NI 43-101 Technical Report Mineral Resource and Preliminary Economic Assessment of the Sunshine Silver Mine Project

Big Creek, Idaho

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Effective Date of Resource: October 2012
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I Rex Clair Bryan, Ph.D., do hereby certify that:

a. I am currently employed by Tetra Tech, Inc. at:

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b. This certificate applies to the Technical Report titled "NI 43-101 Technical Report Mineral Resources and Preliminary Economic Assessment of the Sunshine Silver Mine Project" (Technical Report), effective December 21, 2012, issued December 21, 2012.

c. I graduated with a degree in Engineering (BS with honors) in 1971 and a MBA degree in 1973 from the Michigan State University, East Lansing. In addition, I graduated from Brown University, Providence, Rhode Island with a MS degree in Geology in 1977, and The Colorado School of Mines, Golden, Colorado, with a graduate degree in Mineral Economics (Ph.D.) in 1980. I have worked as a resource estimator and geostatistician for a total of thirty-one years since my graduation from university; as an employee of a leading geostatistical consulting company (Geostat Systems, Inc. USA), with large engineering companies such as Dames and Moore, URS, and Tetra Tech and as a consultant for more than 30 years. I am a Registered Member (#411340) of the Society for Mining, Metallurgy, and Exploration, Inc. (SME). I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

d. I have visited and inspected the subject property from February 9, 2012 to February 10, 2012. I am responsible for sections 1-12, 14, 20, 23, 24 and jointly responsible with Richard Kunter and Neil Prenn for sections 25-26 and with Neil Prenn for Sections 27-28 of this Technical Report.

e. I am independent of Sunshine Silver Mines Corporation, as set out in section 1.5 of NI 43-101.

f. I have had no prior involvement with the properties that are the subject of the Technical Report.

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

h. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

Dated this 21st day of December 2012

"Rex Clair Bryan, Ph.D." - Signed

Signature of Qualified Person

Rex Clair Bryan, Ph.D.

Print name of Qualified Person

Richard S. Kunter, MS, FAusIMM (CP) QP

Metallurgical Engineer

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I, Richard S. Kunter, BS (Met Engr.), MS (Met Engr.) do hereby certify that:

a. I am currently employed as Chief Process Engineer-Mining & Metals by Samuel Engineering, Inc. at:

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Greenwood Village, Colorado 80111

b. This certificate applies to the Technical Report titled "NI 43-101 Technical Report Mineral Resources and Preliminary Economic Assessment of the Sunshine Silver Mine Project" (Technical Report), effective December 21, 2012, issued December 21, 2012.

c. I graduated with a Metallurgical Engineering degree in 1967 from the University of Idaho. I have practiced in my profession since 1967. I am a member in good standing of: Fellow, Australasian Institute of Mining and Metallurgy; Mining and Metallurgical Society of America; Society of Mining, Metallurgy, and Exploration (SME); Minerals, Metals and Materials Society (TMS); American Society for Metals; and Society of Sigma Xi. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

d. I have personally visited and inspected the subject property from February 22-25, 2012 and March 20-22, 2012. I am responsible for sections 13 and 17, and jointly responsible with Rex Bryan and Neil Prenn for sections 25-26 of this Technical Report.

e. I am independent of Sunshine Silver Mines Corporation, as set out in Section 1.5 of NI 43-101.

f. I have had no prior involvement with the properties that are subject to the Technical Report.

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

h. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

Dated this 21st day of December 2012

"Richard S. Kunter" - Signed

Signature of Qualified Person

Richard S. Kunter

Print name of Qualified Person

Neil B. Prenn, P.E.

Mining Engineer

Mine Development Associates

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I, Neil B. Prenn, P.E., do hereby certify that:

a. I am currently employed as Principal Engineer by Mine Development Associates at:

210 S. Rock Boulevard

Reno, Nevada 89502

b. This certificate applies to the Technical Report titled "NI 43-101 Technical Report Mineral Resources and Preliminary Economic Assessment of the Sunshine Silver Mine Project" (Technical Report), effective December 21, 2012, issued December 21, 2012.

c. I graduated with an Engineer of Mines degree in 1967 from the Colorado School of Mines. I have worked as a mining engineer for a total of 46 years since my graduation from university. I am a Registered Professional Mining Engineer in the state of Nevada (#7844) and a member of the Society of Mining Engineers and Mining and Metallurgical Society of America. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. My relevant work experience includes 16 years with Cyprus Mines Corporation, two years with California Silver, and 24 years with Mine Development Associates, completing numerous resource and reserve calculations.

d. I have personally visited and inspected the subject property from May 8-11, 2012. I am responsible for sections 15, 16, 18, 19, 21, 22, and jointly responsible with Rex Bryan and Richard Kunter for sections 25-26 and with Rex Bryan for sections 27-28 of this Technical Report.

e. I am independent of Sunshine Silver Mines Corporation, as set out in Section 1.5 of NI 43-101.

f. I have had no prior involvement with the properties that are subject to the Technical Report.

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

h. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

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

Dated this 21st day of December 2012

"Neil Prenn" - Signed

Signature of Qualified Person

Neil Prenn, P.E.

Print name of Qualified Person

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

1.0 SUMMARY

1.1 Introduction

The Sunshine Mine Project is a silver project located within the well-known Coeur d'Alene Mining District in northern Idaho and is owned by Sunshine Silver Mines Corporation (SSMC), based in Denver, Colorado. This technical report (Technical Report) has been prepared for SSMC by, or under the supervision of, Qualified Persons within the meaning of National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101) in support of SSMC's disclosure of scientific and technical information for the Sunshine Mine Project.

SSMC contracted Tetra Tech, MTB Project Management Professionals, Inc. (MTB) and Samuel Engineering, Inc. (SE) of Denver, Colorado, and Mine Development Associates (MDA) of Reno, Nevada to prepare a Preliminary Economic Assessment (PEA) and Independent Technical Report, compliant with NI 43-101 for the Sunshine Mine Project.

Tetra Tech completed the resource estimate and environmental, permitting, and related sections of this Technical Report; SE completed the metallurgy and process sections; and MTB completed estimates and potential economic evaluation, and compiled remaining sections with contributions from Tetra Tech, SE, and MDA. MTB's work was completed under the supervision of Neil Prenn, a Qualified Person within the meaning under NI 43-101.

The Sunshine Mine is an existing underground mine with more than 110 years of operating history. The mine has largely been suspended throughout a succession of bankruptcies. Since acquiring the property in 2010, SSMC has embarked on a program to rehabilitate the mine and supporting infrastructure in preparation for executing its plan to return the mine to commercial production in the third quarter 2014.

The sections following summarize the scope, methodologies, and results of this PEA, including recommendations for future work.

1.2 Location

The Sunshine Mine is located in northern Idaho at Big Creek, between Kellogg and Wallace. It is 37 miles east of Coeur d'Alene along U.S. Interstate 90 (I-90).

1.3 Ownership

The Sunshine Mine is located in the Coeur d'Alene Mining District, which contains one of the world's largest concentrations of silver. The Sunshine Mine property is comprised of patented and unpatented mining claims which are both owned and leased from third parties, for a total project area of 9,637 acres. Some of the properties are subject to royalities with the royalty terms varying for each property. The majority of the Sunshine Mine property is subject to a net smelter return (NSR) royalty under a settlement with the United States Government and the Coeur d'Alene Indian Tribe for remediation, restoration, and other actions to address environmental damages to the Couer d'Alene River and other natural resources in the Silver Valley. Table 1.1 sets out the various mineral rights that comprise the Sunshine Mine property and their acreage.

Table 1.1 Property Mineral Rights and Claims

Property	Owner	Patented Claims	Unpatented Claims	Acres
Sunshine	SSMC	165	118	4313
Sun South	SSMC	0	158	3111
Silver Hill	SSMC	0	28	578
CAMP Project	SSMC (below -900' elevation)	20	12	402
Metropolitan	Metropolitan Mines Corporation	2	70	1020
Chester	Chester Mining Company	6	0	106
Bismark	Chester Mining Company	3	0	62
Mineral Mountain	Mineral Mountain Mining and Milling Company	4	0	45
TOTAL		**200**	**386**	**9637**

1.4 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The Sunshine Mine is located 37 miles east of Coeur d'Alene, Idaho along I-90. From I-90, the property is accessed at the Big Creek exit south approximately two and one-half miles via a paved county road which parallels Big Creek. The property is located about four and one-half miles southeast of the town of Kellogg, Idaho which hosts a full complement of services and is home to many of the mine staff. The closest major airport and metropolitan center are located in Spokane, Washington, approximately 90 miles west of Kellogg.

