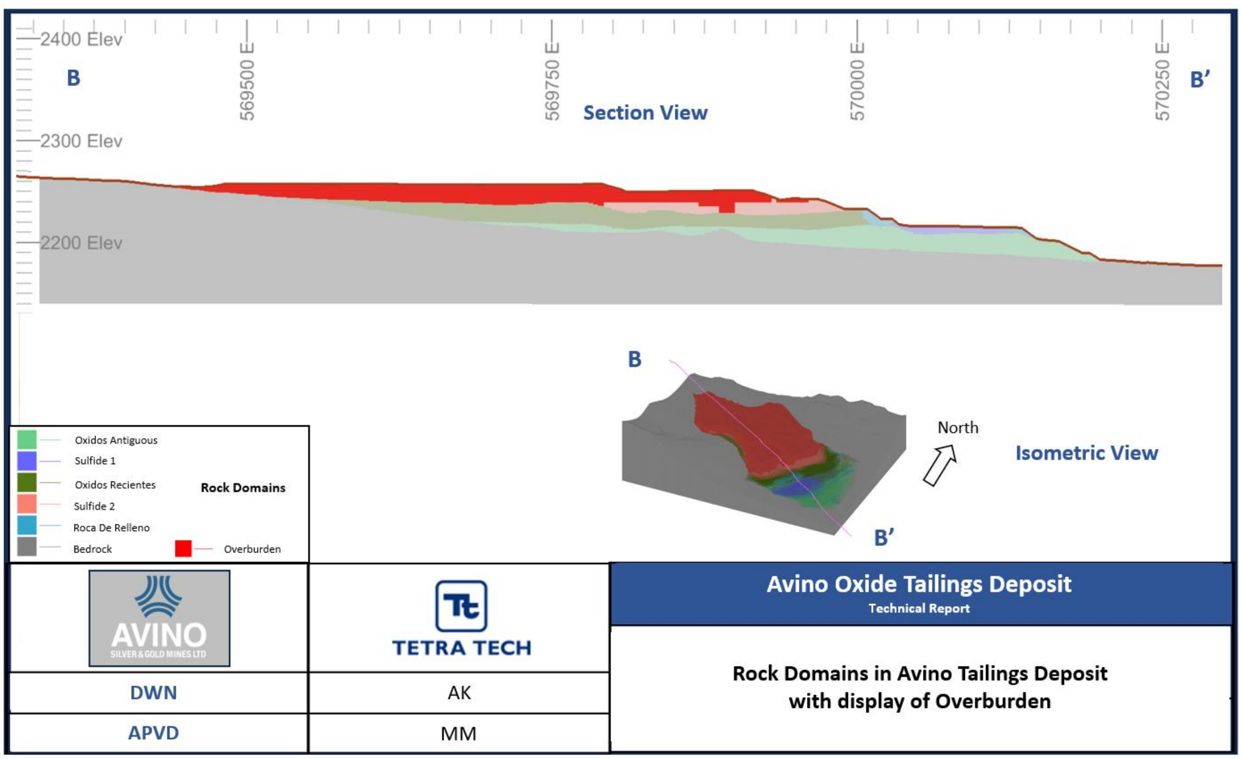

Figure 15-2: Cross Section BB' Viewing Overburden Areas

The mining benches were designed to incorporate the bedrock surface as the pit floor to maximize reserve capture while honoring geotechnical bench constraints. Other infrastructure that was included in the mine design and schedule included existing mill infrastructure and roads on the northeast of the deposit as shown in Figure 15-3.

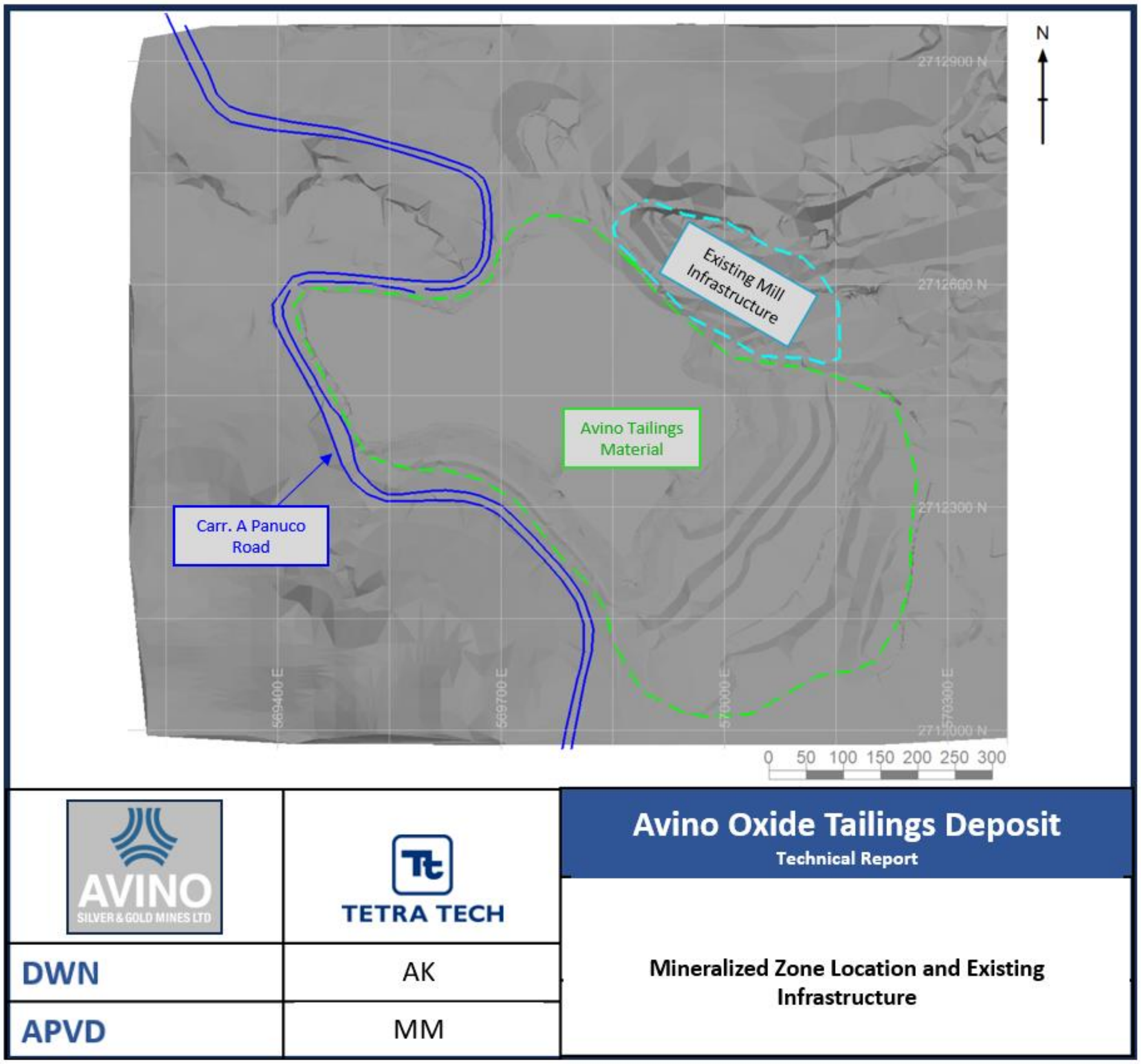

Figure 15-1: Location of Mineralized Material and Existing Infrastructure

15.3 Reserve Estimation Parameters

15.3.1 Geotechnical Analysis

A simplified geotechnical assessment was carried out upon reviewing and adopting geotechnical parameters and descriptions for select material types from the geotechnical study by GEIC Ingenieria Civil y Ciencias de la Tierra (GEIC 2018). This approach modelled bench and pit slopes for each of the 3 most prevalent tailings materials independently, thereby assessing the stable bench and pit geometries that each tailings material could achieve. An example of this is shown in Figure 15-4.

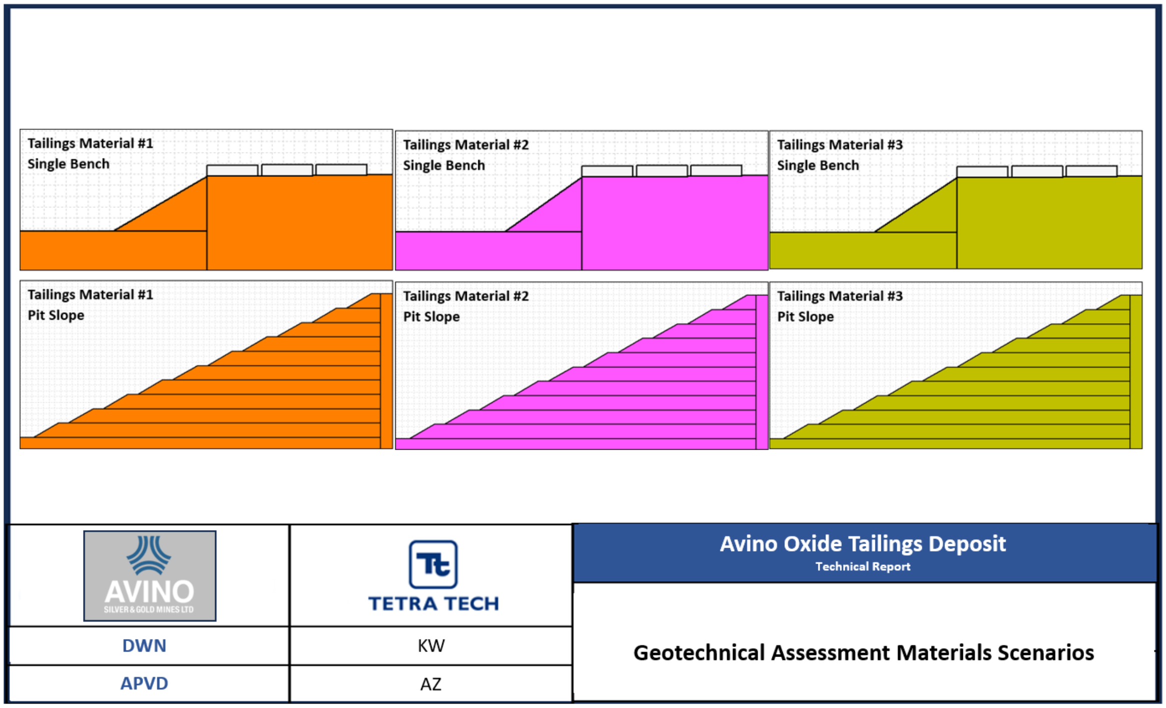

Figure 15-2: Geotechnical Assessment Materials Scenarios

15.3.1.1 Geotechnical Parameters

The geotechnical parameters and material descriptions for the 3 tailings materials are summarized in Table 15-1.

Table 15-1: Geotechnical Material Parameters and Descriptions

Material	Unit Weight (γ – kN/m^3)	Cohesion (c – kPa)	Friction Angle (ϕ – degrees)
Tailings Material (“Jal”) 1 – Very Loose to Loose Compactness	14.37	7	23
Tailings Material (“Jal”) 2 –Loose to Medium Compactness	14.58	4	31
Tailings Material (“Jal”) 3 – Medium Compactness	15.2	2*	35

Note: * - GEIC (2018) reports a cohesion of 0 kPa. This was increased to 2 kPa by Tetra Tech since the stability analysis used probabilistic parameters, and therefore modelled cohesion values ranging from 0 kPa to 12 kPa. This is discussed in Section 15.3.1.2.

15.3.1.2 Bench and Pit Design

Individual benches were modeled and designed with the requirement of achieving a minimum Factor of Safety (FoS) of 1.0 and a Probability of Failure (PoF) below 10%. Multiple scenarios were modeled which considered:

- Loading trucks from the bottom of the bench.
  - Resulting in no crest surcharge.
- Loading trucks from the top of the bench with an excavator weight of 50.91kN/m^3.
  - 3 crest surcharge scenarios were assessed (Figure 15-3): excavator 0 m back from crest, excavator 5 m back from crest, and excavator 10 m back from crest.
  - No surcharge was included for the haul trucks.

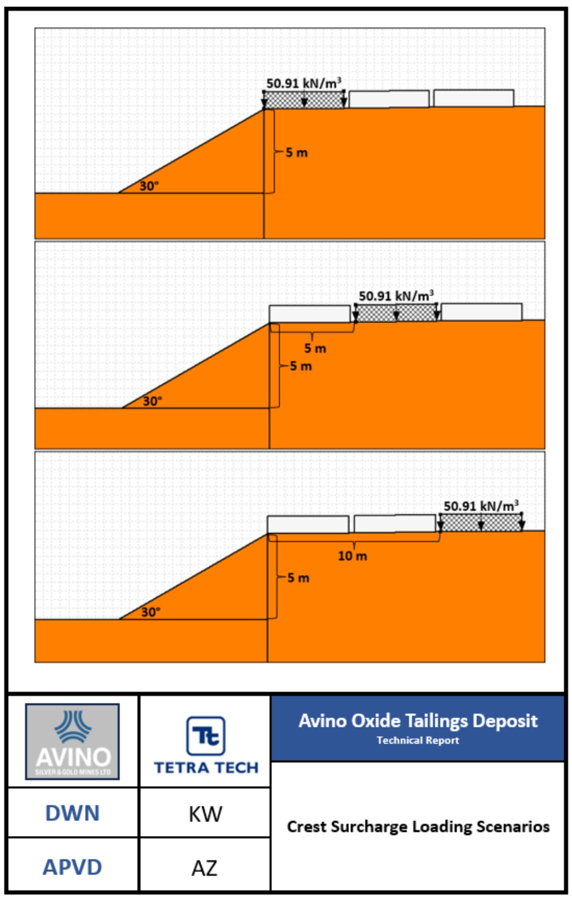

Figure 15-3: Crest Surcharge Loading Scenarios

The required FoS was achieved under drained condition of the materials. For all scenarios, a bench height of 5 m was considered to accommodate mining equipment operation.

The tailings will be mined in single benches by an excavator loading haul trucks on 5 benches and a bench face angle up to 30 degrees. It also allows the excavator to load from the top or bottom of the bench to suit the evolving mining work area.

From this single bench stability analysis, it was also determined that the required mining bench width for the final pit slope is 3.5 m. The resulting pit slope geometry is shown in Figure 15-4. This can be further optimized in future phases of study when more information becomes available. The 3.5 m bench width extends beyond the potential failure surface extents of the tailings, therefore avoiding a multi-bench failure.

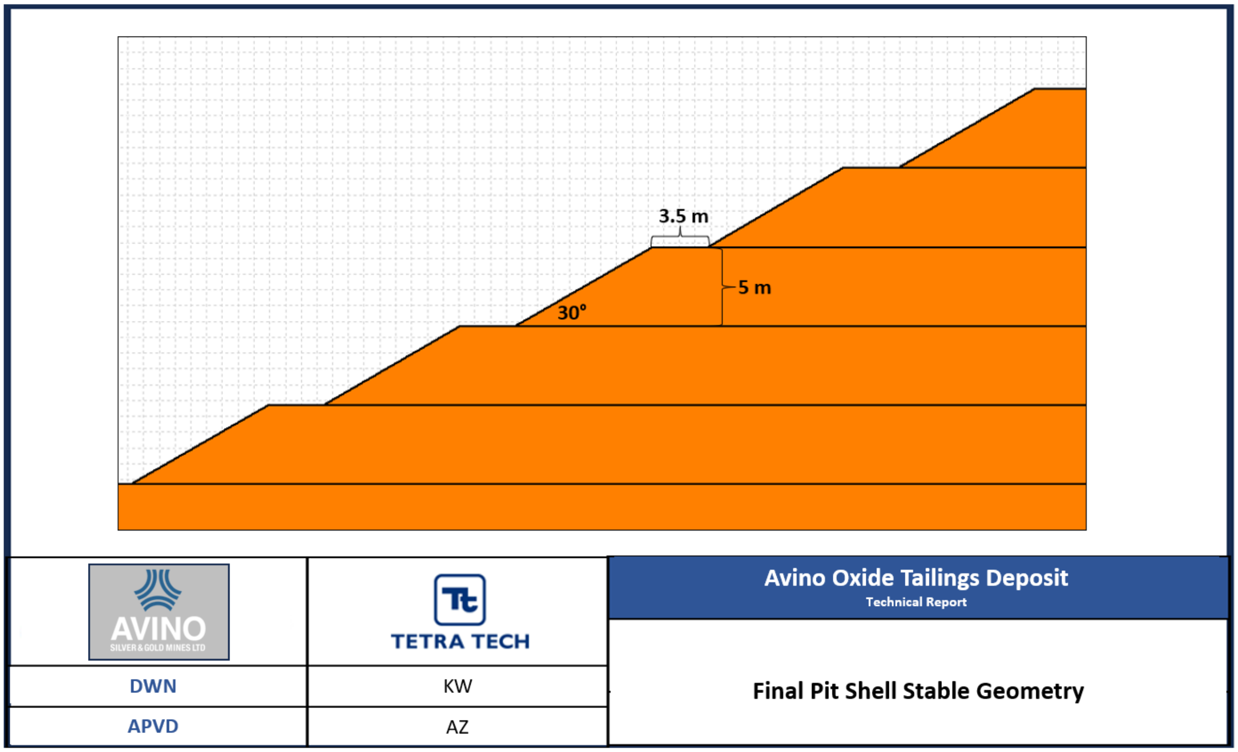

Figure 15-4: Final Pit Shell Stable Geometry

15.4 NSR Model

A Net Smelter Return (NSR) model was created for the Mineral Reserves and used the parameters summarized in Table 15-2 below.

NSR models determine the worth of a tonne of ore that is mined, processed, concentrated, and delivered to a smelter or process plant. The NSR represents the anticipated revenue from every tonne of mill feed, considering factors such as mill recoveries and deductions for transportation to the smelter, as well as treatment and refining costs. For the Avino Oxide Tailings Project, the NSR and profit calculation can be expressed as the following:

$$\boldsymbol{NSR} = (\mathbf{Metal}\ \boldsymbol{Price} * \boldsymbol{Payablity} - \boldsymbol{Shipping\ Cost}) * \ \mathbf{Mass} * \mathbf{Ore\ Grade} * \mathbf{Process\ Recovery} * (\mathbf{1} - \mathbf{Insurance})$$

$$\boldsymbol{Profit} = \mathbf{NSR} - \mathbf{Mill\ Cost} - \mathbf{Processing\ Cost} - \mathbf{G\&A}$$

Tetra Tech used the NSR model to estimate the value of blocks in the tailings deposit and determine a cut-off value for reserve calculations.

Table 15-2: NSR Model Parameters

Parameter	Unit	Value
Gold Price	$ USD/tr. Oz	1,850
Silver Price	$ USD/tr. Oz	22.00
Mining Cost	$ USD/t mined	1.00
Mining Dilution	%	1
Mining Recovery	%	99
Processing Cost	$ USD/t	18.00
G&A	$ USD/t	3.00
Process Recovery - Gold		
Ancient Oxides	%	85
Recent Oxides	%	80
Sulphides	%	80
Process Recovery - Silver		
Ancient Oxides	%	86
Recent Oxides	%	80
Sulphides	%	73
Mill throughput	t/y	821,250
Selling Cost - Gold	$ USD/tr. Oz	5.00
Selling Cost - Silver	$ USD/tr. Oz	0.50
Shipping Cost	$ USD/tr. Oz	0.20
Insurance	%	0.13
Percent Payable - Gold	%	99
Percent Payable - Silver	%	95
NPV Discount Rate	%	5

Due to the multi-element nature of the tailing deposit, the cut-off is expressed as an NSR value rather than the grade of silver or silver equivalent. The profit and loss can be calculated from revenue that can be assigned to mining activities, processing, and selling the final product. As part of cut-off calculation, the NSR cutoff will be expressed as the value of profit is greater than zero. Based on Table 15-2, the following equation defines the cut-off calculations:

$$\mathbf{NSR\ Cutoff = Processing\ Cost + G\&A = 18.00 + 3.00 = \$21.00}/\boldsymbol{t\ processed}$$

15.5 Ultimate Pit Design and Pushbacks

To help identify the most profitable reserve and mining pushbacks, Tetra Tech used Datamine's™ NPVS software to design the ultimate pit. Datamine™ uses a calculation called the LG Algorithm to determine the economic mining limit. Furthermore, the ultimate pit design incorporated key factors such as pit access, and geotechnical slope considerations including bench height, face angle, berm width, and ramp design parameters.

The Datamine's™ NPVS optimization process involves the creation of nested pit shells by applying scaling revenue factors (RF). Each block's anticipated base case revenue is then multiplied by these factors and processed using the LG Algorithm. As a result, a sequence of pit shells is generated, reflecting the influence of the revenue factors. These nested pit shells play a key role in determining the mining sequence for the ore body and establishing the ultimate pit limit. Tetra Tech determined approximately 1.1 million tonnes of uneconomical tailings deposit will be left in place, located north and west perimeter of the tailings deposit. (Figure 15-9)

Based on the nested pit shells, five cashflow-positive pushbacks were designed to allow for operational flexibility while stripping the overburden and targeting high grade material (Figure 15-8). Pushback 1 targets the highest profit per tonne mined with the lowest strip ratio, located at the east end of the deposit. All remaining pushbacks are designed in order of profitability and strip ratio and progress the mining of the deposit from East to West. Figure 15-7 visualizes the expansion of nested pit shells beginning in the northeast end of the deposit based on incrementally increasing revenue factors. The pushback design summary is in Table 15-3. Furthermore, for each pushback, the haulage ramps were designed to be consistent with the equipment fleet currently used at the mine site. Double-lane haul roads are designed at a width of 10.0 m and a gradient of 12% for all ramps within the pit. To better facilitate higher haul truck productivity, some of the mining benches were designed at 10m wide (instead of 3.5m) so that they can tie in with the existing access road to reduce cycle time.

Table 15-3: Pushback Design Summary

Pushbacks	Ore (t)	Waste (Mt)	Strip Ratio	Profit per tonne (USD $/t)
1	673,429	135,602	0.2	218
2	1,101,834	514,445	0.5	169
3	2,436,857	1,344,012	0.6	145
4	1,995,745	1,490,784	0.7	116
5	500,728	216,131	0.4	158
Total	**6,708,593**	**3,700,974**	**0.6**	**146**

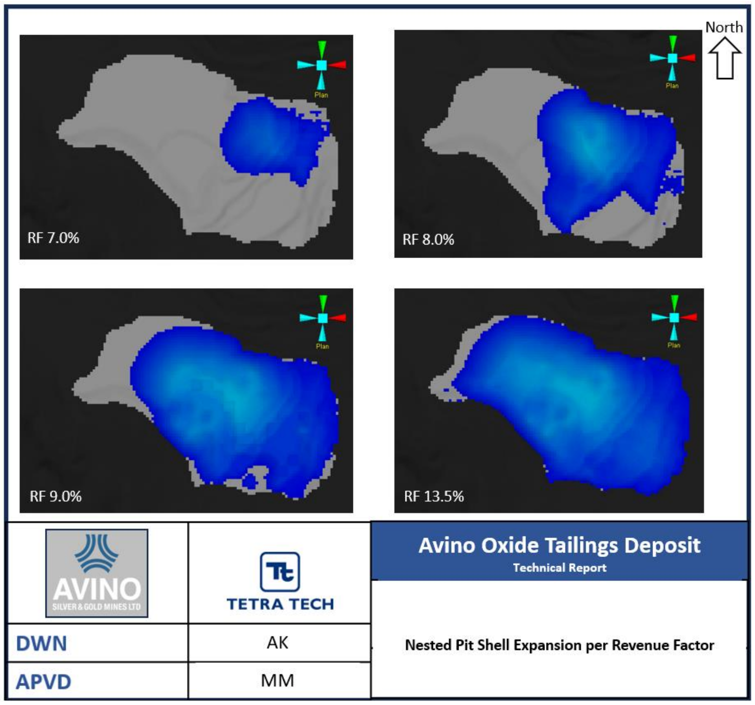

Figure 15-5: Expansion of Nested Pit Shells with Revenue Factors

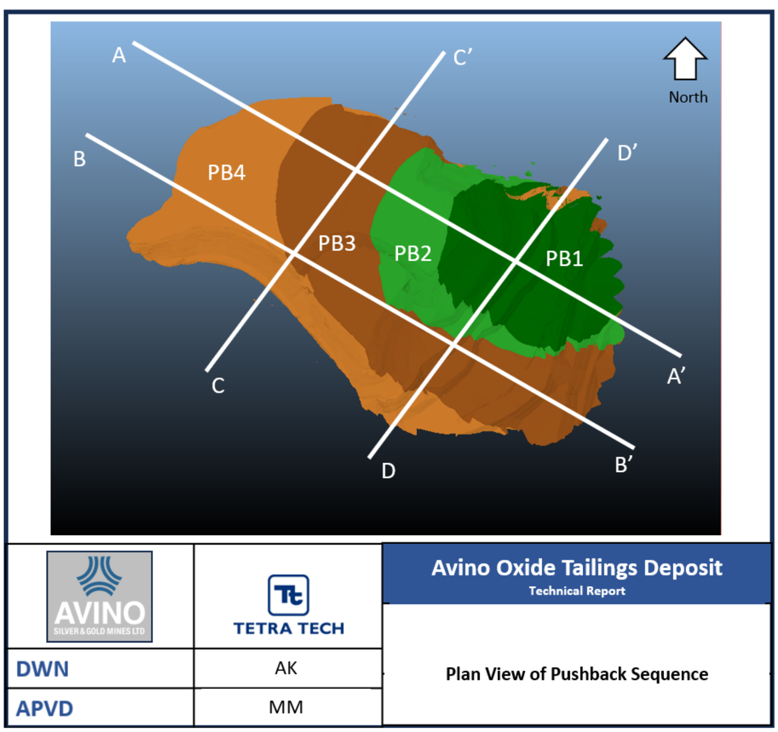

Figure 15-6: Pushback Sequence

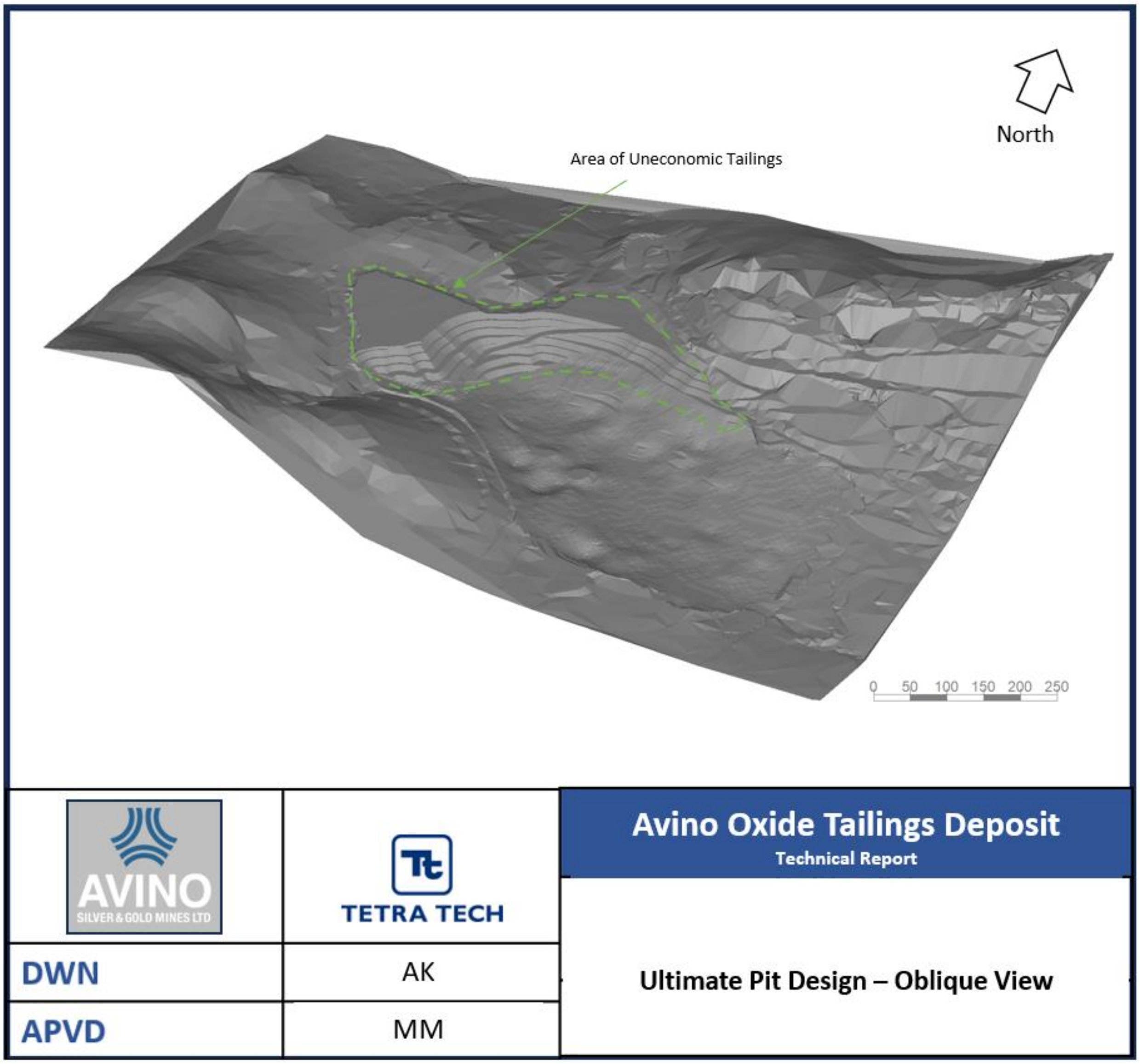

Figure 15-7: Ultimate Pit Design - Oblique View

Cross-sections of each pushback are provided in Figure 15-8 through Figure 15-16.

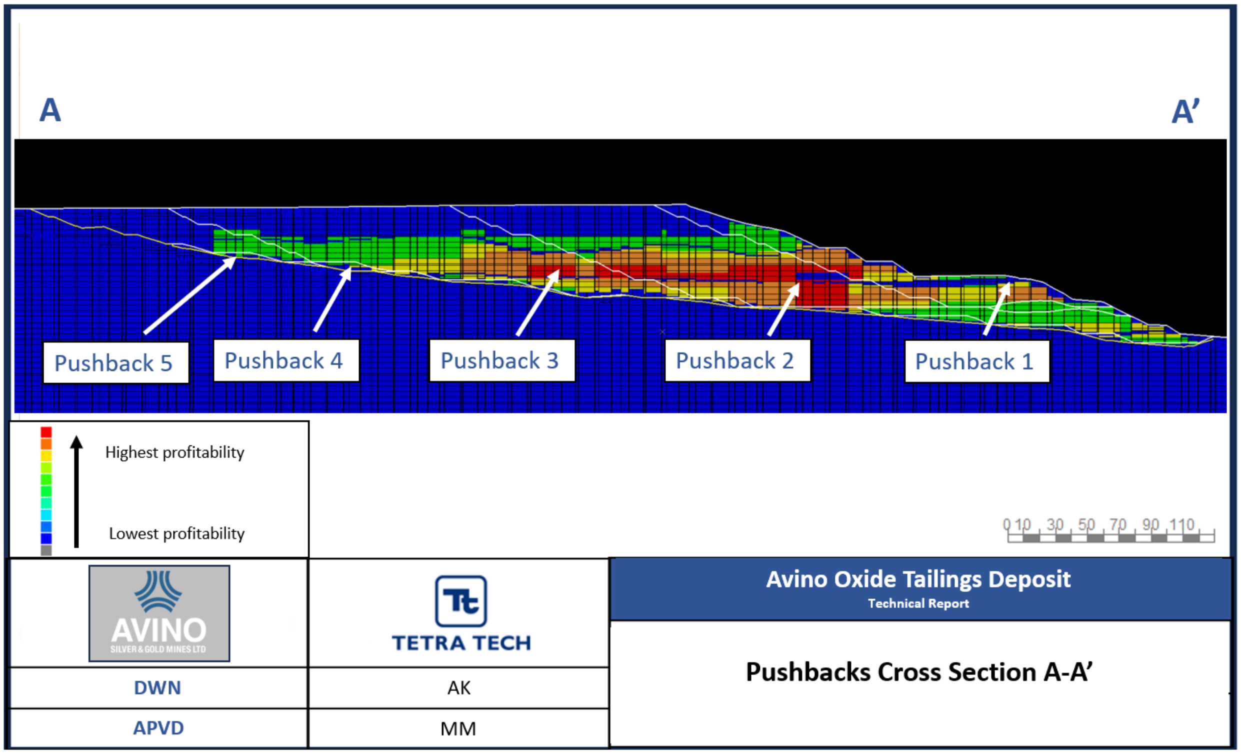

Figure 15-8: Cross-Section AA'

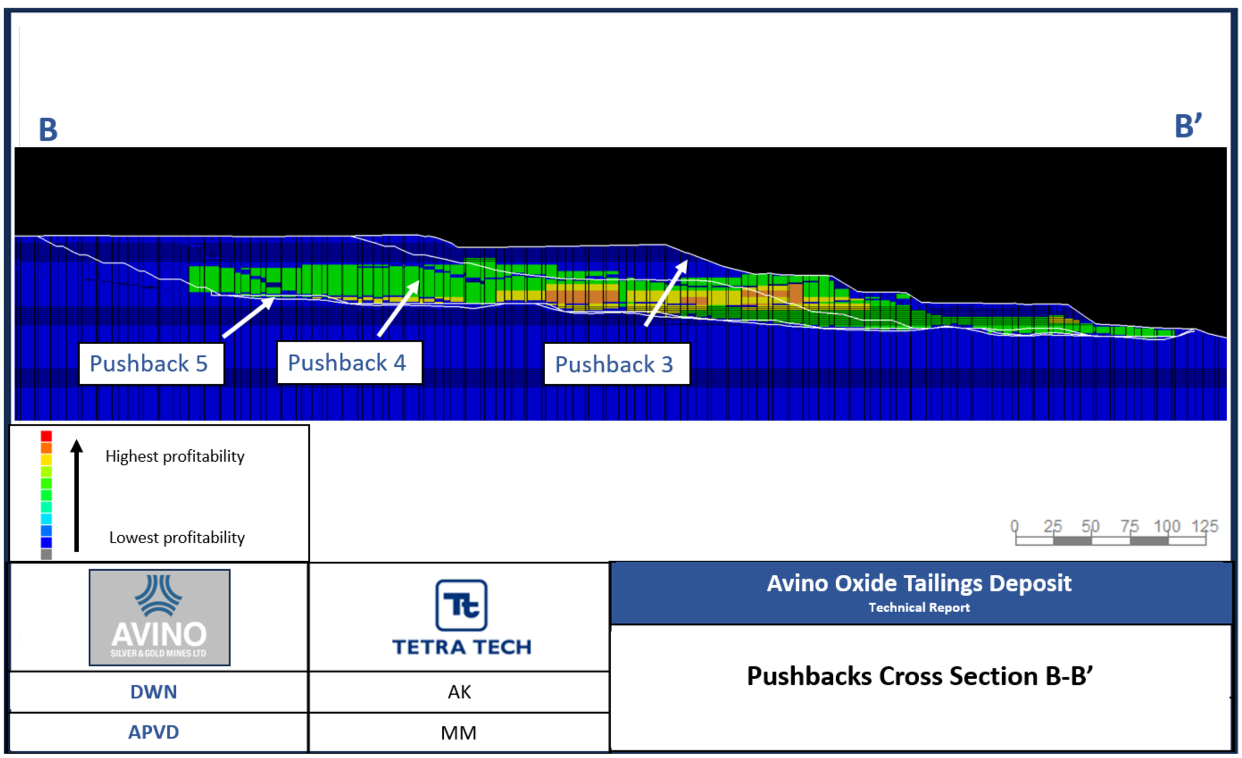

Figure 15-9: Cross-Section BB'

