Report to:

SILVER STANDARD RESOURCES INC.

NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study

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Report to:

SILVER STANDARD RESOURCES INC.

NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY

SEPTEMBER 2009

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03	Sept. 21, 2009	A.S. Sept. 21, 2009	J.R. Sept. 21, 2009	P.W. Sept. 21, 2009	Updated final report revised per SEDAR requirements.

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GLOSSARY

UNITS OF MEASURE

Above mean sea level .... amsl
Acre .... ac
Ampere .... A
Annum (year) .... a
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
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) .... US$

Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
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
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 tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
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
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
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
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

ABBREVIATIONS AND ACRONYMS

abrasion resistant	AR
ACI Engineering Ltd.	ACI
Alphair Ventilating Systems Inc.	Alphair
ammonium nitrate	AN
atomic absorption spectrophotometer	AAS
Bond Work Index	BWi
Cambio de Uso de Suelo	CUS
capital cost estimate	CAPEX
Construction Management	CM
Contract Support Services Inc.	CSS
Delcan International Corp.	Delcan
digital control system	DCS
Engineering, Procurement, and Construction Management	EPCM
Environmental Management Plan	EMP
fixed exchange rate	FXR
free board marine	FOB
free carrier	FCA
fuel oil	FO
G&T Metallurgical Services Ltd	G&T
general and administrative	G&A
Health and Safety Management Plan	HSMP
Health, Safety, and Environmental	HSE
high-density polyethylene	HDPE
inductively-coupled plasma	ICP
input/output	I/O
internal rate of return	IRR
Knight Piésold Consulting Ltd.	Knight Piésold
La Cuesta International, Inc.	LCI
load-haul-dump	LHD
London Metal Exchange	LME
Manifestación de Impacto Ambiental	MIA
material takeoffs	MTOs
methyl-isobutyl carbinol	MIBC
Monarch Resources de Mexico	Monarch
motor control centre	MCC
net present value	NPV
net smelter return	NSR
net smelter royalty	NSR
P&E Mining Consultants Inc.	P&E
Petroleos Mexicanos	PEMEX
Piping and Instrumentation Diagrams	P&IDs
Pocock Industrial, Inc.	Pocock
process design criteria	PDC
programmable logic controller	PLC

Project Management System	PMS
Project Management Team	PMT
Project Procedures Manual	PPM
Quality Assurance/Quality Control Plan	QA/QC
Request for Proposal	RFP
reverse-circulation	RC
rock mass rating	RMR
run-of-mine	ROM
Secretaría de Medio Ambiente y Recursos Naturales	SEMARNAT
Securities & Exchange Commission	SEC
sewage treatment plant	STP
Silver Standard Mexico S.A. de C.V.	SSM
Silver Standard Resources Inc.	Silver Standard
sodium isobutyl-xanthate	SIBX
sodium isopropyl xanthate	SIPX
sodium metabisulphite	SMBS
tailings storage facility	TSF
Traffic and Logistics	T&L
unconfined compressive strength	UCS
Unité de Recherche et de Service en Technologie Minérale	URSTM
variable frequency drive	VFD
Ventsim Mine Ventilation Simulation Software	Ventsim
Wardrop Engineering Inc., A Tetra Tech Company	Wardrop
work breakdown structure	WBS
Workplace Hazardous Materials Information Systems	WHMIS
x-ray fluorescence spectrometer	XRF

1.0 SUMMARY

This technical report on the Pitarrilla property, which is held by Silver Standard Mexico S.A. de C.V., (SSM) a wholly owned subsidiary of Silver Standard Resources Inc. (Silver Standard), has been prepared to comply with the standards outlined in the National Instrument 43-101 (NI 43-101).

SSM's proposed Pitarrilla project is located on the eastern flank of the Sierra Madre mountain range in the central part of Durango State, Mexico. The Pitarrilla property is located in the Municipality of Inde, about 175 km north-northwest of the city of Durango within the state of Durango, Mexico (Figure 1.1). The major city of Torreón lies approximately 150 km (93 miles) east of the project. The town of Casas Blancas is located adjacent to the property. Road access is good, with paved highways extending to within 20 km (12 miles) of the centre of the property. Through SSM, Silver Standard holds a 100% interest in the mineral rights to the approximately 136,192 ha Pitarrilla claim block. Silver Standard also holds significant surface rights in the area.

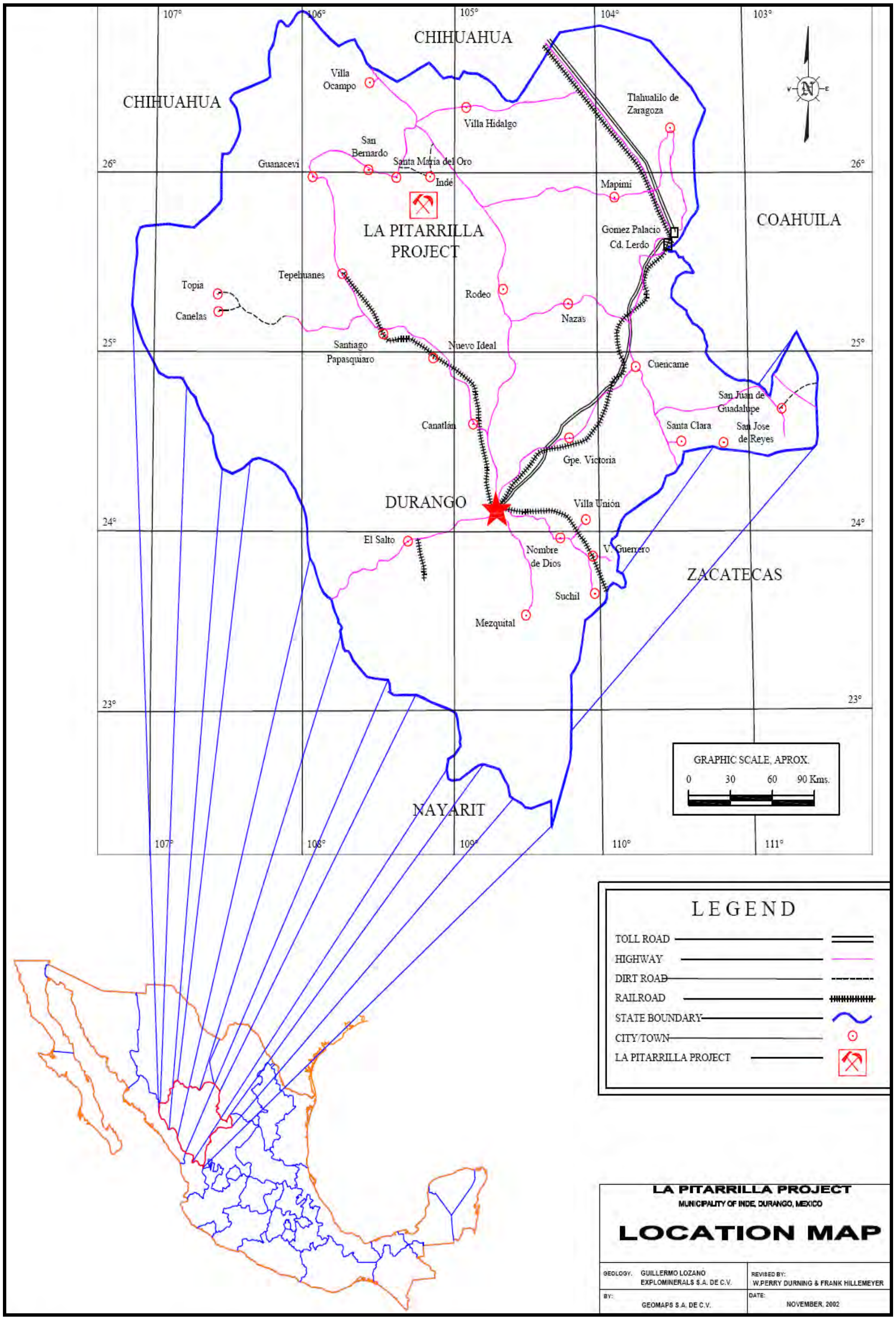
Explominerals, S.A. de C.V. obtained the Pitarrilla concessions on behalf of SSM in November and March of 2002, and June of 2003. SSM obtained the Peña and America claims in May of 2005. The claims have been legally surveyed, as required in Mexico and are currently in good standing.

In May 2008, SSM commissioned a team of engineering consultants to complete the component studies of this Technical Report for the project. The following consultants were commissioned to complete the component studies for this Technical Report:

- Wardrop Engineering Inc., A Tetra Tech Company (Wardrop) – processing, mining, infrastructure, and financial analysis
- P&E Mining Consultants Inc. (P&E) – mineral resource estimate
- Knight Piésold Consulting Ltd. (Knight Piésold) – tailings handling, waste rock and water management, environmental and geotechnical design
- Delcan International Corp. (Delcan) – access road study
- G&T Metallurgical Services Ltd. (G&T) – metallurgical testing.

The proposed mine will be an underground operation with ore processed in the conventional milling plant. Each concentrate of lead, zinc, silver, and minor copper will be thickened and filtered, and will be stored in its corresponding stockpile for subsequent shipping to smelters. The ore production rate will be 4,000 t/d. The mine life is estimated at 12 years, not including the 2 years of pre-production.

Figure 1.1 Pitarrilla Property Location



Resource and reserve tonnages and grades were derived from a geological block model provided by P&E. The four main rock types presented in the block model (Andesite, Basal Conglomerate, Sediments, and C-Horizon) have differing metallurgical recoveries for each of the significant metals: silver, lead, and zinc. The metallurgical assumptions and metal prices of US$11.00/oz for silver, US$0.50/lb for lead, and US$0.70/lb for zinc were used in the net smelter return (NSR) calculation.

A cut-off grade of US$50 NSR was selected as showing the best economic result based on the economic analysis performed for different scenarios for ore reserves at different cut-off grades.

The mineral reserves for the mine are estimated to be 16,673,893 tonnes at an average of 171 g/t Ag, 2.57% Zn, 1.12% Pb.

Only measured and indicated mineral resources as defined in NI 43-101 were used to establish the probable mineral reserves. No reserves were categorized as proven.

Two mining methods were selected based on ore body geometry: room and pillar for mining blocks that have dips of less than 55° and longhole stoping for those steeper than 55°. Both stoping methods provide high productivity from a small number of working faces.

A mine production rate of 4,000 t/d was chosen based on 16,673,893 tonnes of reserves, the geometry of the orebody, stoping productivities, and stope availability.

The design of underground openings, ground support, and mining sequences of this report is based on the geotechnical assessment of the ground conditions at the Pitarrilla mine performed by Knight Piésold. The Knight Piésold report entitled "Silver Standard Resources Inc. La Pitarrilla Project, Geomechanical Input into Underground Mine Design" is available in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices".

Geotechnical considerations have led to longhole stope sizes of 15 m along strike, up to 15 m transverse and sublevel to sublevel heights of 30 m. A primary secondary mining sequence will be adopted in a bottom up sequence.

The ore extraction for room and pillar was estimated to be 89%. Stope recoveries of 95% were applied for both methods.

A currently driven exploration decline will be utilized as the main access for equipment, personnel, and materials, and will also be a major intake airway.

Ore and waste will be conveyed from the underground crusher station mine via conveyor decline. The decline will also act as a major exhaust airway.

The ventilation system designed for the Pitarrilla mine is an intake system delivering approximately 253 m^3/s.

Wardrop designed the Pitarrilla project process plant to treat lead/zinc sulphide ores mined from underground at a rate of 1,400,000 t/a, or 4,000 t/d for 350 d/a. The plant design is based on metallurgical testwork performed by G&T and Pocock Industrial Inc. The testwork results showed that saleable lead and zinc concentrates could be produced using conventional comminution and flotation processes.

The feed to the process plant will contain sulphide minerals composed of lead, zinc, and silver.

Knight Piésold has completed the design of tailings dam, which will be sufficient for the duration of mine life. It is sited upstream of the proposed mill site. Locations for waste dumps have been selected to be compatible with plans for surface water management, which include a seepage recovery dam and pond downstream of the main dam structure.

General information for the project is summarized below:

- mine life..12 years
- milling rate..4,000 t/d
- tonnage milled...1.4 Mt/a
- pre-production capital ....US$277,439,630
- average operating cost ....US$33.81/t milled.

The base case prices supplied by Silver Standard were as follows:

- silver – US$11.00/oz
- zinc – US$0.70/lb
- lead – US$0.50/oz.

A pre-tax economic model has been developed from the estimated costs and the mine production schedule. The base case has an internal rate of return of 10.9% and a net present value of US$107.4 M at a 5% discount rate for the 12-year mine life. The payback of the initial capital is within 6.2 years.

2.0 INTRODUCTION

This NI 43-101 compliant report has been prepared by Wardrop based on work by the following independent consultants:

- Knight Piésold
- P&E
- Delcan
- G&T.

Peter Wells (SAIMM, Fellow) and Jacqueline McAra (P.Eng.) visited the site on behalf of Wardrop on February 17, 2008.

A summary of the qualified persons (QPs) responsible for each section of this report is detailed in Table 2.1. Certificates of QPs are included in Section 21.0.

Table 2.1 Summary of QPs

Report Section	Company	QP
1.0 – Summary	Wardrop	Jacqueline McAra
2.0 – Introduction	Wardrop	Jacqueline McAra
3.0 – Reliance on Other Experts	Wardrop	Jacqueline McAra
4.0 – Property Description and Location	P&E	Dr. Wayne Ewert
5.0 – Accessibility, Climate, Local Resources, Infrastructure and Physiography	P&E	Dr. Wayne Ewert
6.0 – History	P&E	Dr. Wayne Ewert
7.0 – Geological Setting	P&E	Dr. Wayne Ewert
8.0 – Deposit Types	P&E	Dr. Wayne Ewert
9.0 – Mineralization	P&E	Dr. Wayne Ewert
10.0 – Exploration	P&E	Dr. Wayne Ewert
11.0 – Drilling	P&E	Dr. Wayne Ewert
12.0 – Sampling Method	P&E	Tracy Armstrong
13.0 – Sample Preparation, Analysis and Security	P&E	Tracy Armstrong
14.0 – Data Verification	P&E	Tracy Armstrong
15.0 – Adjacent Properties	P&E	Dr. Wayne Ewert
16.0 – Mineral Processing and Metallurgical Testing	Wardrop	Andre de Ruijter
17.0 – Mineral Resource & Mineral Reserve Estimation		
17.1: P&E 2008 Resource Estimate	P&E	Eugene Puritch
17.2: Mineral Reserve Estimate	Wardrop	Iouri Iakovlev

table continues…

Report Section	Company	QP
18.0 – Other Relevant Data and Information		
18.1: Mining Operations	Wardrop	Iouri Iakovlev/ Hasan Ozturk/ Miloje Vicentijevic
18.2: Process Plant	Wardrop	Andre DeRuijter
18.3: Surface Facilities	Wardrop	Jacqueline McAra
18.4: Tailings, Waste Rock, and Water Management	Knight Piésold	Daniel Friedman
18.5: Geotechnical Design	Knight Piésold	Robert Mercer
18.6: Project Execution Plan	Wardrop	Peter Wells
18.7: Environmental	Knight Piésold	Daniel Friedman
18.8: Taxes	Silver Standard	N/A
18.9: Capital Cost Estimate	Wardrop/ Knight Piésold	Peter Wells/ Daniel Friedman
18.10: Operating Cost Estimate	Wardrop	Andre de Ruijter
18.11: Financial Analysis	Wardrop	Scott Cowie
19.0 – Conclusions and Recommendations	All	All
20.0 – References	N/A	N/A
21.0 – Certificates of Qualified Person	N/A	N/A

3.0 RELIANCE ON OTHER EXPERTS

This technical report is based upon published and unpublished data, primarily from geological reports, as described in Section 6.0 (History) and 20.0 (References). Some of these reports were written prior to the implementation of the standards relating to NI 43-101. However, as persons experienced in geology or related fields have prepared the reports, the reports and relevant data are considered to be of high quality.

Silver Standard's employees and consultants provided further information used in the completion of this report and for database compilation and resource modelling.

Technical data provided by Silver Standard for use by Wardrop in this report is the result of work conducted, supervised, and/or verified by Silver Standard professional staff or their consultants. Wardrop provides no guarantees or warranties with respect to the reliability or accuracy of information provided by third-parties.

Silver Standard retained Mr. James A. McCrea (P.Geo.) during January 2007 to update the resource estimate for the Pitarrilla property. Mr. McCrea, a QP under NI 43-101, visited the Pitarrilla property and surrounding area from February 13 to 14, 2004, August 13 to 15, 2005, and again in March of 2007.

As outlined in Section 2.0, this Technical Report has been completed by independent consulting companies. Certificates of QPs are included in Section 21.0. Wardrop disclaims responsibility for reliance on information provided by the following sources who are not QPs:

- Monty Reed of Silver Standard has been relied on for advice on matters relating to market and contracts.

4.0 PROPERTY DESCRIPTION & LOCATION

The Pitarrilla property is located in the municipality of Inde, about 175 km north-northwest of the city of Durango within the state of Durango, Mexico (Figure 4.1). It can be located on the San Francisco de Asis topographic map sheet G13D-31. The property consists of 12 claims totalling 136,192 ha. Explominerals, S.A. de C.V. obtained the Pitarrilla concessions on behalf of SSM in November and March of 2002, and June of 2003. SSM obtained the Peña and America claims in May of 2005. The claims have been legally surveyed, as required in Mexico, and are currently in good standing. Claim details are listed in Table 4.1 and shown in Figure 4.2.

Table 4.1 Summary of Mineral Claims – Pitarrilla Property Land Tenure, May 2009

Claim Number	File #	Title #	Surface (ha)	Municipality (Durango State)	Staking Date
LA PITARRILLA	30749	218323	1,395.4696	El Oro	05/11/2002
LA PITARRILLA 2	31124	220231	5,771.2504	El Oro	24/06/2003
LA PITARRILLA 3	31254	221576	4,200.0000	Inde	02/03/2004
LA PITARRILLA 4	31845	226715	17,960.3850	Inde	21/02/2006
AMERICA	321.1/1-111	183518	198.0000	El Oro	26/10/1988
PEÑA	27442	216381	73.1967	El Oro	14/05/2002
PEÑA 1	27443	216382	62.0818	El Oro	14/05/2002
PITARRILLA 5	25/32978	231034	98,796.3590	El Oro & Inde	
PITARRILLA 6	25/33079	230335	81.0000	El Oro & Inde	16/08/2007
PITARRILLA 7	25/33421		6,241.7758	El Oro & Inde	
PITARRILLA 7 Fracc A	25/33421		114.5648	El Oro & Inde	
PITARRILLA 7 Fracc B	25/33421		1,298.1527	El Oro & Inde	
TOTAL			**136,192.2358**		

The claims cover the major targets of interest: the Cordon Colorado Zone, the La Peña Dike Zone, the Javelina Creek Zone, the Breccia Ridge Zone, and the South Ridge Zone. The property also covers Monarch's gold zones, the Peña de Guerrero target, part of the Fluorite Mine target, and it surrounds the claims that cover the remainder of the Fluorite Mine target area of Crown Resources. The claim map is shown in Figure 4.2.

There are no known royalties, back-in rights, payments, or other agreements and encumbrances to which the property is subject. The property has no known environmental liabilities or outstanding issues.

Information relating to land tenure as noted above was verified by Mr. Kenneth McNaughton, Silver Standard's Vice President Exploration, and P&E has relied on the integrity of such data.

Figure 4.1 Pitarrilla Location Map

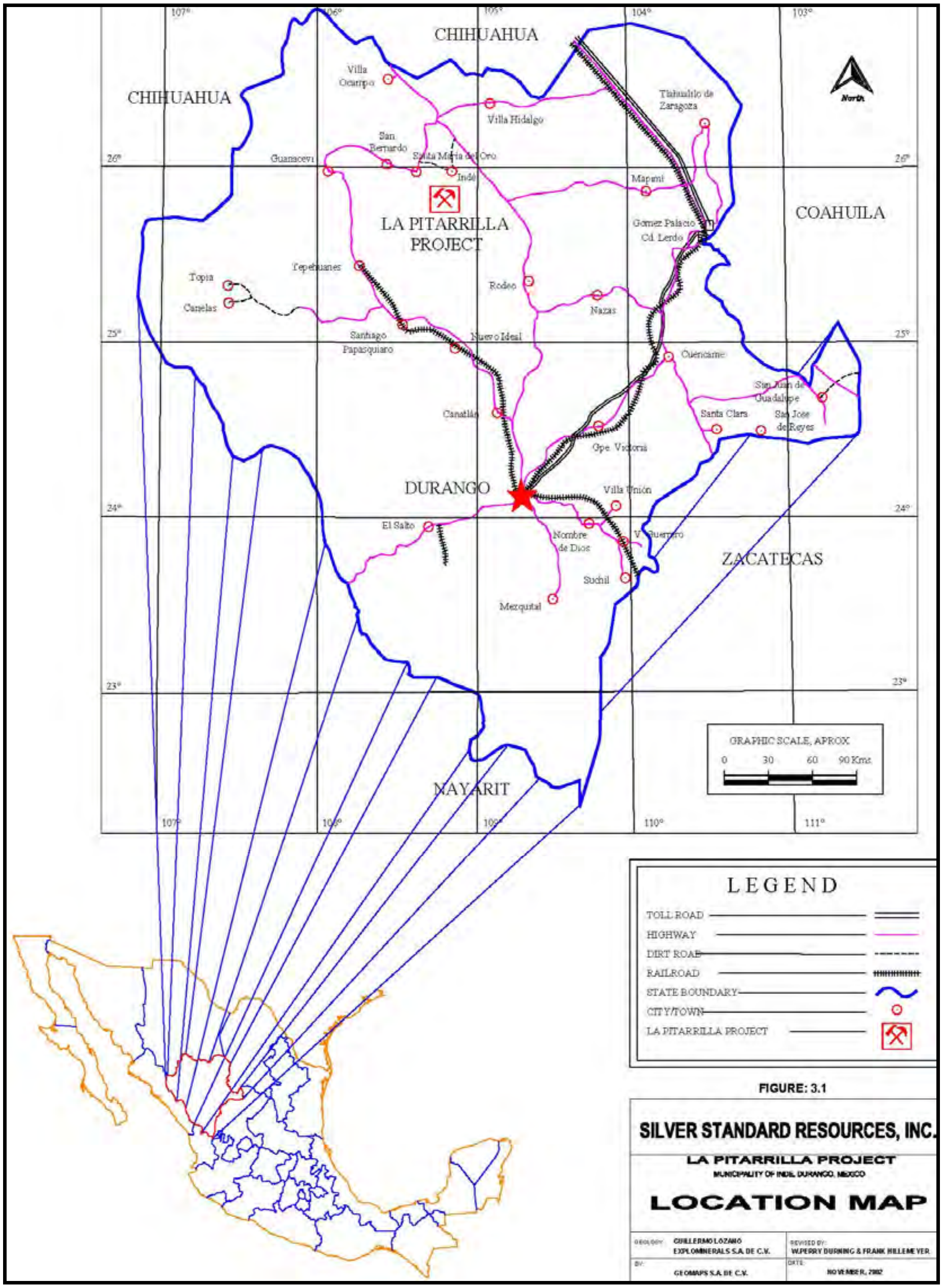
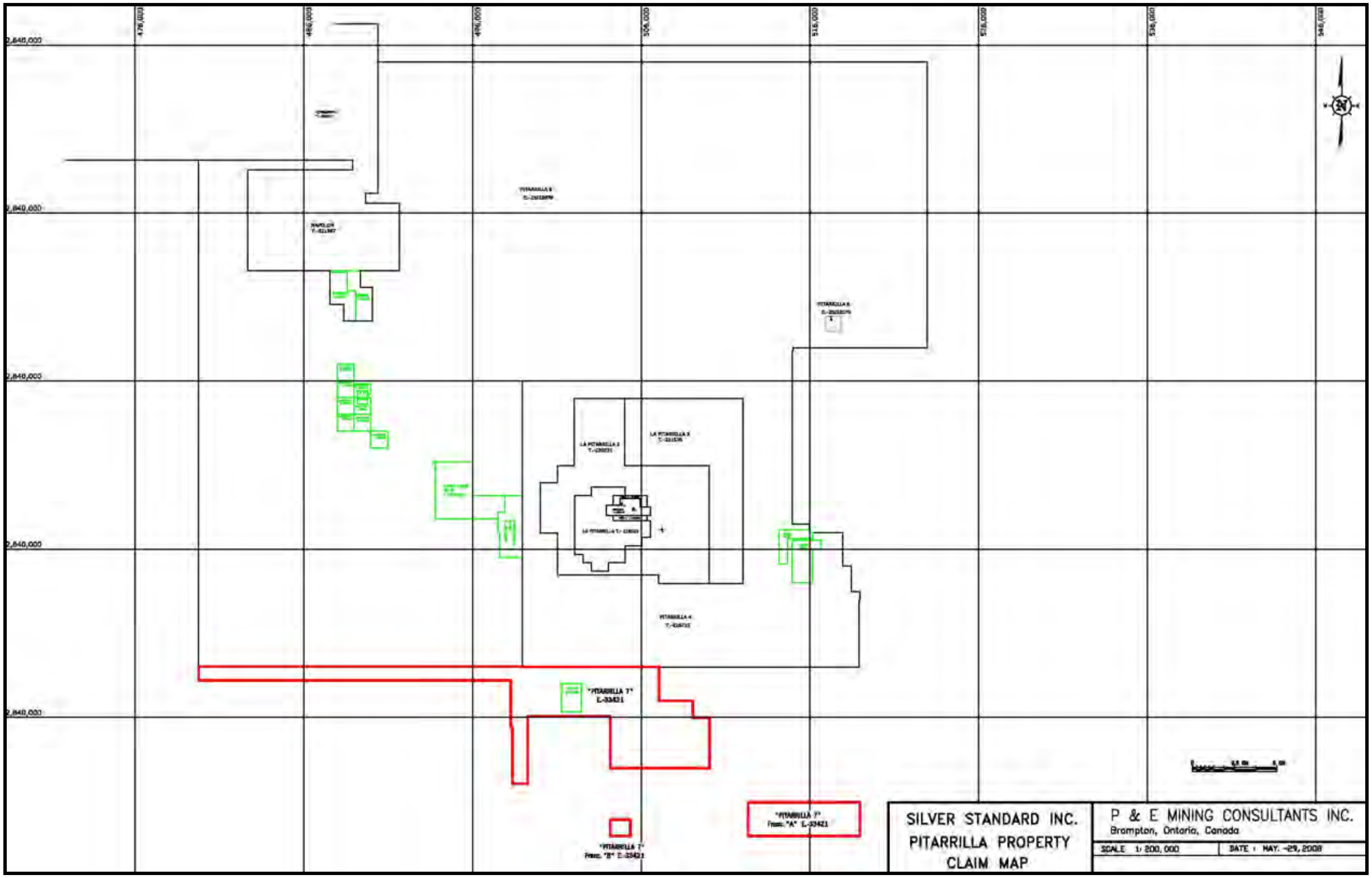


Figure 4.2 Pitarrilla Claim Map



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

The Pitarrilla property is easily accessible year-round. The required driving time from the city of Durango is about 5 hours (approximately 260 km). Traveling along all-weather roads from either San Francisco de Asis or from the town of El Palmito via Casas Blancas can access the property.

The climate is generally hot and dry with average temperatures of approximately 18°C. Rainfall is limited to approximately 500 mm annually. The summer months of July and August have very hot, arid conditions with temperatures reaching 45°C.

Vegetation in the area consists of numerous species of cacti, mesquite, and other thorny bushes. The target areas are located around Cerro La Pitarrilla with local relief in the areas of interest of approximately 250 m. Absolute relief on the property varies from approximately 1575 masl in the valley bottom to the west of Cerro La Pitarrilla to 2120 m at the top of Cerro La Pitarrilla.

Supplies and resources, including fuel and groceries, are available in the town of Rodeo two hours drive to the south. Limited supplies are available in San Francisco de Asis and no services are available in Casas Blancas. Company accommodations and warehouse are located in Casas Blancas. Most large items, assay laboratories, and air transportation are available in the city of Durango.

6.0 HISTORY

6.1 PAST EXPLORATION WORK

The exploration history of the Pitarrilla property is documented by McCrea (2007) and the reader is referred to this reference for additional information.

Exploration of the Pitarrilla property is considered sporadic, with the only significant activity occurring between about 1996 and 2007. A documented history of the property, including exploration activities both on the property and the immediate vicinity, are briefly outlined below and summarized in Table 6.1.

Table 6.1 Summary of Prior Work

Year	Description of Work
1996	Monarch Resources de Mexico, S.A. de C.V. completed a rock-chip and soil sample grid and a detailed stream-sediment survey. (Durning and Hillemeyer, 2002). Monarch completed a 22 reverse-circulation drillhole program, totalling 2,840 m (Durning and Hillemeyer, 1997b). Monarch's exploration was concentrated on the Fluorite Mine target, north of the current property location.
1997	La Cuesta International Inc. (LCI) acquired the La Pitarrilla concessions from Monarch and collected a total of 30 rock-chip samples in a follow-up program (Durning and Hillemeyer, 1997a).
1998	LCI collected 14 channel and grab samples. The samples were sent to Chemex Labs Inc. for chemical analysis (Thurow, 1998).
1999	LCI completed a detailed reconnaissance sampling and mapping program. Samples were sent to Bondar-Clegg in Hermosillo, Sonora to be analyzed for Au, Ag, Pb, Zn, Mo, Bi, As, Sb and Hg (Durning and Hillemeyer, 1999).
2002	Guillermo Lozano Chavez acquired a Pitarrilla concession in the name of Exploминerals, S.A. de C.V. on behalf of Silver Standard. LCI collected 34 rock-chip samples in a work program for Silver Standard (Durning & Hillemeyer, 2002).
Jan.-May 2003	Silver Standard collected 335 rock-chip samples and sent them to Inspectorate Labs for analysis (Lozano, 2003).
June 2003	Road construction commenced and has been ongoing since. Throughout construction road-cut chip samples were collected.
July 2003	LCI collected 5 rock-chip and 8 stream-sediment samples in a work program for Silver Standard to test the Casas Blancas ASTER anomaly. The samples were sent to Inspectorate for analysis (Durning and Hillemeyer, 2003).

table continues…

Year	Description of Work
Sept. 2003-Apr. 2004	Silver Standard has completed four phases of drilling, defining and prioritizing targets of interest. A total of 97 RC drillholes have been completed totalling 10,801 m.
March 2004	Road-cut chip and surface chip sampling completed on the La Colorado portion of the property.
May-June 2004	Silver Standard has completed phases five and six of the drill program for an additional 33 reverse-circulation drillholes. Total drilling is now 130 holes for 14,196 m.
August 2004	Silver Standard has completed phase seven of the drill program with 8 holes completed for 1,219 m and a total of 15,415 m.
April 2005	Silver Standard reports the discovery of two new zones of silver mineralization: Breccia Ridge and Javelina Creek.
July 2005	Silver Standard reports the discovery of a new zone of silver mineralization: South Ridge.
December 2005	Silver Standard has completed phase eight of the drill program with the completion of 73 diamond drillholes and 47 RC drillholes for 22,402 m and a total of 38,060 m drill to date.
March 2006	Phase nine of the drill program is under way with the completion of 17 diamond drillholes for 3,606 m and a property total of 41,666 m in 186 RC drillholes and 90 diamond drillholes.
August 2006	Phase nine continues with the completion of 65 diamond drillholes for 19,044 m and a property total of 60,710 m in 186 RC holes and 155 diamond drillholes.
April 2007	The continuation of phase nine with the completion of 34 diamond drillholes for 27,867 m and a property total of 88,577 m in 186 RC holes and 199 diamond drillholes.
May 2008	The continuation of drilling from August 2007 to May 2008 resulted in the completion of 63,790 m of diamond core drilling in 98 holes. In addition 20,445.69 m of RC were completed. In order to augment the drilling data at the Breccia Ridge Zone, Silver Standard initiated a program of underground ramp development. By May 2008, a total of 707.32 m of ramping had been completed: 616.08 m along the original strike direction and 91.24 m along a branch to the NW.

6.2 EXISTING RESOURCE ESTIMATES

An NI 43-101 compliant resource estimate for the various zones comprising the Pitarrilla property was completed by McCrea in 2004, which was subsequently updated in 2006 and 2007. The resource estimates, which are briefly described below, are based on sectional interpretations for the South Ridge, South Ridge East, Breccia Ridge, Cordon Colorado, Peña Dike, and Javelina Creek zones. There are no historic resource estimates.

6.2.1 SUMMARY OF EXISTING RESOURCE ESTIMATES

MCCREA – APRIL 16, 2004

A 3D block model resource estimate was completed for the Cordon Colorado and Peña Dike zones. Resources for the Pitarrilla Property, using a 40 g/t cut-off, were 10.3 Mt at 114.0 g/t silver in the indicated category and 3.4 Mt at 110.99 g/t silver in the inferred category. Total resources in all categories at a 40 g/t cut-off were reported to be 13.7 Mt at 113.25 g/t silver.

MCCREA – MARCH 13, 2006

An updated 3D block model resource estimate was completed for the Cordon Colorado, Peña Dike, Javelina Creek, and South Ridge zones. Resources for the Pitarrilla property (cut), using a 40 g/t cut-off, were reported to be 18.6 Mt at 112.4 g/t silver in the indicated category and 32.4 Mt at 124.8 g/t silver in the inferred category. Total resources in all categories at a 40 g/t cut-off were reported as 51.0 Mt at 120.30 g/t silver.

MCCREA – SEPTEMBER 28, 2006

McCrea completed an updated 3D block model resource estimate for the South Ridge, South Ridge East, and Breccia Ridge zones with the Cordon Colorado, Peña Dike, and Javelina Creek zones re-classified since the March 13, 2006 update. Resources for the Pitarrilla property (cut), using a 40 g/t cut-off, were 27.2 Mt at 120.6 g/t silver in the measured category, 35.6 Mt at 112.5 g/t silver in the indicated category, and 64.4 Mt at 92.7 g/t silver in the inferred category. Total resources in all categories at a 40 g/t cut-off were reported to be 127.2 Mt at 104.2 g/t silver.

MCCREA – MAY 21, 2007

McCrea prepared an updated 3D block model resource estimate for the Breccia Ridge Zone using a 40 gram silver equivalent cut-off, which was calculated using US$7.00/oz silver, US$0.65/lb zinc, US$0.37/lb lead, and US$1.48/lb copper. Total resources in all categories at a 40 g/t cut-off were 192.3 Mt at 88.0 g/t silver.

7.0 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Pitarrilla property is situated on the eastern flank of the Sierra Madre (Sierra Occidental) mountain range that extends for more than 1,500 km in a north-westerly direction through the northern half of Mexico. This mountain range is largely formed by one of the earth's most voluminous accumulations of intermediate to felsic volcanic rocks which were deposited during the Cenozoic Era. A large number of medium- to high-level hydrothermal systems variably enriched in Ag, Au, Pb, Zn and (to a lesser extent) Cu, Sb, As, and F were generated during this extended period of felsic magmatism including the epithermal systems that formed the great Mexican silver ore systems at Guanajuato, Zacatecas, Fresnillo, and Santa Barbara-San Francisco del Oro. The silver mineralization found on the Pitarrilla property is strategically located within the central portion of the globally important Central Mexican Silver Belt, a linear 900 km-long metallogenetic province defined by the four previously noted silver districts among several others, including the mining camps of Parral, Santa Maria del Oro, and Sombrerete-Chalchihuites (Burk, 2005, pers. comm.).

The unconformity between the Tertiary volcanic succession that partially hosts the extensive silver mineralization and the underlying Cretaceous or older marine sediments is exposed at an elevation of about 1685 masl on the Pitarrilla property. This regional unconformity is generally marked by various facies of polymictic conglomerate with these deposits reaching thicknesses of tens of metres. Overlying the conglomerate, or marine sedimentary units where the conglomerate is absent, is a 200- to 300-m thick sequence of volcaniclastic and subaerial volcanic lithologies that are predominantly andesitic in the lower portion of the sequence and dacitic to rhyolitic in the upper parts (Burk, 2005, pers. comm.)

Structurally, the Pitarrilla area is marked by at least two sets of prominent normal faults, one set trending NW-SE and the other striking NE-SW. Regional extension appears to have been focussed in the area surrounding Pitarrilla. It is hypothesized that the high-level intrusion of a felsic magma body caused, or was at least associated with, this extensional tectonism.

7.2 PROPERTY GEOLOGY

The Pitarrilla silver deposits are distributed around and beneath a felsic flow-dome complex, which was emplaced into a package of intermediate volcaniclastic rocks.

Underlying these siliceous flows and breccias, is a 600-m thick succession of andesitic, dacitic, and rhyolitic volcanic rocks, summarized as follows:

- There is a 150- to 200-m thick package of felsic and intermediate pyroclastic and polylithic breccia rocks.
- This is underlain by an accumulation of volcaniclastic sandstone and conglomerate units that attain thicknesses of 50 to 100 m
- Immediately down-section, these units are gradually replaced by a 75- to 150-m thick package of dacitic to andesitic, fine- to coarse-grained volcaniclastic rocks. In the vicinity of rhyolite dikes and sills, the fine-grained dacite is intensely brecciated, displaying features suggestive of a diatreme breccia.
- Based on relatively deep drilling in the area of the Breccia Ridge Zone Ag-Zn-Pb deposit, it has been determined that a 100- to 130-m thick sill of massive, fine-grained andesite was emplaced sub-conformably at or close to the base of the main package of volcaniclastic rocks.
- The andesite sill overlies a polymictic conglomerate formation, up to 60 or 80 m in places, containing cobbles of micritic limestone in addition to clasts of siltstone and variously textured andesite. This conglomerate has proven to be a key lithology because it hosts important base metal sulphide-associated silver mineralization at the core of the Pitarrilla mineralizing system.
- Beneath the potential mineralization-hosting cobble conglomerate is a succession of interbedded siltstone, sandstone, shale, marl, and thin units of intraformational pebble conglomerate. Contact relationships suggest that mineralization was deposited in a basal conglomerate which separates the interbedded, probably marine sedimentary rocks from the overlying rocks (most likely early to middle Tertiary volcaniclastic and pyroclastic).
- This succession of volcanic rocks is intruded by a complex system of rhyolite sills and dikes, ranging in thickness from less than 2 m up to as much as 100 m. These rhyolite intrusive bodies, which can be more than 100-m thick, are a key rock type at Pitarrilla because most mineralization is either hosted in the rhyolites or in the enclosing brecciated host rock.

The oxidized and disseminated silver mineralization that forms the six known zones of mineralization at Pitarrilla is largely hosted by three main lithologies: a feldspar-porphyritic rhyolite, a polylithic breccia, and a crystal-lithic tuff.

The intrusive feldspar-porphyritic rhyolite is by far the most important lithology at Pitarrilla. Massive, fine-grained, and weakly quartz- and feldspar-porphyritic rhyolite sills and dikes host more than 95% of the silver resources of the Cordon Colorado and Peña Dike deposits along with major portions of the South Ridge and Breccia Ridge Zones.

Probably the second most important host rock for the oxidized silver mineralization is represented by the polylithic breccia that is spatially associated with mineralized rhyolite porphyry at the South Ridge Zone. It is estimated that in this deposit the total amount of disseminated silver mineralization is roughly distributed in the proportion of 70/30 between the intrusive rhyolite host and the adjacent mineralized polylithic breccia units.

The third of these lithologies, the crystal-lithic tuff, is known to host more than about 70% of the silver mineralization at the Javelina Creek deposit. The tuff is a relatively thick unit that contains flattened or welded dacite lapilli set in a groundmass rich in quartz crystal fragments. The remaining 30% of the resource is found in a unit of thinly interlayered ash and medium- to coarse-grained rhyodacite tuffs that immediately overlie the crystal-lithic tuff.

7.3 PITARRILLA SILVER DEPOSITS

Silver Standard has identified six zones of significant silver mineralization on the Pitarrilla property (refer to Figure 11.1). From west to east, they are known as the Cordon Colorado, Peña Dike, Breccia Ridge, South Ridge, South Ridge East, and Javelina Creek zones.

CORDON COLORADO ZONE

The Cordon Colorado Zone is a relatively flat-lying, ‘pie-shaped’, tabular deposit with a southeast orientated apex. The deposit lies very close to surface and would be very amenable to open-pit mining. The NW-SE axis of maximum length is approximately 450 m, with the NE-SW axis being about 350 m. The deposit ranges in thickness from 30 to 85 m, with the average being about 50 m.

PEÑA DIKE ZONE

The Peña Dike Zone lies 500 m north of Cordon Colorado beneath a relatively small, but nevertheless prominent, north westerly trending ridge. The basic shape of the deposit, based on some 40 drillholes, is one of an elongated ellipsoid where the primary axis is oriented sub-horizontally in the NW-SE direction. The length of this axis is approximately 250 m, with the secondary axis being 100 m or less. Potentially ore-grade oxidized silver mineralization outcrops on surface; however, the silver-rich core of the deposit lies roughly 60 m below surface.

BRECCIA RIDGE ZONE

While the northern end of the Breccia Ridge Zone appears to be connected to the Peña Dike Zone by sub-economic silver mineralization, the bulk of the deposit lies beneath the topographic saddle that separates the two peaks of Cerro La Pitarrilla. Diamond drillholes have indicated that this zone of silver plus significant Pb-Zn mineralization could ultimately be the largest of the Pitarrilla deposits. Presently, it is known to extend from about 50 m below surface to vertical depths in excess of 750 m. This 250-m wide zone has been traced laterally for more than 500 m along its primary NNW-SSE axis.

As well as the oxidized mineralization close to the surface at Breccia Ridge, diamond drilling has determined that a number of sub-zones of mineralization can be distinguished in the lower levels of the Breccia Ridge deposit. A brief description of these sub-zones is given below.

Basal Conglomerate Zone

This is one of the most economically important mineralized zones at Pitarrilla. It lies sub-horizontally, consisting of replacement-style iron and base metal sulphide mineralization hosted by a polymictic conglomerate, which unconformably separates the Cretaceous sedimentary rocks from the overlying Tertiary volcanic rock units. The mineralized zone is typically from 15 to 25 m in thickness obtaining a maximum thickness of 40 m in the central part of the zone. This sub-zone can be traced along strike for at least 700 m.

C Zone

The limits of the C Zone are not well constrained; however, it does appear to be developed in a mafic sill above the central part of the Basal Conglomerate sub-zone and could have lateral dimensions that exceed 100 m. Locally, the upper margin of this mafic unit is marked by silver-rich Cu-Pb-Zn sulphide mineralization comprising the C Zone. The mineralization consists of a few metres of massive to sub-massive chalcopyrite or sphalerite with high concentrations of galena.

V1, V2, and V3 Sub-zones

These three deep and clearly structurally-controlled sub-zones of mineralization are steeply east-dipping and follow the margins of two rhyolite dikes that cut through the Cretaceous sedimentary sequence. The mineralization consists of small veins and impregnations of pyrite, marcasite, sphalerite, and galena, along with minor amounts of chalcopyrite, arsenopyrite, and tetrahedrite.

South Ridge Zone

The core of the South Ridge Zone is situated about 600 m south-southeast of the central part of the Breccia Ridge Zone, but future drilling may ultimately show that these two deposits can be viewed as a single very large deposit. It is worth noting that the package of volcanic rocks hosting the silver mineralization of this zone lies at a higher stratigraphic position than those rocks hosting the Breccia Ridge mineralization. Overall, the South Ridge Zone trends NNW-SSE and its presently defined maximum dimensions measure 525 m by 470 m perpendicular to strike and down dip respectively. The deposit is found at surface along its north-western margin and dips into the southern ridge-spur of Cerro La Pitarrilla. Vertical thickness of the deposit varies considerably, from less than 20 m up to more than 100 m. The body of mostly oxidized Ag (-Zn) mineralization appears to plunge at a moderate angle to the south-southeast and appears to be cut off by topography on the southern flank of Cerro La Pitarrilla.

South Ridge East Zone

The South Ridge East Zone is situated immediately east of the South Ridge Zone. Geological evidence suggests that the two zones are connected by relatively narrow low-grade zones of oxidized silver mineralization. This zone is best described as a strongly elongated ellipsoid that has its primary axis oriented NNW-SSE. The deposit is at least 700 m long and on the order of 75 to 100 m in thickness.

Javelina Creek Zone

The Javelina Creek Zone underlies the western slope of the north-westerly trending ‘whale-back’ ridge that exists approximately 900 m northeast of the centre of the Breccia Ridge Zone. Oxidized silver mineralization is disseminated in two separate zones hosted within thinly bedded felsic tuffs and underlying quartz-crystal lithic tuffs. The two zones merge to the east beneath the crest of the ridge.

8.0 DEPOSIT TYPES

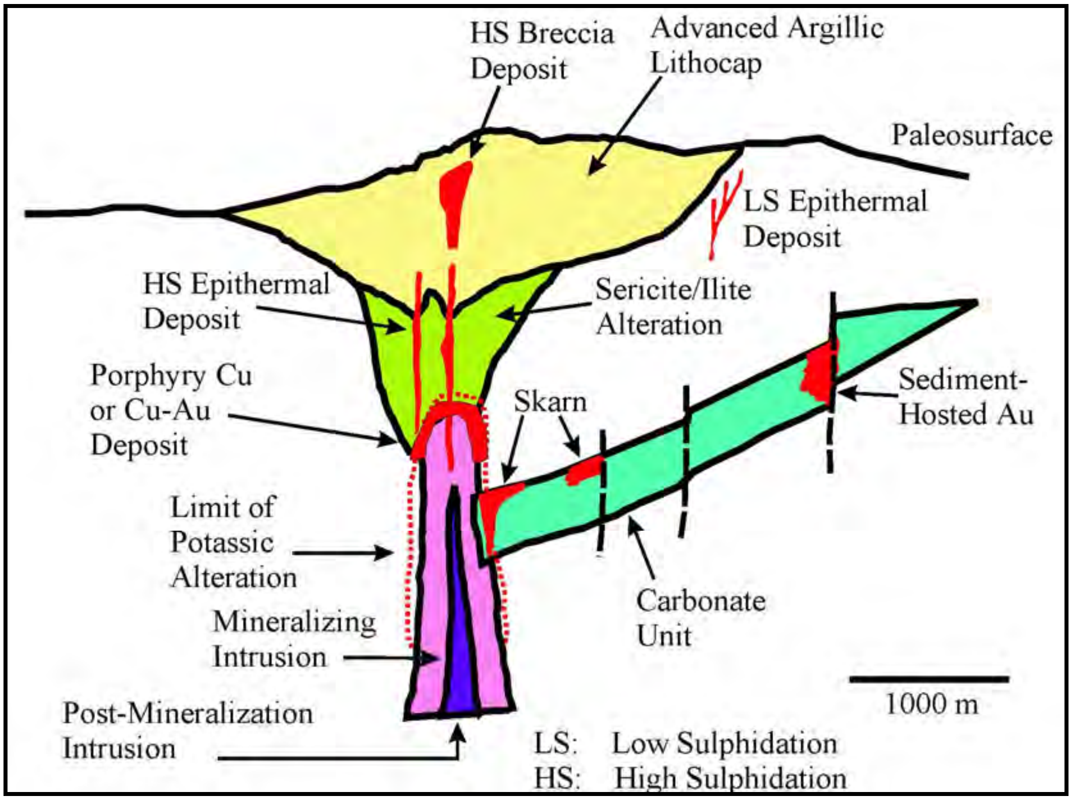
While the Pitarrilla property is centrally located within the Central Mexican Silver Belt, the recently discovered Pitarrilla silver deposits display many features that are more or less unique to the major silver districts of Mexico. The style and, to some degree, the setting of the historically important silver deposits of Mexico differ markedly from a large percentage of the silver resources found at Pitarrilla where the mineralization is mostly microscopically disseminated (commonly in a very uniform manner) in mainly Middle Tertiary subvolcanic rhyolite sills and dikes, and locally in volcanic breccias and tuffs. Sulphide-associated silver mineralization at Pitarrilla is essentially restricted to the Breccia Ridge Zone, especially in the deeper parts of the deposit where the most recently completed diamond drilling has been concentrated. The silver resources contained in the other five deposits are characterized by the ubiquitous presence of disseminated and fracture-controlled hematite and/or limonite.

Notwithstanding that there are considerable differences between Pitarrilla and other major silver deposits of Mexico (if not in the world), the oxidized mineralization on the Silver Standard property is seen to have at least two characteristics that induce it to be classified as an intermediate sulphidation type of epithermal deposit (Sillitoe and Hedenquist, 2003) as shown in Figure 8.1.

Firstly, the Pitarrilla silver mineralization is presumed to be Middle Tertiary in age (32-28 Ma) and genetically related to felsic magmatism (i.e. the rhyolite volcanism and hypabyssal intrusive activity that was regionally extensive in Mexico during Cenozoic time when compressional tectonics and subduction where occurring off the western coast of the country) (Burk, 2005, pers. comm.; Camprubí et al., 2003). Secondly, the metal suite of Ag-Pb-Zn-Cd-Sb (-As-Cu) and the proportions of the main trace elements that characterize the mineralization at Pitarrilla are comparable to those found in the other 'classic' Mexican deposits, with the exception that gold is essentially absent in the disseminated silver deposits at Pitarrilla.

The Breccia Ridge Zone contains economically significant zones of argentiferous Fe-Zn-Pb (-As-Cu) sulphide mineralization at depths of between 500 and 800 m. The mineralization occurs as massive veins up to a few metres thick, as extensive networks of veinlets, and as a large body of replacement mineralization hosted by a basal polymictic conglomerate. These forms of silver-bearing sulphide mineralization are similar to ore types that have been mined in many of the productive mining districts of Mexico, making the Pitarrilla property both typical and atypical in its styles of intermediate sulphidation epithermal mineralization. Silver Standard views these zones of volcanic- and high-level intrusive-hosted, oxidized, and disseminated silver mineralization at Pitarrilla as being the upper manifestations of a vertically extensive ore system that at depth, beyond the limits of supergene oxidation, is represented by argentiferous base-metal veins and replacement bodies of the type most commonly mined in Tertiary volcanic terrains of Mexico, Peru, and other countries (Burk, 2005, pers. comm.). It is possible that Pitarrilla is a special case in the broad spectrum of epithermal deposits, where a complete intermediate-sulphidation silver ore system is preserved (Burk, 2005, pers. comm.).

Figure 8.1 Potential Deposit Types in an Intrusion-centred Hydrothermal System



Modified after Sillitoe (1995b).

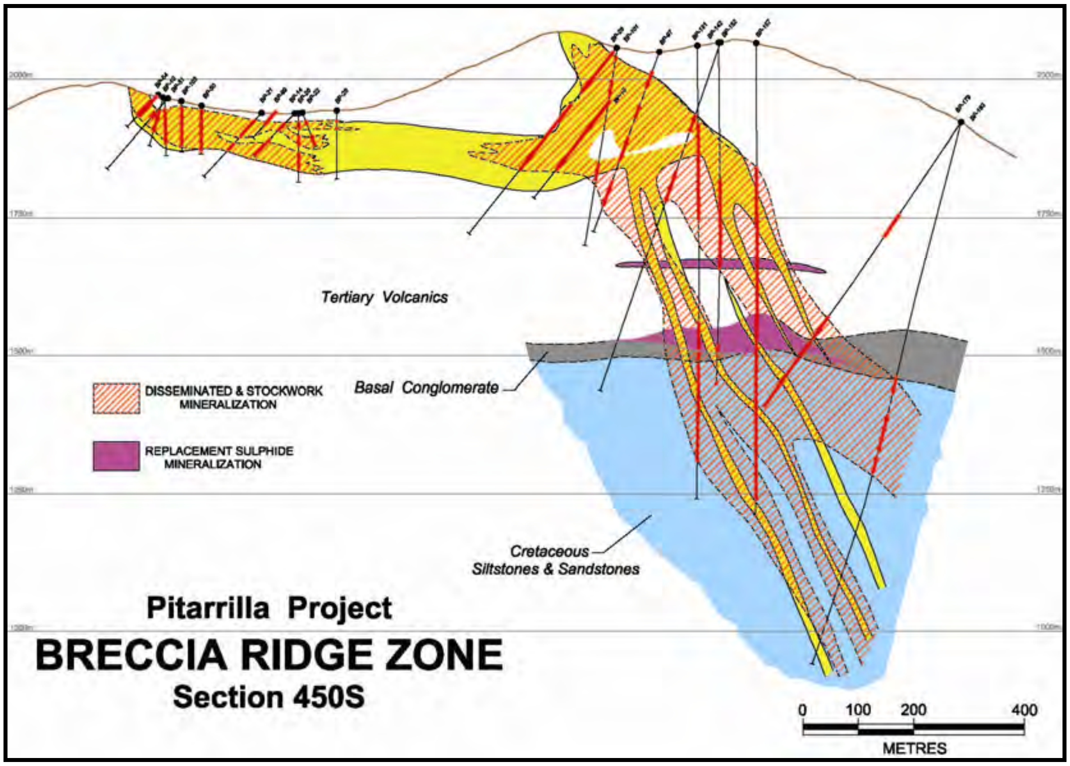
9.0 MINERALIZATION

To date, optical petrographic analyses have only rarely succeeded in identifying the silver mineral species in the oxidized mineralization at Pitarrilla. Clearly, for the zones of oxidized mineralization, silver is associated with moderate to strong iron oxide alteration, both hematite- and limonite-dominated, though it should not be assumed that the precious metal is entirely fixed, chemically or physically, to these oxides.

Detailed petrographic analysis has identified silver halides, specifically chlorargyrite, iodargyrite, bromargyrite, and in lower grade samples the silver sulphide acanthite. The Ag-sulphide grains were seen occurring along minute fractures in anhedral quartz grains, interstitial to quartz-potassium feldspar intergrowths, and sparingly as submicron specks in iron oxide pseudomorphs after subhedral pyrite. In addition to the fine-grained acanthite trace amounts of Ag-Hg sulphide (imiterite) and silver, selenides (aguilarite and naumannite) have been noted.

Only limited work has been done on the silver-bearing sulphide mineralization found mostly in the Breccia Ridge Zone at depths in excess of 250 m. Silver Standard has noted that sulphide-associated silver is mainly found with fine grained sphalerite, galena, and pyrite-marcasite mineralization. Pyrrhotite is locally abundant in the mineralized basal polymictic conglomerate found 400 to 450 m below surface, apparently occurring as an early sulphide phase that was replaced by later sphalerite, galena, pyrite, and minor chalcopyrite. A cross section (4+50S) through the Breccia Ridge Zone is presented in Figure 9.1.

Figure 9.1 Breccia Ridge Cross Section 4+50S



Other sulphides that have been observed in minor amounts include arsenopyrite, tetrahedrite-tennantite, and stibnite. Lead-antimony-arsenic-silver sulphosalt minerals, including jamesonite, boulangerite and acanthite, have also been reported. Sulphide-rich samples indicate that freiburgite, the silver-rich form of tetrahedrite, is one of the main silver-bearing minerals.

The zones of silver-bearing sulphide mineralization in the lower levels of the Breccia Ridge Zone are enriched in zinc, lead and, to lesser degrees, copper, arsenic, and antimony. Weak to moderate enrichments were also noted in cadmium and, locally, bismuth. Gold is only rarely enriched in sulphide mineralization and then only weakly (generally less than 200 ppb Au). Based on empirical observations, higher silver contents tend to occur where the mineralized rock is rich in galena (Burk, 2007, pers. comm.).

In contrast to the sulphide mineralization, metal enrichments in the deposits of oxidized mineralization are generally associated with elevated contents in lead, antimony, zinc, cadmium, and arsenic. These trace metals are presumed to occur in the molecular structures of various iron oxides and sulphates. In these zones of oxidized mineralization, there is an overall negative correlation between silver and zinc, although many samples with high-grade silver contents are also marked by relatively high zinc contents (1 to 8%).

As previously noted, the most evident form of hydrothermal alteration at Pitarrilla is represented by the relatively abundant disseminated and fracture-controlled hematite and/or limonite in the mineralized lithologies. Mostly, the iron oxides are replacing disseminated pyrite, sphalerite, and minor galena that were deposited prior to the extensive oxidation event. Commonly, the iron oxide alteration is accompanied by a pervasive argillization of the originally feldspathic intrusive and volcanic rocks.

Geochemical analyses consistently show the mineralized rocks being depleted in calcium, sodium, and magnesium, while being relatively enriched in potassium, suggesting that potassic clays such as kaolin and illite formed in the silver-enriched rocks at the expense of plagioclase (Burk, 2005, pers. comm.). Another phase of potassic alteration in the form of K-feldspar may be developed in the rhyolite porphyry sills and dikes, especially those that are mineralized with silver, although it has not been conclusively determined if the bulk of this feldspar has a secondary, hydrothermal origin or if it was an original constituent of the rock (Leitch, 2005).

Another lithology strongly affected by the hydrothermal fluids that deposited silver at Pitarrilla is the polymictic conglomerate that occurs at the base of the Tertiary succession on the property. Drillhole intercepts of this unit from the Breccia Ridge Zone show that the matrix of the conglomerate is moderately to entirely replaced by intergrowths of dark green chlorite and beige coloured iron carbonate, either siderite or ankerite. The chlorite-iron carbonate assemblage also partially replaces cobble-sized clasts; however, iron and base metal sulphides preferentially replace the various mineral components forming these clasts, especially pebbles and cobbles of limestone.