The geographic coordinates of the Sunshine Mine are latitude 47°30'6" north and longitude 116°4'10" west.

The Sunshine Mine is located in the Big Creek Valley at an approximate elevation of 2,600 feet above sea level. The topography is typical of northern Idaho's countryside, hilly to mountainous and forested. Sunshine's main production shaft, the Jewell Shaft, and the mill are located above the base of a steep mountain, while the hoist room and other infrastructure facilities are located on a relatively level piece of property at the base of the mountain.

Climate in the area is considered to be typical of the northern-US with snow, rain, and fog in the winter. Snowfall in the winter and the changing topography can restrict access to some surface facilities at higher elevations. Average rainfall in the area is approxmately 33 inches annually.

Kellogg, Idaho, with a population of approximately 2,000, and Wallace, Idaho, with a population of 784, are the two nearest towns to the mine and are home to many of the mine staff. The mining history of the Idaho Silver Belt ensures a ready source of skilled and unskilled labor. Efforts are made to stimulate the local economies as much as possible, with the local area having numerous vendors that supply services to the mining industry such as welding, steel

supply, transportation, and project consumables. Spokane, Washington is the largest city in the area which has an international airport and many industry supplies and services are obtained here.

Electrical power is supplied by AVISTA, a large northwest United States power supplier with long historical ties to the mining industry in the Coeur d'Alene district. The district is tied to the main northwest power grid, and outages are rare. Power supply is ample for the life of the Sunshine Mine. The main power source for the mine is a power line that parallels Big Creek Road and terminates at the AVISTA substation on the Sunshine Mine property. From the substation, power is distributed to numerous smaller substations throughout the property.

1.5 History

The Sunshine Mine property had its beginning in 1884, when the Blake brothers, Dennis and True Blake, arrived from Maine and staked the Yankee Load (sic) mining claim on the east slope of Big Creek. Throughout its 128-year history, the Sunshine Mine was able to remain in production by continually advancing new resources. Between 1934 and 2007, published historical reserves have remained between 20 and 50 million contained ounces. A total of 365 million ounces have been produced from mines on what is now the Sunshine Mine property.

1.6 Geologic Setting and Mineralization

The district is hosted by the rocks of the Precambrian Belt Supergroup. These Middle Proterozoic age sedimentary rocks were deposited approximately 1.47 to 1.6 billion years ago. At various times, these rocks were faulted, leached, altered, and re-mineralized. The Belt Supergroup has been divided into the Pre-Ravalli group, Ravalli group, Piegan group, and Missoula group. Within the Coeur d'Alene Mining District, rocks of the Pre-Ravalli, Ravalli, and Piegan groups can be found. The formations comprising the Ravalli group are the preferred host rocks for silver mineralization in the district. The Ravalli formations are from older to younger Burke Formation, Revett Formation, and St. Regis Formation.

Ore deposits of the Coeur d'Alene Mining District occur in veins hosted in weakly-metamorphosed sedimentary rocks of the Belt Supergroup. Most of the ore production is from the Revett and St. Regis Formations of the Ravalli group. This thick sequence (up to 12.4 miles) of middle Proterozoic-age strata covers a large area of northern Idaho, western Montana, and southeastern British Columbia. The sedimentary rocks are predominately fine-grained siliciclastics with subordinate carbonate-bearing units.

Over 30 veins have been named and mined at the Sunshine Mine. The Sunshine and Chester Veins have each produced over 90 million ounces of silver. The majority of veins strike east-west and dip about 65° to the south. Locally, dips range from 45° to 90°. Strike lengths locally exceed 2,000 feet and dip lengths are two to three times greater than the strike length. Major veins are located between the faults at an angle of 25° to the bounding faults. Veins vary in width from a few inches to over 30 feet, but are typically between one to five feet thick. Ore minerals include tetrahedrite and galena, with siderite and quartz as the principal gangue minerals. Accessory minerals include bournonite, pyrargyrite, and magnetite.

1.7 Deposit Types

The Sunshine Mine mineral deposits are narrow, high-grade vein deposits, which characteristically strike east-west and dip steeply (average 65°) to the south. The combination of

faults, folds, fractures, and favorable host rocks created suitable conditions for mineral emplacement by silver-rich and silver-base metal veins. Figure 8. shows the general relationship between the principal productive veins and the four major faults. Historically, underground drilling and drifting were the most productive exploration tools.

1.8 Exploration

The objectives of the current exploration program at the Sunshine Mine are to discover new high-grade veins and ore shoots in areas that already have nearby development, explore for new large veins in unexplored or under-explored areas, and to systematically replace reserves as they are mined. All of the exploration work carried out at the Sunshine Mine created historic resources. It is necessary to describe this historic exploration work as it includes the methods utilized by prior owners/operators Sunshine Mining Company, Sterling, and current owner SOP. Exploration work presently underway is using the same methods as practiced by Sunshine Mining Company up through 2001, except that defined resource and reserve categories will now be classified in accordance with the CIM Definition Standards on mineral resources and mineral reserves (CIM Definition Standards), as most recently adopted by the CIM Council on November 27, 2010. Mine staff completed surveys and exploration work in the past and this practice continues incorporating new advancements in practice, method, and technology. A significant example of this, the Sterling Tunnel, was driven to join the Sunshine and ConSil Tunnels and has allowed underground diamond drilling exploration to be resumed in the Upper Country mine area.

1.9 Drilling

The current drill database contains information from approximately 3,536 underground drill holes. Additional drill holes are being identified and added to the master database. The longest underground hole is 3,000 feet. It is not uncommon for holes to be 1,500 to 2,000 feet long. Long underground exploration holes are required to locate structures and veins because the majority of historical development, except in the West Chance, has been on the vein structures themselves; thus, drilling platforms for shorter holes at appropriate angles to the targets have not been available.

The current drilling for exploration, delineation, and development conducted at SSMC has been performed with diamond core drills. Work is continuing with a local contract core drilling company, Dynamic Drilling Inc. from Osburn, Idaho. They currently operate one Hagby Onram-1000 diamond drill underground in the Sterling Tunnel. Down-hole surveys are attempted on all diamond drill holes. The primary survey tool is a Reflex EZ-AQ multi-shot down-hole survey camera. Core diameters range from 1.062 to 1.875 inches. Core recovery is generally very good, exceeding 90%. Core recovery can be difficult in certain faulted or shear areas. The diamond drillers will change from wireline tools to conventional tools before encountering proven areas of loss, which significantly improves recovery.

All drill hole and sample information is stored in an Access® database for reporting purposes and in a Gemcom database for three dimensional (3D) evaluations in support of resource modeling. When drill hole samples are used for polygonal or accumulation methods of resource modeling, they are calculated back to true horizontal thickness. Diamond drill holes are typically designed to intersect mineralization as close to perpendicular as possible. Down-hole directional surveys are conducted on all drill holes, since hole deviation is common. A Reflex EZ-AQ multi-shot down-hole survey instrument is used for deviation surveys.

To date, three new vein structures have been defined with drilling from the Sterling Tunnel elevation. Two new silver-copper veins have been defined in the immediate hanging wall of the historic Sunshine Vein, named the “West Chance Link Vein” and the “South Yankee Boy Split Vein”. A total of 16 holes have been drilled by SSMC targeting the West Chance Link vein and all have encountered silver mineralization. Additionally, a new lead-silver vein named the “10 Vein” has been recently discovered 200 feet within the footwall of the Sunshine Vein. All veins carry economic silver-copper or lead-silver values. Drilling is continuing to define the vertical and lateral limits of the new vein structures.