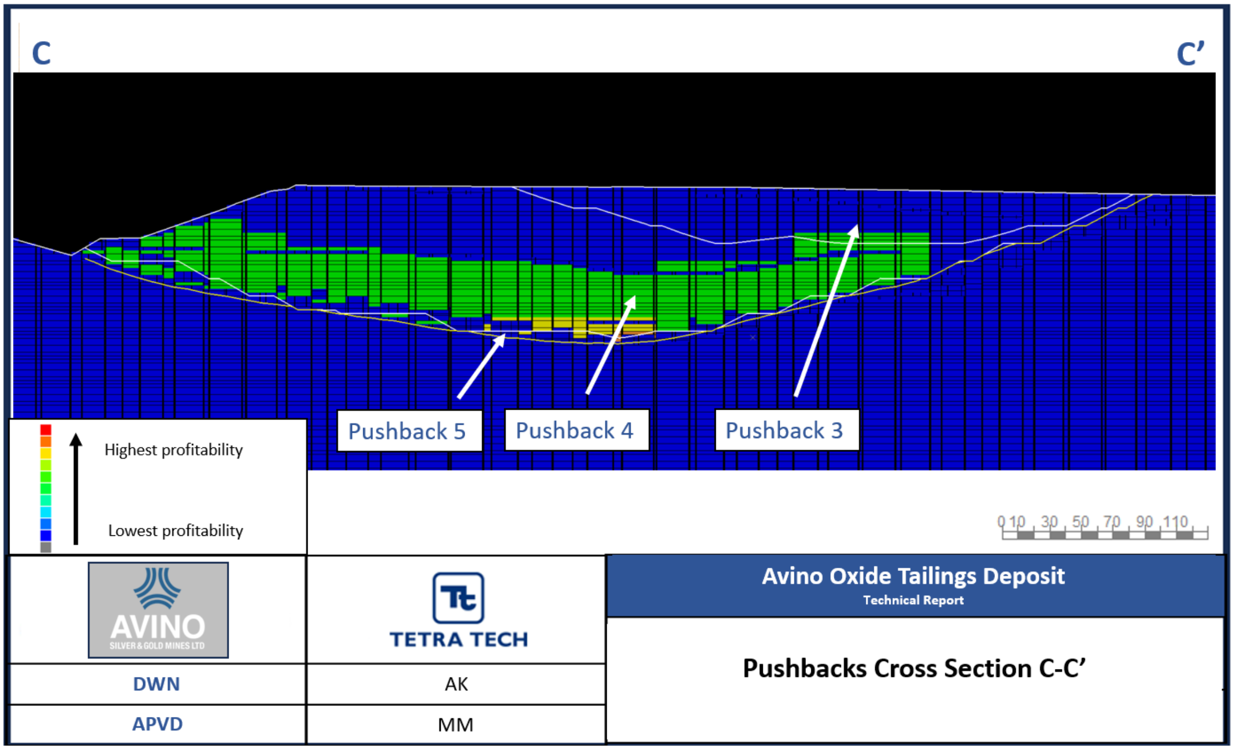

Figure 15-10: Cross-Section CC'

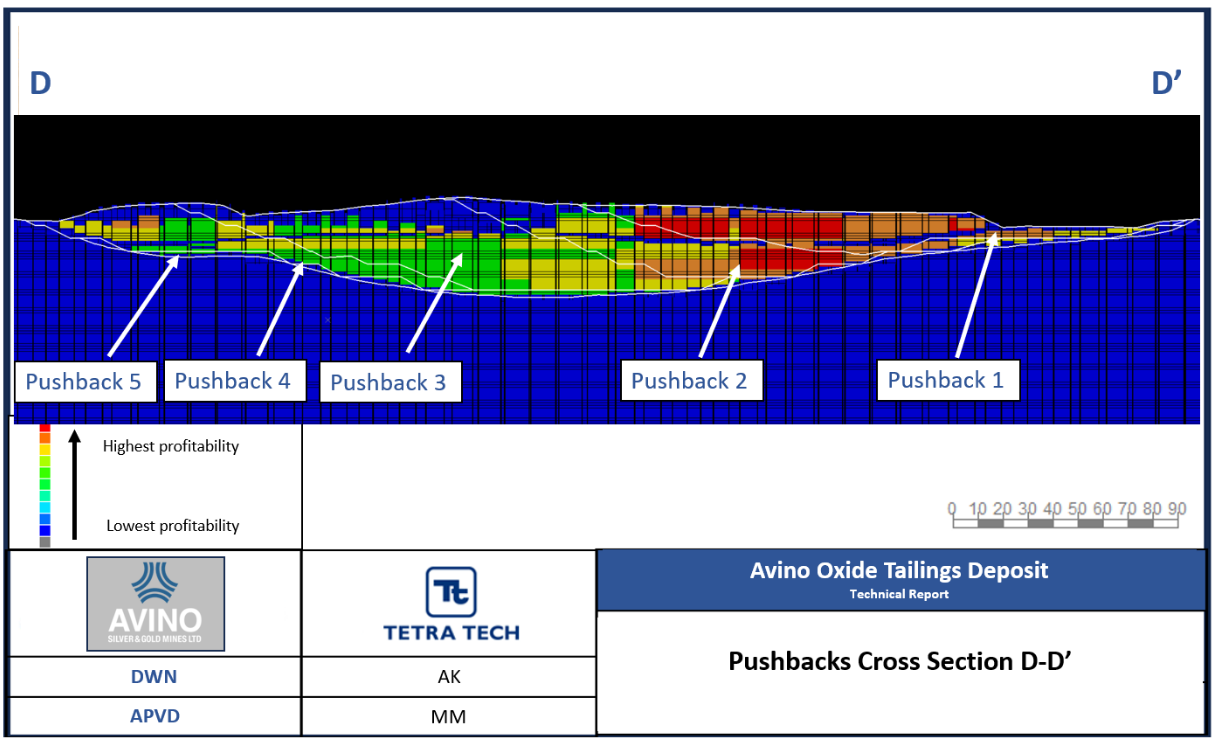

Figure 15-11: Cross Section DD'

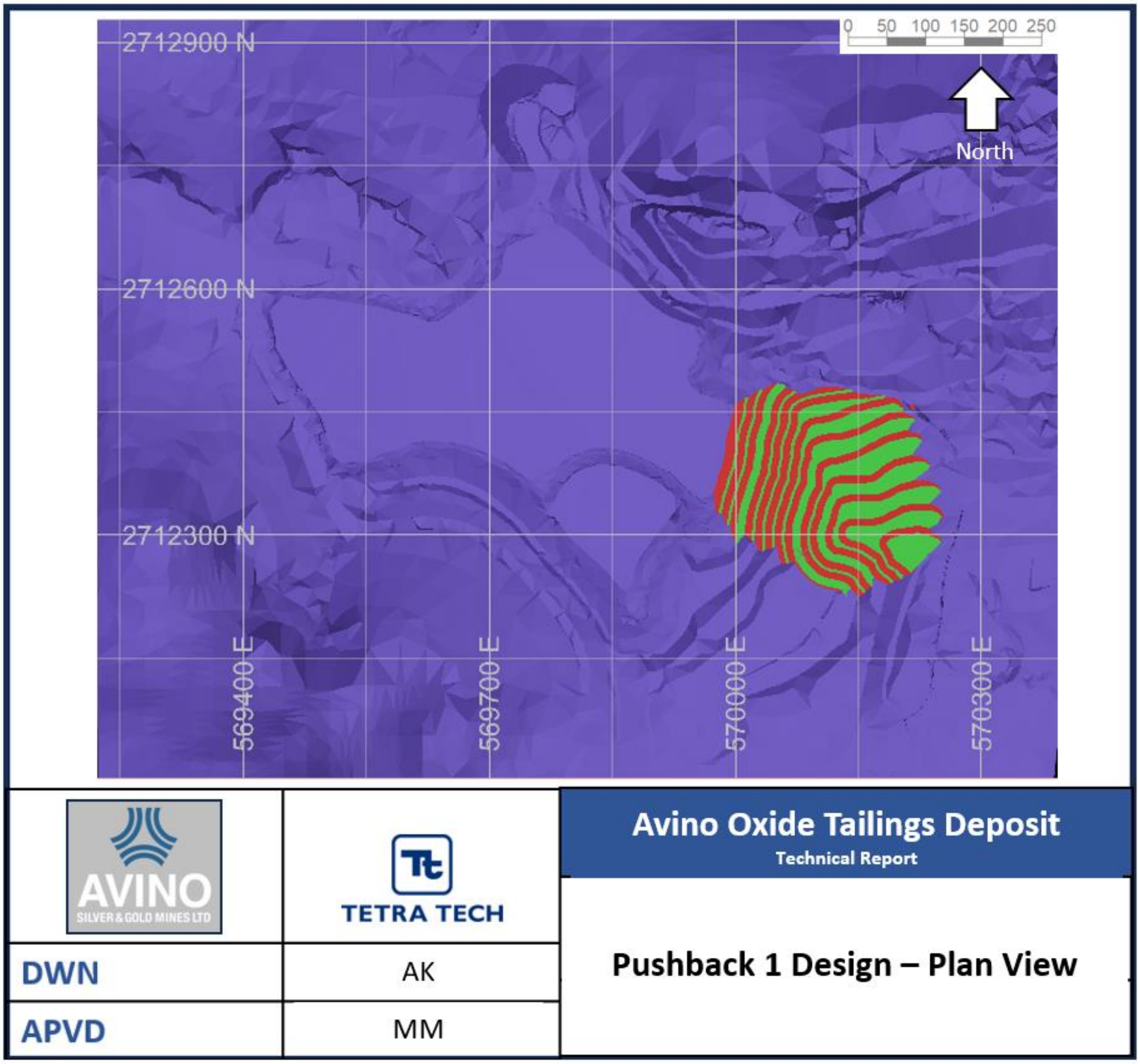

Figure 15-12: Pushback 1 Design

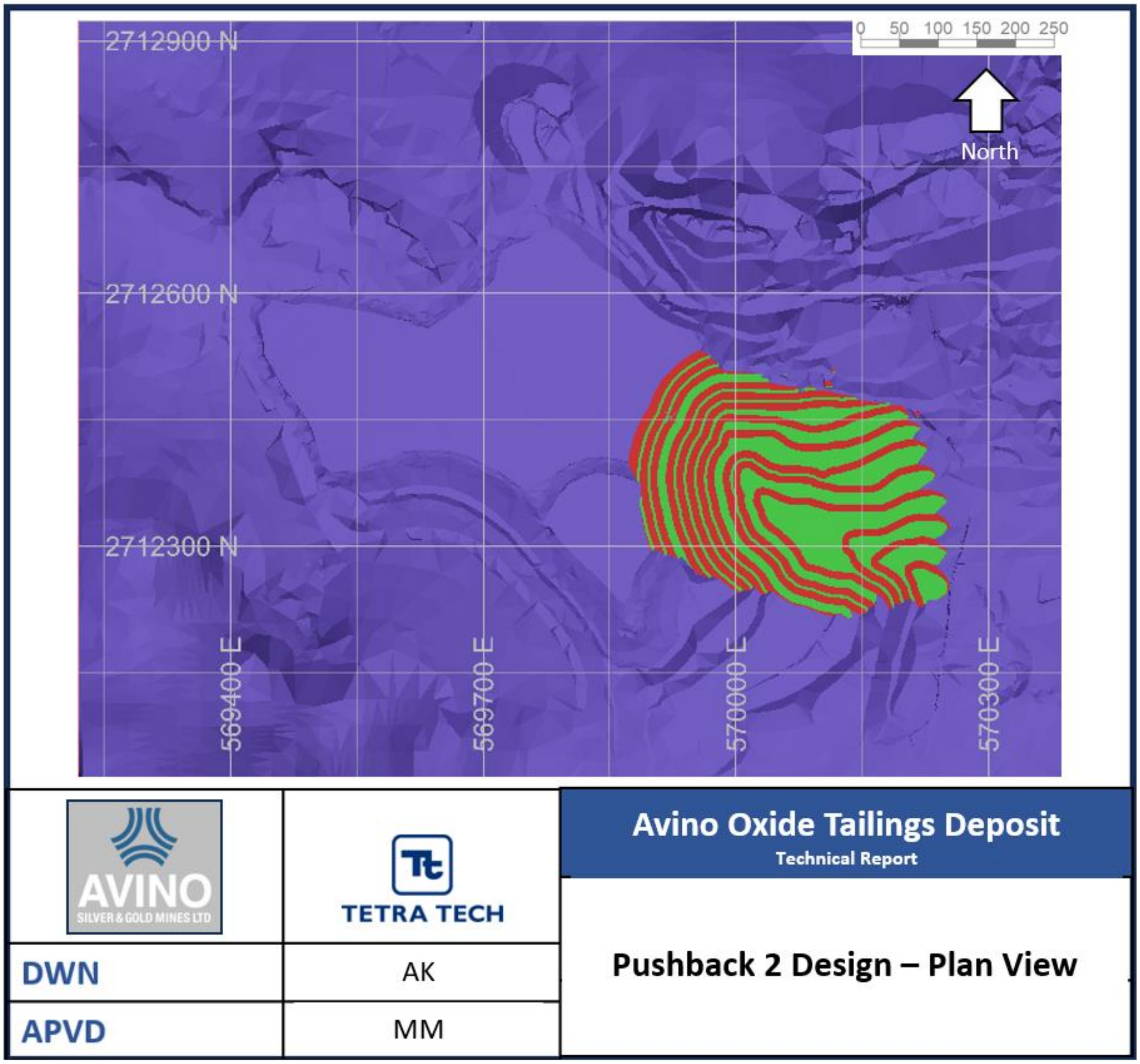

Figure 15-13: Pushback 2 Design

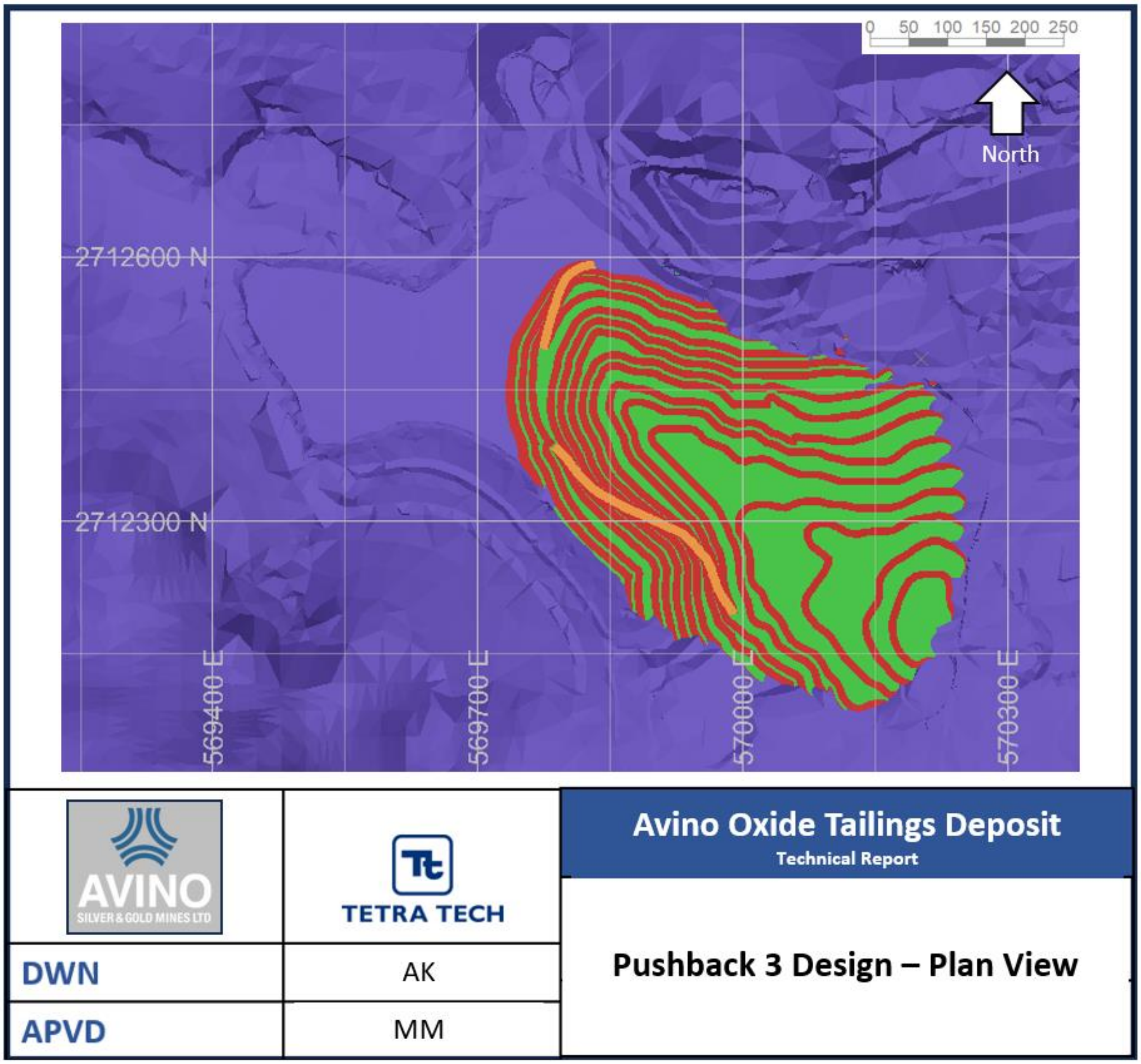

Figure 15-14: Pushback 3 Design

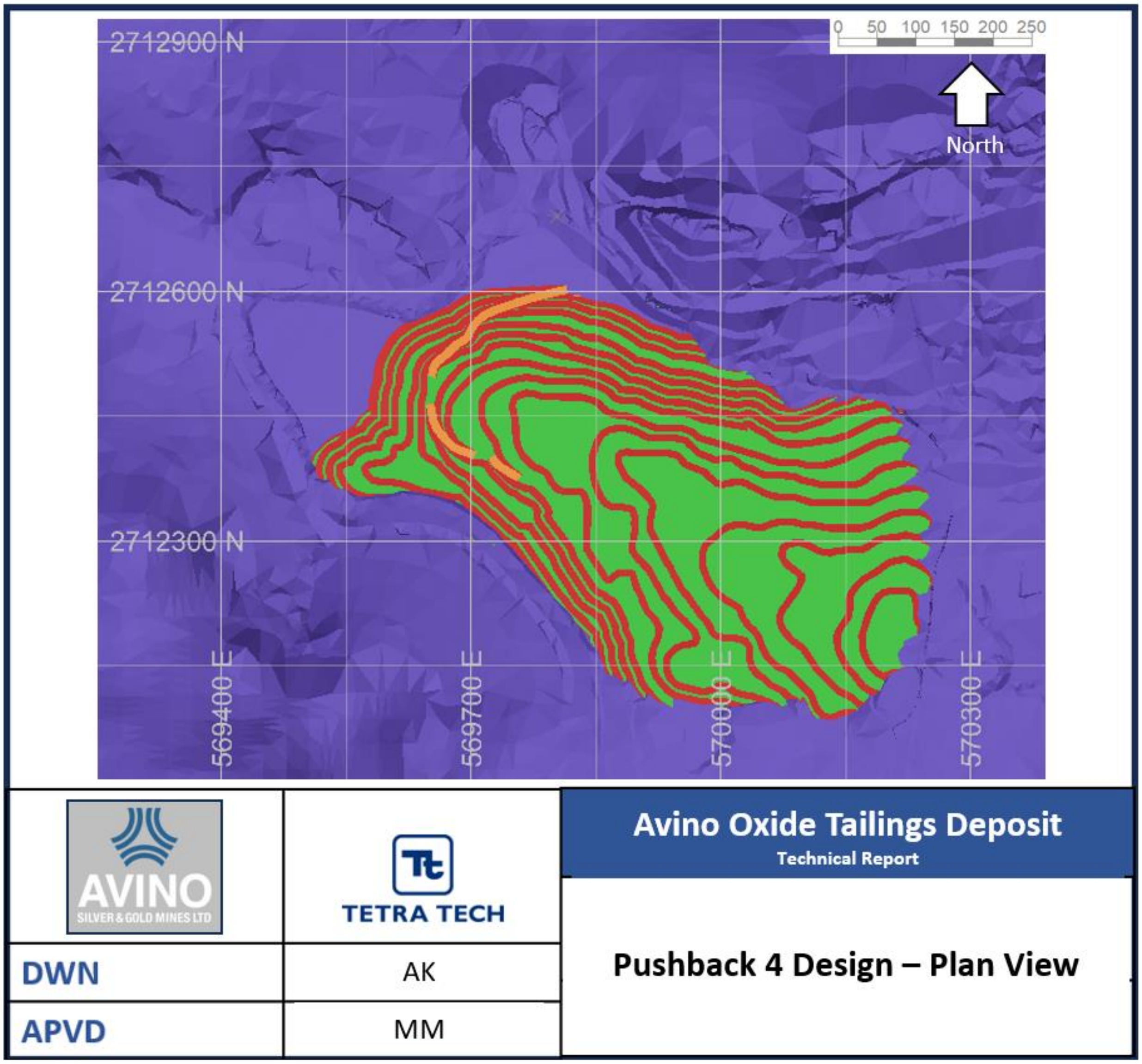

Figure 15-15: Pushback 4 Design

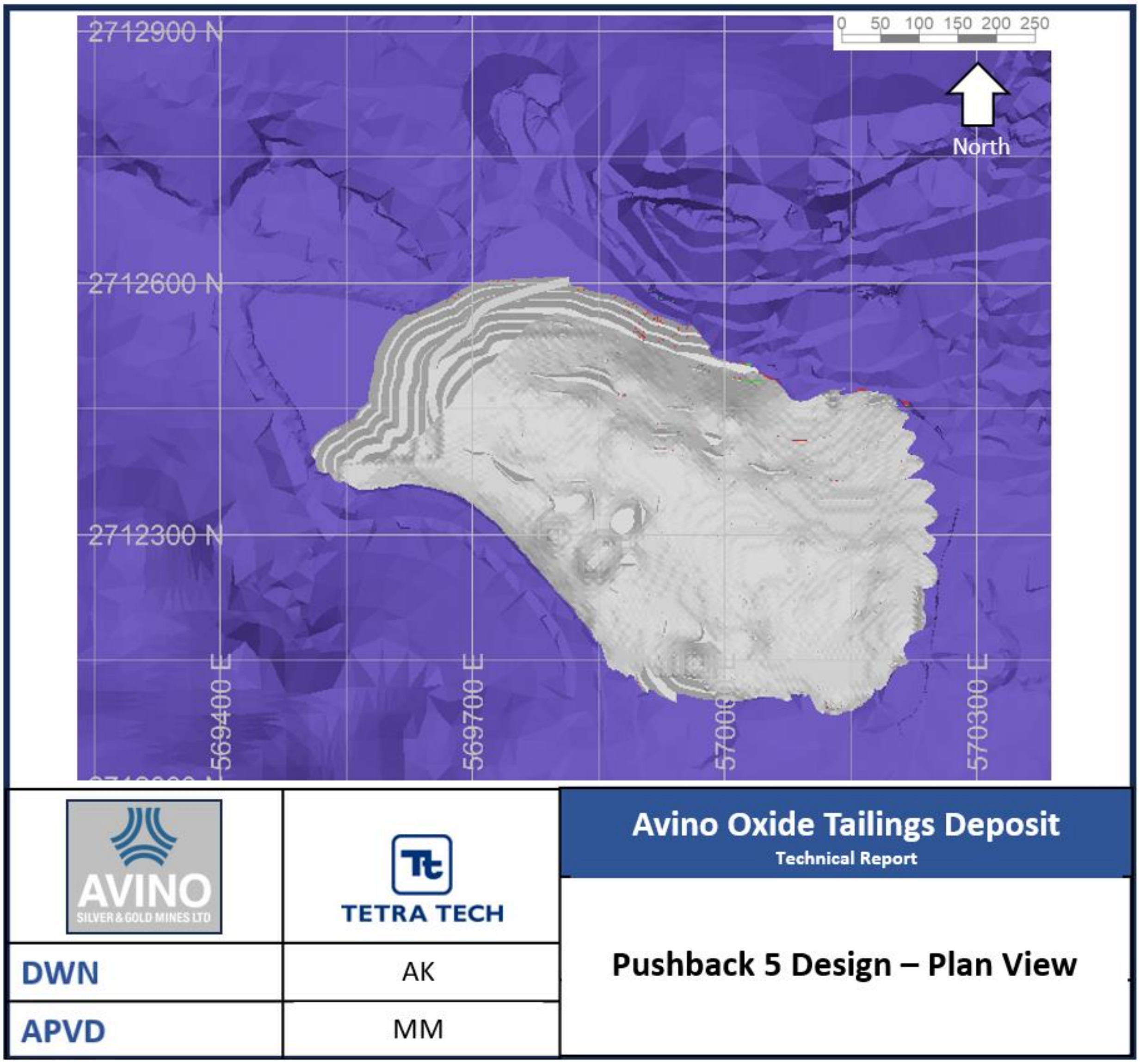

Figure 15-16: Pushback 5 Design

15.6 Mineral Reserve Statement

The Mineral Reserves were estimated using both oxide and sulphide tailings and are based on Measured and Indicated Resources only. The pit design used for the estimation was at the PFS level.

The tailings material to be mined at the Avino Project was deposited upon bedrock. The visual difference between the bedrock and tailings material will allow for recovery of most of the tailings while minimizing potential dilution when mining the final bench on bedrock.

To account for this, a mining dilution of 1% and mining recovery of 99% is included within the pit optimization model.

Mineral reserves were classified based on resource categories defined during resource estimation. Measured Resources were converted to Proven Reserves, and Indicated Resources were converted to Probable Reserves. No Measured Resources were included within Probable Reserves. No Inferred Resources were included within the reserve classification.

Proven and Probable Mineral Reserves are summarized in Table 15-4.

Table 15-4: Mineral Reserve Statement of the Avino Oxide Tailings Project

Reserve Category	Quantity (Million tonnes)	Average Ag Grade (g/t)	Average Au Grade (g/t)	Contained Ag Metal (Million tr. Oz)	Contained Au Metal (Thousand tr. Oz)
Proven	4.27	61	0.47	8.37	65.01
Probable	2.43	43	0.47	3.38	36.53
Total	6.70	55	0.47	11.75	101.54

Notes:

1. The effective date of the Mineral Reserve estimate is January 16, 2024. The QP for the estimate is Junjie (Jay) Li, P.Eng. of Tetra Tech
2. The Mineral Reserve estimates were prepared with reference to the 2014 CIM Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.
3. Reserves estimated assuming open pit mining methods
4. Reserves are reported on a dry in-situ basis
5. Reserves are based on a gold price of US $1850/tr oz., and silver price of US $22/tr oz, mining cost of US$1.00/t mined, milling costs of US$18.00/t feed, and USG&A cost of US$3.00/t feed.
6. Mineral Reserve includes consideration for 1% mining dilution and 99% mining recovery.
7. Ore-waste cut-off was based on US$21.00/t of NSR.

15.7 Comments on the Mineral Reserve Statement

Changes in the following factors and assumptions could affect the Mineral Reserve Estimate:

- Gold and Silver prices
- Interpretations of mineralization geology
- Geotechnical and hydrogeological assumptions
- Process method and recoveries
- Operating cost and sustaining capital cost assumptions and price escalation
- Ability to meet and maintain permitting and environmental license conditions.

The current Mineral Reserve estimates are based on the most current knowledge, permit status, and engineering constraints. The QP believes that the Mineral Reserves have been estimated using industry best practices.

16.0 MINING METHODS

16.1 Introduction

The Avino Oxide Tailings Project will be extracted using conventional surface mining techniques with an excavator, wheel loader, and trucks, operating 365 days per year with three 8-hour shifts per day. Based on the Measured and Indicated Resource, the mine plan includes transporting ore to the ROM feed located north of the deposit, and the waste material will be placed at a waste storage structure located east of the Project. Equipment selection and requirements are based on the existing equipment list provided by Avino and the design parameters of the pit and annual material movement, respectively.

16.2 Geotechnical Pit Slope Parameters

The geotechnical pit slope parameters are presented in Section 15.

16.3 Site Water Management

A prefeasibility level site-wide Water Management Plan (WMP) was developed based on site surface data available at the time this report is submitted. This WMP, titled *"Avino Silver & Gold Mines Ltd. Prefeasibility Water Management Plan"*, contains catchment delineations for watersheds that are expected to contribute to flow which either results from precipitation falling on mining operations (contact water), or which originally does not fall on mining operations (non-contact water) but would be expected eventually flow into and mix with contact water if not intercepted.

Sub-catchments identified from the watershed delineation were based on a DEM constructed from the Civil3D file *"Survey Avino-2023.dxf",* provided by Avino. This delineation includes identification of existing watercourses on site.

Tetra Tech identified key locations requiring diversion ditches or swales to convey water safely to existing watercourses or receiving waterbodies. These swales are typically located along existing road structures, which serve to separate contact and non-contact water and subsequently reduce the volume of water to be treated. It is assumed that contact water, e.g. runoff from existing tailings structures and future mine waste dump(s), will need treatment prior to release into receiving waterbodies. Consequently, Tetra Tech has proposed several locations for water storage ponds which, at a later project stage, could be designed to facilitate removal of sediments and other contaminants from contact water.

At this time, a groundwater study has not been conducted on the mine site. Groundwater plays an important role in understanding how certain mining operations will be impacted by flood events. Additionally, understanding groundwater flow facilitates a more thorough design of the previously mentioned swales and storage ponds, as it allows consideration of a base flow and infiltration rates. Tetra Tech strongly recommends Avino consider conducting a groundwater study of the site to augment future water management implementations and mining operations.

16.4 Phase Designs

To maximize the NPV of the project, mining phases (pushbacks or PB) have been designed and incorporated into the mining sequence to bring higher grade material forward and to defer waste stripping. The phase designs were guided by the lower revenue factor pit shells from the pit optimization analysis. A total of five phases have been designed. The mining phases were designed to provide operational flexibility while meeting geotechnical constraints and mining equipment available on site. For the LOM, there are at least two active phases being mined at one time to reduce the operational risk from geotechnical failures. Figure 16-1 shows the pushback schedule for the LOM.

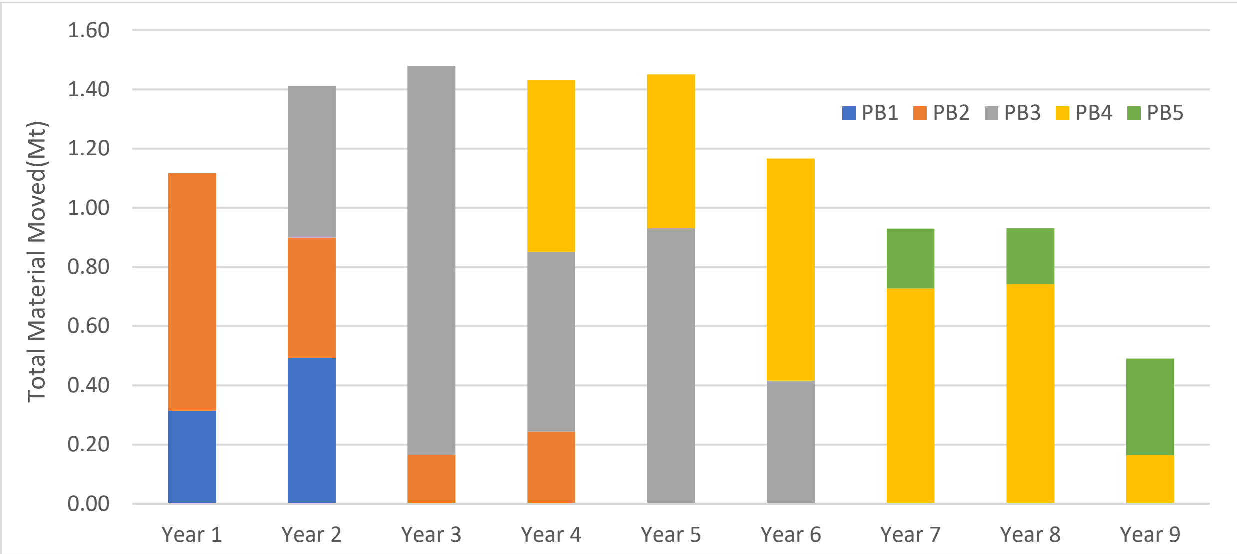

Figure 16-1: Mining Phases Schedule

16.5 Mine Waste Storage Facility

Over the LOM, 3.7 Mt of waste material will be stored in the mine waste storage facility located east of the mining area, sitting at a final elevation of 2220 masl. Figure 16-2 illustrates the location and its access haul road alignment. Dumps are designed to minimize haulage distances from the pits while also honoring geotechnical offsets from the ultimate pit and infrastructure.

For the design of the waste dump geometry, the assumption was made that the tailings materials used for the dump would have the same geotechnical properties as the tailings materials they will be sourced from. Therefore, the waste dump design used the same bench configuration as the pit geometry discussed in Section 15.3.1:

- 3.5 m berm width.
- 30° face angle.
- 5 m high bench lift.