It is noteworthy that true silicification, the introduction into hydrothermally altered rocks of significant amounts of silica, is not widespread or well developed in the various silver deposits found on the Pitarrilla property. Remarkably, few quartz veins have been observed that are mineralized with silver and the associated base metals.

10.0 EXPLORATION

Explominerals, S.A. de C.V. (Guillermo Lozano Chavez) obtained the Pitarrilla concessions for Silver Standard in November 2002 and June 2003.

Since the date(s) of acquisition, exploration on the Pitarrilla property by Silver Standard has included extensive mapping and surficial sampling (rock-chips, soils and channel/stream sediments) programs, as well as extensive reverse circulation (RC) and diamond core drilling. No geophysical surveys have been recorded.

Immediately upon initial possession of the claims, Silver Standard confirmed the presence of anomalous silver and zinc values with follow-up sampling and mapping programs conducted by La Cuesta International Inc. (LCI). Follow-up work involved the collection of 34 rock-chip samples at previously identified anomalies.

Silver Standard completed a 335 rock-chip sample program from January to May 2003 to evaluate the property (Lozano, 2003). Three main targets were identified based on the presence of anomalous silver zones: the Cordon Colorado, Javelina Creek, and Peña Dike targets.

Road construction commenced in June of 2003 and has been ongoing as exploration continues and expands. Throughout the road construction road cut chip sampling was conducted.

In July of 2003, approximately 3 km west-southwest of Cerro La Pitarrilla, 5 rock-chip and 8 stream-sediment samples were taken to test the mineral potential of the Casas Blancas ASTER anomaly (Durning and Hillemeyer, 2003).

Silver Standard commenced Phase I drilling at Pitarrilla in September of 2003. Since then, the drill has not left the property and additional equipment has been added. Silver Standard has completed eight phases of drilling, with a ninth phase currently underway.

In March of 2004, road-cut chip sampling, along with additional surface chip sampling, was completed on the La Colorado area of the property. The surface sampling was located to the north west of the Cordon Colorado Zone where the deposit crops out and forms part of a ridge.

In the fall of 2007, Silver Standard commenced construction on a 2.5 km-long decline to provide underground drilling stations for the high-grade silver and base metal mineralization of the Breccia Ridge Zone. As of August 2008, a total of 707.32 m of underground ramping had been completed (616.08 m along the original azimuth of the ramp and 91.24 m along the NW branch).

11.0 DRILLING

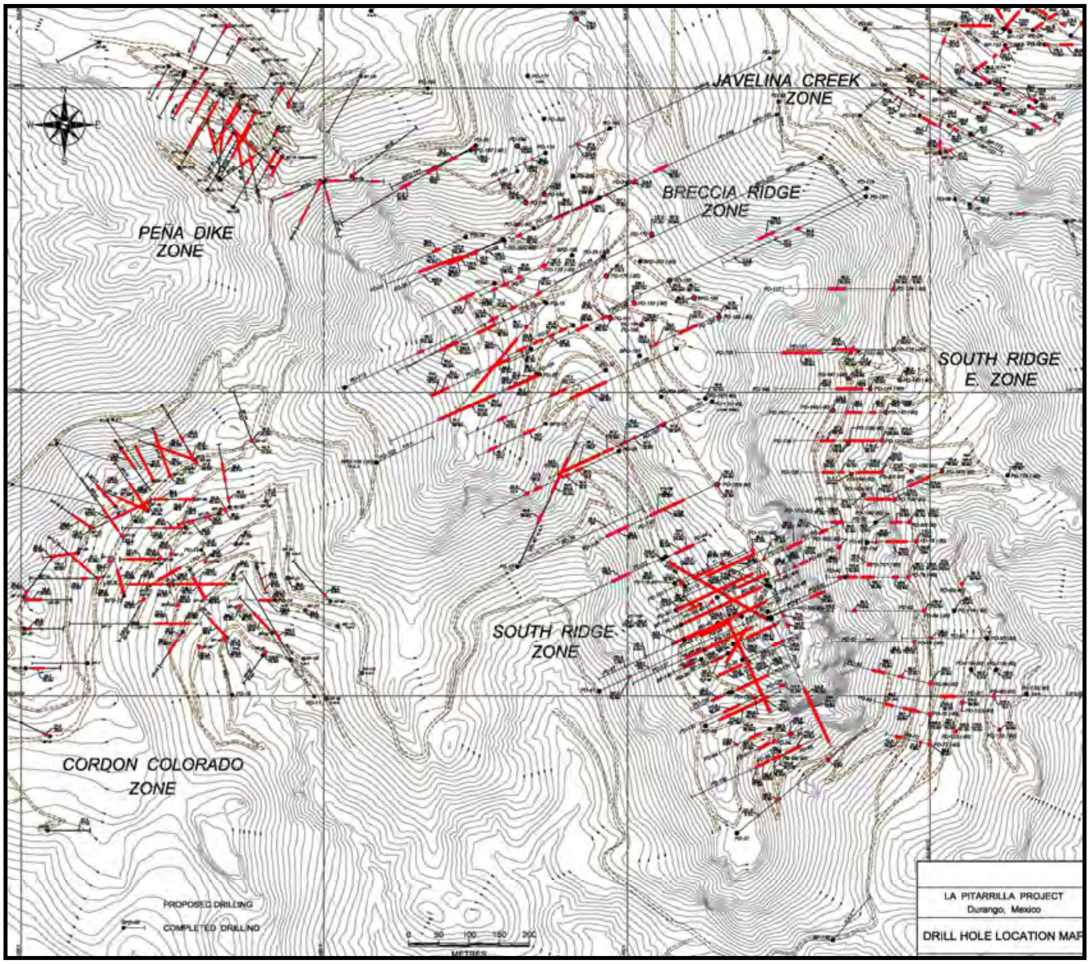
From September 2003 to August 2008, Silver Standard completed eight phases of drilling on the Pitarrilla property and is currently conducting a ninth phase. As of August 2008, 186 RC drillholes and 299 diamond drillholes have been drilled, totalling 159,457 m. The first five phases were concentrated on the Cordon Colorado and Peña Dike zones. The sixth phase of drilling included a third identified silver zone, Javelina Creek, but the majority of the drilling was on Peña Dike. Phase seven of the drill program saw more drilling on Peña Dike. The eighth phase of drilling started the use of diamond drill equipment with the continuation of the RC drilling. Phase eight included the discovery of the Breccia Ridge and South Ridge zones.

The drill contractor for the first seven phases of RC drilling was Dateline Drilling of Mexico; Phase eight included Major Drilling of Mexico as the diamond drilling contractor, and phase nine consists entirely of diamond drilling. As of the effective date of this report, P&E does not have any information pertaining to the additional drilling completed after drillhole number PD-343 in August 2008.

The relationship of sample length to true thickness is variable and dependent on the individual silver zones. The drillholes are not drilled at a consistent azimuth and dip because the individual zones all have different orientations and the zones are also structurally complex. At this time, no direct conclusion can be drawn between drillhole intercepts and true width.

Figure 11.1 shows the 2007 drillhole location map.

Figure 11.1 2007 Drillhole Location Map



12.0 SAMPLING METHOD AND APPROACH

Sampling on the Pitarrilla property has been extensive, including the collection of rock-chip, soil, drillhole, channel, and stream-sediment samples in a large variety of widths from various rock types. Only RC and diamond drillholes were used to estimate the current resource.

RC drill samples were collected at the drill site at 1 m intervals from collar to total depth. Sampling intervals were dependent on the drilling equipment selected and not based on geological controls or other features of the zone of interest.

The RC samples were split three times using a Jones Splitter down to 1/8 size. The sample size ranged from approximately 2 to 10 kg, and every 20th sample had a second (field duplicate) sample collected. All samples were stored in the company warehouse in Casas Blancas. Periodically, staff from ALS Chemex laboratories picked up the samples and transported them to Guadalajara for sample preparation; however, after mid-August of 2005, the samples were shipped to Chihuahua for preparation.

Diamond drill core samples were collected after the core was logged. The core was split using a diamond saw. Sample lengths were based on geologic contacts and drilling equipment and were 1.52 to 1.53 m to fit the core tube length of the drill. The maximum sample length was just over 3 m in zones believed to be poorly or unmineralized. The split core samples were bagged and, as with the RC samples, the staff from ALS Chemex laboratories picked up the samples and transported them to Guadalajara. After mid-August of 2005, the samples were shipped to Chihuahua for sample preparation.

There are no obvious drilling, sampling, or recovery factors that would impact the reliability of the samples.

13.0 SAMPLES PREPARATION, ANALYSES, AND SECURITY

Project crews delivered the RC and diamond core drill samples to the ALS Chemex preparation laboratory in Guadalajara or Chihuahua, Mexico. The samples were crushed to 70% passing -2 mm and split using a riffle splitter to produce a representative 250-g split for pulverization. The sample split was pulverized to better than 85% passing 75 µm. The pulps were shipped to ALS Chemex laboratories in North Vancouver, BC for analysis by four acid "near total" digestion and inductively coupled plasma (ICP) analysis of 27 elements. ALS Chemex laboratory is a large international laboratory with an ISO certification.

Mercury analysis was added to the standard package and analyzed by cold vapour atomic absorption (AA). The analyses were completed on a standard 30-g split. Element over limits were rerun by atomic absorption for zinc, lead, and copper, and a fire assay with a gravimetric finish for silver. Gold analyses were requested during the early stages of the program but were later dropped for lack of results. Gold analyses are occasionally requested in deep drillholes in base metal zones.

Silver Standard's early quality control program included shipping field duplicates to BSI Inspectorate in Durango. Every 20th RC drill sample had a second split collected and analyzed as described above. In 2005, Silver Standard implemented a quality assurance/quality control (QA/QC) program comprising certified reference materials, (also known as standards), blanks, and field duplicates. The rate of insertion of QC samples was approximately 1 in 20. Approximately 5% of the pulps were sent to a secondary laboratory as a monitor on the principal lab.

No aspect of the sample preparation was conducted by an employee, officer, director, or associate of Silver Standard.

14.0 DATA VERIFICATION

A site visit was undertaken by P&E co-authors Eugene Puritch (P.Eng.) and Antoine Yassa (P.Geo.) from March 26 to April 2, 2008. P&E visited several drill collars, took GPS coordinates, and inspected the core storage and sampling facilities. The core was examined and 14 samples were taken in 11 holes by taking ¼ splits of the half remaining core. An effort was made to sample a range of grades. At no time were any employees of Silver Standard advised as to the identification of the samples to be chosen during the visit.

The samples were selected by Mr. Yassa, ¼ sawn by the technician, and placed by Mr. Yassa into sample bags, which were sealed with tape, packed in reinforced bags, and transported by road to DHL International Courier in Torréon. They were shipped to the P&E office in Brampton, Ontario, and from there delivered to SGS Mineral Services in Toronto.

P&E's sampling confirms that the Pitarrilla system is mineralized.

14.1 QUALITY CONTROL PROGRAM

14.1.1 PROPERTY REFERENCE MATERIALS

Silver Standard used a total of 11 reference materials to monitor laboratory accuracy. Eight of the standards were made up in batches from material left over from the initial RC drilling program. Each batch of reference material was given a number and called Standards 1, 2, and 6 through 11 (Standards 3, 4 and 5 were certified reference materials). The number of samples in each round robin characterization of these standards was found to be too few to provide a representative mean. As the data set provided by Silver Standard contained at least 200+ samples and at most 700+ samples, P&E recalibrated the mean for each of the property standards.

Recalibrating the mean on large data sets is common practice and provides much more reliable mean and standard deviation values, even though the data may all be from the same laboratory. In order to recalibrate the mean and standard deviations, the initial mean for each standard was calculated, as well as the standard deviation. All values greater or less than 2 standard deviations from the new mean were removed from the data set and a new mean and standard deviation were calculated. Results from this were graphed in order to view the warning limits (±2 standard deviations from the mean) and the tolerance limits (±3 standard deviations from the mean). All values falling within the warning limits were considered acceptable and those falling outside the tolerance limits were declared failures.

Of the 3,523 data points in the 8 property standards, there were 6 failures for Ag (0.002%), 11 failures for Pb (0.003%), and 0 failures for Zn. The impact these failures have on the data quality is zero.

14.1.2 CERTIFIED REFERENCE MATERIALS

Silver Standard also purchased 3 certified reference materials which they interspersed with the property standards. These standards were known as Standards 3, 4, and 5. The initial round robin characterization for these certified materials involved only 2 laboratories and from between 5 samples total to 16 samples total. P&E is of the opinion that the lack of data in the round robin led to a characterization of these standards that was completely insufficient for monitoring laboratory accuracy. Silver Standard's own data set for these standards (Standard 3 = 172 data points, Standard 4 =122 data points, and Standard 5 =166 data points) contained enough data that recalibration of the mean and standard deviation for each standard was possible and yielded much more representative statistics.

14.1.3 BLANKS

Silver Standard used three different blanks throughout the drill programs. Blank 1 came from an area immediately west of the South Ridge Zone and immediately south of the Breccia Ridge Zone before these zones were discovered. After this material was found to return values greater than three times the detection level for Ag, Pb, and Zn, Blank 1 was dispensed. Silver Standard procured a second blank material (Blank 2) from a dacite tuff located approximately 3.8 km west of the mineralized areas. The third blank material (Blank 3) was sourced from an intermediate volcanic located approximately 6 km west of the mineralized areas.

14.1.4 FIELD DUPLICATE DATA

The field duplicate data have very poor precision, particularly for Ag. The deposit statistics confirm this with coefficients of variation ranging from 1.6 to 3.7 (a well behaved disseminated deposit has a coefficient of variation from between 1.0 to 1.5). Based on the field duplicate results and the statistics, there is likely a nugget effect responsible for this. Silver Standard was advised in August 2008 to investigate this further.

It is P&E's opinion that the sample preparation, security, and analytical procedures were satisfactory.

15.0 ADJACENT PROPERTIES

Some of the information and geological knowledge regarding the Pitarrilla property is extracted from past reports, which involved an area larger than the present day property. Specifically, Monarch Resources trenched and drilled a gold fluorite showing called Fluorite Mine that is part of the present Pitarrilla claim block. The Fluorite Mine area and two other showings in that area are the main gold occurrences in the Cerro Pitarrilla area.

16.0 METALLURGICAL TESTING AND MINERAL PROCESSING

16.1 METALLURGICAL TESTING

16.1.1 INTRODUCTION

Wardrop designed the Pitarrilla project process plant to treat sulphide ores mined from underground at a rate of 1,400,000 t/a, or 4,000 t/d for 350 d/a. The plant design is based on metallurgical testwork performed by G&T Metallurgical Services Ltd. (G&T) (2007, 2008), Pocock Industrial, Inc. (Pocock) (2008), and Contract Support Services Inc. (CSS) (2008). The testwork results showed that saleable lead and zinc concentrates could be produced using conventional comminution and flotation processes.

Silver Standard initially produced five composite samples of different ore types from the mineralized zone on the Pitarrilla property for metallurgical testwork. Subsequently, additional samples were received and tested. Although initial information received from Silver Standard indicated that the processing plant feed material would consist of only Basal Conglomerate and Sediments ore, the revised mine plan of April 2008 also included small amounts of C-Horizon and Andesite ore types. This section will focus on the results obtained from all the ore samples.

The feed to the process plant will contain sulphide minerals composed of lead, zinc, silver, and minor copper. Lead and zinc are present as galena and sphalerite, respectively. Silver was generally recovered with galena, although no specific silver minerals were identified. However, as a result of the presence of arsenic and antimony, silver bearing sulphosalts are a possibility (G&T, 2008-I). The copper occurs predominantly as chalcopyrite.

The feed to the plant was initially planned to have a nominal head grade of 0.87% Pb, 1.96% Zn, and 102 g/t Ag. The metal recoveries were estimated accordingly to be as follows:

- lead recovery of 86.5%, with a grade of 63% Pb
- zinc recovery of 94.1%, with a grade of 49% Zn
- silver recovery of 83.6%, with grades of:
  - 5,980 g/t Ag (71.3% recovered), in the lead concentrate
  - 355 Ag g/t (12.3% recovered), in the zinc concentrate.

The changes to the mine plan resulted in the average life of mine feed grade changing to have a nominal head grade of 1.12% Pb, 2.57% Zn, and 171 g/t Ag. The metal recoveries for the revised mine plan were estimated to be as follows:

- lead recovery of 89.6%, with a grade of 60.7% Pb
- zinc recovery of 93.2%, with a grade of 48.5% Zn
- silver recovery of 88.4%, with grades of:
  - 8,087 g/t Ag (77.2% recovered), in the lead concentrate
  - 402 Ag g/t (11.2% recovered), in the zinc concentrate.

The bulk of the silver will be recovered in the lead concentrate, which will add to the value of this product, while the silver present in the zinc concentrate will garner an economical bonus depending on the smelter that the zinc concentrate is sold to.

16.1.2 Metallurgical Testwork Review

Process Design Testwork Description

The metallurgical testwork carried out for the Pitarrilla project covered:

- the aspects of mineralogical evaluation
- rougher flotation processes
- concentrate regrinding processes
- cleaner flotation processes
- reagent use for selective flotation of the relevant sulphide minerals
- flocculant screening
- gravity sedimentation/thickening
- pulp rheology
- pressure and vacuum filtration.

The purpose of the testwork was to determine the optimum processing treatment route for the Pitarrilla material.

The initial G&T testwork reported in January and April 2007 (G&T, 2007-I and -II respectively) showed favourable flotation for the desired minerals contained in the given composite samples.

Testwork reported in March 2008 (G&T, 2008-I) focused on five ore composites. The goals of the initial tests of this portion of the test program were to achieve optimum flotation results from the ore samples supplied from the Pitarrilla project. The new composite samples were considered to be more representative of the orebody, and were used in all the subsequent testwork.

The testwork carried out in August 2008 by G&T (G&T, 2008-II) focused on improving metallurgical processes as well as confirming the testwork results previously obtained from the five composite samples developed during the March 2008 testwork program (G&T, 2008-I).

Pocock performed sedimentation and filtration testwork in 2008 using sample material generated by G&T during the pilot plant test program, which followed the August 2008 test program (G&T, 2008-III).

CSS performed simulations in 2008 for optimum mill sizing using test results received from SGS Mineral Services, Santiago, Chile, for Basal Conglomerate and Sediments composite samples.

16.1.3 Reports Reviewed

Wardrop developed the process design criteria (PDC) for the Pitarrilla project based on the following testwork reports.

January 2007 – G&T

G&T initially conducted batch rougher and open circuit cleaner flotation tests on composite samples from the Breccia Ridge mineralized zone. The results showed that moderate grade lead and zinc concentrates could be produced from the three composite samples tested (G&T, 2007-I).

April 2007 – G&T

G&T conducted locked cycle flotation tests, and regrinding tests of the bulk copper/lead and zinc concentrates, with the three Breccia Ridge composite samples. The locked cycle tests confirmed acceptable flotation results, and produced a lead concentrate with an average grade of about 54% Pb and containing about 9 kg/t Ag, and recoveries of approximately 83 and 73%, respectively. The zinc concentrate averaged around 50% Zn with a recovery of almost 88% (G&T, 2007-II).

March 2008 – G&T

G&T completed batch rougher flotation tests, cleaner flotation tests, and additional locked cycle tests on five composite samples ground to a nominal particle size P_{80} of 120 µm. The average batch flotation cleaner testwork resulted in 91% of the lead and 85% of the zinc recovered into lead and zinc concentrates assayed at 64% Zn and 51% Pb, respectively. The locked cycle tests confirmed earlier results, with 90% of the lead and zinc recovered at a grade of 68% Pb and 50% Zn. The copper minerals identified as present in the ore samples were chalcopyrite, with chalcocite, covelite, enargite, and tetrahedrite/tennantite (G&T, 2008-I).

AUGUST 2008 – G&T

G&T conducted batch rougher flotation and cleaner flotation tests, and locked cycle tests on Basal Conglomerate and Sediments composite samples ground to P_{80} values of between 100 µm and 276 µm.

The best testwork results for the Sediments composite sample gave a lead concentrate grade of 67.2% Pb at 87% recovery and 5,898 g/t Ag at 74.3% recovery, and a zinc concentrate grade of 48.3% Zn with 92% recovery. The ore was ground to a nominal P_{80} of 218 µm.

The best testwork results for the Basal Conglomerate composite sample, ground to a nominal P_{80} of 191 µm, gave a lead concentrate grade of 59.7% Pb at 85.9% recovery and 6,055 g/t Ag with 68.2% recovery, and a zinc concentrate grade of 49.5% Zn with 96.2% recovery. Table 16.11 shows further detail (G&T, 2008-II).

AUGUST 2008 – POCOCK

Pocock performed sedimentation and filtration testwork using sample material generated during testwork conducted by G&T during August 2008. Pocock gave a recommended thickener underflow of between 58 and 62% solids (Pocock, 2008).

SEPTEMBER 2008 - CSS

This report provided simulation information on optimum mill sizing based on process criteria of a plant operating at 10,000 t/d. The simulation was based on Basal Conglomerate and Sediments composite samples supplied by Silver Standard to SGS Lakefield, Chile. The report confirmed that a grinding circuit, consisting of a single 8.53 m diameter x 3.66 m (28 ft x 12 ft) SAG mill followed by a single 5.03 m diameter x 7.32 m (16.5 ft x 24 ft) ball mill, would achieve a throughput of 438.6 t/h at a final grind size P_{80} of 200 µm (CSS, 2008). The results were not used in the design of the mills for the 4,000 t/d plant.

16.1.4 *TESTWORK PROGRAM COMPONENTS*

Wardrop selected the following testwork components from the testwork reports for use in the design of the Pitarrilla processing facility:

- origin of samples
- head analyses and specific gravity determination
- mineralogical studies

- comminution
- flotation
- settling and filtration testwork:
  - flocculant screening
  - gravity sedimentation
  - pulp rheology
  - pressure filtration.

16.1.5 ORIGIN OF TEST SAMPLES

Testwork was reported to have been conducted on assay laboratory rejects and drill core samples from the Breccia Ridge mineralized zone. G&T received the samples from Silver Standard in July and August 2007.

16.1.6 HEAD ANALYSES AND SPECIFIC GRAVITY DETERMINATION

INTRODUCTION

Laboratory assay rejects and drill core samples were used to prepare five composite samples for testwork. However, according to Silver Standard, the initial processing of plant feed would consist predominantly of only Basal Conglomerate and Sediments ore types. Therefore this report will focus mainly on the results obtained in the testing of only these ore type samples. A detailed discussion is presented in the following sections.

HEAD ASSAY ANALYSIS RESULTS

March 2008 – G&T (G&T, 2008-I)

Table 16.1 shows the head assay results of five composite samples. According to the project report by G&T in 2008, composite D was a C Zone laboratory assay reject sample and is therefore not described in detail in this report. The procedure involved a stage screening and crushing to 4 mm. The crushed material was homogenized and prepared into 2 kg charges, with a representative cut being removed from each composite for head assay sample chemical analysis. The samples were sealed under nitrogen in sample bags to inhibit oxidation.

Table 16.1 shows the head assays of the various samples highlighting the Basal Conglomerate and Sediments average values obtained, respectively, as well as the overall average head grades. The overall average grade of the five composite samples was 0.37% Cu, 1.57% Pb, 2.99% Zn, and 290 g/t Ag. Samples A and B (Basal Conglomerate) and C and E (Sediments) were considered to be the most representative of the orebody, with average grades of 0.75% Cu, 1.60% Pb, 3.45%

Zn, and 472 g/t Ag, and 0.14% Cu, 1.87% Pb, 3.34% Zn, and 185 g/t Ag, respectively. These grades are significantly higher than the head grades given in the original and revised mine plan.

Table 16.1 Head Assay Results for Pitarrilla Ore Testwork (G&T, 2008-I)

Composite	Head Assay Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Sample ID	Ore Type
A	0.17	1.89	4.50	646	PD 186 BC Reject	BC Laboratory Reject
B	1.33	1.30	2.40	297	PD 181 BC Reject	BC Laboratory Reject
Average	**0.75**	**1.60**	**3.45**	**472**	**Basal Conglomerate Average**	
C	0.09	2.67	5.23	169	PD 157 Seds Reject	Seds Laboratory Reject
E	0.19	1.07	1.44	200	PD 182 Seds 46762 - 46789 Core	Seds Drill Core
Average	**0.14**	**1.87**	**3.34**	**185**	**Sediments Average**	
D	0.05	0.94	1.39	139	P 173 C-Zone Reject	C-Zone Laboratory Reject
BC & Seds Average	0.45	1.73	3.39	328	* not including C-Zone	
Total Average	**0.37**	**1.57**	**2.99**	**290**	*** including C-Zone**	

Note: BC = Basal Conglomerate Composite Sample
Seds = Sediments Composite Sample

Table 16.2 shows the effect of head ore variability in metal recovery. The results are from locked cycle tests that were conducted and are intended to simulate conditions projected to be present in the continuous flotation processing of the full production facility. The results from the Basal Conglomerate sample indicate that the zinc recovery rate will decrease with an increasing copper head ore grade. The results are shown graphically in Figure 16.1.

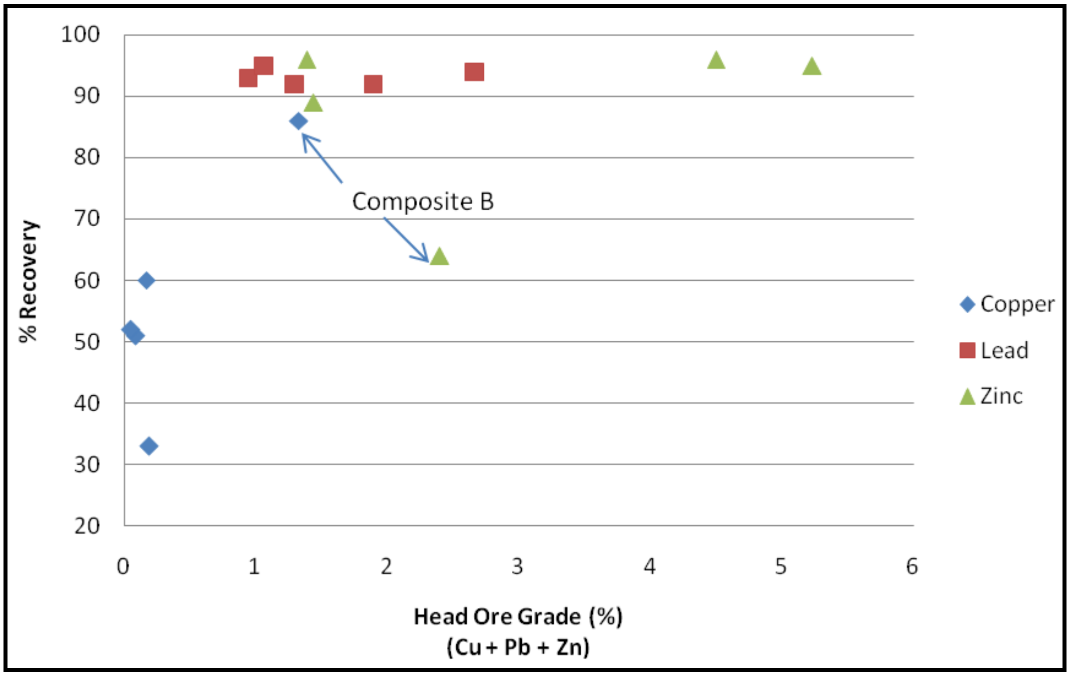
Table 16.2 Locked Cycle Test Performance Data (G&T, 2008-I)

Comp.	Sample ID	Product Concentrate	Assay Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Distribution (%) Cu	Pb	Zn	Ag
A	PD 186 BC Reject	Pb	3.66	64.10	4.10	18482	60	92	3	83
		Zn	0.70	0.79	51.70	1055	33	3	96	14
B	PD 181 BC Reject	Pb	18.80	19.00	6.90	4164	86	92	19	49
		Zn	2.43	0.45	44.90	385	6	1	64	4
C	PD 157 Seds Reject	Pb	1.43	73.30	4.38	3995	51	94	3	84
		Zn	0.43	1.05	51.40	150	39	3	95	8
E	PD 182 Seds Core	Pb	4.13	66.30	1.42	9046	33	95	1	69
		Zn	4.02	0.43	47.10	1666	58	1	89	23
D	DP 173 C Zone Reject	Pb	1.78	69.40	2.02	10624	52	93	2	89
		Zn	0.64	0.65	51.10	316	38	2	96	5

Note: Abbreviations are the same as Table 9.1.

Figure 16.1 illustrates that the lead and zinc recoveries appear to be insensitive to fluctuations of head ore grade. The variability in the grades of individual samples is apparent.

Figure 16.1 Relationship between Ore Grade and Recovery (G&T, 2008-I)



August 2008 – G&T (G&T, 2008-II)

This portion of the testwork performed by G&T utilized samples from the Basal Conglomerate and Sediments ore types. Samples used in the testwork were assayed for head analysis and the results are shown in Table 16.3.

It is apparent that the material tested in this test program has a significantly lower grade for both Basal Conglomerate and Sediments ore types when compared with the earlier testwork material. The grades are also significantly lower than those given in the revised mine plan.

Table 16.3 Head Assay Results for Pitarrilla Ore Testwork (G&T, 2008-II)

Sample ID – KM2232	Assay			
	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)
23 BC	0.17	0.99	2.61	129
24 BC	0.15	0.92	2.47	120
25 BC	0.16	0.97	2.59	121
29 BC	0.15	1.01	2.65	115
35 BC	0.16	1.02	2.78	123
37 BC	0.16	1.06	2.82	120
41 BC	0.17	0.97	2.84	121
47 BC	0.18	1.07	2.83	137
BC Average	**0.16**	**1.00**	**2.70**	**123**
17 Seds	0.07	0.68	1.12	78
18 Seds	0.08	0.73	1.17	80
31 Seds	0.07	0.67	1.1	73
32 Seds	0.07	0.73	1.1	74
38 Seds	0.07	0.69	1.13	74
42 Seds	0.07	0.67	1.13	66
50 Seds	0.07	0.66	1.08	66
Seds Average	**0.07**	**0.69**	**1.12**	**73**
Average of BC & Seds	**0.12**	**0.86**	**1.96**	**100**

Note: Abbreviations are the same as Table 9.1.

SPECIFIC GRAVITY RESULTS

Table 16.4 lists the measured specific gravity values obtained for the various composite samples, along with the arithmetic average.

Table 16.4 Specific Gravity Measurements (G&T, 2008-I)

Sample ID	Specific Gravity
PD 181 BC-reject	3.00
PD 186 BC-reject	3.02
BC Average	**3.01**
PD 157 Sediments-reject	2.98
PD 182 Sediments-core	2.85
Sediments Average	**2.91**
PD 173 C Zone-reject	2.65
Average of BC & Sediments	**2.96**
Average	**2.85**

Note: Abbreviations are the same as Table 9.1.

The arithmetic average specific gravity value used in the process design is 2.85.

DISCUSSIONS AND CONCLUSIONS

Sample preparation was conducted under standard conditions and unused samples were packed in a nitrogen atmosphere to minimize oxidation.

Based on information initially obtained from Silver Standard, Wardrop used lead, zinc and silver head grades of 0.87%, 1.96%, and 102 g/t, respectively, in the design of the Pitarrilla processing facility. These values correlated very well with the average head assay results from the G&T 2008-II testwork as shown in Table 16.3. However, the April 2008 revised mine plan head grades were significantly higher; subsequently, the interpretation of testwork data and design criteria were amended to reflect these higher ore grades.

The specific gravity of the ore used in the process design criteria was 2.85, which equals the arithmetic average of the values of samples measured in the 2008 testwork (G&T, 2008-I). However this value may be low when compared with the average value for the Basal Conglomerate and Sediments, which is 2.96.

16.1.7 *MINERALOGICAL STUDIES*

INTRODUCTION

G&T conducted basic mineralogical assessments on the various samples from the Pitarrilla property, and the results of their investigations are discussed below.

G&T, 2007-I

Initial testwork on the Breccia Ridge mineralized region was performed on three composite samples made up of Basal Conglomerate, Sediments, and Breccia Contact samples. The mineralogical evaluation found the dominant mineral to be pyrite, with secondary galena and sphalerite. No silver minerals were found although strong correlations were found between silver and galena contents during the flotation testwork. This indicates that the silver is mainly hosted by the galena as small inclusions or possibly as solid-solution substitution in the galena lattice. The presence of up 700 g/t Ag in the zinc concentrate indicates that a different silver, or silver-bearing mineral, may be present.

The average liberation of sphalerite and galena was found to be between 50 and 60% when ground to P_{80} values of between 91 and 103 µm. It was concluded that the ore could be successfully processed with a standard flotation process at a generally coarse feed size, estimated at between 150 and 200 µm. All of the testing was conducted at a P_{80} of 100 µm, with a recommendation that future testwork should include a coarser grind size.

G&T, 2008-I

G&T conducted mineral fragmentation studies of the five composite samples, particularly with respect to liberation and mineral composition. G&T used the percent liberation by size and class data to determine the minerals in the composites (G&T, 2008-I). The minerals found in the composite samples are listed in Table 16.5.

Table 16.5 Mineral Composition of Feed Sample (G&T, 2008-I)

Sample ID	Composite	Mineral Assay Cs (wt%)	Ga (wt%)	Sp (wt%)	Py (wt%)	Gn (wt%)
PD 186 BC Reject	A	0.5	2.1	7.4	3.0	87.0
PD 181 BC Reject	B	3.3	1.2	3.6	9.9	82.0
PD 157 Seds Reject	C	0.3	2.5	7.1	3.5	86.6
PD 182 Seds 46762 - 46789 Core	E	0.7	1.6	2.0	1.8	93.9
P 173 C Zone Reject	D	0.2	1.2	2.6	4.4	91.6

Note:
Cs: Chalcopyrite, Tetrahedrite/Tennantite, Enargite, Chalcocite, and Covellite
Ga: Galena
Sp: Sphalerite
Py: Pyrite, Pyrrhotite and Arsenopyrite
Gn: Gangue, inclusive of Magnetite, Hematite, and Goethite.

Table 16.6 shows the mineral liberation estimates in two dimensions for the composite samples.

Table 16.6 Mineral Fragmentation for Feed Composites (G&T, 2008-I)

Sample ID	Composite	Primary Grind P_{80} (µm)	Liberation (% in 2 Dimensions) Cs	Ga	Sp	Py	Gn
PD 186 BC Reject	A	116	30	72	72	62	97
PD 181 BC Reject	B	128	49	37	53	52	98
PD 157 Seds Reject	C	113	20	91	83	48	98
PD 182 Seds 46762 - 46789 Core	E	100	51	72	61	52	97
P 173 C Zone Reject	D	111	42	70	86	58	98

Note: Abbreviations are the same as Table 9.1.

A mineralogical evaluation at a nominal P_{80} size of 120 µm confirmed that a coarser flotation feed size could be used with virtually no loss in rougher flotation performance (G&T, 2008-I) as 98% of the non-sulphide gangue host minerals had been effectively liberated at this stage. The results also indicated that flotation plants typically operate satisfactorily with galena and sphalerite liberation levels of 50 to 60%, which are significantly lower than those obtained at the 120 µm grind size evaluated. This observation implies the prospect of effectively using a coarser primary grind when processing this ore. Also, it was found that a significant portion of unliberated sphalerite and galena was locked in binary or multiphase assemblages rich in gangue (G&T, 2008-I) and would therefore require a regrind stage to help liberate and reject gangue thereby upgrading the rougher flotation concentrates.

16.1.8 COMMINUTION

G&T and CSS conducted basic grinding testwork on the Basal Conglomerate and Sediments composite samples.

G&T, 2007-II

G&T performed formal ball mill work index testing on the Basal Conglomerate and Sediments composite samples. The testwork gave ball mill work index values of 16 kWh/t and 20 kWh/t, for the Basal Conglomerate and Sediments composites, respectively. A comparative Bond Work Index (BWi) test was undertaken with the Breccia Contact sample, as shown in Table 16.7, giving an average BWi of 21.4 kWh/t.

Table 16.7 Bond Work Index Test Results (G&T, 2007-II)

Sample Origin	P_{80}	BWi	Comparitive Wi (average)
Basal Conglomerate	87	15.8	-
Sediments	106	20.0	-
Breccia Contact	125	-	21.4
Average BC and Seds		**17.7**	

A Bond Ball Mill Work Index value of 17.7 kWh/t has been assumed in the design of the grinding circuit based on the average test results obtained for the Basal Conglomerate and Sediments samples.

CSS, 2008

CSS conducted simulation tests to determine the optimum sizing for the SAG and ball mills using the following design parameters:

- SAG feed rate ....10,000 t/d
- SAG feed.......................................150 mm
- ball mill work index (assumed).......21.0 kWh/t
- plant availability..............................95%.

The simulation determined that the minimum dimensions for the SAG mill should be 8.53 m diameter x 3.66 m (28 ft x 12 ft), and 5.03 m diameter x 7.32 m (16.5 ft x 24 ft) for the ball mill for a 10,000 t/d throughput. This design was not utilized since the tonnage and work index used for the final project design changed to 4,000 t/d and 17.7 kWh/t, respectively.

16.1.9 Flotation

Introduction

The preliminary testwork performed by G&T (G&T, 2007-I&II) found favourable flotation results of the targeted sulphide minerals contained in the three composite samples tested. Throughout 2007, resource development progressed and additional samples became available for testwork, which were subsequently used in further testwork conducted by G&T in 2008. The results obtained from the five composite samples tested are summarized in this section.

Composite D is included for reference only and is not included in the discussion of results since it is a C-Zone (C-Horizon) sample, which will not be used for the plant feed (as per Silver Standard) although the April 2008 mine plan incorporates a small amount of this type of ore.

Initially only the results from Basal Conglomerate and Sediments samples were reviewed but this been revised to incorporate the results obtained from the testing of higher grade samples and all the ore types.

Sulphide Flotation

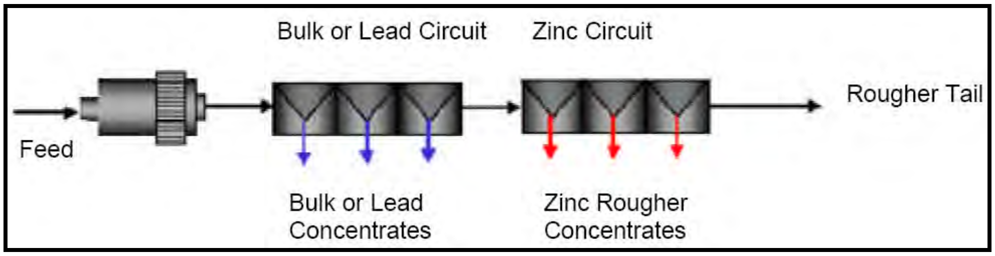
The testwork conducted by G&T (G&T, 2008-I) on the five composite samples included the following tests:

- rougher tests (open cycle)
- cleaner tests (open cycle)
- locked cycle tests.

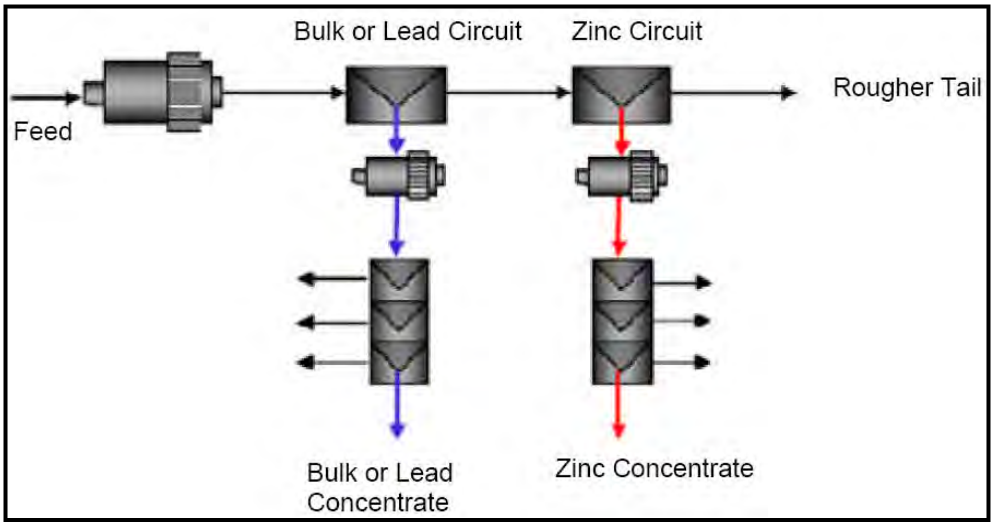
The three flowsheets for the processes are displayed in Figure 16.2. The rougher tests were used to produce a bulk copper/lead concentrate, and a zinc rougher concentrate. The cleaner tests were conducted on reground concentrates subjected to various stages of cleaning to give final cleaner concentrates. The locked cycle tests recycled the second and third cleaner tailings to simulate the full plant process. The collector reagent 3418A, a dithiophosphinate, was used as the principal collector for the copper and lead sulphide flotation, while sodium isopropyl xanthate (SIPX), was used as the collector reagent in the zinc sulphide flotation. Lime and sodium cyanide were used to depress the flotation of pyrite and sphalerite into the copper/lead concentrate, while copper sulphate was used as the activator for the zinc flotation stage. Lime was used for pH control.

Figure 16.2 Process Flowsheets used in G&T Testwork (G&T, 2008-I)

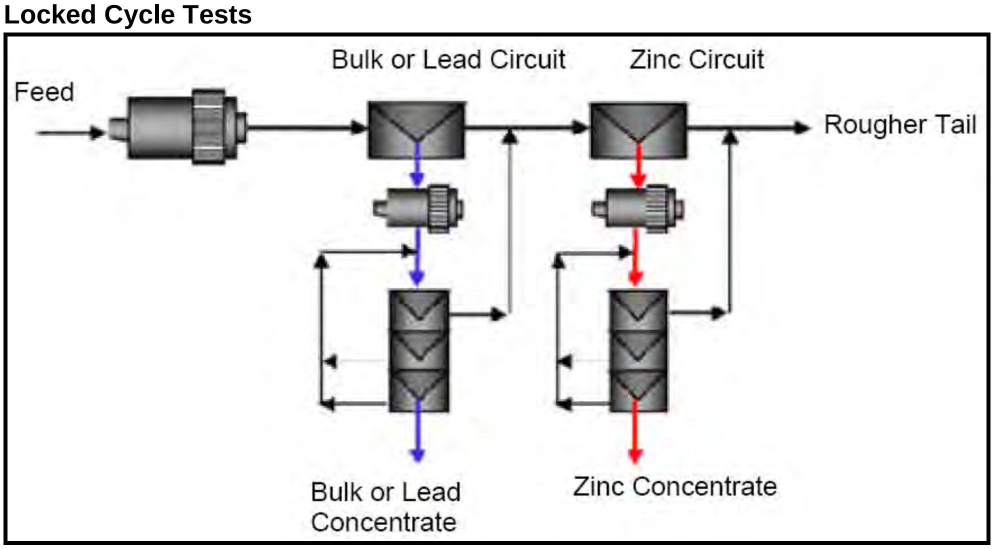
Batch Rougher Tests



Batch Cleaner Tests



Locked Cycle Tests



Rougher Test Results

The five composite samples were tested in the batch rougher flotation process. Table 16.8 shows the solids mass and metal recoveries into the bulk copper/lead and zinc rougher concentrates.

Table 16.8 Summary of Rougher Flotation Results – G&T, 2008-I

		Bulk or Copper/Lead Concentrate					Zinc Rougher Concentrate		
			Grade		Percent Recovery			Grade	Percent Recovery
Sample ID	**Comp.**	**Mass (%)**	**Cu (%)**	**Pb (%)**	**Cu**	**Pb**	**Mass (%)**	**Zn (%)**	**Zn**
PD 186 BC Reject	A	9.5	1.8	19.9	83	96	12.7	25.8	99
PD 181 BC Reject	B	15.7	7.8	7.2	92	95	9.8	9.5	99
Basal Conglomerate Average		**12.6**	**4.8**	**13.6**	**88**	**96**	**11.3**	**17.7**	**99**
PD 157 Seds Reject	C	9.1	0.9	30.4	68	97	15.6	27.6	99
PD 182 Seds Core	E	5.1	4.0	22.6	92	97	10.9	11.6	77
Sediments Average		**7.1**	**2.5**	**26.5**	**80**	**97**	**13.3**	**19.6**	**88**
P 173 C Zone Reject	D	8.0	0.5	10.2	77	94	9.2	11.6	99
Average Total		**9.5**	**3.0**	**18.1**	**82**	**96**	**11.6**	**17.2**	**95**
Basal Conglomerate and Sediments Average		**9.9**	**3.6**	**20.0**	**84**	**96**	**12.3**	**18.6**	**94**

For the open cycle batch rougher tests, the Basal Conglomerate Average shows a lead recovery rate of 96% in approximately 12.6% mass, and a zinc recovery of 99% in 11.3% recovered mass. The results from Table 16.1 also indicate that feed grade variations between the two samples leads to variations in copper, lead and zinc grade, and the amount of mass recovery.

The Sediments Average shows a lead recovery of 97% in 7.1% mass recovery and a zinc recovery of 88% in a 13.3% mass recovery. Variations in recoveries between Sediments samples can be traced back to variations in feed grade.

The average of the five composites gave a lead recovery of 96% containing 9.5% of the feed mass, and a zinc recovery of 95% at 11.6% of the feed mass. The average of the Basal Conglomerate and Sediments resulted in a lead recovery of 96% with a 9.9% portion of the feed mass, and a zinc recovery of 94% having 12.3% of the feed mass. When compared with the other results, the Basal Conglomerate Average and Sediments Average had higher grades and percent feed mass than the total average, because of the lower head grade of the C-Zone composite sample. The Basal Conglomerate Average and Sediments Average had a lower zinc grade and recovery than the Sediments Average as a result of the Sediments samples containing low head values of copper. The Basal Conglomerate Average and

Sediments Average returned a lower copper grade and recovery than the Basal Conglomerate Average, as a result of the Basal Conglomerate samples containing a higher head grade of copper.

Cleaner Test Results

The five composite samples were tested in the open circuit cleaner process. Table 16.9 summarizes the results of the tests.

Table 16.9 Batch Cleaner Average Test Results – G&T, 2008-I

Sample ID	Composite	Concentrate	Grade				Recovery (%)			
			Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu	Pb	Zn	Ag
PD 186 BC Reject	A	Lead	3.8	64.1	4.4	18,350	57	90	3	77
		Zinc	0.5	0.4	50.4	866	21	2	82	10
PD 181 BC Reject	B	Lead	20.9	18.8	4.4	4,188	88	87	11	77
		Zinc	1.2	0.4	43.9	327	3	1	54	3
Basal Conglomerate Average		**Lead**	**12.4**	**41.5**	**4.4**	**11,269**	**73**	**89**	**7**	**77**
		Zinc	**0.9**	**0.4**	**47.2**	**597**	**12**	**2**	**68**	**7**
PD 157 Seds Reject	C	Lead	1.3	63.1	6.4	3,711	45	93	5	84
		Zinc	0.5	0.5	50.2	130	33	2	83	6
PD 182 Seds Core	E	Lead	2.0	65.8	3.2	9,764	49	86	3	84
		Zinc	0.5	0.6	51.1	243	22	2	83	4
Sediments Average		**Lead**	**1.7**	**64.5**	**4.8**	**6,738**	**47**	**90**	**4**	**84**
		Zinc	**0.5**	**0.6**	**50.7**	**187**	**28**	**2**	**83**	**5**
P 173 C-Zone Reject	D	Lead	4.2	64.0	2.0	8,666	62	94	2	75
		Zinc	0.8	0.3	50.8	792	23	1	91	13
Average - Total		**Lead**	**6.4**	**55.2**	**4.1**	**8,936**	**60**	**90**	**5**	**79**
		Zinc	**0.7**	**0.4**	**49.3**	**472**	**20**	**2**	**79**	**7**
Basal Conglomerate and Sediments Average		**Lead**	**7.1**	**53.0**	**4.6**	**9,004**	**60**	**90**	**6**	**81**
		Zinc	**0.7**	**0.5**	**49.0**	**392**	**20**	**2**	**76**	**6**

For the open cycle cleaner tests, the Basal Conglomerate Average results gave concentrates of 53% Pb and 49% Zn with 90% lead and 76% zinc recoveries, respectively. The lead concentrate contained on average 7.1% Cu, and 9 kg/t Ag. The zinc concentrate contained on average 0.7% Cu. Overall, it was found that the cleaner circuit could produce saleable grade lead and zinc concentrates based on the results obtained from the Sediments Average, which produced concentrates containing low contaminants, particularly the zinc concentrate with 0.5% Cu and 0.6% Pb. The Basal Conglomerate Average produced a lead concentrate with 12% Cu. This concentrate would possibly benefit from further processing to remove the copper either into a high grade copper concentrate, or by removing as much of the copper as possible into a recycle product should smelter conditions demand this.

Locked Cycle Tests

The composite samples were tested in locked cycle tests made to simulate the full production process. Table 16.10 reviews the results. Not included in the average results was Composite B, as it gave a bulk concentrate with 19% Cu, 19% Pb and 7% Zn, with 86%, 92% and 19% recoveries, respectively. Also, Composite B gave a low quality zinc concentrate with 45% Zn at 64% recovery. Ignoring the results of Composite B because of the high copper grade, the other composite samples gave an average lead concentrate of 68% Pb, with 94% recovery, and containing 3% Cu and 3% Zn. The average zinc concentrate contained 50% Zn, with 94% recovery, containing 1.7% Cu and 957 g/t Ag.

Table 16.10 Data Summary for Locked Cycle Tests – G&T, 2008-I

Sample ID	Product	Mass (%)	Assay Cu (%)	Assay Pb (%)	Assay Zn (%)	Assay Ag (g/t)	Distribution (%) Cu	Distribution (%) Pb	Distribution (%) Zn	Distribution (%) Ag
PD 181 BC Reject	**Comp A (Tests 35, 41)**									
	Flotation Feed	100.0	0.2	2.0	4.5	639	100	100	100	100
	Pb Concentrate	2.9	3.7	64.1	4.1	18,482	60	92	3	83
	Zn Concentrate	8.4	0.7	0.8	51.7	1,055	33	3	96	14
PD 186 BC Reject	**Comp B (Tests 34, 40)**									
	Flotation Feed	100.0	1.3	1.2	2.2	310	100	100	100	100
	Pb Concentrate	5.9	18.8	19.0	6.9	4,164	86	92	19	49
	Zn Concentrate	3.1	2.4	0.5	44.9	385	6	1	64	4
BC Average										
Pb Concentrate		4.4	11.2	41.6	5.5	11,323	73	92	11	66
Zn Concentrate		5.8	1.6	0.6	48.3	720	20	2	80	9
PD 157 Seds Reject	**Comp C (Test 36)**									
	Flotation Feed	100.0	0.1	3.2	5.7	196	100	100	100	100
	Pb Concentrate	4.1	1.4	73.3	4.4	3,995	51	94	3	84
	Zn Concentrate	10.5	0.4	1.1	51.4	150	39	3	95	8
PD 182 Seds Core	**Comp E (Test 51)**									
	Flotation Feed	100.0	0.2	1.1	1.5	201	100	100	100	100
	Pb Concentrate	1.5	4.1	66.3	1.4	9,046	33	95	1	69
	Zn Concentrate	2.8	4.0	0.4	47.1	1,666	58	1	89	23
Sediments Average										
Pb Concentrate		2.8	2.8	69.8	2.9	6,521	42	95	2	77
Zn Concentrate		6.7	2.2	0.7	49.3	908	49	2	92	16
PD 173 C-Zone reject	**Comp D (Test 33,39)**									
	Flotation Feed	100.0	0.2	1.1	1.5	201	100	100	100	100
	Pb Concentrate	1.5	4.1	66.3	1.4	9,046	33	95	1	69
	Zn Concentrate	2.8	4.0	0.4	47.1	1,666	58	1	89	23

table continues…

Sample ID	Product	Mass (%)	Assay Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Distribution (%) Cu	Pb	Zn	Ag
Average										
Pb Concentrate		2.8	3.1	67.9	3.3	10,508	48	94	2	79
Zn Concentrate		7.2	1.7	0.8	50.1	957	43	2	93	15
Basal Conglomerate and Sediments Average										
Pb Concentrate		3.6	7.0	55.7	4.2	8,922	58	93	7	71
Zn Concentrate		6.2	1.9	0.7	48.8	814	34	2	86	12

When compared with the open cycle cleaner flotation tests, the locked cycle Basal Conglomerate Average and Sediments Average produced very similar lead concentrate grades and recoveries. The zinc concentrate from the locked cycle test contained higher copper than the open cycle cleaner tests. The Sediments Average zinc concentrate contained a copper grade of 2.2%, which could possibly garner a smelter penalty. The Basal Conglomerate Average produced a lead concentrate with 11% Cu, similar to the Basal Conglomerate Average for the cleaner tests, but which could also lead to a smelter penalty.

Although the locked cycle Basal Conglomerate Average and Sediments Average gave a higher copper grade in the zinc concentrate, the test still confirms that the production of saleable concentrates from the Pitarrilla samples which were tested, is feasible.

G&T (G&T, 2008-II) tested Basal Conglomerate, Sediments, Andesite, and C-Horizon composites. Table 16.11 summarizes the results of the final locked cycle tests done on the four composite types. The results are included for completeness of reporting.

Table 16.11 Test Results Summary for Locked Cycle Tests – G&T, 2008-II

Composite/Test #	Mass %		Pb Concentrate Grade				Pb Concentrate Recovery				Zn Concentrate Grade				Zn Concentrate Recovery			
	Pb Conc	Zn Conc	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu %	Pb %	Zn %	Ag %	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu %	Pb %	Zn %	Ag %
Andesite/LCT30	0.63	1.44	0.49	48.4	17	5,120	24.4	70.4	10.8	51.9	0.28	1.9	49.4	661	32.7	6.5	72.3	15.4
Andesite/LCT49	0.64	1.96	0.52	48.6	13.1	5,394	17.9	69.6	8.0	45.2	0.35	2.1	43.3	731	36.9	9.3	81.0	18.7
C-Horizon/LCT36	0.74	2.47	2.45	53.8	7.4	7,989	35.8	71.8	3.4	58.1	0.87	1.1	50.1	551	42.2	4.9	76.5	13.3
C-Horizon/LCT48	0.76	3.24	2.50	53.5	8.2	5,734	35.8	70.4	3.4	47.3	0.77	1.1	48.8	766	47.1	6.4	87.1	26.9
BC/LCT37	1.78	4.86	6.94	47.8	6.4	4,972	67.6	77.0	3.9	72.1	1.00	0.5	49.9	246	26.7	2.2	83.4	9.8
BC/LCT47	1.50	5.45	5.35	59.7	3.0	6,055	51.8	85.9	1.6	68.2	1.19	0.7	49.8	303	41.6	3.5	96.3	12.4
Sediments/LCT42	0.90	2.13	3.65	65.7	2.8	5,898	42.9	86.7	2.2	74.3	1.78	1.5	49.1	406	49.7	4.6	92.1	12.1
Sediments/LCT50	0.90	2.16	2.87	65.2	2.4	5,563	37.4	87.7	2.0	74.9	1.56	1.2	47.7	367	48.7	3.9	95.6	11.8

The results from the tests on Basal Conglomerate and Sediments composite samples, which will make up the initial feed to the processing plant, have been averaged for use in the process plant design. For the testwork, the Basal Conglomerate and Sediments samples were ground to a nominal primary grind P_{80} of 191 µm and 218 µm, respectively, based on the findings and recommendations from the G&T testwork (G&T, 2008-I). Table 16.12 displays the results of the locked cycle tests that will be used for the design of the process plant, namely locked cycle tests numbered 47 and 50.

Table 16.12 Data Summary for Locked Cycle Tests – G&T, 2008-II

Composite (Test No.)	Product	Weight (%)	Grade				Percent Recovery			
			Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu	Pb	Zn	Ag
Basal Conglomerate (47)	Lead	1.50	5.4	59.7	3.0	6,055	51.8	85.9	1.6	68.2
	Zinc	5.45	1.2	0.7	49.8	303	41.6	3.5	96.3	12.1
Sediments (50)	Lead	0.90	2.9	65.2	2.4	5563	37.4	87.7	2.0	74.9
	Zinc	2.16	1.6	1.2	47.7	367	48.7	3.9	95.6	11.8
Average	**Lead**	**1.20**	**4.1**	**62.5**	**2.7**	**5809**	**44.6**	**86.8**	**1.8**	**71.6**
	Zinc	**3.80**	**1.4**	**1.0**	**48.8**	**335**	**45.2**	**3.7**	**96.0**	**12.1**

The average results from the two locked cycle tests give a lead concentrate containing 63% Pb at a recovery of 87%. The lead concentrate also contained 4.1% Cu, 2.7% Zn, and 5,809 g/t Ag, with a silver recovery of 72%. The averaged zinc concentrate grade was calculated at 49% Zn at a recovery of 96%. The zinc concentrate also contained 1.4% Cu, 1.0% Pb, and 335 g/t Ag. The results obtained are similar to the calculated grades and recoveries projected in the initial mine plan. However, the changes to the mine plan in April 2008 resulted in the review of the test data; the results are reported in a later section.