1.10 Mineral Resource Estimates

This mineral resource estimation was completed by Tetra Tech in MicroMine® mining software utilizing data supplied by SSMC. This estimation is the first time a digital resource estimate has been calculated for the Sunshine Mine and is a direct result of an extensive data digitizing effort by SSMC. Tetra Tech was commissioned by SSMC to develop a digital 3D estimation that includes all available information and can be evaluated at various cutoffs.

Utilizing drill hole and channel assays data, Tetra Tech created best fit vein surfaces and estimated mineral resources along those vein surfaces. Ordinary Kriging was used to estimate 3D points along a string type block model for 38 veins. The results of the mineral resource estimate are tabulated in Table 1.2. This mineral resource has been diluted to a fixed mine width of 6.5 ft and is an insitu resource estimate. A base-case cutoff has been applied but no other economic parameters have been considered and all metal recoveries are assumed to be 100%. Royalties do exist in certain areas of the mine and have not been considered for this mineral resource estimate.

Table 1.2 Mineral Resource Estimate Sunshine Silver Mine

Resource Class	Cutoff Ag Opt Diluted	Tons Diluted	Grade Ag Opt Diluted	Ag Contained Ounces	Cu %	Pb %	Zinc %
Measured	10	1,215,000	24.6	29,890,000			
Indicated	10	1,960,000	21.6	42,410,000			
Measured + Indicated	10	3,175,000	22.8	72,300,000			
Inferred	10	9,115,000	24.4	222,050,000	0.26	0.35	0.02

1.11 Sources of Data

Assay data necessary to facilitate this resource estimate was derived from three sources: drill holes, level drift channel samples, and stope channel samples. Drill hole assays were provided by SSMC from a collated drill hole database, encompassing historic and modern drilling. Level drift channel sampling was digitized from historic level plan maps. Stope channel samples were digitized from historic stope production sheets. In addition to assay data sources, level plan vein sketches, development triangulations, and vein map long sections were essential for resource estimation.

1.12 Mining

A mine plan and production schedule were completed by MDA using the mineral resource estimates (with an effective date of October 2012) completed by Tetra Tech.

MDA completed a mine production schedule using a cutoff grade of 10 opt Ag. Alimak slusher mining is expected to be used for areas where stopes are generally less than 6.5 feet and mechanical cut and fill mining are expected to be used for areas where the stopes are generally greater than 6.5 feet. The majority of the ore and waste is expected to be delivered to the surface via the hoist in the Jewell Shaft with other material coming from the Sterling Tunnel.

A mine development schedule was also generated showing the estimated amount of development needed before mining can commence, and thereafter to sustain the planned mine production. Before development work can begin, the Silver Summit shaft must be rehabilitated to provide a secondary escapeway for the mine and a new ventilation system must be installed. The ventilation system requires two new vent raises and a bypass drift on the 3100 level. It is expected that the Silver Summit rehabilitation work will be completed during the second quarter of 2014 and the ventilation system development will be completed at mid-year 2016. According to current plans, mining of the Upper Country can begin immediately.

Mine ore production is scheduled to start during the second quarter of 2014 however, this material is planned to be stockpiled until the mill start-up scheduled for the third quarter of 2014. Production from the mine is limited to the Upper Country stopes until the ventilation and secondary escapeway are completed by mid-year 2016. Mine production is expected to increase when the stopes are available to sustain a 1,000 tpd operation currently scheduled to occur in the second half of 2016. The mine development schedule is shown in Table 16.5 and the mine production schedule is shown in Table 16.6.

Mining is presented in more detail in Item 16.0.

1.13 Processing

Preliminary processing assumptions are based on a flowsheet in which the process plant will produce two flotation concentrates; one silver and copper, and the other lead. While the grade and mineralogical characteristics for the silver are fairly constant, there are areas of the mine where lead is more prevalent and the amount of lead feed to the concentrator can be expected to be variable over time.

The concentrates will be produced from the crushing, grinding, and flotation of freibergite-rich ore. The anticipated flotation system will use three rougher and scavenger systems with separate, three-stage silver cleaning and lead cleaning systems. Once the ore has been ground by the ball mill, it then proceeds through the three rougher-scavenger flotation systems in series. All tailings then depart from the third-stage scavenger and report to the tailings, thickening, filtration, and disposal systems.

The concentrates are expected to be processed initially by offsite commercial refiners and then in the second year of production the silver-copper concentrates will be processed at an onsite refinery, which will produce copper cathodes and silver doré. Lead concentrate will continue to be refined by an offsite smelter.

1.14 Infrastructure

Located two and a half miles from I-90, the Sunshine Mine and surface facilities (including tailing storage facility (TSF) and waste rock storage facility (WRSF)) are situated within a narrow valley adjacent to Big Creek. Because of the age of the current infrastructure, demolition and replacement of certain facilities is planned allowing for the construction of, among other facilities, a new process plant while improving use of available area.

Process/industrial water for the mine is available via four water rights including three directly from Big Creek. Potable water is provided via a local water district. Power is supplied from a dedicated power line and is maintained by a local utility company. Back-up power will be provided via generators planned for installation. Surface water run-on and septic effluent are effectively and properly managed reducing the need for further treatment. Onsite fire protection is provided from water drawn directly from Big Creek. Supplemental fire protection is provided via the local fire district (Kellogg Fire Department).

1.15 Environmental

Environmental permitting, compliance, and stewardship are important considering the location and long operational history of the Sunshine Mine. With the proximity to several watersheds (including Big Creek and the South Fork of the Coeur d'Alene River (South Fork)), areas of population (four and one half miles from Kellogg, Idaho), other mining operations, and the Bunker Hill Mining and Metallurgical Complex Superfund Site, maintaining protection of the environment and regulatory compliance is critical. For the purposes of this PEA and to gain a better understanding of Sunshine Mine's environmental component, historical compliance, and permitting data were reviewed and studies were conducted including:

- Historical and current monitoring and compliance reporting
- Historical studies by prior operators
- Surface and groundwater analyses from sources within and adjacent to the operation
- Waste rock and mine tailings analyses from sources associated with the Sunshine and ConSil Mines
- Current permitting requirements

The results of these efforts confirmed there are no environmental issues existing or anticipated that could materially impact the ability to reopen the Sunshine Mine. As the operation progresses, SSMC will be required to maintain or renew existing or acquire new approvals and permits. However, at this time all environmental permits, agreements, and approvals necessary to commence surface and subsurface operations are in place.

1.16 Project Economics

1.16.1 Capital Costs

Capital costs were estimated for the proposed mining and processing operations of the Sunshine Mine Project. The total estimated initial cost to design, procure, construct, and commission the facilities described in this Technical Report is $130.3 million, in third quarter 2012 United States dollars (USD). A contingency of $12.8 million has been included in the

capital cost. This contingency is based on the level of definition that was used to prepare the estimate.

The total life of mine (LOM) sustaining capital costs are estimated to be approximately $215.4 million. Sustaining capital costs mainly relate to equipment replacement at end of useful life, additional equipment required to meet production levels, and phased expansion of facilities, such as the TSF. Sustaining capital costs are detailed in Table 22.6.

The capital cost estimate has been developed to a level sufficient to assess/evaluate the project concept, various development options, and the potential overall project viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy in the range of plus/minus 35 percent.

Table 1.2 summarizes the capital costs by major area.

Capital costs are discussed in more detail in Item 21.0.