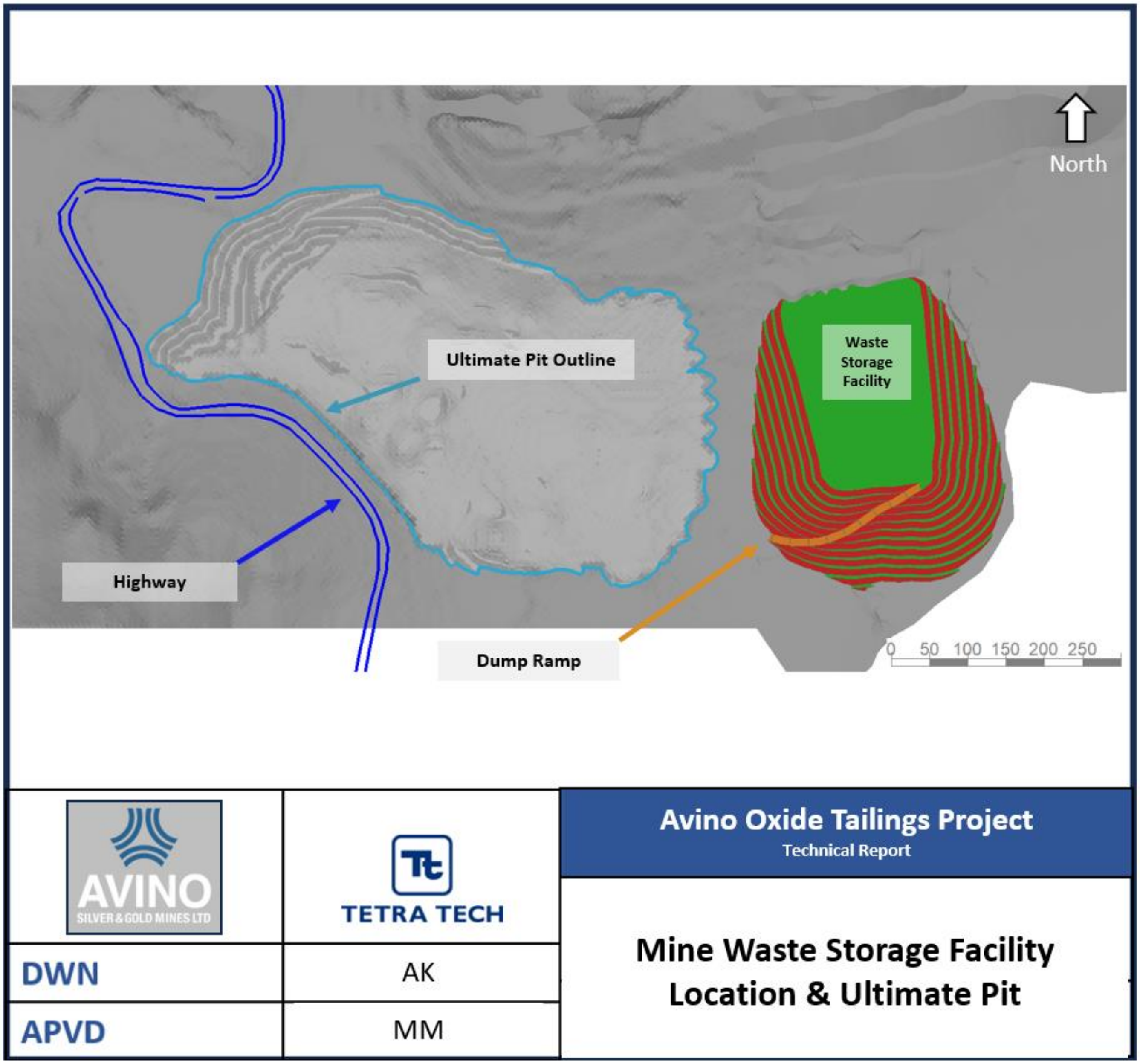

Figure 16-2: Mine Waste Storage Design Layout

16.6 Mine Production Plan

The mine production plan has been prepared using Datamine'sTM Studio OP and NPVS software. Provided with economic input parameters and operational constraints such as phase sequencing, maximum bench sink rates, mining and milling capacities, the software determines the optimal mining sequence.

The objective of the Avino LOM schedule was to maximize the early cash flow from the pit by targeting high-grade material to be fed to the processing plant while removing the overburden material. The optimum throughput to the processing plant was determined to be 821,250 tonnes per year with a 1 year ramp-up period. (Figure 16-3). The mining strip ratio averages 0.55 overall for the LOM.

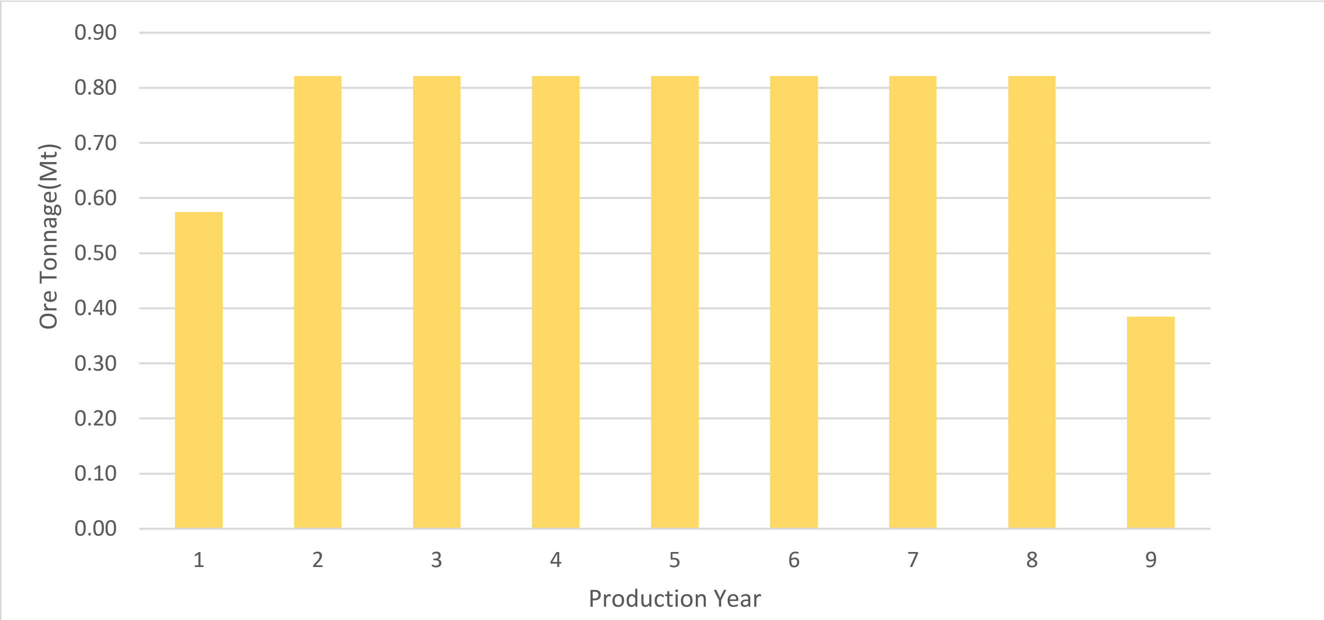

Figure 16-3 Mill Feed Schedule Over the Life of Mine

Other considerations of the production schedule included:

- Balance annual strip ratio to reduce equipment fleet fluctuations,
- Mining multiple phases to allow saturated areas of the pit to dewater and become trafficable. A saturated surface is expected as the mining benches are advanced deeper. This may present challenges for mining equipment to operate safely and efficiently around the pit areas. Due to the semi-arid climate of the region, it can take approximately 2-3 months for the saturated surface to become fully trafficable. As such, mining multiple phases and assigning a maximum vertical sinking rate will not only allow improved equipment trafficability but also reduce risks from geotechnical stability.

The mine life of the project is expected to be approximately 9 years. The mining rate will ramp up to around 1.4 Mt in years 2 to year 5 to accommodate a high strip ratio to remove the majority of the overburden and will start to ramp down in later years as the strip ratio decreases. Table 16-1, Figure 16-4 and Figure 16-5 illustrate the life of mine schedule and grade. Figure 16-6 and Figure 16-7 show year-end progression over the life of mine.

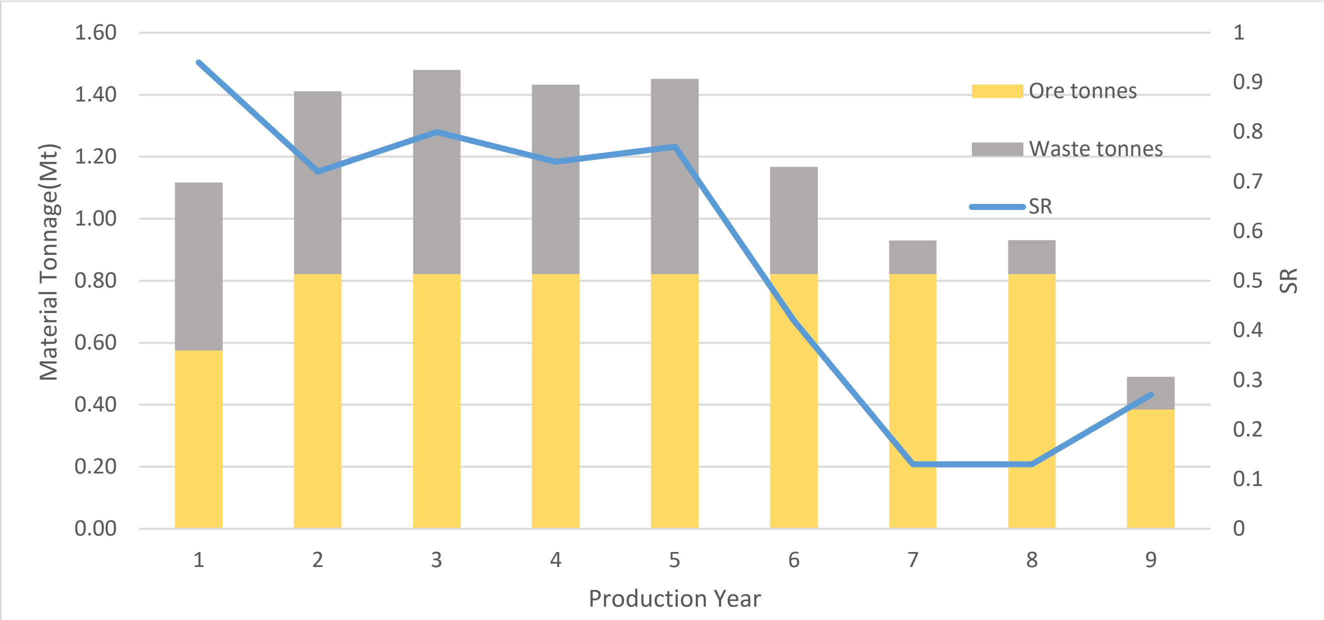

Figure 16-4: Ore and Waste Mining Schedule

Table 16-1: Life of Mine Schedule with Grade

Year	Ag Grade (g/t)	Au Grade (g/t)	Cu Grade (g/t)	SR	Total Material (t)	Waste (t)	Ore (t)
1	33.7	0.42	0.09	0.94	1,116,607	541,679	574,927
2	65.5	0.56	0.13	0.72	1,411,114	589,851	821,263
3	34.32	0.45	0.10	0.80	1,480,062	658,852	821,209
4	53.15	0.56	0.13	0.74	1,432,503	611,222	821,282
5	74.93	0.48	0.12	0.77	1,451,156	629,949	821,207
6	67.03	0.36	0.11	0.42	1,166,837	345,538	821,299
7	32.92	0.42	0.11	0.13	930,027	108,760	821,267
8	54.68	0.53	0.14	0.13	930,630	109,404	821,226
9	82.67	0.42	0.14	0.27	490,632	105,719	384,913
Total	**54.46**	**0.47**	**0.12**	**0.55**	**10,409,567**	**3,700,974**	**6,708,593**

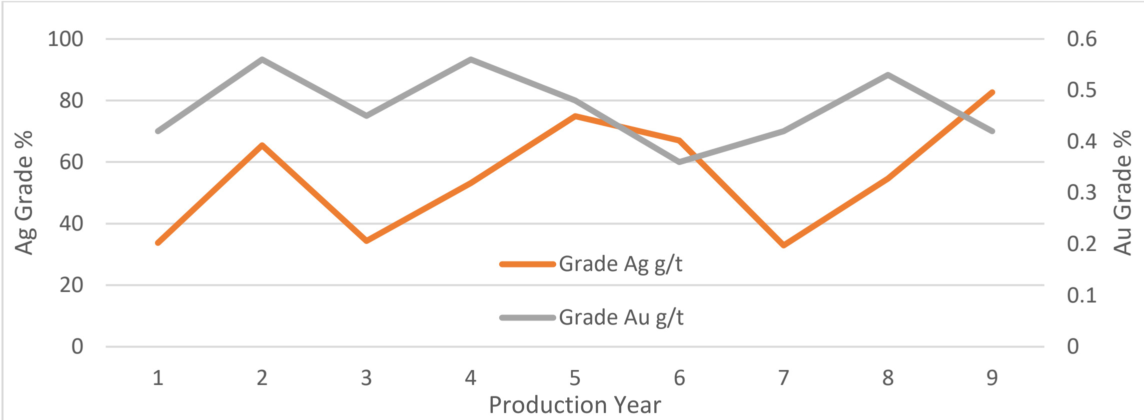

Figure 16-5: Ag and Au Feed Grade

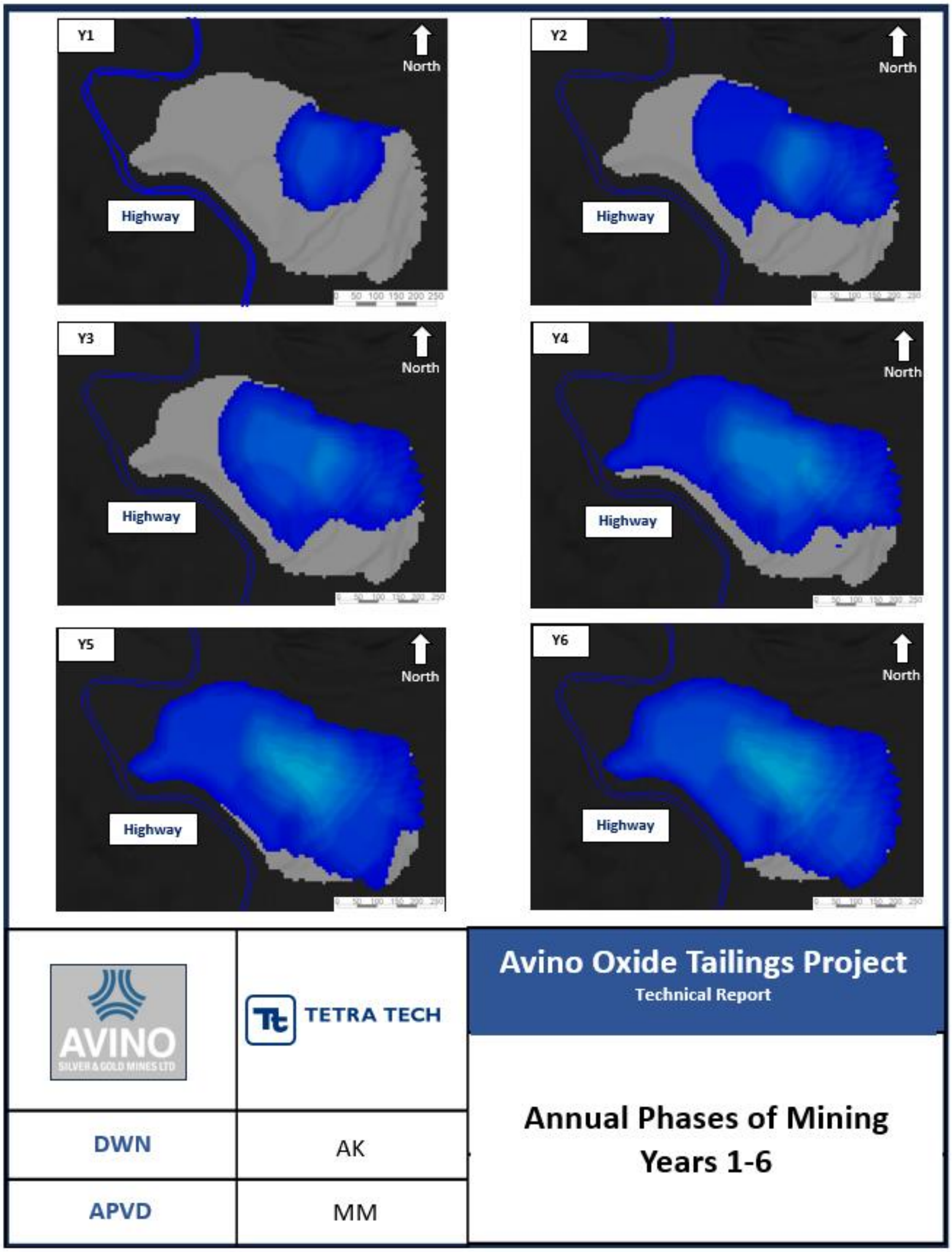

Figure 16-6: LOM Year-End Progressions Year 1 to 6

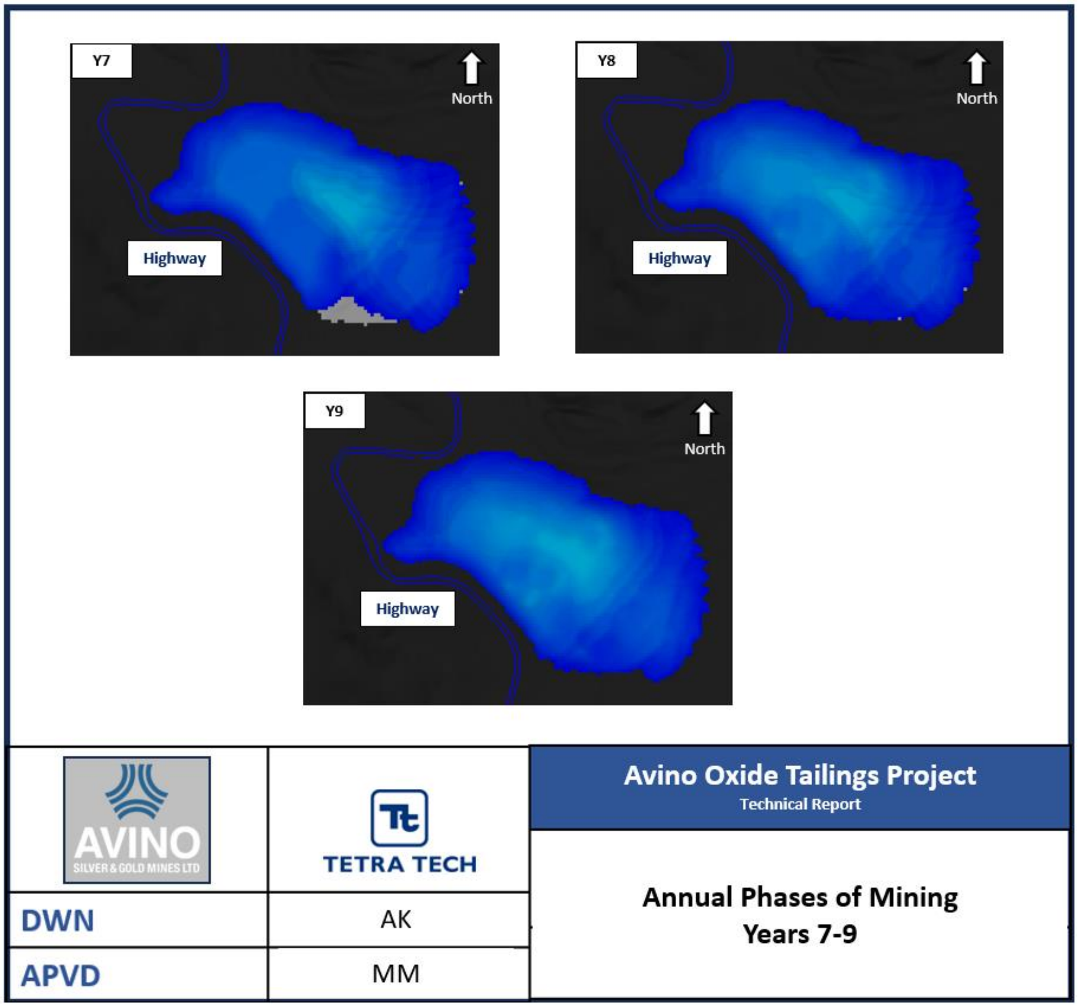

Figure 16-7: LOM Year-End Progressions Year 7 to 9

16.7 Mine Operation and Equipment Selection

Mining is to be carried out using conventional surface mining techniques with an excavator, wheel loader, and trucks in a bulk mining approach with 5 m benches. Equipment requirements are based on the design parameters of the pit and production rate requirements. Equipment availability and utilization is based on Tetra Tech's experience and vendor guidance. For determining the number of each piece of equipment required, the following parameters are considered:

- Annual material movement,
- Existing surface mining equipment on site provided by Avino,
- Existing access road profile and haul roads in pit and dump,
- Existing truck fleet operating cycle time, including spot, load, haul, dump, and maneuvering time,
- Equipment mechanical availability, utilization, and overall efficiency,
- Engine life of the equipment

16.7.1 Hauling

Tetra Tech understands that Avino currently employs a contract fleet (Freightliner medium-duty dump truck) to rehandle filtered tailings material for backfilling the open pit (Figure 16-8). Additionally, most of the existing access roads are either 8 or 10 meters wide which is determined to be acceptable for the current fleet operation. As such, haulage analysis was conducted based on the contract fleet. The truck fleet productivity was estimated in annual haulage distance and average travel speed (Table 16-2 and Table 16-3). Two haulage routes, including the ore and waste destinations, were digitized for each pushback at a given production year. The average travel speed was estimated based on site visit observation and on-site operation consultations.

Figure 16-8: Existing Contract Fleet On-Site

Table 16-2: Haul Truck One-way Travel Distance (Meters)

Destination	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Ore	1200	1300	1200	1200	1500	1300	1100	1700	1700
Waste	900	1000	900	1100	1100	1100	1100	950	820
Average	1100	1200	1100	1200	1300	1300	1100	1600	1500

Table 16-3: Haul Truck Productivity Assumptions

Description	Unit	Truck
Model	--	Freightliner M2 106 Plus
Engine Life	Hours	35,000
Maximum Payload	Tonnes	17
Effective Capacity	m^3	9.6
Fixed Cycle Time (Loading, Dumping, and Queuing)	minutes	8
Average Speed - Loaded	km/h	15
Average Speed - Empty	km/h	17
Operating Efficiency	%	70
Physical Availability	%	85

The total net operation hours by period combined with the truck's productivity was used to determine the number of trucks required (Figure 16-9). The maximum number of trucks required for the LOM is 5 units and then declines as mining quantity decreases.

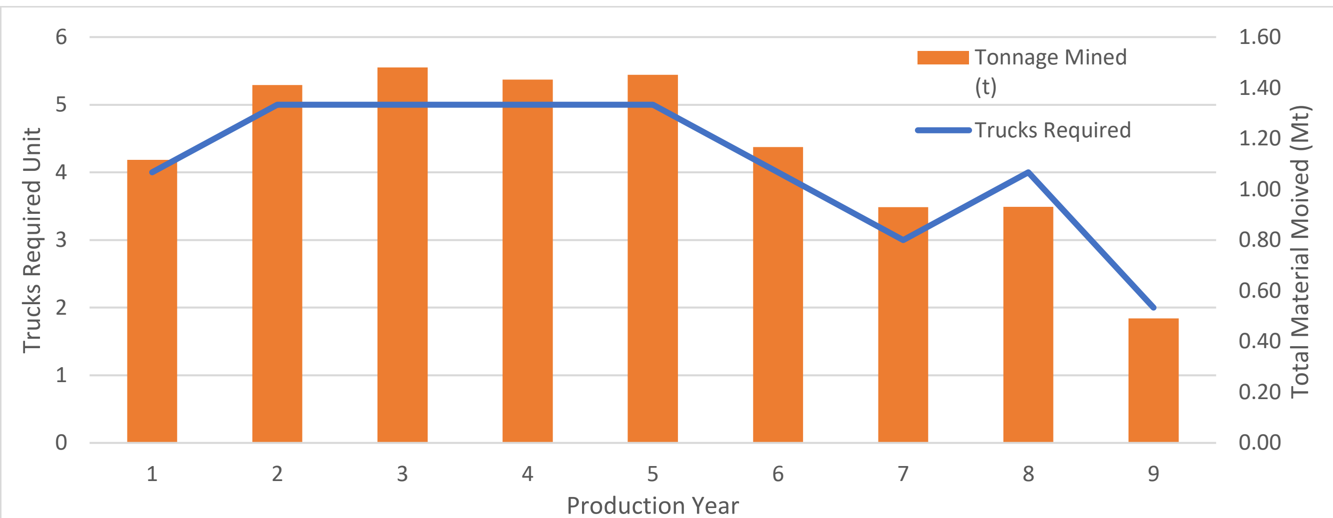

Figure 16-9: Truck Requirement and Total Mining Quantities

16.7.2 Loading

Tetra Tech reviewed the existing loading equipment list provided by Avino (Table 16-4) and determined that the loading will be done primarily using a Caterpillar 330 excavator which will be matched with a fleet of Freightliner medium-duty dump trucks. A Caterpillar 950L wheel loader will be responsible for the ROM feeder stockpile rehandling activities while complementing the excavator to improve the loading unit's physical availability and occasionally facilitate the ramp construction to improve trafficability. The loading productivity assumptions and requirements for both types of loading equipment are presented in Table 16-5 and Table 16-6, respectively.

Table 16-4: Existing Loading Unit On-Site

Model	Machine Type	Machine Label	Commissioning Date
Caterpillar 430F	Small loader	RE02	01/04/2016
PAUS RL852T5L2.6	Small loader	AM01	01/09/2022
Caterpillar - 330	Excavator	EX04	10/03/2023
Caterpillar 924H	Large Loader	CR01	27/04/2011
Caterpillar 320D	Excavator	EX01	23/07/2012
Caterpillar 420F	Small loader	RE01	12/07/2013
Caterpillar 980H	Large Loader	CR03	23/12/2014
Caterpillar 980L	Large Loader	CR05	01/08/2017
Caterpillar 950L	Large Loader	CR06	04/07/2019
Hyundai R220LC-9S	Excavator	EX03	19/09/2018
Volvo 330 2006	Excavator	EX02	31/12/2017
Caterpillar 330	Excavator	EX04	10/03/2023

Table 16-5: Loading Unit Productivity Assumptions

Description	Unit	Excavator	Front End Loader
Model		CAT 330	CAT 950L
Nominal Bucket Size	m^3	1.8	3.4
Cycle time per load	Seconds	30	40
Truck Spot Time	Seconds	30	30
Passes to Load a Truck	Number	6	3
Cycle time per truck	Minutes	3.5	2.5
Physical Availability	%	75%	25%[1]
Operating Efficiency[2]	%	70%	70%
Nominal Productivity	m^3/Year	722,000	351,000

Notes:.

1. Share the same unit with the mill feeder stockpile rehandle
2. Includes mobilizing delay between pushbacks, weather delays, and other operation delays

Table 16-6: Loading Unit Operating Hours

Loading Units	Unit Req'd	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Excavator	1	3,831	4,600	4,600	4,600	4,600	4,004	3,191	3,193	1,684
Front End Loader	1	766	920	920	920	920	801	638	639	337

16.7.3 Support Equipment

Surface mining haul road, ramps, and mining benches will be maintained with a fleet of support equipment. Tetra Tech understands that Avino has provided existing support equipment on-site and it is determined that the primary support equipment is acceptable for the mining operation, as follows:

- At the waste storage facility, a Caterpillar D5 track dozer will be used to maintain the desired bench geometry. The operating hours were estimated based on the material quantity handled and the rate of rise for the storage structure.
- At active mining benches, either a Caterpillar D6R or Komatsu D275AX-5E0 track dozer will be used for bench cleanups, facilitating ore and waste selectivity, and increasing loading unit productivity. When the loading units are down for maintenance, the track dozer can be used for dewatering control berms, ramp development, pit floor clean-ups, etc.
- A 12 ft blade grader (Caterpillar 120K2) and a water truck will be designated for haul roads which will improve trafficability and dust suppression, respectively.

Table 16-7 and Table 16-8 show the existing primary support equipment on-site operating hours throughout the mine life.

Table 16-7: Existing Support Equipment On-Site

Model	Machine Type	Machine Label	Commissioning Date
Caterpillar D4	Dozer	TR04	3/10/2023
Caterpillar D6R	Dozer	TR01	4/25/2011
Caterpillar D5	Dozer	TR03	8/16/2022
Komatsu D275AX-5E0	Dozer	X181	6/30/2018
Caterpillar 120K2	Grader	MN01	11/30/2015

Table 16-8: Support Equipment Operating Hours

Working Area	Unit Req'd	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Dozer at Waste Dump	1	1,075	538	538	538	538	376	134	134	134
Dozer at mining Bench	1	2,682	3,389	3,555	3,441	3,486	2,803	2,234	2,235	1,178
Grader	1	2,190	2,190	2,190	2,190	2,190	2,190	2,190	2,190	1,095
Water Truck	1	1,460	1,460	1,460	1,460	1,460	1,460	1,460	1,460	1,460

16.7.4 In-pit Dewatering

As identified from the Avino Prefeasibility Water Management Plan (WMP 2023), prepared by Tetra Tech, storm events that release high quantities of precipitation in short periods of time can pose significant operational risks to the mining. Tetra Tech used a 1:200-year, 24-hour precipitation event for the dewatering study. Based on the annual mining footprint and rainfall precipitation, a total of four 125 hp class submersible pumps will be required over the mine life.

16.7.5 Mine Operation Labour

The open pit will operate continuously, with three 8-hour shifts daily, 365 days per year. The life-of-mine labour complement was calculated from first principles based on the number of units of equipment required to achieve the planned production schedule. Over the life of mine, excluding the contract truck fleet, the complement will be 22 full-time equivalent personnel per day, shown as in Figure 16-10.

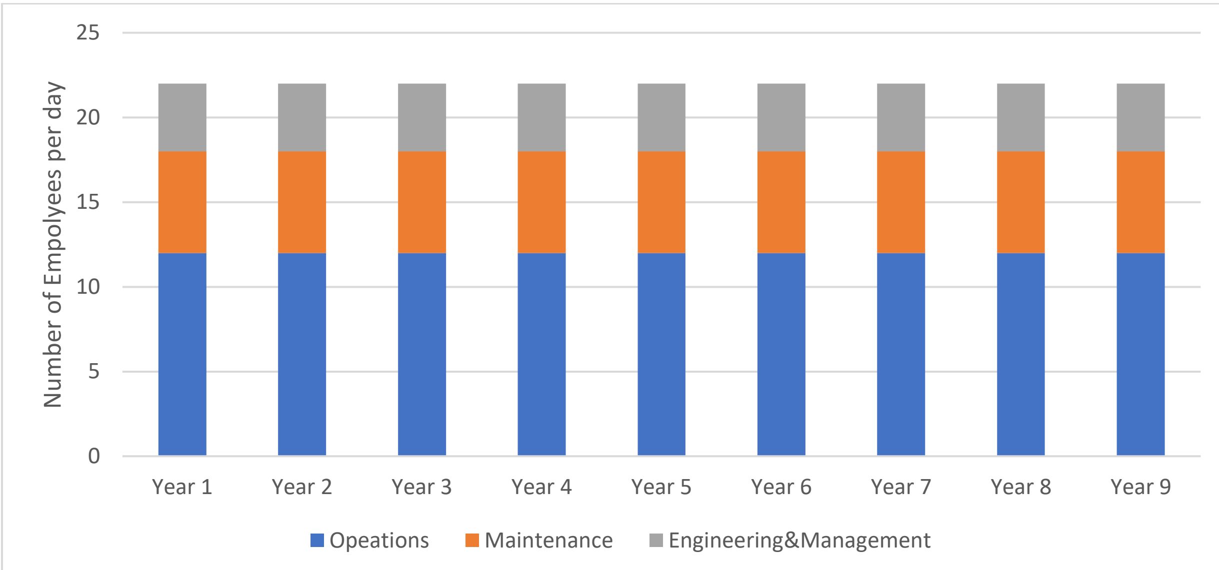

Figure 16-10: Life of Mine Labour Requirement

17.0 RECOVERY METHODS

17.1 Introduction

17.1.1 Avino Area

The Avino area hosts multiple mineralizations, namely Avino ET, San Gonzalo, Guadalupe and La Potosina and potential tailings resources from previous operations.

- The Avino ET vein deposit is currently being mined and processed at the Avino processing facility. The description and production statistics for the current processing facility are presented in Section 17.2.
- The San Gonzalo vein deposit entered commercial production in October 2012 and stopped its operation in Q4 2019 as it reached the end of its resources.
- The Guadalupe and La Potosina veins deposits are currently exploration targets. Currently, there is no operation to recover metals from these resources.
- The tailings deposit comprises historic mineral processing plant tailings material deposited during the different periods of open pit mining of the Avino Vein. The oxide tailings are partially covered by younger sulphide tailings. This PFS focuses on silver and gold recovery from the tailings resources. The processing plant design information is presented in 17.3.

17.1.2 La Preciosa Area

The La Preciosa deposit is an exploration target, and there is no operation to recover metals from the La Preciosa resources.

17.2 Existing Processing Facility

The Avino ET Vein was mined during 27 years of open pit and underground production prior to 2001. The mine was closed in November 2001 due to low metal prices and the closure of a key smelter. Prior to the mine closure in 2001, the processing plant averaged 1,000 t/d and achieved up to 1,300 t/d operations, producing copper concentrate that was sold to a smelter in San Luis Potosi. During the final 3 full years of operation, production averaged 1.7 Moz of silver equivalent annually. Following several years of redevelopment, Avino completed the Avino mine and plant expansion in Q4 2014. Full-scale operations commenced on January 1, 2015, and commercial production was declared effective April 1, 2016, following a 19-month advancement and test period. In 2018, Avino completed an expansion to its processing plant, increasing its combined throughput capacity to 2,500 t/d.

The ROM feed is processed in a conventional flotation circuit, producing copper flotation concentrate containing silver and gold. Avino operates four separate circuits with a combined throughput capacity of 2,500 t/d. The four circuits include the following:

- Circuit #1 has a nominal processing capacity of 250 t/d. It was previously used to process ore from the San Gonzalo vein, producing a lead concentrate containing gold and silver and a zinc concentrate. The circuit has now been modified to process materials from the Avino ET vein to produce a copper flotation concentrate containing silver and gold.

- Circuit #2 and #3 have a nominal processing capacity of 250 t/d and 1,000 t/d, respectively.
- Circuit #4 has a nominal processing capacity of 1,000 t/d and is currently processing ore from the Avino ET vein, although it also processed low-grade materials from historic stockpiles before Q2 2020.

17.2.1 Circuit Description

The current circuit at Avino utilizes a conventional three-stage crushing followed by a ball mill grinding and flotation circuit to produce copper concentrate containing silver and gold.

17.2.1.1 Crushing Facility

The existing crushing plant is capable of handling plant feed at the rate of 2,500 t/d. ROM feed is processed using a conventional three-stage crushing circuit, and the crushed ore is conveyed to separate bins/stockpiles dedicated to the different grinding and flotation circuits.

The ROM ore is transported to the processing plant using haul trucks and dumped in the ROM bin. The ore is crushed in a primary jaw crusher. The jaw crusher product is screened using a double deck primary vibrating screen with a 50 mm aperture size for the top deck and a 12.5 mm aperture size for the bottom deck. The screen oversize (+12.5 mm) is further crushed in a secondary standard head cone crusher, whereas the screen undersize (-12.5 mm) is sent directly to storage bins/stockpiles. The secondary cone crusher product is screened using a double-deck secondary vibrating screen with a 19 mm aperture size for the top deck and a 9.5 mm aperture size for the bottom deck. The screen oversize is further crushed using tertiary cone crushers, and the tertiary cone crusher product is recycled back to the secondary screen. The secondary screen undersize and the primary screen undersize are conveyed to storage bins/stockpiles.

The crushing facility includes:

- One 0.8 m x 1.1 m primary jaw crusher with an installed power of 112 kW,
- One 0.5 m x 0.9 m primary jaw crusher (standby unit) with an installed power of 56 kW,
- One 1.5 m wide x 4.9 m long double deck primary vibrating screen,
- One secondary standard head cone crusher with an installed power of 224 kW,
- Two 2.4 m wide x 6.1 m long double deck secondary vibrating screens,
- One Nordberg tertiary cone crusher with an installed power of 373 kW,
- Other ancillary equipment, such as feeders, surge bins, conveyors, belt magnets, and metal detectors.

17.2.1.2 Grinding/Gravity Concentration/Flotation

As discussed previously, the processing plant has four grinding and flotation circuits to recover valuable metals from the Avino ET deposit. The crushed material is fed to the grinding circuits, each consisting of a ball mill and cyclone cluster to grind the crushed material to a size of approximately 55% to 60% passing 75 µm (200 mesh). Lime is added in the ball mill to raise the slurry pH to about 10.5 to depress pyrite.