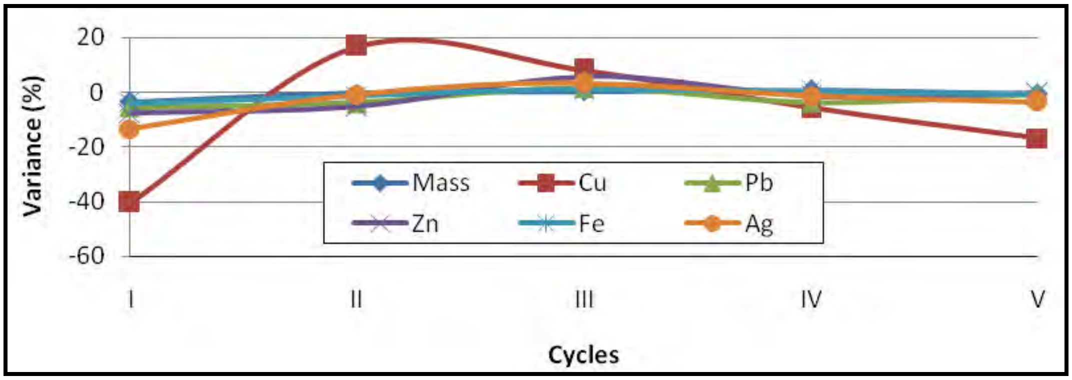
Locked Cycle Stability

G&T performed a stability analysis on all the locked cycle tests conducted in the test program. The stability of the test is determined by comparing the metals entering the process, and the metals leaving the process in the tailings and concentrate. The difference between the metal entering and leaving is the amount of metal being accumulated in the middlings streams. This amount of metal in the middlings stream is the measure of stability. The details for the Basal Conglomerate Test 47 are shown in Table 16.13 and Figure 16.3. Details for the Sediments composite Test 50 are shown in Table 16.14 and Figure 16.4.

Table 16.13 Basal Conglomerate Locked Cycle Stability Test – G&T, 2008-II

	Mass		Calculated Head Assay					Metal Unit Variances (%)				
Cycles	g/cycle	% Var.	Cu (%)	Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	Cu	Pb	Zn	Fe	Ag
Test KM2232-47 Basal Conglomerate												
I	1925	-3.5	0.11	1.05	2.71	15.1	123	-40.5	-5.5	-7.6	-4.1	-13.4
II	1990	-0.3	0.21	1.03	2.69	15.1	136	16.9	-3.9	-5.3	-1.3	-0.8
III	2001	0.3	0.19	1.08	2.98	15.3	141	8.0	1.6	5.7	1.0	3.3
IV	2010	0.7	0.16	1.02	2.77	15.2	134	-5.8	-3.7	-1.5	0.3	-1.3
V	1983	-0.6	0.15	1.07	2.83	15.2	133	-16.8	-0.7	-0.5	-0.7	-3.6
Average III to V	1998	0.1	0.17	1.06	2.86	15.2	136	-4.9	-0.9	1.2	0.2	-0.5
Total	**1996**	**-**	**0.18**	**1.07**	**2.83**	**15.2**	**137**	**-7.6**	**-2.4**	**-1.8**	**-1.0**	**-3.2**

Figure 16.3 Basal Conglomerate Locked Cycle Stability Test – G&T, 2008-II



Locked cycle tests are the preferred method to use for metallurgical projections. The initial indications are that the Basal Conglomerate locked cycle test was reasonably stable. This conclusion is based on the consistent amount of mass recovered to the final concentrates, and the relatively steady analysis of the metals in the products of the last three cycles.

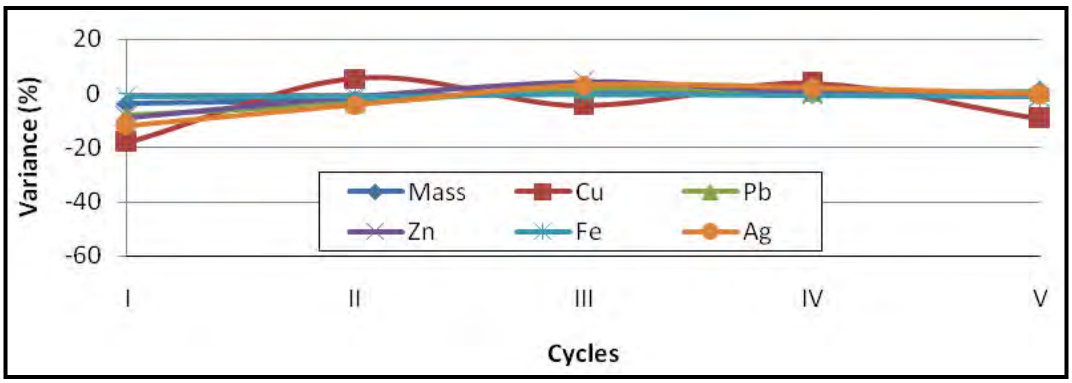
The steady state of the test is determined by both mass and metal reconciliation to the final products combined with metal stability (constancy). The reconciliation for the mass recovered to the final concentrate shows the test to be stable over the last three cycles. The reconciliation for copper metal units was reasonable while the reconciliation for lead, zinc, and silver units was considered to be good. The results showed that, with the exception of zinc, there was a small amount of build-up of metals occurring in the middling streams, apparently continuing even after the five cycles have been completed. However, this build-up is rather small, indicating that this test can be considered stable and constant thereby allowing for confident use of the results in metallurgical projections.

Figure 16.3 shows that some variations occurred over the entire test. The variation was minimal during the last three cycles, which points to good stability of the overall test.

Table 16.14 Sediments Locked Cycle Stability Test – G&T, 2008-II

	Mass		Calculated Head Assay					Metal Unit Variances (%)				
Cycles	g/cycle	% Var.	Cu (%)	Pb (%)	Zn (%)	Fe (%)	Ag (g/t)	Cu	Pb	Zn	Fe	Ag
Test KM2232-50 Sediments												
I	1950	-2.6	0.06	0.63	1.01	11.0	60	-18.1	-8.1	-8.8	-1.5	-11.9
II	1979	-1.1	0.08	0.65	1.08	10.8	64	5.5	-3.6	-1.2	-1.3	-4.3
III	1998	-0.2	0.07	0.68	1.13	10.8	68	-4.5	1.7	4.3	-0.3	3.0
IV	2002	0	0.07	0.67	1.08	10.8	68	3.8	0.6	0.0	-0.7	2.0
V	1991	-0.5	0.06	0.67	1.07	10.7	66	-9.3	0.5	-1.4	-1.4	-0.5
Average III to V	1997	-0.2	0.07	0.67	1.09	10.8	67	-3.3	0.9	1.0	-0.8	1.5
Total	**2002**	**-**	**0.07**	**0.66**	**1.08**	**10.8**	**66**	**-4.5**	**-1.8**	**-1.4**	**-1.0**	**-2.3**

Figure 16.4 Sediments Locked Cycle Stability Test – G&T, 2008-II



For the Sediments test, the initial indications showed this test could be regarded as reasonably stable. This conclusion is based on the steady mass recovery to the final concentrate, and the consistent analysis of metal assays in the products of the last three cycles.

The reconciliation for the mass recovered to the final concentrate showed that the test was stable over the last three cycles, and also stable over the whole test. The reconciliation for copper, lead, zinc, and silver metal units over the three cycles was found to be good. The results showed that there was a small amount of build-up of copper occurring in the middling streams, apparently even continuing after the five test cycles. However, the extent of this build-up was rather small while the lead, zinc, and silver metals do not show any build-up. All these factors indicate that the test can be considered to be stable and constant allowing for confident use of the results in metallurgical projections.

Figure 16.4 shows that some variations occurred over the entire test, but these variations are seen to be fairly minimal. The minimal amount of variation, particularly over the last three cycles, shows the test to be quite stable for the elements analyzed.

Locked Cycle – Primary Grind Size

Since primary grind is important in the process design, an evaluation of tests performed at the two grind extremes was undertaken, namely at 100 µm and 200 µm. Table 16.15 shows the locked cycle test data from the Basal Conglomerate and Sediments composites comparing results from the two grind sizes.

Table 16.15 Comparison of Locked Cycle Test Results with Primary Grind Size

			Grade				Percent Recovery			
Test Phase	**Concentrate**	**Weight (%)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**Ag (g/t)**	**Cu**	**Pb**	**Zn**	**Ag**
Fine Grind Size - P_{80} = 100 µm										
G & T, 2008-I	Lead	3.6	7.0	55.7	4.2	8922	58	93	7	71
	Zinc	6.2	1.9	0.7	48.8	814	34	2	86	12
Coarse Grind Size - P_{80} = 200 µm										
G & T, 2008-II	Lead	1.2	4.1	62.5	2.7	5809	45	87	2	72
	Zinc	3.8	1.4	1.0	49.0	335	45	4	96	12

In evaluating the results from the coarse grind size test (at 200 µm), the copper grade in the final lead concentrate was found to be reduced by 2.5% while the lead grade increased by 7.3%, when compared to earlier results. Zinc recovery into the zinc concentrate increased 8% with the coarser grind size, while the lead concentrate showed a slight decrease in copper and lead recovery. In total, the coarser primary grind did not seem to adversely affect the recovery rate or the grades of the concentrates produced.

LOCKED CYCLE TESTS - DISCUSSION

Flotation testwork conducted during 2008 (G&T, 2008-I) indicated that an average lead recovery of 96% and zinc recovery of 95% was attainable with rougher flotation testwork. These tests showed that further cleaning would be beneficial and would improve the concentrate purity. The test results show an improvement over the testwork originally conducted in 2007, where the lead recovery was estimated at 78 to 87% and zinc recovery was between 83 and 92% (G&T, 2007-II).

The subsequent cleaner tests confirmed that saleable lead and zinc concentrate grades could be produced from the Pitarrilla samples, namely 53% Pb in the lead concentrate and 49% Zn in the zinc concentrate.

Confirmatory testwork, including additional locked cycle tests conducted in 2008 (G&T, 2008-II), verified that the test process simulating plant conditions would produce saleable concentrate products. However, the two concentrates produced contained relatively high quantities of contaminating minerals. Thus the lead concentrate contained 7.0% Cu and 4.2% Zn, while the zinc concentrate had 1.9% Cu and 0.7% Pb. Other trace elements present in the concentrates are discussed further in this section. The effect of these concentrations of impurity elements may require more consideration regarding treatment, separation, and/or smelter penalties.

Furthermore, the presence of copper was seen to influence the zinc recovery and the grade of both concentrates, namely with the Basal Conglomerate Composite B (see Table 16.10 and Figure 16.1). This composite contained a copper head grade of 1.3%, whereas the other composite samples tested contained between 0.1 and 0.2% Cu. When compared with Basal Conglomerate Composite A, Composite B showed a reduction of lead grade by 45% and an increase of contaminating minerals, namely 15% Cu and 3% Zn, in the lead concentrate. The zinc concentrate showed an increase of 1% Cu, and a grade decrease of 7% Zn, when compared to Composite A. This shows that a further cleaning of the lead concentrate for the recovery of copper is required when the mine processes high grade copper ore in order to minimize the losses of lead and zinc through the increased presence of copper.

Locked cycle test data from the Basal Conglomerate and Sediments composites at two grind sizes was analyzed. The coarser primary grind at a P_{80} of 200 µm did not seem to adversely affect the recovery rate or the grades of the concentrates produced when compared with the fines grind P_{80} of 100 µm. Wardrop used a plant product size of P_{80} 200 µm, based on these results in the design of the processing facility.

The stability of the locked tests was also studied. Both the Basal Conglomerate and Sediments tests were found to be stable, with little metal accumulation in the middling streams.

The locked cycle tests confirmed that saleable lead and zinc products could be produced from the Basal Conglomerate and Sediments composite samples but that the presence of contaminating elements may result in smelter penalties being imposed.

The concentrates produced during the locked cycle tests were assayed and contained 1.0 to 4.1% contaminating minerals, which included copper and zinc in the lead concentrate, and copper and lead in the zinc concentrate (G&T, 2008-II). At such concentrations of contaminants, possible smelter fines could be imposed. Wardrop recommends additional testwork to further refine the metallurgical process in order to improve the selective flotation of sulphide minerals. To produce a higher grade product, further removal of contaminant minerals from the final concentrates may be required.

16.1.10 *Smelter Considerations*

Introduction

It is probable that the lead and zinc concentrates produced will be sold to the Torréon Smelter in Mexico. For the purpose of this study, no approach has been made by Silver Standard regarding smelter payment terms, smelter charges, and penalties for impurity elements present in the concentrates. Therefore, the following is only a general discussion of anticipated terms and conditions.

Analyses of Flotation Concentrates

The pilot plant test program conducted during 2008 produced sufficient quantities of lead and zinc concentrates for a detailed analytical evaluation in order to determine the impurity elements present. Table 16.16 gives a summary of the results obtained, while the results of the detailed analyses have been presented in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices". Two sets of results are shown for the zinc concentrates produced, one from Basal Conglomerate samples, and the other from Sediments samples. The lead concentrate was produced from Sediments samples.

The basic treatment charge will apply to both concentrates. This charge will depend on the grade of the concentrate, and will also be smelter dependent. The data in Table 16.16 provides an indication of the anticipated grades of the two concentrates that will be produced by the Pitarrilla plant; this will be discussed in general terms in the following section.

Table 16.16 Flotation Concentrates – Impurity Elements Analysis

Sample	Element									
	Ag (g/t-ICP)	As (ppm)	Au (g/t)	Bi (ppm)	Cl (ppm)	Cu (%)	Co (ppm)	F (ppm)	Fe (%)	Hg (ppm)
Lead Concentrate	5810	959	0.63	129	23	3.22	37	138	5.1	1
Zinc Concentrate - Basal Conglomerate	402	733	0.09	6	23	1.48	64	100	12.1	26
Zinc Concentrate - Sediments	309	264	< 0.01	16	30	1.17	43	33	13.3	6

Sample	Element									
	Mg (%)	Mo (ppm)	Ni (ppm)	Pb (%)	S (%-ICP)	S (%-Leco)	Sb (%)	Se (ppm)	Tl (ppm)	Zn (%)
Lead Concentrate	0.20	55	409	62.3	16.2	15.6	0.85	360	4	2.91
Zinc Concentrate - Basal Conglomerate	0.15	29	227	1.71	31.7	29.6	0.09	60	3	47.55
Zinc Concentrate - Sediments	0.04	19	51	1.16	34.1	29.8	0.09	90	3	49.00

Lead Concentrate

Generally, the lead smelter terms that could be anticipated are as follows:

- Pb: paid at 95% of content with a minimum deduction of 3%
- Ag: paid at 95% of content with a minimum deduction of 50 g/t
- Au: paid at 95% of content with a minimum deduction of 1 g/t.

The typical major penalty elements are:

- As: maximum 0.5% with penalty incurred for every 0.1% exceeding 0.5%
- Sb: maximum 0.5% with penalty incurred for every 0.1% exceeding 0.5%
- Bi: maximum 0.15% with penalty incurred for every 0.01% exceeding 0.15%
- Zn: maximum 10.0% with penalty incurred for every 1.0% exceeding 10.0%
- Hg: maximum 50 g/t with penalty incurred for every 10 g/t exceeding 50 g/t
- Cu: penalty limits for copper have not been ascertained.

Zinc Concentrate

Generally, the zinc smelter terms that could be anticipated will be based on the following:

- Zn: paid at 85% of content with a minimum deduction of 8%
- Ag: paid at 70% of the balance after 100 g/t has been deducted
- Au: paid at 70% of the balance after 1.5 g/t has been deducted.

The typical major penalty elements are:

- Fe: maximum 8.0% with penalty incurred for every 1.0% exceeding 8.0%
- Se: maximum 300 g/t with penalty incurred for every 100 g/t exceeding 300 g/t
- Hg: maximum 100 g/t with penalty incurred for every 100 g/t exceeding 100 g/t up to a maximum of 500 g/t, then an increased penalty applies for every 100 g/t exceeding 500 g/t
- As: maximum 0.1% with penalty incurred for every 0.1% exceeding 0.1%
- F: maximum 100 g/t with penalty incurred for every 1 g/t exceeding 100 g/t
- Mg: 0.3% with penalty incurred for every 0.1% exceeding 0.3%
- Cu: penalty limits for copper have not been ascertained
- Pb: penalty limits for lead have not been ascertained.

DISCUSSION

Assuming that the lead and zinc concentrates will be sold to a smelter having the above terms and penalty limits, a brief discussion follows.

Lead Concentrate

Credit will be obtained for the lead in the concentrate, as well as for the silver present. No credit will accrue from the gold present as this is present in quantities that are too low.

Regarding penalty elements, antimony will be subject to a penalty, although no penalty would be incurred for the presence of arsenic, bismuth, zinc, and mercury in the lead concentrate since these are all below the penalty limits.

In the absence of specific information, it is not known whether the relatively high amounts of copper present, namely 3.22%, will induce a penalty.

Zinc Concentrate

Credit will be obtained for the zinc in the concentrate, as well as for the silver present in excess of 100 g/t.

A penalty will be incurred for iron since this exceeds the 8.0% content limit. Fluorine will be a borderline penalty element if predominantly Sediments material is processed, although a blend of Basal Conglomerate and Sediments will reduce the possibility of penalties being imposed. No penalty limits would apply for selenium, mercury, arsenic, and magnesium.

In the absence of specific information, it is not known whether any form of penalty will be induced for the presence of copper, namely between 1.2 and 1.5%, or lead at levels between 1.2 and 1.7%.

The effect of impurity elements present in the Andesite and C-Horizon material slated for processing has not been established.

16.1.11 Revised Mine Plan – April 2008

INTRODUCTION

As mentioned previously, the mine plan was revised in April 2008 and this increased the grade of the plant feed material. The testwork was reviewed to obtain relevant metal recovery and concentrate grade information that could be related to the higher grade feed material.

Certain ore types were tested by G&T, which contained results arising from a range of feed grade values. A relationship was established linking the feed grade with the metal recoveries and the product grades. These revised recoveries and grades were used as indicated in the mine plan, affording an annual summary of results based on the ore types and head grades being treated.

In certain cases, insufficient data was available to establish any meaningful relations between the feed grades and metal recoveries and concentrate grades. In these cases, the best available results were used. The mine plan schedule has been included as Table 16.17. The test data used from the various locked cycle tests have all been summarized in Table 16.18.

As can be seen from the results, the recoveries and grades for lead and zinc have not changed very much, but the silver values are significantly different.

Table 16.17 Production Schedule including Assays, Recoveries, and Expected Concentrate Grades for Each Ore Type over the Life of Mine

Mining Method	Rock Type	Units	Production Year 1	2	3	4	5	6	7	8	9	10	11	12	Total (tonnes)
Longhole		t/a	724,000	820,000	820,000	820,000	820,000	820,000	820,000	820,000	820,000	820,000	820,000	814,368	9,738,368
Room & Pillar		t/a	550,000	580,000	580,000	580,000	580,000	580,000	580,000	580,000	580,000	580,000	580,000	585,525	6,935,525
Total		**t/a**	**1,274,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,399,892**	**16,673,892**
	Sediment	t	**724,000**	**431,453**	**120,000**	**338,638**	**599,602**	**940,000**	**820,000**	**1,070,264**	**936,133**	**880,406**	**1,084,037**	**993,372**	**8,937,905**
	Ag	g/t	185	194	185	169	193	178	174	175	170	148	159	165	172
	Zn	%	1.83	1.81	1.83	1.99	1.67	1.71	1.75	1.76	1.70	1.81	1.86	1.80	1.78
	Pb	%	1.21	1.23	1.21	1.44	1.07	1.15	1.14	1.02	1.10	1.30	1.23	1.19	1.17
	NSR	$/t	$70.82	$72.95	$70.82	$69.87	$70.16	$67.35	$66.68	$66.14	$65.03	$61.97	$64.61	$65.40	$66.71
	Basal Cong.	t	**550,000**	**968,547**	**1,280,000**	**1,061,362**	**490,000**	**460,000**	**580,000**	**329,736**	**463,867**	**206,583**	**120,016**		**6,510,110**
	Ag	g/t	220	146	148	152	167	167	162	157	171	192	123		162
	Zn	%	2.72	4.03	3.88	4.00	3.14	3.14	3.54	3.93	3.14	1.93	4.76		3.59
	Pb	%	1.02	0.97	0.98	0.97	0.99	0.99	0.99	0.99	0.94	0.88	0.59		0.97
	NSR	$/t	$80.87	$74.72	$73.89	$75.78	$72.18	$72.18	$74.31	$76.40	$72.64	$66.74	$72.60		$74.49
	Andesite	t					**310,398**						**99,234**	**356,210**	**765,843**
	Ag	g/t					249						182	187	211
	Zn	%					3.12						3.19	3.26	3.19
	Pb	%					1.19						1.57	1.69	1.47
	NSR	$/t					$70.21						$62.16	$64.17	$66.36
	C-Horizon	t										**313,011**	**96,712**	**50,310**	**460,034**
	Ag	g/t										195	240	303	216
	Zn	%										2.51	2.98	2.34	2.59
	Pb	%										1.52	1.69	2.00	1.61
	NSR	$/t										$65.07	$78.13	$74.18	$68.81
Sediment															
	Ag	% Rec to Pb	81.4	81.9	81.4	80.5	81.9	81	80.8	80.8	80.5	79.3	79.9	80.3	80.7
		% Rec to Zn	10	10	10	10	10	10	10	10	10	10	10	10	10
		Grade in Pb- Ag,g/t	8,005	8,186	8,005	7,682	8,166	7,863	7,783	7,803	7,702	7,258	7,480	7,601	7,742
		Grade in Zn, Ag, g/t	450	450	450	450	450	450	450	450	450	450	450	450	450
	Zn	% Rec , Zn	95.6	95.6	95.6	95.6	95.6	95.6	95.6	95.6	95.6	95.6	95.6	95.6	95.6
		Grade, % Zn	47.7	47.7	47.7	47.7	47.7	47.7	47.7	47.7	47.7	47.7	47.7	47.7	47.7
	Pb	% Rec , Pb	95.38	95.69	95.38	98.92	93.22	94.45	94.3	92.45	93.68	96.76	95.69	95.07	94.76
		Grade, % Pb	65.2	65.2	65.2	65.2	65.2	65.2	65.2	65.2	65.2	65.2	65.2	65.2	65.2
Basal Conglomerate															
	Ag	% Rec to Pb	76.3	75	75.1	75.1	75.4	75.4	75.3	75.2	75.5	75.8	74.6		75.3
		% Rec to Zn	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4		12.4
		Grade in Pb- Ag,g/t	10,955	7,931	8,031	8,228	8,922	8,922	8,698	8,466	9,097	9,951	6,666		8,698
		Grade in Zn, Ag, g/t	303	303	303	303	303	303	303	303	303	303	303		303
	Zn	% Rec , Zn	88.21	92.82	92.3	92.72	89.69	89.69	91.1	92.47	89.69	85.43	95.39		91.28
		Grade, % Zn	48.9	50.6	50.4	50.6	49.5	49.5	50	50.5	49.5	47.9	51.5		50

table continues…

Mining Method	Rock Type	Units	Production Year												Total (tonnes)
			1	2	3	4	5	6	7	8	9	10	11	12	
	Pb	% Rec , Pb	85	84.7	84.7	84.7	84.8	84.8	84.8	84.8	84.5	84.1	82.1		84.7
		Grade, % Pb	55.7	55.3	55.3	55.3	55.5	55.5	55.5	55.5	55.1	55.1	52.6		55.4
Andesite															
	Ag	% Rec to Pb					45.2						45.2	45.2	45.2
		% Rec to Zn					18.7						18.7	18.7	18.7
		Grade in Pb- Ag,g/t					5,394						5,394	5,394	5,394
		Grade in Zn, Ag, g/t					731						731	731	731
	Zn	% Rec , Zn					81						81	81	81
		Grade, % Zn					43.3						43.3	43.3	43.3
	Pb	% Rec , Pb					69.6						69.6	69.6	69.6
		Grade, % Pb					48.6						48.6	48.6	48.6
C-Horizon															
	Ag	% Rec to Pb										88.8	88.8	88.8	88.8
		% Rec to Zn										5.4	5.4	5.4	5.4
		Grade in Pb- Ag,g/t										10,625	10,625	10,625	10,625
		Grade in Zn, Ag, g/t										316	316	316	316
	Zn	% Rec , Zn										95.8	95.8	95.8	95.8
		Grade, % Zn										51.1	51.1	51.1	51.1
	Pb	% Rec , Pb										92.7	92.7	92.7	92.7
		Grade, % Pb										69.5	69.5	69.5	69.5
Weighted Average															
	Ag	% Rec to Pb	79.2	77.1	75.6	76.4	71.5	79.2	78.5	79.5	78.8	80.9	77.6	71.7	77.2
		% Rec to Zn	11.0	11.7	12.2	11.8	12.8	10.8	11.0	10.6	10.8	9.3	10.5	12.0	11.2
		Grade in Pb- Ag,g/t	9,279	8,010	8,029	8,096	7,816.0	8,211	8,162	7,959	8,164	8,408	7,480	7,148	8,087
		Grade in Zn, Ag, g/t	387	348	316	339	460.9	402	389	415	401	398	448	517	402
	Zn	% Rec , Zn	92.4	93.7	92.6	93.4	90.3	93.7	93.7	94.9	93.6	94.1	94.6	91.9	93.2
		Grade, % Zn	48.2	49.7	50.2	49.9	47.4	48.3	48.7	48.4	48.3	48.5	47.9	46.7	48.5
	Pb	% Rec , Pb	90.9	88.1	85.6	88.1	85.0	91.3	90.4	90.6	90.6	94.0	92.5	88.5	89.6
		Grade, % Pb	61.1	58.4	56.1	57.7	58.1	62.0	61.2	62.9	61.9	64.7	63.2	61.1	60.7

Table 16.18 Test Results Summary for Locked Cycle Tests used in the April 2008 Mine Plan Schedule Calculations

	Mass %		Pb Concentrate Grade				Pb Concentrate Recovery				Zn Concentrate Grade				Zn Concentrate Recovery			
Composite/Test #	Pb Conc.	Zn Conc.	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Ag (%)
Andesite/LCT49	0.64	1.96	0.52	48.6	13.1	5,394	17.9	69.6	8.0	45.2	0.35	2.1	43.3	731	36.9	9.3	81.0	18.7
C-Horizon/LCT33	1.27	2.51	1.83	68.8	2.3	10,651	54.0	93.0	2.2	89.2	0.63	0.6	50.9	291	36.6	1.5	96.1	4.8
C-Horizon/LCT39	1.19	2.48	1.73	70.1	1.8	10,598	49.3	92.4	1.6	88.4	0.65	0.8	51.2	340	38.5	2.0	95.5	5.9
BC/LCT21	1.35	6.79	4.64	54.4	7.6	10,949	55.7	85.9	2.8	83.1	0.57	0.6	49.4	269	34.8	4.5	92.1	10.3
BC/LCT27	1.20	6.70	4.24	56.1	6.9	11,009	47.5	87.7	2.2	78.0	0.70	0.6	49.2	378	43.6	5.4	87.1	14.8
BC/LCT35	2.96	8.30	3.86	63.2	4.3	18,801	62.3	93.9	2.8	83.5	0.67	0.8	51.9	1,071	30.4	3.4	95.5	13.4
BC/LCT37	1.78	4.86	6.94	45.6	6.4	4,972	67.6	76.2	3.9	72.1	1.00	0.5	49.5	246	26.7	2.3	83.7	9.8
BC/LCT41	2.79	8.44	3.47	65.0	3.9	18,152	56.8	90.8	2.4	82.8	0.73	0.8	51.5	1,036	35.8	3.2	95.8	14.3
BC/LCT47	1.50	5.45	5.35	59.7	3.0	6,055	51.8	85.9	1.6	68.2	1.19	0.7	49.5	303	41.6	3.5	95.6	12.4
Sediments/LCT22	1.82	2.54	2.82	48.8	3.2	7,524	39.7	88.1	3.9	80.0	2.23	1.0	48.6	520	46.7	2.8	91.6	8.7
Sediments/LCT25	1.80	2.50	2.95	49.0	2.5	7,241	39.9	90.3	2.7	77.7	2.20	0.9	48.7	534	45.3	2.5	92.1	9.2
Sediments/LCT28	2.10	2.50	2.50	38.5	3.2	6,370	40.3	89.5	4.0	79.0	2.22	0.8	49.3	478	44.9	2.3	91.4	7.8
Sediments/LCT36	4.13	10.49	1.43	73.3	4.4	3,995	51.4	93.6	3.2	84.0	0.43	1.1	51.4	150	38.9	3.4	95.0	8.0
Sediments/LCT42	0.90	2.13	3.70	67.2	2.8	5,898	42.9	87.0	2.2	74.3	1.78	1.5	48.3	406	49.7	4.5	92.0	12.1
Sediments/LCT50	0.90	2.16	2.87	65.2	2.4	5,563	37.4	87.7	2.0	74.9	1.56	1.2	47.7	367	48.7	3.9	95.6	11.8
Sediments/LCT51	1.54	2.78	4.13	66.3	1.4	9,046	33.3	95.2	1.5	69.4	4.02	0.4	47.1	1,666	58.4	1.1	89.3	23.1

ANDESITE

Only one relevant test was conducted using Andesite ore sample material. This was an optimized locked cycle test and the results are considered to be reliable. The results from this test have been used in the development of the metal recoveries of the overall mine plan.

The results from this test have been provided in Table 16.18, Test LCT49.

C-HORIZON

Only two tests were found to be relevant to developing the recoveries and grades of lead, zinc, and silver from this ore type. However, a plot linking the results of these two tests gives unrealistic lead recoveries; therefore, this relationship cannot be used. The most reliable data was obtained by averaging the results of the two tests. The test results have been reported in Table 16.18, C-Zone, Composite D, Tests LCT33 and 39.

BASAL CONGLOMERATE

Sufficient test data was available to enable relationships to be established linking the feed grade of the samples with the metal recoveries in the concentrates and the respective concentrate grades. The results of six locked cycle tests were used to develop equations for lead and silver in lead concentrate.

The zinc results had to be reviewed in a different manner. In the case of zinc, the results were grouped into pairs of tests with similar grades, and the results averaged to give three points that provided realistic recovery values from the relationship.

The test numbers used to develop the recovery and grade values relationships were as follows: Tests 21 and 27, Tests 35 and 41, and Tests 37 and 47 (all which have been detailed in Table 16.18).

For zinc, the results from Tests 21 and 27 were grouped and averaged, as well as Tests 35 and 41, and Tests 37 and 47. The three data points were used to obtain the required relationships.

For the silver recovery and grade in the zinc concentrate, the average of all six tests was used.

The relationships developed, including the correlation coefficients obtained, were as follows:

- Lead: $y = 6.6095x + 78.24$, where x = lead feed grade and y = lead recovery; $R^2 = 0.38$

- Lead: y = -0.012x2 = 3.112x -122.1, where x = lead recovery and y = lead concentrate grade; R^2 = 0.86
- Silver in lead concentrate: y = 0.0173x + 72.518, where x = silver feed grade and y = silver recovery in the lead concentrate; R^2 = 0.46
- Silver in lead concentrate: y = 7376.7ln(x) – 28832, where x = silver recovery and y = silver concentrate grade in the lead concentrate; R^2 = 0.94
- Zinc: y = 3.5202x + 78.638, where x = zinc feed grade and y = zinc recovery; R^2 = 0.72
- Zinc: y = 1.2884x + 45.41, where x = zinc recovery and y = zinc concentrate grade; R^2 = 0.66
- Silver in zinc concentrate: recovery = 12.5% (average value)
- Silver in zinc concentrate: grade = 551 g/t Ag (average value).

SEDIMENTS

Although a number of tests had been conducted using Sediments samples, the data was not sufficiently reliable to enable all the relationships to be established linking the feed grade of the samples with the metal recoveries in the concentrates, and the respective concentrate grades. The results of seven locked cycle tests were used to develop only three equations, namely for the lead recovery and for silver in lead concentrate. The zinc results also had to be reviewed in a different manner. In the case of zinc, the results from Test 50 were also used for the estimation of grade and recovery values. The silver values were taken as the average of six locked cycle tests.

The test numbers used to develop the recovery and grade values relationships, or average values, were as follows: Tests 22, 25 and 28, Tests 36 and 51, and Tests 42 and 50 (all presented in Table 16.18).

For zinc, the results from the tests were found to be relatively insensitive to feed grade and concentrate grade, and the correlations obtained were extremely poor. The results from Test 50 were used to provide the required grade and recovery values.

For the silver recovery and grade in the zinc concentrate, the averages of six of the seven locked cycle tests were used.

The relationships developed, including the correlation coefficients obtained, were as follows:

- Lead: y = 15.387x + 76.759, where x = lead feed grade and y = lead recovery; R^2 = 0.66
- Lead: the relationship between lead recovery and lead concentrate grade was very poor: used 65.2% Pb concentrate grade from Test 50

- Silver in lead concentrate: y = 0.0572x + 70.812, where x = silver feed grade and y = silver recovery in the lead concentrate; R^2 = 0.84
- Silver in lead concentrate: y = 20.18x + 4271.2, where x = silver recovery and y = silver concentrate grade in the lead concentrate; R^2 = 0.92
- Zinc: recovery = 95.6% (results from Test 50)
- Zinc: grade = 47.7% Zn (results from Test 50)
- Silver in zinc concentrate: recovery = 10.0% (average value)
- Silver in zinc concentrate: grade = 450 g/t Ag (average value).

DISCUSSION

An analysis of the relevant locked cycle test data has enabled metal grades and recoveries to be calculated in accordance with the ore feed grades anticipated over the life of the mine. Although some of the relationships developed from the data has given relatively low correlation coefficients, it is considered to be the best available estimate in the absence of additional information. In some cases, the results from one particular test only have been used as the sole source of relatively reliable information. The data obtained as described above, and presented in Table 16.18 , has been used in the financial analysis.

16.1.12 SETTLING AND FILTRATION TESTWORK

INTRODUCTION

Wardrop used the following testwork components from the 2008 Pocock report:

- flocculant screening
- gravity sedimentation
- pulp rheology
- pressure and vacuum filtration tests.

These tests were conducted on flotation tailings samples from the Basal Conglomerate and Sediments composite samples which were produced during the pilot plant testing by G&T in 2008. The results are summarized in the following sections.

FLOCCULANT SCREENING

Testwork was performed by Pocock on the Basal Conglomerate and Sediments composite tailings samples. The tests were conducted at 20°C with a feed solids concentration of 10%. The flocculant chosen for best overall performance was a medium to high molecular weight, medium charge density (between 7 and 15%)

anionic polyacrylamide. This type of flocculant allows for many mechanisms of flocculation, such as polymer bridging, charge patch attraction, or specific ion adsorption, due to its high molecular weight and anionic charge. The Basal Conglomerate samples had best settling rates and supernatant clarity with between 15 and 20 g/t of flocculant. The Sediments samples showed their best performance when 15 to 30 g/t of flocculant was added.

GRAVITY SEDIMENTATION

Pocock carried out gravity sedimentation testwork on the Basal Conglomerate and Sediments composite samples. Both static (conventional) and dynamic thickening tests were used. The tests found the optimum range of feed solids and flocculant to be in the range of 15 to 20%, and 15 to 30%, respectively. With this range of feed solids and flocculant, the optimum thickener unit area was found to be between 0.125 and 0.25 m^2/t/d. Table 16.19 shows a summary of the static test results.

Table 16.19 Summary of Static Thickening Test Results – Pocock, 2008

Feed Size	Feed Solids	Sample	Flocculant Dose (g/t)	Flocculant Conc. (g/L)	Rise Rate (m^3/m^2h)	Maximum Test Density/Time (%/min)	Solids Conc. (wt %)	Unit Area (m^2/t/d)
100 µm	15	BC	15	0.1	7.59	63.5 / 52	60	0.137
		Seds			5.89	50.3 / 50	60	0.211
	20	BC	10	0.1	1.23	61.4 / 60	60	0.406
		Seds			0.63	55.2 / 60	60	0.674
	20	BC	15	0.1	4.14	64.0 / 60	60	0.159
		Seds			2.32	61.1 / 60	60	0.327
	20	BC	20	0.1	7.46	64.0 / 60	60	0.125
		Seds			3.92	61.5 / 60	60	0.248
	25	BC	20	0.1	2.85	63.9 / 60	60	0.154
		Seds			0.84	62.1 / 60	60	0.341
200 µm	15	BC	15	0.1	20	69.7 / 60	60	0.062
		Seds			10.97	65.5 / 60	60	0.100
	20	BC	10	0.1	2.43	63.9 / 60	60	0.335
		Seds			0.89	54.2 / 60	60	0.742
	20	BC	15	0.1	9.24	69.1 / 60	60	0.098
		Seds			3.79	65.4 / 60	60	0.151
	20	BC	20	0.1	10.91	67.6 / 60	60	0.063
		Seds			7.5	65.4 / 60	60	0.107
	25	BC	20	0.1	5.09	65.3 / 60	60	0.095
		Seds			6.16	66.0 / 60	60	0.079

An overall summary of recommended thickener design parameters is shown in Table 16.20 based on all thickening test results obtained.

Table 16.20 Summary of Thickener Design Parameters

Sample Material	Feed Size P_{80} (µm)	Feed % Solids	Flocculant Type	Minimum Flocculant Dosage (g/t)	Underflow % Solids	Unit Area* (m^2/t/d)	Net Feed Loading Rate** (m^3/h)/m^2
BC	100	15-20	Hychem 304	20-25	58-62	0.15-0.25	3.0-4.0
BC	200	15-20	Hychem 304	20-25	60-64	0.125-0.15	3.5-4.5
Sediments	100	15-20	Hychem 304	20-30	54-58	0.25-0.35	2.5-3.5
Sediments	100	10-15	Hychem 304	20-30	54-58	0.25-0.35	4.5-5.5
Sediments	200	15-20	Hychem 304	20-30	58-62	0.15-0.25	3.0-4.0
Composite	100	15-20	Hychem 304	15-20	60-65	0.125-0.15	4.0-5.0
Composite	200	15-20	Hychem 304	15-20	65-69	0.125	4.5-5.5

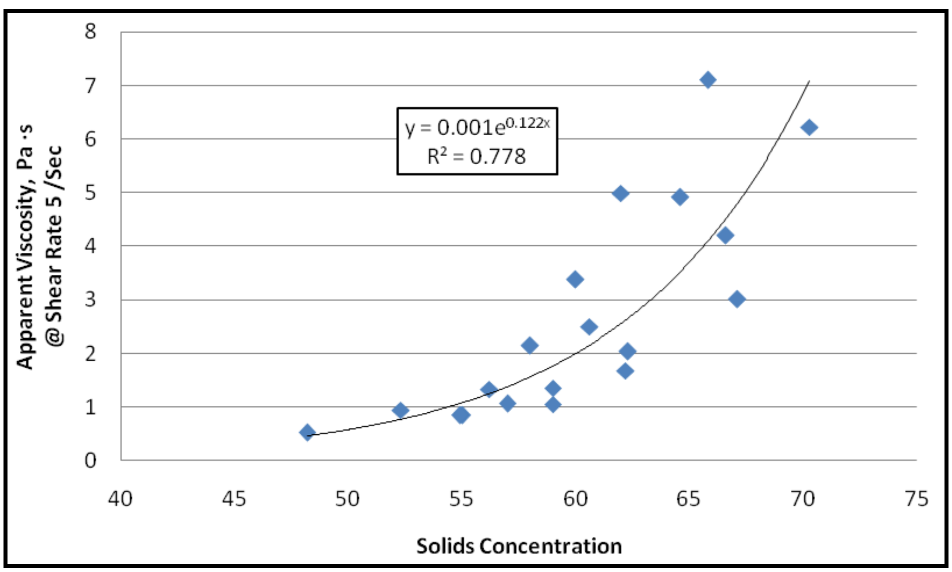
***** recommended minimum Unit Area range to be used for **conventional** type thickener design only, and includes a 1.25 scale-up factor.
****** recommended maximum Net Feed Loading Rate to be used for high-rate type thickener design only.

A high-rate thickener average value for the Basal Conglomerate and Sediments sample is 3.5 (m^3/h)/m^2. This Net Feed Loading Rate value has been used as a design parameter for the tailings thickener. The Composite sample results have been included for report completeness but were not taken into consideration for design purposes. These tests results would not have overly affected the design outcome.

PULP RHEOLOGY

The Pocock pulp rheology tests gave a maximum recommended thickener underflow solids concentration of between 58% and 62%. Figure 16.5 shows the sensitivity of apparent viscosity with increased solids concentration.

Figure 16.5 Apparent Viscosity vs. Solids Concentration – Pocock, 2008



As the solids concentration increases, the apparent viscosity increases exponentially. The slope of the exponential curve reaches a critical point around 62% solids, which corresponds with Pocock's maximum recommended solids concentration.

There was a direct correlation of decreasing viscosity with increasing shear rate. This indicates that the pulp exhibits a shear thinning behaviour, putting the pulp into the Pseudo-plastic class of non-Newtonian fluids. Therefore a specific velocity gradient is needed to initiate and maintain flow. The report gave tested yield values of the pulp with various percent solids. Within the recommended solids concentration given, the yield stress range is between 6 and 15 Pascal•seconds (Pa•s) for 200 µm samples, and 21 to 38 Pa•s for 100 µm. Pocock noted that specialized equipment and design engineering would be required for yield stresses above 30 Pa•s. This highlights the previous recommendation of not exceeding 60% solids in the thickened tailings.

PRESSURE FILTRATION

Pocock performed pressure and vacuum filtration tests on the Basal Conglomerate and Sediments composite tailings samples. The testwork results indicated that the minimum possible pressure filter cake moisture content was between 10.8 and 15.7%. Normal design pressure filter cakes were between 12.0 and 16.8%.

The Pocock testwork found the vacuum filter cake maximum moisture content to be from 17.4 to 27.0% with no flocculant addition, or between 24.4 and 31.7% with flocculant addition. Of note, the filter cakes with added flocculant were deemed stackable, despite the higher moisture content. The vacuum filter cakes with no flocculant addition were not considered to be stackable.

SETTLING AND FILTRATION DISCUSSION

The flocculant screening tests gave the recommended flocculant of Hychem 304, or any other flocculant with similar properties. These key properties include a medium-to-high molecular weight, a medium charge density of 7 to 15%, and an anionic polyacrylamide classification. Wardrop chose Magnafloc 10 in the PDC since Magnafloc 10 has similar properties to Hychem 304.

The thickener and pulp rheology testwork gave a maximum recommended thickener underflow solids concentration between 58 and 62%. Wardrop used a solids concentration of 55% in the PDC. The thickener testwork also gave a recommended thickener unit area between 0.125 and 0.350 m^2/t/d. For design purposes, Wardrop used the average of the Basal Conglomerate and Sediments results for a high rate thickener at the P_{80} of 200 µm grind size, namely 3.5 (m^3/h)/m^2.

Based on the information from the pulp rheology testwork given about the pseudo-plastic nature of the pulp, Wardrop decided to use the larger primary grind size of 200 µm in the PDC, previously set at 125 µm. This will largely nullify the need for specialized equipment and design engineering to pump the thickener underflow should the density exceed 60% solids.

The vacuum filter tests found the maximum possible filter cake moisture content to vary between 17.4 and 27.0% when no flocculant was added. Wardrop selected to use 22% in the PDC based on these results.

16.1.13 METALLURGICAL TESTING CONCLUSIONS

The Pitarrilla process plant design is based on the testwork reviewed in this report. This is based primarily on the results obtained from the testing of the Basal Conglomerate and Sediments ore samples, since this material will constitute the greatest proportion of feed to the plant, and was therefore tested more extensively than the other ore types.

The results showed that saleable flotation concentrate products can be produced from conventional comminution and flotation processes. The plant feed will initially be crushed and milled, then subjected to flotation to produce two separate high grade lead and zinc concentrates. In the event that higher copper grade ores are treated at a future date, a copper flotation circuit should be installed. This circuit will run off the lead flotation circuit, producing a third product (namely a copper flotation concentrate).

The PDC for settling and filtration was based on the slurry properties found during testwork by Pocock, principally the thickener underflow solids concentration and the filter cake percent moisture.

16.2 MINERAL PROCESSING

16.2.1 INTRODUCTION

The Pitarrilla concentrator is designed to process a nominal 4,000 t/d of predominantly lead/zinc ore. The concentrator will produce a marketable concentrate of lead and zinc. The lead concentrate will contain an associated valuable metal content of gold and silver.

16.2.2 SUMMARY

The unit processes selected were based on the results of metallurgical testing performed at G&T, based on resources set out by Silver Standard. The metallurgical processing procedures have been designed to produce saleable high grade lead and zinc concentrates. The possibility of processing copper-rich ores to produce a high grade copper concentrate has not been fully investigated but should be metallurgically feasible. The design of a copper flotation circuit has not been incorporated into the design at this time.

As designed, the treatment plant will contain an underground crushing stage and surface comminution processes, followed by a two-step flotation process to upgrade lead and zinc to saleable concentrate grades. As shown in the simplified flowsheet (Figure 16.6), each flotation concentrate will be thickened and filtered. Each concentrate will be sent to its corresponding stockpile for subsequent shipping to smelters.

The final flotation tailings will alternatively be treated for use as paste backfill in the underground workings, and deposited as thickened slurry in a tailings impoundment facility.

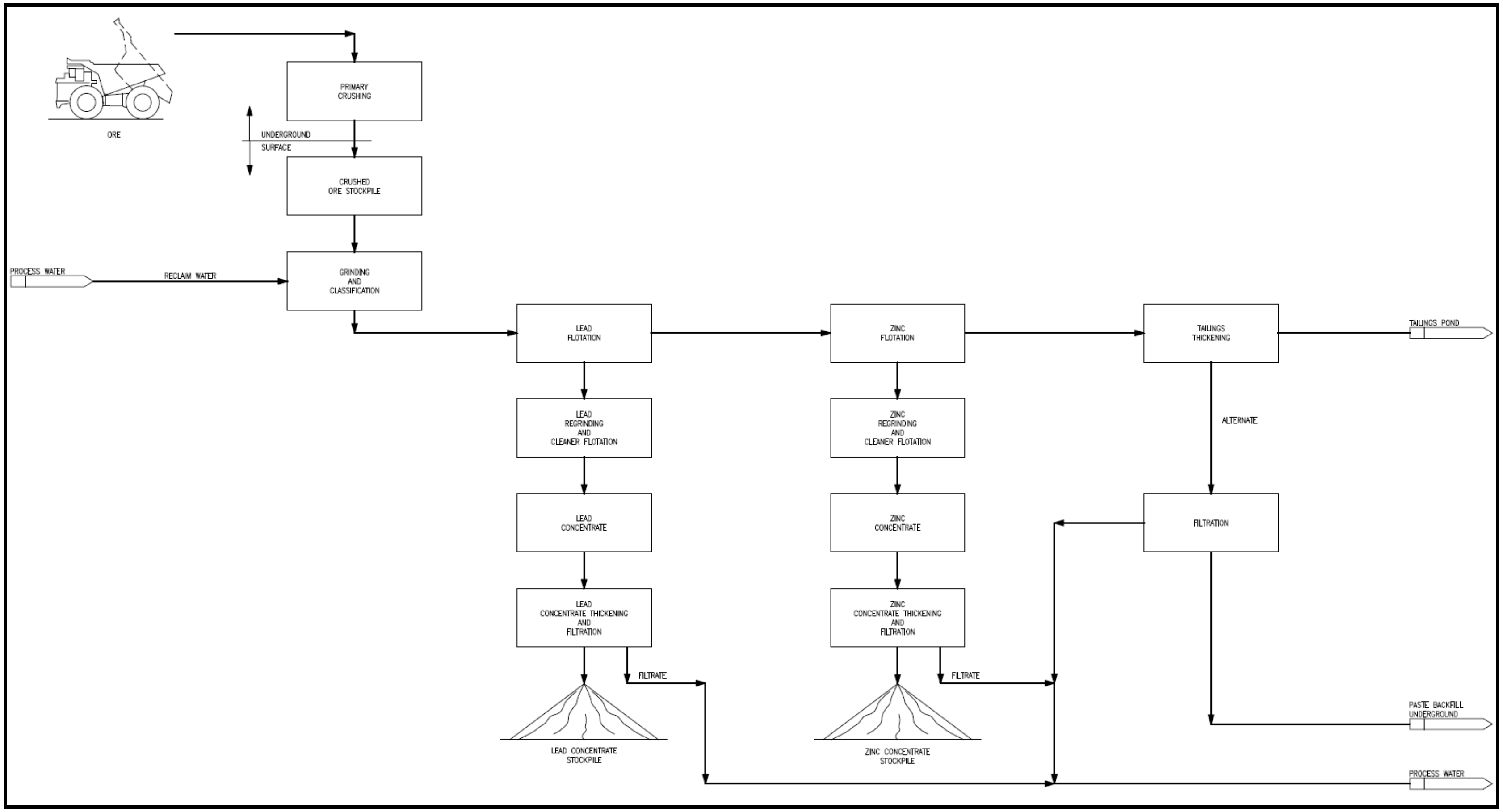
Process water will be recycled from the concentrate and tailings thickener overflows as well as from the tailings impoundment facility. Fresh water will be used for gland service, reagent preparation, and as required for process water make-up.

The process plant will consist of the following unit operations and facilities:

- run-of-mine (ROM) ore feed pass and feed conveyors
- primary crushing – underground operation
- crushed ore stockpile – above ground
- ore reclaim
- a SAG and ball mill grinding circuit incorporating cyclones for classification
- lead rougher flotation, regrinding, and cleaner flotation stages
- lead concentrate thickening, filtration, and dispatch
- zinc rougher flotation, regrinding, and cleaner flotation stages
- zinc concentrate thickening, filtration, and dispatch
- tailings thickening and discharge to the tailings pond
- water return from the tailings pond facility
- intermittent tailings filtration for paste backfill
- tailings stockpile
- tailings conveyor to underground paste plant
- paste distribution.

The simplified flowsheet is shown in Figure 16.6. The detailed process flowsheets are available in “Pitarrilla Pre-feasibility Study Volume 2 – Appendices” as Drawings A0-09-02 to A0-09-13.

Figure 16.6 Simplified Flowsheet



16.2.3 MAJOR PROCESS DESIGN CRITERIA

The concentrator was designed to process 4,000 t/d, equivalent to 1,400,000 t/a. The major criteria used in the design are outlined in Table 16.21. The complete design criteria are available in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices".

Table 16.21 Major Process Design Criteria

Criteria	Unit	
Operating Year	d	350
Crushing Availability	%	65
Grinding and Flotation Availability	%	95
Primary Crushing Rate	t/h	260
Milling & Flotation Process Rate	t/h	175
SAG Mill Feed Size, 80% Passing	µm	150,000
SAG Mill Transfer Size, 80% Passing	µm	900
Ball Mill Circulating Load	%	300
Ball Mill Grind Size, 80% Passing	µm	200
Bond Ball Mill Work Index	kWh/t	17.7
Bond Abrasion Index	g	0.10
Concentrate Regrind Size, 80% Passing		
Lead Concentrate	µm	20
Zinc Concentrate	µm	35

The design parameters are based on testwork results obtained by G&T, particularly from the tests performed in 2008 using the results from the Basal Conglomerate and Sediments samples (G&T, 2008-II). The results from the Pocock report have been used in the thickener and filtration design.

The crushers were sized based on the Bruno (Metso) simulation. The grinding mills were sized based on the Bond Work Index data for SAG and ball mills. The regrind mills were sized using the conventional Bond Work Index equation for ball mills, and using the standard tower mill to ball mill efficiency factor.

The flotation cells were sized based on the optimum flotation times as determined during the laboratory testwork. Typical scale-up factors have been applied.

16.2.4 PLANT DESIGN

OPERATING SCHEDULE AND AVAILABILITY

The primary crushing will be designed on a basis of two 8-hour shifts per day, for 350 d/a. Ore will be stored underground in an ore-pass during the off-crushing hours. The surface plant will be designed to operate on the basis of three 8-hour shifts per day, for 350 d/a.

The jaw crusher overall availability will be 65% and the grinding and flotation circuit availability will have a running time of 95%. This will allow for a potential increase in crushing rate and will allow sufficient downtime for the scheduled and unscheduled maintenance of the crushing and process plant equipment.

16.2.5 *Process Plant Description*

Primary Crushing

A conventional jaw crusher facility will be designed to crush ROM ore underground to reduce the size of the rocks in preparation for the grinding process on surface at an average rate of 257 t/h.

The major equipment and facilities in this area includes:

- stationary grizzly
- an ore pass and Ross Feeder
- vibrating grizzly
- jaw crusher – 1,000 mm x 850 mm
- apron feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scale
- dust collection-fogging system.

The ROM ore will be crushed underground in the primary jaw crusher that will be fed from an ore pass driven by a Ross Feeder system. The Ross Feeder will be equipped with a stationary grizzly with 600 mm x 600 mm square apertures to prevent oversize rocks from entering the circuit.

The ore will feed a vibrating grizzly for sizing and minus 150 mm material will bypass the jaw crusher and report directly to the crushed ore stockpile via the stockpile feed conveyor system. Oversize ore will be reduced to 80% minus 150 mm in the jaw crusher and will be discharged onto the stockpile feed conveyor via the jaw crusher discharge conveyor. The crushed ore stockpile will be situated on the surface.

The total capacity of the crushed ore stockpile will be approximately 4,000 live tonnes. From the stockpile, apron feeders will discharge onto a 600 mm wide conveyor which will transfer the ore to the SAG mill.

The crusher facility will be equipped with a dust collection system to control fugitive dust that will be generated during the crushing and conveying operations.

CRUSHED ORE STOCKPILE AND RECLAIM

The crushed ore stockpile will have a live capacity of 4,000 t. The ore will be reclaimed from this stockpile by apron feeders at a nominal rate of 175 t/h. The apron feeders will feed a 600 mm wide conveyor which in turn feeds the SAG mill. The conveyor belt will be equipped with a belt scale.

The crushed ore stockpile will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the ore.

GRINDING AND CLASSIFICATION

The grinding circuit will consist of a SAG-ball mill combination circuit. It will be a two-stage operation with the SAG mill in open circuit and the ball mill in closed circuit with the classifying cyclones. The grinding will be conducted as a wet process at a nominal rate of 175 t/h of material. The grinding circuit will include the following equipment:

- conveyor feed belt
- conveyor belt weigh scale
- SAG mill – 6.40 m diameter x 2.90 m long (21 ft x 9.5 ft)
- ball mill – 3.96 m diameter x 5.49 m long (13 ft x 18 ft)
- mill discharge pumpbox
- cyclone feed slurry pumps
- cyclone cluster – 4 x 500 mm
- mass flow meter
- sampler system.

The ore on the crushed ore stockpile will be reclaimed under controlled feed rate conditions using apron feeders. These feeders will discharge the material onto a conveyor belt feeding the SAG mill. A belt scale will control the feed to the SAG mill. Water will be added to the SAG mill feed material to assist the grinding process. The SAG mill will operate at a critical speed of 75%.

The SAG mill discharge end will have a trommel screen with 65 mm apertures to remove the tramp oversize material and will discharge the slurry into the mill discharge pumpbox.

The discharge from the ball mill will also be discharged into the common mill discharge pumpbox. The slurry in the mill discharge pumpbox will be pumped to a cyclone cluster for classification. The cut size for the cyclones will be at a particle size of P_{80} of 200 µm, and the circulating load will be 300%. The cyclone underflow will be returned to the ball mill as feed material.

Three cyclones will be operating and one will be a standby unit. The milling rate will be 175 t/h and this will constitute the feed rate to the lead flotation circuit. The ball mill will operate at a critical speed of 75%. Dilution water will be added to the grinding circuit as required.

The cyclone overlow will gravity feed into the lead flotation conditioning tank ahead of the flotation process. The pulp density of the slurry will be approximately 33% solids.

Provision will be made for the addition of lime to the SAG mill for the adjustment of the pH of the slurry in the grinding circuit prior to the flotation process.

Grinding media will be added to the mills in order to maintain the grinding efficiency. Steel balls will be added periodically, to each mill, using a ball charging kibble.

FLOTATION CIRCUIT

The milled ore will be subjected to two stages of sequential flotation to respectively recover the lead and the zinc minerals into high-grade metal concentrates. Conventional mechanical and column flotation cells will be utilized.

Lead Flotation Circuit

The lead flotation circuit will include the following equipment:

- conditioning tank – 3.5 m diameter x 4.5 m
- flotation reagent addition facilities
- conventional mechanical rougher flotation cells – 6 x 15 m^3
- vertical regrind mill – 9,780 mm (H) x 2,670 mm (L) x 2,310 mm (W)
- classification cyclone cluster
- conventional mechanical 1st cleaner flotation cells – 3 x 1.5 m^3
- conventional mechanical 2nd cleaner flotation cells – 3 x 1.5 m^3
- column flotation 3rd cleaner cell – 1 m diameter x 4 m
- pumpboxes
- slurry and concentrate pumps
- particle-size monitor
- sampling system.