Table 1.2 Summary of Initial Capital Costs

Description	Cost	Total
Mine	21,830,000	
Crushing / Ore Handling	4,463,700	
Concentrator	15,518,500	
Tailings	1,709,400	
Tailings Paste Backfill (Surface Facility)	2,488,300	
Reagents	554,400	
Utilities	4,714,300	
General & Infrastructure	5,244,600	
Total Contracted Directs		**56,523,100**
Process Facilities Contractor Indirects	1,924,600	
Construction Equipment	684,200	
Freight and Duties	801,300	
EPCM - Process Facilities	5,204,000	
Commissioning Support	255,000	
Third Party Testing Services	240,000	
Vendor Representatives	353,300	
Spare Parts and Initial Fills	2,600,300	
Total Contracted Indirects		**12,062,600**
Mining and Ancillary Equipment	13,164,000	
Preproduction Mine Development	17,768,200	
Mobile Equipment & Light Vehicles	770,000	
Other (equipment, furniture, software, etc)	150,000	
Temporary Facilities	200,000	
Medical, Security and Safety	25,000	
Total Owner Direct Cost		**32,077,200**
Client Management	2,377,700	
Preproduction Employment & Training	7,010,000	
Utilities/Site Overhead Expenses	2,660,000	
Insurance	2,025,000	
Outside Services (Legal & Accounting)	400,000	
Corporate Overhead (Travel and Expenses)	2,400,000	
Total Owner Indirect Cost		**16,872,700**
Subtotal Project Cost		117,535,600
Contingency	12,805,700	
Total Initial Capital Costs (USD)		**$ 130,341,300**

1.16.2 Operating Costs

In connection with this PEA, operating costs have been estimated for the following areas of the Sunshine Mine Project: mining, processing, refining, general and administrative (G&A), and mine reclamation and closure. Operating costs are estimated to average approximately $77 million per year over the estimated life of mine, or 27.5 years. With an estimated LOM throughput of 9.65 million tons of ore feed to the process plant, the average operating cost per ton of ore is estimated to be $219.39 per ton. A 20% contingency allowance, or approximately $203 million has been included in mine operating costs as shown in Table 21.4.

Table 1.3 LOM Operating Cost Summary

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore
Mining	1,287,050,608	46,801,840	133.34
Processing	238,809,316	8,683,975	24.74
Refining	380,902,297	13,850,993	39.46
General & Administration	209,552,915	7,620,106	21.71
Mine Reclamation & Closure Cost	1,350,000	**	0.14
Total Operating Cost (USD)	**$ 2,117,665,135**	**$ 76,956,914**	**$ 219.39**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*

*** No Average Annual Cost indicated as cost is considered incurred after LOM in Year 28 and after.*

1.16.3 Mine Development Costs

Mine development during preproduction will be performed by a contractor, as will construction of a bypass drift on the 3100 Level and all vertical development during production years. All other development during production will be performed by SSMC employees. LOM mine development is detailed in Table 16.5.

Estimated preproduction mine development costs ($17.8 million) are included within initial capital costs discussed above in Table 1.2 and depreciated accordingly. A 5% contingency allowance has been included in estimated preproduction mine development costs. Estimated production mine development costs totaling $456.5 million for production years 1-26 are expensed in the year in which the related ore is mined at an average rate of approximately $17 million per year. A 10% contingency allowance, or $40.6 million has been included in estimated mine development costs.

1.16.4 Economic Evaluation

Long term metals prices of $25.00/oz, $2.75/lb, and $0.85/lb were applied to the payable silver, copper, and lead quantities shown in Table 1.4 below, respectively.

The results of this PEA estimate an internal rate of return (IRR) of 28.6% for the Sunshine Mine Project. Assuming a discount rate of five percent over an estimated mine life of 27.5 years, the after-tax net present value (NPV) is estimated to be approximately $732 million. The NPV using the same discount rate of five percent is expected to turn negative with silver price at or below $15.79/oz. Based on the results of this PEA, payback is estimated to occur late in the fourth year of mine life, approximately 3.9 years after the start of production.

This PEA is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable

them to be categorized as mineral reserves, and there is no certainty that the PEA results described herein will be realized. Mineral resources that are not mineral reserves have no demonstrated viability.

Table 1.4 Economic Model Inputs

Description	Values			
Construction Period	15 months			
Preproduction Period	1.25 years			
Mine Life (after Preproduction)	27.5 years			
LOM Ore (ton)	9,652,519			
LOM Silver Concentrate (tons)	131,997			
LOM Lead Concentrate (tons)	232,309			
LOM Silver Con Grade (% Cu)	10.10%			
LOM Silver Con Grade (% Pb)	8.69%			
LOM Lead Con Grade (% Cu)	0.61%			
LOM Lead Con Grade (% Pb)	4.60%			
Avg. Annual Process Production Silver (troy ozs)	8,391,830			
Avg. Annual Process Production Copper (lbs)	1,072,549			
Avg. Annual Process Production Lead (lbs)	1,611,990			
Market Price	***Year 1***	***Year 2***	***Year 3***	***Year 4 >***
Silver Price ($/Toz Ag)	*32.50*	*29.91*	*26.15*	*25.00*
Copper Price ($/lb Cu)	*3.40*	*3.10*	*2.75*	*2.75*
Lead Price ($/lb Pb)	*0.94*	*0.95*	*0.85*	*0.85*
Silver Price (Average LOM $/Toz Ag)	25.27			
Copper Price (Average LOM $/lb Cu)	2.77			
Lead Price (Average LOM $/lb Pb)	0.85			
Cost and Tax Criteria				
Estimate Basis	3rd Qtr 2012 USD			
Inflation/Currency Fluctuation	None			
Leverage	100% Equity			
Income Tax - Federal, Idaho state, and local *(effective tax rate including depletion)*	27.7%			
Depreciation	Straight Line			
Royalties				
Coeur d'Alene Tribe	7.0%			
Chester Mining Co*	4.0%			
Hecla Mining Co*	4.0%			
Metropolitan Mines Corp* **on select revenue*	4.0%			
Transportation Charges				
Truck/Rail Freight - Silver Con to Horne ($/wst)	181.40			
Road Freight - Lead Con (to Trail, British Columbia) ($/wst)	53.70			
Freight and Insurance - Silver Dore to Utah ($/Toz)	0.02			
Road Freight - Copper Cathode to Missouri ($/lb)	0.14			
Payment Terms				
Provisional upon Bill of Lading	90%			
Settlement within 3 months after Bill of Lading	10%			

A more detailed discussion of project economic performance is presented in Item 22.0.

1.17 Conclusions and Recommendations

1.17.1 Conclusions

The Qualified Persons for this Technical Report have made the following conclusions:

- Overall, the results of the PEA indicate that the Sunshine Mine Project is a robust silver project at this stage of development and warrants further work toward the next stage of development. The exploration program continues to demonstrate the potential for future growth of the resource. Risks, as well as significant opportunities (identified in Item 25.2), can be evaluated in the feasibility stage of the project.
- All sources of historic data are internally consistent, have supported several decades of mining, and are suitable for use in resource estimation.
- The Sunshine Mine is complying with CIM Definition Standards best practice requirements for sample handling QA/QC.
- The sample preparation, security, and procedures followed by SSMC are adequate to support a mineral resource estimate.
- Assay data provided by SSMC was represented accurately and is suitable for use in resource estimation.
- Based on over 90 years of production history, there are no known factors which should have a negative economic effect on metallurgical recoveries.
- As the operation progresses and reclamation or environmental legislation/regulation requirements evolve, SSMC will be required to maintain, renew existing, or possibly acquire new approvals and permits. However, at this time, all environmental permits, agreements, and approvals necessary to commence surface and subsurface operations are in place.
- There are no existing or anticipated environmental issues that could materially impact the ability to reopen the Sunshine Mine.
- There are no known factors related to metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect the mineral resource estimates.
- This PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that these inferred resources will ever be upgraded or that this PEA will be realized. Mineral resources that are not mineral reserves have no demonstrated economic viability.

1.17.2 Recommendations

Based on results of this PEA, the authors recommend that SSMC complete a Feasibility Study (FS) to further define the Sunshine Mine Project in order to: more accurately assess its economic viability; support permitting activities; and, ultimately, support project financing should the FS results be positive.

A detailed breakdown of recommendations and estimated costs can be found in Item 26.0.

1.18 Units of Measurement

Historically all data and measurements at the mine have been collected in US standard units, feet (ft), tons, ounces per ton (opt), and cubic feet per ton (cu ft/ton). Currently units have

remained in US standard units to most easily incorporate historic data and avoid conversions of large datasets.

Easting and northing location coordinates are described in feet using a local mine grid; elevation coordinates are described in feet relative to sea level. The Jewell Shaft is located at approximately 74,527 East, -79,158 North and 2,722 feet above sea level. In discussion of location only, it is common for elevations to be referenced to the nearest level offset from a relative 0 elevation.