Circuits #1 and #2 utilize 2.4 m diameter by 1.8 m long ball mills, each with an installed power of 168 kW and a cyclone cluster consisting of two 380 mm diameter Krebs cyclones. Circuits #3 and #4 utilize 3.2 m diameter by

4.6 m long ball mills, each with an installed power of 746 kW and a cyclone cluster consisting of two 500 mm diameter Krebs cyclones.

A centrifugal Falcon gravity concentrator, processing ball mill discharge, has been installed in all four processing circuits to recover a high-grade gold/silver concentrate suitable for dispatching directly to a smelter.

The cyclone overflows from different circuits are fed to their flotation circuits, consisting of rougher flotation, scavenger flotation, and cleaner flotation. The rougher flotation tailings report to the scavenger flotation. The scavenger flotation concentrate is recycled back to the rougher flotation, while the tailings are pumped to the tailings thickener. The concentrates from the rougher and scavenger circuits are upgraded in a single stage of cleaner flotation to produce the final concentrate grading of 20% to 25% copper. The cleaner flotation tailings are recycled back to the rougher flotation. Flotation reagents include Aero 404 and Aerophine 3418A as collectors and CC-1065 (Dowfroth 250 equivalent) as frother.

The flotation circuit includes:

- Circuit #1:
  - Four 2.8 m^3 rougher flotation cells and four 2.8 m^3 scavenger flotation cells,
  - Two 1.4 m^3 cleaner flotation cells.
- Circuit #2:
  - One 17 m^3 rougher flotation cell and one 8.5 m^3 scavenger flotation cell.
  - One 8.5 m^3 cleaner flotation cell.
- Circuit #3 and #4 (each):
  - Three 37 m^3 rougher flotation cells and one 17 m^3 scavenger flotation cell
  - One 17 m^3 cleaner flotation cell.
- Other ancillary equipment, such as pumps, conditioning tanks, agitators and pump boxes.

The flotation concentrate from Circuit #1 is pumped to a 9 m diameter concentrate thickener, whereas flotation concentrates from Circuit #2, #3 and #4 are pumped to a 12 m diameter concentrate thickener. The thickener underflow from both thickeners is combined and pumped to two concentre pressure filters (13 plates each with dimensions of 1.2 m x 1.2 m) for further dewatering. The final concentrate is dewatered to approximately 9% moisture prior to being shipped to an off-site smelter.

The flotation tailings from all four flotation circuits are pumped to one conventional tailings thickener (24 m diameter). The tailings is thickened to an underflow solids density of 55-60% (by wt.) and then further dewatered to a solids density of 80% (by wt.) using two pressure filters (98 plates each with dimensions of 2.0 m x 2.0 m). Diluted flocculant solutions are added into concentrate and tailings thickeners to assist in the settling of fine particles.

The dry stack tailings management facility with pressure filters was installed in Q1 2019 and in operation since Q2 2019 to conserve the process water consumption. The thickener overflow and the decant water from the permitted tailings impoundment areas are reclaimed for process use. The tailings cake is stored on the surface of the existing tailings management facility. The dry-stack tailings improved the overall tailings facility safety and stability.

The simplified flowsheet of the current operation is shown in Figure 17-1.

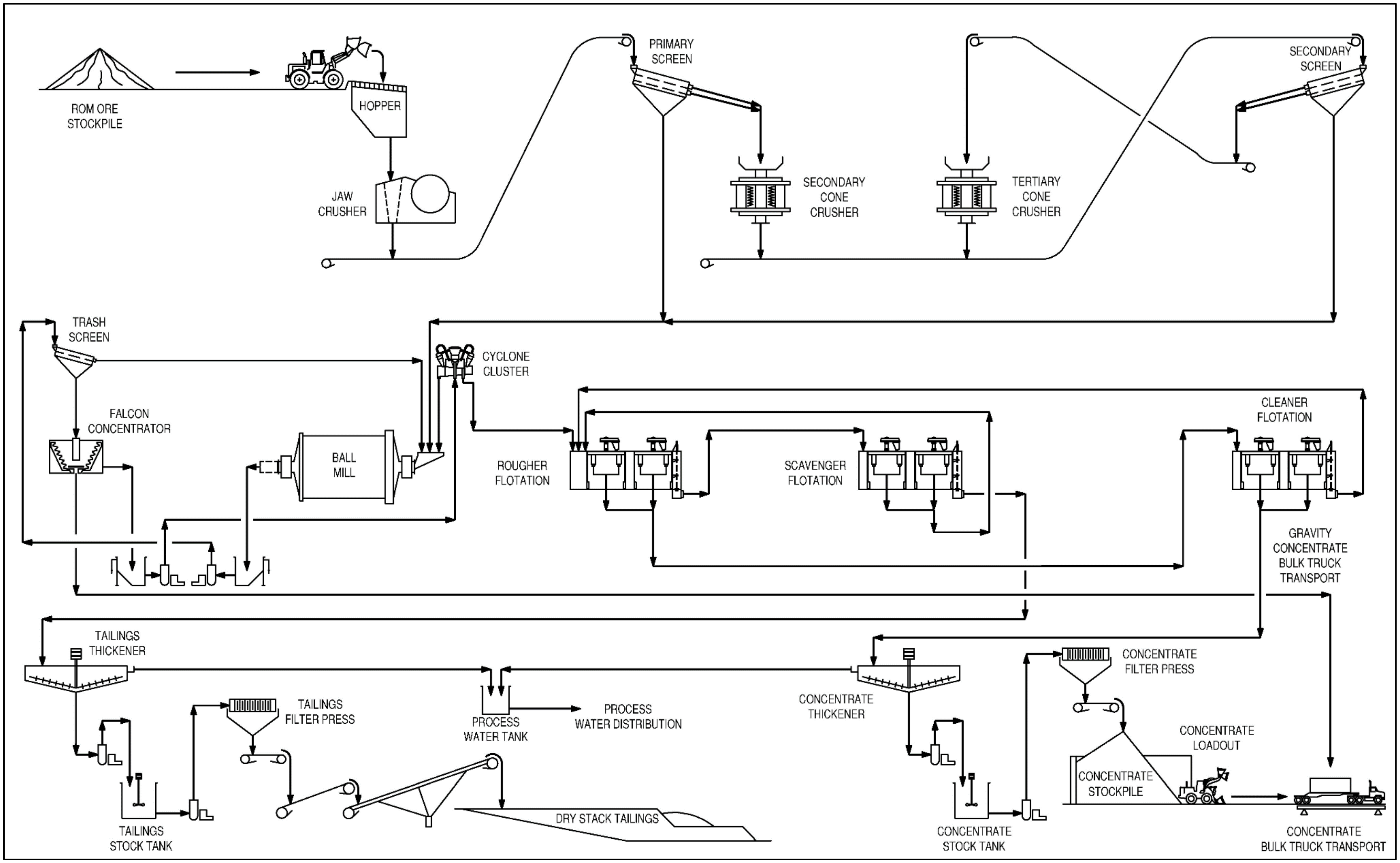

Figure 17-1: Simplified Flowsheet – Avino Processing Plant (Current Operation)

The flotation circuit has air blowers and compressors to provide the air required for the flotation cells and the filter presses. A Courier Model 5 online sampling system has been installed to monitor production performance in Circuits #3 and #4. This system samples the feed, tailings, and concentrate streams of the two circuits. The Courier Model 5 online sampling system complements the existing six sampling points. The system continuously measures copper, lead, zinc, and iron contents, as well as the slurry density of the streams.

Hand samples are collected at predetermined times for sample preparation and analysis. The plant samples are routinely analyzed by an AAS for copper, lead, zinc, bismuth, antimony, and iron and by fire assay for gold and silver.

17.2.1.3 Production Statistics

Table 17-1 summarizes the productions from Avino since 2020. The San Gonzalo Mine went through a planned shutdown in Q4 2019. As part of the ramp-up of operations, 10,806 t of historic above-ground stockpile material was processed during Q3 2021.

Table 17-1: Total Production (Avino, 2024)

Description	2023	2022	2021*
Feed Tonnage			
Tonnes Milled (dry t)	615,373	541,823	165,304
Feed Grade			
Silver (g/t)	54	62	53
Gold (g/t)	0.51	0.42	0.84
Copper (%)	0.47	0.61	0.57
Recovery			
Silver (%)	87	92	87
Gold (%)	72	78	75
Copper (%)	83	89	88
Total Metal Produced			
Silver Produced (oz)	928,643	985,195	245,372
Gold Produced (oz)	7,335	5,778	3,386
Copper Produced (lbs)	5,304,808	6,504,177	1,869,306

* After a period of operational suspension, the Avino Mine restarted production during Q3 2021.

17.3 Tailings Resources

Two types of tailings are produced from previous mining operations: oxide tailings and sulphide tailings. Currently, there is no operation to recover metals from both tailings resources. The PFS focused on recovering silver and gold from the tailings resources using cyanidation treatment in a new processing facility.

The tailings resources will be mined using front-end loaders and trucked to a new processing facility for treatment. The tailings will be processed by tank cyanide leaching, followed by the Merrill-Crowe process to recover silver and gold. The residual material will be detoxified, filtered, and deposited in a lined dry stack tailings management facility. This section outlines the major design criteria and describes the unit processes of the flowsheet.

17.3.1 Flowsheet Development

The proposed processing plant has been designed to process ore from the historic oxide tailings at a nominal throughput of 2,250 t/d, producing silver-gold doré. The LOM average plant feed grade is estimated to be 54.5 g/t silver and 0.47 g/t gold. The LOM average silver and gold recovery is estimated to be 77.2% and 74.9%, respectively. The processing flowsheet has been developed based on the test results discussed in Section 13.

The processing plant will consist of the following:

- A trommel for the tailings repulping and thickener
- Cyanide leaching of the repulped tailings
- Countercurrent decantation (CCD) washing and pre-clarification of the pregnant leach solution (PLS)
- Pregnant solution clarification, de-aeration and gold and silver precipitation by zinc dust (Merrill-Crowe)
- Gold and silver precipitate melting to produce doré
- Cyanide destruction of residual tailings
- Leach residue filtration and deposition to the dry stack tailings facility.

The simplified process flowsheet for the tailings reprocessing is shown in Figure 17-2 and detailed in the following sections.

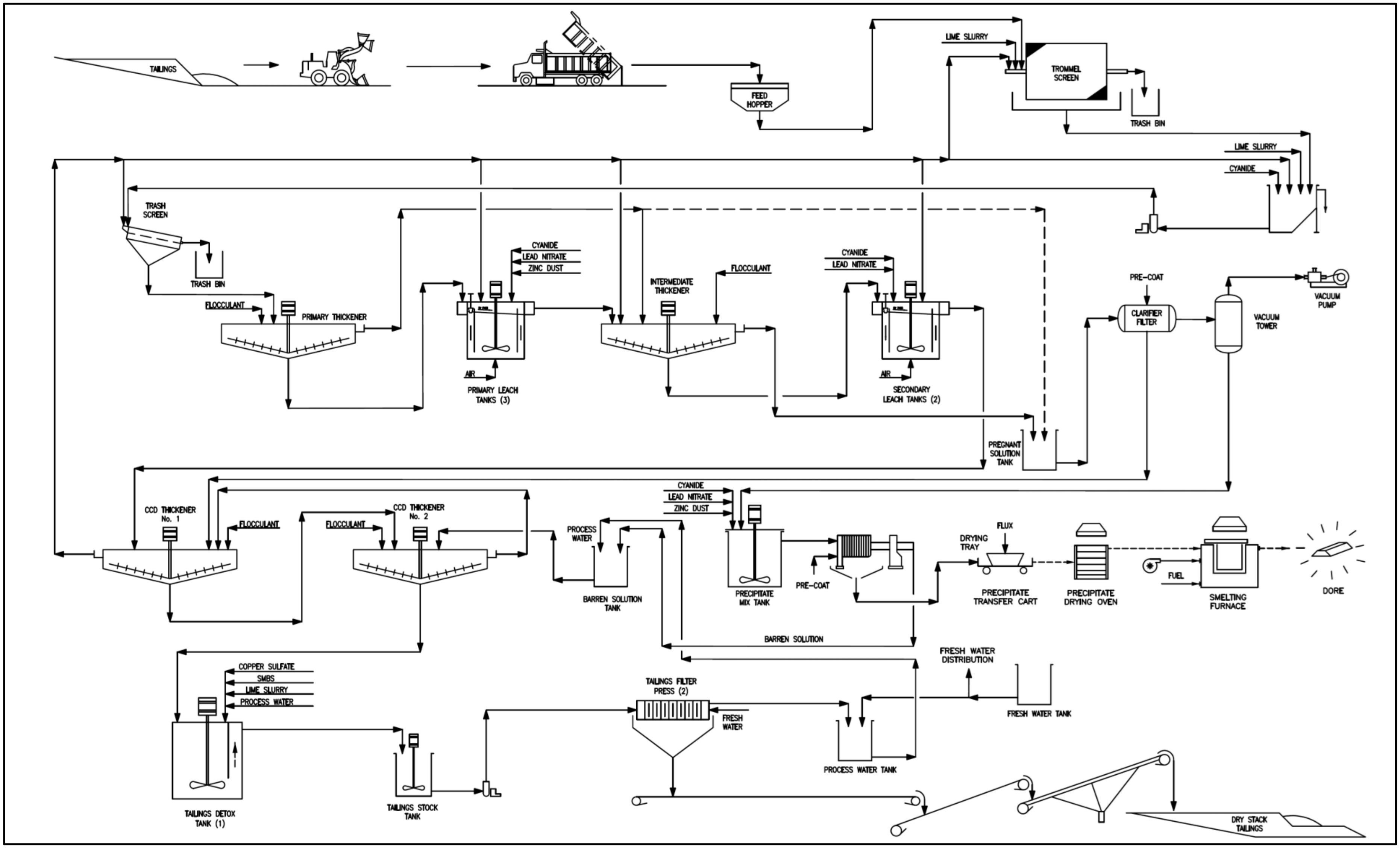

Figure 17-2: Simplified Process Flowsheet for Tailings Reprocessing

17.3.2 Process Design Criteria

The processing plant is designed at a nominal throughput of 2,250 t/d for an average annual throughput of 821 kt at full capacity. The major criteria used to design the processing plant are listed in Table 17-2.

Table 17-2: Major Plant Design Criteria

Description	Unit	Value	Source
Ore Characteristics			
Specific Gravity	g/cm^3	2.74	Testwork
Bulk Density	t/m^3	1.71	Industry Experience
Moisture Content	% (by wt.)	10.0	Industry Experience
Bond Ball Mill Work Index (75^{th} Percentile)	kWh/t	13.9	Testwork
Abrasion Index (Average)	g	0.66	Testwork
Operating Schedule			
Shift/Day	--	3	Client
Plant Hours/Shift	h	8	Client
Plant Hours/Day	h	24	Client
Days/Year	days	365	Client
Plant Throughput			
Overall Plant Feed	t/d	2,250	Client
Overall Plant Feed	kt/y	821	Calculation
Plant Availability	%	92	Industry Experience
Nominal Plant Throughput	t/h	102	Calculation
Grades and Productions Estimates			
Average LOM Head Grade	g/t Ag	54.5	Mine Plan
Average LOM Head Grade	g/t Au	0.47	Mine Plan
Average LOM Silver Recovery	%	77.2	Calculation
Average LOM Gold Recovery	%	74.9	Calculation
Annual Silver Production (LOM Average)	koz/y	1,008	Calculation
Annual Gold Production (LOM Average)	koz/y	8.45	Calculation

17.4 Process Description

The processing plant is designed to treat 2,250 t/d of the historical tailings. The electricity power required for the process plant will be provided from an onsite substation located northwest of the process plant. Details of the power supply are described in Section 18.

17.4.1.1 Tailings Repulping

The exiting tailings will be mined using a front-end loader and trucked to the feed dump surge bin in the processing plant area. The feed bin will have a live capacity of 36 t. The material will be reclaimed using a belt feeder when lime will be added and conveyed to a rotating trommel screen with an aperture opening of 6.35 mm. The overflow, containing cyanide and dissolved gold and silver, from downstream CCD circuit thickener no. 1 will be added to the trommel for repulping. The trommel screen undersize will report to a pump box at a solids density of 50% (by wt.) and then be pumped to the vibrating screen with an aperture size of 1 mm for controlling the leaching circuit feed particle size. The trommel and size control screen oversize materials, which are anticipated to be entrained rocks and other foreign materials, will be stored in the waste dump. The size control screen undersize will report to a pump box and then be pumped to the primary thickener. The CCD thickener no. 1 overflow will be added to the primary thickener as washing water. The thickener overflow will be pumped to the downstream intermediate thickener, whereas the thickener underflow at 50% solids density (by wt.) will be pumped to the primary leaching circuit.

The key equipment in the repulping circuit will include:

- One 3.0 m diameter x 8.0 m length trommel screen,
- One 1.6 m wide x 3.7 m long vibrating screen,
- One 23 m high-rate thickener,
- Related feeding systems and auxiliary equipment (pumps, sump pumps, pump boxes).

17.4.1.2 Primary Leaching and Thickening

The repulped tailings will be leached in a train of three mechanically agitated carbon steel tanks equipped with dual impellers, providing 36 hours of residence time for the primary leaching stage. Pulp will flow by gravity cascading from the No. 1 leach tank sequentially through to the No. 3 leach tank. It will be possible to bypass any tank for maintenance.

The CCD thickener no. 1 overflow will be added to the first leach tank to reduce the slurry solids density to 47% (by wt.). Cyanide will be added in solution form to the first leach tank for bearing gold and silver extraction . Compressed air will also be sparged to all the leach tanks to assist with leaching kinetics. The primary leaching circuit discharge will be pumped to an intermediate thickener to recover leached silver and gold. The primary thickener overflow will be added to the thickener as washing water. The thickener overflow will report to the PLS tank, whereas the thickener underflow at 60% solids density (by wt.) or higher will be pumped to the secondary leaching circuit.

The key equipment in the primary leaching circuit will include:

- Three 13.7 m diameter x 13.7 m height leach tanks equipped with dual impellers,
- One 26 m high-rate thickener,
- Related feeding systems and auxiliary equipment (pumps, sump pumps, pump boxes).

17.4.1.3 Secondary Leaching and Counter-Current Decantation

The intermediate thickener underflow will be re-pulped and leached in a train of two mechanically agitated carbon steel tanks, each equipped with dual impellers, providing 24 hours of residence time for the secondary leaching stage. The pulp will flow by gravity cascading from the No. 1 secondary leach tank to the No. 2 secondary leach tank. It will be possible to bypass the tanks for maintenance.

The CCD thickener no. 1 overflow will be added to the first secondary leach tank to reduce the slurry solids density to 47% (by wt.). Cyanide solution will be added to the first leach tank to assist with leaching. Compressed air will also be sparged to all leach tanks to assist with leaching kinetics. The secondary leaching circuit discharge will be pumped to the CCD circuit to recover dissolved silver and gold. The CCD circuit will consist of two high-rate thickeners. The secondary leaching circuit discharge will be pumped to CCD thickener no. 1. CCD thickener no. 2 overflow will be recycled to CCD thickener no. 1 as washing water. The CCD thickener no. 1 overflow will be pumped to a surge tank and then pumped to the tailings repulping and primary leaching circuit. Barren solution recovered from the gold and silver recovery circuit using the Merrill-Crowe process will be added to the CCD thickener no. 2 as washing water. The CCD thickener no. 2 overflow will be pumped to CCD thickener no. 1, whereas the thickener underflow at 60% solids density (by wt.) or higher will be pumped to the tailings detoxification circuit.

The washing ratio, washing solution volume to feed tonnage, will be 3.5:1 to achieve an overall CCD washing performance efficiency of 99%.

The key equipment in the secondary leaching circuit will include:

- Two 13.7 m diameter x 13.7 m height leach tanks equipped with dual impellers,
- Two 26 m high-rate thickeners,
- One 9.1 m diameter x 9.7 m height CCD thickener overflow surge tank,
- Related feeding systems and auxiliary equipment (pumps, sump pumps, pump boxes).

17.4.1.4 Merrill Crowe Circuit

The PLS from the primary and intermediate thickeners will be treated using the Merrill-Crowe process to recover the contained precious metals by zinc-dust cementation. The barren solution will then be reused in the CCD washing circuit as a washing solution. The nominal solution feed rate to the Merrill-Crowe precipitation circuit will be approximately 320 m^3/h.

The PLS from the primary and intermediate thickener will be discharged to the PLS tank. The PLS will then be pumped into a leaf clarifier filter precoated by a diatomaceous earth filter aid to remove suspended solids. Pre-coat will be required to enhance the capture of the fine solids at the start of each cycle. At the end of the filtration cycle, the clarifying filter sludge will be pumped back to the CCD circuit via the clarifying filter sump pump to minimize any losses of precious metals in the entrained solution. The clarified solution will then be pumped to a de-aeration tower, where the solution will be deoxygenated.

The discharge from the de-aeration tower will be mixed with a slurry of zinc dust and lead nitrate in the precipitate mixing tank where the precipitation reactions occur. Cyanide can be added to the process as required to maintain adequate free cyanide for the precipitation reaction. The slurry with the gold and silver precipitates will be pumped through a pre-coated filter press, where the gold-silver precipitates and other solids will be removed. The filtrate (barren solution) will be pumped to the barren solution tank and reused as the washing water for the CCD washing circuit.

The precipitation efficiency is estimated to be approximately 98% for both silver and gold. A filter aid will be required for the leaf clarification filter and the precipitate filter press. A small amount of lead nitrate will also be added to improve the precipitation efficiency. At the end of the filtration cycle, feed pumps will be shut down, filters drained, and compressed air may be used to further dewater the cake. The filter cake, containing approximately 50% (by wt.) precious metals, will be dropped onto precipitate carts for transfer to the refining circuit.

The key equipment in the Merrill Crowe circuit will be a vendor package and include:

- One 9.1 m diameter x 10.0 m height PLS tank,
- Two rotating disk filters as clarifier filters,
- One de-aeration tower,
- Two de-aeration tower vacuum pumps,
- One zinc powder feeding system, including a hopper and a feeder,
- One lead nitrate solution preparation system, including hopper, feeder, mixing tank and associated pumps,
- Two precipitation filter press units,
- One pre-coat preparation system,
- One body feed preparation system,
- One 9.5 m diameter x 10.0 m height barren solution tank,
- Related feeding systems and auxiliary equipment (pumps, sump pumps, pump boxes).

17.4.1.5 Refining Circuit

Gold and silver precipitate from the Merrill-Crowe circuit will be further treated by smelting into gold-silver doré for sale. The refining process will be performed in a batch mode. The circuit will be in a secure enclosed area with CCTV cameras and restricted access.

The precipitate filter cakes from the Merrill-Crowe circuit will be dried and calcified at approximately 730°C. Fluxing agents will be mixed with the calcined materials prior to the smelting process, which will be conducted in an electric furnace at a temperature of approximately 1,250°C. The liquid metals will be poured into moulds to form gold-silver doré bars. The slags generated from the refining process will be retreated separately to recover residual gold and silver or be sold for the precious metal recovery.

For a healthy work environment, sufficient ventilation and off-gas handling will be provided in the gold room. Fume and dust exposure for the melting furnace and material handling will be controlled through a ventilation system installed in the gold room, including hoods, enclosures and wall fans to follow the local regulations/guidelines.

Gold-silver doré products will be stored in a dedicated safe in the gold room. Doré products will be shipped by contractors by armored transport. An inventory record book will be maintained in the gold room to record all the doré product movements into and out of the safe.

The smelting circuit will be a vendor package, and the main equipment will include:

- One 3 kW (6.9 ft^3) drying oven,
- One 10 kW (1.18 ft^3) smelting furnace,

- Flux dosing and flux mixer system,
- One gold-silver doré safe,
- Associated material handling and other systems (moulds, dryers, dust collection system).

17.4.1.6 Tailings Cyanide Detoxification

The washed leach residue slurry from the CCD washing circuit will be treated using a sulphur dioxide (SO_2)-air process to reduce the weak acid dissociable (WAD) cyanide to less than 5 ppm. The CCD thickener no. 2 underflow, with a solid density of approximately 60% (by wt.), will be diluted with process water to approximately 47% (by wt.) and pumped to the cyanide destruction tanks where sodium metabisulfite (SMBS), copper sulphate (catalyst), and lime slurry reagents will be added to reduce the WAD cyanide content to the target level. This process is expected to reduce the WAD cyanide in the tailings to less than 5 ppm. The treated tailings will be pumped to the tailings filter press stock tank. The cyanide detoxification circuit will consist of one mechanically agitated tank equipped with an air sparge device, providing a residence time of 1 hour. The cyanide detoxification circuit will be serviced by a dedicated sump pump. Any spillage within this area will be returned to the tailings stock tank feed pump box.

The cyanide detoxification circuit will include:

- One 6.1 m diameter x 5.5 m height agitated cyanide detoxification reaction tank,
- Associated material handling systems (pumps, pump boxes, sump pumps).

17.4.1.7 Tailings Filtration and Disposal

Detoxified tailings will be filtered, and the filtered solids will be impounded in an on-site storage facility.

Detoxified tailings, at approximately 47% solids (by wt.), will be pumped to an agitated filter feed tank prior to being pumped to a filtration circuit for dewatering. This tank will provide about 4 hr of surge capacity for the pressure filters. Two vertical plate pressure filters were selected for this purpose to increase the solid density of the tailings from approximately 47% (by wt.) to approximately 82% (by wt.), after which the tailings will be conveyed to the DSTMF. Filtered solids will be impounded at the designated DSTMF, located east of the process plant. Filtrate will be pumped to the process water tank for distribution throughout the process facilities. Any spillage within this area will be returned to the sump pump in the cyanide detoxification area and, in turn, will be pumped to the tailings stock tank feed pump box.

The tailings filtration circuit will include:

- One 9.1 m diameter x 8.5 m height agitated filter press stock tank,
- Two 2.0 m x 2.0 m, 124 chamber plate pressure filters,
- One 3.7 m diameter x 4.3 m high tails filter filtrate tank,
- Associated material handling systems (pumps, pump boxes, sump pumps).

17.4.2 Reagents and Consumables

Table 17-3 shows the reagents proposed for the process plant. Anti-scaling chemicals may be required to minimize scale built-up in the process water supply lines.

Table 17-3: Summary of Reagents and Annual Consumption

Reagents	Preparation Method	Use	Consumption* (t/y)
Sodium Cyanide	Received as powder in 1 t bags; mixed to 20% strength; transferred to a storage tank and dosed to the intensive leaching, cyanide leaching and elution circuits	Leaching agent	930
Lime	Received as powder in 1 t bags, mixed to 20% strength; transferred to a storage tank and dosed to the cyanide leaching and detoxification circuit.	pH control added as required	820
Flocculant	Received as powder in 25 kg bags; mixed to 0.5% storing strength; transferred to a storage tank and dosed after further inline dilution to all thickeners.	Flocculation of thickener feed	30
Sodium Hydroxide	Received as powder in 25 kg bags; mixed to 10% strength; transferred to a storage tank and dosed to the cyanide preparation circuit.	pH control in cyanide preparation	75
Zinc Powder	Received as powder in 20 kg drums. Dosed to Zn mixing cone through a feeder at a specific rate in Merrill-Crowe circuit.	Precipitation regent	130
Lead Nitrate	Received as powder in 1 t bulk bags, mixed to 10% strength, and transferred to a storage tank. Dosed directly to the cyanide leaching circuit and Merrill-Crowe circuit.	Leaching aid in cyanidation and a co-precipitation regent in the Merrill-Crowe process	60
Diatomaceous Earth	Received as powder in 25 kg bags; mixed to about 5% solution strength. Dosed to the clarifier and precipitate filters in the Merrill-Crowe circuit.	Precoat and body feed in the Merrill-Crowe circuit	250
Flux	Received as powder in bulk; mixed with calcined charges for smelting	Fusion agent	130
Sodium Metabisulfite	Received as powder in 1 t bags; mixed to 20% strength; transferred to a storage tank and dosed to the cyanide destruction circuit.	Reactant in the cyanide detoxification process	1,400
Copper Sulphate	Received as powder in 25 kg bags; mixed to 10% strength; transferred to a storage tank and dosed to the cyanide destruction circuit.	Catalyst in the cyanide detoxification process	25
Antiscalant	Delivered in liquid form in IBC totes. Dosed neat without dilution to PLS tank and process water tank	To minimize scale build-up	4

*The annual consumption is an approximate value.

A forklift with a drum handler will be used for reagent handling. Electric hoists servicing in the reagent area will lift the reagents to the respective reagent mixing area located above the mixed reagent storage area. The reagent handling system includes unloading and storage facilities, mixing, stock, transfer pumps, and feeding equipment.

The solution storage tanks will have level indicators and instrumentation to ensure that spills will not occur during normal preparation operations. Appropriate ventilation, fire, and safety protection will be provided at the facility. Material safety data sheets (MSDS) will be provided to the operating staff as a training and reference source. Each tank, reagent line, and addition point will be labelled following the Workplace Hazardous Materials Information Systems (WHMIS) standards. All operational personnel will receive WHMIS and additional training for safely handling and using the reagents.

Other main consumables include trash screen decks, pressure filter cloths and laboratory supplies. Maintenance spares for leaching, thickeners, reagent preparation, and assaying will also be provided.

17.4.3 Plant Services

17.4.3.1 Water Supply and Distribution

Fresh water and process water will be required to operate the process plant. Fresh water from the existing processing facility will be pumped to a fresh/fire water storage tank and further pumped for various application points, including reagent preparation. Process water will be made up of recovered filtrate from the pressure filters and fresh make-up water. Process water will be stored in a tank and pumped to various application points.

- Freshwater will be used primarily for the following:
- Firewater for emergency use,
- Reagent preparation,
- Dust suppression,
- Potable water supply.

The barren solution from the Merrill-Crowe circuit will be reused in the CCD circuit as washing water.

17.4.3.2 Air Supply and Distribution

Air service systems will be provided at the plant site for the following applications:

- Plant air: high-pressure air will be provided for the process plant for various maintenance by dedicated air compressors (one operating and one standby).
- Filtration and cyanide detoxification: dedicated air compressors will provide high-pressure air for pressure filters and cyanide detoxification (one operating and one standby).
- Leaching: dedicated air compressors will provide high-pressure air for primary and secondary leaching tanks (one operating and one standby).
- Instrumentation: dried and oil-free instrument air will come from the plant air compressors and be stored in a dedicated air receiver.

17.4.3.3 Instrumentation and Process Control

A distributed control system (DCS) will be designed and installed in the process plant. The process control system will consist of individual locally mounted control panels located near the equipment and a PC-based operator interface station (OIS) located in a centralized control room. The local control panels will be a point for monitoring and controlling the nearby equipment and instrumentation. Alarm annunciation will be local to the major equipment or located on the local control panel. The DCS and OIS will perform process control and data management through equipment and processing interlocking, control, alarming, trending, event logging, and report generation. In this manner, the process plant will be monitored and operated automatically from operator workstations in conjunction with control systems.

17.4.3.4 Quality Control

The existing metallurgical and assay laboratories, which service the current operations will service the tailings leaching facility as well. The laboratories will conduct daily assays for quality control and optimize process performance. The assay laboratory will provide all the routine assays for additional geological samples, leaching plant samples, and samples taken for environmental monitoring. The metallurgical laboratory will undertake all basic test work to monitor metallurgical performance and to improve the process flowsheet and efficiencies.

17.4.4 Annual Production Estimate

The processing plant will generate silver-gold doré during the proposed 9-year LOM. The annual metal production rate has been projected based on the mine plan and metallurgical performance projections. Table 17-4 provides the overall gold production at the 2,250 tpd nominal processing capacity.

Table 17-4: Projected Annual Production Estimate

Year	Ore Processed (kt)	Head Grade		Recovery		Production	
		Silver (g/t)	Gold (g/t)	Silver (%)	Gold (%)	Silver (koz)	Gold (koz)
Year 1	574.9	33.7	0.42	70.9	74.3	441.8	5. 8
Year 2	821.3	65.5	0.56	77.8	74.7	1,345.1	11.0
Year 3	821.2	34.3	0.45	72.6	74.3	658.0	8.9
Year 4	821.3	53.2	0.56	76.8	74.5	1,077.4	10.9
Year 5	821.2	74.9	0.48	79.5	76.0	1,573.4	9.7
Year 6	821.3	67.0	0.36	78.1	75.2	1,383.0	7.1
Year 7	821.3	32.9	0.42	74.8	74.1	650.4	8.2
Year 8	821.2	54.7	0.53	78.0	75.0	1,125.5	10.5
Year 9	384.9	82.7	0.42	80.0	76.1	818.5	3.9
Total LOM	**6,708.6**	**54.5**	**0.47**	**77.2**	**74.9**	**9,073.0**	**76.0**

Note: Numbers may not add due to rounding.

17.4.5 Processing Plant Staffing

Personnel requirements are developed based on operational requirements, including shift, equipment attendance, safety, training, and maintenance requirements. Average annual process plant staffing requirements are summarized in Table 17-5. The staffing is based on three 8-hour shifts per day.

Table 17-5: Processing Plant Staffing Requirements

Area	Personnel Required*
Management	8
Operations	47
Metallurgical and Assay Laboratory	4
Process Plant Maintenance	27
Total	86

*Includes cross shifts

18.0 PROJECT INFRASTRUCTURE

18.1 Site Access

The Property is easily accessible by road and is an important part of the local community from which skilled workers are available. Access is provided by Highway 40, a four-lane highway leading from Durango, past the airport and on to the city of Torreon in Coahuila. Successive turn-offs for the Property are at Francisco I Madero, Ignacio Zaragoza, and San Jose de Avino (Slim 2005d). The Avino mineral concessions are covered by a network of dirt roads which provide easy transport access between the San Gonzalo deposit and the mill at the main Avino Mine (Gunning 2009). In 2008, a 1.7 km road accessing to the San Gonzalo deposit was widened and upgraded so it would be suitable for use by the ore haul trucks and heavy equipment.

A well-established network of internal access roads exists at the Property (Figure 18-1). The primary internal access roads are used for hauling supplies and heavy equipment access while the secondary internal access roads are used by lighter vehicles for other operational activities. Figure 18-2 shows a typical section of the primary internal access roads.