The initial flotation circuit will be a lead flotation circuit. The overflow from the classification cyclones in the grinding circuit will be the feed to the lead rougher flotation circuit. The slurry will be conditioned in the lead conditioning tank at the design feed rate of 175 t/h. Lime, as well as flotation reagents, will be added to the conditioning tank as defined through testing. The flotation reagents added will be A3418 as the collector (a dithiophosphinate), and possibly sodium metabisulphite (SMBS) for activation; methyl-isobutyl carbinol (MIBC) will be added as the frother reagent. Sodium cyanide will be added to the lead circuit as a depressant reagent for the zinc minerals. Provision will be made for the staged addition of the reagents in all the flotation stages.

The conditioned slurry will overflow into the lead rougher flotation bank of cells. A lead concentrate will be selectively floated and discharged into the regrind circuit cyclone feed pumpbox from where it will be pumped to the regrind classification cyclone. The lead flotation rougher tailings will be discharged to the lead tailings pumpbox and will be the feed to the zinc flotation circuit.

To completely liberate the fine sized grains of the various minerals from each other and from the gangue constituents, a regrind mill will be incorporated in the lead and the subsequent zinc flotation circuits. The lead regrind circuit cyclone will separate the finely ground rougher lead flotation concentrate into a cyclone overflow product according to the design particle size P_{80} of 20 µm. The coarser cyclone underflow will be feed for the lead regrind mill. The regrind mill will discharge the finely milled material into the cyclone feed pumpbox, and together with the rougher flotation concentrate will constitute the feed for classification by the cyclone.

The cyclone overflow from the lead regrind circuit will be discharged to the lead first cleaner flotation stage. The concentrate from the first cleaner circuit will feed the second cleaner flotation stage with the second cleaner concentrate reporting to the third cleaner flotation stage. The concentrate from the third cleaner flotation stage will be the final lead concentrate and will feed directly to the lead concentrate thickener. The tailings from the third cleaner stage will be returned to be combined with the feed to the second cleaner stage. Tailings from the second cleaner flotation stage will be recycled back to the first cleaner flotation stage. Tailings from the first cleaner flotation stage will report to the lead tailings pumpbox. The option to return this material to either the lead conditioning tank or the lead scavenger flotation stage should be included. The lead tailings stream will be sampled automatically and this will constitute the feed material to the zinc flotation circuit.

Conventional mechanical flotation cells will be used for the rougher, the first cleaner, and the second cleaner flotation stages. A column cell will be used for the third cleaner flotation stage.

Provision will be made for the use of lead concentrate thickener overflow water to be re-used in the lead flotation circuit as dilution water providing this does not have a deleterious effect on the flotation of lead minerals.

Zinc Flotation Circuit

The zinc flotation circuit will include the following equipment:

- three conditioning tanks – 3.5 m diameter x 4.1 m
- flotation reagent addition facilities
- conventional mechanical rougher flotation cells – 6 x 15 m^3
- regrind ball mill – 2.7 m diameter x 4.3 m long
- classification cyclone cluster
- four conventional mechanical 1st cleaner flotation cells – 5 m^3
- column flotation 2nd cleaner cell – 1.5 m diameter x 7 m
- column flotation 3rd cleaner cell – 1 m diameter x 6 m
- pumpboxes
- slurry and concentrate pumps
- particle-size monitor
- sampling system.

The zinc flotation circuit will be similar to the lead flotation circuit, but will have different quantities and volumes of flotation cells in the various flotation stages. The conditioning time required for the zinc flotation circuit will be longer and will require three conditioning tanks to facilitate the stage-wise addition of reagents. The product size from the regrind circuit will differ from the lead circuit with a P_{80} size of 35 µm. There will be three stages of cleaning and the third cleaner concentrate will be the final concentrate product. The cleaner tails from each stage will be returned to the preceding stage except for the first cleaner tailings which will be discharged as final tailings depending on operating conditions. The concentrate from the third cleaner stage will be the final concentrate and will be pumped to the zinc concentrate thickener.

Reagents will be added to the zinc conditioning tanks as defined through testing. This consists of copper sulphate activator, sodium isobutyl-xanthate (SIBX) collector and MIBC frother. Lime will be added for pH control.

Conventional mechanical flotation cells will be used for the rougher and the first cleaner flotation stages. Column flotation cells will be used for the second and third flotation stages.

The zinc tailings stream, after the final tailings pumpbox, will be sampled automatically and this will constitute the final tailings leaving the plant.

Water recycling from the zinc concentrate thickener overflow will be re-used in the zinc flotation circuit where applicable.

CONCENTRATE HANDLING

The flotation cleaner concentrate for each product will be thickened and then filtered and stored prior to shipment to the smelters. Each of the concentrate handling circuits will have the following equipment:

- concentrate thickener
- overflow standpipe
- concentrate slurry pump
- process water tank and pumps
- concentrate stock tank

The following items of equipment will be common to both concentrates:

- common concentrate filter press
- reversible conveyor feeding the concentrate storage feed conveyor
- concentrate storage and dispatch facilities
- common dust collection system.

The flotation concentrate from each of the two products will be treated in a similar manner, although the equipment sizes for each concentrate will differ.

The concentrate produced will be pumped from the cleaner flotation stage to its respective concentrate thickener. Flocculant will be added to the thickener feed to aid the settling process. The thickened concentrate will be pumped to the concentrate stock tank using thickener underflow slurry pumps. The underflow density will be 60% solids. The concentrate stock tank will be an agitated tank which will serve as the feed tank for the common concentrate filter.

The concentrate filter will be a filter press unit. The unit will perform dual duty and will be used to filter both concentrates on a campaign basis. Since filtration with a filter press unit will be a batch process, the concentrate stock tank will also act as a surge tank for the filtration operation. The filter press will dewater each concentrate to produce a final concentrate with a moisture content of about 8%. The filtrate will be returned to the respective concentrate thickener. The filter press solids will be discharged to the respective concentrate stockpile. Each of the dewatered concentrates will be stored in a designated storage facility. Each concentrate will periodically be loaded into trucks for dispatch off the property. The filter will be equipped with flush water facilities to ensure no contamination of concentrates occurs during the change-over form one concentrate to the other.

The thickener overflow solution from each concentrate thickener will be collected in the respective process water tank for recycling within the respective flotation circuit. Excess overflow solution will be discharged to the common concentrate thickeners overflow standpipe and pumped to the tailings thickener overflow standpipe for pumping to the process water tank. Alternatively, the overflow solutions may be sent to the tailings thickener feed.

TAILINGS HANDLING

The final flotation tailings will either be treated for use as paste backfill in the underground workings, or alternatively deposited as thickened slurry in a tailings impoundment facility. The tailings handling circuit will have the following equipment:

- tailings thickener – 18 m diameter
- thickener underflow pumps for thickened slurry transportation
- thickener underflow filter feed slurry pumps
- vacuum disc filter – 3.2 m diameter x 14 discs
- vacuum pumps
- filtrate receiver tank and pumps
- tailings conveyor belt.

The flotation tailings from the zinc flotation circuit will be the final plant tailings. The tailings will be used intermittently as paste backfill and, during the periods that the paste backfill will not be required, the tailings will be pumped as thickened slurry to the tailings impoundment area. On average, tailings produced will be used in the preparation of paste backfill 38% of the time. By difference, tailings will report to the tailings storage facility 62% of the time.

The tailings will be discharged to the tailings thickener via a collection box for the thickener. Flocculant will be added to facilitate the settling of the solids to a density of about 55% solids. As required for paste plant operation, the thickened tailings will be directed from the thickener to the tailings vacuum disc filter feed pan. The filter will dewater the tailings slurry to a moisture content of about 22%. The dewatered solids will be conveyed either to the paste backfill preparation plant, or to the surface tailings paste stockpile area. The filtrate from the disc filter will be collected in the filtrate and will be pumped to the tailings thickener. The thickener overflow solution from the tailings thickener will be discharged to the tailings thickener overflow standpipe for discharging to the process water tank.

The tailings will be pumped to the tailings storage facility as thickened slurry when the tailings are not required for paste make-up. The tailing pond supernatant will be recycled to the concentrator for re-use as process water.

Solution from the concentrate thickener overflows will be re-used in their respective flotation circuit but excess solution, together with the overflow from the tailings thickener, will normally be pumped to the process water tank for recycling. In the event that there will be an excess of process solution, or a build-up of flotation reagents, the water will be discharged to the tailings pond via the tailings thickener for later reclamation. This will allow the various reagents to oxidize and/or degrade prior to returning the water to the plant as reclaimed process water.

TAILING DISPOSAL – PASTE BACKFILL PLANT

Dewatered tailings will be conveyed underground where they will be mixed with cement and water to the target mix design prior to distribution into excavated stopes.

The following are the major paste backfill equipment:

- surface to underground tailings conveyor and weigh scale, 1.5 km long
- surface cement storage silo, 115 m^3
- underground cement silo with discharge screw feeder, 22.7 m^3
- underground cement weigh conveyor and screw feeder
- two blowers (surface and underground silos) – 1,064 m^3/h at 110 kPa (626 ft^3/min at 16 psi)
- wet scrubber
- paste mixer – 3,000 L wet
- paste hopper – 12 m^3
- paste pump
- emergency flush system – 27.2 m^3/h at 10,000 MPa (120 gal/min at 1,450 psi)
- portable flushing system (diesel powered) – 2.3 m^3/h at 103 MPa (10.2 gal/min at 15,000 psi)
- tailings stockpile area
- tailings reclamation system suitable for front end loaders
- weightometers and instrumentation.

The tailings will be provided to the underground paste plant at the rate of 167 t/h, which is equivalent to the rate of tailings produced at the mill. The paste plant will operate for 22 h/d for the number of hours required to fill the stope. Stope size will range from a volume of 1,455 m^3 for cut-and-fill mining to 6,750 m^3 for longhole stoping, requiring between approximately 1,600 t and 9,400 t of tailings, respectively. During stoppages at the paste plant, the process will revert to sending thickened tailings to the tailings pond facility.

A 4,000 t stockpile of tailings will be maintained on surface in the event that supplementation of plant tailings may be required. The 115 m^3 surface cement storage acts as a buffer between cement delivered from the manufacturer and secondary underground trucking to the paste plant.

Approximately 38% of the total tailings produced will be returned underground for paste backfill purposes. For the remaining 62% of the time, the tailings will be discharged to the tailings pond from the tailings thickener.

The underground tailings will be mixed with cement and mine water in the required proportions in the paste mixer. The paste will then be pumped to the mined-out stopes. Mass balance and pastefill plant process details are available in “Pitarrilla Pre-feasibility Study Volume 2 – Appendices”.

Reagent Handling and Storage

Various chemical reagents will be added to the process slurry stream to facilitate the individual flotation processes.

The preparation of the various reagents will generally require the following equipment:

- bulk handling system
- mix and holding tanks
- metering pumps
- flocculant preparation facility
- lime slaking and distribution facility
- eye-wash and safety showers
- applicable safety equipment.

Various chemical reagents will be added to the grinding and flotation circuit to modify the mineral particle surfaces and enhance the floatability of the mineral particles into selective concentrate products. Fresh water will be used in the making up or the dilution of the various reagents that will be supplied in powder/solids form, or which require dilution prior to the addition to the slurry. These solutions will be pumped to the addition points of the various flotation circuits and streams using metering pumps.

The solid reagents will generally be made up to a solution of 10% strength in a mix tank, and then transferred to the holding tank, from where the solution will be pumped to the addition point. The reagents that will be prepared in this manner will be the SMBS, copper sulphate, SIPX, and sodium cyanide. The reagents MIBC and Aero 3418 will not be diluted and will be pumped directly from the bulk containers to the point of addition using metering pumps.

Flocculant will be prepared in the standard manner as a dilute solution with 0.30% solution strength. This will be further diluted in the thickener feed well.

Lime will be delivered in bulk by 40-t trucks and will be off-loaded pneumatically into a silo. The lime will then be prepared in a lime slaking system as a 20% concentration slurry. This lime slurry will be pumped to the points of addition using a closed loop system. The valves will be controlled by pH monitors, which will control the amount of lime added.

To ensure spill containment, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest tank. In addition, each reagent will be prepared in its own bunded area in order to limit spillage and facilitate its return to its respective mix tank. The storage tanks will be equipped with low level indicators and instrumentation to ensure normal operation can be maintained. Appropriate ventilation, fire and safety protection and Material Safety Data Sheet stations will be provided at the facility.

Each reagent line and addition point will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards. All operational personnel will receive WHMIS training, along with additional training for the safe handling and use of the reagents.

Assay and Metallurgical Laboratory

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, the concentrator, and the environment departments. The most important of these instruments includes:

- atomic absorption spectrophotometer (AAS)
- inductively-coupled plasma (ICP) mass spectrometer
- x-ray fluorescence spectrometer (XRF)
- Leco furnace.

The metallurgical laboratory will undertake all necessary testwork to monitor metallurgical performance and, more importantly, to improve process flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory-sized crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

Water Supply

Two separate water supply systems for fresh water and process water will be provided to support the operation.

Fresh Water Supply System

Fresh and potable water will be supplied to a fresh/fire water storage tank from mine underground dewatering and wells.

Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems
- gland service for the slurry pumps
- vacuum pump seal water
- reagent make-up
- process make-up water
- potable water supply.

The fresh/fire water tank will be equipped with a standpipe that will ensure the tank is always holding at least 40 m^3 of water, equivalent to a 2-hour supply of fire water.

The potable water from the fresh water source will be treated and stored in the potable water storage tank prior to delivery to various service points.

Process Water Supply System

Some process water generated in the individual flotation circuits as concentrate thickener overflow solution will be re-used in the respective flotation circuit. Excess concentrate thickener overflow water will be discharged to the process water tank for redistribution; alternatively, the concentrate thickener overflow will be directed to the tailings thickener feed. The tailings thickeners overflow solution will be directed to the process water tank via the tailings thickener overflow standpipe. Additional process water will be reclaimed from the tailings pond and will be pumped to the process water tank for distribution to the points of usage.

AIR SUPPLY

Separate air service systems will supply air as follows:

- low-pressure air for flotation cells will be provided by air blowers
- high-pressure air for the filter press and drying of concentrate will be provided by dedicated air compressors
- high-pressure air for the dust suppression (fogging) system and other services will also be provided by a separate air compressor
- instrument air will come from the plant air compressors, and will be dried and stored in a dedicated air receiver
- high-pressure air for cleaning the paste backfill lines will be required via an air receiver form the underground air compressor.

ON-LINE SAMPLE ANALYSIS

An on-line analyzer will analyze each flotation stage for each circuit. A sufficient number of samples will be taken so that each circuit can be balanced by analytical results and calculation as required. The samples that will also be taken for metallurgical accounting purposes will be the flotation feed to each circuit, the final tailings, and the final concentrate samples.

An on-stream particle size monitor will determine the P_{80} particle size of the primary cyclone overflow, and the regrind circuit product.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 P&E 2008 RESOURCE ESTIMATE – BRECCIA RIDGE ZONE

17.1.1 *INTRODUCTION*

The 2008 P&E resource estimate was prepared as an independent NI 43-101 compliant estimate of the resources contained strictly within the Breccia Ridge Zone and does not explicitly or implicitly refer to resources contained in any of the other mineralized zones contained within the Pitarrilla property. This resource estimate was undertaken by Eugene Puritch (P.Eng.) and Antoine Yassa, (P.Geo.) of P&E in Brampton, Ontario. The effective date of this resource estimate is August 3, 2008.

17.1.2 *DATABASE*

All drilling data was provided by Silver Standard in the form of a Microsoft Access database, Excel files, drill logs, and assay certificates. There were 28 drill cross sections developed on a local grid looking northwest on an azimuth of 335° on a 25 m spacing named from 25-S to 700-S. A Gemcom database was provided by Silver Standard containing 151 diamond drillholes and 5 RC drillholes, of which 131 were utilized in the resource calculation. The remaining data were not in the area that was modelled for this resource estimate. A surface drillhole plan is shown in Figure 17.1.

The database was validated in Gemcom with minor corrections required. The Breccia Ridge assay table of the database contained 51,284 assays for Ag, Cu, Pb, and Zn. All data are expressed in metric units and grid coordinates are in the NAD 27 UTM reference system.

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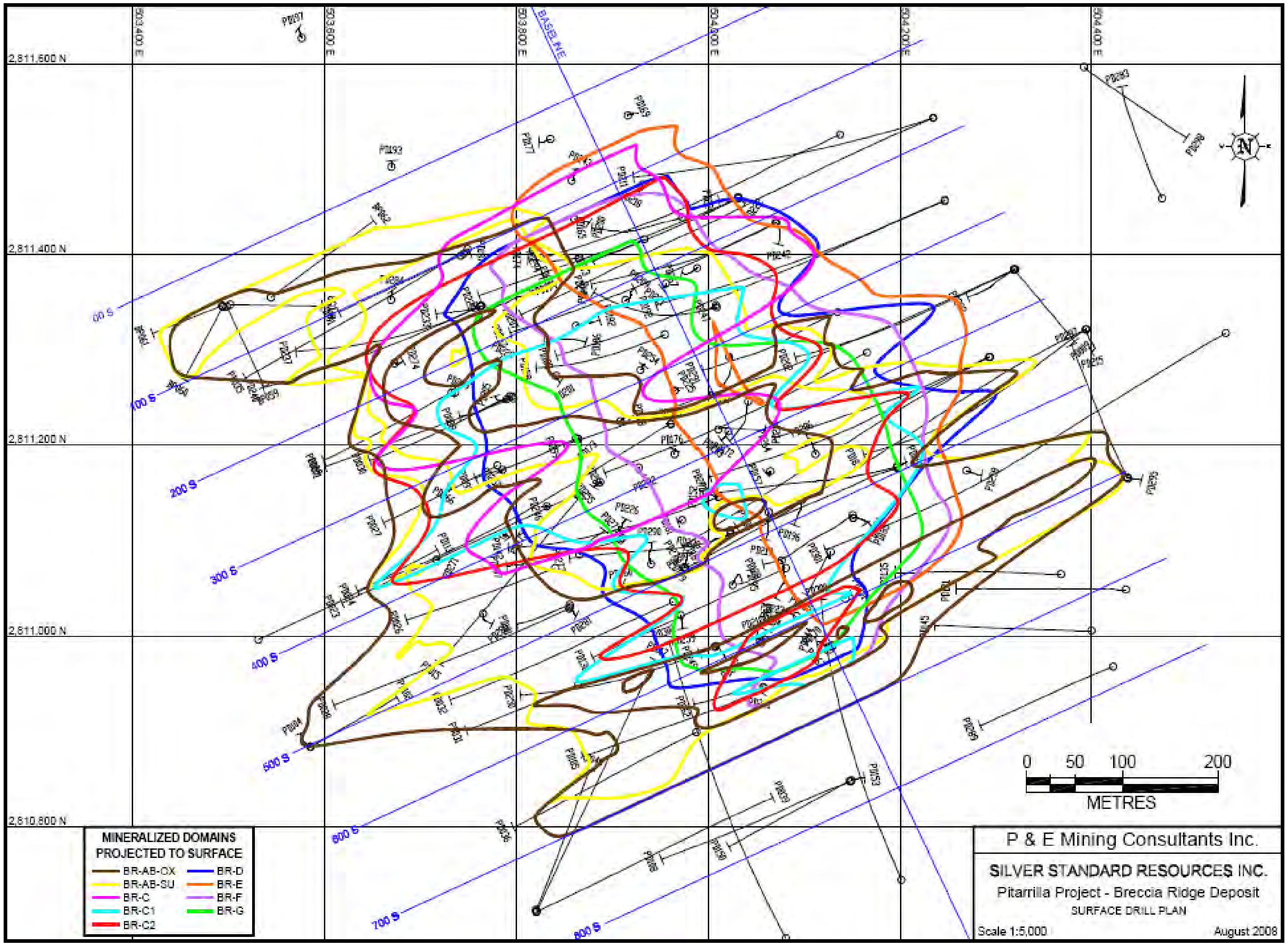
SILVER STANDARD

Figure 17.1 Breccia Ridge Deposit Surface Drillhole Plan



17.1.3 DATA VERIFICATION

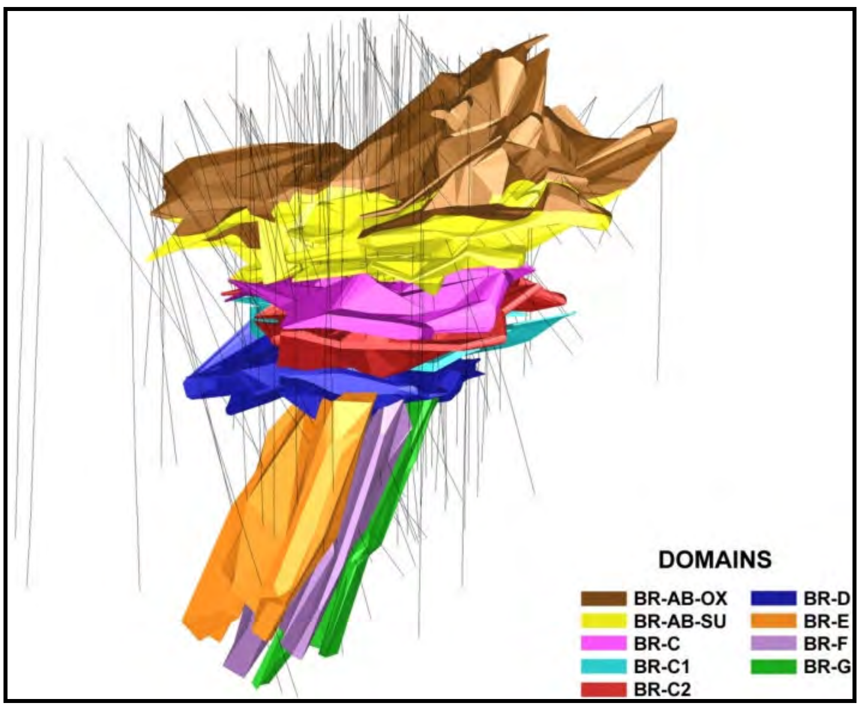
Verification of assay data entry was performed on 17,351 assay intervals for Ag, Cu, Pb, and Zn. A few very minor data entry errors were observed and corrected. The 17,351 verified intervals were checked against digital assay laboratory certificates from ALS Chemex of Vancouver. The checked assays represented 76% of the data to be used for the resource estimate and approximately 34 % of the entire Breccia Ridge database.

17.1.4 DOMAIN INTERPRETATION

Domain boundaries were determined from lithology, structure, and Ag equivalent (AgEq) boundary interpretation from visual inspection of drill hole sections. Nine domains were created named BR-AB-OX, BR-AB-SU, BR-C, BR-C1, BR-C2, BR-D, BR-E, BR-F, and BR-G. These domains were created with computer screen digitizing on drillhole sections in Gemcom.

On each section, polyline interpretations were digitized from drillhole to drillhole but not extended more than 25 m into untested territory. Minimum constrained true width for interpretation was approximately 2.0 m. The interpreted polylines from each section were "wireframed" in Gemcom into a 3D domain. The resulting solids (domains) were used for statistical analysis, grade interpolation, rock coding, and resource reporting purposes (Figure 17.2).

Figure 17.2 Breccia Ridge Deposit 3D Domains



17.1.5 ROCK CODE DETERMINATION

The nine rock codes used for the resource model were derived from the mineralized domain solids.

17.1.6 COMPOSITES

Length weighted composites were generated for the drillhole data that fell within the constraints of the above-mentioned domains. These composites were calculated for Ag, Cu, Pb, and Zn over 1.5 m lengths. The very few un-assayed intervals that were encountered were assigned one-half assay laboratory detection limit values. Any composites calculated that were less than 0.5 m in length were discarded so as to not introduce a short sample bias in the interpolation process. The composite data were transferred to Gemcom extraction files for the grade interpolation as X, Y, Z, Ag, Cu, Pb, and Zn files.

17.1.7 GRADE CAPPING

Grade capping was investigated on the raw assay values within the constraining domains to ensure that the possible influence of erratic high values did not bias the database. Grade capping values are shown in Table 17.1.

Table 17.1 Grade Capping Values

Element	Capping Value	No. of Assays Capped	Cumulative Percent for Capping	Raw COV*	Capped COV*
BR-AB-OX Domain					
Ag	800 g/t	12	99.8	1.62	1.20
Cu	0.15%	62	98.9	2.81	1.53
Pb	6.00%	8	99.9	2.22	1.79
Zn	5.00%	55	99.1	1.99	1.41
BR-AB-SU Domain					
Ag	650 g/t	47	98.6	2.18	1.50
Cu	0.18%	69	97.7	6.77	1.76
Pb	7.50%	4	99.9	2.04	1.77
Zn	10.00%	6	99.8	1.58	1.34
BR-C Domain					
Ag	750 g/t	11	98.5	2.50	1.72
Cu	1.00%	15	97.9	2.52	2.00
Pb	2.50%	4	99.5	1.61	1.28
Zn	4.50%	25	96.6	1.78	1.24
BR-C1 Domain					
Ag	800 g/t	12	98.9	1.87	1.59
Cu	0.60%	11	99.0	2.49	2.11

table continues…

Element	Capping Value	No. of Assays Capped	Cumulative Percent for Capping	Raw COV*	Capped COV*
Pb	7.50%	8	99.4	2.57	1.93
Zn	12.50%	7	99.4	1.84	1.51
BR-C2 Domain					
Ag	800 g/t	27	98.1	3.35	1.79
Cu	0.90%	29	98.0	5.90	1.90
Pb	9.00%	6	99.6	3.10	2.27
Zn	12.00%	18	98.7	1.90	1.73
BR-D Domain					
Ag	1,500 g/t	14	99.5	2.04	1.71
Cu	2.50%	6	99.8	2.85	2.32
Pb	7.00%	18	99.4	2.27	1.50
Zn	20.00%	7	99.8	1.36	1.32
BR-E Domain					
Ag	1,750 g/t	3	99.9	2.39	1.97
Cu	0.50%	23	99.1	5.41	1.67
Pb	9.00%	3	99.9	2.09	1.69
Zn	15.00%	5	99.8	1.53	1.42
BR-F Domain					
Ag	1,600 g/t	11	99.7	2.84	2.00
Cu	1.00%	5	99.9	2.35	2.12
Pb	10.00%	12	99.6	1.92	1.81
Zn	No Cap	0	100.0	1.72	1.72
BR-G Domain					
Ag	1,000 g/t	8	99.6	3.67	1.62
Cu	No Cap	0	100.0	2.23	2.23
Pb	7.00%	7	99.6	2.29	1.62
Zn	No Cap	0	100.0	1.62	1.62

17.1.8 VARIOGRAPHY

Snowden's Supervisor software (Version 7.10) was used to evaluate the spatial continuity of the Ag, Cu, Pb, and Zn mineralization using capped composites within constrained domains.

Standardized traditional semi-variograms were used to model the grade continuity. Nugget effects were estimated from true downhole semi-variograms. Correlations between grade-elements within individual domains were also examined when determining the ranges of the semi-variograms.

Oriented semi-variogram fans were used to determine the major, semi-major, and minor axis of grade continuity. Anisotropic directional semi-variograms were then defined for resource estimation using Gemcom's Azimuth-Dip-Azimuth convention. Where a principle direction of grade continuity could not be determined an omni-directional semi-variogram was modelled.

A total of 36 experimental semi-variograms were developed, incorporating four grade-elements in nine domains. Directional semi-variograms were modelled for the BR-E, BR-F, and BR-G domains, with the remainder of the domains being modelled as isotropic. Modelling of Ag and Cu produced good experimental semi-variograms; Pb produced moderate quality experimental semi-variograms, and Zn produced poor experimental semi-variograms.

17.1.9 *Bulk Density*

The bulk density used for the resource model was derived from 172 measurements of bulk density for select samples sent to ALS Chemex. The bulk density block model was coded with one simple spherical interpolation pass. The resulting average bulk densities within the constraining domains utilizing these samples were calculated to be 2.59 t/m^3 for the open pit resource and 2.73 t/m^3 for the underground resource.

17.1.10 *Block Modelling*

The resource model was divided into a 3D block model framework. The block model has 39,040,000 blocks that were 5 m in the X direction, 5 m in the Y direction, and 5 m in the Z direction. There were 400 columns (X), 400 rows (Y), and 244 levels (Z). The block model was rotated 25.0163 degrees counter-clockwise. Separate block models were created for rock type, density, percent, class, Ag, Cu, Pb, Zn, and AgEq.

The percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside each constraining domain. As a result, the domain boundaries were properly represented by the percent model ability to measure infinitely variable inclusion percentages within a particular domain.

The Ag, Cu, Pb, and Zn composites were extracted from the Microsoft Access database composite table into separate files for each mineralized zone. Inverse distance squared (ID2) grade interpolation was utilized. There were three interpolation passes performed on all domains for each element for the measured, indicated, and inferred classifications.

17.1.11 *Resource Classification*

For the purposes of this resource, classifications of all interpolated grade blocks were determined from the Ag grade interpolations for measured, indicated, and inferred due to Ag being the dominant revenue producing element in the AgEq calculation.

17.1.12 RESOURCE ESTIMATE

The resource estimate was derived from applying an AgEq cut-off grade to the block model and reporting the resulting tonnes and grade for potentially mineable areas. The following calculations demonstrate the rationale supporting the AgEq cut-off grade that determines the potentially economic portion of the mineralized domains.

AGEQ CUT-OFF GRADE CALCULATION COMPONENTS

All currency is shown in US dollars.

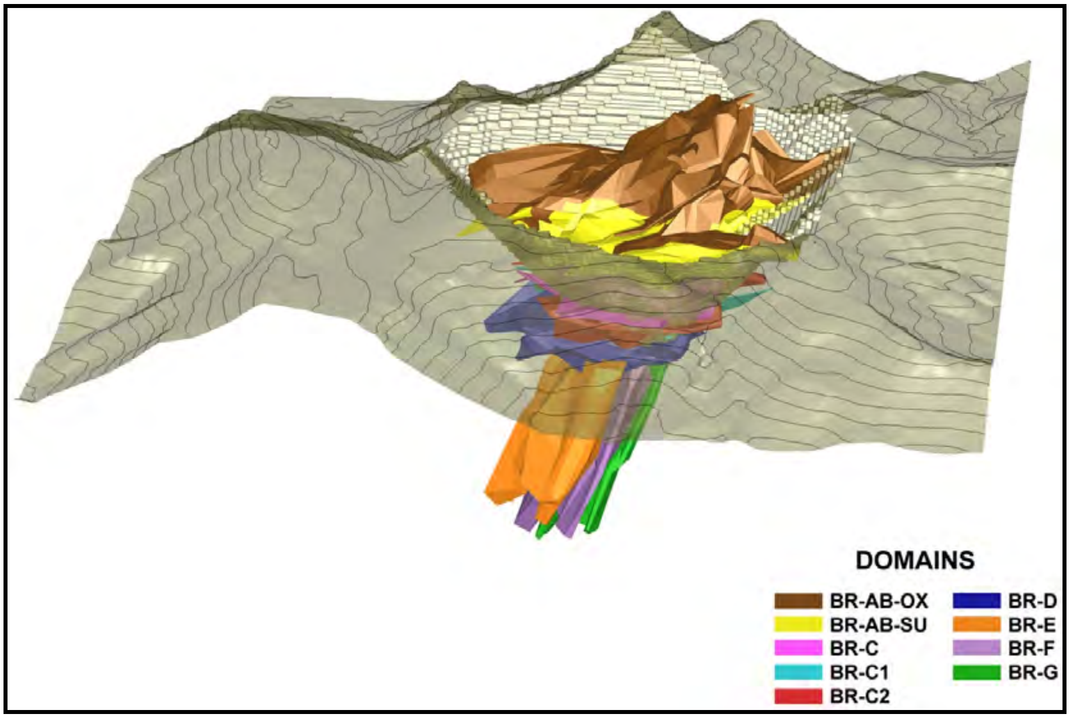
- Ag Price................$11.00/oz
- Cu Price................$2.00/lb
- Pb Price................$0.75/lb
- Zn Price................$1.05/lb
- Ag Payable...........90%
- Cu Payable...........60%
- Pb Payable...........60%
- Zn Payable...........60%.

***AgEq** = (Ag g/t x 90%) + (Cu% x [($2.00 x 22.046)/($11.00/31.1035)] x 60%) + (Pb% x [($0.75 x 22.046)/($11.00/31.1035)] x 60%) + (Zn% x [($1.05 x 22.046)/($11.00/31.1035)] x 60%)*

In the anticipated 10,000 ore t/d underground operation of the Breccia Ridge deposit, the mining, processing, and G&A costs combine for a total of ($13 + $6.00 + $2.00) = $21.00/t milled which combined with an $11.00/oz Ag price and 90% recovery yield an AgEq cut-off $21.00(($11.00/31.1035) x 90%) = 65.98 g/t – **use 65 g/t.**

The Breccia Ridge open pit resource estimate was derived from applying an AgEq cut-off grade to the block model within a Whittle 4X optimized pit shell and reporting the resulting tonnes and grade for potentially open pit mineable portion of the deposit (Figure 17.3).

Figure 17.3 Breccia Ridge Deposit Resource Pit Shell



In the anticipated 40,000 ore t/d open pit operation of the Breccia Ridge deposit, the processing and G&A costs combine for a total of ($5.00 + $1.00) = $6.00/t milled which, combined with an $11.00/oz Ag price and 80% recovery, yields an AgEq cut-off of $6.00 (($11.00/31.1035) x 80%) = 21.21 g/t - **use 20 g/t**.

The resulting underground and open pit resource estimate can be seen in Table 17.2.

Table 17.2 Breccia Ridge Resource Estimate

	Tonnes	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	Ag (million oz)
Underground 65 g/t AgEq Cut-off							
Measured	18,486,000	91.6	0.07	0.70	1.24	155.8	54.5
Indicated	48,469,000	89.2	0.07	0.66	1.68	170.1	139.1
Meas & Ind	**66,955,000**	**89.9**	**0.07**	**0.67**	**1.56**	**166.13**	**193.5**
Inferred	19,265,000	51.3	0.07	0.54	1.12	110.5	31.8
Open Pit 20 g/t AgEq Cut-off							
Indicated	105,630,000	63.6	0.03	0.31	0.68	94.7	215.8
Inferred	5,538,000	72.5	0.06	0.24	0.67	102.8	12.9
Total							
Measured	18,486,000	91.6	0.07	0.70	1.24	155.8	54.5
Indicated	154,099,000	71.6	0.04	0.42	0.99	118.4	354.9
Meas & Ind	**172,585,000**	**73.8**	**0.04**	**0.45**	**1.02**	**122.44**	**409.3**
Inferred	24,803,000	56.0	0.07	0.47	1.02	108.7	44.7

1. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues. There is no guarantee that Silver Standard will be successful in obtaining any or all of the requisite consents, permits, or approvals, regulatory or otherwise for the project or that the project will be placed into production.
2. The quantity and grade of reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and further exploration drilling is required to determine whether they can be upgraded to an Indicated or Measured mineral resource category.

It should be noted that the mineral resources in this estimate were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council December 11, 2005.

In order to investigate the sensitivity of the mineral resource estimate to cut off grade, the block model was reported at several AgEq cut-off grades in all classification categories for the underground and open pit portions of the deposit (Table 17.3 and Table 17.4).

Table 17.3 Breccia Ridge Underground Resource Estimate Sensitivity

AgEq g/t Cut-Off	Tonnes	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)
100	54,560,834	105.5	0.08	0.79	1.84	195.3
95	58,167,956	102.0	0.08	0.77	1.78	189.2
90	62,746,872	97.9	0.08	0.75	1.71	182.2
85	67,126,014	94.3	0.08	0.72	1.66	176.0
80	72,160,514	90.5	0.08	0.70	1.60	169.5
75	76,875,952	87.2	0.08	0.68	1.55	163.8
70	82,020,727	83.8	0.07	0.66	1.50	158.1
65	**86,219,575**	**81.3**	**0.07**	**0.64**	**1.46**	**153.7**
60	90,000,352	79.1	0.07	0.63	1.42	149.9
55	93,653,733	77.1	0.07	0.61	1.39	146.3
50	96,948,463	75.3	0.07	0.60	1.36	143.1
45	100,052,768	73.7	0.07	0.59	1.33	140.1
40	102,628,785	72.4	0.07	0.58	1.31	137.7
35	104,839,021	71.2	0.06	0.57	1.29	135.6
30	106,495,274	70.3	0.06	0.57	1.28	134.0
25	107,678,199	69.7	0.06	0.56	1.27	132.8

Table 17.4 Breccia Ridge Open Pit Resource Estimate Sensitivity

AgEq g/t Cut-Off	Tonnes	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)
75	58,186,460	88.5	0.04	0.45	0.94	132.0
70	64,301,102	84.8	0.04	0.43	0.90	126.3
65	71,060,813	81.1	0.04	0.40	0.86	120.7
60	78,331,291	77.5	0.04	0.38	0.82	115.3
55	85,877,874	74.2	0.03	0.37	0.78	110.2
50	93,107,323	71.3	0.03	0.35	0.75	105.7
45	99,395,321	68.8	0.03	0.34	0.73	102.1
40	103,966,518	67.0	0.03	0.33	0.71	99.5
35	107,134,466	65.8	0.03	0.32	0.70	97.6
30	109,195,170	64.9	0.03	0.32	0.69	96.4
25	110,403,652	64.4	0.03	0.31	0.68	95.7
20	**111,168,054**	**64.0**	**0.03**	**0.31**	**0.68**	**95.2**
15	111,673,748	63.8	0.03	0.31	0.68	94.8
10	111,936,185	63.6	0.03	0.31	0.68	94.6
5	112,098,013	63.5	0.03	0.31	0.68	94.5

17.1.13 Confirmation of Estimate

As a test of the reasonableness of the estimate, the block model was queried at a 0.01 g/t Ag cut-off grade with blocks in all classifications summed and their grades weight averaged. This average is the average grade of all blocks within the mineralized domains. The values of the interpolated grades for the block model were compared to the length weighted capped average grades and average grade of composites of all samples from within the domains. The results are presented in

Table 17.5 Comparison of Weighted Average Grade of Capped Assays and Composites with Total Block Model Average Grade

Category	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Capped Assays	73.5	0.05	0.46	1.06
Composites	70.4	0.04	0.44	1.00
Block Model	65.9	0.04	0.42	0.94

The comparison above shows the average grade of the Ag, Cu, Pb, and Zn blocks in all domains to be similar to the weighted average of all capped assays and composites used for grade estimation. The moderate reduced and smoothing grade effect of the block model interpolation removed some minor data clustering and provides a more reliable estimate.

In addition, a volumetric comparison was performed with the block volume of the model vs. the geometric calculated volume of the domain solids:

- All Domains:
  - block model volume: 83,408,702 m^3
  - geometric domain volume: 83,877,405 m^3
  - difference: 0.56%.

17.2 Mineral Reserve Estimate

Wardrop received the block model that was used for the P&E resource estimate then applied mining and economic parameters to the model in order to form the basis of the reserve estimate.

NI 43-101 defines a mineral reserve as “the economically mineable part of a Measured or Indicated Mineral Resource”. Only measured and indicated resources have been used to establish the probable mineral reserves; no mineral reserves were categorized as proven. Inferred resources were considered to be waste and to have zero grade in this study.

The orebody is polymetallic with the most significant metals being silver, lead, and zinc. Four main rock types, which present differing metallurgical recoveries for each of the significant metals, are being identified as follows:

- Andesite
- Basal Conglomerate
- Sediments
- C-Horizon.

The planned processing operations envisage production of two concentrates, namely, a lead and a zinc concentrate that will be shipped to smelters. Table 17.6 provides metallurgical assumptions for each of the concentrates for each of the rock types.

The oxide and sulphide zones were not considered in the reserve estimation as metallurgical testwork is ongoing for those rock types. Silver Standard considered those resources to be mined by open pit method but the final decision will be made after completion of the metallurgical testwork.

Table 17.6 Metallurgical Assumptions

Rock Type	Pb Recovery (%)	% Pb in Concentrate	Ag Recovery into Pb Concentrate (%)	Ag in Pb Concentrate (ppm)	Zn Recovery (%)	% Zn in Concentrate	Ag Recovery into Zn Concentrate (%)	Ag in Zn Concentrate (ppm)
Sediments	87.7	65.2	74.9	5,563	95.6	47.7	11.8	367
Basal Conglomerate	85.9	59.7	68.2	6,055	96.3	49.8	12.4	303
Andesite	69.6	48.6	45.2	5,394	81	43.3	18.7	731
C-Horizon	70.4	53.5	47.3	5,734	87.1	48.8	26.9	766

Since the deposit is polymetallic, it was decided to estimate the net smelter return (NSR) for each block in the model in order to design the stope outlines and evaluate economic viability.

The NSR value was calculated assuming metal prices of US$11.00/oz for silver, US$0.50/lb for lead, and US$0.70/lb for zinc, metallurgical metal recoveries as provided in Table 17.6, assumed smelting charges of $160 per tonne of lead concentrate and $225 per tonne of zinc concentrate, refining charges of $0.4/oz of silver, and metal deductions, as per assumed treatment terms. Factors for each contributing metal were calculated for each rock type depending on metal recoveries, smelter charges, and metal prices, as follows:

- Sediments:
  - NSR = 0.248 x Ag (g/t) + 8.015 x Pb (%) + 8.264 x Zn (%)
- Basal Conglomerate:
  - NSR = 0.227 x Ag (g/t) + 7.842 x Pb (%) + 8.412 x Zn (%)
- Andesite:
  - NSR = 0.163 x Ag (g/t) + 6.279 x Pb (%) + 7.095 x Zn (%)
- C-Horizon:
  - NSR = 0.181 x Ag (g/t) + 6.389 x Pb (%) + 8.013 x Zn (%)

These factors were input into the block model to calculate the NSR for each block within the model.

In order to determine which resources would be viable for mining, estimates of the onsite operating costs (including mining, milling, and general and administrative) and off-site cost (including concentrate trucking to smelter cost) were developed to determine the break-even cut-off grade.

The onsite operating cost of $33.81/t required for a 4,000 t/d operating mine was estimated from first principles for each cost category such as mining, processing, and general and administrative. The off-site cost of $2.86/t of ore was estimated based on concentrate trucking cost of $40/t of concentrate. The break even cut-off grade was considered to be $36.63/t.

The sensitivity of the mineral resource estimate to cut-off grade and preliminary economic analysis were provided for several cut of grades above break even.

The cut-off grade of $45/t was selected for stope design based on results of preliminary economic analysis of resources at different cut-off grades. Stope outlines, preliminary reserves, and a production schedule were prepared and the average NSR of the reserve was estimated to be $64.25/t.

Economic analysis of this schedule gave poor results so it was decided to raise the stope design NSR to $50/t in order to improve overall reserve grade and therefore project economics. Stopes were redesigned using this cut-off; a reserve with an average NSR of $69.79/t was achieved and a production schedule prepared.

The economic model was run with the $50/t NSR reserve and showed better result than $45/t NSR reserve. Accordingly, the $50/t NSR cut-off model was adopted for the study.

It is unusual for a design cut-off that is significantly higher than the break-even cut-off to be used in designing the stope outlines for the reserve estimate. The reason that the $50/t NSR provides a better economic result is believed to be due to the grade distribution within the orebody. The grade distribution within stopes is not heterogeneous, and fairly significant internal dilution at below the break-even cut-off NSR is included into the reserve. It is not possible to design the stopes to only encompass paying material, so the artificially higher than break-even design cut-off is used to reduce the internal unpaying dilution. Thus, when the $45/t NSR reserve option was checked economically, it gave a less favourable economic result than the $50/t NSR reserve because there was too much internal dilution at a lower NSR included within the stope boundaries.

Table 17.7 shows the effect on the internal dilution by raising the cut-off from $45/t to $50/t. It is interesting to note that, on average, the internal dilution is not making money on the $45/t case as the NSR of $34.02/t is less than the break-even cost of $36.63/t; whereas, for the $50/t case, the average of the internal dilution contributes as $38.21/t, which is higher than the break-even cost of $36.63/t.

Table 17.7 Mineral Resources & Internal Dilution at $45/t and $50/t Cut-off

	Tonnes	**Average NSR ($US/t)**
NSR ≥$45		
Resources within Stopes	18,266,835	74.02
Internal Dilution	2,002,795	34.02
NSR ≥$50		
Resources within Stopes	15,435,594	79.08
Internal Dilution	1,820,630	38.21

There is room to optimize the reserve further at a higher or lower NSR cut-off than the US$50/t finally adopted, but this would best be undertaken during the next phase of study when the resources are better defined. At the next phase of study, cut-off grades for each mining method and for mining above and below the crusher should be used to define the stopes rather than a blanket approach.

Table 17.8 provides a listing of the resources with greater than the US$50/t NSR design cut-off that were used to create the stoping outlines and estimate reserves.

Table 17.8 Mineral Resources at US$50/t NSR Design Cut-off

Rock Type	Rock Code In Block Model	Class	Tonnes	Density (t/m^3)	Ag (g/t)	Zn (%)	Pb (%)	NSR ($/t)
C-Horizon	30	Indicated	164,296	2.477	315	1.399	0.555	71.68
	40	Indicated	473,069	2.637	217	2.776	1.632	72.02
Andesite	50	Indicated	811,145	2.841	229	3.432	1.632	71.98
Basal Cong.	60	Indicated	6,687,198	2.854	177	3.79	1.056	80.27
Sediments	70	Measured	1,132,365	2.604	206	2.055	1.218	77.75
		Indicated	1,372,134	2.626	200	2.039	1.151	75.72
	80	Measured	2,023,639	2.748	205	2.108	1.452	79.93
		Indicated	2,242,615	2.792	202	1.988	1.327	77.13
	90	Measured	648,828	2.703	175	2.037	1.378	71.37
		Indicated	906,436	2.764	175	2.081	1.229	70.55
Sub-total		Measured	3,804,832	2.698	200	2.08	1.37	77.82
		Indicated	12,656,893	2.798	190	3.067	1.179	77.57
Grand Total			**16,461,725**	**2.775**	**193**	**2.839**	**1.223**	**77.63**

The size of the resource is very sensitive to the metal prices used in the estimate. Depending on the metal prices that are applied, the size of the resource has the potential for a significant increase in tonnage at higher metal prices.

Table 17.9 shows the mineral resources at US$50/t NSR design cut-off estimated using the following metal prices: US$14/oz Ag, US$0.6/lb Pb, and US$0.85/lb Zn.

Table 17.9 Mineral Resources at US$50 NSR Cut-off – US$14/oz Ag, US$0.6/lb Pb, and US$0.85/lb Zn

Rock Type	Rock Code	Class	Tonnes	Density (t/m^3)	Ag (ppm)	Zn (%)	Pb (%)	NSR ($/t)
C-Horizon	30	Indicated	326,912	2.492	236	1.396	0.566	75.42
	40	Indicated	890,322	2.627	178	2.29	1.287	76.25
Andesite	50	Indicated	1,517,612	2.854	188	2.828	1.242	76.27
Basal Cong.	60	Indicated	10,083,705	2.856	146	3.303	0.913	88.18
Sediments	70	Measured	1,923,749	2.603	163	1.832	1.004	82.60
		Indicated	2,301,047	2.622	159	1.829	1.021	81.33
	80	Measured	3,213,225	2.751	166	1.866	1.249	86.10
		Indicated	3,863,744	2.792	158	1.768	1.144	81.60
	90	Measured	1,235,564	2.708	144	1.661	1.089	75.19
		Indicated	1,762,111	2.783	144	1.672	1.005	74.41
Sub-total		**Measured**	**6,372,538**	**2.698**	**161**	**1.816**	**1.144**	**82.93**
Sub-total		**Indicated**	**20,745,453**	**2.796**	**155**	**2.607**	**1.01**	**83.44**
Grand Total			**27,117,991**	**2.773**	**157**	**2.421**	**1.042**	**$83.32**

17.2.1 *Dilution and Recovery*

Dilution is defined as the ratio of waste to ore. Two sources of dilution will be generated, namely internal or planned dilution and external or unplanned dilution.

Internal Dilution

Internal dilution is also known as planned dilution and derives from material with grades that are less than the break-even cut-off grade that falls within a designed stope boundary (i.e. it will be drilled and blasted within the stope during mining). Practical mining considerations usually make the inclusion of internal waste unavoidable.

In particular with the Pitarrilla orebody, the grade distribution within stopes is not heterogeneous, and fairly significant internal dilution at below the break-even cut-off NSR is included into the reserve.

The mining blocks designed for the longhole stoping areas contain internal dilution so the amounts have been estimated and reported separately as internal dilution in the long hole stoping reserves (Table 17.11).

In the room and pillar blocks, less detailed design has been undertaken of the mining blocks. It has been assumed that, in many cases where less than cut-off grade zones are encountered, they may be left as pillars due to the highly flexible nature of room and pillar mining. Thus no internal dilution has been included in the room and pillar stoping block reserves.

External Dilution

External dilution is also known as unplanned dilution and derives from low or zero grade material from beyond the stope design boundaries. This material is not drilled and blasted during the mining process but derives from blasting overbreak due to inaccurate drilling, high local powder factors, adverse geological structure, failure within zones of weak rock, etc. External dilution is almost always generated and an allowance is always made for it during the reserve estimation process.

An in situ waste rock density of 2.8 t/m^3 and backfill density of 1.94 t/m^3 were used to estimate the dilution tonnage by accordingly applying those densities to estimated dilution volume. Wardrop conservatively assumed that the external dilution has no mineral value.

Room and Pillar External Dilution

External dilution of 6.6% for room and pillar mining has been estimated for an average stope size by adding 0.15 m overbreak to stope dimensions in waste rock (roof and rock floor) of zero grade, and 0.3 m backfill dilution when mucking on a backfill floor.

Longhole Stoping External Dilution

In the primary longhole stopes, generally only two walls will have exposure to external dilution (in some cases just one wall, when there will be a few stopes in an extraction row), as there will be secondary stopes on the sides. 0.5 m in waste rock from beyond design stope outlines has been allowed for at zero grade as well as 0.4 m from an exposed fill wall at zero grade, and 0.3m from a backfill floor at zero grade. A typical stope configuration was used and applied to all primary stopes. An average external dilution for primary stopes was estimated to be 5.5% at zero grade.

Since secondary stopes will typically have two fill wall exposures, external dilution will be higher for these stopes. In secondary longhole stopes, 0.5 m in waste rock from beyond stope outlines has been allowed for at zero grade as well as 0.4 m from an exposed fill wall at zero grade, and 0.3 m from a backfill floor at zero grade. A typical stope configuration was used and applied to all secondary stopes.

An average external dilution for secondary stopes was estimated to be 9.2% at zero grade, totalling the average longhole external dilution to be 7.3%. More detailed approach for dilution estimate should be a done at the next phase of study including detail investigation of the size and the exposure to waste and backfill dilution of individual stopes.

Table 17.10 shows the estimated average external dilution applied to longhole stoping mining blocks.

Table 17.10 Longhole Dilution

	Stope Size (m)			Density (t/m^3)	Tonnage	Dilution (%)
	H	W	L			
Ore in Average Longhole Stope	30	15	15	2.8	18,900	
Primary Stopes						
Waste Dilution from Walls	30	15	0.75	2.69	908	4.8
Backfill Dilution from Floor	0.3	15	15	1.94	131	0.7
Total Primary Stope Dilution						5.5
Secondary Stopes						
Waste Dilution from Walls	30	15	0.75	2.69	908	4.8
Backfill Dilution from Floor	0.3	15	15	1.94	131	0.7
Backfill Dilution from Walls	30	15	0.8	1.94	698	3.7
Total Backfill Dilution						4.4
Total Secondary Stope Dilution						9.2
Average Longhole External Dilution						7.3
Average Longhole Internal Dilution						23.56

MINING RECOVERY

All ore blasted within a stope generally does not find its way to the process plant. This is due to a number of causes such as:

- Underbreak – the ore is not blasted and remains on the stope walls.
- Ore loss within stope – the blasted ore is left in the stope due to poor access for the loader, buried by falls of waste rock from walls, left on the floor, blasted but does not fall from flatter lying walls, or gets mixed into the backfill floor and is left behind.
- Ore loss in the handling process – the mucked ore is dropped into a waste pass rather than an ore pass.

The contribution of each of these losses varies depending on the particular mine. For the purposes of this study, an average mining recovery of 95% was assumed for both the room and pillar and longhole stoping methods.

17.2.2 *MINERAL RESERVE ESTIMATE*

The cut-off mining NSR was estimated to be US$50/t. In order to convert resources to reserves, a mine plan must be prepared determining how the resources may be mined and then appropriate dilutions and recoveries applied to arrive at estimated reserves.

In this study, the block model was interrogated to graphically show all blocks with NSR greater than US$50 in the Surpac mining software package. The orebody was divided into sublevels of 30 m; each sublevel was examined and stoping outlines were drawn around the >US$50 NSR blocks. In order to maintain practical stope outlines, some lower NSR blocks are unavoidably included within stope outlines as internal dilution and some higher NSR blocks will be excluded from design stope outlines. Small volume outliers of higher grade blocks are also excluded as either being impractical to mine or not worth the cost of development to reach them.

Mining blocks that have dips of less than 55° will become room and pillar blocks or post-pillar cut and fill blocks; those steeper than 55° will become longhole stoping blocks. A minimum thickness or height of 3 m was applied to outline flatter lying room and pillar blocks, and 3 m thickness and 30 m height was applied for the longhole stoping blocks.

LONGHOLE STOPING RESERVES

More detail was applied when defining the steeper longhole stoping blocks since the method is relatively inflexible and requires good definition of the stoping blocks. The stope outlines have minimum dips of 55° on the footwalls to be sure the blasted ore will run during mucking. Outlines were optimized by visual judgement to include lower grade material and or exclude higher grade material.

Once the outlines were finalized, the block model was interrogated to determine all the material contained within each designed stoping block outline, regardless of NSR. The tonnages and grades for blocks above and below the cut-off NSR of US$50 were determined and tabulated. A typical average external dilution of 7.3% was then applied as well as a mining recovery of 95%. It was assumed that the primary and secondary sequencing using paste backfill will allow 100% extraction of the longhole stoping blocks.

ROOM AND PILLAR RESERVES

The room and pillar stoping blocks were more loosely defined since this mining method is very flexible and allows lower grade areas to be left during mining. Accordingly, once the stoping blocks were defined, it was assumed that only the blocks above the cut-off NSR of US$50 would be mined. The block model was interrogated for blocks >US$50. An extraction ratio of 89% was then applied to allow for losses in the permanent pillars. The typical average estimated external dilution of 6.6% was then applied, and finally a mining recovery of 95% to arrive at the reserve.

RESERVE STATEMENT

The reserve statement is provided in Table 17.11 on the following page. Reserves have been defined between the 1670 m and 1160 m elevations within the orebody. Some above cut-off material lies above and some lies below these elevation limits but were considered insufficient to warrant the further development necessary to access and mine them.

Table 17.11 Pitarrilla Mineral Reserves at US$50 NSR Cut-off

Rock Type	Mining Method	Class	Tonnes	Density (t/m^3)	Ag (ppm)	Zn (%)	Pb (%)	NSR ($/t)
C-Horizon	Room & Pillar	Probable	511,182	2.62	230	2.76	1.71	73.33
		(Recovered)	431,665	2.62	230	2.76	1.71	73.33
		(Diluted)	460,034	2.60	216	2.59	1.61	68.81
	Sub-total	**Probable**	**460,034**	**2.60**	**216**	**2.59**	**1.61**	**68.81**
Andesite	Longhole	Probable	273,046	2.83	280	3.51	1.37	79.25
		(Internal dilution)	31,335	2.81	154	1.88	0.50	41.52
		(Recovered)	289,162	2.83	267	3.35	1.28	75.37
		(Diluted)	310,398	2.80	249	3.12	1.19	70.21
	Room & Pillar	Probable	506,083	2.84	198	3.46	1.77	67.92
		(Recovered)	427,359	2.84	198	3.46	1.77	67.92
		(Diluted)	455,445	2.81	186	3.24	1.66	63.73
	Sub-total	**Probable**	**765,843**	**2.81**	**211**	**3.19**	**1.47**	**66.36**
Basal Cong.	Longhole	Probable	1,660,810	2.75	158	5.42	1.07	89.86
		(Internal dilution)	184,081	2.75	69	2.47	0.54	40.66
		(Recovered)	1,752,646	2.75	149	5.13	1.02	84.95
		(Diluted)	1,881,362	2.73	139	4.78	0.95	79.14
	Room & Pillar	Probable	5,143,386	2.89	182	3.31	1.05	77.37
		(Recovered)	4,343,304	2.89	182	3.31	1.05	77.37
		(Diluted)	4,628,749	2.85	171	3.11	0.98	72.60
	Sub-total	**Probable**	**6,510,110**	**2.82**	**162**	**3.59**	**0.97**	**74.49**
Sediments	Longhole	Probable	5,795,101	2.72	207	2.08	1.39	79.63
		(Internal dilution)	1,605,214	2.71	88	1.21	0.77	37.86
		(Recovered)	7,030,299	2.71	181	1.89	1.26	70.57
		(Diluted)	7,546,608	2.69	169	1.76	1.17	65.74
	Room & Pillar	Probable	1,545,986	2.72	202	2.01	1.24	76.65
		(Recovered)	1,305,499	2.72	202	2.01	1.24	76.65
		(Diluted)	1,391,298	2.70	190	1.89	1.16	71.93
	Sub-total	**Probable**	**8,937,905**	**2.70**	**172**	**1.78**	**1.17**	**66.71**
Total	**Longhole**	**Probable**	**9,738,368**	**2.70**	**165**	**2.39**	**1.13**	**$68.47**
	Room & Pillar	**Probable**	**6,935,525**	**2.80**	**179**	**2.84**	**1.10**	**$71.63**
Grand Total		**Probable**	**16,673,893**	**2.74**	**171**	**2.57**	**1.12**	**$69.79**

18.0 OTHER RELEVANT DATA AND INFORMATION

18.1 MINING OPERATIONS

18.1.1 GEOTECHNICAL CONDITIONS

A geotechnical study has been completed by Knight Piésold and stoping dimensions have been based on their recommendations. For details, refer to Section 18.5. The complete report by Knight Piésold entitled “Silver Standard Resources Inc. La Pitarrilla Project, Geomechanical Input into Underground Mine Design” dated August 29, 2008 is available in “Pitarrilla Pre-feasibility Study Volume 2 – Appendices”.