2.0 INTRODUCTION

The Sunshine Mine Project is a silver project in located within the well-known Coeur d'Alene Mining District in northern Idaho and is owned by SSMC, based in Denver, Colorado. This Technical Report has been prepared for SSMC by, or under the supervision of, Qualified Persons within the meaning of NI 43-101 in support of SSMC's disclosure of scientific and technical information for the Sunshine Mine Project.

Tetra Tech Inc. is responsible for completion of the mineral resource estimate available at the property of the Sunshine Mine. MTB Project Management Professionals, Inc. (MTB), a project management consulting firm, is responsible for the coordination and compilation of this Technical Report. MTB assisted in completing this Technical Report under the supervision of the Qualified Persons listed below, and under the overall supervision of Neil B. Prenn, P.E.

The below-listed Qualified Persons are responsible for the information provided in the indicated items.

Neil B. Prenn, P.E., of MDA, is responsible for the information provided in Items 15-16, 18, 19, 21, 22, and portions of 25-28.

Rex C. Bryan, Ph.D., of Tetra Tech, is responsible for the information provided in Items 1-12, 14, 20, and portions of 23-28.

Richard S. Kunter, MS, FAusIMM (CP) QP, of Samuel Engineering Inc., is responsible for the information provided in Items 13, 17, and portions of 25-26.

The purpose of this report is to provide an estimation of the mineral resources at the property that complies with NI 43-101 requirements and to provide and develop information to a level sufficient to assess/evaluate the project concept, various development options, and the potential overall project viability.

All data used in this mineral resource estimate was provided by SSMC technical staff. Assay data provided was derived from three sources: drill holes, level drift channel samples, and stope channel samples. Drill hole assays were provided by SSMC from a collated drill hole database, encompassing historic and modern drilling. Level drift channel sampling was digitized from historic level plan maps. Stope channel samples were digitized from historic stope production sheets. In addition to assay data sources, level plan vein sketches, development triangulations, and vein map long sections were essential for resource estimation.

3.0 RELIANCE ON OTHER EXPERTS

Information relied upon in producing this Technical Report was, in part, provided by the following SSMC staff and consultants for the Sunshine Mine.

- Chief Mine Geologist, Mr. Greg Nickel
- Chief Engineer, Mr. Guy Sande
- Senior Resource Geologist, Mr. William Hudson
- Regional Exploration Manager, Mr. Hugh Smith
- Chief Financial Officer, Mr. Roger Johnson

It is important to note that the status of the owned and leased patented and unpatented claims that constitute the Sunshine Mine property, as described in this report, has had some uncertainty in the past due to the bankruptcy of Sterling. SSMC believes all issues have been adequately resolved. Tetra Tech has relied on SSMC's representation of the ownership of the property titles, mining claims, and licenses to produce, and transfer of permits from the previous operator. No attempt was made to confirm the legality of licenses conferring the rights to mine, explore, and produce silver and other metal products and accordingly, the Qualified Persons disclaim any responsibility or liability in connection with such information or data. The authors are not qualified to express any legal opinion with respect to the property titles and current ownership and possible encumbrances, and therefore, disclaim direct responsibility for such titles and property status representations.

It is also important to note that federal, state, and local taxes, including depletion, were reduced by SSMC to an average life-of-mine effective tax rate for relative simplicity at this early phase of project assessment. The effective tax rate reportedly considered revenue, cost, and drawdown of resources throughout mine life. MTB and the Qualified Person for Item 22, Economic Analysis, have relied on SSMC's representation of the effective tax rate for use in the financial model. MTB and the Qualified Person, Mr. Neil Prenn, are not qualified to express any accounting and tax opinion with respect to the accuracy of the effective tax rate provided. Hence, MTB and the Qualified Person for Item 22 disclaim any responsibility or liability in connection with such information or data.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Ownership and Agreements

In May 2010, SOP acquired from Sterling, through Sterling's bankruptcy proceedings, the majority of the operating facilities and equipment at the Sunshine Mine. Also included in this purchase, was Sterling's lease from SPMI on the mine and a purchase option in the lease for title to the Sunshine Mine, which had been exercised by Sterling prior to the sale to SOP. SOP closed on the exercise of the purchase option of the lease from SPMI in July 2010 to obtain title to the mine and the facilities. SOP is owned by SSMC.

The Sunshine Mine is located in the Coeur d'Alene Mining District which contains one of the world's largest concentrations of silver. The Sunshine Mine property is comprised of patented and unpatented mining claims which are both owned and leased from third parties, for a total project area of 9,637 acres. Some of the properties are subject to royalities with the royalty terms varying for each property. The majority of the Sunshine Mine property is subject to an NSR royalty under a settlement with the U.S. government and the Coeur d'Alene Indian Tribe for remediation, restoration, and other actions to address environmental damages to the Coeur d'Alene River and other natural resources in the Silver Valley.

The Sunshine Mine property also includes the Metropolitan, Chester, Bismark, and Mineral Mountain properties that are leased by SSMC. Table 4.1 sets out the various mineral rights that comprise the Sunshine Mine property. Figure 4.1 shows SSMC mineral tenure.

Table 4.1 Property Mineral Rights and Claims

Property	Owner	Patented Claims	Unpatented Claims	Acres
Sunshine	SSMC	165	118	4313
Sun South	SSMC	0	158	3111
Silver Hill	SSMC	0	28	578
CAMP Project	SSMC (below -900' elevation)	20	12	402
Metropolitan	Metropolitan Mines Corporation	2	70	1020
Chester	Chester Mining Company	6	0	106
Bismark	Chester Mining Company	3	0	62
Mineral Mountain	Mineral Mountain Mining and Milling Company	4	0	45
TOTAL		**200**	**386**	**9637**