Figure 18-1: Existing Road Network at Site (Tetra Tech, 2023)

Figure 18-2: Typical Internal Access Road at Site, Panoramic View (Tetra Tech, 2023)

18.2 Site General Arrangement

The proposed site general arrangement is presented in Figure 18-3. The major areas of Oxide Tailings Project facilities consist the following:

1. Plant Feed Stockpile area
2. Processing Plant area (refer to Section 17.4 for details)
3. DSTMF area (refer to Section 18.6 for details)

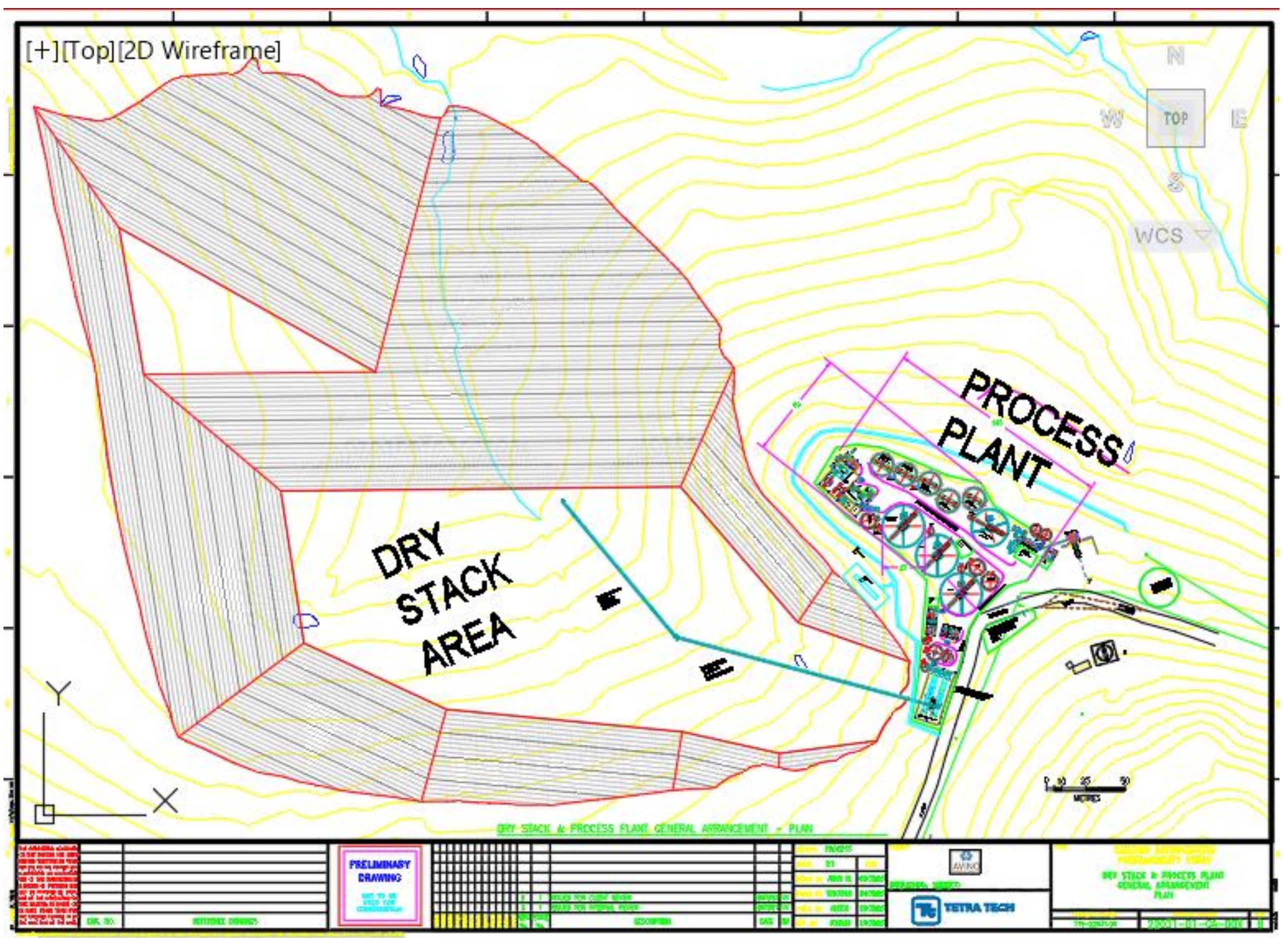

Figure 18-3: General Arrangement of Processing Plant Area and DSTMF (Tetra Tech, 2023)

18.2.1 Plant Feed Stockpile Area

The Plant Feed Stockpile area comprises a 25 m diameter (3,700 t) ROM plant feed stockpile. LHD trucks coming from the historic TMF offload the plant feed to the plant feeding dump hopper, which is reclaimed into the trommel feed conveyor hopper for feeding the processing plant, or to the plant feed stockpile. A 3.85 m^3 front end loader (FEL) transfers the plant feed from the stockpiled plant feed to the plant feeding dump hopper.

18.2.2 Processing Plant Area

The Processing Plant area consists of mineral processing related equipment, buildings, utilities, and services, which are presented in Section 17.

18.2.3 Dry Stack TMF Area

The Dry Stack TMF comprises the dry stack TMF, overland conveyors and stacking conveyors for handling the tailings material and related water management facilities, which are presented in Section 18.6.

18.3 Power

The Avino mine site was connected to the local power grid with a line capacity quoted at 4 MW when the mine last operated in 2001. With the shutdown, much of this excess power was diverted to the surrounding towns in the district. Before 2016, the existing power line provided only 1,000 kW of power, with 500 kW servicing the processing plant, 400 kW for San Gonzalo, and the balance for the well at Galeana, the employee accommodation facility, and water reclaim from the tailings dam. The San Gonzalo power line was built in 2009 to replace the contractor's diesel generator used during mine development.

According to the information provided by Avino, the current total running power available to Avino operation is 7 MW, 4.5 MW of which is being consumed by the current operation, therefore, 2.5 MW excess power is available. The new oxide tailings reprocessing operation is expected to consume 1.8 MW running power, which can be fulfilled by the existing power supply system.

A modular, prepackaged E-house will function as a power distribution center in the area. The E-house will be located close to the trommel screening circuit. Step down transformers step reduce the high voltage from the site power supply to 4,160V, 600V and 240/120V to power equipment and facilities. A 350kW emergency diesel generator provides critical back up power in the event of power outage. Sensitive electronic equipment will have surge protection and UPS.

18.4 Ancillary Facilities

Existing ancillary service facilities on site, such as administration offices, maintenance shop, warehouse, laboratories, fuel storage, can also provide support services to the OTP operations.

The Avino Property is surrounding by communities where accommodations are readily available. Staff accommodation camp on site is not required.

18.5 OTP Plant Utilities and Services

OTP plant utilities and services, such as powerlines, water lines, sewage lines and communication fiber optics, will be tied to the existing site-wide systems for use by the OTP operations. (Refer to Section 17.4.3 for details.)

18.6 Dry Stack Tailings Management Facility

The DSTMF involves dewatering of the slurry tailings using large scale pressure filters and deposition of the resulting tailings 'filter cake' in an engineered filtered DSTMF. DSTMF was selected for the Project due to the following benefits:

- A DSTMF occupies a smaller footprint compared to a conventional slurry tailings impoundment due to the higher densities achieved by dewatered tailings.
- Pressure filtration of tailings results in improved process water recycling, which reduces freshwater demand for the process plant.
- The unsaturated nature of dewatered tailings reduces infiltration through the tailings mass and into the foundation.
- Structural failures of dewatered tailings are less likely to have significant environmental impact and runout distance due to the absence of a water pond and the low pore pressures within the tailings mass.
- The dewatered tailings management option allows concurrent reclamation of the DSTMF, thus reducing potential environmental impacts, notably fugitive dust.
- The dewatered tailings management option minimizes post-closure long-term water management requirements.

The DSTMF was designed to accommodate 6.7 Mt of tailings at an assumed in-situ tailings dry density of 1.5 t/m^3 over the operational life of 9 years. The tailings to be stored on dry stack will be thickened, filtered, and transported by a series of conveyor and stacked at DSTMF, located west of the process plant. The proposed geometry and key features of the DSTMF are shown in Figure 18-6.

As shown in Figure 18-6, the proposed DSTMF is located in the valley west of the process plant. The total footprint of the DSTMF is approximately 43 acres (174,015 m^2), which is situated in close proximity to the processing plant, minimizing the distance required for tailings transport.

18.6.1 DSTMF Design Elements

The proposed DSTMF will be constructed in three sequential stages (see Figure 18-4), starting with Phase 1 from the north end of the footprint (bottom of the valley). The construction will progress southward and towards higher elevations. Stage one is designed with a capacity of 2.25 Mt and a projected final elevation of 2,203 masl. Subsequently, Stage two will be constructed on top of Stage 1, extending further south, yielding a cumulative capacity of 4.65 Mt with a projected elevation of 2,221 masl. The final stage of FSTF will be raised on top of Stage 2 and will reach the maximum projected elevation of 2,250 masl. This final stage is designed to store the ultimate capacity of 6.7 Mt of tailings, with an overall slope of 3:1 (H:V). The phased development of the DSTMF provides opportunity for progressive reclamation effort.

The DSTMF will have other associated infrastructure including seepage and contact water drainage networks, contact/stormwater collection ponds, access roads for operation, maintenance, and rehabilitation purposes.

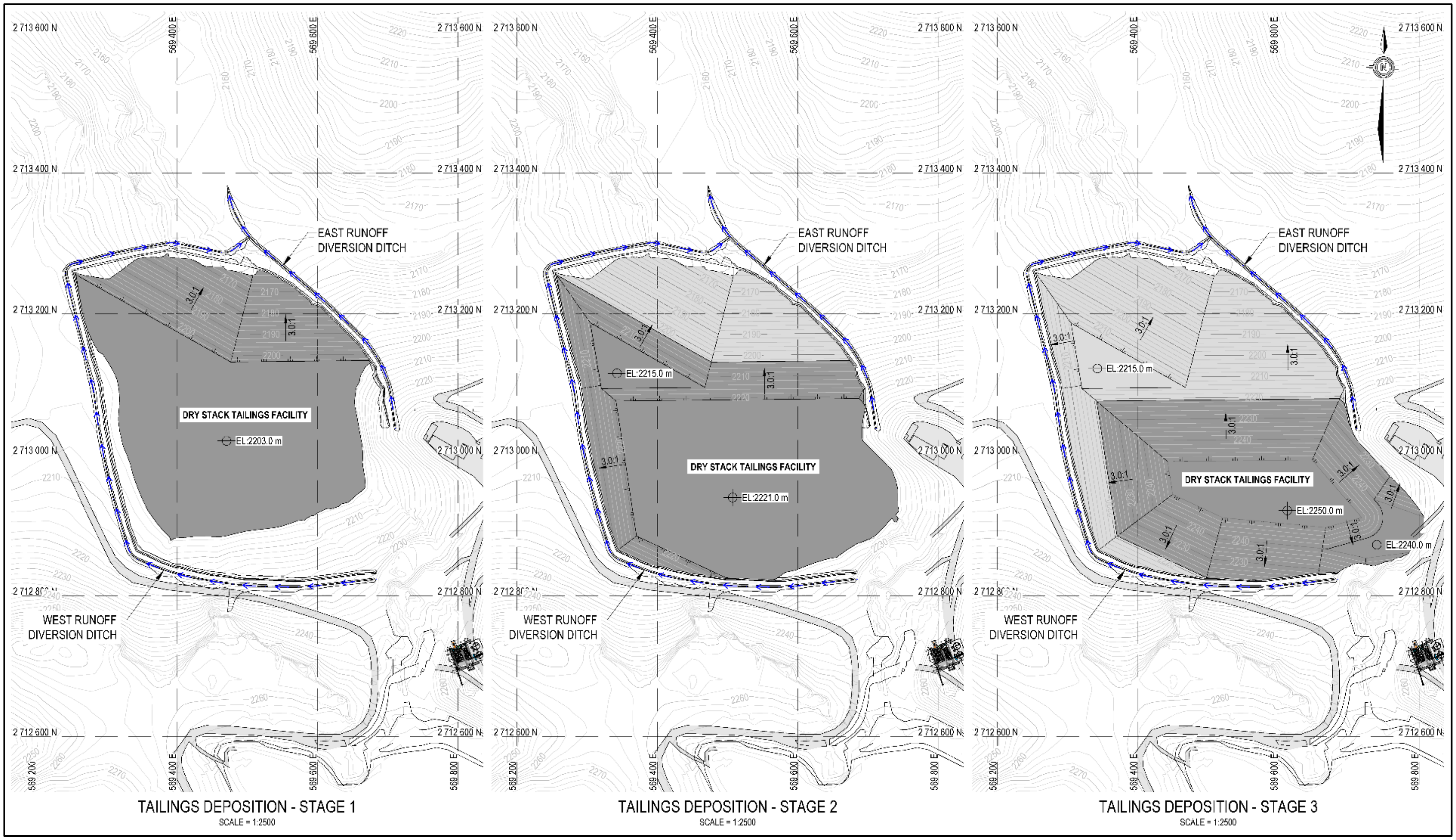

Figure 18-4: DSTMF Progression Over LOM

18.6.2 Geotechnical Analyses

Geotechnical analyses were conducted based on the proposed DSTMF design. Liquefaction potential and stability analysis were completed as part of the assessment.

18.6.2.1 Liquefaction Potential

Liquefaction refers to the sudden loss of strength, and results in failure of structure; This phenomenon can be triggered by the seismic event i.e. earthquake (cyclic liquefaction), or from an undrained soil (static liquefaction). The failure in general tends to be progressive and rapid.

The tailings at the DSTMF are considered at low risk of liquefaction under current design and operation assumptions where:

- Tailings are placed at 15% moisture content.
- Compaction of the tailings during placement with emphasis on creating a min 30 m structural zone at the stack perimeter.
- Underdrainage system at the base of the tailings stack to facilitate tailings drainage.
- Arid climate of the site and design of a surface water management system to divert water from the DSTMF.
- Overall DSTMF design slope of 3H:1V.

However, it is recommended that during the next phase of the study, a geotechnical investigation program which includes a field-testing program such as Cone Penetration Test (CPT) /Seismic Cone Penetration Test (SCPT) be implemented to understand the in-situ properties of the tailings and assess its susceptibility to liquefaction in detail.

18.6.2.2 Stability Analyses

Geotechnical stability analysis was conducted using the commercially software GeoStudio Slope-W 2021. The analysis was conducted on representative cross-section (see Figure 18-5) selected based on the slope of the proposed DSTMF and available geotechnical data. The soil material properties were derived from available geotechnical investigation data/report conducted by others and industry experience with similar materials. A summary of the material properties used in the slope stability analysis is provided in Table 18-1.

Table 18-1: Material Properties Input for Slope Stability Analysis

Material	Model	Unit Weight	Cohesion’	Phi
		kN/m^3	kPa	deg
Bedrock Foundation	Impenetrable			
HDPE Liner	Mohr-Coulomb	20	0	17-26
Compacted Tailings Liner	Mohr-Coulomb	17.8	0	38
Rockfill	Mohr-Coulomb	19.6	0	39
Dry Stack Tailings	Mohr-Coulomb	17.5	0	35

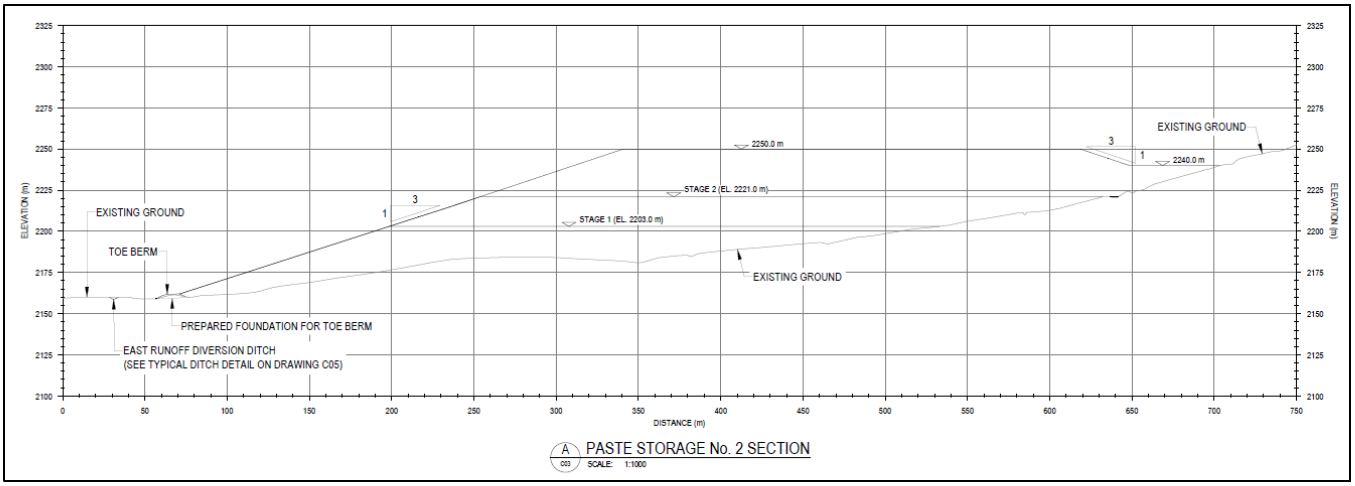

Figure 18-5: Cross Section of the Proposed DSTMF

The stability analyses were carried out for the maximum height cross-section of the DSTMF to the end-of-life stage for the DSTMF with following conditions:

- Static loading conditions
- Pseudo static loading conditions
- Post-seismic loading conditions

The proposed configurations for the DSTMF met the design criteria for stability assessment with reference to the performance criteria of *Application of Dam Safety Guidelines to Mining Dams* (CDA 2019).

18.6.3 Foundation Preparation

The foundation preparation will include clearing of vegetation, shrubs, bushes, debris, stumps. Topsoil rich in organic matter will be stored onsite in a designated area for future use during the closure activities.

Any unsuitable materials such as historic fill, soft saturated materials, and other potentially contaminated soils within the foundation area, if encountered, will be removed, and replaced with suitable competent material, as required. The DSTMF footprint will be graded, compacted sufficiently to achieve the required foundation conditions.

18.6.4 Perimeter Dyke

The initial lift of tailings stack will be contained within a perimeter dyke constructed of mine waste rocks to provide stability and erosion protection along the boundary of the DSTMF. The perimeter dyke will have a 1.5 m wide crest and height of 3 m with overall slope of 2:1 (H:V), as shown in Figure 18-6.

Rockfill for the perimeter dyke will be end dumped from mine haul trucks, spread with bulldozers to achieve design grades, and compacted to the requirements. The rock material used for construction will be hard, durable, angular, non-potentially acid generating, and suitable for end-dump construction.

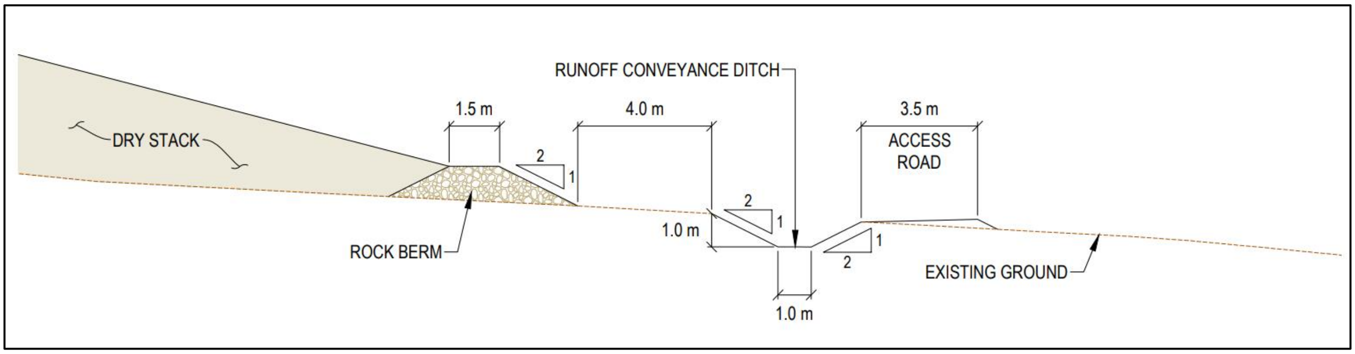

Figure 18-6: Typical Cross Section of the perimeter dyke along with run-off ditch and access road

18.6.5 Contact Water Collection System

The contact water within the DSTMF boundary such as run-off water and seepage will be captured in a lined runoff conveyance ditch outside the perimeter dyke (see Figure 18-6) and collected in the containment pond (see Figure 18-6). The containment pond, where all the contact water is collected, will be double lined with a 60 mil HDPE geomembrane and a leak detection system. The conveyance ditch will also receive seepage within DSTMF collected by the over drain system, as explained in Section 18.6.6. The contact water in the containment pond will be pumped back to the process plant for re-use, as required.

Please refer to section 18.7 site wide water management for water management outside DSTMF boundary.

18.6.6 Seepage Containment

The primary seepage containment layer for the proposed DSTMF consists of an 0.5 m thick (minimum) compacted tailings layer constructed on the prepared foundation including the inner slopes of the perimeter dyke. Vibratory smooth drum compactors will be used to achieve the required compaction in the containment layer in lifts not exceeding 150 mm in thickness after compaction.

A network of gravel over drains will be installed on top of the liner in a herringbone pattern to provide hydraulic relief. The secondary drains will consist of a drainage course enclosed within a non-woven geotextile for filtration purposes. The secondary drains will be connected to primary drains consisting of six-inch diameter perforated HDPE collector pipes enclosed within a drainage course and a heavy duty non-woven geotextile. The primary drains will be collected to the geomembrane-lined run-off conveyance ditch outside the perimeter dyke and collected in the containment pond.

18.6.7 Dry Stack Tailings Placement

Once the foundation preparation takes place, and construction of the contact water collection channels, subdrain, containment ponds, perimeter dike and seepage containment layer is completed, the facility will be ready to receive dry stack tailings. The moisture content of the filtered tailings is expected to be 15%. The geotechnical design calls for the filtered tailings to be compacted at 95% of the maximum Proctor density value ±3% of the optimum moisture content.

Filtered tailings discharged from the conveyor will be spread and levelled by bulldozer and then compacted by a smooth drum compactor. Compaction effort will create a minimum 30 m wide structural zone around the stack perimeter.

Achieving optimum moisture content and compaction density during lift placement are two main design assumptions critical to the stability of the stacked lifts and needed to be closely monitored during construction. The dry stack lifts progression plan will include:

- A stringent QA/QC program to ensure construction of the lifts meets design assumptions.
- Protocols to allow sufficient drying time of the tailings after placement, (2) manage tailings deposition during upset operations or poor weather conditions (i.e. excessive rainfall event), and (3) facilitate thin lift placement and adequate compaction of the structural zone within 30m of the perimeter.
- Geotechnical testing program to better understand in-situ properties, identify deviations, facilitate design or operational changes if required.
- Considerations to limit excess traffic on the DSTMF to limit the dust generation.

18.6.8 DSTMF Construction

The construction sequence of DSTMF is as follows:

- Foundation preparation by clearing vegetation and grading the footprint area.
- Construction of perimeter dike and contact water collection system.
- Installation of containment layer, HDPE liner, and subdrain system.
- Filtered tailings placement, grading, and compaction.
- Progressive implementation of closure plan.

18.7 Site Water Management

The Avino mine site contains an existing tailings management facility along 2712500 mN, beginning at 569500 mE and terminating just before 570500 mE. Coordinates are provided for UTM Zone 13R. This facility is situated within a small valley between two mountains to the northeast and southwest. Mine operations facilities are located on the northeastern mountain and include the processing plant and office space for site personnel.

The newly proposed tailings management facility described in Section 18.6 is a ‘dry stack’ facility, meaning the slurry from mineral extraction is dewatered prior to being layered and compacted within the facility. Additional Dewatering Tailings (DW) storage locations have been proposed to the southeast and northwest of the existing tailings management facility.

Tetra Tech has assessed the water management options for the Mine with regards to hydrologic events to provide prefeasibility-level mitigation measures where necessary to protect both existing infrastructure as well as potential expansion areas. Effort was made to separate contact from non-contact water wherever possible to reduce potential sediment loading in the proposed sedimentation ponds.

18.7.1 Hydrology

Tetra Tech applied a Gumbel distribution to a publicly available Durango City dataset from the National Centre for Environmental Information to develop an Intensity Duration Frequency (IDF) curve. This distribution provides return periods and their corresponding total 24-hour precipitation. For development of a water management plan, Tetra Tech used a 1:200-year, 24-hour precipitation event for model simulation. After accounting for climate change, this provided a total precipitation of 195.6 mm.

18.7.2 Stormwater Management Infrastructure

A comprehensive stormwater management plan has been developed for the Avino mine site to effectively manage both contact and non-contact water. This plan includes the installation of drainage swales at strategic locations to prevent the mixing of contact water with non-contact watercourses. The contributing sub-catchments for these water types were determined based on contour interpretations, derived from a DEM created from the Civil3D file "Survey Avino-2023.dxf" provided by Avino Silver & Gold Mines Ltd. This analysis was completed using Global Mapper v24. However, due to limited available information regarding the site's existing stormwater systems and drainage patterns, as well as the scope of the provided survey, the delineated watershed might not encompass the entire drainage catchment area.

The design of the site's stormwater management features incorporates several key assumptions. The identified sub-catchments were based on the DEM from the Civil3D file, and the actual drainage area is likely larger than initially estimated due to the limited extent of the survey data. The layout of proposed and existing site features, such as the tailings management facility processing plant, waste dump, and the Avino camp & hotel, were extracted from Tetra Tech's technical and mineral resource reports on the Avino Property. The capacity of the proposed drainage swales is designed solely to handle direct precipitation within the catchment areas, not accounting for external inflows like grey water, groundwater, and recycled process water.

The proposed drainage swales are presented in Figure 18-7 as yellow channels; existing, naturally formed channels are presented as pink channels; and existing smaller watercourses are presented as magenta dotted lines. These proposed drainage swales are ideally placed parallel to the outer-most roadways and around proposed/existing infrastructure which produces contact runoff. The design slope should ideally match that of the existing topography to prevent any excessive earthwork cutting and filling during the construction stage. Should the design slope be changed from what is assessed within this report, the capacity of each channel must be reassessed and adjusted to account for reductions in slope. The main purpose of these drainage swales is to capture and convey any surface contact water runoff generated within the mine site. This water can then undergo processing through sedimentation ponds prior to being released just downstream from one of the natural discharge locations, also presented in Figure 18-7.

The geometries of the proposed drainage swales have trapezoidal cross sections with a width of 0.6 – 1 m, height of 2 m, side slopes of 2H:1V, and lined with minimum Class 250 kg riprap underlaid with non-woven geotextile. Gradation and intermediate dimensions of the rock for the Class 250 kg riprap will be based on applicable federal or state laws and regulations.

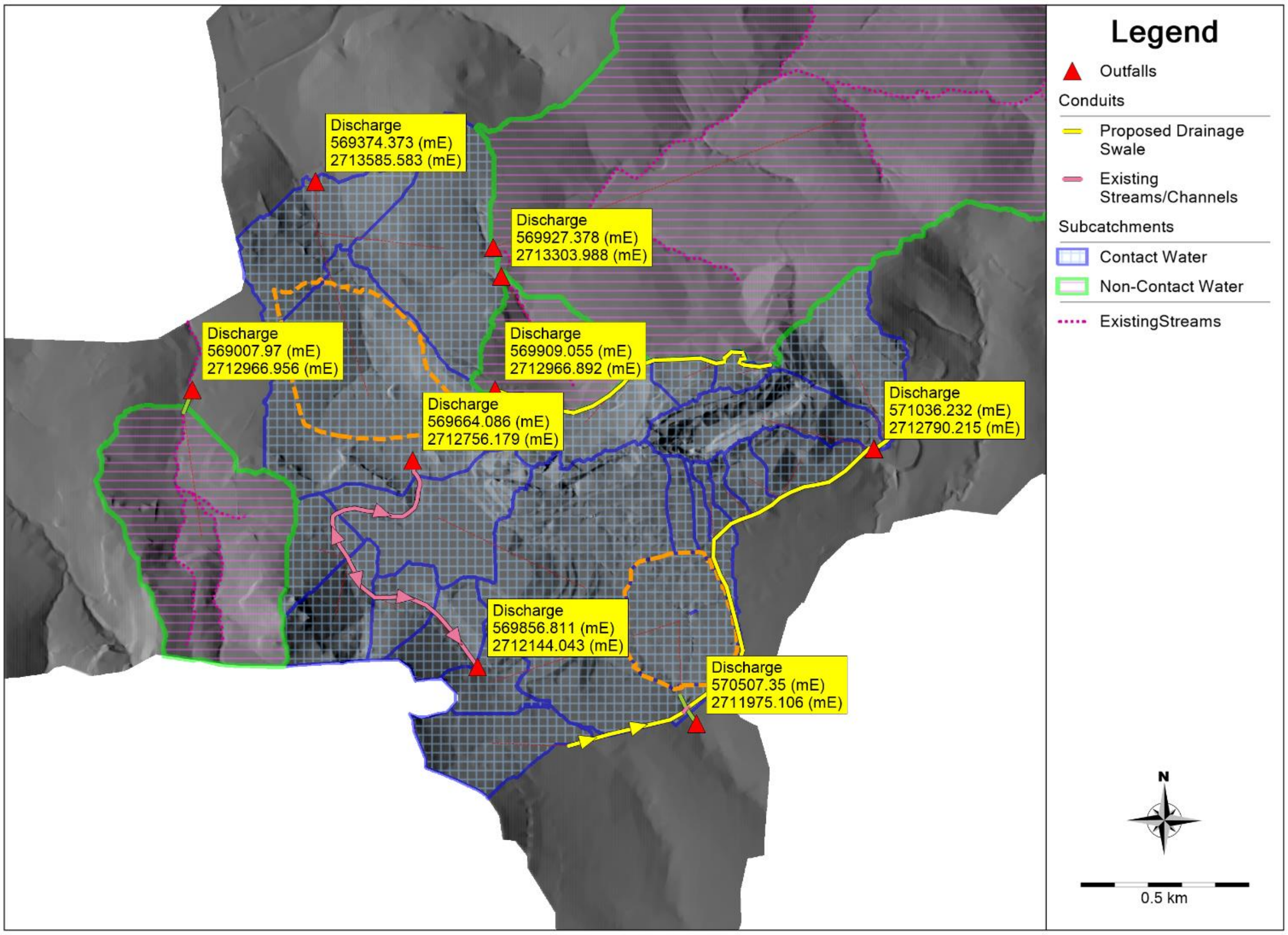

Figure 18-7: Proposed Drainage Swales (Yellow) and Existing Watercourses

In addition, water storage ponds are proposed at selective outfall locations throughout the mine site to store either contact or non-contact water and are presented in Figure 18-8 and Figure 18-9. These outfall locations represent the most downstream point/discharge location of the existing streams and the proposed drainage swales around the mine site. Dependent on whether the water being intercepted is contact or non-contact, the water storage ponds serve the following purposes:

- Contact Water: capture for onsite treatment prior to release into the natural environment; or
- Non-Contact Water: capture and prevent from entering the mine influenced area and become contaminated. Tetra Tech recommends a pump or gravity driven system to be connected to the storage pond to safely convey and discharge into a nearby natural drainage path outside of the mine-influenced area.

Additionally, corrugated steel pipe culverts are proposed at natural-stream roadway crossings. Figure 18-10, Figure 18-11, Figure 18-12 present the locations of these culverts along with the corresponding configuration to safely convey the 24-hour design storm event. The locations of these culverts are a best estimate based on available site infrastructure information at this stage. It is expected that site infrastructure will develop and change as the project progresses out of a prefeasibility stage. In any case, the culverts proposed here should be reassessed at each stage of project detail. Proposed culverts are summarized in Table 18-2.