18.1.2 MINING METHODS

Bearing in mind the geotechnical considerations, inspection of the block model showing the blocks with NSR higher than US$50 reveals that there are two main trends of mineable mineralization. The upper part of the orebody tends to be tabular and flat-lying which would lend itself to room and pillar type mining methods. The lower part of the orebody is moderately to steeply dipping with good widths, which is amenable to bulk mining methods such as longhole stoping. Some sections of the upper part of the orebody also have some sections amenable to longhole stoping and, conversely, some sections of the lower part of the orebody are amenable to room and pillar mining.

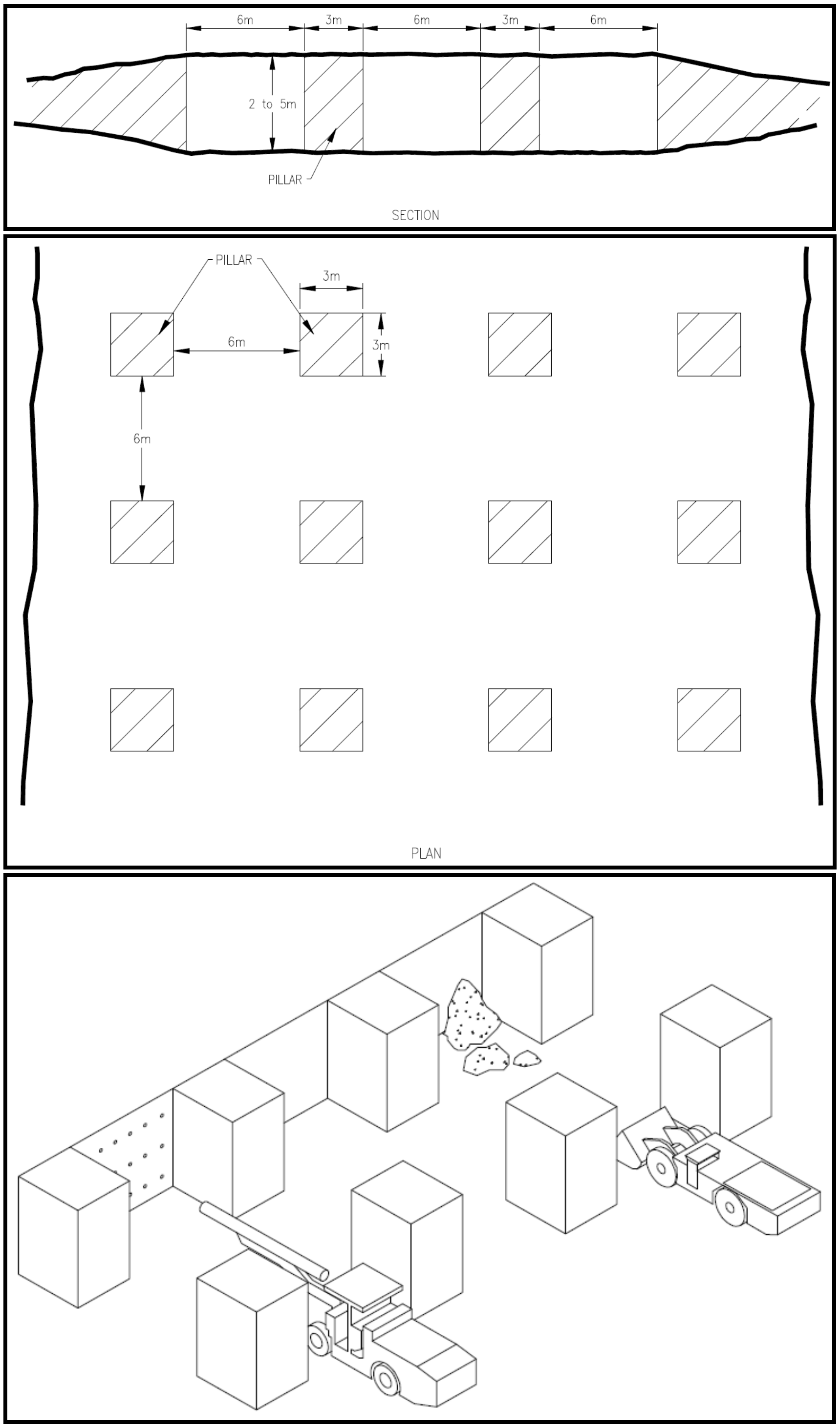
ROOM AND PILLAR

The room and pillar method proposed will be a hybrid post pillar cut-and-fill and true room and pillar method. The reason for adopting this approach is that the room and pillar stopes will be backfilled after mining is completed. The backfilling allows the cutting of smaller pillars, as long-term stability will be provided by the backfilling thus removing the requirement for long-term pillar stability.

Rooms have been designed 5 m high and 6 m wide. Pillars will be 3 m by 3 m on side and 5 m high. This configuration allows an extraction of at least 89%. Some room and pillar blocks are thicker than 5 m and some are moderately inclined. In these blocks, true post pillar mining will be undertaken and the blocks mined with successive cuts after backfilling, carrying the 3 m by 3 m post pillars vertically upward through the cuts (Figure 18.1).

This method is highly productive, flexible, and requires minimal access development before production starts. The main disadvantage is that it is development-style mining and an entry method requiring rock support on every cut.

Figure 18.1 Room and Pillar Mining Method



LONGHOLE STOPING

Mining blocks having dips of at least 55° on the footwall side may be mined using bulk longhole stoping methods.

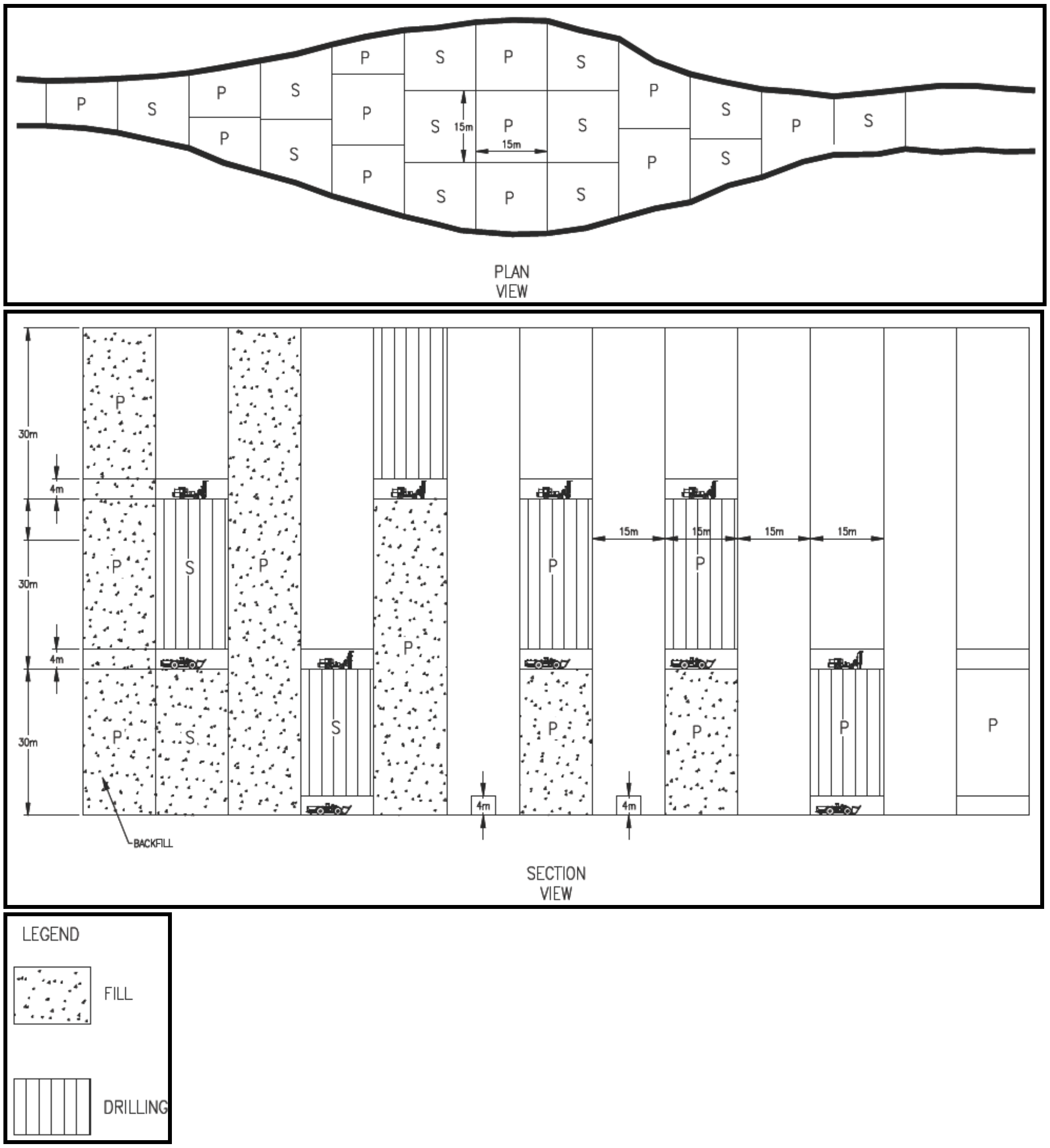
Geotechnical considerations have led to stope sizes of 15 m along strike up to 15 m transverse and sublevel to sublevel heights of 30 m. Where orebody widths are less than 15 m, the stopes will be mined to the orebody width. Where the orebody widths are greater than 15 m, multiple transverse stopes (each with up to 15 m transverse width) will be mined in sequence across the orebody from hanging wall to footwall. The hanging wall stope will be mined first, then filled with pastefill and the next 15 m by 15 m stope mined alongside, retreating toward the footwall.

A primary-secondary mining sequence will be adopted in a bottom-up sequence. After primary stopes are mined, they will be filled with cemented pastefill of adequate strength to allow exposure of a 35 m high by 15 m wide fill wall within the secondary stopes that will be mined alongside. The primary-secondary stoping sequence with the paste backfilling will allow 100% extraction in the longhole stoping blocks.

Two lifts of primary stopes will be mined before the first secondary stopes may be started to allow the drilling drifts to be reused as mucking drifts for the next sub-level up (Figure 18.2).

Primary and secondary stopes will be developed by driving a 5 m wide by 4 m high access drift central to the stope. The central drive will be supported and then slashed to the full stope width of 15 m. The slashes will also be supported. The stope will then be drilled off and mined.

Figure 18.2 Longhole Mining Method



MINE PRODUCTION RATE

The following factors were considered in the estimation of the underground mine production rate:

- reserve tonnage and grade
- geometry of the orebodies
- the amount of the required development
- the sequence of the mining and stopes availability.

The optimum mine production rate can be theoretically determined by applying Taylor's formula, where:

$$\text{Optimum Production Rate} = \frac{5\ \text{x}\ (\text{Expected Reserves})^{3/4}}{(\text{Production Days per year})}$$

Based on Taylor's formula, with an expected reserve of 16,673,893 t ore at 350 production days, the optimum production rate for underground operation can be 3,728 t/d. However, 4,000 t/d was considered appropriate due to the high degree of mechanization, high productivities of selected stoping methods, and provided proper on-going development and stoping schedule.

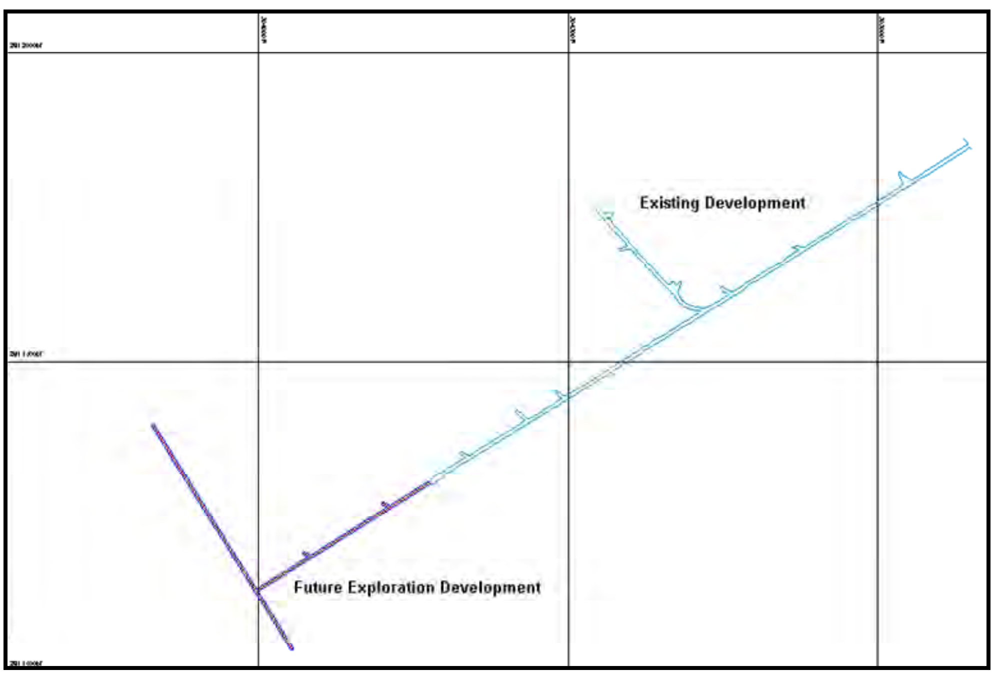
Underground mine life was estimated at 12 years, not including 2 years of pre-production.

18.1.3 *MINE DESIGN*

EXISTING DEVELOPMENT

A 5 m by 5 m exploration decline is currently being driven and, in March 2009, had almost reached the orebody. Once the decline reaches the orebody at approximately 1600 m elevation, an exploration drilling level will be driven along strike for the full length of the currently defined mineralization. It is planned to utilize this exploration development for ventilation purposes during pre-production development and operation of the mine.

Figure 18.3 Existing and Future Exploration Development



MINE ACCESS

It was decided to access the mine via declines. The terrain above the orebody is quite rugged and the oxide / sulphide cap may be mined using open pit methods. The plant site has been located in a valley some distance in the hanging wall side of the orebody. As the orebody is located beneath a high hill, it is convenient to drive declines from the valley of the plant site into the orebody. Shaft access would otherwise need to be located some distance from the orebody to avoid the potential open pit, which would also be inconveniently located with respect to the process plant location for ore delivery to the plant.

Conveyor Decline

A crusher station will be established underground. Ore and waste will be hauled from the mine via conveyor in a dedicated decline from the crusher at 1250 m elevation to the plant site. The decline will have dimensions of 5.0 m (W) by 5.5 m (H) and will also act as a major exhaust airway. Ore below the crusher station level will be trucked up a ramp to the ore and waste bin systems for the crusher station.

Backfill Decline

The backfill preparation plant will be located underground near the top of the orebody. Dry filtered tailings will be conveyed to the underground backfill plant on a conveyor. This conveyor will be located in a dedicated 5.0 m (W) by 5.5 m (H) decline. Part of the conveyor will be overland to the backfill decline portal. The decline will also be used for delivery of cement to the backfill plant as well as serving as a major exhaust airway and access to the top levels of the mine.

Exploration Decline

The exploration decline will be used for access for men, equipment, and materials, and will also be a major intake airway. A drive from the exploration drift will be driven around the southern side of the orebody to access the main mine ramp, which will be located in the footwall of the orebody at its southern end.

Intake Ventilation Raise

A second air intake will be required on the north side of the orebody to augment air entering the exploration decline. This will be a raise bored with 3.0 m diameter shaft located just beyond the limits of the potential open pit. A 5.0 m by 5.0 m access to the bottom of the shaft location will be driven from the northern extremity of the exploration drill drift on 1600 m elevation. The access will be driven upward at 15% to reduce the length of the shaft.

Footwall Ramp

A 5.0 m by 5.0 m footwall ramp will be used to access sublevels within the mine developed at 30 m vertical spacing. It will connect upper mining blocks down to the crusher level at 1250 m elevation, as well as to mining blocks below the crusher elevation. Ore from the mining blocks below the crusher elevation will be truck-hauled up to the crusher via the footwall ramp.

Fresh Air Distribution Raises

A fresh air raise will be driven between sublevels on the northern end of the orebodies. The southern side of the sublevels will be supplied with fresh air via the main access ramp.

Exhaust Air Raises

An exhaust air raise will be located in the centre of the orebody connecting upwards to the backfill decline exhaust airway and downward to the conveyor decline exhaust airway.

Ore and Waste Passes

Ore and waste passes will be centrally located to transfer directly to ore and waste bins located above the crusher station.

Figure 18.4 Mine Access Development – Plan View

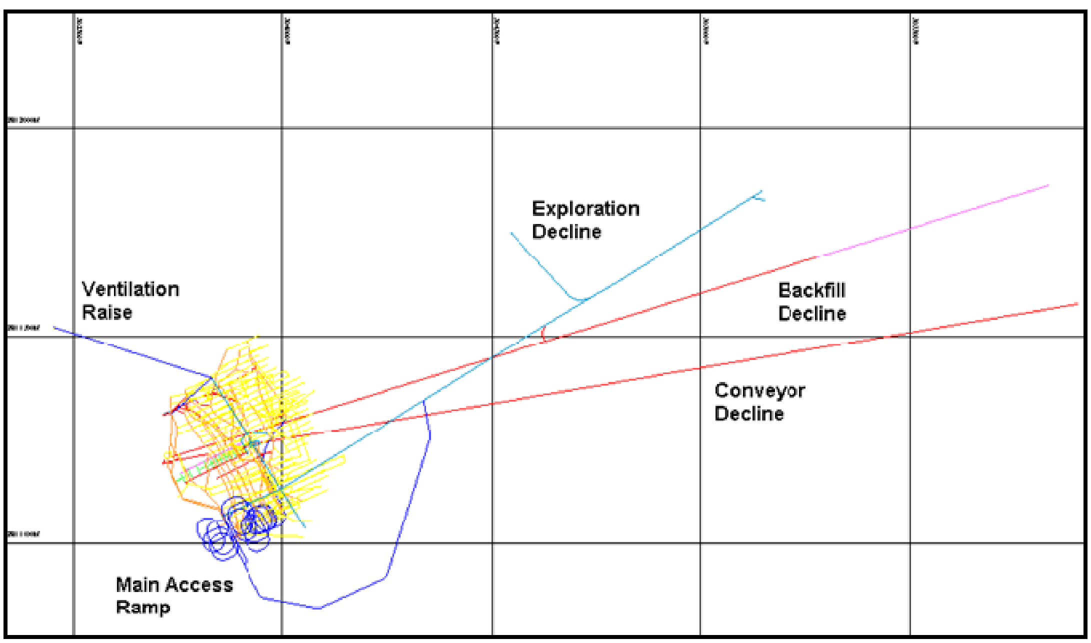
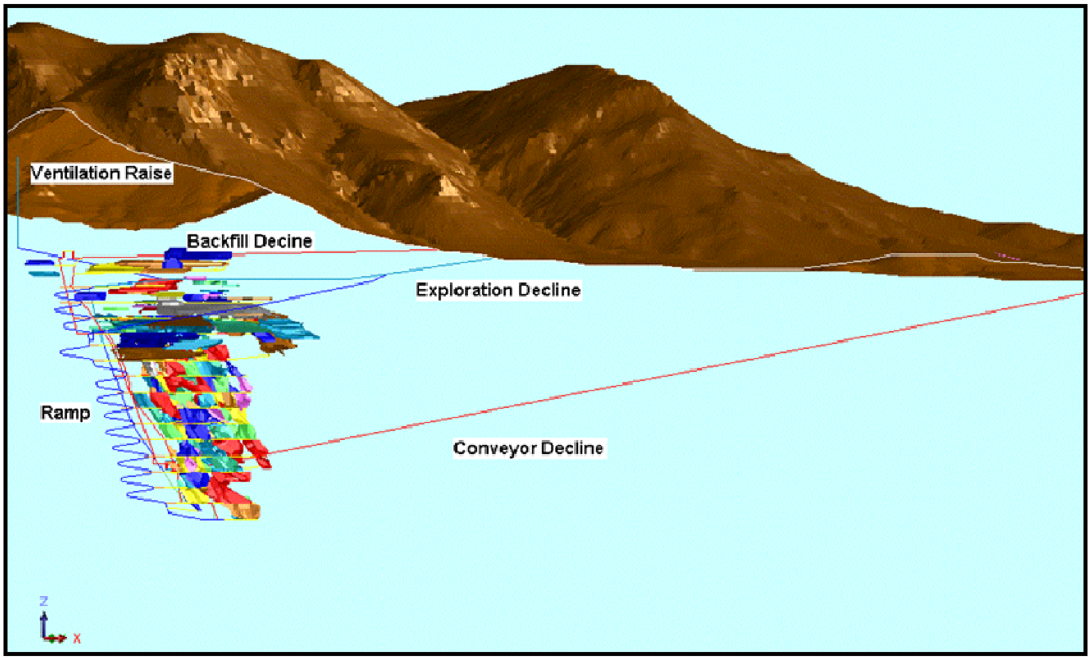


Figure 18.5 Mine Development and Stoping Design – Section View



18.1.4 DEVELOPMENT SCHEDULE

The objective of the mine scheduling was to achieve early ore production from higher-grade areas. The mine development was divided into two periods:

- pre-production development (prior to mine production)
- ongoing development (during production).

Pre-production development will be scheduled to:

- provide access for trackless equipment
- provide ventilation and emergency egress
- establish ore and waste handling systems
- install mining services (backfill distribution, power distribution, communications, explosives storage, fuel storage, compressed air supply, water supply, mine dewatering)
- provide optimum sub-level development in advance of start-up to develop sufficient ore reserves to support the mine production rate.

The underground pre-production development schedule does not include any allowances for obtaining the environmental and mining permits, construction of surface access roads, power lines, and site infrastructure construction.

It was assumed that all underground pre-production development will be contracted by a Canadian contractor in order to ensure the required advance rates of development, and was costed accordingly.

Development cycle times were estimated for each development heading (Table 18.1). Considering estimated development advance rates and North American contractor practices, it was assumed that a jumbo crew advance rate at a main decline development can be approximately 130 m per month per single heading. It was assumed that for the purpose of completing the development schedule on time, an additional jumbo crew might be hired when multiple headings will be available.

The vertical and inclined development of ventilation raises, as well as ore and waste passes, will be done by raiseboring and Alimak crews. An advance rate of 120 m per month of Alimak All underground infrastructure development, such as the backfill plant, crusher chamber, maintenance shop, and dewatering sump, will be completed prior to production.

raise development was used in mine development schedule. It was assumed that a raiseboring crew can drill approximately 450 m per month of pilot holes, and ream it to the 2.4 m diameter at an advance rate of 90 m per month.

It was estimated that all pre-production development will be completed in two years.

Table 18.1 Development Cycle Times

	Unit	Decline (5.0 x 5.5)	Ramp (5.0 x 5.0)	Cross-cut (5.0 x 4.0)	Ore Drift (5.0 x 4.0)	Drift Slash (15.0 x 4.0)	Vent Raise (3.0 x 3.0)	Ore Pass (2.4 m dia.)	Slot Raise (2.5 x 2.5)
Design Criteria									
Width	m	5	5	5	5	15	3	2.4	2.5
Height	m	5.5	5	4	4	5	3	2.4	2.5
Gradient	%	17	15	0.2	0.2	0.2	90	90	70
Summary Cycle Times per Shift									
Drilling	h	4.24	4.24	3.81	3.81	4.36	3.58		37.44
Blasting	h	1.83	1.83	1.58	1.58	2.00	1.61		44.51
Re-entry	h	0.50	0.50	0.00	0.00	0.00	0.50		
Mucking	h	2.10	1.94	1.00	1.00	2.67	1.39		3.49
Support	h	6.20	5.76	3.75	3.75	7.22	2.07		
Services	h	1.32	1.04	0.52	0.52	0.00	1.20		
Secondary Mucking	h	3.43	2.64	0.00	0.00	0.00			
Trucking	h	5.76	4.16	0.00	0.00	0.00	1.93		0.00
Single Heading									
Critical Path Cycle Time	h	16.18	15.30	10.65	10.65	16.26	10.35		85.45
Advance Per Shift	m	2.10	2.22	3.18	3.18	2.38	2.03		2.75
Advance Per Day	**m**	**4.19**	**4.43**	**6.37**	**6.37**	**4.77**	**4.06**	**4.00**	**5.50**

18.1.5 *PRODUCTION SCHEDULE*

The production schedule has been prepared based on a 42% room and pillar and 58% longhole stoping mix, which roughly represents the distribution of reserves in each mining method. Equipment fleets and personnel for each mining method should thus be kept stable during operation of the mine.

After Year 3 of production, a steady flow of ore from mining blocks below the crusher is maintained, again to maintain a constant fleet of haulage trucks.

Mining commences at the 1490 level and at the 1370 levels. These are logical start locations, which also access higher grades at the start of the mine life. Ore geometry is suitable for room and pillar mining on the 1490 level and for longhole mining on the 1370 level, which will allow the required split of ore production from the two different mining methods.

It was assumed that 2 or 3 production sub-levels will be available at anytime during the mine life, ensuring that there will always be enough stopes available to satisfy the 4,000 t/d mine production rate.

Table 18.2 Mine Production Schedule

Mining Method/Rock Type	Units	Production Year												Total
		1	2	3	4	5	6	7	8	9	10	11	12	
Longhole – All Rock Types	**t**	**724,000**	**820,000**	**820,000**	**820,000**	**820,000**	**820,000**	**820,000**	**820,000**	**820,000**	**820,000**	**820,000**	**814,368**	**9,738,368**
Room & Pillar– All Rock Types	**t**	**550,000**	**580,000**	**580,000**	**580,000**	**580,000**	**580,000**	**580,000**	**580,000**	**580,000**	**580,000**	**580,000**	**585,525**	**6,935,525**
Total– All Rock Types	**t**	**1,274,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,400,000**	**1,399,892**	**16,673,892**
Ag	g/t	200	161	151	156	196	174	169	171	170	165	163	176	**171**
Zn	%	2.22	3.34	3.71	3.51	2.50	2.18	2.49	2.27	2.18	1.98	2.28	2.19	**2.57**
Pb	%	1.13	1.05	1.00	1.08	1.07	1.10	1.08	1.01	1.04	1.29	1.23	1.34	**1.12**
Average NSR	US$/t	75.16	74.17	73.63	74.35	70.88	68.94	69.84	68.56	67.55	63.37	66.05	65.40	**$69.79**
Sediments	t	724,000	431,453	120,000	338,638	599,602	940,000	820,000	1,070,264	936,133	880,406	1,084,037	993,372	**8,937,905**
Ag	g/t	185	194	185	169	193	178	174	175	170	148	159	165	**172**
Zn	%	1.83	1.81	1.83	1.99	1.67	1.71	1.75	1.76	1.70	1.81	1.86	1.80	**1.78**
Pb	%	1.21	1.23	1.21	1.44	1.07	1.15	1.14	1.02	1.10	1.30	1.23	1.19	**1.17**
NSR	US$/t	70.82	72.95	70.82	69.87	70.16	67.35	66.68	66.14	65.03	61.97	64.61	65.40	**$66.71**
Basal Conglomerate	t	550,000	968,547	1,280,000	1,061,362	490,000	460,000	580,000	329,736	463,867	206,583	120,016		**6,510,110**
Ag	g/t	220	146	148	152	167	167	162	157	171	192	123		**162**
Zn	%	2.72	4.03	3.88	4.00	3.14	3.14	3.54	3.93	3.14	1.93	4.76		**3.59**
Pb	%	1.02	0.97	0.98	0.97	0.99	0.99	0.99	0.99	0.94	0.88	0.59		**0.97**
NSR	US$/t	80.87	74.72	73.89	75.78	72.18	72.18	74.31	76.40	72.64	66.74	72.60		**$74.49**
Andesite	t					310,398						99,234	356,210	**765,843**
Ag	g/t					249						182	187	**211**
Zn	%					3.12						3.19	3.26	**3.19**
Pb	%					1.19						1.57	1.69	**1.47**
NSR	US$/t					70.21						62.16	64.17	**$66.36**
C-Horizon	t										313,011	96,712	50,310	**460,034**
Ag	g/t										195	240	303	**216**
Zn	%										2.51	2.98	2.34	**2.59**
Pb	%										1.52	1.69	2.00	**1.61**
NSR	US$/t										65.07	78.13	74.18	**$68.81**

18.1.6 ORE HAULAGE

The waste rock from the development headings will be mucked by a 10-t load-haul-dump (LHD) to remuck bays located up to 150 m from the face, and then hauled by 30-t trucks to either surface, a waste pass, or to an assigned mined-out secondary stope.

The broken ore will be mucked from the stopes above the crusher by 17-t LHD and hauled directly to an ore pass. The average hauling distance from the stopes to the ore pass system is about 250 m. A sloping parallel bar grizzly with 450 mm openings will be installed at each sublevel dumping point to prevent oversize from entering the ore pass system. A mobile hydraulic rock breaker will be used to break oversized material at the grizzlies.

The ore from the stopes below the crusher will be mucked by 10-t LHD and loaded at the closest cross-cut and footwall drift intersections onto 30-t underground trucks. It will then be hauled to the bins above the crusher. The selection of LHD and trucks was based on LHD and truck box capacities, to allow for loading the trucks with a maximum of three passes. Remote LHD capability is required for longhole stope mucking.

The ore pass discharge will be equipped with a Ross feeder system to control the rate of discharge onto a vibrating grizzly and into the underground jaw crusher. The conveyor belt, approximately 2 km long at approximately +18%, will carry the ore from the underground crusher to the mill. The conveyor belt would be hung from threaded rebar epoxy grouted into the back to facilitate mechanized clean up with a small LHD.

18.1.7 PASTE BACKFILL

INTRODUCTION

The paste backfill system will be designed to deliver 229 to 236 t/h of backfill to underground openings at an average cement content of 3 to 6% from the underground paste plant.

Pastefill will be utilized to provide wall stability and reduce convergence after ore extraction, act as dilution control, and increase recovery by allowing secondary stopes to be mined with minimal ore loss in regional pillars.

The pastefill plant at Pitarrilla will operate for 22 h/d and will be capable of filling stopes measuring 30 m (H) by 15 m (W) by 15 m (L) in approximately 2.5 days. The stope will be filled to the floor level of the drill drive (upper sill) for mining of the consecutive mining panel above. This is approximately 6,750 m^3 of effective fill volume.

METHODOLOGY

Tailings separated in the flotation cells at the mill will be pumped to the disk filters for further dewatering to 85% solids.

Discharge from the disk filters will be transported to either the 4,000 t surface tailings stockpile, directly underground to the paste plant, or to the dry stack tailings disposal facilities.

Filtered material for the paste plant will be directed onto the underground tailings conveyor and, during paste plant shutdown, the material will be either discharged onto the surface tailing stockpile or conveyed to the dry stack. The recovery of the stockpile material will be by front-end loader onto to the underground tailings conveyor.

The underground tailings conveyor will be designed to deliver nominal 166.7 dry tonnes per hour (dt/h) of filter cake. Cement will be delivered underground via cement tanks by low profile flatbed trucks and metered into the mixer by the underground cement storage system.

The filter cake and cement will be discharged directly into a continuous mixer where additional process water will be metered into the tailings for percentage solids and strength, depending on its applications before final delivery to open stopes.

ESTIMATED TAILINGS PARAMETERS

A summary of the design parameters for Pitarrilla tailings is listed in Table 18.3.

Table 18.3 Design Parameters for Tailings and Pastefill Materials

Description	Unit	Cement Addition		
		3%	4%	6%
Tailings Parameters				
Milling Tonnage	t/d	4,000	4,000	4,000
Estimating Tailings Produced	t/d	3,800	3,800	3,800
Backfill Plant Operating Hours	h/d	22	22	22
Tailings Required for Backfill	t/d	3,800	3,800	3,800
Solid Tailings (per hour)	dt/h	166.7	166.7	166.7
Density of Solid Tailings	t/m^3	2.80	2.80	2.80
Solid Tailings (Dry)	m^3/h	59.53	59.53	59.53
Percentage Solids of Tailings	%	85	85	85
Water in Tailings	m^3/h or t/h	29.41	29.41	29.41
Tailings Pulp (Tonnage)	t/h	196.10	196.10	196.10
Density of Tailings Pulp		2.20	2.20	2.20

table continues…

Description	Unit	Cement Addition		
		3%	4%	6%
Cement Parameters				
Density of Cement	t/m^3	3.15	3.15	3.15
Binder Addition Estimated	%	3	4	6
Binder Consumption	dt/h	5.2	6.9	10.6
Binder Consumption per Day (time to fill)	t/d	113.4	152.8	234.1
Binder Consumption	m^3/h	1.64	2.20	3.38
Pastefill Parameters				
Target Percentage Solids	%	75	75	75
Solid of Pastefill per Hour	dt/h	171.8	173.6	177.3
Density of Solid Pastefill	t/m^3	2.81	2.81	2.82
Solid Tailings	m^3/h	61.17	61.73	62.91
Water in Pastefill	m^3/h or t/h	29.41	29.41	29.41
Percentage Solids of Pastefill	target %	75	75	75
Wet Scrubber Discharge	m^3/h or t/h	2	2	2
Water Addition to Yield Target % Density	m^3/h or t/h	25.84	26.46	27.69
Total Water Addition	m^3/h or t/h	27.84	28.46	29.69
Pastefill (Pulp Tonnage/Hour)	t/h	229.09	231.50	236.43
Pastefill (Pulp Tonnage/Day)	t/d	5,040.08	5,093.06	5,201.42
Pastefill (Pulp Volume/Day)	m^3/d	2,605.28	2,631.38	2,684.27
Pastefill Density (Pulp)	t/m^3	1.93	1.94	1.94
Pastefill (Volume/Hour)	m^3/h	118.42	119.61	122.01
Paste Fill Mixer				
Mixer Capacity (Tonnage)	t/h	254.55	257.23	262.70
Mixer Capacity (Volume/Hour)	m^3/h	131.58	132.90	135.57

SYSTEM COMPONENTS

The following is a list of the system components for the Pitarrilla paste plant:

- underground tailings conveyor system
- cement storage and feed system
- mixing system
- pump feed arrangement and emergency discharge system
- paste pumping system
- emergency flushing system and portable flushing system
- paste distribution system
- control, monitoring, and instrumentation
- underground pressure breaking system

- plant piping system
- electrical system.

System Performance

Underground Tailings Conveyor System

The underground tailings conveyor system will be approximately 1,617 m in length with a -2.0% grade ramp. The conveyor system will be equipped with a hopper for material feed directly from the mill tailings conveyor, and from a front-end loader with a capacity of 166.7 t/h during recovery from the stockpile. Filter cake will be discharged onto the belt and a belt scale to measure the material mass flow.

There will be two belt scales on the conveyor at:

- 3 m from the feed hopper rotary system to measure discharge mass of the cake at the surface
- 3 m before the continuous mixer to measure the final mass of the cake for cement addition underground and for water addition before discharging into the mixer.

The belt scales will be regulated by the digital control system (DCS) in the control room to automatically discharge cement from the cement storage system into the mixer and water addition at the mixer.

The conveyor capacity will range from 150 to 184 t/h controlled by the variable frequency drive (VFD) at the DCS panel. The amount of material transported will depend on the mix type and percentage cement addition (Table 18.3). The accuracy of the belt scales is 0.5% of the throughput.

The conveyor system will be enclosed to reduce exposure to environmental factor such as wind, dust, and water contamination from rain on the surface and seepage from underground.

Cement Storage and Feed System

The cement storage system will include one storage tanker on the surface with an underground storage silo complete with feed system, dust house, and blower.

The storage tanker size will be 115 m^3 (4,050 ft^3) with an estimated load capacity of 136 t/unit; the underground storage silo size will be 22.65 m^3 (800 ft^3) with a load capacity of 30 t/unit.

The delivery of cement from the manufacturer to the underground paste plant will consist of two stages as follows:

- cement will be delivered to site by surface trucks and pneumatically discharged into the surface storage tanker
- cement from the surface tanker will be pneumatically discharged into the cement tanker for delivery underground by low-profile flatbed truck.

The total cement storage capacity (166 t) is based on an approximately 8% surge capacity for the system at a consumption rate of 153 t per operation of a 22 h shift for 4% cement addition. Additional cement capacity can be achieved at the two 20 m^3 cement tankers.

The cement addition system will be DCS-controlled and capable of adding cement at a rate from 0 to 10%.

Cement will be discharged onto the weigh belt controlled by the variable speed rotary feeder. The weigh belt will deliver weighted cement to a screw feeder before discharging onto the mixer. A blower located at the rotary feeder and weigh belt will blow cement dust back into the silo. The system will be enclosed to reduce dust generation.

The screw conveyor will be connected to the continuous mixer with a flexible rubber boot, and a wet scrubber (located at the end of the screw conveyor) will control dust generated. The wet scrubber will discharge 2 m^3/h of cement slurry into the mixer during operation.

The rotary feeder, weigh belt, and screw conveyor will be driven by variable speed motors and controlled by the DCS system. The cement system will have the capacity to deliver cement ranging from 5.0 to 11.0 t/h.

The cement storage silo will be pneumatically loaded from 27 t cement tankers.

Mixing System

The continuous mixing system will be capable of producing paste at 130 to 136 m^3/h at a 152 to 203 mm slump at 75 to 80% solids sized to 90% availability.

The feed streams into the mixer are:

- filtered tailings and cement
- water for additional dilution for slump control.

The piping of water for additional dilution to control slump will be designed to provide flows ranging from 25.0 to 33.0 m^3/h to achieve a target 75% solids for the paste mixes.

Types of Mix

The mass balance for all three types of fill at 3%, 4%, and 6% cement is presented in Table 18.3. The 4% cement addition is the estimated average consumption and the latter is for capping, high strength applications, etc. The amount of cement addition presented in the mass balance is an estimated value. The target pastefill percentage solid for all the mixes is at 75%.

Pump Feed Arrangement and Emergency Discharge System

Discharge from the mixer will be directed into the paste hopper. The paste hopper unit will be equipped with a gate valve and chute for emergency discharge of material into a lined container or the sump. A retention time of approximately 5 minutes (12 m^3) is designed for the paste hopper.

Paste Pumping System

Incorporation of the paste pumping system into the mine design depends on the location of the stopes to be filled. The approximate maximum distance from the underground paste plant to the furthest fill stopes is 500 m horizontally, based on the mine design layout.

The underground paste plant is located above the initial stopes to be mined. This allows gravity to assist the delivery of fill to the stopes within the ratio of two to three times the horizontal-over-vertical drop.

One piston pump will be required to deliver the design tonnage of pastefill to stopes greater than three times the horizontal-to-vertical drop. The paste pumps will be capable of continuously delivering up to 135 m^3/h of paste backfill (262 t/h at a specific gravity of 1.94) to all parts of the mine.

The paste backfill pump will be conservatively designed to allow continuous delivery of paste backfill to all required locations in the mine at no greater than 95% of rated working pressure of the backfill pump and the hydraulic pressure of the associated hydraulic power pack.

The paste pump will be capable of delivering continuous paste at 132 m^3/h to a maximum of 135 m^3/h at 152 to 203 mm slump at 75 to 80% solids. The maximum peak pressure is estimated at 3,500 kPa (35 bar), and a continuous operating pressure at 2,338 kPa (23.38 bar). The pumping rate is varied at the hydraulic power pack to maintain a target level in the pump feed hopper.

A friction loss ranging from 5 kPa ± 1kPa is targeted during lateral transportation of paste in the pipeline.

Emergency Flushing and Portable Flushing System

A high-pressure water pump, capable of delivering 3500 kPa (35 bar), will be installed parallel to the piston pump. In the case of piston pump failure, the emergency water pump will be utilized to purge the line. The emergency flush pump has to be capable of delivering process water at 65 to 70 m^3/h.

An additional portable high-pressure water blaster unit will be utilized to purge a blocked line.

Paste Backfill Line Flushing

The paste backfill line will be flushed by water before and after paste backfill is being delivered underground.

Pre-flushing ensures that the line is sleek or lubricated before material is delivered. This will also ensure that the piping is correctly connected to the stope to be poured. The pre-flush is estimated to consume approximately 30.5 m^3 of water per flush.

A complete flush of 60 m^3 is required when delivery of pastefill is completed and before a shut down during a shift change. This allows most of the material in the pipe line to be cleaned out.

It is estimated that one complete pre-flush and flush are required per day, or pour yielding 91.5 m^3 of water delivered to the underground sump. The average flush water generated by flushing is 4.16 m^3/h per day or pour. This estimation is based on 709 m of total line length from the paste plant at 1654 elevation to the 1490 level.

Flush water increases as mining approaches greater depth and the network of piping increases.

Paste Distribution System

The paste distribution system will consist of a series of boreholes and 8" lateral piping. The line will be required to transport 132 to 135 m^3/h of pastefill at a target velocity of 1.24 to 1.28 m^3/s for variable cement addition.

The estimated main-run piping and clamps are 9241.4 kPa (1,340 psi), 8"diameter, Sch. 40, Grade-A rating pipes with Victaulic HP70 couplings rated at 5,500 kPa (800 psi).

The piping will be designed on the basis of 25% reduction in pressure rating for allowance of corrosion and wear. Thus, the maximum working pressure for 8"diameter, Sch. 40, Grade-A pipes is 9,931.0 kPa (1,005.0 psi) for this application.

The permanent and temporary pipe schedules will be selected to match the duty with 25% pressure de-rating allowance for corrosion and wear when flow simulation is determined.

Secondary and close-to-stope delivery lines will be a lower rating pipe schedule to HDPE pipes.

The paste distribution system includes rupture discs to prevent design pressures from being exceeded. The design of the system and operating procedures will prevent pipe rupture due to excessive pressure and will therefore protect personnel from injury as well as maintain the integrity of the piping system.

Control, Monitoring, and Instrumentation

The control system is a stand-alone automated DCS to control the following components:

- underground tailings conveyor system
- cement storage and feed system
- mixing system
- pump feed arrangement and emergency discharge system
- paste pumping system
- emergency flushing system and portable flushing system
- paste distribution system
- monitoring and instrumentation
- underground pressure breaking system
- plant piping system
- electrical system.

Density, flow meters, and pressure transducers are installed at strategic locations to monitor the mix design and pressure generated during transportation.

Underground Pressure Breaking System

The underground distribution piping is equipped with strategically placed pressure relief devices in the form of rupture discs. These discs are fabricated to fail at a pre-determined pressure, which is below the rated pressure of the upstream distribution system components.

Failure of a rupture disc diverts paste to a containment area. The failure is enunciated at the fill plant, causing an automated shutdown of fill production, followed by an automated system flush. However, the distribution system downstream of a failed rupture disc must be manually flushed. The rupture disc locations will be determined when the underground mine planning is defined.

Electrical System

All plant equipment will operate according to the manufacturer power requirement system.

Operating Environment

The project is located on the eastern flank of the Sierra Madre mountain range in the central part of Durango State, Mexico, approximately 150 km east of the city of Torréon with an arid and dry climate. Most of the backfill plant and equipment will be located underground except for the cement storage system.

Operation

The paste plant is designed to operate with full-time attended/automated operation 22 h/d for 350 d/a, with a design overall availability of 90%.

BACKFILL STRENGTH ESTIMATION

An average 4% by weight of cement addition is estimated for pastefill for the blasthole stoping mining at Pitarrilla. This estimate is made to provide initial parameters for mine design and cost analysis estimation.

In general, compared to secondary stopes, primary stopes will require higher cement content because of the free-standing characteristics of the fill. Some secondary stopes will require similar cement addition to primary stopes if the stopes are exposed to another adjacent secondary stope for a thick orebody. Capping and special applications will require higher cement addition to achieve the strength requirement for equipment transiting on the fill and paste plug applications.

The design backfill strength for Pitarrilla is estimated based on 3D analysis for cemented tailings backfill proposed by Mitchell (1981). For design purposes, the simplified equation is as follows:

$$\boxed{UCS = \frac{\gamma H}{(1 + H/L)}}$$

Where:

- *UCS* = unconfined compressive strength of cemented fill (kPa)
- γ = unit weight of fill material

- H = vertical fill height of exposed face (m)
- L = strike length of exposed face (m).

The estimated critical and design strength determined for Pitarrilla is presented in Table 18.4 and Figure 18.6, where a factor of safety of 1.2 is incorporated into the design strength.

The design strength can be estimated by the best fit curve as:

$$UCS_{Fill\ (Design)} = (-0.213 \times W^2) + (16.74 \times W) + 25.01$$

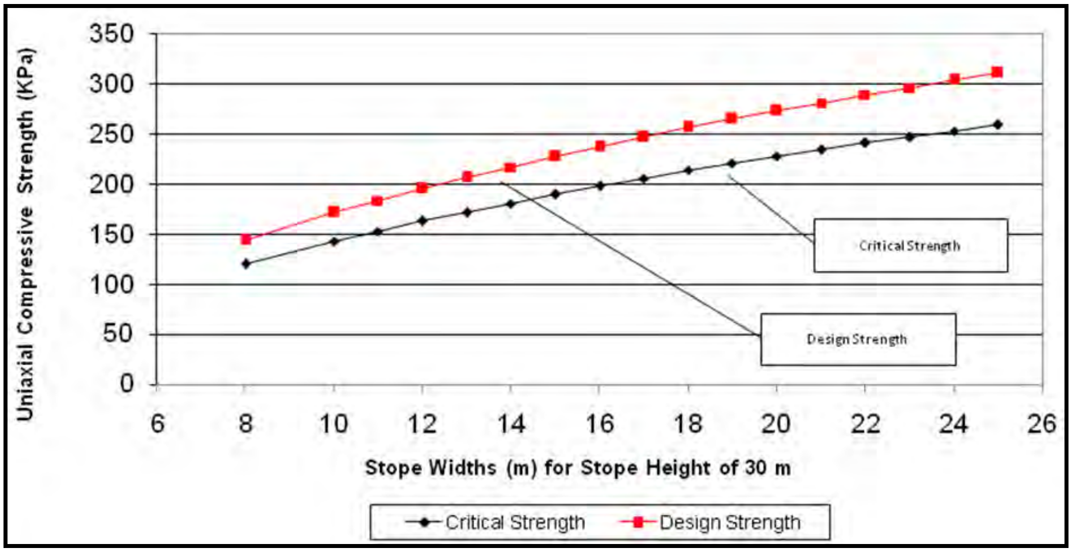
Where W is the width of the exposed face in metres.

For a stope height of 30 m and width of 15 m, the required design strength for the fill is 228 kPa. Table 18.4 and Figure 18.6 present stope exposed length ranging from 8 to 25 m.

Table 18.4 Preliminary Paste Backfill Strength Estimate

Estimated Fill Height (m)	Exposed Face Strike Length (m)	Critical UCS (kPa)	Design UCS (kPa)
30	8	120	144
	10	143	172
	11	153	184
	12	163	196
	13	172	206
	14	181	217
	15	190	228
	16	198	238
	17	206	247
	18	214	257
	19	221	265
	20	228	274
	22	241	289
	23	247	296
	24	253	304
	25	259	311

Figure 18.6 Estimated Strength of Pastefill based on Stope Dimension



Paste backfill plant strength and cement consumption generally vary depending on their applications to the characteristic of the tailings. In the case of the Brunswick mine, for a maximum stope size of 30 m (W) by 30 m (H), the initial 28-day target strength is 1.0 MPa with 5% binder addition. Backfill plugs are poured with 5% cement and a minimum 2% cement addition to the paste in isolated areas to minimize liquefaction potential.

Pastefill strength testing with variable cement additions is recommended to determine the amount of cement addition required to achieve the design fill strength.

The test results will assist in the detailed engineering design of the pastefill plant design, cement storage capacity and distribution system, optimization of the cement content, and cost reduction to the overall operating costs of the paste backfill plant.

PASTEFILL TEST RESULTS

Strength testing of tailings and cement was performed by Unité de Recherche et de Service en Technologie Minérale (URSTM) in Rouyn-Noranda, Quebec, with two tailings samples from Pitarrilla and four locally sourced binder type samples provided by Silver Standard.

Table 18.5 outlines the proposed testing matrix.

Table 18.5 Paste Backfill Test Program (Tailings Types and Binder Selections)

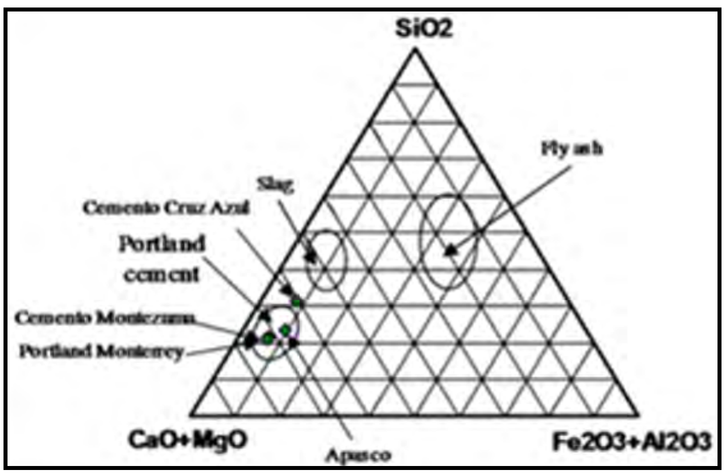
Recipe	Tailings Types	Binder Types		Total Binder Addition (%)	Curing Days					Replicates
		Cement 1 (%)	Cement 2 (%)							
1	Basal	100		3	14	28	56	120	365	3
2	Basal	100		6	14	28	56	120	365	3
3	Basal		100	3	14	28	56	120	365	3
4	Basal		100	6	14	28	56	120	365	3
5	Sediments	100		3	14	28	56	120	365	3
6	Sediments	100		6	14	28	56	120	365	3
7	Sediments		100	3	14	28	56	120	365	2
8	Sediments		100	6	14	28	56	120	365	2

Cement Screening

Cement screening was performed to identify the suitability of the four cement types provided by Silver Standard. The cements are Portland Monterrey, Cemento Cruz-Azul, Cemento Montezuma, and Apasco.

Results from the screening provided by URSTM are plotted in the ternary diagram provided in Figure 18.7. A comparison to Portland cement and the hydraulicity index for each binder is listed in Table 18.6.

Figure 18.7 Local Cement Plotted Ternary Diagram



The hydraulicity index is identified as:

$$I = \frac{SiO_2 + Al_2O_3}{CaO + MgO}$$

Table 18.6 Local Cement Hydraulicity Index

Cement Type	Hydraulicity Index (I)
Portland Monterrey	0.395
Cemento Cruz-Azul	0.653
Cemento Montezuma	0.387
Apasco	0.494

URSTM has suggested the use of Cemento Cruz-Azul and Apasco binders for the study since these binders are near or inside the cement Portland area. They also have a higher hydraulicity index (representing pozzolanic potential) and are more resistant to sulphate attack. Thus, these cements were used in the strength testing.

Test Results and Interpretation

Table 18.7 elaborates on the nomenclature system incorporated for the tailings and laboratory classification system by batch number, binder details, and peak strength on the testing performed. The test results from 14 to 120 days indicate that:

- In general, Apasco cement provides better strength for both tailings types compared to Cruz-Azul.
- Apasco cement at 6% addition provides strength greater than 200 kPa for both tailings types.
- A strength loss is observed with Basal tailings on Apasco cement at 6% addition at 120 days and at 3% addition.
- Cruz-Azul provides higher 14 and 28-day strength to Sediments compared to Apasco (Batch 784 vs. Batch 786) but the incremental increase in strength is lower overall than Apasco cement.
- A strength decrease is observed on Basal tailings with 3% cement addition tested with both Apasco and Cruz-Azul at 56 days (Batch 790 and 788).
- Cruz-Azul cement at 6% addition achieves strength greater than 200 kPa for Basal tailings.
- A strength loss is observed on Basal tailings at 3 and 6% with Cruz-Azul cement addition, and on Sediments tailings at 3%.

The Basal tailings seem to be reactive to both cement types used for testing where there is a strength decrease of 9.2% and 9.7% for Apasco and Cruz-Azul cement, respectively, at 6% addition for 56 to 120 days.

Preliminary results indicate that the use of Apasco cement will yield strengths of 253.8 kPa for Sediments tailings at 56 days, and 251.53 kPa for Basal tailings at 28 days.

Depending on the tailings types used as backfill material, the mining sequence will have to be coordinated to meet these curing periods. Further tailings tests will have to be performed to accommodate for the tailings blend utilized as backfill material during operation. The final test results for 365-day strength testing should be available around December 2009.

Figure 18.8 and Figure 18.9 present a graphical comparison of the test results performed by URSTM up to 120 days of the curing period.