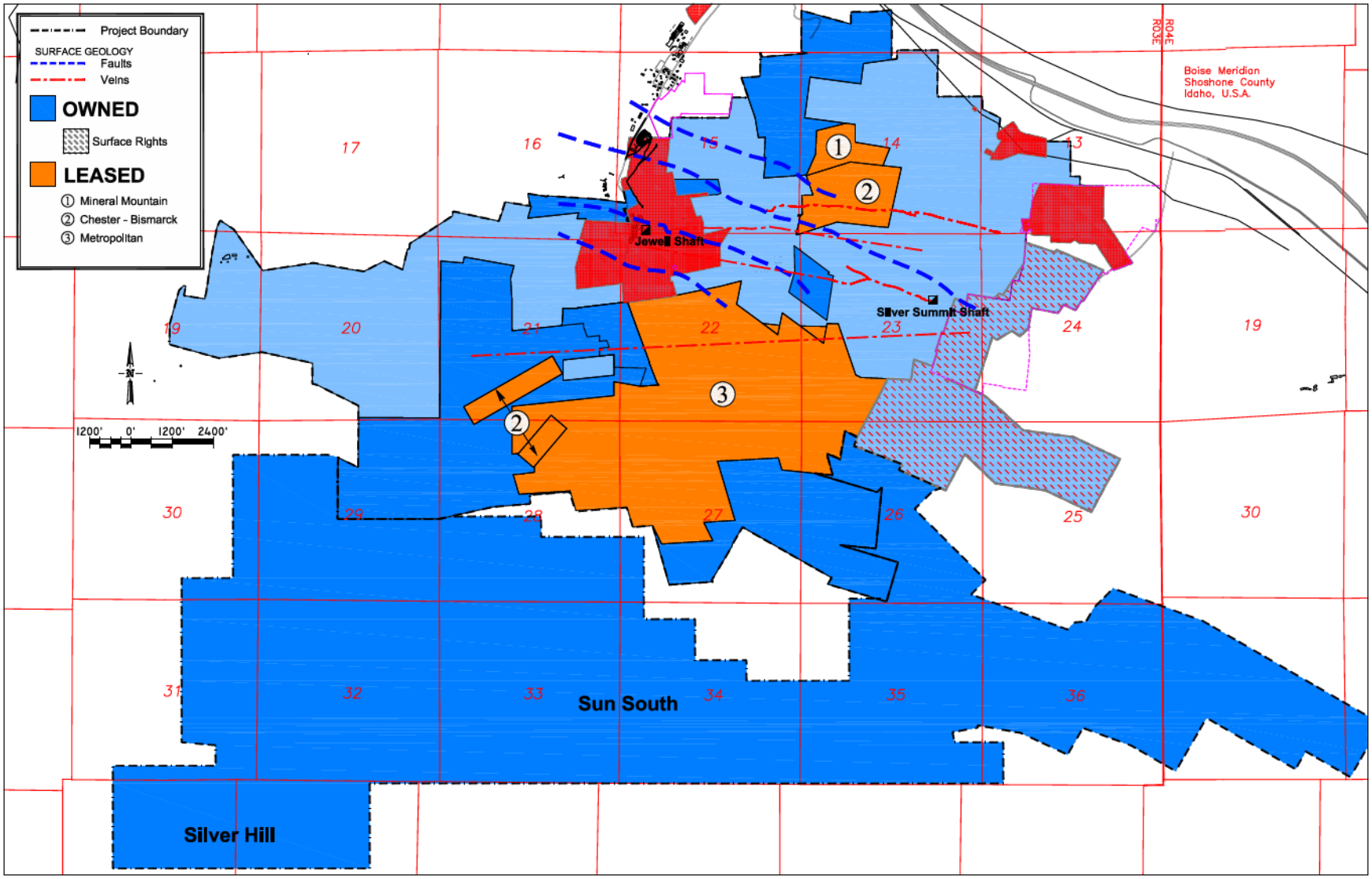


Figure 4.1 Property Mineral Rights and Claims Map

4.1.1 Sunshine and CAMP Project

SSMC owns 185 patented and 316 unpatented mining claims covering 8,404 acres at the Sunshine Mine property, including the CAMP Project claims below -900 ft elevation. This property includes the Sunshine Mine and mill, the Jewell Shaft, surface facilities, a TSF, and extensive underground workings, including shafts, levels, raises, and ramp systems, extending to a depth of over 6,000 ft below the surface. The property also includes the Silver Summit/ConSil Mine and mill, the Silver Summit Shaft, and related buildings and equipment.

4.1.2 Metropolitan

The Metropolitan property consists of two patented and seventy unpatented mining claims covering 1,020 acres. These claims lay immediately south of the primary workings of the Sunshine Mine and immediately to the west of the Silver Summit/ConSil Mine. At depth the claims intersect several veins that were historically mined from the Sunshine Mine.

4.2 Royalties

Many parts of the Sunshine Mine property are subject to royalties that are payable to parties from whom mineral rights are acquired or to others who have a right to royalties on certain areas of the property. Certain of these agreements have royalty payments that are triggered when SSMC begins producing and selling metal-bearing concentrate. These royalties are based upon proceeds paid by smelters less certain costs, including costs incurred to transport the concentrates to the smelters, or NSR, for ore produced in the property area subject to the royalties.

The royalties calculated are the aggregate of all potential royalties to all third parties, and represent a conservatively high estimate of the actual royalties that may be paid from production. A proportionate share of yearly production was assumed in calculating royalties on an annual basis.

4.2.1 Sunshine Mine

SSMC is required to pay between a 0% (at a silver price below $6.00/oz) and 7% (at a silver price of $10.00/oz or higher) NSR royalty under a Consent Decree entered by SPMI with the U.S. government and the Coeur d'Alene Indian Tribe in 2001. All funds from the royalty must be used to pay for the remediation, restoration, and other actions to address certain environmental damage to the Coeur d'Alene River and other natural resources located in the Silver Valley of Idaho. The area subject to the royalty covers substantially all of the Sunshine Mine property, owned or leased by SSMC, and purports to extend outward within a one mile boundary of the property as set forth in the 2001 settlement agreement.

4.2.2 Metropolitan Mines Corporation Mining Claims

SSMC's lease with Metropolitan Mines Corporation requires the Company to pay advanced royalties of $12,000 annually until such time as ore is produced from the Metropolitan property. Upon ore production, Metropolitan Mines Corporation is to be paid either 16% or 50% of the net proceeds from the sale of materials produced from the ore processed from these claims, depending upon the location of production.

4.2.3 Chester Mining Company Mining Claims

SSMC's lease with Chester Mining Company, or Chester, requires the Company to pay an advance royalty of $7,200 annually until such time as an NSR royalty of 4% or royalty of 20% of net profits on ore processed is payable. The net profit royalty is in lieu of and not in addition to the advance royalty and the NSR royalty. The lease also provides Chester with the option to acquire a 20% working interest in all ores, concentrates, metals, or other mineral substances produced from the property. Chester may exercise this option by releasing the Company from its obligation to pay the 20% net profits royalty and by tendering an amount of cash equivalent to 20% of the then-current working capital fund. The initial lease term ends in 2029 and is renewable for an additional 25 years.

4.2.4 Mineral Mountain Mining Claims

SSMC's lease with Mineral Mountain Mining and Milling Company or Mineral Mountain, requires the Company to pay a royalty of $3,600 annually or a royalty of 3% net profits, if net profits from the ore processed from these claims exceeds such amount. The lease also provides Mineral Mountain with the option to acquire a 3% working interest in all ores, concentrates, metals, or other mineral substances produced from the property. Mineral Mountain may exercise this option by releasing the Company from its obligation to pay the 3% net profits royalty and by tendering an amount of cash equal to 3% of the then-current working capital fund. The initial lease term ends in 2029 and is renewable for an additional 25 years.

4.2.5 Silver Summit/ConSil Mine

SSMC is required to pay between a 2% (at a silver price below $5.00/oz) and 4% (at a silver price of $7.00/oz or higher) NSR royalty to Hecla Mining Company. The area subject to the royalty surrounds the Silver Summit/ConSil Mine.

4.3 Environmental Liabilities and Permitting

4.3.1 Environmental and Permitting Requirements

In the ordinary course of operations, SSMC is required to obtain, maintain, and occasionally renew approvals and permits, such as National Pollutant Discharge Elimination System (NPDES) discharge permits, Storm Water Pollution Prevention Plan (SWPPP), TSF reclamation plan and bond, from environmental regulatory bodies. Evolving reclamation or environmental legislation/regulation/requirements may result in future liabilities currently not identified but which may require new permits and/or approvals.

Potential environmental liabilities for current and future operations would most likely be associated with the surface facilities (shops, storage yards, mill, etc.), WRSF, and TSF. All waste water, mine water, process waste water, non-contact cooling water and storm water is captured on the mine property and treated at the TSF which has an NPDES permitted discharge. Waste rock from mining activities is stored at the WRSF located approximately one-quarter mile north of the mine. Operational access to the WRSF is via a haul road located on Sunshine property. Petroleum products, cleaners, lubricants, fuels, etc. are stored at the mine within structures, on secondary containment, under cover, and/or in double-walled containers. Based on these activities, the potential environmental liabilities at the mine have been identified and minimized.

At this time, the Sunshine Mine has all environmental permits, agreements, and approvals necessary to commence surface and subsurface operations. As certain activities commence, such as the abatement of asbestos from a specific structure, the required permits and licenses will be obtained in a timely manner. However, these are limited to a case-by-case basis and can only be pursued as the different activities arise.

4.4 Other Significant Factors and Risks Affecting Access or Title

The authors of this Technical Report are unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Location and Access

The Sunshine Mine is located within the well-known Coeur d'Alene Mining District in northern Idaho and is 37 miles east of Coeur d'Alene, Idaho along I-90. From I-90, the property is accessed at the Big Creek exit south approximately two and one-half miles via a paved county road which parallels Big Creek. The property is located about four and one-half miles southeast of the town of Kellogg, Idaho which hosts a full complement of services and is home to many of the mine staff. The closest major airport and metropolitan center are located in Spokane, Washington, approximately 90 miles west of Kellogg.

The geographic coordinates of the Sunshine Mine Project are latitude 47°30'6" north and longitude 116°4'10" west.

5.2 Physiography

The Sunshine Mine is located in the Big Creek Valley at an approximate elevation of 2,600 feet above sea level with peaks around 4,800 feet above sea level. The topography is typical of northern Idaho's countryside, hilly to mountainous and forested. Forests primarily contain scrub/shrubs and tree species of douglas fir, lodgepole pine, western larch, western white pine, grand fir, and western red cedar. Wildlife inhabiting the area are typical for the Rocky Mountain region including fish, bird, and mammal species.