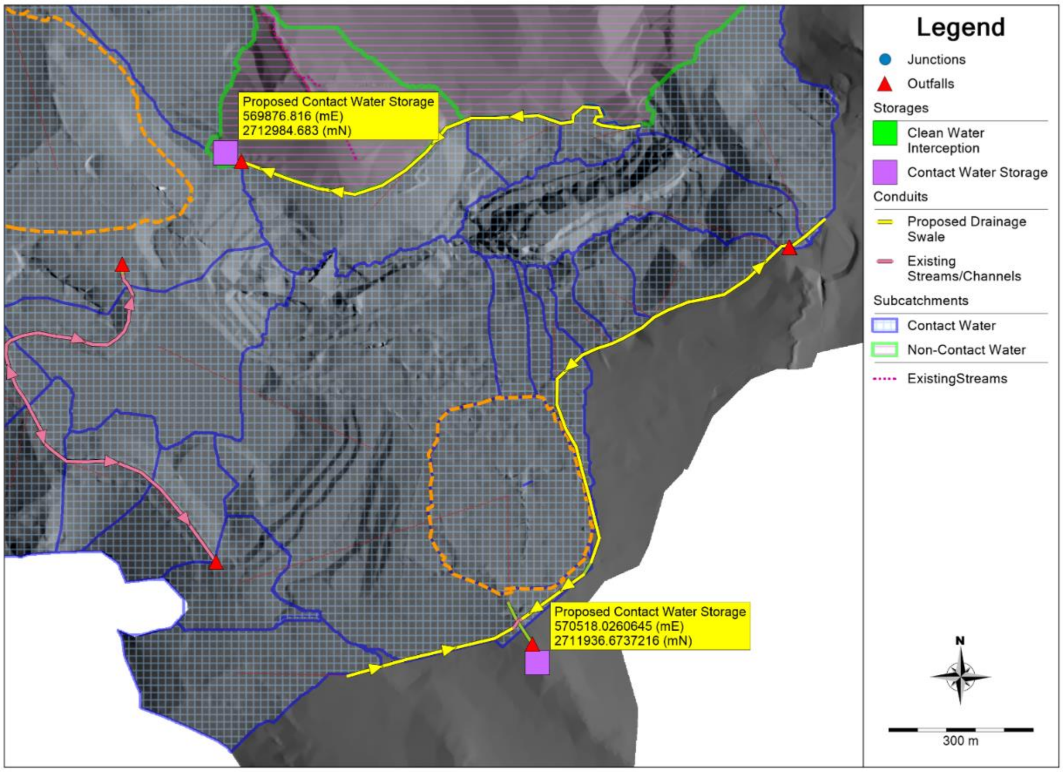

Figure 18-8: Proposed Stormwater Management Features (South)

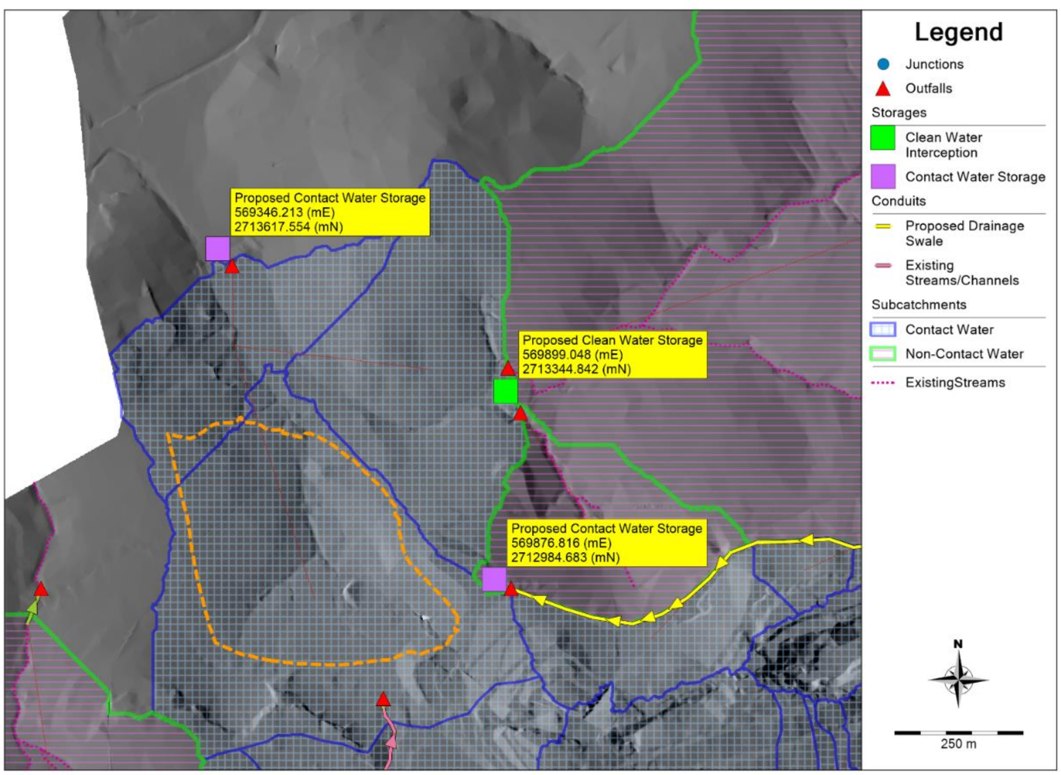

Figure 18-9: Proposed Stormwater Management Features (North)

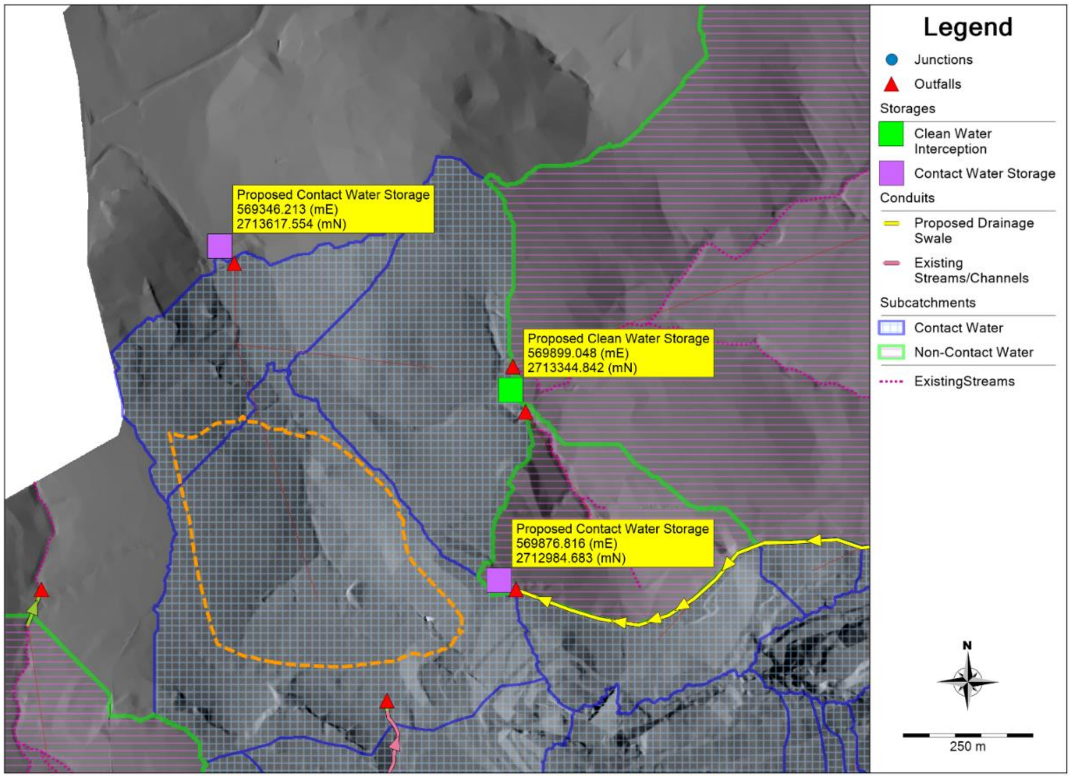

Figure 18-10:Proposed Stormwater Management Features (North)

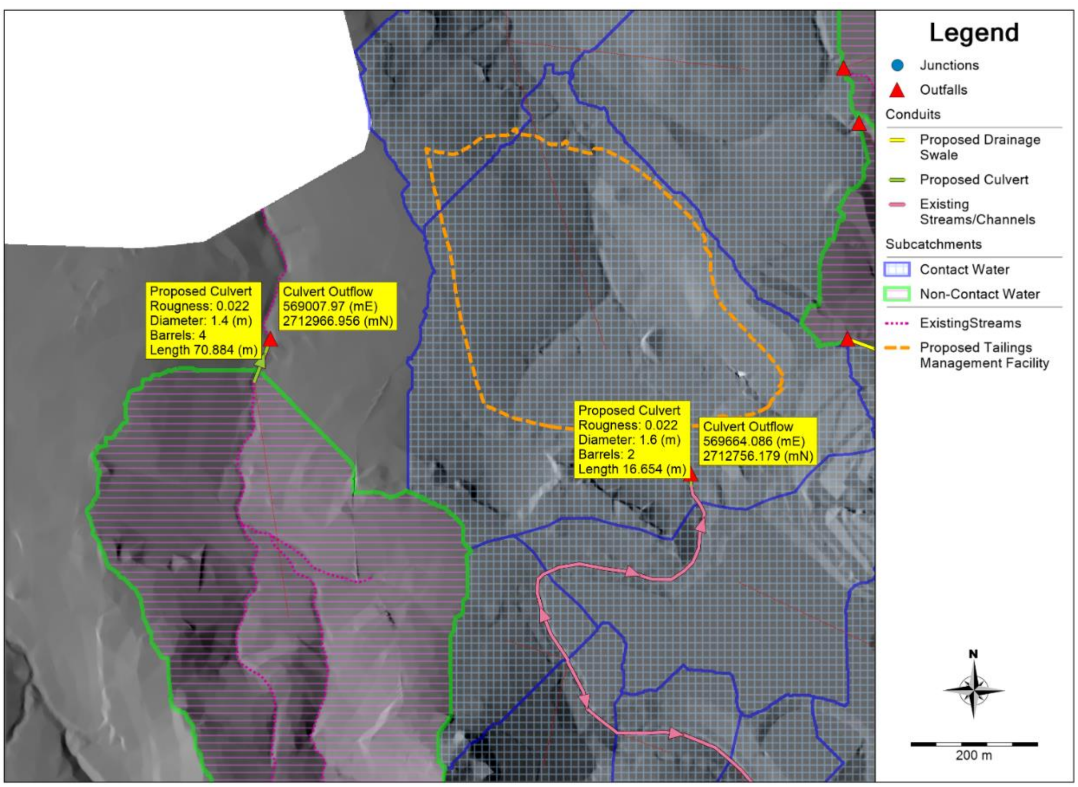

Figure 18-11: Proposed Culvert Locations (Northwest)

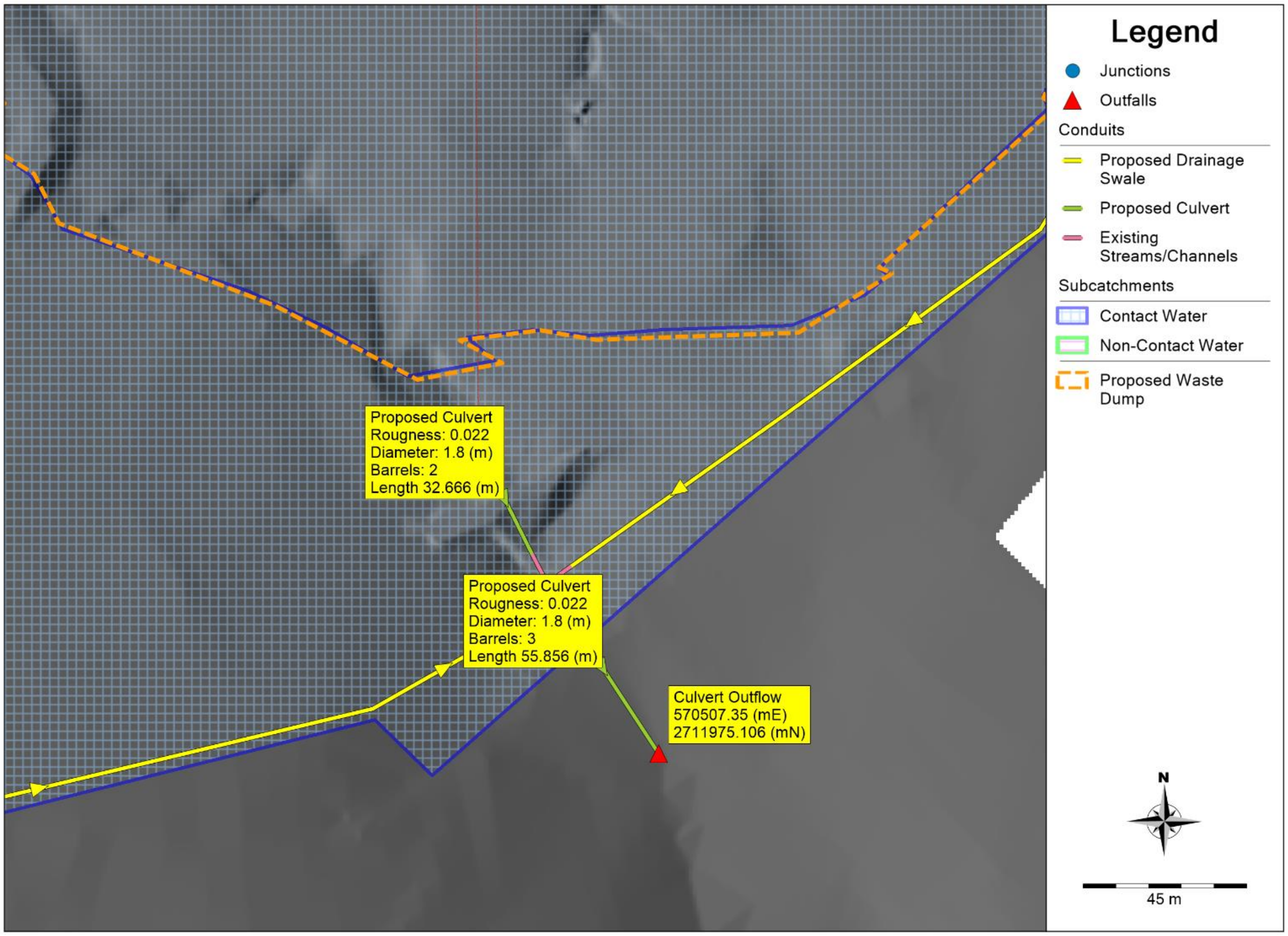

Figure 18-12: Proposed Culvert Locations (Southeast)

Table 18-2: Proposed Culvert Characteristics

Discharge Location	Manning's n	Length (m)	Barrels	Diameter (m)	Slope (m/m)
(569007.97 mE, 2712966.96 mN)	0.022	70.9	4	1.4	0.071
(569664.09 mE, 2712756.18 mN)	0.022	16.7	2	1.6	0.061
(570507.35 mE, 2711975.11 mN) (Upper Portion)	0.022	32.7	2	1.8	0.072
(570507.35 mE, 2711975.11 mN) (Bottom Portion)	0.022	55.9	3	1.8	0.030

19.0 MARKET STUDIES AND CONTRACTS

19.1 Flotation Concentrates

There is a ready market for Avino copper, silver, and gold flotation concentrates. The concentrates are currently being sold mainly to Samsung C&T UK Ltd until December 2024, however sometimes other trading firms are utilized. The concentrate sale agreement was extended in November 2018 from the original contract announced on July 9, 2015. The terms and conditions of these contracts are based on industry norms, and the terms have been used to establish the revenues from various mining operations, including the previous operation at the San Gonzalo Mine, which has now stopped operation.

Under the terms of the agreement, the concentrates are delivered by truck to the Port of Manzanillo, located on the Pacific coast of Mexico, loaded into containers, and shipped to smelters overseas.

The metal prices used for the payable metals, namely copper, silver, and gold, are based on the average market prices of the first month after the delivery to the loading port.

19.2 Gold-Silver Doré

For the doré produced from the proposed oxide tailings retreatment project, currently, there are no letters of interest or letters of intent from potential smelters or buyers of gold and silver doré. The large numbers of available gold and silver purchasers allow for the gold and silver product to be sold on a regular and predictable basis.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL COMMUNITY IMPACT

Avino has not based its production decisions on any FS or Mineral Reserves demonstrating economic and technical viability, and as a result, there is increased uncertainty and multiple technical and economic risks of failure that are associated with these production decisions. These risks, among others, include areas that would be analyzed in more detail in an FS, such as applying economic analysis to Mineral Resources and Mineral Reserves, more detailed metallurgy, and a number of specialized studies in areas such as mining and recovery methods, market analysis, and environmental and community impacts. Information in this section was provided by Avino. The information presented in subsequent sections were based on excerpts summarized from a previous technical report (Tetra Tech 2018) and revised with more recent press release material to reflect the activities at the Project site.

20.1 Avino Mine Area

20.1.1 Environmental Studies

Construction of the new TSF by depositing the tailings in a historical open pit is ongoing, and the facility is in operation now. The new TSF construction was conducted based on the recommendations in the 2013 PEA (Tetra Tech 2013), intended to advance the tailings resource towards a production decision for a Merrill-Crowe precipitation heap leach operation.

In November 2015, to get a head start on the assessment work, Avino began a program of sampling the lower oxide bench in areas not in use. The program used a hydraulic drill with a 2 m split spoon auger to drill vertical holes to a depth of 20 m to 30 m; 12 holes were drilled by the end of 2015, totalling 227 m. By the end of February 2016, 40 holes had been drilled, totalling over 650 m; assays have been received and compiled.

Avino will decommission the current TSF and begin installing wells that will be used to pump out the retained water in the dam. This will speed up the sonic drilling program planned for the upper benches, provide samples for the metallurgical program, and increase confidence in the oxide tailings resource below the sulphide tailing.

20.1.1.1 Environmental Setting

Flora and fauna of the surrounding San Gonzalo Property are anticipated to be similar to what may be found in the area of oxide tailings, although the presence of these species has not been confirmed at the oxide tailings site. Vegetation observed on the San Gonzalo Property at the time of permitting includes catclaw mimosa; cactus species, such as paddle cactus and desert Christmas cactus; needle bush, gobernadora; and persimmon trees.

Within the adjacent San Gonzalo Mine Project area, there were 15 species of major mammals, 51 species of birds, 10 species of reptiles, and 3 species of amphibians reported at the time of permitting. Of these species, 4 mammal species, 14 species of birds, 9 reptiles, and 3 amphibians species are listed by Official Mexican Standard NOM 059-SEMARNAT-2010 or in the Convention on International Trade in Endangered Species of Wild Fauna and Flora (CITES) (Ministry of Environment and Natural Resources [MENR] 2008a) (Table 20-1 to Table 20-4).

Table 20-1: Mammal Species Listed by NOM-059-SEMARNAT-2010 or in CITES within the San Gonzalo Mine (MENR 2008a)

Common Name	Gender and Species	Status under NOM-059-SEMARNAT-2001 or CITES
Abert's Squirrel	*Sciurus aberti*	Resident. Endemic. Special Protection.
White-throated Woodrat	*Neotoma albigula*	Resident. Endemic. Threatened.
Swift Fox	*Vulpex velox*	Resident. Endemic. Threatened.
American Badger	*Taxidea taxus*	Resident. Threatened.

Table 20-2: Bird Species Listed by NOM-059-SEMARNAT-2010 or in CITES within the San Gonzalo Mine (MENR 2008a)

Common Name	Gender and Species	Status under NOM-059-SEMARNAT-2001 or CITES
Great Blue Heron	*Ardea herodias*	Migratory. Special Protection.
Blue-winged Teal	*Anas discors*	Migratory. Special Protection. Hunting.
Common Black Hawk	*Buteogallus anthracinus*	Resident. Special Protection. Indicator.
Red-Tailed Hawk	*Buteo jamaicensis*	Resident. Indicator.
Great Horned Owl	*Bubo virginianus*	Resident. Threatened.
American Kestrel	*Falco sparverius*	Resident. Indicator.
Scaled Quail	*Callipepla squamata*	Resident. Endemic. Self-consume.
Blue Mockingbird	*Melanotis caerulescens*	Resident. Endemic. Threatened. Esthetic.
Northern Mockingbird	*Mimus polyglottos*	Resident. Esthetic.
Curve-billed Thrasher	*Toxostoma curvirostre*	Resident. Esthetic.
Gray Silky-flycatcher	*Ptilogonys cinereus*	Resident. Endemic.
Golden Vireo	*Vireo hypochryseus*	Resident. Endemic.
Desert Cardinal	*Cardinalis sinuatus*	Resident. Esthetic.
Painted Bunting	*Passerina ciris*	Migratory. Esthetic.

Table 20-3: Reptile Species Listed by NOM-059-SEMARNAT-2010 or in CITES within the San Gonzalo Mine (MENR 2008a)

Common Name	Gender and Species	Status under NOM-059-SEMARNAT-2001 or CITES
Mexican Spinytail Iguana	*Ctenosaura pectinata*	Resident. Endemic. Threatened.
Bolsón Night Lizard	*Xantusia bolsonae*	Resident. Endemic. Threatened.
Horrible Spiny Lizard	*Sceloporus horridus*	Resident. Endemic.
Mexican Black-bellied Garter Snake	*Nerodia melanogaster*	Resident. Endemic. Threatened.
Mexican Pine Snake	*Pituophis deppei*	Resident. Endemic. Threatened.
Mexican Garter Snake	*Thamnophis eques*	Resident. Threatened.
Western Diamondback Rattlesnake	*Crotalus atrox*	Resident. Special Protection.
Black-tailed Rattlesnake	*Crotalus molossus*	Resident. Special Protection.
Red-eared Slider	*Chrysemys scripta*	Resident. Special Protection.

Table 20-4: Amphibian Species Listed by NOM-059-SEMARNAT-2001 or in CITES within the San Gonzalo Mine (MENR 2008a)

Common Name	Gender and Species	Status under NOM-059-SEMARNAT-200 or CITES
Tarahumara Salamander	*Ambystoma rosaceum*	Resident. Endemic. Special Protection.
Sinaloa Toad	*Bufo mazatlensis*	Resident. Endemic.
Mexican Cascades Frog or White-striped Frog	*Lithobates (prev. Rana) pustulosa*	Resident. Endemic. Special Protection.

20.1.2 Environmental Permitting

Permits and authorizations required for the Project operation include:

- An Environmental Impact Assessment (EIA) by Ministry of Environment and Natural Resources
- An Environmental Risk Studies (ERA) by Ministry of Environment and Natural Resources
- An Accident Prevention Programs (APP) by Ministry of Environment and Natural Resources
- An application for surface tenures
- A wastewater discharge registration by National Commission of Water
- A hazardous waste generator's registration by National Commission of Water

To obtain authorization regarding EIA Avino must prepare an MIA-R (Manifestación de Impacto Ambiental Regional) based on articles 4°, 5° fractions II, X y XI, 15° fractions I, IV, V, VI, XII, and VXI, 28° first paragraph and fractions III and VII, 35° paragraphs first, second and last, as well as fraction I, II and 147° of the Ley General de Equilibrio Ecológico y la Protección al Ambiente (LGEEPA); 2°, 3° fractions I, XIII and XVII, 4° fraction I, VI and VII, 5° section L and O, 11° fraction III, 13°, 14°,17°, 18°, 37°, 38°, 44°, 45° first paragraph and fraction II, 47°, 48°, 49° of the regulation of the LGEEPA in matter of Environmental Impact Assessment.

Additional surface tenures will likely be required to relocate any tailings to areas outside the current surface tenure rights.

20.1.2.1 Current Permits for the Oxide Tailings

There are no current operating permits for the mining and exploitation of the oxide tailings. However, a conditionally approved Environmental Impact Statement (EIS) (Manifestación de Impacto Ambiental [MIA]) for the exploitation and associated transmission line is in place for the Avino mine site where the tailings are located. Changes to the operating methods may be required if mining of the tailings was not included in the original mining plan. Based on this information, revisions to the permits will be required. An EIA and EIS (MIA) will be mandatory if new operating permits are required.

20.1.2.2 Current Permits

To obtain authorization regarding environmental impact matters, Avino must prepare an EIS or MIA. Avino prepared an EIS, known as “Manifestación de Impacto Ambiental, modalidad Particular” (MIA-P) for the San Gonzalo Mine and submitted it to the MENR in August of 2008. The applicable regulations fall under federal jurisdiction, Article 28, sections II, III, and VII of the LGEEPA and the Reglamento en Materia de Evaluacion del Impacto Ambiental (REIA), sections K, L, and O (Environmental Impact Assessment Matter Regulation).

Given the planned activities for the site, the Ministry also required an assessment in “Environmental Impact Matter for Change of Land Use” (Materia de Impacto Ambiental para el Cambio de Uso de Suelo) for forested areas and mining infrastructure and electrification for a surface area of 9.08 ha.

The authorization from the Ministry also requires the mine to present mitigation measures for all potential environmental impacts, as per Article 30, LGEEPA and Article 44, REIA, which Avino detailed in its EIS to the authorities.

Based on the information provided by Avino to the Mexican authorities, a conditional authorization was granted, subject to additional prevention and mitigation measures to avoid, minimize, or compensate for any environmental impacts during the different stages of the adjacent San Gonzalo Mine (Article 35, section II, LGEEPA), which include an assessment of the “Environmental Impact Matter for Change of Land Use” described above. This permit is valid for 11 years from the date it was issued to perform various activities onsite. Any modification to the Project must be sent to the MENR in writing before commencing changes.

Aside from complying with all prevention, protection, control, and mitigation measures laid out in the proposed MIA-P, Avino must develop an Environmental Quality Monitoring Program (EQMP) or Programa de Seguimiento de la Calidad Ambiental. The proposed EQMP must be presented to the MENR within six months of receiving the conditional authorization. Once the MENR has assessed the monitoring program, Avino needs to deliver progress reports semi-annually for a period of at least five years. Lastly, Avino must obtain proper authorization from the MENR for “Change of Land Use” as well as the corresponding “Change of Use for Forested Ground to Mining Infrastructure”.

It is important to note that the current conditional authorization can be cancelled for many reasons; one of them includes improper disposal of liquid/solid waste (hazardous or non-hazardous).

A second permit for “Change of Forest Land Use to Mining Infrastructure” (Cambio de Utilización de Terreno Forestal a Infraestructura Minera) was requested by the SEMARNAT and granted in September of 2008 for the adjacent San Gonzalo Mine. The corresponding legislation is Article 62, section IX of the Ley General de Desarrollo Forestal Sustentable (General Law for Sustainable Forest Development) and Article 27 of the

Regulation. In addition, the Official Mexican Standards NOM-060-SEMARNAT-1994 and NOM-061-SEMARNAT-1994 must be adhered to. As per the authorization, Avino must complete its change in land use within 18 months of the date of the permit.

20.1.2.3 Applicable Legislation

To remain in compliance with current permits, the following eight applicable Official Mexican Standards for the Project must be complied with:

- Official Mexican Standards NOM-001-SEMARNAT-1996, which establishes the maximum limits allowed for contaminants in wastewater discharges in national waters and goods.
- Official Mexican Standard NOM-041-SEMARNAT-1999, which establishes the maximum limits allowed for the emission of polluted gas generated from the exhaust pipe of automotive vehicles circulating, which utilize gas as fuel.
- Official Mexican Standard NOM-043-SEMARNAT-1993, which establishes the maximum levels allowed for emissions from fixed sources of solid particles to the atmosphere.
- Official Mexican Standard NOM-045-SEMARNAT-1996, which establishes the maximum levels of emission (smokes opacity) generated from automotive vehicles circulating which utilize diesel or mixtures that include diesel as fuel.
- Official Mexican Standard NOM-052-SEMARNAT-2005, which establishes the characteristics, the process of identification, classification, and listing of hazardous waste.
- Official Mexican Standard NOM-054-SEMARNAT-1993, which establishes the procedure to determine the incompatibility between two or more types of residues considered harmful by NOM-052-SEMARNAT-2005.
- Official Mexican Standard NOM-059-SEMARNAT-2010, which regulates the environmental protection of Mexico's native species of wild flora and fauna and specifications for their inclusion, exclusion, or change-list of species in risk.
- Official Mexican Standard NOM-060-SEMARNAT-1994, which establishes protection measures for forestry grounds.

In addition, other Official Mexican Standards regarding the change in land use and mining must be followed and may include:

- Official Mexican Standard NOM-061-SEMARNAT-1994, which refers to the specifications to mitigate the adverse effects caused to Wild Animals and Uncultivated Vegetation as a result of forestry utilization, and which nomenclature was modified.
- Official Mexican Standard NOM-062-SEMARNAT-1994, which establishes specifications to mitigate adverse effects on biodiversity that are caused by the change of land use in forested areas.
- Official Mexican Standard NOM-120-SEMARNAT-1997, which establishes environmental protection specifications for mining exploration activities in dry and temperate climate regions.
- Official Mexican Standard NOM-141-SEMARNAT-2003, which establishes requirements for tailings characterization and specifications and criteria for site preparation, design, construction, operation, and post-operation of tailings dams.

Dependent on the mining plan, additional Official Mexican Standards for mining operations will also be required for the Project:

- Official Mexican Standard NOM-147-SEMARNAT/SSA1-2004, which establishes criteria for determining the concentrations of remediation of soils contaminated with arsenic, barium, beryllium, cadmium, hexavalent chromium, mercury, nickel, silver, lead, selenium, thallium, and/or vanadium; published in the Official Gazette on March 2, 2007.
- Draft Official Mexican Standard PROY-NOM-157-SEMARNAT-2009, which establishes the elements and procedures to implement management plans for mining waste.
- Draft Official Mexican Standard NOM-155-SEMARNAT-2007, which establishes environmental protection requirements for systems leaching gold and silver ores.
- General Law for the Prevention and Management of Waste (Ley General para la Prevención y Gestión Integral de los Residuos [LGPGIR]) and applicable regulations, which regulated the following registrations and authorizations:
    1. Hazardous Waste Generator's Registration and other compliance documents such as Manifest, Monthly Log of Hazardous Waste Generation; Ecological Waybills for the Importation and/or Exportation of Hazardous Materials and Wastes; Semi-annual Report on Hazardous Wastes Sent to Recycling, Treatment or Final Disposition; and Accidental Hazardous Waste Spill Manifest
    2. LGEEPA
        a. Official Mexican Standard NOM-023-STPS-2003, which establishes standards for work in mines and health and safety conditions at these sites
        b. Official Mexican Standard NOM-055-SEMARNAT-2003, which establishes the requirements to be met by sites that will use a hazardous waste landfill
        c. Official Mexican Standard NOM-147-SEMARNAT/SSA1-2004, which establishes criteria for determining the concentrations of remediation of soils contaminated by arsenic, barium, beryllium, cadmium, and chromium

20.1.3 Environmental Monitoring and Reporting

The conditional authorization sets out the requirements for environmental monitoring and reporting, on a semi-annual basis, for a minimum of five years. Details are provided in Section 20.2.

20.1.4 Environmental Management

Environmental liabilities (pasivos ambientales) of brownfields, or site recycling as it is called within the Mexican environmental legislation, are regulated by Articles 68, 69, and 70 of the LGPGIR or General Law for the Prevention and Comprehensive Management of Waste. It is based on the "polluter pay" principle, according to the LGEEPA and the LGPGIR. The federal government coordinates with provincial and municipal authorities to manage the environmental liabilities, whether the sites are orphaned or not. The LGPGIR requires a complete clean-up of contaminated sites.

20.1.5 Water Management

Fresh water for the Project is available from a well drilled in 1996, west of the mine site, and surface water from a dam, which is divided 60%/40% with the town of Panuco de Coronado. The Project has previously been charged annually for water use. Piping infrastructure from these water sources is still in place.

Additional water was also obtained from underground workings and re-circulation from the tailings thickener and tailings dam. There is potential for the water from the underground workings to be acid-producing (Slim 2005d). Treatment of water from the underground workings is ongoing prior to use, depending on the water quality.

20.1.6 Sulphide Tailings Management

ABA tests have indicated that mild acid generation may already have started on the tailings dam. A gap analysis and additional tests to further characterize the current conditions of the tailings should be completed to properly design a tailings management plan.

Three preliminary options have been identified for the management of the sulphide tailings:

- Reprocessing the tailings
- Retreating of the tailings on the heap
- Relocation and treatment for remediation

The feasibility of these options is not known at this stage.

The absence of complete sulphide tailings metallurgical information makes identification of the feasibility of the options difficult. A detailed trade-off study should be undertaken to characterize the current conditions of the tailings and to determine whether the retreatment of this material would contribute to the profitability of the Project. However, at this stage, only limited metallurgical test data is available since no detailed metallurgical test work was undertaken on this material during the MMI 2004 test program.

Alternatively, the treatment of the sulphide tailings for gold recovery will afford an opportunity to recover silver and gold from the material as well as treating this material with the lime to ensure that this material will not be a net acid producer. Indications are that the sulphide tailings will also require treatment for environmental remediation purposes in the future. These costs could be partially or completely offset by treating this material separately or together with the oxide tailings material by the heap leach process.

Relocating the sulphide tailings may afford a more expedient option to address this potential environmental problem. For this Technical Report, it will be assumed that the sulphide tailings will be moved to another location northeast of the proposed site for the leach pad.

20.1.7 Mine Closure and Reclamation

An updated mine closure plan and reclamation will be required for the Project. The mine closure plan should include information such as:

- Justification for the closure plan considering technical, environmental, and legal aspects
- Objectives and how they will be met
- Photo evidence and details of the environmental situation prior to commencing closure activities

- Schedule of activities
- The progressive reclamation of the site during the life of the operation
- The design of tailings disposal areas
- The reclamation and re-vegetation of the surface disturbances wherever practicable
- A cost estimate of the work required to close and reclaim the mine
- A plan for ongoing and post-closure monitoring and reporting at the site

No cost estimates have been generated at this time to ensure the Project meets the environmental requirements once the processing of the heap material has been terminated.

As per federal regulations, under LGEEPA, both the SEMARNAT and Procuraduría Federal de Proteccíon al Ambiente (PROFEPA) (Federal Attorney for Environmental Protection) ministries require Avino to present, in its first semi-annual report, a general plan to remediate the site with dates, activities, techniques, and costs that will guarantee restoration of affected areas, considering complete reforestation of impacted sites, removal of foundations and infrastructure that are no longer useful, roads that no longer have any use, removal and proper disposal of all rubbish, closing off adits that are no longer needed, and restoration of the TSF when its operational life is finished. Avino will also need to present a reforestation program for the entire surface area affected during mining operations. This program will include caveats to safeguard flora and fauna.

20.1.8 Socio-economic and Community Considerations

This socio-economic section of the Technical Report:

- Identifies communities that may potentially be affected by the development of the Project
- Identifies potential positive and adverse effects of the Project on local communities
- Advises on further study requirements

20.1.9 Project Location

The Project is located approximately 82 km northeast of the City of Durango, in Durango. The Property is located within the municipalities of Pánuco de Coronado and Canatlán, and is approximately 85 km by existing road, northeast of the city of Victoria de Durango.

20.1.10 Consultation with Communities

The community is currently being consulted regularly in conjunction with respect to both the dry stack tailings project and the fresh-water requirements for local agriculture. In addition, Avino provides several resources for schools and churches within the adjacent towns. A list of activities and related costs are summarized in Table 20-5 to Table 20-12.

Table 20-5: Apoyos Realizados en Zaragoza, 2017

Unit	Activities	Cost (MXN$)
-	Apoyo del dia del niño primaria y Kinder	4,010
60 pza	Costal de cemento iglesia	10,200
-	Acondicionamiento de baño presidencia	7,000
-	Tablets	16,000
-	Material 20-9ouple20-9vo secundaria	3,000
-	Material didactico educacion inicial	4,000
-	limpia de basureros	-
-	Emparejado de calles	29,000
-	**Total**	**73,210**

Table 20-6: Apoyos Realizados en Avino, 2017

Unit	Activities	Cost (MXN$)
-	Apoyo del dia del niño primaria y Kinder	9,779
-	Pintura para la iglesia	12,713
-	Regalos dia del padre	12,000
-	50% pago de recibo del energia del pozo	11,000
-	20-9ouple dresen 4"	900
-	Instalacion de soket de medidor del pozo	500
-	Tablet graduaciones	13,000
-	Bastones y sillas de ruedas	20,000
-	Emparejado de calles	29,000
-	**Total**	**108,892**

Table 20-7: Apoyos Realizados en Zaragoza, 2018

Unit	Activities	Cost (MXN$)
-	Apoyo del dia del niño primaria y Kinder	3,900
4 lata	Pintura acrilica iglesia	8,727
lote	Material par construccion de porton primaria	6,791
-	Apoyo para la polvora	3,000
-	Apoyo para dia de las madres	2,500
-	Regalos de graduacion primaria	1,700
-	Limpia de basureros	4,000
-	Brecha para alambrado para recuperacion de pastos	10,000
-	Aguinaldo posadas	3,900
-	**Total**	**44,518**

Table 20-8: Apoyos Realizados en Avino, 2018

Unit	Activities	Cost (MXN$)
-	Apoyo del dia del niño primaria y Kinder	3,900
-	Cuetes para la iglesia	1,400
-	Apoyo para el dia de las madres	2,500
-	Desmonte en area de panteon	2,000
-	Desmonte y emparejado del camino al ranchito	6,000
-	Desmonte y aplanillado del ranchito	7,000
-	Limpieza del campo de beibol	2,000
-	Mantenimiento a caminos de parcelas	18,000
-	Aguinaldo posadas	3,900
-	**Total**	**46,700**

Table 20-9: Apoyos Realizados en Panuco, 2018

Unit	Activities	Cost (MXN$)
Lote	Apoyo de $ 3,000.00 anual por héctarea de ocupación temporal para ayuda de la comunidad	180,000
Lote	Agua de la mina San Gonzalo para abrevaderos (Incluye bomba, transformador, manguera, conexiones, etc.)	188,265
-	**Total**	**368,265**

Table 20-10: Apoyos Realizados en Zaragoza, 2019

Unit	Activities	Cost (MXN$)
-	Apoyo del dia del niño primaria y Kinder	3,900
4 lata	Pintura acrilica iglesia	8,727
-	Apoyo para la polvora	3,000
-	Apoyo para dia de las madres	2,500
-	Limpia de basureros	5,000
-	Emparejado de calles	10,000
-	Apoyo para fiestas de septiembre	1,000
-	Aguinaldo posadas	3,900
-	**Total**	**38,027**

Table 20-11: Apoyos Realizados en Avino, 2019

Unit	Activities	Cost (MXN$)
-	Apoyo del dia del niño primaria y Kinder	3,900
-	Polbora para la iglesia	3,000
-	Apoyo para el dia de las madres	1,000
-	Construcion de abrebadero	40,000
-	Camisetas para equipo de basebol	2,000
-	Instalacion de linea de tuberia de 3" para agua	260,000
-	Limpieza del campo de beibol	2,000
-	Limpieza del Basurero	1,000
-	Mantenimiento a pozo de agua potable	5,000
-	Mantenimiento a caminos de parcelas	20,000
-	Refacciones para lineas de agua potable	15,000
-	Tuberia de 2" para ampliacion de red agua	15,000
-	Aguinaldo posadas	3,900
-	**Total**	**371,800**

Table 20-12: Apoyos Realizados en Panuco, 2019

Unit	Activities	Cost (MXN$)
Lote	Apoyo de $ 3,000.00 anual por héctarea de ocupación temporal para ayuda de la comunidad	180,000
-	Agunaldos para la secundaria	3,000
-	Apoyo con la Ambulacia para carrera	1,000
-	Donacion de Bomba de agua, arrancador y tuberia para abrebadero	80,000
-	Emparejado de calles y caminos	20,000
-	Recipiente para agua	50,000
-	Limpieza del Basurero	2,000
-	Reparacion de bordos de abrebaderos	66,000
-	**Total**	**402,000**

The implementation of an effective community engagement program is fundamental to the successful environmental permitting of mining projects. A comprehensive community engagement program should be initiated as soon as possible. The consultation will include addressing concerns about the heap-leach pile that may be present within or adjacent to the Property.