Table 18.7 Tailings and Laboratory Classification System on Backfill Testing

Batch No.	Sample Notes	Curing Period (days)	Percent Binder	Binder Details	Peak Stress (kPa)
786	Sediments	14	3	Apasco : 100	56
		28	3	Apasco : 100	58.53
		56	3	Apasco : 100	65.9
		120	3	Apasco : 100	64.37
787	Sediments	14	6	Apasco : 100	180.57
		28	6	Apasco : 100	214
		56	6	Apasco : 100	253.8
		120	6	Apasco : 100	281.2
790	Basal	14	3	Apasco : 100	78.43
		28	3	Apasco : 100	73.63
		56	3	Apasco : 100	35.83
		120	3	Apasco : 100	37.9
791	Basal	14	6	Apasco : 100	217.83
		28	6	Apasco : 100	251.53
		56	6	Apasco : 100	287.7
		120	6	Apasco : 100	261.27
784	Sediments	14	3	Cruz Azul : 100	70.4
		28	3	Cruz Azul : 100	82.57
		56	3	Cruz Azul : 100	95.6
		120	3	Cruz Azul : 100	81.53
785	Sediments	14	6	Cruz Azul : 100	117.4
		28	6	Cruz Azul : 100	147.3
		56	6	Cruz Azul : 100	167.5
		120	6	Cruz Azul : 100	235.77
788	Basal	14	3	Cruz Azul : 100	56.2
		28	3	Cruz Azul : 100	62.57
		56	3	Cruz Azul : 100	38.13
		120	3	Cruz Azul : 100	22.47
789	Basal	14	6	Cruz Azul : 100	180
		28	6	Cruz Azul : 100	200.77
		56	6	Cruz Azul : 100	237.43
		120	6	Cruz Azul : 100	214.3

Figure 18.8 Pastefill Strength Testing Results up to 120 Days – Apasco Cement Addition at 3% and 6%

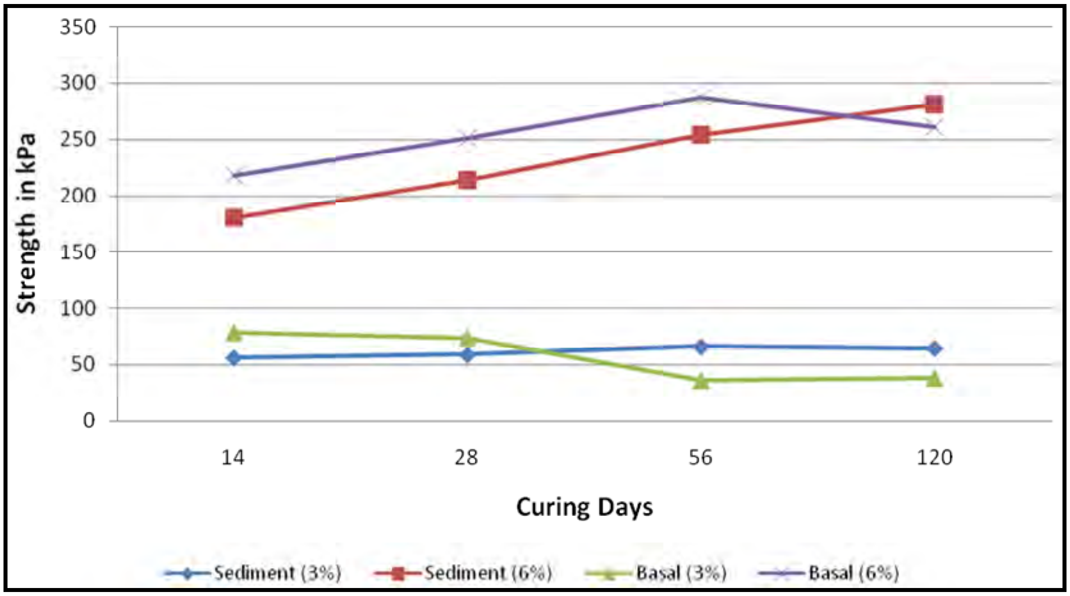
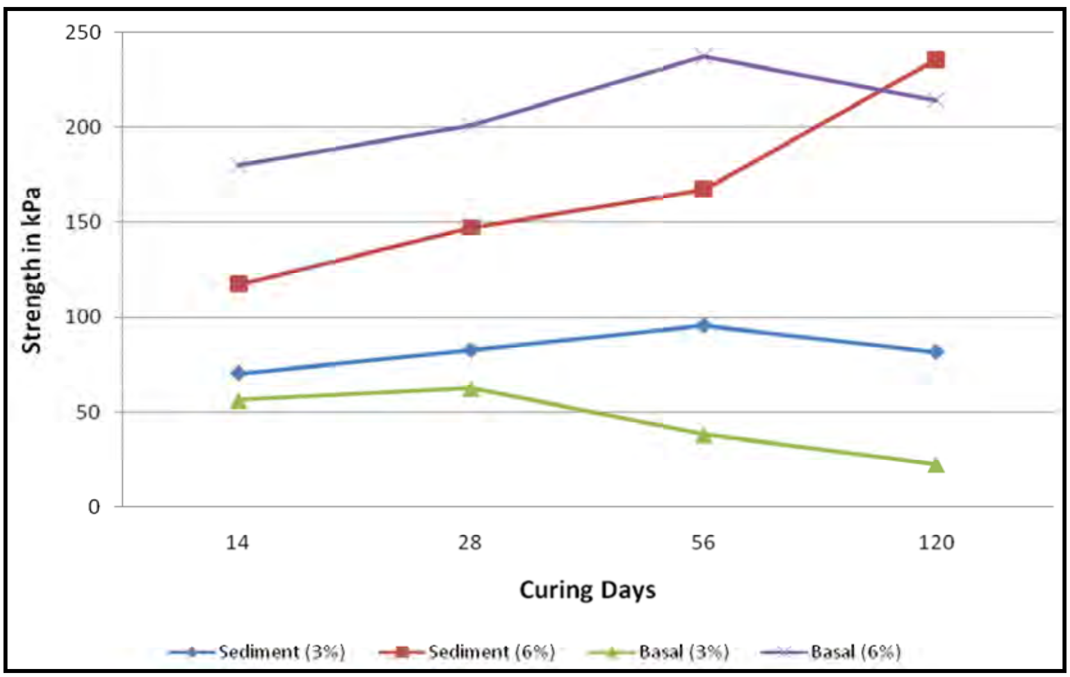


Figure 18.9 Pastefill Strength Testing Results up to 120 Days – Cruz-Azul Cement Addition at 3% and 6%



COST ESTIMATION

Pastefill Operating Cost Estimate

The pastefill operating costs at Pitarrilla are based on an average tailings consumption of 3,800 dt/d.

An average operating cost of $2.99/t of ore is estimated for 4% cement addition at 3,800 t/d of tailings delivered underground for tailings preparation (Table 18.9).

The 4% weighted average cement addition is made based on the assumption that tonnage from the mine will be 42% from room and pillar mining, and the remaining 58% from longhole stoping (Table 18.8). Out of the 58% from longhole stoping, 70% of the ore will be mined from primary stopes while the remaining will be from secondary stopes. The estimated cement consumption is:

- room and pillar at 2% cement addition
- longhole stope (primary) at 6% cement addition
- longhole stope (secondary) at 4% cement addition.

The cement cost utilized for this estimation is US$107.74/t ($1,600 Pesos/t), including transportation to site.

The operating cost estimate for 3 and 6% addition is also included in Table 18.9 for comparison. More than 71% of the total operating cost is related to cement costs.

Detailed investigation by strength testing and screening of cement with the possibility of adding ground blast furnace fly ash will assist in reducing costs and provide better strength results for tailings as constituents for backfill material.

Table 18.8 Estimated Weighted Average Cement Addition by Mining Method

Description	Tonnage Split*	Estimated Cement Addition* (%)	Sum Product (Tonnage Split x Cement Addition)
Room and Pillar* (42.0%)	42.0%	2	0.8
Long Hole Stoping*(58.0%)			
- Primary (70% of 58%)	40.6%	6	2.4
- Secondary (30% of 58%)	17.4%	4	0.7
Total	**100%**		**4.0% weighted average cement addition**

* "Split" and "% Addition" are estimated.

Table 18.9 Estimated Pastefill Operating Costs

Description	Units	Cement Addition					
		3%		4%		6%	
Ore Production	t/d	4,000		4,000		4,000	
Tailings Production for Backfill	dt/d	3,800		3,800		3,800	
Cement Costs	US$/t	107.74		107.74		107.74	
Mined Volume	m^3/d	1,455		1,455		1,455	
Cement Tonnage per Paste Vol.	t/m^3	0.04		0.06		0.09	
Cement	t/d	63.32		84.46		126.84	
Supply Costs per Tonne							
Cement	US$/t ore	1.706	71.4%	2.275	76.0%	3.416	76.6%
Cement Transport Underground	US$/t ore	0.114	4.8%	0.153	5.1%	0.235	5.3%
Pump Maintenance	US$/t ore	0.043	1.8%	0.036	1.2%	0.036	0.8%
Cement Handling	US$/t ore	0.30	12.4%	0.30	9.9%	0.30	6.6%
Maintenance	US$/t ore	0.23	9.7%	0.23	7.8%	0.23	10.7%
Total	**US$/t ore**	**2.39**	**100%**	**2.99**	**100%**	**4.21**	**100%**

18.1.8 MINE SERVICES

VENTILATION

The ventilation system designed for the Pitarrilla mine is an intake system delivering approximately 253 m^3/s. Three main intake fans and a number of auxiliary fans will control the primary ventilation circuit for full production of 4,000 t/d. The main ventilation fans will be installed on surface. During production, fresh air will be downcast through the ventilation raise and the main ramp, and up-cast through the backfill decline and conveyor drift.

The ventilation system designed for the Pitarrilla underground mine is consistent with regulations applied by the Mexican occupational health and safety standards (Mexico-Nom-023-Stps- 2003) and Canadian standards.

Primary Ventilation

Primary ventilation will be by stand-alone three vane axial intake fans located on the surface (one on the ventilation raise, two parallel on the decline).

Airflow in the mine will be controlled by ventilation regulators and doors placed appropriately for the mining taking place at any time. These will be double-door airlock-type to allow vehicle passage without interrupting mine ventilation.

As the mine develops and deepens, operating pressures and air volumes required to ventilate the mine will increase. Fan performance will be adjusted to meet these changing requirements by changing fan speeds, blade pitch, and the number of fans operating.

The mine ventilation requirements were derived from the diesel equipment list and based on the requirement of 0.06 m^3/s/kW. Air velocity is restricted on the haulage level to a minimum of 0.25 m/s and a maximum of 6 m/s (Mexico-Nom-023-Stps-2003).

Table 18.10 lists the total air volume required. The full production ventilation requirements are 253 m^3/s. A utilization factor was applied to the diesel-electric-hydraulic equipment. Ventilation losses are included at 15% of the total ventilation requirements.

Table 18.10 Ventilation Requirements at Full Production

Equipment Detail	Quantity	kW	Utilization (%)	Total kW
Development Jumbo (2 boom)	2	111	10	22
Production Jumbo (2 boom)	2	111	10	22
Longhole DTH Drill	2	74	10	15
Secondary Breaking System	1	112	10	11
Rockbolter	3	149	20	89
Cablebolter	2	111	20	44
Development LHD, 5.4 m^3	2	220	100	440
Production LHD, 7 m^3	3	354	100	1,063
Haulage Truck, 30 t	4	298	100	1,193
Grader	1	149	70	104
ANFO Loader	2	93	50	93
Explosive Truck	1	112	20	22
Mechanics Truck	1	93	30	28
Supervisor Vehicle	4	93	20	75
Electrician Vehicle - Scissor Lift	1	93	30	28
Survey Vehicle	1	93	20	19
Scissor Lift	1	111	50	56
Cassette Carrier	3	149	50	224
Fuel/Lube Truck	1	112	50	56
Forklift	1	112	50	56
Total kW				**3,660**
Air Required (m^3/s)	220			
15% Losses (m^3/s)	33			
Total Ventilation Requirements (m^3/s)	**253**			

The ventilation system design was modelled using Ventsim Mine Ventilation Simulation Software (Ventsim). This software allows input parameters including resistance, k-factor (friction factor), length, area, perimeter, and fixed quantities (volume) of air. Underground ventilation control requires several sets of ventilation control doors, regulators, and auxiliary fans to direct air quantities to the workings.

The ventilation circuit can be seen in Figure 18.10. Intake and exhaust air quantities (m^3/s) and air velocities (m/s) are presented in Figure 18.11 and Figure 18.12, respectively.

Figure 18.10 Full Production Ventilation Circuit

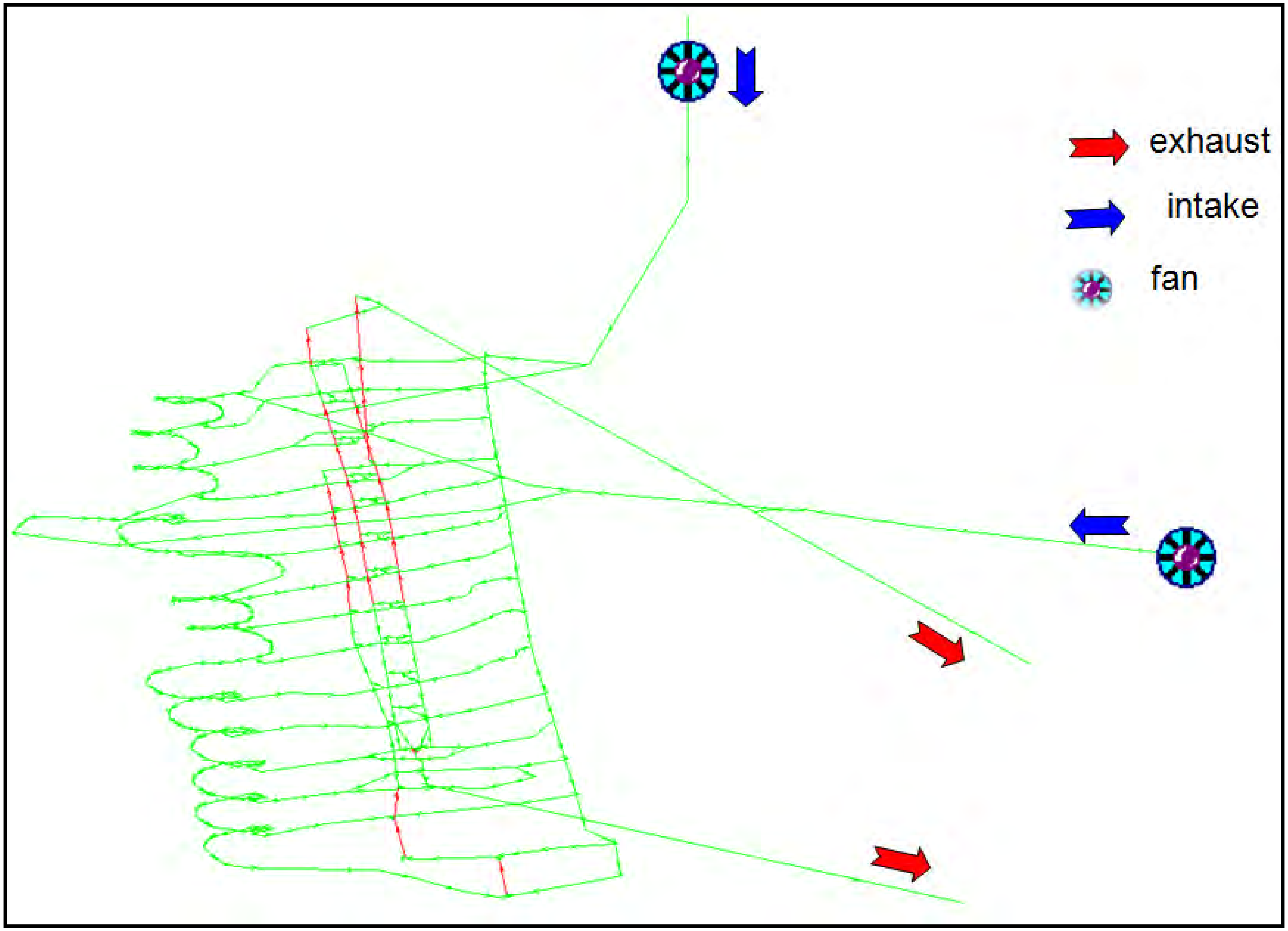


Figure 18.11 Full Production Ventilation Circuit Air Quantities (m^3/s)

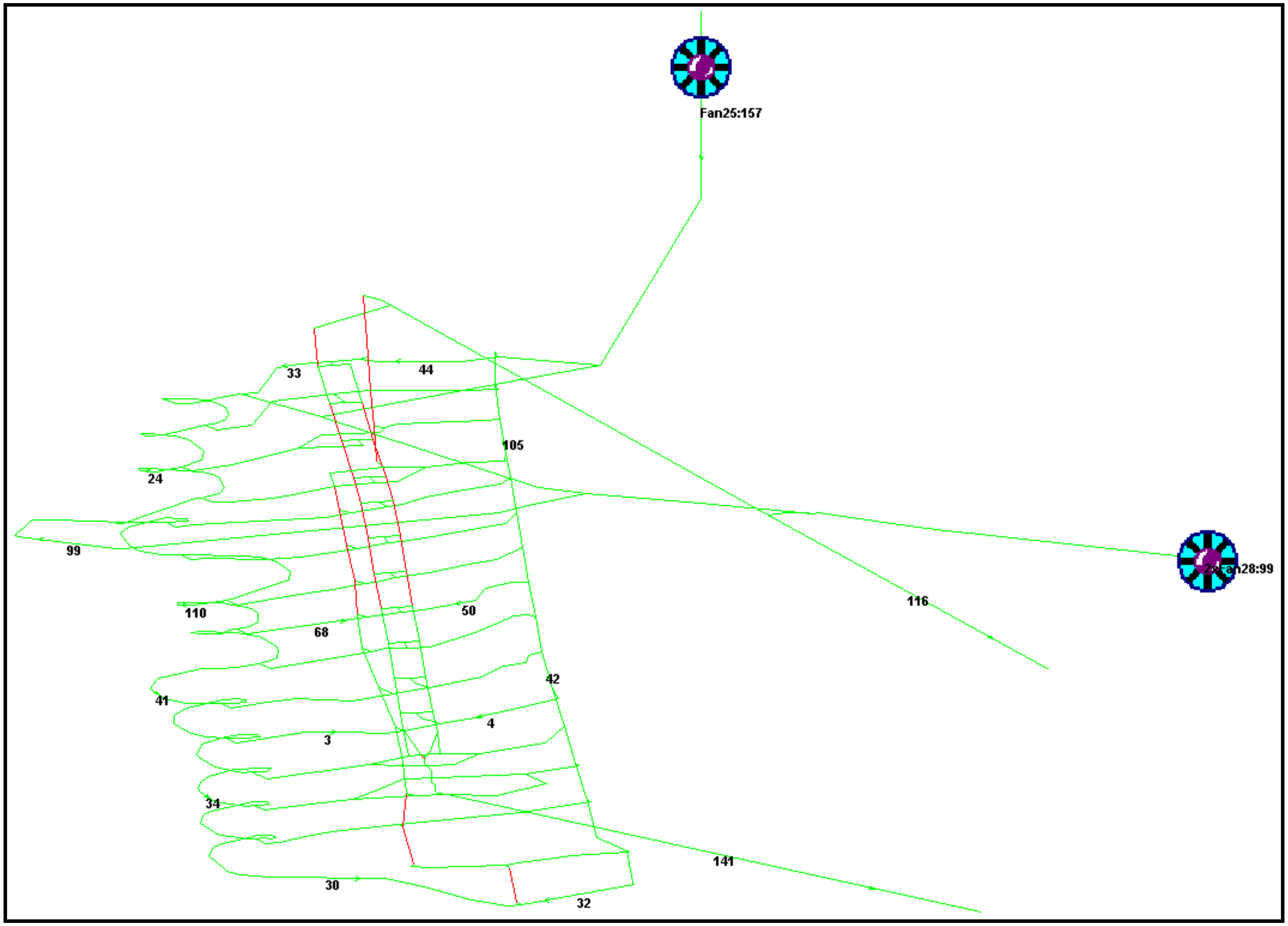
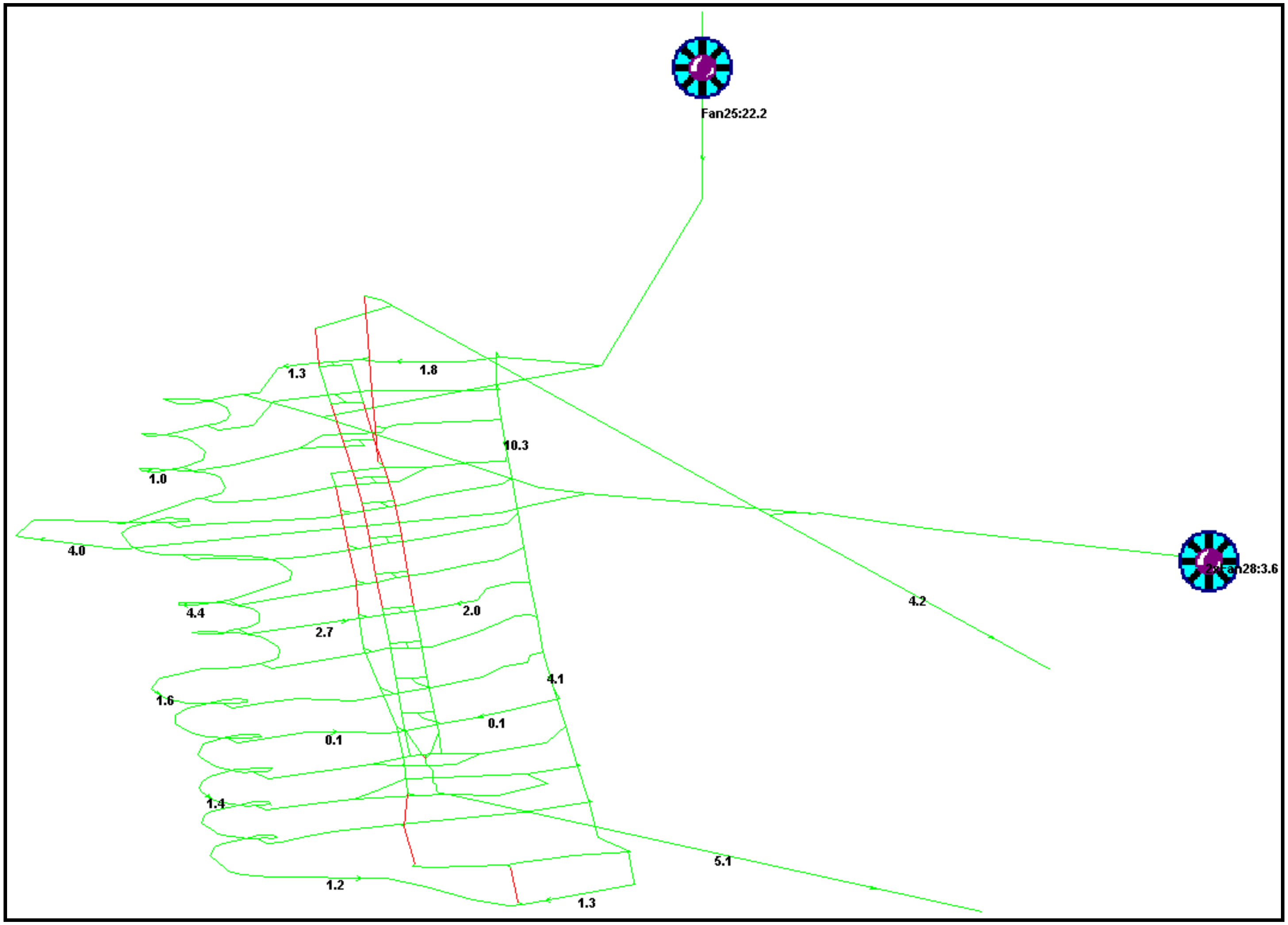


Figure 18.12 Full Production Ventilation Circuit Air Velocities (m/s)



Ventilation of Headings during Development

Auxiliary fans will maintain a 32 m^3/s airflow in the development headings. This airflow rate is required to dilute and remove exhaust from the 5 m^3 LHD, 30 t haul truck, and a double-boom jumbo. Table 18.11 provides the detailed calculations.

Table 18.11 Production Ventilation Requirements for Development

Description	Qty	Diesel (hp)	Utilization (%)	Utilized (hp)	Air Volume (m^3/s)
LHD LH 410	1	295	100	295	13
Sandvik EJC 533	1	400	100	315	18
Tamrock DD420-40C	1	149	10	15	1
Total					**32**

The development ventilation system is designed to distribute air for a 1,000 m long opening. The Alphair Ventilating Systems Inc. (Alphair) Model 4800-VAX-2100 fan (1.2 m diameter, 19° blade angle, 90 hp), or a fan with similar capacities of another manufacturer, has been selected for the auxiliary ventilation during development.

The conveyor decline development (2,000 m long) will be ventilated with two Alphair Model 4800-VAX-2100 fans (1.2 m diameter, 15° blade angle, 75 hp) in series, or fans with similar capacities of another manufacturer. The flexible duct will have a diameter of 55″.

Fan Selection

The selection of the main intake fan is based on the maximum operating duty for specific conditions. For the main fan, Wardrop assumed an average fan efficiency of 75% and 1.36 kg/m^3 air density.

Two axial flow fans with variable pitch control will be used for the permanent ventilation system (one fan at the ventilation raise and one at the main decline). The simultaneous blade adjustment provides a wide range of fan settings within a single speed; the fan volume flow rate can also be adjusted to accommodate varying conditions. Fan selection is based on the total pressure needed to deliver the required quantity of air. A possible main surface ventilation raise fan for the underground ventilation system is by Alphair with the following specifications:

- Model 8400 AMF 5000 (2.1 m diameter)
- 30° blade angle
- 1,180 rpm
- 450 hp motor.

The two possible main ramp intake parallel fans are by Alphair with the following specifications:

- Model 11200 AMF 6100 (2.8 m diameter)
- 30° blade angle
- 590 rpm
- 250 hp motor.

A possible underground booster fan for the stopes is by Alphair with the following specifications:

- Model 4800 VAX 2100 (1.2 m diameter)
- 30° blade
- 1,780 rpm
- 110 hp motor.

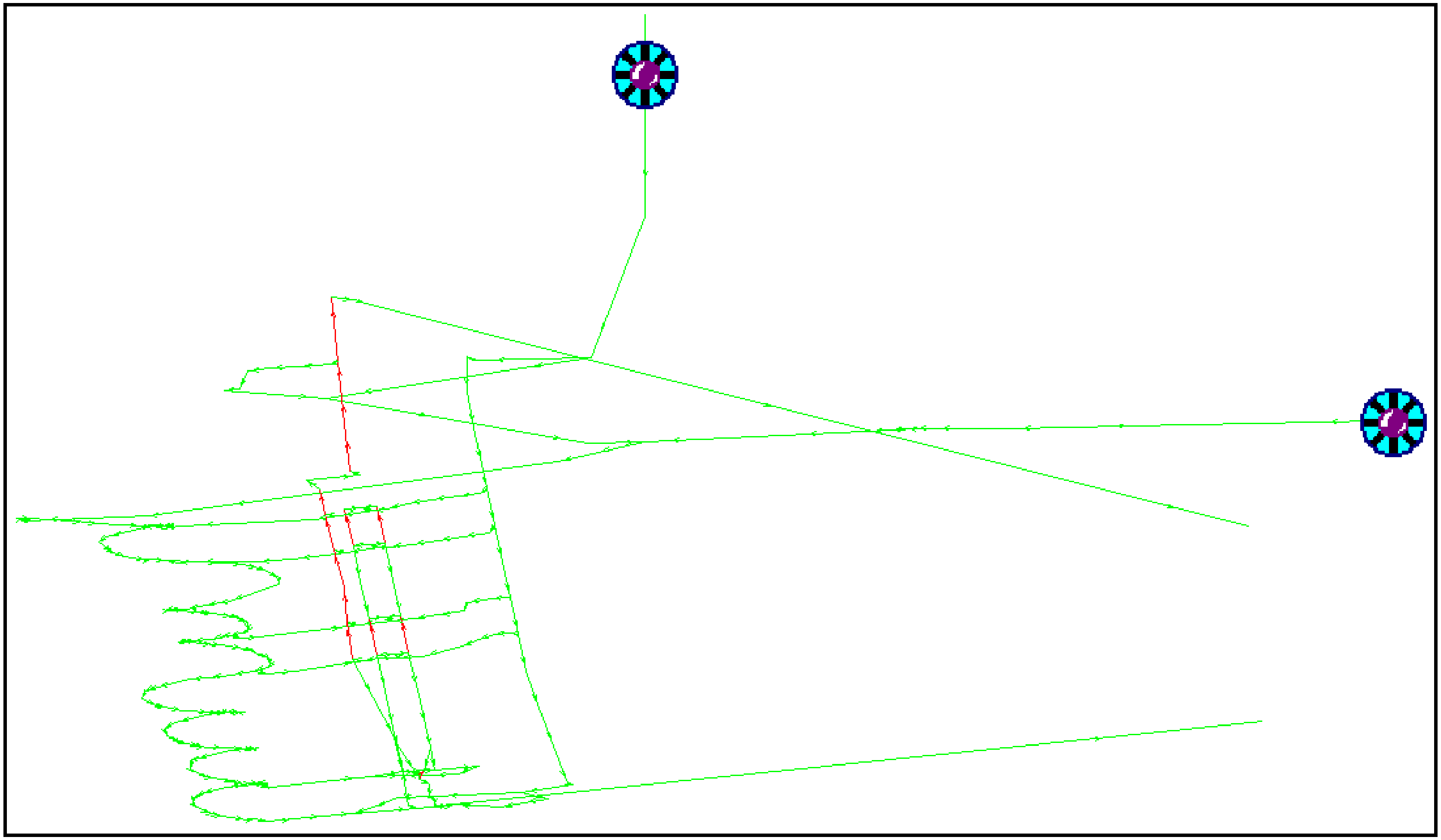
PRE-PRODUCTION DEVELOPMENT

Pre-production ventilation design is based on the following equipment list. The pre-production ventilation circuit is presented in Figure 18.13.

Table 18.12 Pre-production Equipment List

Equipment Detail	Quantity	kW	Utilization (%)	Total kW
Development Jumbo (2 boom)	4	111	10	44
Rockbolter	4	149	20	119
Development LHD, 5.4 m^3	2	220	100	440
Haulage Truck, 30 t	4	298	100	1,193
Grader	1	149	40	60
ANFO Loader	2	93	30	56
Mechanics Truck	1	93	30	28
Supervisor Vehicle	2	93	20	37
Electrician Vehicle - Scissor Lift	1	93	30	28
Survey Vehicle	1	93	30	28
Scissor Lift	1	111	50	56
Cassette Carrier	2	149	50	149
Fuel/Lube Truck	1	112	50	56
Forklift	1	112	50	56
Total kW				**2,350**
Air Required (m^3/s)	141			
15% Losses (m^3/s)	21			
Total Ventilation Requirements (m^3/s)	**162**			

Figure 18.13 Pre-production Ventilation Circuit



Two intake fans (one at the ventilation raise and one at the main decline) will be used for pre-production ventilation.

A possible main surface ventilation raise fan for pre-production ventilation system is by Alphair with the following specifications:

- Model 8400 AMF 5000 (2.1 m diameter)
- 30° blade angle
- 1,180 rpm
- 450 hp motors.

A possible main decline intake fan is by Alphair with the following specifications:

- Model 4800 VAX 2100 (1.2 m diameter)
- 25° blade angle
- 1,780 rpm
- 90 hp motors.

UNDERGROUND ELECTRICAL POWER DISTRIBUTION SYSTEM

The major electrical power consumption in the mine will be from the following:

- main and auxiliary ventilation fans
- underground backfill plant
- drilling equipment
- mine dewatering pumps
- underground maintenance shop
- underground primary crusher
- conveyor system.

High voltage cable will enter the mine via the conveyor decline and be distributed via the ventilation raise to electrical sub-stations located on each sublevel. The power cables will be suspended from the back of development headings. All equipment and cables will be fully protected to prevent electrical hazards to personnel.

High voltage power will be at 13.8 kV or 4.16 kV, and reduced to 600 kV at electrical sub-stations. All power will be three-phase.

Lighting and convenience receptacles will be single phase 127 kV power.

UNDERGROUND COMMUNICATION SYSTEM

A leaky feeder communication system will be installed as the communication system for mine and surface operations. Telephones will be located at key infrastructure locations such as the underground crusher station, underground maintenance shop, backfill plant, electrical sub-stations, refuge areas, lunchrooms, and pumping stations.

Key personnel (such as mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (such as loader, truck, and utility vehicle operators) will be supplied with an underground radio for contact with the leaky feeder network.

EXPLOSIVES AND STORAGE HANDLING

Explosives will be stored on surface in permanent magazines. Detonation supplies (NONEL and electrical caps, detonating cords, etc.) will be stored in a separate magazine on surface.

Underground powder and cap magazines will be prepared on the 1490 level. Day boxes will be used as temporary storage for daily explosive consumption.

Explosives will be transported from the surface magazines to the underground magazines in mine supply trucks.

Ammonium nitrate (AN) and fuel oil (FO) will be used as the major explosive for mine development and production. Packaged emulsion will be used as a primer and for loading lifter holes in the development headings. Smooth blasting techniques are recommended in development headings, with the use of trim powder for loading the perimeter holes.

During the pre-production period, blasting in the development headings will be done at anytime during the shift when the face is loaded and ready for blast. All personnel underground will be required to be in a designated Safe Work Area during blasting. After the pre-production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of the shift.

FUEL STORAGE AND DISTRIBUTION

The underground mobile equipment has a consumption rate of approximately 4,441 L/d during the first 3 years of production, and 5,934 L/d after additional truck haulage will be required from the stopes below crusher.

Haulage trucks, LHDs, and all auxiliary vehicles will be fuelled at fuel stations. The fuel/lube cassette will be used for the fuelling/lubing of face equipment.

COMPRESSED AIR

The mobile drilling equipment such as jumbos, rockbolters, longhole production drills, and scissor lifts with ANFO loaders will be equipped with their own compressors. No reticulated compressed air system will be required.

Two portable compressors will be required to satisfy compressed air consumption for miscellaneous underground operations such as: blasthole cleaning, jackleg and stoper drilling, secondary pumping with pneumatic pumps.

Another compressor will be installed in underground maintenance shop to satisfy miscellaneous needs.

WATER SUPPLY

Industrial-quality water will be distributed in 4″ and 2″ diameter pipelines throughout the underground workings for drilling equipment, dust suppression, and fire fighting. Flexible hoses will be used to connect water pipelines to drilling equipment at working faces.

The major water consumption will be by drilling equipment such as jumbos, longhole production drills, cablebolters with pump capacities of 100 l/min as per equipment technical specification. The rockbolters and exploration drills will require 30 l/min of water. It is assumed that each mucking LHD will require 20 l/min for dust suppression. The average estimated mine water consumption will be approximately 161,000 l/day.

A water tank located on surface near the main access decline will provide fresh and fire water. Fresh water will be obtained from a series of wells.

MINE DEWATERING

The main sources of water inflow to the underground mine will be groundwater and water from drilling operations. There is currently no information available on the required water quantities to be pumped. A hydrogeological study is required to estimate underground water inflow rates.

The main sump will typically be a two-bay design to allow suspended solids to settle out of the water before pumping. It will be located at the underground crusher station level at 1250 m elevation. Another permanent sump will be located at the bottom of the mine at 1160 m elevation.

Water will be pumped from the main sump by a high-pressure pump through a 6" diameter steel pipe located in the conveyor decline to the final tailing pump box on surface. It might require a few pumping stages. Each sump will be equipped with two high-head submersible pumps – one for operation and one on standby.

TRANSPORTATION OF PERSONNEL AND MATERIALS UNDERGROUND

Supplies and personnel will access the underground via the main access decline.

Three Maclean CS-3 Carriers with personnel cassettes will be used to shuttle men from surface to the underground workings and back during shift changes. Supervisors, engineers, geologists, and surveyors will use diesel-powered Toyota trucks as transportation underground. Mechanics and electricians will use the mechanics' truck and maintenance service vehicles.

A Maclean CS-3 boom deck with a Hiab 095 10-t crane will be used to move supplies, drill parts, and other consumables from surface to active underground workings.

UNDERGROUND CONSTRUCTION AND MINE MAINTENANCE

A mine service crew will perform the following:

- mine maintenance and construction work
- ground support control and scaling
- road checking and maintenance
- construction of ventilation doors, bulkheads, and concrete work
- mine dewatering
- safety work.

An underground grader and scissor lift will be utilized to maintain the main declines and active work areas.

EQUIPMENT MAINTENANCE

Mobile underground equipment will be maintained in an underground mechanical shop located on the 1490 level. The maintenance shop will contain the following:

- five maintenance bays (two heavy repair crane bays and three service bays) equipped with overhead and jib cranes
- tire repair bay
- welding bay
- electrical bay
- lube and wash bay
- warehouse
- office.

A 50 m long and 2.5 m wide slash of the maintenance shop access will provide a parking area for mobile equipment.

A maintenance supervisor will provide a daily maintenance work schedule, ensure the availability of spare parts and supplies, and provide management and supervision to maintenance crews. He will also provide training for the maintenance workforce.

A maintenance planner will schedule maintenance and repair work, as well as provide statistics of equipment availability, utilization and life cycle, mine efficiency, and personnel utilization. A computerized maintenance system will facilitate planning.

The equipment operators will provide equipment inspection at the beginning of the shift and perform small maintenance and repairs as required.

A mechanics truck will be used to perform emergency repairs underground.

Major rebuild work will be conducted off site.

MINE SAFETY

Fire Prevention

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the underground crusher station, electrical installations, pump stations, conveyors, service garages, fuelling stations, and wherever a fire hazard exists.

A suitable number of fire extinguishers will be provided and maintained at each stationary diesel motor, transformer substation, and any splitter panel. Every vehicle will carry at least one fire extinguisher of adequate size and proper type.

Underground mobile vehicles and conveyor belts will be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system will be installed at the main intake mine entries to alert underground workers in the event of an emergency.

Mine Rescue

A mine rescue Emergency Response Plan will be developed, kept up to date, and followed in an emergency. Two fully trained and equipped mine rescue teams will be established.

A mine rescue room will be provided in the administration building. A trailer with mine rescue equipment and a foam generator will be located on site. The mine rescue teams will be trained for surface and underground emergencies.

Refuge Station

The portable refuge stations will be provided in the main underground work areas. The refuge stations will be equipped with compressed air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line, compressed air, and emergency lighting. The stations will be capable of being sealed to prevent the entry of gases. A plan of the underground workings showing all exits and the ventilation plan will be provided.

The refuge station locations will move as the working areas advance, eliminating the need to build new refuge stations.

Emergency Egress

The main access and conveyor declines will provide primary access and auxiliary exits. The ventilation raise will have a dedicated manway for personnel to travel between levels in case of emergency, providing the secondary exit.

Dust Control

Broken ore will be wet down after blasting and during production using sprays on the extraction level.

18.1.9 MINE EQUIPMENT

UNDERGROUND EQUIPMENT

Table 18.13 lists underground mobile equipment by type and quantity.

Table 18.13 Underground Mobile Equipment List

Equipment	Type	Quantity
Drilling Equipment		
Development Jumbo (2 boom)	Tamrock DD320-26C	2
Production Jumbo (2 boom)	Tamrock DD320-26C	2
Rockbolter	Tamrock DS 310	3
Cablebolter	Tamrock DS 420	2
Longhole Drill	Tamrock DL310-7	3
Secondary Breaking System	Maclean SB-6 Blockholer	1
Exploration Drill	Diamec 252/1600U4PHC	by contractor

table continues…

Equipment	Type	Quantity
Loading & Hauling Equipment		
Waste Development LHD, 10 t	TORO LH410	2
Ore Dev/Production LHD, 17 t	TORO LH517	3
Haulage Truck, 30 t - Waste Development	EJC 533	2
Haulage Truck, 30 t - Ore below Crusher	EJC 533	2
Service Vehicles		
Grader	GR 12 H	1
Explosive Truck		1
ANFO Loader	Toyota HZJ79	2
Mechanics Truck	Maclean MT-3	1
Supervisor/Engineering Vehicle	Toyota HZJ79	4
Electrician Vehicle - Scissor Lift	Toyota HZJ79	1
Scissor Lift	Maclean SL-3	1
Cassette Carrier	Maclean CS-3 Carrier	3
- Flat Deck Cassette	Maclean CS-3 Flat Deck	2
- Hiab 095 Boom/Deck Cassette	Maclean CS-3 Boom Deck	1
- Fuel/Lube Cassette	Maclean CS-3 Fuel - Lube	1
- Fuel Tank Cassette	Maclean CS-3 Fuel Tank	1
- Personnel Cassette	Maclean CS-3 Personnel	3
Shotcrete Robo	Mamba	1
Transmixer	Getman	2
Forklift	Toyota	2

18.1.10 PERSONNEL

The mining employees at the Pitarrilla underground operation are divided into two categories: salaried personnel, and hourly labour.

The personnel requirement estimates are based on the following:

- a 4,000 t/d production rate
- the production schedule
- a crew rotation of two 10-h shifts per day
- two crews working on site for four days and one crew off for two days.

A mining contractor will begin work in the pre-production development stage of the mine life to allow time for the Owner to recruit staff for the project. The labour and personnel requirements described in this section do not include pre-production development, which will be performed by the contractor.

Underground staffing requirements peak at 224 personnel during full production.

Table 18.14 Technical and Supervisory Staff

Staff Mine Operation	Quantity
Mine Operating Staff	30
Mine Maintenance Staff	9
Total Mining Staff	**39**

Hourly personnel were estimated based on production and development rates, operation productivities, and maintenance requirements. Personnel productivities were estimated for all main activities by developing cycle times for each operation.

Table 18.15 Hourly Labour

Labour Description	Personnel per Shift	Personnel per Day	Total Payroll
Hourly Mine Labour			
Production	11	22	33
Development	10	20	30
Haulage and Ore Handling	12	24	37
Mine Services & Safety	10	20	30
Mine Maintenance	18	36	55
Total Mine Operating	**61**	**122**	**185**

18.1.11 Underground Mining Capital Cost

The capital cost estimate was prepared at a pre-feasibility study level, which is considered a ±25% level of accuracy, based on the following:

- basic equipment list
- budget quotes obtained from equipment manufacturers and development contractors
- in-house database
- Western Mining estimation references
- preliminary project development plan.

Mining capital is divided into equipment capital cost and mine development cost categories.

The mining equipment capital cost estimate includes the purchase of permanent mining equipment required for production. It is estimated that underground production will start in Year 3, when all pre-production development will be completed by the contractor and all mine services will be established.

It is assumed that all pre-production development will be done by a contractor. The pre-production development cost is based on required amount of underground development, contractor rates per metre of development of each type of heading, and included contractor mobilization to the mine site.

The underground mining capital cost is estimated to be US$74.9 million as indicated in Table 18.16.

Table 18.16 Underground Mining Capital Cost

Cost Distribution	US$
Drilling Equipment	9,688,335
Loading and Hauling Equipment	5,899,035
Service Vehicles	4,216,543
Underground Stationary Equipment	5,458,072
Mine Engineering and Safety equipment	1,149,720
Spare Parts (5% of Equipment Capital)	1,320,585
Sub-total Underground Mining Equipment	**27,732,290**
Underground Pre-production Development in Year -1	21,417,700
Underground Pre-production Development in Year -2	25,747,773
Sub-total Underground Pre-production Development	**47,165,474**
Total Underground Mining Capital Cost	**$74,897,764**

18.1.12 Underground Mining Operating Costs

The mining operating cost required for a 4,000 t/d operating mine was estimated for each cost category such as development, production, haulage, maintenance, mine services, and labour.

Productivities, equipment operating hours, labour, and supply requirements (including drill and steel supplies, explosives, ground support, and services supplies) were estimated for each type of underground operation.

Total hourly labour requirements were estimated to achieve the daily mining production rate based on 2 shifts at 10 h/d with 3 crews. An 83.3% hourly efficiency (50 minutes per working hour) and 1.5 hours of non-productive time per shift were applied to estimate an effective work time per shift of 7.1 hours per 10-hour shift.

The stoping cost were estimated based on an average typical stope size, and the haulage cost – for an average haulage distance.

The mine exploration cost was estimated based on the assumed amount of exploration and delineation drilling required, and contractor drilling rates.

Total underground mining operating cost is shown Table 18.17.

Table 18.17 Underground Mining Operating Cost

Cost Distribution	US$/t
Longhole Stope Production	2.83
Longhole Stope Development (Ore)	3.18
Longhole Stope Access Development (Waste Crosscuts)	0.72
Sub-total Longhole Stoping Cost (58% of total production)	**6.73**
Room & Pillar Stoping Cost (42% of total production)	**9.18**
Average Stoping Cost	**7.75**
Truck Haulage from Below Crusher (16% of total production)	1.27
Exploration	0.27
Services	1.09
Maintenance	0.22
Mine Safety, Training, Mine Rescue	0.12
Miscellaneous (5%)	0.15
Sub-total Mine Services Cost	**1.85**
Labour	2.05
Labour Maintenance	1.20
Salaried Personnel	1.51
Sub-total Mine Labour	**4.75**
Total Average Mining Operating Cost per Tonne	**$14.55**

18.2 PROCESS PLANT

18.2.1 MILL SERVICES

The mill services will consist of water distribution and building services.

WATER DISTRIBUTION

Process Water System

Details on the process water system can be found in Section 16.2.5 of this report.

Fresh Water Systems

Fresh water is distributed by a pump for start-up and emergency purposes, gland seal water, reagents, flotation cleaning stages, process water make up, and potable water. Details on the process water system can be found in Section 16.2.5 of this report.

Potable Water System

Distributed fresh water is collected in a potable water storage tank. The water is pumped from this tank through a hydro chlorinator and into a distribution piping ring to serve all the potable water users in all facilities. The main users of potable water include the plant workshop, the administration building, washrooms, and safety showers in the processing area.

BUILDING SERVICES

The service areas of the truck shop will be provided with ventilation systems with exhaust fans.

The assay laboratory will be provided with dust collection systems in the sample preparation area as well as local laboratory exhaust systems.

The truck wash will be provided with an exhaust fan.

Dust Control

Dust control will be provided with aspirated systems, with dust hoods and ducting being connected to dry dust collectors at the following locations:

- primary crushing (below crusher)
- stockpile reclaim area
- secondary crushing
- concentrate load out.

Water sprays will be utilized at the truck dump pocket and at the feed to the coarse ore stockpile.

Fire Protection

The plant site will be provided with a fire protection system as follows:

- A fire water tank will be located above the processing plant facilities. The tank will hold a dedicated quantity of water for fighting fires.
- Firewater will be piped to all main facilities by gravity.
- In addition, all buildings will be equipped with hand held fire extinguishers of two types. General purpose extinguishers for inside plant areas and dry-type extinguishers for inside electrical and control rooms.

- Conveyors:
  - crushed ore conveyor – portable fire extinguishers and access by fire truck along length of conveyor.
- Truck Shop and Warehouse/Tire Change and Truck Wash:
  - wet Class 2 standpipe hose system will be provided throughout the building
- Administration/Dry:
  - the IT room will be protected by an automotive clean agent suppression system.
- Assay Laboratory:
  - automatic wet sprinklers will be provided throughout the building.

18.2.2 INSTRUMENTATION AND PROCESS CONTROL

OVERVIEW

The plant control system will consist of a Distributed Control System (DCS) with PC based Operator Interface Stations (OIS). The DCS, in conjunction with the OIS, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation

PRIMARY CRUSHER

A control room in the primary crushing building will be provided with a single OIS. Control and monitoring of all primary crushing, conveying, and stockpiling operations will be conducted from this location.

MILL

A central control room in the mill building will be provided with three OIS. Control and monitoring of all processes in the mill building will be conducted from this location.

All the plant interlocking, monitoring and control functions are implemented within the DCS. This will comprise graphical operator workstations in the control room, windows based hardware and software application, and a system of manufactured I/O cabinets to allow the control system to interface with the motor starter, valves, and associated equipment that perform the actual plant control.

18.3 SURFACE FACILITIES

18.3.1 INTRODUCTION

Surface facilities, infrastructure, and ancillary facilities will comprise the following and are designed to conform to locally available materials and methods of construction:

- process plant
- administration building
- maintenance/warehouse
- camp accommodations
- mine dry
- open storage area
- fuel storage and distribution
- power supply and distribution
- communications system
- truck wash
- sewage system
- access road
- water supply.

18.3.2 SURFACE FACILITIES

PROCESS PLANT

The mill site will be located approximately 2,000 m east of the underground crushing chamber. The proposed project site will accommodate the following:

- crushing circuit
- grinding
- flotation
- thickening
- filtration
- tailings disposal
- reagents area
- water supply and distribution.

The locations of the main project facilities are shown on the overall site plan (Figure 18.14).

The infrastructure has been designed to conform to locally available materials and methods of construction. The 7 m wide site access road is designed to accept semi-trailer truck traffic with a tandem-axle loading of 20 t and will be adequate for transportation of material supplies.

Careful attention was given to the placement of the facilities in order to minimize the overall footprint and required excavation. The layout as a whole takes advantage of the natural slope in the area.

The crushing circuit will store material in the course ore stockpile. Dust will be contained by using fogging spray nozzles along the belt conveyors running from the jaw crusher to the coarse ore stockpile.

The mill building will house the grinding, flotation, thickening, filter presses, and reagents. The building will be open-walled to allow for ventilation, while the roof will protect the inside from sunlight and rain. This roofed-structure will rise approximately 50 m above grade at its highest point. Vehicle access points will allow delivery of reagents as well as pickup of lead and zinc concentrates.

The mill building layout also takes advantage of gravity as the ore will be processed following a top to bottom path, eliminating where possible the necessity for pumps. All equipment within the building will be accessible either from the floor or the stair-connected platform system. Bridge mounted cranes will access all areas of the building.

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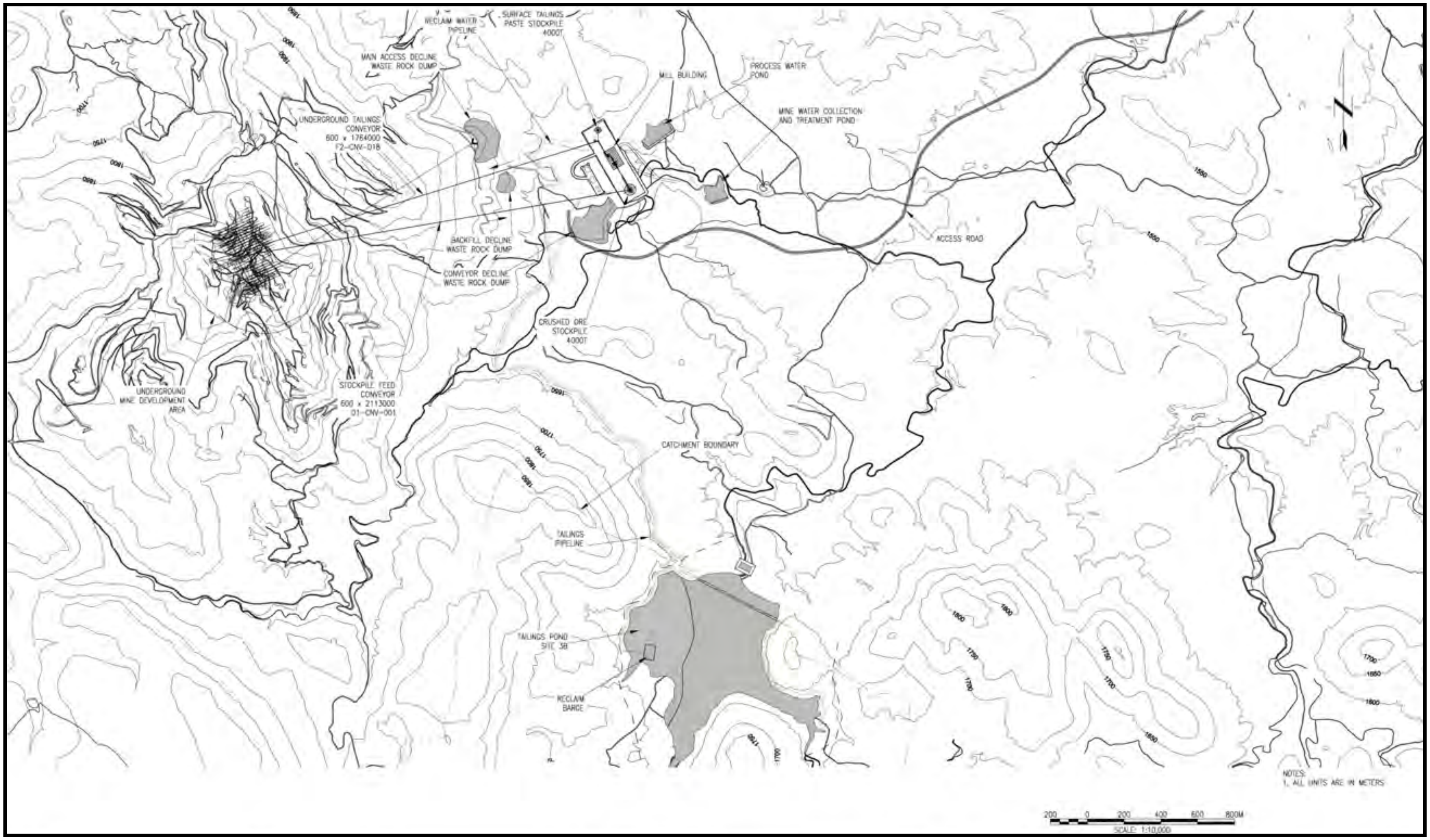
SILVER STANDARD

Figure 18.14 Overall Site General Arrangement Plan



Administration Building

The administration buildings will provide a working space for management, supervision, geology, engineering, and other operations support staff.

Maintenance/Warehouse

A Maintenance/Warehouse facility is provided to service the mobile equipment and for storage of equipment spares. The maintenance area includes an overhead crane and is fully equipped including lube racks, washer system, etc.

Propane Storage

An open area with a slab on grade has been provided to accommodate propane storage.

Open Area Storage

A fenced-off open storage area is provided for equipment and materials that can be stored outside.

Truck Wash

Trucks will be washed in an open area, complete with slab on grade and plastic side panel framing. It comprises a truck wash machine and oil water separator.

Mine Dry

A separate mine dry facility is provided, which includes lockers and shower facilities.

Canteen

Canteen facilities include cooking and dining facilities.

Assay Laboratory

A small facility has been included to house the laboratory facilities.

Camp Accommodations

The construction camp and bunkhouses will be upgraded to accommodate 450 operations personnel.

FUEL STORAGE AND DISTRIBUTION

Fuel will be transported to site and stored at a Petroleos Mexicanos (PEMEX) owned and operated station located adjacent to the maintenance facility. The pumping station allows for vehicle refuelling.

POWER SUPPLY AND DISTRIBUTION

Power to the mine site will be via a 115 kV overhead line from the Santa Maria del Oro substation located about 80 km away. A new step-down substation located near the plant site will reduce the voltage to 13.8 kV, which will be used as the main distribution voltage level for the plant. The power line and step down system is based on information available from the area electrical utility (Comisión Federal de Electricidad).

The mine's incoming substation includes the utility equipment (primary disconnecting means and main transformer) and a secondary 15 kV switchgear line-up, which provides the means to selectively distribute power to the various plant areas.

The power distribution method to load areas will be either by overhead power line for the more distant loads or by a 15 kV cable system for the loads that are relatively close by.

Each plant area, where loads are logically grouped, will be provided with an electrical room where the 13.8 kV distribution voltage will be stepped down to the process level distribution voltages of 4.16 kV and 480 V. The plant power system design includes 4.16 kV switchgear (breakers and starters) for distribution and large motors, with 480 V MCCs provided for smaller motor loads.

Local emergency generators are included to provide standby power to equipment that is designated as critical. A Critical Process MCC is provided in each electrical room (where critical loads are identified) and connected to the area's stand-alone generator so that, should utility power fail, the critical equipment can be restarted after the generator start-up time.

COMMUNICATIONS SYSTEMS

Communication System

The site communications systems will be supplied as a design build package. The base system will be installed during the construction period then expanded to encompass the operating plant.

Various parts of the operation will be connected together through a site telephone system. Two-way radios will be used for communication between supervisors, mobile equipment operators, crusher operators, and conveyor operators.

Fibre Optic Network

A fibre optic network will be installed around the site to facilitate plant control system and communication between process areas. Generally, the network routing will follow that of the site power distribution.

Plant Control System

A programmable logic controller (PLC)-based system will be provided for monitoring and control of the entire plant. PLC input/output (I/O) cabinets located in electrical rooms throughout the plant will be used to interface with all field instrumentation, equipment, and motor controls. Starting and stopping of equipment and set-point changes will be done via PC-based operator workstations located in at the primary crushing control room and the mill building control room.

WASTE DISPOSAL

Solid Waste Disposal

Hydrocarbon wastes such as used oil and lubricants, oil filters, rags, and hydrocarbon-contaminated soil are expected to make up the major proportion of hazardous wastes generated during project life. Used oil and lubricants will be sent to an authorized recycling facility, and oil filters and rags will be incinerated by an authorized facility. Soil remediation facilities will be constructed on site to manage oil-contaminated soils and sludge from wash pad facilities.

Other hazardous wastes such as batteries, certain lamps, and pathological wastes are expected to be generated in small quantities, and will be sent to an authorized facility for disposal.

Recyclable wastes including scrap metal, glass, wood, paper, and plastics will be reused within the project to the extent possible, and the remainder sorted and then transferred to external recycling facilities.

Sewage

The sewage facility comprises a series of tile septic fields.

SITE ACCESS ROAD

Road access is good and easily accessible all year round with paved highways extending to within 20 km (12 miles) of the center of the property. Most large items, assay labs, and air transportation (Torréon International Airport) are available in the city of Torréon. The required driving time from the city of Torréon is about four hours traveling along all-weather roads from San Francisco de Asis.

Detailed information on the access route is available in the Delcan International Corp. report entitled "Pitarrilla, Durango State, Mexico – Access Road Scoping Study, Phase 2" dated September 2008, which is provided in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices".

WATER SUPPLY – POTABLE AND SAFETY

Water Supply

The current operation uses three sources of water for operations: surface water reservoirs, groundwater wells, and pumping from the nearby Lazaro Cardenas reservoir, or a combination of these. Based on average conditions, the required fresh and make-up water for the plant operations can be provided from just two of the three water supply reservoirs. However, a third reservoir has been included to provide a contingency for dry years.

Details on the existing water supply system from the river can be found in the report by Knight Piésold entitled "Silver Standard Resources Inc. La Pitarrilla Project, 4,000t/d Pre-feasibility Study, Tailings, Waste Rock and Water Management" dated May 27, 2009, which is provided in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices"

18.4 TAILINGS, WASTE ROCK, AND WATER MANAGEMENT

18.4.1 *DESIGN BASIS*

The design criteria for tailings, waste rock, and water management are based on the proposed mine development plan, the preliminary hydrometeorological data, site reconnaissance and investigations, limited laboratory testing, and assumptions where no data are available. The data available at this early stage of the project are limited in some areas.

Key aspects of the design basis are:

- An underground mine producing 4,000 t/d (about 1.4 Mt/a) over a 12-year mine life. During this time, the underground mine will yield approximately 17 Mt of ore.
- Approximately 40% of the tailings will be used to produce paste backfill to support the underground mine openings. The remainder of the tailings will report to a surface Tailings Storage Facility (TSF) as a dilute slurry of approximately 54% solids by weight.
- A total of 10 Mt of tailings will be stored in the TSF at an assumed dry density of 1.5 t/m^3.

- Waste rock from pre-production mining will be stored in three surface dumps in the vicinity of the mine portals.
- Waste rock produced after the mine is operational will generally be used as backfill in the underground mine openings. Waste rock that is not backfilled to the mine will be managed in surface dumps.
- Any potential acid generating waste rock will be hauled to the TSF where it will be stored.
- The region is relatively arid with an estimated mean annual rainfall of 450 mm and evaporation of 1,840 mm.