Sunshine's main production shaft, the Jewell Shaft, and the mill are located above the base of a steep mountain, while the hoist room and other infrastructure facilities are located on a relatively level piece of property at the base of the mountain.

5.3 Climate

Climate in the area is considered to be typical of the northern-US with snow, rain, and fog in the winter. Snowfall in the winter and the changing topography can restrict access to some surface facilities at higher elevations. Average rainfall in the area is approximately 33 inches annually.

5.4 Local Resources

Kellogg, Idaho, with a population of approximately 2,000, and Wallace, Idaho, with a population of 784, are the two nearest towns to the mine and are home to many of the mine staff. The mining history of the Idaho Silver Belt ensures a ready source of skilled and unskilled labor. Efforts are made to stimulate the local economies as much as possible, with the local area having numerous vendors that supply services to the mining industry such as welding, steel supply, transportation, and project consumables. Spokane, Washington is the largest city in the area which has an international airport and many industry supplies and services are obtained here.

5.5 Infrastructure

Electrical power is supplied by AVISTA, a large northwest United States power supplier with long historical ties to the mining industry in the Coeur d'Alene district. The district is tied to the main northwest power grid, and outages are rare. Power supply is ample for the life of the

Sunshine Mine. The main power source for the mine is a power line that parallels Big Creek Road and terminates at the AVISTA substation on the Sunshine Mine property. From the substation, power is distributed to numerous smaller substations throughout the property.

Water is abundant from Big Creek, which flows immediately through the mine yard. Big Creek flows into the South Fork of the Coeur d'Alene River, then into Coeur d'Alene Lake, and out of the lake via the Spokane River to the Columbia River. Big Creek is the principal fresh water source for the mine and mill. A treatment facility, adjacent to Big Creek, is used to chlorinate all water on the surface as well as water piped down the Jewell Shaft for distribution as required.

The Sunshine Mine is well served by regional infrastructure with I-90 approximately two and one-half miles from the mine, and Spokane, Washington is approximately 90 miles west of Kellogg. Ready access to I-90 allows for easy transport of mining equipment, supplies, and consumables to and from the mine.

SSMC has a permitted waste rock storage facitlity located approximately one-quarter mile north of the mine along Big Creek Road. A TSF is located on the Sunshine Mine property about one mile north of the mine and the historic Sunshine mill is located at the mine site.

6.0 HISTORY

Historic resource/reserve estimates, production figures, and costs contained in this section have not been verified by the author and are not considered current.

The Sunshine Mine property had its beginning in 1884, when the Blake brothers, Dennis and True Blake, arrived from Maine and staked the Yankee Load (sic) mining claim on the east slope of Big Creek. In 1909, the Blake brothers were granted patent on four mining claims and one millsite claim based upon work including ten tunnels, numerous open cuts, and a shaft for a total valuation of $18,140 exclusive of raises and stopes. Beginning in 1914, the mine was operated by a series of leasers until 1919.

The Sunshine Mining Company then came into control of the property, consisting of fifteen patented mining claims and one unpatented mining claim. Construction of a 25 tpd mill and concentrator began in 1921, with modest expansions increasing the capacity to 500 tpd. Development financed by corporate bonds continued, and their efforts were rewarded on the 200 Level with a high-grade discovery. The stopes above the 500 Level, known as “Chinatown”, allowed $145,000 of debt to be retired several months ahead of schedule and enabled a dividend to be paid for the first time in 1927.

Encouraged by the success of the Chinatown stopes, downward development of the vein continued. An inclined shaft was sunk from the 500 Level to follow the dip of the vein, while at the same time mining was conducted upward to connect to the Sunshine Tunnel level. Stations were cut at 200 foot intervals until the shaft reached the 1300 Level. Silver prices had continued to drop, so with funds running low the 1500 Level was bypassed in favor of advancing to the 1700 Level. Soon after cutting the 1700 station in 1931, drift crews driving on the path of the vein discovered a bonanza of the first order – vein widths of 20-25 feet showing a solid face of high grade silver ore. What had previously been known as the Yankee Boy Vein was re-christened the Sunshine Vein on 1700, and the Sunshine Mine became the largest silver mine in Idaho in 1931 and the second largest in the United States. By 1935, the Sunshine Mine would become the richest and most profitable silver mine in the world.

The year 1935 also saw significant infrastructure and facility improvements. The concentrator was upgraded with modern ball mill grinding units and flotation cells, which increased capacity to 1,000 tpd while attaining an excellent silver recovery of 98%. A new four compartment vertical shaft, named the Jewell Shaft, was sunk from surface, and deepened 2,079.7 feet during the year. The 2300 Level was reached in 1936, making it possible to begin hoisting ore. Numerous other concrete and steel buildings were erected to support the rapidly growing operation, including a modern 480-man dry, machine shop, compressor house, warehouse, and hoist house for the new Jewell double-drum hoist.

By the end of the 1930s, Sunshine metallurgists began to experiment with a unique method of extracting antimony from concentrate. In 1942, a plant to extract antimony from the Sunshine Mine’s argentiferous tetrahedrite concentrate went on stream using a new, Sunshine-patented process of caustic leach and electrolytic deposition. The production of antimony, as well as lead, allowed the Sunshine Mine to remain open throughout World War II.

In 1943, with the high-grade sections of the Sunshine Vein playing out, management began an aggressive exploration program. That effort paid off when a drift crew, drifting east on 2700 Level following the Syndicate Fault, broke into a very high grade vein of ore splitting the fault in an east-northeast direction. This was the discovery of the famous Chester Vein.

Four main vein systems were being developed by the early 1950s; the Syndicate Vein, the Chester Vein, the Sunshine Vein, and the Yankee Girl Vein.

The most significant of the internal shafts is the No. 10 Shaft. Crews began work on the No. 10 Shaft on the 4000 Level in 1959. The shaft heading was raised to 3100 Level and eventually sunk to an elevation equivalent to the 6000 Level.

In 1960, sandfilling operations were introduced underground. The mill tailings were classified so that the coarser material, approximately 45% of the total mill feed, could be sent back underground to be used for backfill in the stopes. This sand material was hydraulically placed at about 65% solids into the voids created by the stoping operations. The sandfill provided a much better support than the old gob (waste rock and rubbish) fill method and greatly reduced dilution from caving stope walls.

Operations at the Sunshine moved steadily deeper during the 1960s. The increasing complexity of mining ore at the Sunshine was summed up in a comment by Mine Manager John Brandon; “The bulk of the mining activity is now 6,000 feet from the Jewell Shaft. It is like working a mine through another mine.” An exception to discoveries remote to the Jewell Shaft came when exploration during the late 1960s revealed good prospects at depth near the Jewell Shaft. Plans were made and implemented to sink a new winze called the No. 12 Shaft from the 3700 Level to 4800 Level, but only the pilot borehole was completed prior to the tragic fire of May 2, 1972 resulting in the deaths of 91 men.

The Sunshine Mine was slow to recover from the fire. Labor relations worsened, coming to a climax in a bitter, year-long strike that began in March of 1976. Then, in 1980 with silver prices skyrocketing, the mine again went on strike in March and operations were down until November when an agreement was finally reached. These events led to a series of high cost labor agreements.

By early 1986, silver prices had dropped to below $5.50 per ounce, making it impossible for the mine to remain profitable. Management and labor failed to come to an agreement that would lower operating costs and allow continued production, and the mine was shut down and the property put on care and maintenance until May 1988 when a new labor contract was signed.

By the end of 1988 the mine had reached full production. Ore production was primarily from mining the Chester Vein systems serviced by the No.10 Shaft and the remnants of the Sunshine and Rambo vein stopes referred to as the Footwall Area on 3700 and 3400 Levels. The 4000 and 4200 Level Copper Vein was under development from the No.12 Shaft. There were small amounts of ore coming from this development activity.