Consultation and development of a working relationship with local communities typically involve the development of a series of agreements that lay the groundwork for conversations. These include:

- Memorandums of understanding
- Protocol agreements
- Community consultation / participation agreements

As project exploration and development proceeds, other agreements will become necessary, including:

- Socioeconomic/community economic benefits agreements
- Environmental monitoring agreements
- Training agreements
- Accommodation/impact benefit agreements

20.1.10.1 Potential Positive Effects on Local Communities

Potential positive effects of the proposed project development include:

- Long-term, meaningful employment in mining operations and related positions (e.g., environmental monitors, service industry sector)
- Economic development and contract opportunities for local communities (existing and new businesses), and community infrastructure improvements

20.1.10.2 Potential Adverse Effects on Local Communities

For potential adverse effects of the proposed project development, it will be assumed that the sulphide tailings will be moved to another location northeast of the proposed site for the leach pad. Again, it should also be mentioned that this proposed site is very close to the town of San Jose de Avino, which may result in objections from the local community.

20.2 La Preciosa Area

The area was divided into two study areas for environmental applications: one for the mine area and one for the access, power, and waterlines. All plant and animal studies and clearances, surface and groundwater baseline data, drainage basin studies, and storm water drainage volumes and flows were defined. Local and site studies have been completed for groundwater characterization (water quality, water level, pit-inflow rates), surface water quality, and geochemical characterization of mining wastes (waste rock and tailings). Monitoring and management plans have been developed for groundwater monitoring, waste rock, tailings, prevention, and control of potential petroleum and chemical spills, sediment control plans, and tailing designs were completed based on those studies. A mine closure and reclamation plan and closure cost estimate has been prepared.

21.0 CAPITAL AND OPERATING COSTS

21.1 Avino Current Operation

Avino is currently conducting mining activity, including mineral processing, on the materials from the Avino Mine. There is no cost estimate applicable for the ongoing operations, and all costs below are based on actual expenditure, excluding the proposed tailings reprocessing project.

21.1.1 Capital Costs

The actual capital expenditures for the last three years on the Avino Vein are summarized in Table 21-3. The San Gonzalo Mine ceased its operation at the end of 2019. Mine and mill capital costs were mainly attributed to equipment purchases, construction and site upgrading.

Table 21-1: Capital Costs for the Avino Mine (US$ in 000s) (Source: Avino, 2024)

Description	2023	2022	2021
Office Furniture	78	108	31
Computer and Communication /Automation Enhancement	1,177	136	13
Mill Machinery and Processing Equipment	3,080	4,781	1,130
Mine Machinery and Transportation Equipment	3,271	2,181	1,337
Buildings and Construction	1,042	360	445
ET Mineral Property – Avino	4,827	1,649	(113)
Total Capital Costs	13,475	**9,215**	**2,843**

21.1.2 Operating Costs

The mine and milling operating costs for processing materials from the Avino Mine and historical stockpiles are summarized in Table 21-2. The costs include operating and maintenance labour together with the operation-associated consumable supplies. The cost of electrical power was included in the milling costs. The geological component was mostly related to technical labour. The San Gonzalo Mine ceased its operation at the end of 2019. As part of the ramp-up of operations, 10,806 tonnes of AHAG stockpile material were processed during Q3 2021.

Table 21-2: Operating Costs for Avino Mine (US$ in 000s) (Source: Avino, 2024)

Description	2023	2022	2021
Mining Cost	15,883	13,767	2,683
Milling Cost	10,667	7,486	1,467
Geological and Other	5,280	3,989	1,152
Royalties	1,456	1,505	403
Depletion and Depreciation	2,704	2,046	1,976
Total Direct Costs	**35,990**	**28,793**	**7,681**
G&A (General & Admin.)	7,888	7,179	5,084
Total Operating Costs	**43,878**	**35,972**	**12,765**

21.2 Oxide Tailings Project

The capital and operating costs for retreating the oxide tailings portion of the Property, including reclaiming the oxide tailings and constructing the processing plant and dry stack tailings management facility, were estimated and presented in subsequent sections.

21.2.1 Initial Capital Cost Estimate

The total estimated initial capital cost for the design, construction, installation, and commissioning of the Project is $49.1 million. This total includes all direct costs, indirect costs, owner's costs, and contingency. A breakdown of the initial capital cost is provided in Table 21-3.

Table 21-3: Initial Capital Cost Summary

Description	Cost (Million $)
Site Preparation, Excavations & Demolition	0.2
Mining Equipment	0.5
Processing Plant	26.8
TMF and Water Management	3.3
Site Services and Utilities	4.6
Total Direct Initial Capital Cost	**35.3**
Indirect Initial Capital Costs	7.8
Owner's Cost	0.7
Contingency	5.3
Total Initial Capital Cost	**49.1**

Note: Sums may not add due to rounding. Estimate Information

21.2.1.1 Class of Estimate

This Class 4 cost estimate has been prepared according to AACE International (2020) standards. The expected accuracy range of this initial capital cost estimate is ±25%.

21.2.1.2 Estimate Base Date

This estimate was prepared with a base date of Q4 2023. The estimate does not include any escalation past this date. Budget quotations were obtained for major equipment from suppliers who provided prices, delivery lead times, and spare allowances. Costing is based on in-house data and budgetary quotes for non-major equipment and construction materials. The quotations used in this estimate were obtained in Q3/Q4 2023 and are budgetary and non-binding.

21.2.1.3 Currency and Foreign Exchange

All capital costs are expressed in US dollars. No provision was made for future fluctuations in the currency exchange rates. The currency exchange rates used in the estimate are shown in Table 21-4.

Table 21-4: Currency Exchange Rates

Currency	Exchange
1.00 CAD	0.7700 US$
1.00 EUR	1.1000 US$
1.00 MXN	0.0556 US$

21.2.1.4 Measurement System

The International System of Units (SI) is used in this estimate.

21.2.2 Capital Cost Exclusions

Note that nearly all vendor quotations are budgetary and without validity periods stated in the quotes, which suggests that these quotations are not binding. These quotes may change without notice. Amid increasing uncertainty in global economics and trade, fluctuations in global commodity prices and exchange rates could cause a rapid increase in the cost of items presented in this cost estimate, which may be inevitable due to the significant reliance of the project execution upon imported services and goods. It is not unusual for commodity prices or exchange rates to fluctuate more than 5% within several trading days. The cost estimate presented herein is for information only and is not indicative of the future capital cost estimate produced for the PFS or subsequent studies.

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

- Cost escalation during construction
- Major scope changes
- Interest on loans during construction
- Schedule delays and associated costs

- Unexpected ground conditions
- Extraordinary climate events
- Labour disputes
- Receipt of information beyond the control of EPCM contractors
- Schedule recovery or acceleration.
- Financing costs
- Overtime
- Cost outside PFS battery limits
- Sunk costs
- Corporate expenses
- Research and development costs
- Permitting costs
- Working or deferred capital (included in the financial model)
- Financing costs
- Taxes and duties
- Land acquisition
- Lost time due to severe weather conditions
- Lost time due to force majeure
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resulting from a change in the project schedule
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares
- Any project sunk costs (studies, exploration programs, etc.)
- Mine reclamation costs
- Mine closure costs
- Vendor price fixing/gouging
- Macroeconomic factors
- Currency fluctuations
- Geopolitical tensions or war
- Disruptions of global supply and logistical services

- Pandemics or other natural disasters

21.2.3 Basis of Cost Estimation

21.2.3.1 Estimate Sources

For this PFS, sources of estimated total initial capital costs, including direct, indirects, Owner's costs, and contingencies, can be classified into three categories:

- Budget Quotations
- Estimated/Historical
- Allowances/Factoring

21.2.3.2 Labour Rates

A project labour rate has been estimated and applied to various areas of this PFS. The estimated project construction labour rate is US$8.50/h, including the estimated labour burdens.

21.2.3.3 Labour Productivity

Factors impacting productivity included in the buildup were the general economy, production supervision, labour relationship, job conditions, equipment, and weather. A field labour productivity factor of 1.50 was used in the cost estimate.

21.2.3.4 Person-hours / Work Week

Tetra Tech assumed the person-hours/workday to be 8 h/d for the construction labour and supervision roster.

21.2.4 Elements of Cost

The capital cost estimate consists of the four main parts: direct costs, indirect costs, Owner's costs, and contingency.

21.2.4.1 Direct Costs

Direct costs consist of completing work that is directly attributable to its performance and is necessary for completion. In construction, it includes the cost of installed equipment, material, labour and supervision directly or immediately involved in the physical construction of the permanent facilities. Examples of direct costs include mining equipment, process equipment, and permanent buildings.

The total direct cost for the Project is estimated to be $35.3 million.

Site Preparation

Overall site preparation cost is estimated to be $0.2 million and includes bulk earthworks, site preparation, excavation, and demolition of existing structures as needed. The quantities are based on the layout drawings. The earthmoving unit rates were estimated based on Tetra Tech's historic information for similar projects in the area. Fuel costs are included in the rates.

Mining Equipment

The mining equipment capital cost is estimated to be $0.5 million and provides an evaluation of the capital expenditure required to develop and excavate the existing tailings and overburden storage area and acquire the mining mobile equipment. Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, previous experience of the mining team, and benchmark data from existing operations.

Tetra Tech reviewed the existing equipment list and commissioning year provided by Avino, and determined that some of the current mining equipment could continue to be used which will offset the initial mining equipment purchase. For example, as discussed in Section 16.7.1, Tetra Tech suggests that the contractor truck fleet can continue to be employed on-site to eliminate initial equipment purchases. Costs for mining equipment include the following:

- Loading units
- Primary support equipment, including dozers, grader, water truck and mechanical service truck.
- Scheduled replacement cost when the equipment becomes obsolete or inefficient.

Processing Plant

The processing plant includes tailings rehandling, stockpile, primary and secondary leaching, thickening, Merrill-Crowe circuit, cyanide detoxification and filtration and reagent preparation/storage. The cost estimates also include overall process plant control systems.

Tetra Tech prepared a detailed equipment list with the description, size, and unit cost for each piece in accordance with the process flow sheets. Budget quotations were obtained for the major mechanical equipment based on preliminary specifications. Costing for non-major equipment is based on in-house data. Electric or hydraulic motors are itemized and priced with the equipment. The standard installation man-hour database was used to calculate the installation hours for mechanical equipment.

The processing plant capital cost is estimated to be $26.8 million. A breakdown of the processing plant cost is provided in Table 21-5.

Table 21-5: Processing Plant Capital Cost Summary

Description	Cost (Million $)
Tailings Rehandling (Reclaim & Stockpile)	0.6
Repulping and Thickening	2.6
Primary Leaching & Intermediate Thickener	4.9
Secondary Leaching & Counter Current Decantation	6.7
Merrill-Crowe Circuit	6.4
Tailings Detox/Filtration	4.2
Reagents Preparation	1.4
Total	**26.8**

Note: Sums may not add due to rounding.

Quantities for all plate work and liners for tanks, pump boxes, and chutes were estimated based on preliminary drawings and referencing historical data. Concrete quantities were determined from PFS layout drawings and experience from previous projects of a similar nature. The unit rates for concrete placement and finishing were derived from in-house data from similar projects, and the rates were cross-checked against unit rates provided by regional industrial contractors. Steel quantities are based on preliminary engineering design and general arrangement drawings. Allowances were included for cut-offs, bolts, and connections. Piping and instrumentation costs were estimated as allowances based on a factored approach. The piping estimate includes pipes, valves, fittings, hangers/supports, testing, and installation labour. The electrical cost estimate includes electrical equipment, cables, control wires, bus work, hangers/supports, termination, testing, and installation labour.

TMF and Water Management

The TMF and Water management cost is estimated to be $3.3 million and includes the tailings cake delivery system (conveyors), TMF site preparation and earthworks, including liners and proposed drainage swales, including riprap and liners and culvert crossings. The estimated construction materials and associated quantities for the TMF and water management were measured based on the embankment design shown on the relevant PFS drawings and are "neat line" estimates. The capital cost estimates for the TMF were based on these material and quantity estimates.

A breakdown of the TMF and water management is provided in Table 21-6.

Table 21-6: TMF and Water Management Capital Cost Summary

Description	Cost (Million $)
Tailings Disposal System	1.6
TMF Site Preparation and Earthworks	0.8
Site Water Management	0.9
Total	**3.3**

Note: Sums may not add due to rounding.

Site Services and Utilities

Site services and utilities cost is estimated to be $4.6 million and includes electrical substation and power distribution, water (fresh, fire, process, gland and potable) supply and distribution, and plant and instrument air.

21.2.4.2 Indirect Costs

Indirect capital costs consist of costs not directly attributable to the completion of an activity, which are typically allocated or spread across all activities on a predetermined basis. In construction, (field) indirect costs are costs that do not become a final part of the installation but which are required for the orderly completion of the installation and may include, but are not limited to, field administration, direct supervision, capital tools, start-up costs, contractor's fees, insurance, taxes, cranage and scaffolding, etc.

The indirect cost is estimated to be $7.8 million.

21.2.5 Owner's Costs

Owner's costs are assumed by the Owner to support and execute the Project. The Project execution strategy, particularly for construction management, involves the Owner working with an EPCM organization and supervising the general contractor(s). The Owner's costs include field staffing, field travel, general field expenses, community relations, and the Owner's contingency.

The total Owner's cost for the Project is estimated to be $0.7 million.

21.2.6 Contingency

When estimating costs for a project, there is always uncertainty as to the precise content of all items in the estimate, how work will be performed, what work conditions will be encountered during execution, etc. These uncertainties are risks to a project, and these risks are often referred to as "known-unknowns", which means that the estimator is aware of the risks and, based on experience, and can estimate the probable costs as such. A contingency for each activity or discipline was estimated based on the level of engineering effort, the consensus from the consultants, as well as experience on past projects.

The total Project contingency allowance is $5.3 million.

21.2.7 Sustaining Capital Costs

The sustaining capital costs are all required from Year 1 of operations to sustain the mining operation for the LOM and are estimated to be $5.1 million for the LOM, including the closure and reclamation costs. Details of the total sustaining cost are presented in Table 21-7.

Table 21-7: Sustaining Capital Costs Summary

Description	Cost (Million $)
Mining Equipment	2.0
TMF	1.5
Water Management	0.5
Total Sustaining Capital Costs	**4.0**
Closure & Reclamation	1.1
Total	**5.1**

Note: Sums may not add up due to rounding.

21.2.8 Project Operating Cost Estimate

The project operating cost estimate consists of mining, processing, tailings management, and G&A costs, which are summarized in Table 21-8. The average LOM operating cost is estimated to be $21.34/t processed.

Table 21-8: Project Average LOM Operating Cost Summary

Description	LOM Cost (million $)	Unit Cost ($/t processed)
Mining	16.2	2.41
Processing	102.7	15.31
G&A – Onsite (including Site Services)	22.2	3.31
Tailings Management	2.2	0.32
Total Operating Cost	**143.2**	**21.34**

Note: Sums may not add up due to rounding.

Project operating costs include all recurring costs for payroll, service contractors, maintenance parts and supplies, reagents, consumables, supplies, freight, personnel transportation, etc. The operating cost estimates are based on budget prices obtained in Q4 2023 and the costs from internal databases. The expected accuracy range of the operating cost estimate is ±25%.

The operations schedule is three 8-hour shifts per day.

21.2.9 Mining Operating Cost

Mine operating costs are developed from first principles. Inputs are derived from vendor quotations and historical data collected. This includes quoted cost and consumption rates for inputs such as fuel, lube, tires, undercarriage, machine parts, major components, and operating and maintenance labour ratios. Labour rates for planned hourly and salaried personnel have been supplied by Avino. A summary of the mining operating cost is presented in Table 21-9.

Table 21-9: Mining Operating Cost Summary

Description	LOM Cost (million $)	Unit Cost ($/t mined)
Fuel	3.1	0.30
Maintenance	1.6	0.16
Labour	3.9	0.37
Contract Truck	7.5	0.72
Total Mining Operating Cost	**16.2**	**1.55**

Note: Sums may not add up due to rounding.

21.2.10 Processing Operating Cost

The average processing operating cost is estimated to be $12.4 million per year, or $15.09/t processed (excluding TMF and process water management) for the 2,250 t/d nominal operation or $15.31/t processed (excluding TMF and process water management) for the LOM average. Included in this cost estimate are:

- Hourly and salaried personnel requirements and costs
- Maintenance supplies
- Reagents and operation consumables
- Electrical power consumption.

The processing cost for the 2,250 t/d nominal operation is summarized in Table 21-10.

Table 21-10: Summary of Processing Costs

Description	Unit Cost ($/t milled)	Annual Cost (Million $)
Labour	1.67	1.37
Power	1.78	1.46
Reagents Consumables	8.97	7.36
Other Consumables	0.10	0.08
Maintenance Supplies	1.49	1.23
Operating Supplies	0.36	0.30
Others	0.72	0.59
Total	**15.09**	**12.39**

Note: Sums may not add up due to rounding.

21.2.10.1 Labour Cost

Total process labour costs at the full process rate of 2,250 t/d are estimated to be $1.37 million, or $1.67/t of material processed. The salary/wage levels are based on site information provided by Avino. The payments include base salaries/labour rates and various burdens.

The projected process personnel requirement is 86 persons for a shift rotation schedule of three 8-hour shifts per day, including:

- 10 staff for management and technical support, including laboratory personnel for quality control and process optimization, excluding personnel for sample assaying
- 49 operators servicing overall operations, including personnel for sample assaying
- 27 personnel for equipment maintenance, including the maintenance management team.

21.2.10.2 Power Cost

The average annual power consumption, based on the average processing rate of 2,250 t/d of material processed, is estimated to be 11.5 GWh/a. At an average power unit cost of $0.13/kWh, the annual power cost is estimated to be $1.46 million, or $1.78/t of material processed.

21.2.10.3 Maintenance and Operating Supplies Costs

Maintenance and operating supplies are estimated to be $1.86/t of material processed. Maintenance supplies are estimated to cost $1.23 million per year or $1.49/t of material processed. Operating supplies are estimated to be $0.30 million per year or $0.36/t of material processed.

21.2.10.4 Major Processing Consumables Costs

Major consumable costs are estimated to be $7.44 million per year or $9.06/t of material processed. Major consumable unit prices were quoted from suppliers.

Major processing consumable costs include reagents and trommel screen supplies.

21.2.11 TMF Operating Cost

TMF operating costs include labour, maintenance and consumables. The TMF operating cost is estimated to be \$0.32/t material processed. A summary of the TMF operating cost is presented in Table 21-11.

Table 21-11: TMF Operating Cost Summary

Description	LOM Cost (million \$)
Fuel	0.96
Lube	0.11
Wear Parts	0.12
Labour	0.74
Total TMF Operating Cost	**2.15**

Note: Sums may not add up due to rounding.

21.2.12 General and Administrative Operating Costs

Average LOM G&A and site service operating costs are estimated at \$2.46 million per year or \$3.31/t of material processed, apart from the shared service costs with the existing operations. The operating cost distributions are:

- Labour
- Consumables
- General management
- Personnel rotational travel
- Socio-economic programs
- Communications

21.2.13 Operating Cost Estimate Exclusions

The following items are not included in the operating cost estimate:

- Pre-production
- First fills
- Closure and reclamation
- Escalation.

21.3 La Preciosa Area

Currently, there are no commercial operations on the property. Avino has not conducted capital and operating cost estimates on the property.

22.0 ECONOMIC ANALYSIS

22.1 Avino Vein – Current Operation

Avino is currently conducting mining activity, including mineral processing and concentrate production, on the materials from the Avino Vein. There is no economic analysis performed for this vein.

Avino has not based its production decisions on any FS or Mineral Reserves demonstrating economic and technical viability, and as a result, there is increased uncertainty and multiple technical and economic risks of failure that are associated with these production decisions. These risks, among others, include areas that would be analyzed in more detail in an FS, such as applying economic analysis to Mineral Resources and Mineral Reserves, more detailed metallurgy, and a number of specialized studies in areas such as mining and recovery methods, market analysis, and environmental and community impacts. Information in this section was provided by Avino.

22.2 Oxide Tailings Project

Tetra Tech prepared an economic evaluation of the Oxide Tailings Project PFS based on a pre-tax and a post-tax basis. For the 9-year mine life and 6.7 Mt Mineral Reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 35% IRR
- 2.9-year payback period
- $98 Million NPV at a 5% discount rate

Taxes and depreciation for the Project were modelled based on the inputs from tax consultants engaged by Avino. The following post-tax financial results were calculated:

- 26% IRR
- 3.5-year payback period
- $61 million NPV at a 5% discount rate

Sensitivity analyses and additional metal price scenarios were also developed to evaluate the 2023 PFS economics.

22.2.1 Forward-looking Statements

This document contains “forward-looking information” within the meaning of Canadian securities legislation and “forward-looking statements” within the United States Private Securities Litigation Reform Act of 1995. This information and these statements, referred to herein as “forward-looking statements”, are made as of the date of this document. Forward-looking statements relate to future events or future performance and reflect current estimates, predictions, expectations, or beliefs regarding future events and include, but are not limited to, statements with respect to:

- The estimated amount and grade of Mineral Reserves and Mineral Resources.

- Estimates of the capital costs of constructing mine facilities and bringing a mine into production, operating the mine, sustaining capital, and the duration of payback periods.
- The estimated amount of future production, both material processed and metal recovered.
- Estimates of operating costs, the life of mine costs, net cash flow, net present value (NPV), and economic returns from an operating mine.
- The assumptions on which the various estimates are made are reasonable.

All forward-looking statements are based on the authors' current beliefs, their various assumptions, and the information currently available to them. These assumptions are set forth throughout this Report, and some of the principal assumptions include:

- The presence of and continuity of metals at estimated grades.
- The geotechnical and metallurgical characteristics of rock conforming to sampled results.
- The water quantities and quality available during mining operations.
- The capacities and durability of various machinery and equipment.
- Anticipated mining losses and dilution.
- Metallurgical performance.
- Reasonable contingency amounts.

Although the QPs consider these assumptions reasonable based on currently available information, they may prove incorrect. Many forward-looking statements assume the correctness of other forward-looking statements, such as statements of net present value and internal rates of return, which are also based on most other forward-looking statements and assumptions herein.

By their very nature, forward-looking statements involve inherent risks and uncertainties, both general and specific, and risks exist that estimates, forecasts, projections, and other forward-looking statements may not be achieved or that assumptions do not reflect future experience.

22.2.2 Base Case Assumptions and Inputs

22.2.2.1 General

The following general assumptions and criteria form part of this analysis:

- Real 2024 US Dollars; no inflation applied.
- Two-year construction period.
- 100% equity financing.
- Discount Rate – 5%.
- Equipment salvage value, 10% of the project direct cost, has been considered at the end of mine life.

- A closure and reclamation cost of approximately $100,000/year has been used in the analysis.

22.2.2.2 Metal Pricing

The 2024 Base Case results apply the following key inputs:

- Silver – $23.45/oz
- Gold – $1,839.51/oz

22.2.3 Mine and Metal Production

The production statistics are presented shown in Table 22-1.

Table 22-1: LOM Production Statistics

Description	Unit	Year 1 to 9 (LOM)
Duration	years	9
Ore Mined	kt	6,709
Waste Mined	kt	3,701
Strip Ratio	-	0.55
Ore Processed	kt	6,709
Silver Head Grade	g/t Ag	54.5
Gold Head Grade	g/t Au	0.47
Silver Recovery	%	77.2
Gold Recovery	%	74.9
Silver Recovered	koz	9,073
Gold Recovered	koz	76

22.2.4 Payability, Transportation Cost, and Other Costs

The metal payability, estimated transportation costs and refining charges for the silver-gold doré are listed in Table 22-2.

Table 22-2: Concentrate Terms & Transportation Costs

Description	Unit	Value
Silver Payable	%	99,8
Gold Payable	%	99.8
Refining Charges	$/oz Ag Eq.	0.35
Transportation Cost	$/oz	Included in refining charges

22.2.5 Royalties

Avino advised that there are no private royalties applicable to this project. Therefore, no royalties are considered in this economic analysis.

22.2.6 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative value of the potential Project economics. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations, and, as such, the after-tax results are only approximations. The tax calculations prepared do not include opening tax pools.

Taxes and depreciation for the Project were modelled based on the inputs from tax consultants engaged by Avino.

The following assumptions were used to calculate the taxes in the economic analysis:

- The 7.5% Mining Duty and the 0.5% Special Mining Duty is deductible for income tax purposes in the year paid. The total royalties paid to the Mexican government for the life of the Project amount to $17 million.
- A 30% corporate income tax rate was utilized.
- Capital expenditures are indexed for inflation, which runs through tax depreciation. An annual inflation rate of 4% was used for analysis.

22.2.7 Working Capital

Working capital is based on 30 days of accounts receivable and 90 days of accounts payable. Working capital is reflected in the cash flow as changes in net working capital.

22.2.8 Results of Economic Analysis

Table 22-3 provides a summary of the life-of-mine cash flow.

Table 22-3: Cash Flow Summary

Description	Unit	Value (LOM)
Silver Net Revenue	M $	212.3
Gold Net Revenue	M $	139.5
Total Revenue from Sales	**M $**	**351.9**
Refining Charges	M $	(5.3)
Net Smelter Return	**M $**	**346.6**
Site Operating Cost	M $	(143.2)
EBITDA	**M $**	**203.4**
Initial Capital Cost	M $	(49.1)
Sustaining Capital Cost (inc. Closure)	M $	(5.1)

table continues…

Description	Unit	Value (LOM)
Salvage Value	M $	3.5
Pre-tax Cash Flow (undiscounted)	**M $**	**152.7**
Pre-tax Cash Flow (discounted @ 5%)	**M $**	**97.5**
Total Income Tax	M $	(35.4)
Mining and Special Mining Duty	M $	(17.0)
Total Corporate Tax	**M $**	**(52.4)**
Post-tax Cash Flow (undiscounted)	**M $**	**100.3**
Post-tax Cash Flow (discounted @ 5%)	**M $**	**60.6**

Note: Sums may not add due to rounding

The post-tax discounted annual cash flow and cumulative net cash flow are presented in Figure 22-1.

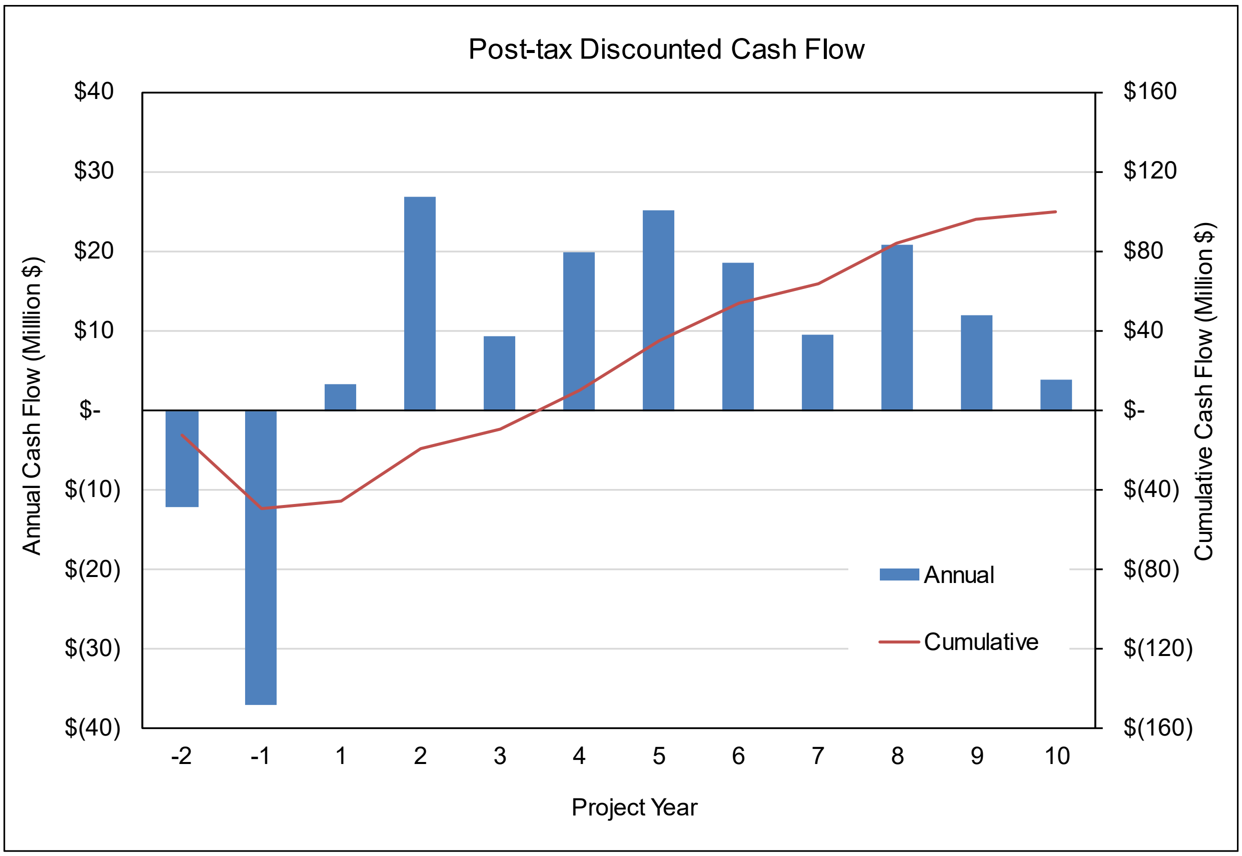

Figure 22-1: Discounted Post-Tax Annual and Cumulative Cash Flow

The financial results for the base case are presented in Table 22-4.

Table 22-4: Summary of Pre-Tax Economic Analysis

Description	Unit	Pre-tax	Post-tax
Undiscounted Net Cash Flow	Million $	152.7	100.3
NPV @ 5% Discount Rate	Million $	97.5	60.6
NPV @ 7% Discount Rate	Million $	74.5	44.1
NPV @ 10% Discount Rate	Million $	62.1	35.3
IRR	%	35	26
Payback Period	years	2.9	3.5

Table 22-5 shows the cash flow for the PFS base case.

Table 22-5: Summary of LOM Annual Cash Flow

Description	Unit	LOM	- 2	-1	1	2	3	4	5	6	7	8	9	10
Ore Processed	Mt	6,709	-	-	575	821	821	821	821	821	821	821	385	-
Feed Silver Grade	g/t Ag	54.5	-	-	33.7	65.5	34.3	53.2	74.9	67.0	32.9	54.7	82.7	-
Feed Gold Grade	g/t Au	0.47	-	-	0.42	0.56	0.45	0.56	0.48	0.36	0.42	0.53	0.42	-
Silver Recovered	koz	9,073	-	-	442	1,345	658	1,077	1,573	1,383	650	1,126	819	-
Gold Recovered	koz	76.0	-	-	5.8	11.0	8.9	10.9	9.7	7.1	8.2	10.5	3.9	-
Payability	%	99.8	-	-	99.8	99.8	99.8	99.8	99.8	99.8	99.8	99.8	99.8	-
Net Revenue from Sales	**M $**	**351.9**	**-**	**-**	**21.0**	**51.6**	**31.7**	**45.3**	**54.6**	**45.3**	**30.4**	**45.6**	**26.4**	-
Refining Charges	M $	(5.3)	-	-	(0.3)	(0.8)	(0.5)	(0.7)	(0.8)	(0.7)	(0.5)	(0.7)	(0.4)	-
Revenue (NSR)	**M $**	**346.6**	**-**	**-**	**20.6**	**50.8**	**31.3**	**44.6**	**53.8**	**44.6**	**29.9**	**44.9**	**26.0**	-
Operating Cost														
Mining	M $	(16.2)	-	-	(1.8)	(2.1)	(2.1)	(2.1)	(2.2)	(1.9)	(1.5)	(1.7)	(0.8)	-
Processing	M $	(102.7)	-	-	(9.2)	(12.4)	(12.4)	(12.4)	(12.4)	(12.4)	(12.4)	(12.4)	(6.7)	-
G&A	M $	(22.2)	-	-	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	(2.5)	-
Tailings	M $	(2.2)	-	-	(0.2)	(0.2)	(0.2)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.1)	-
Total OPEX	**M $**	**(143.2)**	**-**	**-**	**(13.7)**	**(17.2)**	**(17.2)**	**(17.2)**	**(17.3)**	**(17.0)**	**(16.7)**	**(16.8)**	**(10.2)**	**-**
EBITDA	**M $**	**203.4**	**-**	**-**	**7.0**	**33.7**	**14.1**	**27.4**	**36.5**	**27.6**	**13.2**	**28.1**	**15.8**	**-**
Capital Cost														
Initial	M $	(49.1)	(12.2)	(37.0)	-	-	-	-	-	-	-	-	-	-
Sustaining (inc. closure)	M $	(5.1)	(0.0)	(0.1)	(1.3)	(1.5)	(0.2)	(0.2)	(0.6)	(0.6)	(0.2)	(0.2)	(0.2)	(0.1)
Salvage Value	M $	3.5	-	-	-	-	-	-	-	-	-	-	-	3.5
Working Capital	M $	-	-	-	(1.7)	1.6	(1.6)	1.1	0.7	(0.7)	(1.1)	1.2	0.1	0.4
Total CAPEX	**M $**	**(50.7)**	**(12.2)**	**37.1)**	**(3.0)**	**0.1**	**(1.8)**	**0.9**	**0.1**	**(1.3)**	**(1.3)**	**1.0**	**(0.1)**	**3.8**
Cash Flow														
Undiscounted (Pre-tax)	M $	152.7	(12.2)	(37.1)	4.0	33.8	12.3	28.3	36.6	26.3	12.0	29.1	15.8	3.8
Total Taxes	M $	52.4	-	-	0.6	6.9	2.9	8.4	11.4	7.8	2.4	8.3	3.7	-
Undiscounted (Post-tax)	M $	100.3	(12.2)	(37.1)	3.3	26.9	9.4	19.9	25.2	18.6	9.5	20.8	12.1	3.8
Discounted (Post-tax)	**M $**	**60.6**	**(11.6)**	**(33.6)**	**2.9**	**22.1**	**7.4**	**14.9**	**17.9**	**12.6**	**6.2**	**12.8**	**7.1**	**2.1**

22.2.9 Sensitivity Analysis

Tetra Tech investigated the sensitivity of NPV and IRR to the key variables. Using the 2024 PFS Base Case as a reference, each key variable was changed between -30% and +30% in 10% increments while holding the other variables constant.

Sensitivity analyses were carried out on the following key variables:

- Silver price,
- Gold price,
- Capital costs,
- Operating costs.

Table 22-6 shows the economic analysis comparison results for different metals without changing the other base case parameters.

Table 22-6: Economic Result Comparison for Different Metal Prices

Metrics	Gold Price (US$/tr. oz)	Silver Price (US$/tr. oz)	Undiscounted Cashflow (Million $)	NPV @ 5% (Million $)	IRR (%)	Payback Period (Years)
Base Case	1,839.51	23.45	100.3	60.6	25.6	3.5
+30% Case - Silver Price	1,839.51	30.49	141.5	90.0	33.7	2.8
+30% Case - Gold Price	2,391.36	23.45	126.7	79.8	31.4	3.0
-30% Case - Silver Price	1,839.51	16.42	59.3	31.3	16.7	4.3
-30% Case - Gold Price	1,287.66	23.45	74.2	41.7	19.8	4.1
Spot Price*	2,055.65	23.06	108.4	66.5	27.5	3.3

*Spot price: average price on January 12, 2024.