A preliminary dam classification has been carried out for the TSF to enable appropriate design earthquake and flood events to be assigned. The selection of the design earthquake and flood events is based on the classification criteria provided by the Canadian Dam Association's (CDA) "Dam Safety Guidelines" (2007). The TSF has been assigned a SIGNIFICANT hazard classification for the purpose of this study.

The Maximum Design Earthquake (MDE) for the TSF has been selected based on the SIGNIFICANT dam hazard classification defined for the facility and the criteria for design earthquakes provided by the CDA "Dam Safety Guidelines" (2007). The CDA Guidelines require that a SIGNIFICANT dam classification be designed for a probabilistically derived event (known as the Earthquake Design Ground Motion) having an Annual Exceedance Probability (AEP) of 1/1,000. Consequently, the MDE selected for the TSF is the 1-in-1,000 year earthquake. The corresponding maximum acceleration is 0.05 g for the 1-in-1,000 year earthquake. Limited deformation of the confining embankment dam is acceptable under seismic loading from the MDE, provided that the overall stability and integrity of the facility is maintained and that there is no release of stored tailings (ICOLD, 1995).

The Inflow Design Flood (IDF), based on CDA guidelines, for TSF Site 3B is between the 1/100 year flood and 1/1000 year flood.

18.4.2 *Tailings Storage Facility*

Design and Construction

Previous studies identified a number of potential sites for a TSF to support a 10,000 t/d operation. The site selection was reassessed following the reduction in mine throughput to 4,000 t/d. The preferred location for the 4,000 t/d Pre-feasibility Study TSF is Site 3B based on land access constraints, and estimated initial capital and operating costs.

The impoundment shown on Figure 18.15 is located approximately 2.5 km south of the plant site and incorporates a cross valley embankment. A more detailed TSF plan and section is shown on Figure 18.16.

Figure 18.15 Site Layout

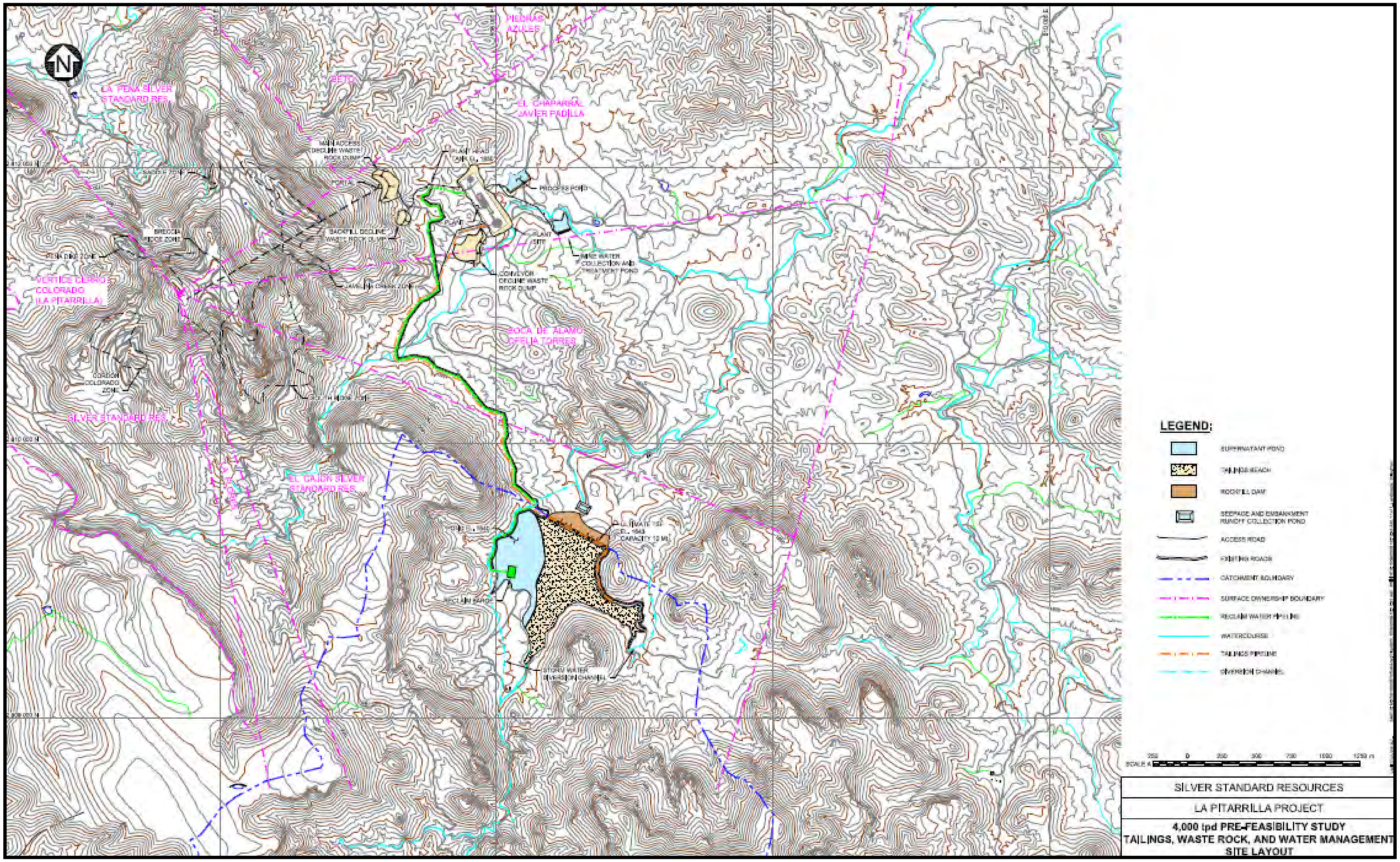
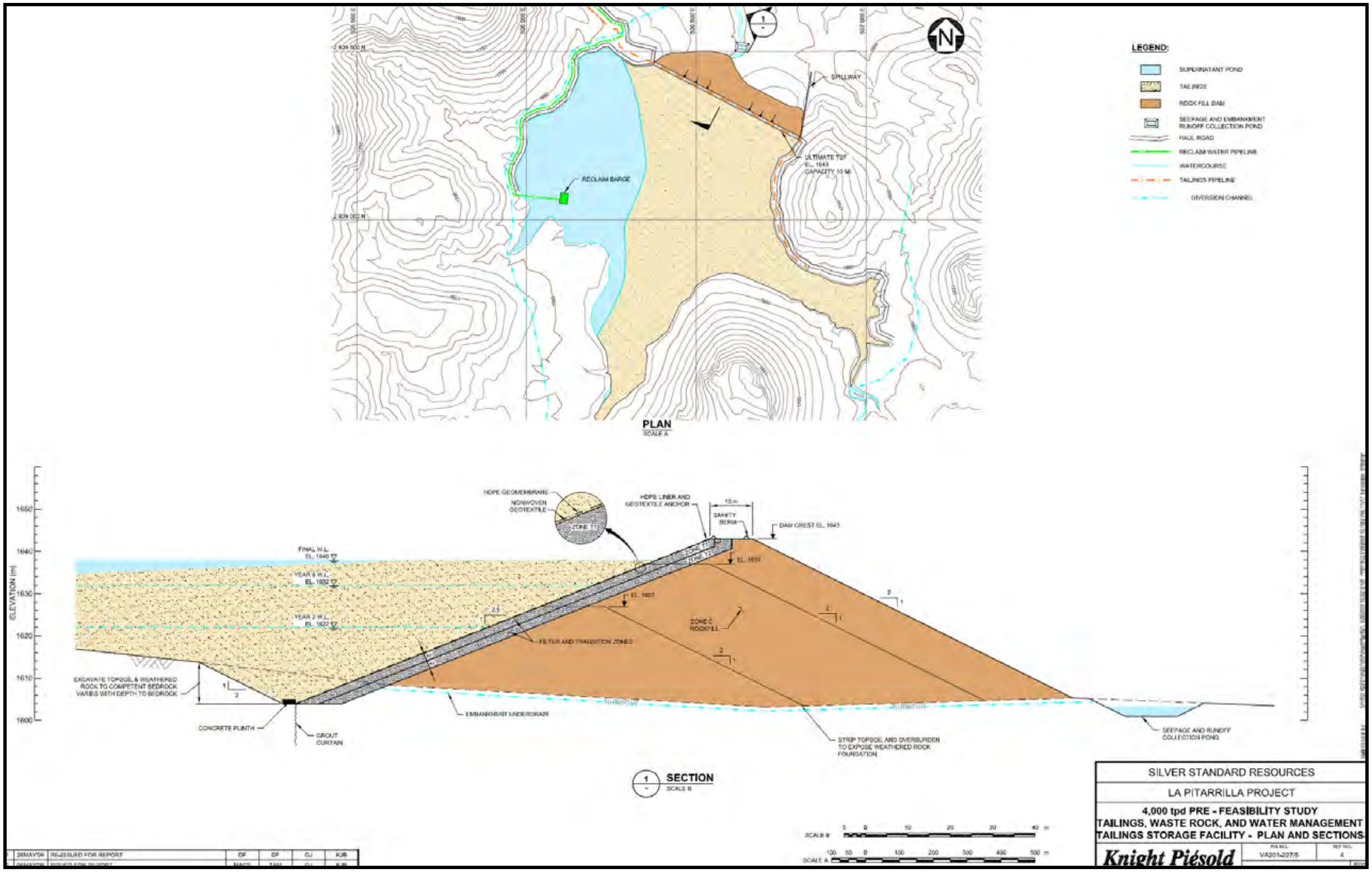


Figure 18.16 Tailings Storage Facility – Plan and Sections



Initial preparation of the TSF basin will comprise clearing and grubbing. The embankment foundation will be prepared by stripping topsoil and the shallow overburden soils to expose a weathered rock foundation. The topsoil will be stockpiled for future reclamation activities.

A trench will be excavated to competent (fresh to slightly weathered) bedrock along the upstream toe of the embankment. This will require removal of approximately 5 m of fine to coarse grained sand that was encountered on the west abutment of the proposed dam location at Site 3B. A concrete plinth will be constructed along the length of the trench, with shallow localized grouting applied if and where required.

Drains will be placed in trenches excavated in original ground at topographic low points within the embankment footprint. The drains will assist with foundation dewatering during construction and function as embankment underdrains during operation. Seepage collected in these drains will report to a collection pond located at a topographic low point downstream of the TSF embankment.

The TSF embankment will be built in three stages over the mine life using the downstream method of construction, as shown on Figure 18.16, with an upstream slope of 2.5H:1V and a downstream slope of 2H:1V. The stages correspond with the tailings elevation at Years 2, 6, and 10, and include a freeboard allowance for storm storage and wave run-up. The main embankment shell will be constructed of rockfill sourced from local quarries or borrow areas. The upstream face of the dam will be lined with a high density polyethylene (HDPE) geomembrane liner. The HDPE liner will be anchored to the concrete plinth at the upstream toe of the dam, and at the crest of each raise, forming a positive hydraulic cut-off along the full length of the dam. The HDPE liner will be placed over a non-woven geotextile and bedding layers comprising filter and transition zone materials as shown on Figure 18.16. This arrangement ensures that the dam can handle seepage flows even if the geomembrane liner is damaged.

Surface water diversion channels design to carry the 1-in-50 year storm runoff will be constructed upstream of the TSF. These diversions will include stop logs at the intake structures to assist in managing the supernatant ponds elevation.

An overflow spillway may be required during operations or at closure to manage the inflow design flood. Future design should include a detailed hydrological analysis of the facility.

TAILINGS STORAGE FACILITY OPERATION

Slurry tailings will be pumped from the plant site to the TSF through an approximately 5 km-long steel and HDPE pipeline. The pipeline will be placed in a lined ditch to provide secondary containment. Tailings will be discharged from offtakes along the dam crest and selected points along the northeast perimeter of the impoundment. The tailings discharge points will be rotated as required to form a gently sloping beach that isolates the supernatant pond from the embankment.

A floating pump station, or reclaim barge, will return supernatant water to the plant site head tank through a HDPE pipeline. The barge will include a covered enclosure, a motor control center, and power transformers. Additional requirements include barge anchoring to cope with windy conditions. Pipeline connections from the barge to the shore will incorporate flexible joints to accommodate changes in pond elevation. Operation of the pumps will be controlled by water levels in the process circuit or in the holding tank above the process plant.

Seepage from the underdrain system and runoff from the embankment face will report to the collection pond constructed downstream of the embankment. The collected water will be monitored and pumped back into the TSF as required.

Geotechnical instrumentation will be installed in the TSF embankment and its foundation. This instrumentation will be monitored during the construction and operation of the facility to assess performance and to identify any conditions different to those assumed during design.

Tailings Storage Facility Reclamation and Closure

The TSF will be required to maintain long term physical and geochemical stability, protect the downstream environment, and passively manage surface water. Upon mine closure, surface facilities will be removed in stages and full reclamation of the TSF will be initiated. General aspects of the closure plan include:

- compaction and grading of the exposed tailings surface (beach) to ensure acceptable surface water runoff and control
- maintenance of a small surface water pond within the TSF to attenuate large runoff events
- construction of a closure spillway to allow safe discharge of storm water
- capping of the tailings surface with inert waste rock or locally quarried material followed by a layer of topsoil, stockpiled from stripping activities
- revegetation of the exposed embankment faces
- dismantling and removal of the pipelines, structures, and equipment not required beyond mine closure
- removal of the seepage monitoring system when water quality is shown to be acceptable
- removal and re-grading of all access roads, ponds, ditches, and borrow areas
- long term stabilization of all exposed erodible materials.

18.4.3 WASTE ROCK MANAGEMENT

The approximately 810,000 tonnes of waste rock produced during the pre-production will be managed at three surface waste dumps near the portals shown on Figure 18.14. For the purpose of this study, it is assumed that 25% of the pre-production waste rock will be potentially acid generating (PAG). All PAG waste from the pre-production period will be hauled to the TSF where it will be submerged by tailings and water.

Runoff and seepage from each waste rock dump will be collected in ditch along the dump toe and directed to the mine water collection and treatment pond downstream of the plant site.

It is expected that waste rock classified as PAG during operations will be encapsulated within the underground paste tailings backfill to prevent oxidation of the sulphide minerals. PAG waste rock may be hauled to the TSF for subaqueous storage where backfill to the underground mine is not feasible. The waste rock management plan should be reviewed in future designs when more detailed classification and scheduling of waste rock is available.

18.4.4 WATER MANAGEMENT

GENERAL

The Pitarrilla project is located in a dry climate where evaporation is estimated to be approximately three to four times greater than annual precipitation. Reliable water supply will therefore be critical for the successful and efficient operation of the mine. The key sources of water are underground mine dewatering and surface runoff from the TSF. The TSF has a large catchment area relative to the tailings impoundment and supernatant pond areas resulting in a net accumulation of water during an average hydrometeorological year. The site-wide water balance has been calculated on an average annual basis to include water consumption and conditions encountered to date in the underground mine.

The Mexican water authority agency, Comisión Nacional del Agua (CONAGUA), is responsible for granting water abstraction permits. Detailed impact studies and remediation plans that are sensitive to the local land tenure are required to support permit applications. Watersheds in protected zones (e.g. hydropower production, irrigation, ecological conservation) may require additional study and analysis, and may have restrictions on the type and volume of abstraction allowed.

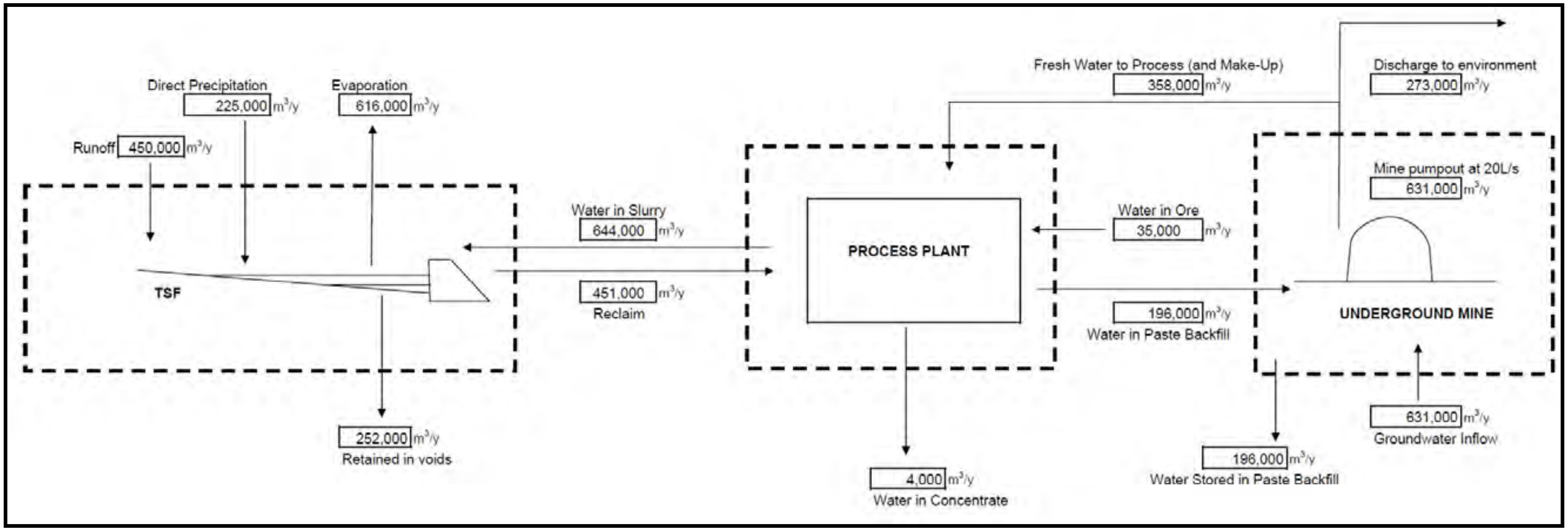
Groundwater abstraction permits are generally easier to obtain than those for surface water, but likewise require supporting study and analysis. An on-going study of regional groundwater targets as potential sources of fracture controlled aquifers is underway, with some success to date. Pump testing and analysis is on-going to investigate the capacity and feasibility of the groundwater resources intercepted to date.

SITE-WIDE WATER BALANCE

A site-wide average annual water balance was prepared to aid in the development of the water management plan, and to estimate the required quantities of fresh and recycled water for mine operations. A schematic flowsheet representation of the water balance model, including results, is shown on Figure 18.17. The water management objectives reflected in the water balance are as follows:

- provide sufficient water for mine operations with a contingency for dry years
- capture and manage all water that has been affected by mine operations
- divert runoff not required for mine operations from undisturbed areas to retain existing creek flows to the extent possible
- effectively manage the TSF supernatant pond and storage ponds to minimize the risks associated with having a deficit or surplus of water.

Figure 18.17 Water Balance Schematic Flowsheet



Notes:

1. All volumes are report in m^3/a.
2. Calculations assume mean average annual conditions.
3. All areas have been calculated for Year 10 of operations.

It was assumed that the catchment areas of the TSF and plant site do not change during the life of the project and the underground mine water flow is consistent. Therefore, a single year water balance is considered suitably representative of the average operating conditions for this level of study.

The principal supply of water for the project will be from mine dewatering, in combination with surface water abstraction from the TSF. It was concluded that supplemental groundwater is not required to make up a significant proportion of the site water requirements.

The site water will be managed by the selective routing of water to the environment or for use in the process circuit. All water introduced to the process circuit will be discharged to the TSF. A large volume of water may be stored in the TSF pond to provide a process water source during the dry season. The pond level in the TSF will be managed by the selective discharge of water to the environment from either the underground mine or diversion of runoff from the undisturbed TSF catchment. Surface water diversion channels constructed upstream of the TSF will include stop log controlled intake structures to allow water to either be diverted around, or let into the TSF, depending on the supernatant pond elevation and available storm storage capacity. Water discharged to the environment from the mine dewatering will be released via a mine water treatment pond. Treatment is expected to comprise settling of suspended solids.

The entire site is estimated to be in a water surplus during an average hydrometeorological year. This water surplus will be managed for the annual and seasonal variation by selective use of water from mine dewatering. The water will be used to offset shortages in the mill process circuit. If excess water is available from mine dewatering, it can be stored in the tailings impoundment for use during prolonged dry conditions, discharged for use in irrigation, or as otherwise determined in future socio-economic evaluations. The key results of the water balance are as follows:

- The water requirement for the mill is approximately 844,000 m^3/a (or ~100 m^3/h)
- The underground paste backfill will consume approximately 196,000 m^3/a of process water (or on average ~23 m^3/h). This preliminary estimate may be high but is adopted as a conservative number.
- The mine dewatering will generate approximately 631,000 m^3/a (or ~72 m^3/h). The potential for increased groundwater recharge to the underground mine workings should be evaluated in subsequent studies based on observations from the exploration adit.
- The mine dewatering will not be able fully supply the mill (i.e. reclaim from the TSF is required).
- The TSF will receive approximately 675,000 m^3/a of water from combined precipitation and runoff.

- The TSF will lose approximately 616,000 m^3/a of water to evaporation.

Based on average conditions, the required fresh and makeup water for the plant operations can be provided from a combination of mine dewatering and runoff collected in the TSF with approximately 270,000 m^3/a surplus.

Prior to start-up, it will be necessary to capture sufficient water in the TSF to allow the mill to begin processing ore. The volume of water required to achieve this will be sensitive to the start-up date in relation to the wet season (i.e. a mill start-up at the onset of the wet season will require less initial water storage than a mill start in the dry season). It is recommended that either one wet season of water is collected in the TSF or three months of pumping water from the underground mine to the TSF is allowed for in the construction schedule to provide sufficient water for the mine operations in the first year. Construction scheduling will need to consider this requirement and it may be necessary to construct the TSF in advance of the other facilities.

SITE WATER MANAGEMENT PLANNING

Process water for the mill will be obtained from the mine water pond. This pond will be replenished from the mine dewatering and either pumped to a holding tank above the plant, or directly into the process circuit. Water recovered from the process and the filter plant will be recycled or held in the process water pond. This pond will be required to store water for the following activities:

- temporary storage until it can be recycled in the plant
- reconstituting paste backfill from filtered tailings; this may require storing several days of waste until an underground block is ready for backfill
- collecting runoff from the plant site that has been in contact with ore reagents (i.e. fine ore stockpile runoff and sump pump-out from the mill)
- any water or tailings needing to be dumped from the tailings or water reclaim lines to the TSF.

The working storage volume for the process water pond is approximately 10,000 m^3, as recommended by Wardrop. An additional 10,000 m^3 of storage capacity is provided for emergency inflows and storm water management from the small catchment around the plant site.

Surface runoff water from the plant site will be collected and directed to the process water pond for use in the process circuit. Seepage collected in the TSF underdrains and runoff from the tailings embankment will be collected in a seepage pond at the downstream toe of the embankment for pump-back into the TSF.

Sediment control measures such as silt fences, check dams, and sediment ponds will be required throughout the site for surface water control.

A number of groundwater monitoring wells have been established downgradient of the proposed plant site and TSF facilities to provide a baseline for water quality. These wells, together with those installed for the construction program, will be monitored during construction and mine operations to confirm that there is no potential for contamination of groundwater.

The site experiences occasional extreme rainfall events during the wet season and it will be important to ensure that the TSF and other facilities are designed to operate in a safe manner during these events. A spillway may be required to pass the design flood.

18.5 GEOTECHNICAL DESIGN

Silver Standard engaged Knight Piésold to review the available background information on the Breccia Ridge Zone and the geomechanical information available from the 200-series drillholes with the objective of providing a first assessment of the rock mass characteristics in and around the Breccia Ridge Zone orebody. This information, combined with laboratory test results, was used as the basis for the required rock mechanics comments on achievable stope dimensions and ground support requirements for the access development and crusher.

With the assistance of the exploration staff, the various lithologies encountered in the drillholes were grouped into domains of similar lithological and geomechanical characteristics. The 4 domains that will host initial mining (between the 1600 and 1200 m elevations) include the Andesites, Basal Conglomerate, Replacement Mineralization, and Sediments. The rock mass characteristics within these domains were summarized from the available logging information and laboratory testing results. The domain characteristics were found to be quite variable, but covered similar value ranges. In all cases, the rock mass rating (RMR) distributions suggest FAIR to GOOD quality rock masses with UCS values between 50 and 70 MPa.

Empirical design techniques were used to determine appropriate stope dimensions for pre-feasibility level design. Target stope heights and widths were provided by the mine design team. After a number of iterations, the results suggest that a 15 m (W) x 20 m (L) x 35 m (H) should generally be achievable assuming reasonable mining practices are employed. Back and End Wall stability can be expected to be an issue if rock mass qualities tend towards the lower bound design values. A 30 m stope height or a 15 m stope length would increase stand-up time and reduce the potential for dilution. Stope performance can be improved operationally with quality backfill installation, appropriate drilling and blasting practices, timely installation of support, and relatively short cycle times. In some cases, cables may be required to maintain Back stability. It would be an advantage if the selected mining method allowed stope lengths to be adjusted to be consistent with actual rock mass performance.

Support recommendations for the permanent access development, associated intersections, and the crusher were developed from a combination of empirical design approaches, rules-of-thumb, and experience at Canadian mining operations. Rebar, welded wire mesh, cables, and shotcrete are expected to be the main elements of the required support systems. Longer support elements will be required in regions of wider spans and shotcrete will be utilized within regions of reduced rock mass quality.

18.6 PROJECT EXECUTION PLAN

18.6.1 *INTRODUCTION*

The Project Execution Plan presents how Silver Standard will successfully complete the Pitarrilla project. The Project Execution Plan specifies the project approach, tasks, and schedule. As well, it identifies and addresses any unique challenges facing the project. The project will be designed and constructed to industry and regulatory standards, with emphasis on addressing all environmental and safety issues. Adherence to the Project Execution Plan will ensure timely and cost effective completion while ensuring quality is maintained.

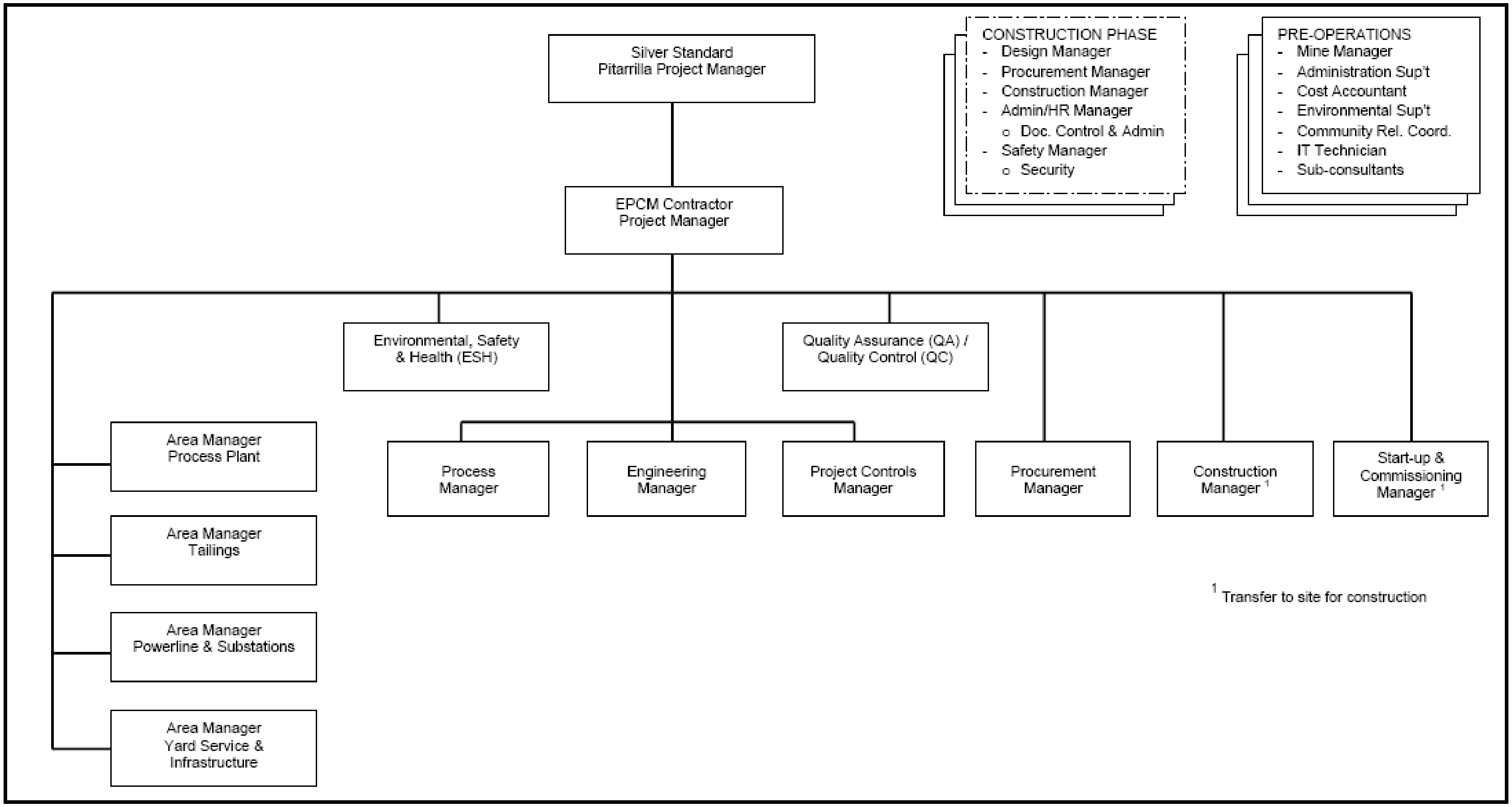
18.6.2 *PROJECT APPROACH*

To achieve successful project execution, Silver Standard will assemble a Project Management Team (PMT). The PMT will be comprised of personnel with appropriate skills, knowledge, and experience and will act with the support of multi-discipline consultants and contractors. The PMT will, with the support of its consultants and contractors, ensure that checks, balances, progress monitoring, regulatory guidance, and quality assurance/control to provide the information to manage effectively are implemented. The execution philosophy is based on Engineering, Procurement, and Construction Management (EPCM). The EPCM contractor (Contractor) will be required to implement the following:

- Project Management System
- Engineering Records System
- Procurement System
- Logistics Plan
- Health and Safety Plan
- Construction Managing and Contract Plan
- Quality Assurance/Quality Control System
- Environmental Management Plan
- Labour Relations Plan.

The Project Management organizational chart is shown in Figure 18.18.

Figure 18.18 Project Management Organization Chart



Project Management System

A proven and integrated Project Management System (PMS) will be utilized by the Contractor to facilitate monitoring and control of the project. The PMS will provide precise and accurate information to the Contractor and Silver Standard, enabling them to make decisions and implement actions for the successful execution of the project. The PMS will also provide reporting of the status of the project, ensure documentation of scope changes, track the budget and schedule; it will also compare actual performance with planned activities and report the effect of anticipated changes on the final date and cost.

Project Controls Personnel

An integral part of the PMS is the project controls function. The personnel assigned to this function will plan and control the schedule and costs of the project by use of an integrated project control system, which will encompass the functions of scheduling, cost control, estimating, change control, monitoring and reporting for the engineering, procurement, construction, and pre-operational testing of the project.

Project Controls personnel will utilize PMS to perform the following functions:

- Planning and Scheduling
- Cost Control
- Cost Engineering/Estimating.

Planning and Scheduling

The project schedule will set out the project's planning and controlling schedules. At the commencement of the project, the following planning and control activities will be undertaken:

- The Project Master Schedule will be developed as the principal control document.
- The Front End Schedule is essentially a schedule produced early in the project to monitor and accumulate detailed activity status and progress.
- The Detailed Project Schedule will be developed as scope definition and work packages are finalized.
- The Control Level Schedules represent the day-to-day tasks which summarize activities and/or deliverables.

Engineering Cost Monitoring and Control

Budgeted, committed, and actual costs of hours for engineering and procurement activities will be monitored within the project cost control system together with other engineering costs and expenses. Monthly reports will be produced from the detailed schedule, man-hour monitoring and forecasting system, and the project cost control system showing the status of the engineering and procurement phase progress and costs.

Cost Control will include cost monitoring, trending, and forecasting in order to measure performance in relation to project budget and schedule.

The Project Control Budget will be formed on the basis of the approved Pre-Feasibility Study estimate. Cost Control personnel will maintain cost trending and forecasting accountability by keeping the originally approved Pre-Feasibility Study capital cost estimate and maintaining an audit track of specific decisions managed through scope changes. The Control Budget will include items such as:

- original contract price
- approved changes
- current contract price
- billings this period and to date
- changes submitted but not approved
- forecasted final contract price.

Cost Estimating

Cost Engineering/Estimating includes developing capital cost estimates for the overall project, estimating in support of value engineering, scope change estimates, and fair bid estimates for construction contracts.

TRENDING/CHANGE REQUESTS

Trends and Change Requests may originate from any member of the project team and/or Silver Standard.

Typically initial sources are identified from:

- design instructions
- minutes of meetings
- performance analysis
- procurement changes due to vendor data, market prices, and supply demand
- construction changes due to soil reports, weather, labour, equipment, material, and field instructions
- environmental changes due to social, economic, and political forces.

An order of magnitude estimate is prepared and schedule impact is assessed. The trend is then reviewed, and if required, corrective action is initiated. If the corrective action is successful and the trend does not affect cost and schedule, the trend log is updated accordingly. If the corrective action is not successful, a trend report is prepared and other affected task force members are notified. The cost impact is incorporated in the project cost forecast and schedules are updated accordingly.

Trend meetings will occur on a regular basis to review changes and strategy for corrective action. All trends will be expeditiously priced. Routine changes will be estimated within two to five days depending on the complexity and the availability of information. Depending on the magnitude or nature of the impact, a Change Request may be required.

18.6.3 Project Execution Summary

A well-managed plan will be initiated from the date that project execution begins. An effective project management system will be implemented to assist in managing project costs and scheduling. The team will ensure that:

- the critical path schedule of construction is met or improved upon
- engineering and procurement activities are completed to support construction requirements
- costs are monitored, controlled, and reported to Silver Standard on a regular basis.

Within six months from the project go-ahead, the following will be completed:

- award of contract
- project control structure including budget, schedule, procedures, and work plans
- bidders lists
- completed flowsheets and material balances
- Project Procedures Manual (PPM)
- process design frozen
- final site layout
- all design criteria including, but not limited to, environmental, applicable codes, materials of construction, and control philosophy
- process equipment list with Request for Proposal (RFP) packages
- the assignment of package contract numbers
- modularizing, pre-assembly, and purchasing strategies
- finalized contracting strategy

- approved training program
- contracts for early construction activities tendered, received, and evaluated
- Health and Safety Management Plan (HSMP)
- Quality Assurance/Quality Control Plan (QA/QC)
- Environmental Management Plan (EMP)
- Construction Plan
- all project management systems in place
- all geotechnical and site survey data completed.

18.6.4 ENGINEERING

The detailed design engineering program will include all disciplines from geotechnical to computerized controls. Each discipline will utilize both recent technological advances and proven techniques as are appropriate for this project.

Once Silver Standard has authorized the project to proceed, the EPCM Contractor will establish the engineering organization and assemble the necessary resources required to meet project demands.

The first step of establishing project standards and procedures melding with those required by Silver Standard and the relevant regulatory bodies has been completed during the Pre-feasibility level design resulting in a set of Design Basis Memoranda. The design basis is based on local requirements, industry guidelines, and North American standards. The design basis addresses all aspects to be considered during the detailed design (e.g. Health & Safety, structural, architectural, environmental, etc.) and specific requirements raised by Silver Standard.

Additionally, and in compliance with the Design Basis Memorandum, the Pre-feasibility level design provided PDC, Process Flow Diagrams, Piping and Instrumentation Diagrams (P&IDs), and general arrangements. During the detailed design, the internal, public, and environmental review process may highlight the need for revision of these documents. Such changes will be noted and any alterations or improvements precipitated by this process will be incorporated in the facility design on a continuous basis.

In addition to detailed design drawings, detailed engineering will provide:

- work scope definitions
- installation specifications
- modularization detail where appropriate
- shipping requirements for larger or more delicate items
- heavy lift instructions.

These will be coordinated with the work packages, construction schedule, and logistics schedule. The Construction Manager will undertake constructability reviews throughout the development of detailed designs.

The list of project activities with a budgeted time for each activity and a corresponding list of deliverables (drawings, specifications, data sheets, requisitions, MTOs, BOMs) will be placed into the Engineering Management System for project control purposes.

Additionally, engineers and technical staff will be assigned to the construction program for drawing interpretation, and updating drawings to an "as built" status. The final engineering step will be the cataloguing and entering of all design and procurement information into the Silver Standard central library including computerized drawing and administration files at the site and head office in Vancouver.

18.6.5 PROCUREMENT

Procurement of goods and services will adhere to the highest ethical standards and will be performed in a transparent manner. The Procurement group will develop and implement procurement policies that:

- comply with project technical requirements
- comply with the Health, Safety, and Environmental (HSE) policy
- comply with legal and regulatory
- deliver goods and services to satisfy project schedule requirements
- where quality, price, and availability are competitive on a global basis, sourced within Canada.

The Procurement group will prepare procurement procedures and a procurement plan for the execution of the project, including procedures for purchasing, inspection, progress monitoring, material control, expediting, batch crating and packaging, transshipping, consolidating, and transportation.

CONTRACTS

Contract Form

Silver Standard will use a widely recognized standard Form of Contract for all tendering including the primary EPCM Contract. Use of such a contract form ensures key aspects of the contract (i.e. arbitration) are not overlooked. As well, Contractors tendering should already be familiar with the Form of Contract so will not require excessive time understanding and assessing the implication of any nuances in a project specific unique contract form.

In support of the standard Form of Contract, project specific Terms of Reference, Scope of Work, and Deliverables will be prepared. These Contract Sections as well as source data will be assembled to comprise a Tender Package.

It is anticipated that the Contract deliverables will include items of each of the following types:

- Lump Sum – fixed amounts for specified works
- Re-measureable – unit prices with re-measureable quantities
- Provisional Sum – fixed amounts for specified works that may or may not be executed.

CONTRACT PACKAGING PLAN

Construction Management (CM) begins with an overall basic EPCM project philosophy. All planning from conceiving project environmental strategy through to the stages of project approval and financing to final design and procurement phases are involved in developing the actual construction program into logical consulting, technical, service, equipment supply and construction contract packages.

LOGISTICS PLAN

The Logistics Plan addresses the need to procure and deliver materials, equipment, and supplies to meet the restricted transportation delivery windows to the site. The team work and implementation of the logistics plan is critical and must be agreed upon by all concerned.

The scope of the logistics plan provides for and encompasses the services necessary for the efficient transport, traffic, warehousing, and marshalling of personnel and all materials and equipment, fuel, and cement required to construct the facilities. The objective of the traffic and logistics plan is to ensure that equipment, materials, and personnel are transported to the project site in a safe, efficient, economical, and timely manner to meet construction schedules. It is imperative that materials and equipment transported during the shipping window arrive at the site without loss or damages and according to the planned window sequences to enable all work to be completed on schedule.

PROCUREMENT AND EXPEDITING

The EPCM contractor's Purchasing Group will provide capital equipment procurement, vendor drawing expediting and, when required, equipment inspection. The procurement department will package the technical and commercial documentation and manage the bidding cycle for equipment and materials to be supplied by Silver Standard to the contractors.

Standard procurement terms and conditions approved for the project will be utilized for all equipment and materials purchase orders. Suppliers will be selected based on locations, quality, price, delivery, and support service.

The CM group will organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, issue tenders, analyze and recommend suitably qualified contractors to Silver Standard, and prepare executed contracts for issue.

LOGISTICS

A freight forwarding company will coordinate with manufacturing facilities, establish shipping points and dates, forward the shipments to the most convenient ports, and complete trans-shipments to the project site.

To execute procurement and materials control across the various parties and work break down areas, certain policies, procedures, forms, and coding structures will be standardized (e.g. vendor communication policy, material requisition forms, material takeoffs (MTO), material status reports, document numbering systems, material reference codes, etc.).

PROCUREMENT SCHEDULES

The procurement activities of preparing and issuing the RFP bid tabulation and the purchase order deliverable items list will be tracked. Milestones will be developed where the typical activities include:

- Expediting:
  - Expediting ensures a continuous flow of equipment and, materials to the marshalling yards, at the scheduled time and in the proper sequence to facilitate timely transport to site.
- Logistics:
  - The Logistics system function will track material and equipment deliveries to multiple project yards and job site lay down areas and maintain a control over multiple inventories at the various sites.
- Materials Control:
  - Materials control evolves from initial definition and packaging of materials by engineering, through purchasing, expediting, fabrication, delivery, receiving, and use. Site materials management involves multi-warehousing, receipts, issues, returns, inventory management, and inter-warehouse transactions.
  - The system provides status reported on purchasing and expediting along with materials controls database to track equipment and materials from the design stage to final delivery and installation.

- Traffic and Logistics Coordination:
  - The Traffic and Logistics (T&L) will consist of support staff, computer systems, communications equipment, leased marshalling yard, and warehousing facilities. Logistics will be the advanced planning for the movement of material from its point of origin to the location where it is required.

18.6.6 CONSTRUCTION

The construction phase is subdivided into three main phases: Construction Management, Field Engineering, and Construction Contracting.

CONSTRUCTION MANAGEMENT

The CM group will be responsible for the management of all field operations. Reporting to Silver Standard, the Construction Manager will plan, organize, and manage construction quality, safety, budget, and schedule objectives. The key CM objectives are:

- Conduct HS&E policy training and enforcement for all site and contractor staff. Site hazard management tools and programs will be employed to achieve the no harm/zero accident objective.
- Apply 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 as well as schedule reporting and control. A cost trending programme will be instigated whereby the contractor will be responsible for evaluating costs on an on-going basis for comparison to budget and forecasting for the cost report on monthly basis.
- Establish a field contract administration system to effectively manage, control, and coordinate the work performed by the contractors.
- Apply an effective field constructability program, as a continuation of the constructability reviews performed in the design office.
- To develop a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.
- Meet the schedule for handover of the constructed plan to the commissioning team.
- Develop a QA/QC plan to set guidelines in terms of plant operability, safety of operation and adherence to all regulatory requirements.

The CM organization chart (Figure 18.19) shows the CM team organization plan for the site.

CONSTRUCTION SCHEDULE

Schedule of Development

The first construction schedule issue will be to identify activities that clearly outline the project logic.

The construction schedule will expand with sub-schedules addressing specific activities and contracts. The construction schedule will typically control all activities. For example, activities in years two and three are essential to the project's timing and limiting commitments; therefore, a sub-schedule will outline the "ramping up" activities in detail. Scheduling will be linked as required to the construction contract packages, which will in turn be required to produce schedules, depending on the nature of the individual contracts. These in turn will be monitored by the EPCM controls staff. A preliminary schedule is presented in Figure 18.20.

Figure 18.19 CM Organization Chart

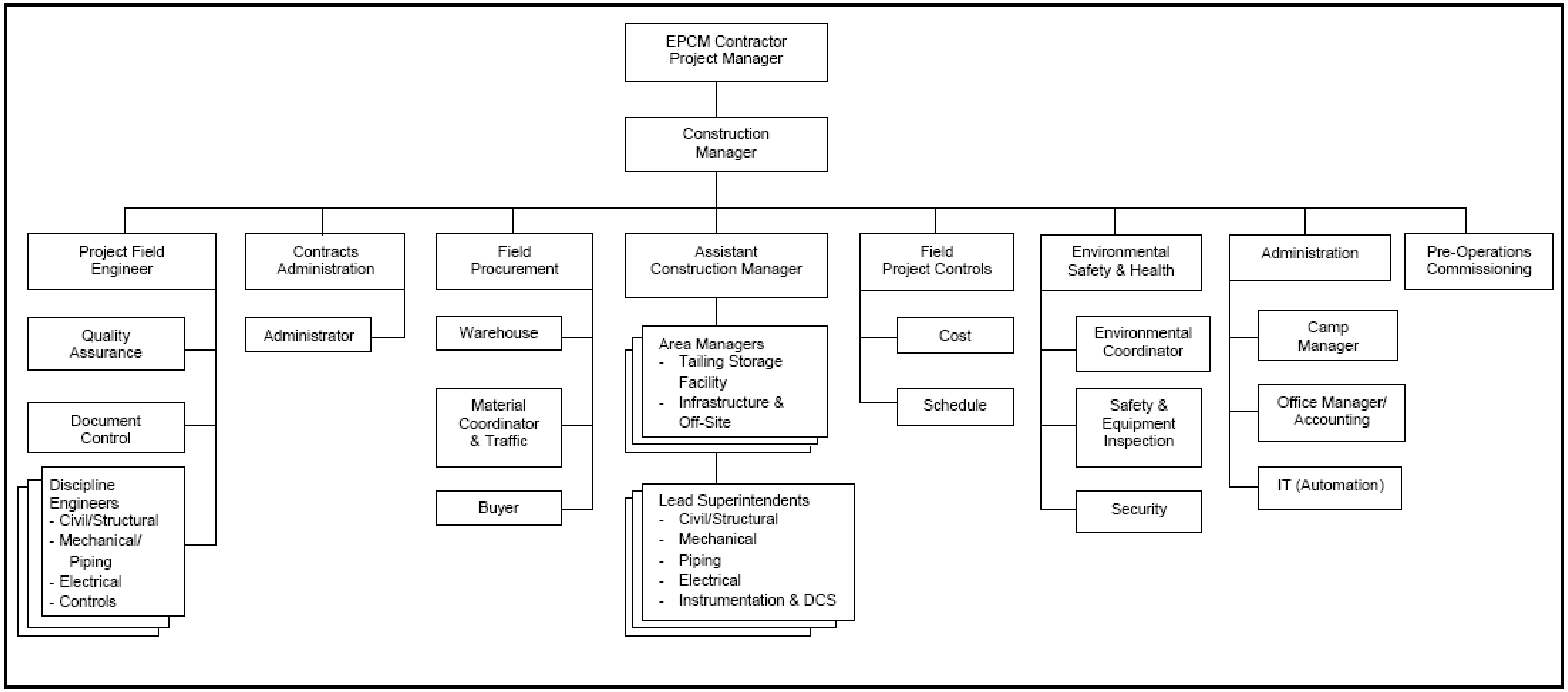
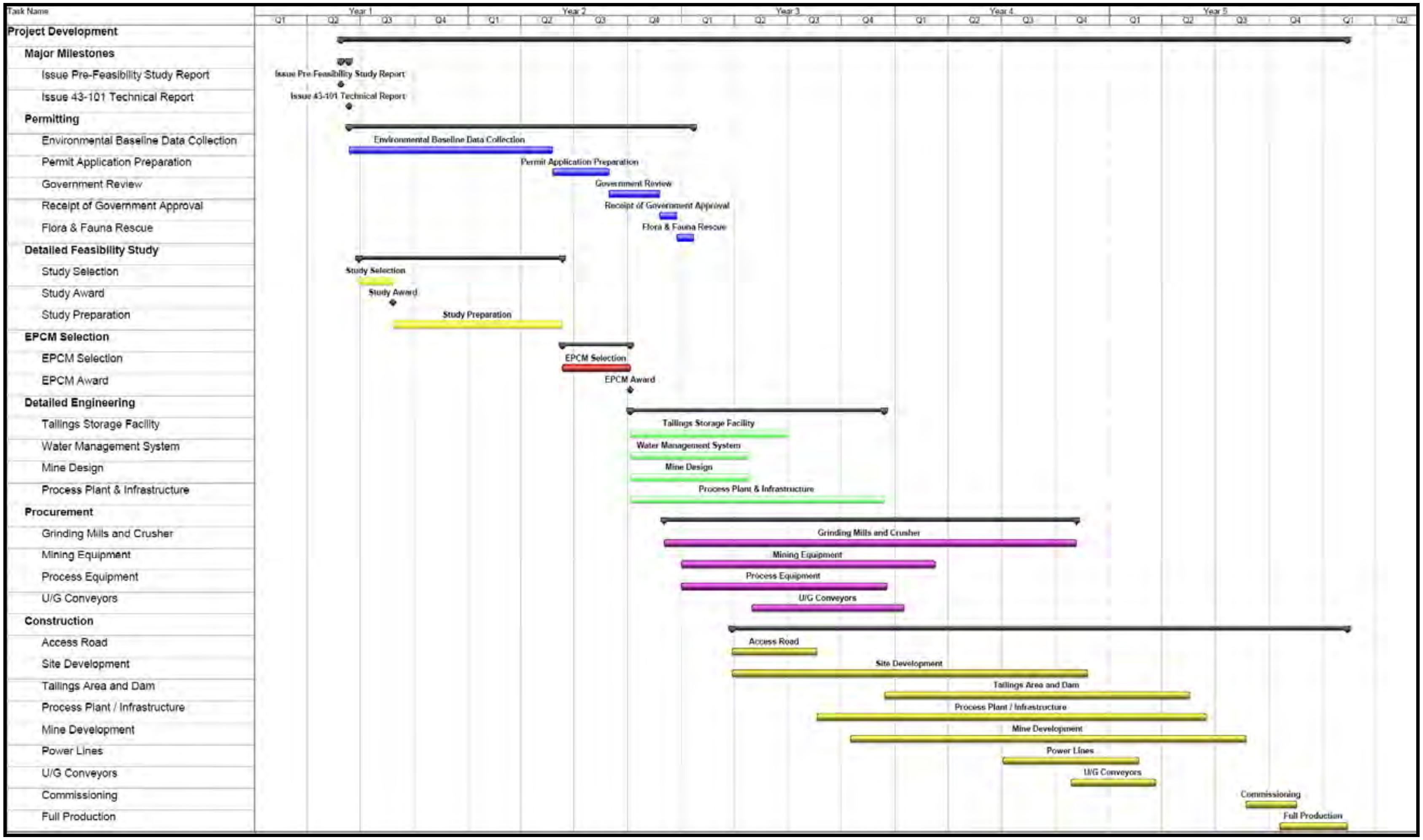


Figure 18.20 Preliminary Project Development Schedule Summary



CONSTRUCTION CONTRACTING

The contracting strategy will be designed to maximize the local labour force, create a responsible and sustainable relationship with the nearby communities, and provide a mix of senior management and specialists to support the safety, quality, schedule, and cost objectives of the project. In addition, contracts will be designed to combine timing, scope, battery limits, and contract value into manageable packages.

Approved contract pro formas will be utilized for construction and service contracts. Construction contractors will be responsible for:

- all construction labour
- all construction equipment
- worker transportation
- site offices and temporary services
- site management
- contractor surveying
- quality control
- contract scheduling
- safety
- environmental safeguarding
- tools and equipment security
- permanent material supplying, as required by contract.

Silver Standard will provide contractors and construction management staff with:

- on-site, project-wide first aid services
- project-wide security
- locations for offices and equipment/material laydown
- local electrical panels and temporary generator sets
- water sources
- diesel fuel, including storage for construction equipment
- sources for all concrete and structural aggregates
- permanent bulk materials and all capital equipment
- quality assurance and control audits
- vendor-representative assistance.

FIELD ENGINEERING

Surveying

The CM survey crew will verify the accuracy of the existing control system before construction begins. Contractors will use only applicable control data for the project. Additional monuments will be set as needed. The Construction Manager will verify surveys prior to construction. Contractors will supervise day-to-day field surveying, and the CM team will provide spot checks.

Quality Control/Quality Assurance

Contractors will establish and observe their own Quality Control program in accordance with the construction technical specifications and the applicable codes and standards. The CM Field Engineering Team will employ independent CSA-qualified Quality Assurance specialists to ensure quality control.

MATERIALS MANAGEMENT

Warehousing

The Site Materials Management group will receive, inspect, and log all incoming materials, assign storage locations, and maintain a database of all materials received and dispensed to the contractors. On-going reconciliation with the procurement system, including reconciliation to the freight consolidation point, will confirm the receipt of materials and payment of suppliers. An allowance for the lease or purchase of warehousing equipment has been included in the construction budget.

Construction Equipment

Individual contractors will be responsible for the equipment required to meet their contract obligations. All equipment must comply with Mine Safety Standards requirements; Silver Standard CM team will perform regular spot checks. Large cranes may be supplied by a single company, managed by the CM team.

TEMPORARY FACILITIES AND CONSTRUCTION SITE INFRASTRUCTURE

Construction Accommodation

Development of a full-service camp for construction contractors will begin immediately following the receipt of construction permits.

The camp, a modular design, will accommodate up to 250 workers. Accommodations types will include single occupancy rooms.

Transportation, potable water, waste management and other support services will be scaled to support the various development stages. The CM team will ensure that catering contractor meets all facilities, staffing, hygiene, food handling, storage, and meal expectations.

Communication

Silver Standard systems manager will determine the appropriate telecommunications technologies for the project. Requirements include voice and data link technologies adequate to support growth construction and plant operation growth.

Construction Power

Permanent power will supply all mine equipment and construction power loads for the duration of the construction phase.

First Aid and Site Security

Silver Standard will provide a fully-equipped first-aid facility and ambulance for project-wide use. The facility will normally be staffed 12 h/d, with on-call services ensuring continuous coverage. The first-aid staff will live at the camp. Contractors will be expected to provide basic first-aid stations at the site.

Silver Standard will supply a 24-hour staffed site security program during the initial field mobilization. Access to the site will be controlled at the principal road entrance and will be limited to personnel who have attended induction training, as well as approved visitors.

Warehousing

Construction warehousing will evolve with the project. All freight delivered will be received at a temporary warehouse and stored there or in designated laydown areas.

Initially, fabric or fold-out type structures will be erected to serve as the light and heavy vehicle maintenance shops and general shop/warehouse area for the relevant contractors.

Laydown Areas

Rapid development of the laydown areas at the site is absolutely essential to tie-in with the arriving loads.

All laydown areas will be clearly marked with sign posts and will be laid out in a grid system to eliminate confusion due to snow cover.

Concrete Batch Plant

The concrete batch plant will be managed by the General Contractor as a service to the project and will be operated by the General or Site Services Contractor.

Water Supply and Treatment Plant

The construction project will require fresh water for the following:

- potable drinking water
- truck washing
- concrete batching
- road dust control
- fire water
- building cleaning
- washroom and cleaning purposes.

Sewage Treatment

A portable temporary sewage treatment plant (STP) will be among the first items shipped to the project. It will be a modular system that is very easy and quick to set up.

18.6.7 *Pre-Operational Testing and Start-up*

Process Plant

When construction is complete on any process unit, the construction organization will turn over responsibility to the Commissioning Manager for pre-operational testing and turnover of the facility to Silver Standard prior to introducing ore into the plant for commissioning and start-up.

Silver Standard's operating personnel will be involved in the pre-operational testing phase to the extent that they will progressively accept responsibility for sections of the plant as they are checked and handed over.

Pre-operations testing of equipment will begin once the equipment items have been delivered to site, erected, and tested by the vendor's engineers.

The pre-operational testing phase for the process facilities will include all aspects of dry mechanical and electrical testing of equipment and water testing of process equipment, including pressure testing of pipework and wet pre-operational testing as far as practicable.

The following procedure and tagging system will be adopted in the execution of assignments as work is being completed.

Visual Inspection

Visual inspection is the non-operational examination of an installation to check that it is in accordance with the engineer's and the manufacturer's drawings, specifications, and manuals.

Pre-Operational Test

A pre-operational test is the initial no-load test of a piece of equipment with test media such as water or air where required.

Visual inspection and pre-operational testing (Yellow Tag) will occur upon completion of installation of plant and equipment where the Construction Contractor will submit one copy of the appropriate pre-operational check forms, which will notify that the plant and equipment are ready for inspection and the following have been put into effect and/or completed.

Checkout and Acceptance (Green Tag)

This procedure allows for the transfer of responsibility from the Contractor to Silver Standard. This procedure establishes that the installation of the equipment and ancillaries has been completed in accordance with the Contractors' drawings, specifications, and codes and the equipment has been energized to prove its readiness for the process commissioning and start-up. On acceptance, Silver Standard assumes responsibility for operation and maintenance.

START-UP

Start-up (introduction of ore) is performed under the direction of the Silver Standard Start-up Manager, and involves a select staff from pre-operations, process specialists, and Silver Standard's operating personnel. This will be the beginning of operations under load conditions and the systematic increase in capacity until process through-put and recovery requirements are met and sustained.

18.7 ENVIRONMENTAL

18.7.1 REGULATORY REQUIREMENTS

Mine permitting in Mexico is primarily administered by the federal government body Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT). Following from objectives outlined in the Ley General del Equilibrio Ecológico y la Protección al

Ambient, an Environmental Impact Assessment (by Mexican regulations called a Manifestación de Impacto Ambiental, or MIA) will be required in order to gain approvals for mine construction and operation. SEMARNAT has indicated to Silver Standard that separate MIAs will be required for each of the road infrastructure, electrical transmission, and mine infrastructure. Additional regulatory triggers, based on anticipated process reagent volumes, will require that the MIA for the mine infrastructure include a Risk Assessment. Additionally, as a part of land tenuring, a Change in Land Use Application (Cambio de Uso de Suelo, or CUS), supported by a Technical Supporting Study (Estudio Técnico Justificativo) will be required for all areas of land disturbance.