The analyses are presented graphically as financial outcomes regarding post-tax NPV and IRR. The NPV is most sensitive to silver price, followed by capital cost, gold price and operating cost, while IRR is most sensitive to capital cost, followed by silver price, gold price and operating cost. Generally, sensitivity to metal price is roughly equivalent to sensitivity to metal grade. The NPV and IRR sensitivities are presented in Figure 22-2 and Figure 22-3, respectively.

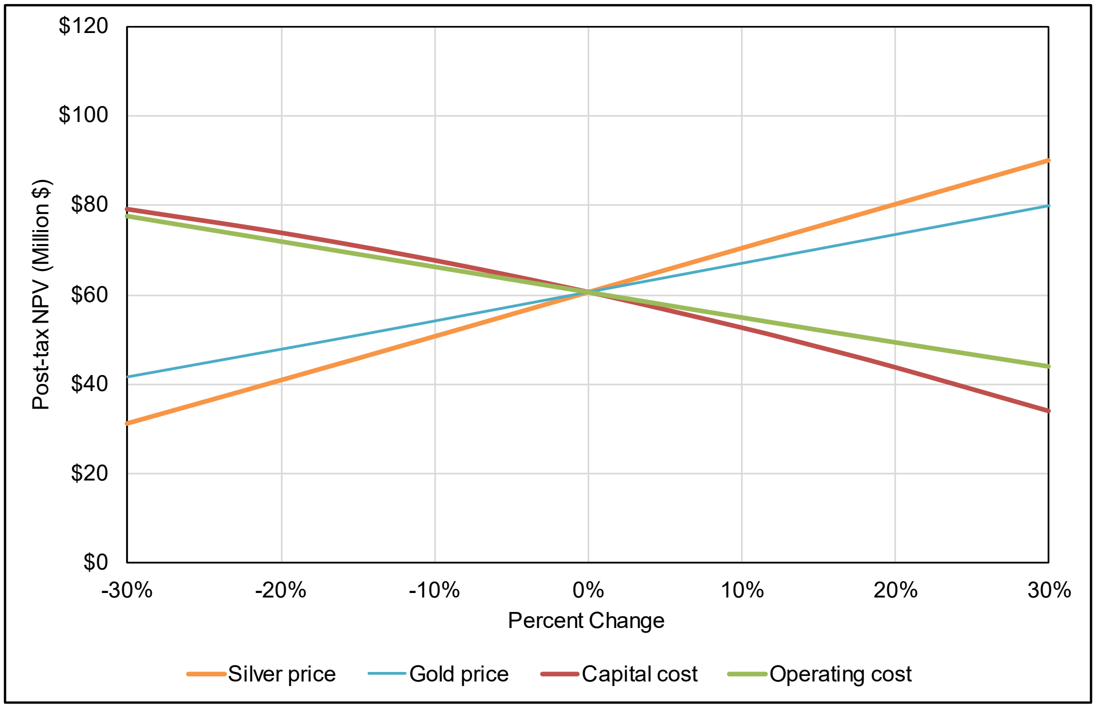

Figure 22-2: Sensitivity Analysis of Post-Tax NPV

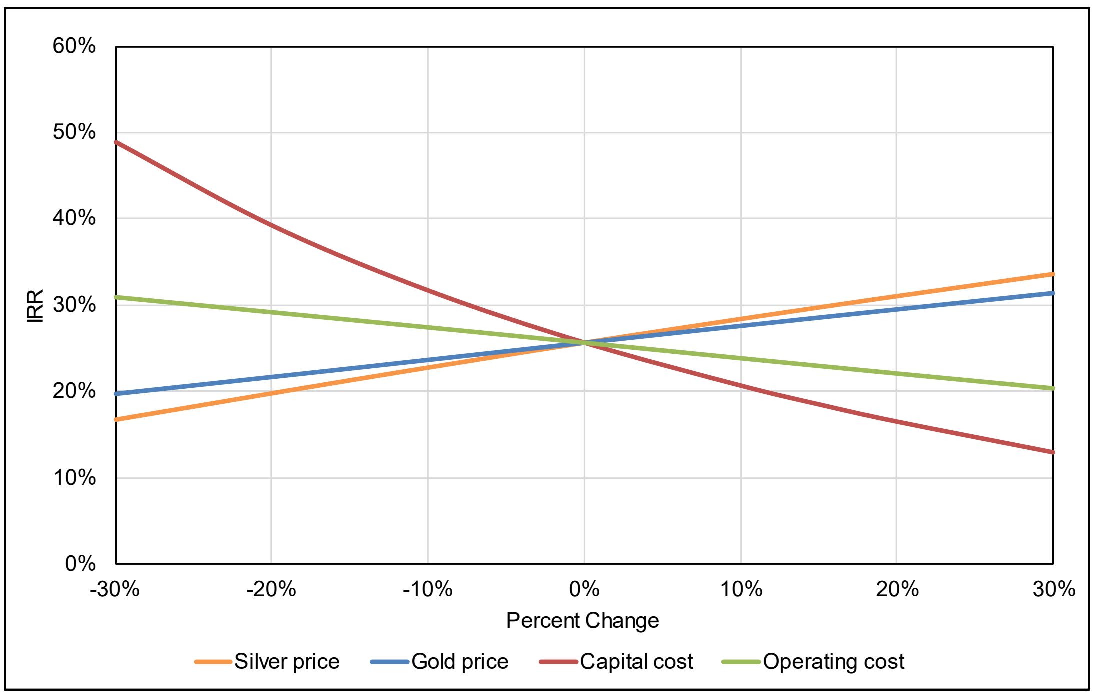

Figure 22-3: Sensitivity Analysis of Post-Tax IRR

22.3 La Preciosa Area

Currently, there are no commercial operations on the property. Avino has not conducted a project economic analysis on the property.

23.0 ADJACENT PROPERTIES

Arcelia Gold Corp., through its wholly owned subsidiary, Arcelia Gold, S.A. de C.V. owns two (2) Mining Concessions (La Peña, Título #204828 and El Niño, Título #236219) that are adjacent to and contiguous with the El Choque Tres, El Choque Cuatro, La Preciosa, and San Juan Mining Concessions of PMLP.

Canasil Resources, Inc., through its wholly owned subsidiary, Minera Canasil, S.A. de C.V. owns two (2) Mining Concessions (Carina, Título #233344 and Reducción Victoria Fracción B, Título #235845) that are adjacent to and contiguous with the San Juan Mining Concession of PMLP.

24.0 OTHER RELEVANT DATA AND INFORMATION

The Project Execution Plan (PEP) describes the strategy for executing the project engineering, procurement, construction, and commissioning phases. This includes the process plant and DSTMF. The PEP is based on the following principles:

- Promote safety in design, logistics, construction and operation for a zero-harm project implementation
- Utilizing contracted services for the procurement, construction and commissioning of critical operational systems and equipment
- Expedite factory site and process components, preassembly, modularization, testing to minimize site construction hours and hazards
- Maximize contracting opportunities for major scope components
- Advance planning and scheduling to compress the construction schedule to enhance the cashflow and return on investment

24.1 PEP Scope Outline

The project's major Work Breakdown Structure (WBS) is divided as follows:

1. Plant Feed Stockpile area comprises a 25 m diameter (3,700 t) ROM plant feed stockpile. LHD trucks coming from the historic TMF offload the plant feed at the plant feed stockpile. Plant feed is transferred from the plant feed stockpile into the trommel feed conveyor hopper for feeding the processing plant.
2. Processing Plant area consists of the primary thickening, primary leaching & intermediate thickening, secondary leaching & counter current decantation, Merrill-Crowe and tailings detox/filtration circuit to facility silver and gold extraction and production on site, support by ancillary process circuits and utilities such as reagent preparation circuit, Power substation and distribution, water services and compressed air services.
3. Dry Stack TMF area comprise the DSTMF and the material transport conveyors between tailings dewatering facilities in the process plant and dry stacking area. Contact water collected in the DSTMF and processing plant areas will be sedimented prior to being pumped to the process water tank for process use.
4. Ancillary building area consists of the administration building, maintenance shop, storage warehouse, assay laboratory and diesel fuel storage. Buildings will be equipped with power, HVAC, plumbing, sewage, fire protection and communication systems where required.

The PEP for each major WBS area is sub-divided into three project phases, from the current prefeasibility study to full production. The four phases are:

1. Feasibility study
2. Detailed engineering and procurement
3. Implementation, construction, and commissioning
4. Operations, including operations preparation, ramp-up and performance testing.

In addition, the project scope also includes the following activities that are required to support the phases above.

- Logistics planning and implementation.
- Operations preparation – to hire and train the operations team.
- Environmental & regulatory – as required by the Government local authorities
- Community and Stakeholder engagement

24.2 Execution Strategy

The primary strategy for the project is to spend the time and effort in the Engineering phase preparing in as much detail as possible to mitigate risks during the implementation phases. Once the project is entirely planned and analyzed for completeness and achievability, the project will be mobilized to the site to complete the implementation phases as rapidly as possible to minimize the costly person-days on-site. This strategy will be realized through the following approaches:

- Prefabricating and assembling process plant equipment, services and ancillary components off-site.
- Completion of all the engineering and procurement well in advance before mobilizing to site.
- Maximizing modularization of concentrator and infrastructure site components.
- Maximizing pre-assembly and commissioning of process and infrastructure components.
- Complete concentrator process wet run commissioning of all major mill equipment and process circuits before shipping to site.
- Logistics planning and scheduling all equipment and components required for each phase.
- Utilizing off-site accommodation construction and operations housing.

This strategy will minimize the project's exposure to unfavourable logistics, weather delay conditions and worker availability risks that will impact productivity on-site and delay the project.

The project also plans to maximize contracting opportunities as part of the procurement strategy. This would help the project apply its 'risk mitigation' contingency funds to the best-equipped parties to address and mitigate the risks. Accordingly, the costs are minimized with the Project Execution Team primarily fulfilling the contracts management role and the suppliers performing most of the project detail design and construction logistics themselves.

24.2.1 Project Schedule

The project construction timeline is expected to be approximately 1.5 years. Year -2 will conclude the Engineering studies and planning. The long lead items and detailed logistics planning will also be carried out in year -2. On-site surveying will be completed as required to support the Engineering work in year -2.

The major tasks comprising the critical path of the project schedule include the following:

- Process plant and DSTMF detailed engineering and design.
- Identification of vendors and development of contracts required for all concentrator, mine and infrastructure process sections

- Procurement of any long lead-time equipment
- Process equipment fabrication
- Completion of detailed engineering, design and procurement
- Logistics planning and execution.

24.3 Project Management Procedures

The Project Management team will combine the experience of Avino personnel with engineering and construction managers who will be responsible for following the PEP to complete the project on time and within budget. The project will be designed and constructed to the highest industry and regulatory body standards, emphasizing environmental and safety considerations. Avino will act as the Owner to manage suppliers and contractors.

In conjunction with Avino, the management group will develop comprehensive procedures and standards to establish the project charter and procedures manual for the execution and administration of the project. The Project Procedures Manual will outline the following:

- Project organization
- Communication matrix
- Responsibility matrix
- Reporting requirements
- Data management
- Document control
- Drawing and specification preparation, including numbering protocols, levels of issue, and transmittal procedures
- Equipment and materials procurement procedures
- Project scheduling requirements, tools, formats, and issue times
- Project accounting methods, including the cost reporting and forecasting systems
- Construction contract procedures, including bidding and awarding the work
- Site administration procedures, including camp administration rules
- Site safety
- Field engineering
- Safety procedures
- Quality assurance expectations
- Site and office personnel rules and regulations
- Emergency site procedures and contact information

- Construction of temporary facilities (power, water, and offices)
- Site housekeeping and hazardous waste management
- Mechanical completion expectations, including lock-out procedures
- Commissioning procedures outline
- Project close-out and handover procedures
- Other administrative matters and issues specific to the project

24.4 Engineering

24.4.1 Engineering Strategy

The suppliers/contracts are responsible for conducting the detailed engineering design work for specific components, including site earthworks, process equipment and DSTMF material conveying system.

24.4.2 Detailed Layout Engineering

The layout engineering provides detailed drawings and specifications for connecting utilities, including power, instrumentation, water, and pipes between different process plant components. The equipment suppliers will provide the detailed drawings for the major and minor equipment, and the Owner's team will provide the interconnections.

24.4.3 Procurement and Contracts

The Owner's Procurement team will provide capital equipment procurement and expediting services. The team will package the technical and commercial documentation and manage the bidding cycle for equipment and materials. Standard procurement terms and conditions approved for the project will be utilized for all equipment and material purchase orders (PO). Suppliers will be selected based on their expertise and experience, location, quality, price, delivery, and support service.

The Procurement team will organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, issue tenders, analyze and recommend suitably qualified contractors to Avino, and prepare executed contracts for issue.

A procurement manager from Avino will manage ongoing construction needs for miscellaneous materials and services and assist with expediting tasks. The field procurement manager will also be responsible for the receipt, storage, and disbursement of purchased materials and equipment at the job site.

24.4.4 Project Direction

The Owner's team will establish an on-site project office within proximity to the Avino Mine operations. This office will provide the project direction and house the Owner's team for the engineering and procurement phases. The office will provide direct day-to-day management of all site activities.

24.5 Construction

The construction materials and equipment required for the site activities must be marshalled, loaded, delivered and offloaded at site prior.

24.5.1 Construction Management

The site/project manager will manage all on-site and off-site construction and process equipment installation operations. Reporting to the Project Director, the execution manager will plan, organize and manage construction quality, safety, budget and schedule objectives. Construction of the project will be performed by contractors under the direction of the site manager, reporting to the Project Director. The key objectives are to:

- Conduct on-site Health, Safety and Environmental (HSE) policy training and enforcement for all site and contractor staff. Site hazard management tools and programs will achieve the no harm/zero-accident objective.
- Apply on-site and off-site contracting and construction infrastructure strategies to support the project execution requirements.
- Develop and implement a construction-sensitive and cost-effective master project schedule.
- Establish a project cost control system to ensure effective cost reporting, monitoring and forecasting and schedule reporting and control. A cost trending program will be instigated whereby the contractor will be responsible for evaluating costs on an ongoing basis for comparison to budget and forecasting for the cost report monthly.
- Establish an on-site and off-site contract administration system to effectively manage, control, and coordinate the work performed by contractors.
- Solicit tenders from the contractors and award the construction contracts to successful contractors.
- Apply an effective field constructability program to continue the constructability reviews performed in the design office.
- To develop a detailed field procurement of bulk materials, expediting, logistics, and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.

24.5.2 Field Engineering

Surveying

The Owner's team will verify the accuracy of the site-based existing control system before construction begins. The Owner's team will supervise day-to-day field surveying, and the site management team will provide spot checks. The contractor for each installation will verify surveys prior to construction.

Quality Control/Quality Assurance

Suppliers and contractors will establish and observe their own quality control program in accordance with the contractual technical specifications and applicable codes and standards.

Document Control

All on-site and off-site contractual drawings, specifications, and other documents will be electronically transferred to the field office, logged in, updated in the master stick-file, and distributed to contractors and their supervisors.

Construction Strategy

The construction contracts for the process plant facilities, process equipment and DSTMF, are expected to be awarded to separate specialized contractors in Y-2.

Construction Labor Roster

The on-site construction establishment labour schedule is based on two 12-hour shifts per day.

24.5.3 Construction Equipment

On-site construction equipment will be the responsibility of each contractor. Contractor equipment safety and operability must comply with the regulatory safety requirements. The Owner's safety personnel will perform spot checks to ensure compliance. No mobile equipment will be permitted to operate on-site unless it complies with the applicable regulations, and no cranes will be permitted to operate unless they have undergone a recent inspection. Equipment modifications must be certified fit for operation, particularly with respect to welding.

24.5.4 Communication

Avino will determine the appropriate on-site construction phase and permanent telecommunications technologies for the project, with input from the team where needed. Requirements will include voice and data link technologies to support construction and plant operation needs.

24.5.5 Construction Power

On-site construction power will be provided to the contractors as requested. The construction power will be supplied through the existing power supply and distribution network on site.

24.6 Commissioning

The contractor will be responsible for commissioning individual components with assistance from vendors' representatives. Upon completion of construction, the Owner's team will take over the custody of the process plant and will start commissioning with the assistance of the engineer, the site management team, and the contractors' labour to ensure that the plant is performing as was designed. The Owner's team will be responsible for training the Owner's personnel.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Property is located in Durango State in North Central Mexico, within the Sierra Madre Silver Belt, and 82 km northeast of Durango City. The current Property is comprised of 23 mineral concessions, totalling 1,103.934 ha.

The Property is located within a large caldera, which hosts numerous epithermal veins and breccias, grading into a "near porphyry" environment. The dominant rock types in the region of the Avino Property include andesitic, rhyolitic, and trachytic pyroclastic rocks. The area was intruded by monzonite dykes and stocks, which appear to be related to mineralization. Silver- and gold-bearing veins crosscut the various lithologies and are generally oriented north-northwest to south-southeast and northwest to southeast. The rocks have been weathered and leached in the upper sections from contact with atmospheric waters, resulting in an oxidized and reduced, or sulphide, portion of the mine.

Five deposits are present on the historic Avino Mine part of the property and have been subjected to exploration drilling, namely: the Avino breccias and veins, the San Gonzalo Vein, Guadalupe vein system, the La Potosina vein system and the tailings dam (which includes an oxide and a sulphide portion).

The Avino-acquired large area of La Preciosa concessions were extensively drilled by Coeuer and earlier operators. The flat-lying, narrow silver and gold-bearing veins in this area represent a significant part of the mineral resources.

25.2 Mineral Resource Estimates

The Mineral Resources of the Property are presented in Section 14.

25.3 Mineral Reserve Estimate

The Mineral Reserve of the Property is presented in Section 15.

25.4 Mineral Processing

Avino is currently conducting mining activities on the Avino Vein at the flotation processing plant at the Avino mine site. The feed from the Avino Vein has been processed using froth flotation to produce a copper concentrate with silver and gold credits. In the 2023 operation, the average silver, gold, and copper recoveries reporting to a silver/gold/copper concentrate and a gravity concentrate were 87%, 72%, and 83%, respectively. The total material processed was 615,373 t.

For the oxide tailings, the preliminary test results indicate that the tailings samples responded well to cyanide leaching, including column leaching treatment. The 2022-2023 test program shows the following test results with regrinding to 80% passing approximately 75 µm:

- Early-stage Oxide Tailings Composite: silver and gold extractions were improved to approximately 90.4% and 88.3% respectively, compared to 82.7% for silver and 78.7% for gold without regrinding.

- Recent Oxide Tailings Composite, silver and gold extractions were improved to approximately 83% or slightly higher respectively, compared to 77.9% for silver and 76.6% for gold without regrinding.

- Sulphide Tailings Composite: silver and gold extractions were improved to approximately 76.1% and 82.8% respectively, compared to 69.1% for silver and 77.0% for gold without regrinding.

According to the tests results, the existing tailings will be processed by tank cyanide leaching, followed by the Merrill-Crowe process to recover silver and gold. The residual material will be detoxified, filtered, and deposited in a lined dry stack tailings management facility.

The proposed processing plant has been designed to process the tailings from the historic oxide tailings at a nominal throughput of 2,250 t/d, producing silver-gold doré. The LOM average plant feed grade is estimated to be 54.5 g/t silver and 0.47 g/t gold. The LOM average silver and gold recovery is estimated to be 77.2% and 74.9%, respectively.

25.5 Mining

Avino is currently conducting mining activity on the Avino Vein. Sublevel stopping mining method is used to feed the processing plant. Mining activities at the San Gonzalo Vein reached the end of its current resources, and underground mining activities at the mine were stopped. However, the mine remains open for continued exploration at different levels of the mine (more information can be found in Section 16).

25.5.1 Avino Oxide Tailings Project

The Avino oxide tailings project will be extracted using conventional surface mining techniques with excavator, wheel loader, and existing contractor truck fleet on site. Five cashflow-positive mining pushbacks or phases were designed to allow for operational flexibility while stripping the overburden and targeting high-grade material. The mine life of the tailings deposit is expected to be approximately 9 years. Over the life of the mine, saturated ground condition is expected as the mining benches are advanced deeper. Numerous practical approaches were considered to address mining equipment trafficability challenges and reduce risks from geotechnical stability. Waste material or overburden material will be placed in dedicated facilities located near the mine.

25.6 Infrastructure

The Avino Property is currently in operation. The Property and sites within the Property are accessible by well-established network of internal and external access roads. On-site utilities and services such as water and power, exist and are readily available for use. Based on the information provided by Avino, the existing power supply system has sufficient capacity to provide the 1.8 MW running power required for OTP. The existing site-wide water system can be extended to provide water to the OTP area.

A new dry stack tailings management facility will be established to store dewatered tailings produced by OTP.

Water management features, such as water diversion ditches and contact water collection channels, will be constructed to manage water flows in the OTP area.

25.7 Capital and Operating Costs

The total estimated initial capital cost for the design, construction, installation, and commissioning of OTP is US$49.1 million. This total includes all direct costs, indirect costs, owner's costs, and contingency.

The sustaining capital costs are all required from Year 1 of operations to sustain the mining operation for the LOM and are estimated to be US$5.1 million for the LOM

The average LOM operating cost is estimated to be US$21.34/t processed. The LOM mining operating cost is estimated to be US$2.41/t processed. The average nominal processing operating cost is estimated to be $12.4 million per year, or $15.09/t processed (excluding TMF) for the 2,250 t/d nominal operation or $15.31/t processed (excluding TMF) for the LOM average. Apart from the costs for the existing operations, the LOM additional cost for G&A and site services for the tailings reprocessing operation is expected to be approximately $2.46 million per year or $3.31/t of material processed.

25.8 Economic Analysis

Tetra Tech prepared an economic evaluation of the Oxide Tailings Project PFS based on a pre-tax and a post-tax basis. For the 9-year mine life and 6.7 Mt Mineral Reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 35% IRR
- 2.9-year payback period
- $98 Million NPV at a 5% discount rate

Taxes and depreciation for the Project were modelled based on the inputs from tax consultants engaged by Avino. The following post-tax financial results were calculated:

- 26% IRR
- 3.5-year payback period
- $61 million NPV at a 5% discount rate

Sensitivity analyses and additional metal price scenarios were also developed to evaluate the 2023 PFS economics.

26.0 RECOMMENDATIONS

26.1 Geology

The QP's recommendations are itemized in the following subsections. The estimated cost for the proposed future work (drilling and remote sensing) is estimated to be approximately Cdn$2,000,000.

26.1.1 Density Sampling and Analysis

The QP recommends that Avino continues to develop the database for bulk density data using drill cores. The QP also recommends that grab samples from controlled Avino Mine underground exposures (location and lithology description) be used to supplement the data. The QP further recommends that some large samples be cut from the faces of the oxide tailings deposit, weighed, and measured to determine the bulk density of the deposit.

26.1.2 QA/QC Sampling

The QP recommends that standards and blank submissions be included in the master database for the Property to avoid the difficulty of locating such data when it resides in separate spreadsheet reports.

The QP recommends that QA/QC performance graphs be updated monthly to allow questionable sample batches to be repeated timeously.

26.1.3 Avino Deep Drilling

The QP recommends that further drilling (8,000 m) be carried out to explore the deep extension of the Avino breccias below 17 level at Elena Tolosa. This drilling can be carried out from a prospect crosscut in the hanging wall of the Avino breccias on 17 level or as long hole drilling from surface. Drilling and sampling costs for this work are estimated at Cdn$1,950,000.

26.1.4 Exploration Optimization

- The recent consolidation of the two parts of the property, namely the historic Avino Mine and the large la Preciosa area, provide an opportunity to optimize exploration effort.
- The large size of the La Preciosa concessions and the flat-lying veins and recent advancements of remote sensing techniques, suggest that geophysical techniques and multi-spectral techniques may be applied to the consolidated property with the following advantages:
- Improved spectral resolution: Hyperspectral imaging can detect subtle variations in the electromagnetic spectrum, allowing for highly detailed and accurate identification of minerals and metals. This improved spectral resolution can reveal even trace amounts of minerals, making it a powerful tool for exploration.
- Enhanced mapping capabilities: Hyperspectral techniques can generate highly detailed maps of mineral and metal distribution, helping to identify new deposits and locate areas of high mineral potential.
- Non-destructive testing: Hyperspectral techniques are non-destructive, meaning that samples do not need to be physically altered or destroyed in order to analyze their composition. This allows for more efficient and cost-effective exploration.

- Rapid data acquisition: Hyperspectral imaging can cover large areas in a relatively short amount of time, making it possible to collect vast amounts of data quickly and efficiently. This allows for more comprehensive and accurate assessments of mineral resources.
- Reduced environmental impact: Because hyperspectral techniques are non-destructive, they can be used without causing significant environmental damage or disruption. This makes them a more sustainable and environmentally friendly option for mineral exploration.
- It is recommended that a suitable specialist consultant be engaged to carry out remote sensing and hyperspectral work to identify key geological features such as alteration, faulting and sub-cropping volcanic centres that can be followed up by drilling. A budget of Cdn$50,000 should be sufficient for hyperspectral work.

26.2 Mining

Tetra Tech recommends that Avino prepare a long-term mine plan for the Avino area mineral resources based on the resource estimate. Mining methods for the historical tailings extraction and for the La Preciosa deposit should be developed and assessed.

The estimated cost for the mine plan development is estimated to be approximately Cdn$300,000.

26.2.1 Avino Oxide Tailings Project

- Up on completion of future grade control drilling or infill drilling, the mining block model should be updated.
- Further investigation if dewatering wells are required to better facilitate mining operations.
- Geotechnical slope:
  - Additional geotechnical investigation and laboratory testing of the oxidized tailings from the historic TMF should be carried out as the excavation progresses.
  - Dewatering measures should be evaluated and installed, to allow sufficient lowering of the phreatic surface prior to excavation.
  - The stability of the historic TMF should be monitored, particularly during the initial production years, and the data will be assessed and results incorporated into future tailings excavation design.
  - The liquefaction assessment should be undertaken as required with consideration of updated information on material properties and updated excavation plan.

The estimated cost for the above mentioned items is estimated to be approximately Cdn$300,000.

26.3 Metallurgy and Process

26.3.1 Avino Vein

Avino is currently conducting mining activities on the Avino Vein, ET Mine, including metal recovery using a flotation process. Tetra Tech recommends that Avino further optimize the processing conditions, including metallurgical tests, to improve metallurgical performances for the Avino Vein plant feeds. Further metallurgical test work should be conducted to investigate separating bismuth from copper-gold-silver concentrate. The bismuth removal should reduce the impurity penalty; however, a trade-off study should also be required to assess copper, gold, and silver loss into the bismuth product and how to handle the high bismuth material.

The costs for the metallurgical tests are estimated to be approximately Cdn$80,000.

26.3.2 Oxide and Sulphide Tailings

Further tests are recommended to evaluate the metallurgical performances of the tailings samples, including the sulphide tailings samples. The test work should be conducted on samples that better represent the tailings Mineral Resources. The test work should focus on the processing condition optimization, including regrinding particle size, cyanide concentration and leaching retention time. Cyanide destruction tests on the leach residues and leach residue characterization should also be further conducted. The water management plan should be further investigated.

The estimated cost for the test work, excluding sampling, is approximately Cdn$250,000.

26.4 Reprocessed Tailings Geotechnical Characterization

The tests should be conducted to characterize geotechnical properties for the filtered leach residue which will be dry stacked. Further geotechnical investigation in the OTP DSTMF area is also recommended.

The estimated cost for the geotechnical characterization mentioned above is estimated to be approximately Cdn$100,000.

26.5 Environmental

The cost of permitting has not been considered at this stage of the oxide tailings project. Government agencies should be consulted prior to the permitting process to determine if the current permits for the San Gonzalo Mine can be revised. The cost of expropriating agricultural land for the leach pad and water, which would have to be redirected to the heap leach project but is currently used for agricultural purposes, has also not been assessed. Once the mine plan, site layout, and tailings management plan are further along and have definitive locations, the cost of these factors should be addressed. The cost of monitoring environmental effects post-mine closure needs to be estimated.

The estimated costs for the studies are estimated to be approximately Cdn$80,000.

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28.0 CERTIFICATES OF QUALIFIED PERSONS

Certificate of Qualified Person

I, Hassan Ghaffari, P.Eng., M.A.Sc. do hereby certify:

- This certificate applies to the technical report entitled "Avino Silver & Gold Mines Ltd. Oxide Tailings Project Prefeasibility Study for the Avino Property, Durango, Mexico, NI 43-101 Technical Report", with an effective date of February 5, 2024 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30408).
- My relevant experience includes 30 years of experience in mining and mineral processing plant operation, engineering, project studies and management of various types of mineral processing, including hydrometallurgical mineral processing for porphyry mineral deposits.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- I conducted a personal inspection of the Avino property most recently on June 27, 2023, and prior to the 2023 personal inspection, on March 30, 2011, December 12, 2017, from August 12 to 14, 2019 and July 20, 2021.
- I am responsible for Sections 2, 3, 18, 20, 21 (except mining related costs and process operating cost estimates), 22, 24, and related disclosure in Sections 1, 25, 26 (for matters related to infrastructure, environmental, permitting, socio-economics and economic analysis) and 27 (only references from sections for which I am responsible) of the Technical Report.
- I am independent of Avino Silver & Gold Mines Ltd. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Avino property that is the subject of the Technical Report, in acting as a Qualified Person for the "Amended Mineral Resource Estimate Update for the Avino Property, Durango, Mexico", with an effective date of February 16, 2023; "Amended Mineral Resource Estimate Update for the Avino Property, Durango, Mexico", with an effective date of February 21, 2018; "Technical Report on the Avino Property, Durango, Mexico", with an effective date of April 11, 2017; "Technical Report on the Avino Property, Durango, Mexico", with an effective date of July 19, 2013; and, Technical Report on the Avino Property", with an effective date of July 24, 2012.
- I have read NI 43-101, and the sections of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 5th day of February 2024

"Signed and Sealed"

Hassan Ghaffari, P.Eng., M.A.Sc.
Director of Metallurgy
Tetra Tech Inc.

Certificate of Qualified Person

I, Jianhui (John) Huang, Ph.D., P.Eng., do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Avino Silver & Gold Mines Ltd. Oxide Tailings Project Prefeasibility Study for the Avino Property, Durango, Mexico, NI 43-101 Technical Report", with an effective date of February 5, 2024 (the "Technical Report").
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000).
- I am a member in good standing of the Engineers and Geoscientists British Columbia (#30898).
- My relevant experience includes over 35 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores, and mineral processing plant operation and engineering, including hydrometallurgical mineral processing for porphyry mineral deposits.
- I am a "Qualified Person" for purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- My most recent personal inspection of the Property was on June 27, 2023.
- I am responsible for Sections 13, 17, 19, 21 (process operating cost estimates), and related disclosure in Sections 1, 25, 26 (for matters related to metallurgy, mineral processing, and product marketing) and 27 (only references from sections for which I am responsible) of the Technical Report.
- I am independent of Avino Silver & Gold Mines Ltd. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Avino property that is the subject of the Technical Report, in acting as a Qualified Person for the "Amended Mineral Resource Estimate Update for the Avino Property, Durango, Mexico", with an effective date of February 16, 2023; "Amended Mineral Resource Estimate Update for the Avino Property, Durango, Mexico", with an effective date of February 21, 2018; and, "Technical Report on the Avino Property, Durango, Mexico", with an effective date of April 11, 2017.
- I have read NI 43-101, and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 5th day of February 2024

"Signed and Sealed"

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Inc.

Certificate of Qualified Person

I, Michael F. O'Brien, P.Geo., M.Sc., Pr.Sci.Nat., FAusIMM, do hereby certify:

- I am an independent consultant and director of Red Pennant Communications Corp., a British Columbia Corporation with a business address at 811380 Pinetree Way, Coquitlam, BC, V3E 3S6.
- This certificate applies to the technical report entitled "Avino Silver & Gold Mines Ltd. Oxide Tailings Project Prefeasibility Study for the Avino Property, Durango, Mexico, NI 43-101 Technical Report", with an effective date of February 5, 2024 (the "Technical Report").
- I am a graduate of the University of Natal (B.Sc. Hons. Geology, 1978) and the University of the Witwatersrand (M.Sc. Engineering, 2002).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#41338).
- I am a member in good standing of the South African Council for Natural Scientific Professions (South Africa, 400295/87). My relevant experience is 36 years of experience in operations, mineral project assessment, and I have the experience relevant to Mineral Resource estimation of metal deposits. I have estimated Mineral Resources for greenstone-hosted gold, diatreme complex epithermal gold deposits, porphyry copper-gold, volcanogenic massive sulphide deposits and shear zone-hosted deposits. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on July 20, 2021 and prior to the most personal inspection, from June 12 to 15, 2017 and from February 12 to 14, 2020.
- I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23 and related disclosure in Sections 1, 25, 26 (for matters related to geology and resource estimate) and 27 (only references from sections for which I am responsible) of the Technical Report.
- I am independent of Avino Silver & Gold Mines Ltd. as Independence is defined by Section 1.5 of NI 43-101.
- I have had involvement with the Avino property that is the subject of the Technical Report, in acting as a Qualified Person for the "Amended Mineral Resource Estimate Update for the Avino Property, Durango, Mexico", with an effective date of February 16, 2023; "Amended Mineral Resource Estimate Update for the Avino Property, Durango, Mexico", with an effective date of February 21, 2018; "Technical Report on the Avino Property, Durango, Mexico", with an effective date of April 11, 2017; "Technical Report on the Avino Property, Durango, Mexico", with an effective date of July 19, 2013; and, Technical Report on the Avino Property", with an effective date of July 24, 2012.
- I have read NI 43-101, and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 5th day of February 2024

"Signed and Sealed"

Michael F. O'Brien, P.Geo., M.Sc., Pr.Sci.Nat., FAusIMM, FSAIMM
Owner
Red Pennant Communication Corp.

Certificate of Qualified Person

I, Junjie (Jay) Li, P.Eng., do hereby certify:

- I am the Senior Mining Engineer with Tetra Tech Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled "Avino Silver & Gold Mines Ltd. Oxide Tailings Project Prefeasibility Study for the Avino Property, Durango, Mexico, NI 43-101 Technical Report", with an effective date of February 5, 2024 (the "Technical Report").
- I graduated in 2010 from the University of Alberta with a B.Sc. in Mining Engineering.
- I am a member in good standing with Engineers and Geoscientists of British Columbia (#57479).
- I am a member in good standing with The Association of Professional Engineers and Geoscientists of Alberta (#96462).
- My relevant experience includes over 10 years of experience working in mine engineering consulting, mine operational planning, strategic planning, and mineral economic evaluation. I have been directly involved in mine design and planning, mine production, Ore and Mineral Reserve estimation, technical reviews of mineral assets and mining capital and operating cost estimation.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for those sections of the Technical Report that I am responsible for preparing.
- My recent personal inspection of the Property was on June 27, 2023.
- I am responsible for Sections 15, 16, 21 (oxide tailings extraction related costs only) and related disclosure in Sections 1, 25, 26 (for matters related to mining and mineral reserve for the oxide tailings project) and 27 (only references from sections for which I am responsible) of the Technical Report.
- I am independent of Avino Silver and Gold Mine Ltd. as Independence is defined by Section 1.5 of NI 43-101.
- I have no prior involvement with the Avino Property that is the subject of the Technical Report.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 5th day of February 2024

"Signed and Sealed"

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Junjie (Jay) Li, P.Eng.
Senior Mining Engineer
Tetra Tech Canada Inc.