In order to ensure acceptability to shareholders and potential financing bodies, the environmental documentation will also comply with the international guidelines presented in the Equator Principles, the International Finance Corporation Performance Standards, and the International Cyanide Management Code.

18.7.2 *Previous Studies*

In order to gain authorization for works conducted in the exploration phase of the Project, an MIA was completed by a local consultant and approved by SEMARNAT. As part of that study, a baseline database was initiated for the following:

- climate
- soils
- surface hydrology
- flora
- fauna
- socio-economic factors.

On-site climate data collection, along with an automated surface water hydrology program, has continued throughout the exploration phase. Additionally, a groundwater hydrology program has been initiated.

18.7.3 *Proposed Studies*

The three MIAs and the CUS required for mine authorization will require updated studies to those conducted to support exploration permitting. Local consultants have initiated the updated studies in 2008, which will be further refined to keep pace with refined mine planning.

18.7.4 *Identified Issues*

Previous studies have identified some cactus species and one rattlesnake species that would require special management. However, none of these species would require measures beyond typical mine mitigation techniques such as relocation or "rescue" programs. No other environmental issues have been identified that would alter or compromise mine planning.

18.8 TAXES

Federal income taxes in Mexico consist of corporate income tax at a rate of 28% on an accrual basis and the Impuesta Empresarial a Tasa Unica (IETU) which is a flat tax of 17.0% in 2009 and 17.5% starting in 2010. Corporate taxpayers must pay the greater of the annual regular income tax and the IETU tax. The excess of an IETU liability over the regular income tax liability is not recoverable.

The Value Added Tax (VAT) in Mexico is 15% but is creditable and should not be considered an expense in most cases.

The only State of Durango Tax is a 2% tax on salaries and benefits which are included in the payroll burdens.

Other payroll taxes included Social Security, Housing Fund, and Pension Fund which are 25%, 5%, and 2%, respectively, on salaries and benefits. These payroll taxes have been included in the labour rates as payroll burden.

18.9 CAPITAL COST ESTIMATE

18.9.1 INTRODUCTION

The Pitarrilla Project Pre-feasibility Study capital cost estimate (CAPEX) was developed with an accuracy of ±25% and is suitable for project evaluation and client budgeting. The CAPEX consists of four main parts: direct costs, indirect costs, contingency, and Owner's costs.

As of April 2009, the CAPEX for the Project is US$277,439,630. The estimate is subject to qualifications, assumptions, and exclusions, as detailed in this report.

The CAPEX has been developed using input from Wardrop, Knight Piésold, and Silver Standard.

Knight Piésold provided capital cost estimates for the following project components:

- roads to the TSF from the plant site
- slurry TSF
- seepage collection and recycle system for the TSF
- tailings transport system

- reclaim water system
- process and mine water ponds.

Details of costs developed by Knight Piésold can be found in the report titled "4,000 tpd Pre-feasibility Study, Tailings, Waste Rock, and Water Management", provided in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices".

The capital cost summary and distribution is shown in Table 18.18. The detailed CAPEX is also provided in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices".

Table 18.18 Summary of Project Capital Costs

Area	Cost ($US)
Direct Works	
A – Overall Site	20,076,140
B – Mining	82,074,817
C – Crushing	2,864,049
D – Fine Ore Storage and Conveyance	9,342,787
E – Grinding and Flotation	49,859,273
F – Tailings	13,415,841
G – Site Services and Utilities	2,948,270
J – Ancillary Buildings	5,389,405
K – Plant Mobile Fleet	2,789,160
M – Temporary Services	3,750,000
N – Off-site Infrastructure and Facilities	8,640,889
Direct Works Subtotal	**201,150,630**
Indirects	
X – Project Indirects	44,974,000
Y – Owner's Costs	2,500,000
Z – Contingency	28,815,000
Indirects Subtotal	**76,289,000**
Total Project	**$277,439,630**

18.9.2 PROJECT AREAS

The estimate has been assembled and coded based on the project specific work breakdown structure (WBS) (Table 18.19). The following areas are in the scope of work for the cost estimate unless otherwise noted.

Table 18.19 WBS for Project Areas

Project Areas	
A – Overall Site	
A1	Plant Site and Roads
A2	Power Supply and Distribution
A3	Control System
A4	Communication
A5	Yard Lighting
B – Mining	
B1	Underground Mining Development
B2	Underground Mobile Equipment
B3	Underground Equipment
B4	Underground Fuel Storage and Delivery
B5	Underground Backfill
B6	Underground Dewatering
B7	Underground Electrical
B8	Underground Communication
B9	Underground Safety
B10	Underground Engineering Equipment
B11	Underground Explosives Storage
B12	Underground Maintenance Shop
C – Crushing	
C1	Primary Crushing
D – Fine Ore Storage and Conveyance	
D1	Crushed Ore Stockpile and Reclaim
E – Grinding and Flotation	
E0	Mill Enclosure
E1	Grinding & Classification
E2	Lead Flotation & Regrind
E3	Copper Flotation (not included)
E4	Zinc Flotation & Regrind
E5	Concentrate Dewatering & Loadout
E7	Reagents
F – Tailings	
F1	Tailings Thickening
F2	Tailings Filtration & Paste Backfill
F3	Dry Tailings Containment (not included)
F4	Dry Stack Equipment (not included)
F5	Process Water Reclaim
F6	Tailings Storage Facility (KP-B1)
F7	Tailings Storage Water Management (KP-B2) (sustaining)
F9	Dry Stack Water Management (KP) (not included)

table continues…

Project Areas	
G – Site Services and Utilities	
G1	Process Water
G2	Potable Water
G3	Gland Water
G4	Fresh/Fire Water Storage & Distribution
G5	Plant & Instrument Air
G6	Sewage Collection & Treatment (Tile Fields)
J – Ancillary Buildings	
J1	Administration Building
J2	Mine Dry
J3	Canteen
J4	Bunkhouse Complex
J5	Maintenance/Warehouse Complex
J6	Assay Laboratory
J7	Tire Change & Oil Separator
J8	Propane Storage
J9	Open Storage Area
J10	Fuel Storage & Distribution
J12	Truck Wash
K – Plant Mobile Fleet	
K1	Plant Mobile Fleet
M – Temporary Services	
M1	Construction Camp
M2	Catering and Housekeeping (Included In Labour Rates)
N – Off-site Infrastructure and Facilities	
N1	Main Access to Site (Supplied by Owner)
N2	Tailings Access Roads (KP-A)
N3	Mine Water Storage (KP-D3)
N4	Process Water Management (KP-D1, D2)
N5	Pipeworks and instrumentation (KP-E1, E2, E3, E4, E5)
N6	Reclamation and Closure (KP-5)
X – Project Indirects	
X1	Construction Indirects
X2	Spares
X3	Initial Fills
X4	Freight and Logistic
X5	Commissioning and Pre-operational Start-up
X6	EPCM
X7	Vendors
Y – Owner's Costs	
Y1	Owner's Costs
Z – Contingencies	
Z1	Contingency

18.9.3 ESTIMATE ORGANIZATION

The CAPEX is assembled and coded with a hierarchical WBS of Major Area, Area, and Section numbers as follows:

X99	99	99.99
Area	Section	Sequence

The area coding is based on the area numbering system described in Section 18.9.2 and the section codes are shown in Table 18.20.

Table 18.20 Section Codes

Code	Description
Direct Works	
1.1	Dry Tailings Containment (Not Included)
1.2	Dry Stack Equipment (Not Included)
1.3	Tailings Storage Facility (KP-B1)
1.4	Tailings Storage Facility Water Management (KP-B2)
1.5	Water Management (KP) (Not Included)
1.6	Tailings Storage Access Roads (KP-A)
1.7	Mine Water Storage (KP-D3)
1.8	Process Water Management (KP-D1, D2)
1.9	Pipeworks & Instrumentation (KP-E1, E2, E3, E4, E5)
1.11	Reclamation and Closure (KP-5) (Sustaining)
2	Earthworks
4	Civil
6	Concrete
8	Structural Steel
10	Architectural
11	Platework
12	Mechanical
13	Piping
14	Building Services
17	Instrumentation and Controls
18	Electrical
20	Surface Mobile Equipment
40	Mining
42	Mining Mobile Equipment
Indirect	
91	Construction Indirects
92	Spares
93	Initial Fills
94	Freight and Logistics

table continues…

Code	Description
95	Commissioning and Pre-operational Start-up
96	EPCM
97	Vendors
98	Owners Costs
99	Contingency

18.9.4 Sources of Costing Information

The CAPEX is based on the following:

- budget quotations for all “tagged” equipment
- pricing from local contractors from the area
- cost books such as RS Means, NECA, and JS Page
- Wardrop’s in-house database
- costing from similar projects.

Comprehensive equipment specifications were prepared and issued for bid to qualified vendors for budgetary quotations for all major equipment. The vendors supplied equipment prices, delivery lead times, freight costs, and spares allowances. In some instances, the vendor also provided an estimate of installation hours for the specified equipment.

All equipment and material costs are included as free carrier (FCA) or free board marine (FOB) manufacturer plant and exclusive of spare parts, taxes, duties, freight, and packaging. These costs, if appropriate, are covered in the indirect section of the estimate.

Equipment items valued under $100,000 are priced from in-house data or from recently updated pricing data from previous projects, unless the equipment was of a specialized nature.

18.9.5 Quantity Development and Pricing

All quantities were developed from conceptual general arrangement drawings, process design criteria, process flow diagrams and equipment lists. Details on the respective discipline quantities are as described in the following sections.

Bulk Earthwork Including Site Preparation

Bulk Earthwork quantities are generated from rough grading designs using Autodesk Land Development Desktop/Civil Design. Structural fill pricing is based on aggregates being produced at site utilizing a portable crushing and screening plant;

the price of the aggregate plant is included in the CAPEX. No allowances were included for bulking or material compaction as these were included in the unit price.

In the bulk earthwork estimate, Wardrop has made the following assumptions:

- Clearing and grubbing was based on minimal clearing and grubbing requirements.
- Stripping of topsoil is included at 300 mm depth.
- Excavate and remove waste to original ground profile; it is assumed that this original ground profile is bedrock.
- Rock excavation is included – 50% rippable, 50% drill and blast.
- Surplus excavated material will be stockpiled on site.
- All roads will have 200 mm thick surfacing material (minus 50) complete with 300 mm thick base (minus 300) and 1,500 mm thick sub-base.

CONCRETE

Concrete quantities were calculated from in-house data, current prices from previous projects, and are based on “neat” line quantities from engineering designs and sketches. For designing purposes, designers have provided quantities to the estimator in the following breakdown:

- lean concrete
- concrete footings
- concrete grade beams
- concrete columns and pedestals
- concrete walls
- concrete slab on grade and curbs
- concrete elevated slabs
- concrete equipment bases, <1 m^3
- concrete equipment bases, >1 m^3
- concrete sumps
- anchor bolts
- embedded metal
- rock anchors
- grout.

Typically all concrete is based on 30 MPa with the exception of lean mix levelling concrete, which is 10 MPa. Wardrop has assumed that a batching plant will be available on site and the batching cost is assumed to be \$350-\$450/m^3.

The total unit price for concrete includes supply (batched) formwork, rebar, placement, and concrete finishing.

STRUCTURAL STEEL

Steel quantities were calculated from in-house data and current prices from previous projects as follows:

- light weight steel sections – 0 to 30 kg/m (tonnes)
- medium weight steel sections – 31 to 60 kg/m (tonnes)
- heavy steel sections – 61 to 90 kg/m (tonnes)
- extra heavy steel sections – >90 kg/m (tonnes)
- stairways (m) including platforms
- grating (m^2)
- handrail comes with kickplate (m)
- ladders (m).

The supply unit rates for fabricated steel ranges from \$2,500 to \$4,800 per tonne depending on the above classification.

Craneage was included according to steel category, between \$175 and \$250 per tonne.

Buildings were calculated in accordance with Mexican standards on a square metre footprint basis.

PLATEWORK AND LINERS

Quantities for all platework and metal liners for tanks, launders, pumpboxes, and chutes have been calculated based on detailed quantity take-offs developed from design drawings and sketches and provided in kilograms of steel. Rubber lining for pumpboxes has been provided on a square metre basis. Abrasion resistant (AR) wear plate and rubber linings are included as appropriate. All quantities for AR plate and rubber linings were all calculated from design sketches based on layouts.

PLUMBING AND DRAINAGE

The estimate was based on costs per square metre, calculated from in-house data based on building function and site-specific climatic conditions.

HVAC and Fire Protection

HVAC and fire protection were based on detailed sheets from Wardrop's HVAC department.

Dust Control

Dust control was based on detailed sheets from Wardrop's HVAC department.

Piping and Valves

The piping allowance was based on a percentage of the equipment cost relevant to each area.

Electrical

Electrical costs were included as per detailed take-offs and quotations obtained by Wardrop's Electrical department, based on single line diagrams and electrical load lists.

Cost estimates were developed using a material and labour approach for:

major electrical equipment (major equipment quoted), including:
- 4 kV switchgear
- power transformers
- 480 V switchgear
- 480 V motor control centres (MCCs)

- electrical infrastructure (based on in house estimate of quantities)
- motor wiring (based on wire and material take-offs with current material pricing – all derived from items on the equipment list – using cable prices obtained specifically for the project)
- estimates are by area where possible; the remaining equipment was designated as "infrastructure"
- factoring and in-house pricing was used for smaller items, as required
- variable frequency drives (VFDs) were included in this area.

Instrumentation

An allowance of $550,000 was included for the plant control system (including hardware, software, and licences)

The field instrument allowance was based as a percentage of the mechanical equipment costs.

18.9.6 ESTIMATE BASE CURRENCY

The estimate has been prepared with US dollars as the base currency. Foreign exchange rates, as noted below are applied as required (Table 18.21).

Table 18.21 Foreign Exchange Rates

Base Currency	Foreign Currency
US$1.00	Pesos$11.00
US$0.89	Cdn$1.00

18.9.7 LABOUR COST DEVELOPMENT

LABOUR RATE

A blended construction labour rate of US$15.00/h is utilized in the estimate, which includes the catering allowance. This is based on hourly rates collected from local contractors. The calculation of the blended labour is as shown in Table 18.22.

Table 18.22 Labour Rate Calculation (April 30, 2008)

	Pesos/ hour	US$/ hour	Weight by Trade		Ratio (%)	Blended (US$/h)
			%	$		
1 – Exchange Rate: Pesos$11.00 = US$1.00						
2 – Unskilled Labour						
Earthworks	37.04	3.37	10	0.34		
Concrete	75.16	6.83	25	1.71		
Steel	56.20	5.11	25	1.28		
Mechanical, Electrical, Instrumentation	75.16	6.83	40	2.73		
Unskilled Labour Subtotal			100	6.06	65	3.94
3 – Skilled Labour						
Earthworks	167.89	15.26	10	1.53		
Concrete	106.14	9.65	25	2.41		
Steel	167.84	15.26	25	3.81		
Mechanical, Electrical, Instrumentation	167.89	15.26	40	6.11		
Skilled Labour Subtotal			100	13.86	35	4.85
Blended Labour Rate (before mark-up) Subtotal						**$8.79**
4 – Add Contractor's Markup						
Field Supervision			13.5			1.19
Site Trailers			1.0			0.09

table continues…

	Pesos/ hour	US$/ hour	Weight by Trade %	Weight by Trade $	Ratio (%)	Blended (US$/h)
Contractor Trucks			4.5			0.40
Transportation of Personnel			1.5			0.13
Small Tools less than $1,500			5.0			0.44
Consumables			2.5			0.22
Safety Supplies and Radio			8.0			0.70
Subtotal						**$11.95**
Home Office & Overhead Costs			15.0			1.79
Profit			10.0			1.19
Subtotal						14.94
5 – All-in Blended Rate/Hour						**US$15.00**

The following activities are included in the rate:

- contractors' field supervision, which includes managers, general foremen, secretarial, and office personnel (overhead)
- office supplies, running costs, and vehicle costs (overhead)
- temporary facilities and utilities such as tool sheds, cribs, small equipment, maintenance facilities, power, water, and sewer (overhead)
- small tools and consumables
- miscellaneous indirect costs such as power and water distribution, clean-up, safety supervision and protection, safety training, welder's certification, etc. (overhead)
- contractors' profit and overhead included as a percentage including home office overhead costs and profit
- sub-contractors mark-ups
- all construction equipment, cranes, and incidental equipment rentals (see line items)
- mobilization and demobilization costs (see Construction Indirects)
- freight costs relating to contractors materials are included in the estimate line items
- scaffolding rental (See Construction Indirects)
- construction camp and catering costs are included in the labour rate
- travel time for personnel, if appropriate (see Construction Indirects).

The following labour rated items will be calculated separately and included in the Indirects section.

Labour Premiums

Scheduled site hours are 7 days x 10 hours = 70 hours (40 hours x 1 and 30 hours x 1.5). Turn-arounds are based on 3 weeks in, 1 week out.

Labour Productivity

The labour man-hour included in the estimate is based on North American standard adjusted with a labour productivity factor of 2.

18.9.8 PROJECT INDIRECTS

Project Indirects include but are not limited to the following items.

X1 – CONSTRUCTION INDIRECTS

Construction indirects are based on a percentage of direct costs (9%) and include the following:

- temporary services
- miscellaneous equipment and craneage requirements (rentals)
- garbage disposal
- waste disposal
- sewage
- surveying
- first aid/medical
- warehousing
- bus transportation
- personnel turn-around costs
- safety
- final clean-up.

X2 – SPARES

Allowances have been included for process (2%) and mining rolling stock (5%).

X3 – INITIAL FILLS

Initial fills for both process and mining were considered part of the working capital, as requested by the Owner.

X4 – Freight and Logistics

The freight and logistics allowance included in the estimate is based on percentages of related equipment and materials. A more accurate estimate for freight can be established when all details of equipment and materials are known. Freight and logistics costs include:

- land and ocean transportation
- loading and offloading including craneage
- marshalling yard
- ocean transportation
- customs duties and brokerage.

X5 – Commissioning and Start-up

The commissioning and start-up calculation is based on a rate of $125/h. The process plant is based on 160 man-weeks and vendors are based on 64 man-weeks.

X6 – EPCM

An EPCM allowance is calculated based on percentages of the direct costs and it includes:

- engineering design
- procurement, expediting, and inspection
- contract administration
- construction management and controls, contract controls
- site trailers and vehicles
- engineering support during construction and commissioning.

X7 – Vendors

Vendor assistance during construction has been included.

Y1 – Owners Costs

Owners costs have been removed from the CAPEX (as requested by the Owner), with the exception of construction insurance and security requirements. A small allowance of $250,000 was included for miscellaneous items.

Z1 – Contingency

Contingency amounts included in the estimate are detailed in Table 18.23.

Table 18.23 Allowances for Contingencies

Section	Description	Percentage
Direct Works		
1.1	Dry Tailings Containment (deleted)	-
1.2	Dry Stack Equipment (deleted)	-
1.3	Tailings Storage Facility (KP-B1)	KP Allowance
1.4	Tailings Storage Facility Water Management (KP-B2)	KP Allowance
1.5	Water Management (KP-B2) (sustaining capital)	KP Allowance
1.6	Tailings Storage Facility Access Roads (KP-A)	KP Allowance
1.7	Mine Water Storage (KP-D3)	KP Allowance
1.8	Process Plant Water Management (KP-D1, D2)	KP Allowance
1.9	Pipeworks & Instrumentation (KP-E1 to KP-E5)	KP Allowance
1.11	Reclamation & Closure (sustaining capital)	KP Allowance
2	Bulk Earthworks	20
4	Civil	16
6	Concrete	15
8	Structural Steel	15
10	Architectural	12
11	Platework	12
12	Mechanical	12
13	Piping	15
14	Building Services	12
17	Instrumentation and Controls	12
18	Electrical	12
20	Surface Mobile Equipment	0
40	Mining	12
42	Mining Mobile Equipment	5%
Indirects		
91	Construction Indirects	10
92	Spares	0
93	Initial Fills	0
94	Freight and Logistics	10
95	Commissioning & Pre-operational Start-up	0
96	EPCM	10
97	Vendors	0
98	Owner Costs	10

18.9.9 Exclusions

The following items are excluded from the CAPEX:

- working or deferred capital
- financing costs
- refundable taxes and duties
- land acquisition
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials and services resultant from a change in project schedule
- warehouse inventories other than those supplied in initial fills
- any project sunk costs including this study
- pre-production costs, excluding pre-stripping
- mine drainage costs
- mine reclamation costs
- mine closure costs
- escalation beyond October 2008
- community relations
- mine equipment fleet at mine
- Owner's risks and exposure.

18.10 Operating Cost Estimate

18.10.1 Summary

Operating costs are estimated to be US$33.81/t of ore processed including mining, processing, power and general and administrative (G&A). The unit costs are based on an annual ore production rate of 4,000 t/d and 350 days of operation. Mine operating costs are summarized in Table 18.24.

Table 18.24 Operating Cost Summary

Area	Unit Cost (US$/t ore)
Mining	14.55
Process	9.15
Power	6.03
G&A	4.08
Total Operating Cost	**$33.81**

18.10.2 *Basis of Estimate*

The accuracy of the estimate is ±25%, which is suitable for the pre-feasibility level of studies. Sources used for establishing labour rates include Western Mining Engineering's 2008 published figures, budgetary quotes, recent Wardrop projects in Mexico, and labour rates at similar existing mining operations in Mexico.

18.10.3 *Mining*

The estimated mining unit cost of US$14.55/t of ore includes mine hourly labour, equipment operation, supplies, administration, sundries, and the activities listed in Table 18.25. Underground equipment operating cost estimation is based on hourly equipment operating costs provided by mine equipment suppliers, Western Mine Engineering published data, and experience of Wardrop engineers. Mining operating costs are summarized in Table 18.25.

Table 18.25 Mining Operating Cost Summary

Mining Operating Cost Distribution	US$/t Ore
Stopes	2.10
Drifts	1.11
Crosscuts	0.15
Draw points	0.44
Access Raises	0.20
Ore Passes	0.01
Ventilation Raises	0.02
Main Haulage	0.79
Back Fill	3.30
Services	1.31
Ventilation	1.17
Exploration	0.07
Maintenance	0.43
Administration	2.07
Miscellaneous	1.38
Total Operating Cost	**14.55**

18.10.4 PROCESSING

All process operating costs are shown in US dollars, unless otherwise specified.

Average annual process operating costs are estimated to be $9.15/t milled. The process operating costs are based on a process rate of 1,400,000 t of ore annually and 350 days of operation.

The estimated process operating costs are summarized in Table 18.26 and include the following:

- personnel requirements including supervision, operating staff, and maintenance (salary/wage levels are based on current labour rates in comparable operations in Mexico)
- liner and grinding media consumption have been estimated from the Bond Ball Mill Work Index and Abrasion Index equations and have been quoted as budget prices, or derived from the Wardrop database
- maintenance supplies are calculated based on major equipment capital costs
- reagents are based on test results and quoted budget prices, or derived from the Wardrop database
- paste backfill operating and maintenance supplies
- other operation consumables including laboratory, filter cloth, and service vehicle consumables.

Table 18.26 Process Operating Cost Summary

Description	Personnel	Unit Cost ($/t milled)
Operating Personnel		
Operating Staff	12	0.85
Operating Labour	44	0.51
Maintenance Labour	32	0.38
Met Lab and Quality Control	16	0.17
Sub-Total Staff	**104**	**1.91**
Supplies		
Operating Supplies		3.45
Maintenance Supplies		0.55
Sub-Total Supplies		**4.00**
Paste Backfill Operating and Supplies		3.24
Total Process Operating Costs (Excluding Power)	**104**	**9.15**

Operating Personnel

The projected personnel requirements are 104 persons, including:

- 12 staff for management and professional services
- 44 operators including paste and backfill operators
- 32 personnel for maintenance
- 16 personnel for quality control, process optimization, and assaying.

Salary/wage rates are based on current rates in Mexico, including base salary, holiday and vacation pay, various benefits, and tool allowance costs. The total estimated personnel cost is $1.91/t milled.

Operating and Maintenance Supplies

The operating cost for major consumables and operating supplies for the process plant (excluding the paste backfill plant) are estimated to be $3.45/t milled. The major consumables include the crusher, mill liners, and reagents.

Reagent consumption has been based on the consumption rates as given in the PDC (provided in "Pitarrilla Pre-feasibility Study Volume 2 – Appendices"). The reagent costs were obtained from current budget prices from potential suppliers.

The maintenance supplies cost has been estimated at $0.55/t milled and excludes the paste backfill plant.

The maintenance and supply costs for the paste backfill plant are estimated at $3.24/t milled, which includes the cost of cement, transportation and handling charges, as well as the maintenance costs.

18.10.5 General and Administrative

G&A costs are estimated at $4.08/t milled, which has been based on similar operations in Mexico and developed by Wardrop. The operating costs include:

- salaries for administrative personnel
- medical and first aid
- safety and training
- office overheads.

18.10.6 POWER

The unit power cost is estimated at US$6.03/t and is not included in the process operating costs. The cost of power for the process plant, along with the mine and surface facilities, is discussed in Section 18.3.2 of this report.

18.11 FINANCIAL ANALYSIS

18.11.1 INTRODUCTION

An economic evaluation of the Pitarrilla Project was prepared by Wardrop based on a pre-tax financial model. For the 12-year mine life and 16.7 Mt reserve, the following pre-tax financial parameters were calculated:

- 10.9% internal rate of return (IRR)
- 6.2 years payback on $277 M capital
- US$107.4 M net present value (NPV) at a 5% discount rate.

The base case prices supplied by Silver Standard were as follows:

- silver – US$11.00/oz
- zinc – US$0.70/lb
- lead – US$0.50/oz.

Sensitivity analyses were carried out to evaluate the project economics for 2-year, 5-year, and current metal prices.

18.11.2 PRE-TAX MODEL

FINANCIAL EVALUATIONS

The production schedule, based on a reserve with an NSR cut-off grade of US$50/t, has been incorporated into the pre-tax financial model to develop annual recovered metal production. Market prices for silver, zinc, and lead have been adjusted to realized price levels by applying smelting, refining, and concentrate transportation charges from mine site to smelter in order to determine the NSR contributions for each metal.

Unit operating costs for mining, process, and G&A were applied to annual milled tonnages to determine the overall mine site operating cost which has been deducted from NSRs to derive annual net revenues.

Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the net revenue to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate; sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and tailing embankment construction.

Working capital has been calculated on the basis of three months mine site operating costs and applied to the first year of expenditures. It will be recovered at the end of the mine life and aggregated with the salvage value contribution and applied towards reclamation during closure.

METAL PRICE SCENARIOS

The financial outcomes for four metal price scenarios have been tabulated for NPV, IRR, and payback of capital. Discount rates of 10%, 8%, and 5% were applied to all cases identified by the following metal price scenarios:

- base case
- 2-year average
- 5-year average
- current prices.

The results are presented in Table 18.27.

Table 18.27 Summary of Pre-tax NPV, IRR, and Payback by Metal Price Scenario

	NPV at Selected Discount Rates (Million Cdn$)			IRR (%)	Payback (Years)
Scenario	10%	8%	5%		
5-year Average	163.6	214.7	310.5	20.3	3.8
Silver Standard (Base Case)	12.8	45.5	107.4	10.9	6.2
2-year Average	317.9	388.6	521.1	28.9	2.8
Spot	168.7	221.3	320.1	20.4	3.9

Wardrop adopted metal prices supplied by Silver Standard for the base case. The prices provided are lower than the historical 3-year average metal prices. The 3-year average is an approach which is considered an industry standard and is in line with the SEC requirements of the United States of America.

The prices used for the financial evaluations are based on the rolling historical average prices from the LME are summarized in Table 18.28.

Table 18.28 Summary of Pre-tax Metal Price Scenarios

Scenario	Silver (US$/oz)	Zinc (US$/lb)	Lead (US$/lb)
5 Year Average	11.16	0.99	0.72
Silver Standard (Base Case)	11.00	0.70	0.50
2 Year Average	14.06	1.01	1.02
Spot	14.02	0.71	0.77

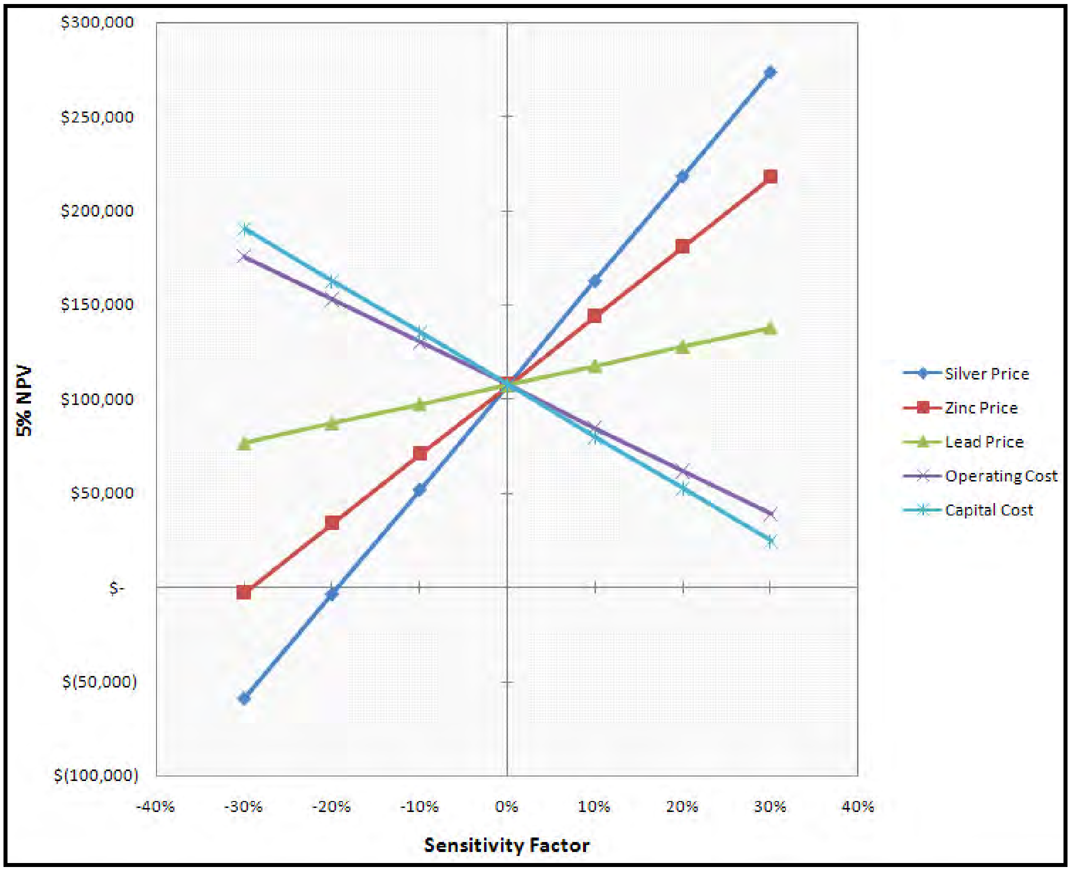
SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- silver prices
- lead prices
- zinc prices
- initial capital expenditure
- mine site operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR (Figure 18.21 and Figure 18.22). The project NPV (5% discount) is most sensitive to the silver price and, in decreasing order: zinc prices, capital cost, operating costs, and lead price.

Figure 18.21 NPV Sensitivity Analysis



Similarly, the project IRR is most sensitive to the silver price and, in decreasing order: zinc prices, capital cost, operating costs, and lead price. The IRR is more sensitive to initial capital when the initial capital and mine site operating costs are tested at minus 10 and 20%. However, when the initial capital and mine site operating costs are tested at plus 10 and 20%, there is an equal effect on the IRR. A reduction of capital at the start of the project reduces the payback period and significantly improves the IRR.

Figure 18.22 IRR Sensitivity Analysis

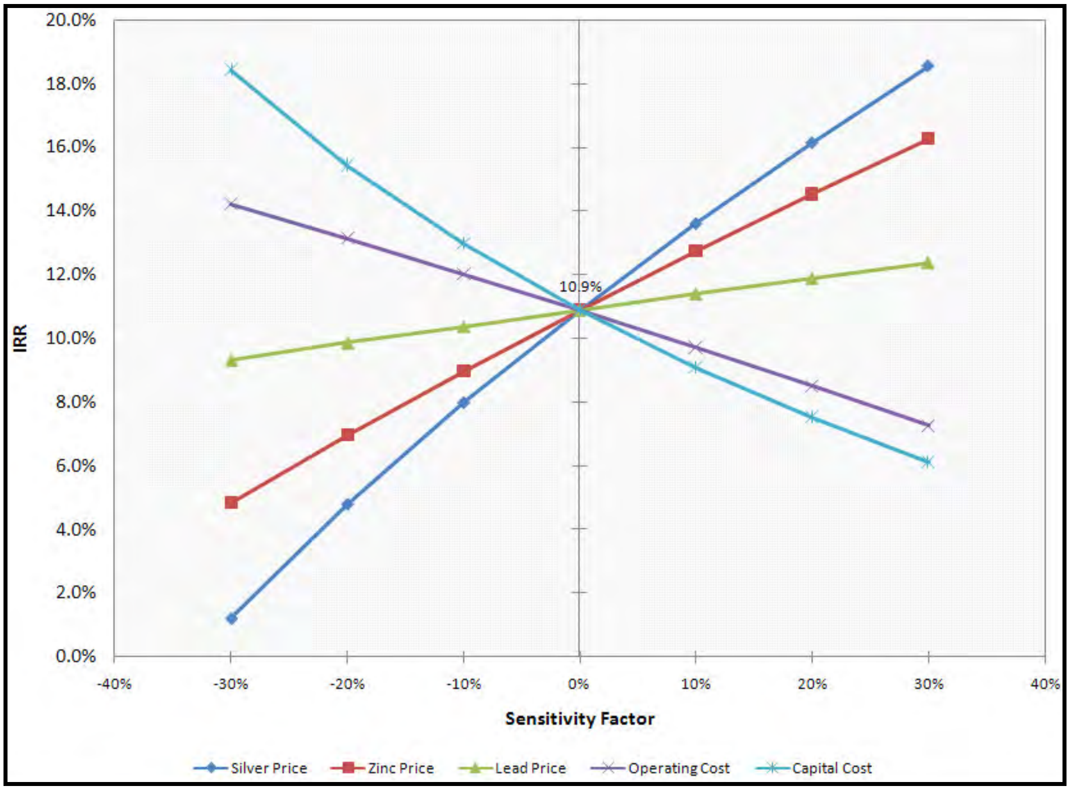
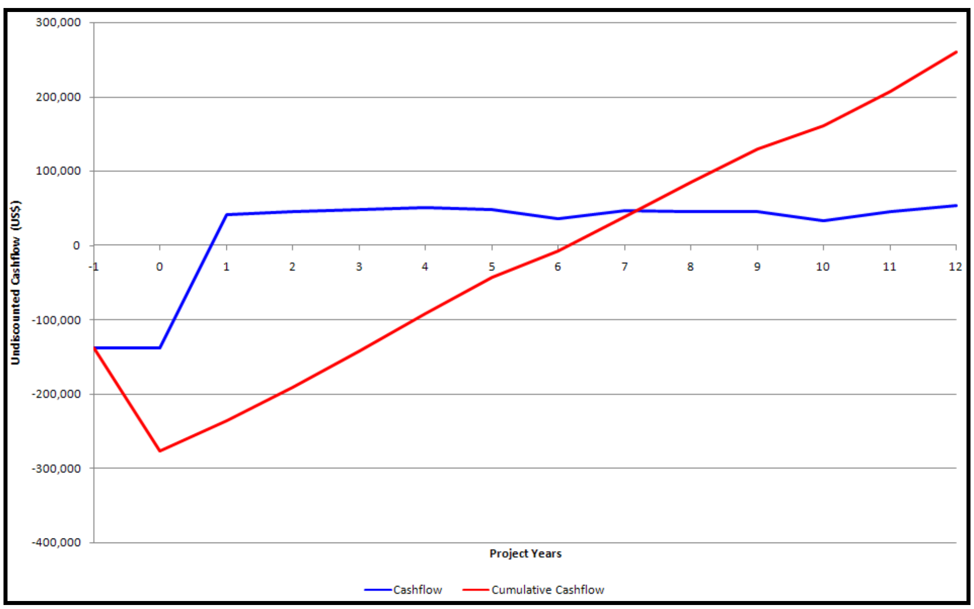


Figure 18.23 Cumulative and Annual Undiscounted Pre-tax Cash Flows



ROYALTIES

There are no royalties on the project.

18.11.3 SMELTER TERMS

In the absence of letters of interest or letters of intent from potential smelters or buyers of concentrate, in-house database numbers were used to benchmark the terms supplied by Silver Standard.

Contracts will generally include payment terms as follows:

- Lead Concentrate:
  - Silver – pay 97% of silver content; a refining charge of $0.40/accountable troy oz will be deducted from the metal price.
  - Lead – pay 95% of lead content.
  - Zinc – pay 0% of zinc content.
  - Treatment Charge – US$160/dmt of concentrate delivered will be deducted. The treatment charge might be subject to both positive and negative price escalation. When the price is over US$0.36/lb, the refining charge will be increased by 15% of the excess. When the price is below US$0.36/lb, the refining charge will be reduced by 15% of the deficit.
  - Impurities – there are no penalties due to impurities.
- Zinc Concentrate:
  - Silver – pay 60% silver content less 3 oz Ag/dmt concentrate; a refining charge of $0.40/accountable troy oz will be deducted from the metal price.
  - Zinc – pay 85% of zinc content.
  - Lead – pay 0% of lead content.
  - Treatment Charge – US$225/dmt of concentrate delivered will be deducted.
  - Impurities – there are no penalties due to impurities.

18.11.4 MARKETS

The project will produce a lead concentrate containing the majority of the recovered silver as well as a separate zinc concentrate. The high silver grade of the lead concentrate will make it a desirable concentrate for smelters. The concentrates will be sold and shipped to Mexican refineries, the closest of which is in Torréon (275 km). Deliveries to overseas smelters and other North American smelters will also be investigated. Ports that can handle containerized concentrates exist on both the Pacific Ocean and Gulf of Mexico. For this study, it was assumed that the concentrate will be transported to Torréon.

18.11.5 CONTRACTS

There are no established contracts for the sale of concentrate currently in place for this project.

CONCENTRATE TRANSPORT LOGISTICS

Concentrate will be truck transported from the mine site at a charge of US$40.00/wmt.

Concentrate Transport Insurance

An insurance rate of 0.15% will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

Owners Representation

An Owners representation rate of US$0.25/wmt will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

Concentrate Losses

Concentrate losses are estimated at 0.5% of the provisional invoice value during shipment from the mine to smelter.

18.11.6 CASH COST ANALYSIS

Average annual revenue from silver for the life of mine is 60% silver, 27% zinc, and 13% lead. The life of mine cost to produce 1 oz silver net of a zinc and lead credit is US$5.74/oz Ag.

The breakeven price for the project is $10.77/oz Ag, when the NPV at 10% is equal to zero.

The breakeven price for the project is $5.33/oz Ag, when the undiscounted operating cash flow is equal to zero.

19.0 CONCLUSIONS & RECOMMENDATIONS

Based on the conclusions from the Pre-feasibility Study, it is recommended that the project proceed to the Feasibility Study stage. In order to advance the project to this stage, additional in-fill drilling should be conducted. The information from this drilling would be used to complete reserve definition, geotechnical studies, and hydro-geologic investigations. The drilling and studies are estimated by Silver Standard to cost approximately US$7,200,000.

Subsequently, the Feasibility Study should proceed. Silver Standard estimates that the budget for Feasibility Study phase of the project will be US$2,200,000.

19.1 METALLURGICAL TESTING AND MINERAL PROCESSING

19.1.1 CONCLUSIONS

Saleable lead and zinc concentrates can be produced from the Pitarrilla ores using conventional mineral processing techniques including crushing, grinding and flotation.

Tailings disposal will be using the backfill paste technique for underground stopes and tailings dam deposition with water reclaim.

19.1.2 RECOMMENDATIONS

At the proposed treatment rate of 4,000 t/d, it should be ascertained whether a conventional 3-stage crushing plant followed by ball mills may be more efficient than the proposed SAG mill/ball mill grinding combination.

The physical parameters of the main ore types should be confirmed and include the determination of the crushing work index, the ball mill work index, and the specific gravity and bulk density determinations. Bulk density data for the concentrates should also be determined.

Variability tests are required to confirm the selected process flowsheet, and also to confirm the relationships deduced with respect to feed grade and concentrate grades and recoveries. This is particularly important given that four ore types will be treated and that the individual flotation characteristics differ.

Filtration tests at the selected primary grind sizes are required to confirm whether a disc or belt filter will be better suited to the application.

The potential influence of the presence of variable amounts of copper in the ore needs to be qualified since this may impact the design of the plant.

The smelter that will purchase the concentrates should be identified, particularly with a view to identifying and understanding the impact penalty elements may have on the revenue and possibly on the process.

19.2 MINING

19.2.1 *GEOTECHNICAL*

The following work will help advance this design work to a level consistent with feasibility-level engineering.

- **Spatial Variations in Key Geomechanical Characteristics** – The downhole variations in the key geomechanical parameters collected from the exploration drill core need to be reviewed and discussed with site geological staff prior to finalizing the geomechanical drillhole positions. An enhanced understanding for the spatial distribution of each domain would also aid design.
- **Geomechanical Drillholes** – Additional geomechanical data needs to be collected to better understand the characteristics of the rock masses to be encountered around the ore body and in the area of key infrastructure. It is anticipated that a series of triple-tubed and oriented geomechanical drillholes will be required, especially in the area of initial mining. It is likely that this work could be combined with the planned underground definition drill program.
- **Laboratory Strength Testing** – Additional laboratory testing will need to be completed in order to better define the intact properties of each rock type.
- **Underground Mapping** – It is understood that there will be an opportunity to complete underground wall mapping of existing development in and around the orebody prior to the completion of the next phase of study. This will provide valuable information on excavation and support performance as well as the larger scale characteristics of the encountered discontinuities.
- **Unwedge Analyses –** With an enhanced understanding of joint-set characteristics from the oriented core and underground mapping, it will be possible to update support recommendations to account for the possibility of blocky type failures. UnWedge would be a suitable program for this type of analysis.

- **Numerical Modelling** – Current analyses have assumed that mining will take place under relatively low stress conditions and that the currently proposed sequence will quickly shed stresses to the abutments. The impact of mine sequencing and alternative far-field stress assumptions on stope, pillar, and permanent infrastructure stability needs to be evaluated. Examine3D and Phase2 would be suitable programs for these types of analyses.

19.2.2 HYDROGEOLOGY

Further study on hydrogeological conditions are required to estimate water handling and underground pumping requirements.

19.2.3 INFILL DRILLING

Further infill drilling is required to improve geological interpretations in order to better define stoping layouts.

19.2.4 PASTE BACKFILL

CONCLUSIONS

Cement screening and strength testing of tailings was performed by Unité de Recherche et de Service en Technologie Minérale (URSTM) in Rouyn-Noranda, Quebec, with two tailings samples from Pitarrilla and four locally sourced binder type samples provided by Silver Standard.

In general, Apasco cement provides better strength for both tailings types compared to Cruz-Azul.

Apasco cement at 6% addition provides strength greater than 200 kPa for both tailings types. The design strength for fill at 30 m H by 15 m W is estimated at 228 kPa.

The Basal tailings seem to be reactive to both cement types used for testing where there is a strength decrease of 9.2% and 9.7% for Apasco and Cruz-Azul cement, respectively, at 6% addition for 56 to 120 days.

Preliminary results indicate that the use of Apasco cement will yield strengths of 253.8 kPa for Sediments tailings at 56 days, and 251.53 kPa for Basal tailings at 28 days.

RECOMMENDATIONS

Depending on the tailings types used as backfill material, the mining sequence will have to be coordinated to meet these curing periods. Further tailings tests will have to be performed to accommodate for the tailings blend utilized as backfill material during operation.

Detailed investigation by strength testing and screening of cement with the possibility of adding ground blast furnace fly ash will assist in reducing costs and provide better strength results for tailings as constituents for backfill material.

19.2.5 TRADE-OFF STUDIES

The following items should be considered for future trade-off studies:

- reconsider early production from upper levels by trucking ore until the conveyor decline, crusher station, and ore and waste passes are developed
- consider less extraction from room and pillar mining as a trade-off against backfilling the room and pillar areas
- reduce the size of the backfill plant from 4,000 t/d to 2,000 t/d.

19.3 TAILINGS, WASTE ROCK, AND WATER MANAGEMENT

RECOMMENDATIONS

Knight Piésold has made the following recommendations with regard to tailings, waste rock, and water management:

- Prepare an estimated schedule of potentially acid generating waste rock production from the mine that includes quantities and timing. This will allow appropriate management plans to be developed in future studies.
- Additional geotechnical site investigations at the TSF site to support feasibility level design of a 4,000 t/d, 9 Mt facility may include:
  - geotechnical drilling in overburden soils and rock
  - hydrogeological testing of soil and rock
  - test pitting to characterise the surficial geology
  - laboratory testing of samples collected in the field.
- Detailed hydrometeorological study to support feasibility level design, including:
  - refined estimate of design storms and floods
  - inputs for a stochastic monthly water balance model
  - analysis of annual variation in climatic conditions for estimating wet and dry year precipitation and runoff.

- There will be an opportunity to quantify groundwater inflow rates during the next phase of development. This should be incorporated into the exploration planning.
- Development of a stochastic water balance model to assess sensitivity to variable wet and dry periods during the mine life.
- Spillway design based on additional hydrometeorology data and the appropriate Inflow Design Flood for the TSF.

20.0 REFERENCES

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21.0 CERTIFICATES OF QUALIFIED PERSONS

Certificates are provided in this section for the following QPs, as outlined in Section 2.0 (Introduction):

- Jacqueline McAra, P.Eng.
- Dr. Wayne D. Ewert
- Tracy J. Armstrong, P.Geo.
- Marinus Andre de Ruijter, P.Eng.
- Eugene Puritch, P.Eng.
- Iouri Iakovlev, P.Eng.
- Hasan Ozturk, Ph.D., P.Eng.
- Miloje Vicentijevic, P.Eng.
- Daniel Friedman, P.Eng.
- Robert A. Mercer, Ph.D., P.Eng.
- Peter Wells, SAIMM (Fellow)
- Scott Cowie, MAusIMM.

CERTIFICATE OF QUALIFIED PERSON

I, Jacqueline McAra, of Vancouver, British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Project Manager with Wardrop Engineering Inc. with a business address at 800-555 West Hastings Street, Vancouver, B.C., V6B 1M1
- I am a graduate of the University of Waterloo (BaSc, 2000).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #29671).
- I have practiced my profession continuously since graduation except during the years 2003.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to this project includes almost 9 years in mining consulting and project design.
- I am responsible for the preparation of Sections 1.0, 2.0, 3.0, and 18.3 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009. In addition, I visited the Property on February 17, 2008.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

"Original Document, Revision 03 signed and sealed by Jacqueline McAra, P.Eng"

Jacqueline McAra, P.Eng.
Project Manager
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Wayne D. Ewert, of Brampton, Ontario, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Principal with P&E Mining Consultants Inc., with a business address at Suite 202, 2 County Court Blvd., Brampton, Ontario.
- I am a graduate of Carleton University (Ph.D. in Geology, 1977).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License # 18965) and of the Association of Professional Geologists of Ontario (License # 0866).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Technical Report is as follows:

- Principal, P&E Mining Consultants Inc.	2004 - Present
- Vice President, A.C.A. Howe International Limited	1992 - 2004
- Canadian Mgr, New Projects, Gold Fields Canadian Mining Ltd.	1987 - 1992
- Regional Manager, Gold Fields Canadian Mining Ltd.	1986 - 1987
- Supervising Project Geologist, Getty Mines Ltd.	1982 - 1986
- Supervising Project Geologist III, Cominco Ltd.	1976 - 1982

- I am responsible for the preparation of Sections 4.0 through 11.0, inclusive, and Section 15.0 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Brampton, Ontario

"Original Document, Revision 03 signed and sealed by Wayne D. Ewart"

Dr. Wayne D Ewert
Principal
P&E Mining Consultants

CERTIFICATE OF QUALIFIED PERSON

I, Tracy J. Armstrong, of Magog, Quebec do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a senior associate geologist with P&E Mining Consultants Inc. with a business address at Suite 202, 2 County Court Blvd., Brampton, Ontario.
- I am a graduate of Queen's University, with a B.Sc. (HONS) in Geological Sciences, 1982.
- I am a member in good standing of the Order of Geologists of Quebec (License #566) and of the Association of Professional Geoscientists of Ontario (License #1204).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Technical Report includes:

- Underground production geologist, Agnico-Eagle Laronde Mine	1988-1993
- Exploration geologist, Laronde Mine	1993-1995
- Exploration coordinator, Placer Dome	1995-1997
- Senior Exploration Geologist, Barrick Exploration	1997-1998
- Exploration Manager, McWatters Mining	1998-2003
- Chief Geologist Sigma Mine	2003
- Consulting Geologist	2003 to present.

- I am responsible for the preparation of Section(s) 12.0, 13.0 and 14.0 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Magog, Quebec

"Original Document, Revision 03 signed and sealed by Tracy .J. Armstrong, P.Geo."

Tracy J. Armstrong, P.Geo.
Senior Associate Geologist
P&E Mining Consultants Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Marinus Andre de Ruijter, of Delta, British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Senior Metallurgical Engineer with Wardrop Engineering Inc. with a business address at 800-555 West Hastings Street, Vancouver, B.C., V6B 1M1
- I am a graduate of the University of Witwatersrand, Johannesburg, South Africa (B.Sc. [Mathematics and Physics], 1969; B.Eng., 1973; M.Eng., 1979).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration #31031).
- I have practiced my profession continuously since graduation, except during the years 2000 and 2004.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to this project included various investigations and studies involving sulphide mineral flotation as pyrite and copper and/or lead and/or zinc minerals.
- I am responsible for the preparation of Sections 16.0, 18.2, and 18.10 of this technical report titled “NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study”, dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

“Original Document, Revision 03 signed and sealed by M.A. de Ruijter, P.Eng.”

M.A. de Ruijter, P.Eng.
Senior Metallurgist
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Eugene Puritch, P.Eng., of Brampton Ontario, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a President of P&E Mining Consultants Inc. with a business address at 2 County Court Blvd, Suite 202, Brampton, Ontario, L6W 3W8.
- I am a graduate of the Haileybury School of Mines (Mining Technologist, 1977) and undergraduate of Queen's University, (Mine Engineering, 1978).
- I am a member in good standing of Professional Engineers Ontario (License #100014010).
- I have practiced my profession continuously since 1978.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Technical Report includes is as follows:

- Mining Technologist - H.B.M. & S. and Inco Ltd.	1978-1980
- Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd	1981-1983
- Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine	1984-1986
- Self-Employed Mining Consultant – Timmins Area	1987-1988
- Mine Designer/Resource Estimator – Dynatec/CMD/Bharti	1989-1995
- Self-Employed Mining Consultant/Resource-Reserve Estimator	1995-2004
- President – P & E Mining Consultants Inc.	2004-2009

- I am responsible for the preparation of Section 17.1 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009. In addition, I visited the Property during the period March 26 and 27, 2008.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Brampton, Ontario.

"Original Document, Revision 03 signed and sealed by Eugene Puritch, P.Eng."

Eugene Puritch, P.Eng.
President
P&E Mining Consultants Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Iouri Iakovlev, of Vancouver, British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Senior Mining Engineer with Wardrop Engineering Inc. with a business address at 800-555 West Hastings Street, Vancouver, B.C., V6B 1M1.
- I am a graduate of the Serbian Industrial University, Novokuznetsk, Russia (M.Sc. Honours, Mining Engineering, 1983).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration #32213).
- I have practiced my profession for more than 15 years.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to mine engineering includes over 15 years of mine engineering and mine operations experience.
- I am responsible for the preparation of Sections 17.2 and 18.1 (excluding backfill and ventilation) of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

"Original Document, Revision 03 signed and sealed by Iouri Iakovlev, P.Eng."

Iouri Iakovlev, P.Eng.
Senior Mining Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Hasan Ozturk, of Vancouver, British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Mining Engineer with Wardrop Engineering Inc. with a business address at 800-555 West Hastings Street, Vancouver, B.C., V6B 1M1.
- I am a graduate from the University of Alberta with a Ph.D. (Mining) in 2005. In addition, I obtained a B.Sc. in 1997 and a M.Sc. in 2000 from the Middle East Technical University, Turkey.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration #32327).
- I have practiced my profession as a Mining Engineer since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to ventilation includes ventilation circuit design and fan selection.
- I am responsible for the preparation of the ventilation portions of Section 18.1 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

"Original Document, Revision 03 signed and sealed by Hasan Ozturk, Ph.D., P.Eng."

Hasan Ozturk, Ph.D., P.Eng.
Mining Engineer
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Miloje Vicentijevic, of Vancouver British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am the Manager of Mining (Vancouver) for Wardrop Engineering Inc. with a business address at #800 – 555 West Hastings St., Vancouver, BC, V6B 1M1.
- I am a graduate of the University of Belgrade (B.Sc. in Mining Engineering, 1990) and University of Alberta (M.Eng. in Engineering Management, 2003).
- I am a member in good standing of the Association of Professional Engineers, Geologists, and Geophysicists of Alberta (#75678).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to this project includes mine planning and design experience.
- I am responsible for the preparation of Section 18.1.7 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009. I rely on technical data and results provided by Barnard Foo (P.Eng.) whom I supervised during the preparation of this report.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

"Original document, Revision 03 signed and sealed by Miloje Vicentijevic, P.Eng., M.Eng."

Miloje Vicentijevic, P.Eng., M.Eng.
Manager of Mining (Vancouver)
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Friedman, P.Eng., of Vancouver, British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Project Engineer with Knight Piésold Ltd. with a business address at Suite 1400, 750 West Pender Street, Vancouver, BC.
- I am a graduate of McGill University (B.Eng., 2003).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #32571).
- I have practiced my profession continuously for over five years since my graduation from university.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to mine waste and water management includes over five years of continuous work in the discipline.
- I am responsible for the preparation of Section 18.4, Section 18.7, and portions of Sections 18.9 and 19.0 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009. In addition, I visited the Property during the periods March 20 to 22, 2007, and August 10 to 15, 2007.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

"Original Document, Revision 03 signed and sealed by Daniel Friedman, P.Eng."

Daniel Friedman, P.Eng.
Project Engineer
Knight Piésold Ltd

CERTIFICATE OF QUALIFIED PERSON

I, Robert A. Mercer, of North Bay, ON, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Specialist Engineer with Knight Piesold Ltd. with a business address at 1650 Main St. W on North Bay, ON (P1B 8G8).
- I am a graduate of Queen's University (Ph.D, 1999).
- I am a member in good standing of the Professional Engineers of Ontario (No. 90521915).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to Mining Rock Mechanics includes over ten years of continuous work in the discipline.
- I am responsible for the preparation of Section 18.5 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at North Bay, ON.

"Original Document, Revision 03 signed and sealed by Robert A. Mercer, Ph.D., P.Eng.

Robert A. Mercer, Ph.D., P.Eng.
Specialist Engineer
Knight Piesold Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Peter Wells, of Vancouver, British Columbia, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a General Manager and Study Manager with Wardrop Engineering Inc. with a business address at 800-555 West Hastings Street, Vancouver, B.C., V6B 1M1.
- I am a graduate of Technicon Witwatersrand (NDT, 1978) and University of South Africa, (B.Comm, 1991).
- I am a Fellow in good standing with the South African Institute of Mining and Metallurgy (Registration#703616).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to Sections 18.6 and 18.9 includes 25 years of working in mining operations and as a consultant.
- I am responsible for the preparation of Sections 18.6 and portions of 18.9 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study" dated September 21, 2009. In addition, I visited the Property on February 17, 2008.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Vancouver, British Columbia

"Original Document, Revision 03
signed by Peter Wells, SAIMM (Fellow)"

Peter Wells, SAIMM (Fellow)
General Manager and Study Manager
Wardrop Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Scott Cowie, of London, United Kingdom, do hereby certify that as a co-author of this **NI 43-101 TECHNICAL REPORT – PITARRILLA PROPERTY PRE-FEASIBILITY STUDY**, dated September 21, 2009, I hereby make the following statements:

- I am a Senior Mining Engineer with Wardrop Engineering Inc., with a business address at Ground Floor, Unit 2, Apple Walk, Kembrey Park, Swindon, UK, SN2 8BL.
- I am a graduate of the University of Queensland (Bachelor of Mining, 2001)
- I am a member in good standing of the Australian Institute of Mining and Metallurgy (Member Number: 206253).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to financial modelling includes completion of scoping study to feasibility study mineral project evaluations for base, industrial, and precious metals.
- I am responsible for the preparation of Section 18.11 of this technical report titled "NI 43-101 Technical Report – Pitarrilla Property Pre-feasibility Study", dated September 21, 2009.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 21st day of September, 2009 at Swindon, UK

"Original Document, Revision 03 signed and sealed by Scott Cowie, MAusIMM"

Scott Cowie, MAusIMM
Senior Mining Engineer
Wardrop Engineering Inc.