In 1989, the mine produced 4.8 million ounces of silver, the first full year of production since the shutdown in 1986. The production from the high-grade Copper Vein stopes began to impact the silver production volumes. During 1990, the mine produced 5.4 million ounces of silver, the highest since 1971. By now the high-grade Copper Vein stopes on 4200 Level were becoming substantial producers, while production from the No. 10 Shaft stopes was dropping off. However, the silver price had dropped to $4.06 per ounce by year’s end.

The silver price continued to slip in 1991 and the operation was losing money. A mining plan was put together to reduce losses substantially while waiting for prices to improve. This plan was referred to as the “small mine plan” and was implemented in June 1991. Operations underground were centralized by eliminating the outlying and more costly production and development headings and limiting operation to day shift only. Mining was consolidated in the

area of the Copper Vein and the most productive headings in the Footwall Area. The mine below 5000 Level was salvaged and allowed to flood with water. Production was cut in half while the work force was reduced by 65%. However, funds were made available to proceed with a limited but focused exploration program.

In 1992, the West Chance Vein was discovered by drifting on 4200 Level. Additional diamond drilling coupled with detailed geological analysis of the vein-to-host-rock relationship indicated the vein would enter more favorable horizons up-dip. Drift crews delineated the West Chance on the 2700, 3100, and 3700 Levels in 1994 and 1995. By 1996 it was clear the ore body was of sufficient size and value to support the mine's return to full production, but only if done via trackless ramp and lateral development methods using Load Haul Dump (LHD) diesel equipment. By July 1997, the mine workings below the 4000 Level were salvaged of all usable equipment and materials.

Peak production occurred in 1998, when 5.9 million ounces were produced, supplemented by ore mined via trackless methods in the Sunshine Vein below the 3100 Level.

Sunshine Mining Company filed bankruptcy in late 2000, and the mine ceased production in the first quarter of 2001 as a result of several factors, including the low price of silver and the lack of concentrate sales due to the abrupt closure of ASARCO's East Helena smelter in Montana. New development and exploration were halted in 1999 as management shifted cash flow from the mine to sustain corporate expenses, debt, and other projects.

The mine remained closed until 2006, when Sterling began development of the Sterling Tunnel after acquiring a lease in 2003. Sterling resumed production in late 2007 but operations terminated in late 2008. In May 2010, SSMC acquired from Sterling, through Sterling's bankruptcy proceedings, the majority of the operating facilities and equipment at the Sunshine Mine, including a lease on the Sunshine Mine that included a purchase option for title to the mine. In July 2010, SOP closed the purchase option in the lease to obtain title to the Sunshine Mine and acquired the remaining operating facilities and equipment.

6.1 Historic Exploration and Resource/Reserve Calculations

From 1884 to 2001, the Sunshine Mining Company carried only one reserve estimate classification for the Sunshine Mine; Proven and Probable. Until recently, the terms Proven Reserves and Probable Mineral Reserves were not used, as they were classified simply as "reserves'. However, the method used to historically estimate and calculate the reserves fully corresponds to the standard practice of estimating vein type reserves traditionally used in the Coeur d'Alene Mining District for this deposit type.

No proven reserves were estimated solely by drill hole data. Proven requires at least one lineal dimension of mineralized vein had to be exposed by mine workings and adequately sampled. The long mining history at Sunshine has shown that both the main ore shoots and subsidiary ore shoots typically have vertical dimensions that are at least two to three times longer than the horizontal dimensions. The ore reserve estimation technique, in the absence of limited diamond drill hole information, was to project a block of ore above and below the developed level for a distance equal to one-half the horizontal dimension. The average weighted grade from chip samples regularly spaced at six feet or at seven feet of the development length (depending on which mining method was being utilized) was assigned to the block. This method is typical of that used in the Coeur d'Alene Mining District and originated when virtually all underground mineral exploration was conducted by drifting on the vein. Sunshine referred to this as the

“McKinstry” method, as it is adapted from the classic ore reserve estimation technique taught by McKinstry early in the last century when vein mining was the norm, not the exception.

With the exploration and development of the West Chance Vein in the mid and late 1990s, the traditional McKinstry method was modified to incorporate increased drilling results and decreased vein drift development to determine the resource of the newly discovered ore. Therefore, in the West Chance where diamond drill hole data was abundant and where actual mined grade data were being generated, a polygonal estimation approach, rather than the McKinstry method, was used. After development of the sill level of the vein, chip samples of the vein were taken at six foot intervals, and an assay “string” for a given strike length along the vein was created. The composite grade was diluted to seven feet, if the vein was less than seven feet wide. Diamond drill hole assays around this development were given a weight equal to one point in the assay string, and the area of influence of the drill hole was calculated by the polygonal technique, diluting to correspond to the mining method width common in the West Chance stopes. The resulting block was then given the average grade of the string plus the one drill hole data point.

Once the reserve block was in production, the grade of the ore reserve block was estimated from the actual grade of the last two stope cuts. The grade of the remaining block was modified by the diamond drill hole assays. Again, each drill hole assay point in the block was given a weight equal to one point in the production assay string and the proper polygonal size of the block was calculated. In this manner, the actual data from the mining were given a greater weight than the single drill hole intercept assay point.

A density factor of 10 cubic feet per ton was used for both ore and waste in reserve calculations prior to 1998. Measurements of representative ore and waste samples from the mine show tonnage factors of 8.3 and 11.4, respectively. Beyond 1998 calculations used an ore tonnage factor of 9.4 and a waste tonnage factor of 11.4.

Throughout its 128-year history, the Sunshine Mine was able to remain in production by continually advancing new resources. Between 1934 and 2007, published reserves have remained between 20 and 50 million contained ounces. A total of 365 million ounces have been produced.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The district is hosted by the rocks of the Precambrian Belt Supergroup. These Middle Proterozoic age sedimentary rocks were deposited approximately 1.47 to 1.6 billion years ago. At various times, these rocks were faulted, leached, altered, and re-mineralized. The Belt Supergroup has been divided into the Pre-Ravalli group, Ravalli group, Piegan group, and Missoula group. Within the Coeur d'Alene Mining District, rocks of the Pre-Ravalli, Ravalli, and Piegan groups can be found. The formations comprising the Ravalli group are the preferred host rocks for silver mineralization in the district. Formations within the Ravalli group from oldest to youngest are the Burke Formation, Revett Formation, and St. Regis Formation.

Ore deposits of the Coeur d'Alene Mining District occur in veins hosted in weakly-metamorphosed sedimentary rocks of the Belt Supergroup. Most of the ore production is from the Revett and St. Regis Formations of the Ravalli group. This thick sequence (up to 12.4 miles) of middle Proterozoic-age strata covers a large area of northern Idaho, western Montana, and southeastern British Columbia. The sedimentary rocks are predominately fine-grained siliciclastics with subordinate carbonate-bearing units. The Cretaceous Gem and Dago Peak stocks and a few mafic dikes (Precambrian) are the only known intrusives in the district.

A major tectonic lineament, the Lewis and Clark Line, defined by strike-slip, normal, and reversed faults, transects the district in a west-northwest direction, with folds north of the fault striking north-south. Early workers suggested that transcurrent movement, along the Lewis and Clark Line, resulted in this change of orientation. Recent interpretations support the hypothesis that there were two folding episodes and that earlier workers did not recognize the north-south folds, south of the line.

Rapid facies changes and variations in thickness suggest that faulting was active during deposition of the Belt sediments. The Osburn Fault is the local expression of the Lewis and Clark Line. The fault has been interpreted to have 15 miles of post-ore-right-lateral strike-slip displacement, periodically active through geologic time.

The district has a history of intense faulting and folding of the rock formations. Two major east-west fault zones, the Osburn Fault and Placer Creek Faults, cut through the district. Although mineralization does not necessarily occur along these fault zones, the district ore bodies are intimately associated with these and other related faulting. The unique geology of the district may display little or no indication of mineralization on the surface. Many of the successful silver mines in the district did not realize their full potential and best grade of ore until after a depth of at least 1,700 feet was reached in their downward development and exploration. Thus, mining claims in the district, in particular if located near major mines and of similar geological setting, often require deep drilling from the surface or underground drilling to determine whether commercial grade ore bodies are present. In many silver-producing areas, a deposit may bottom out at a few thousand feet below surface. However, in the Coeur d'Alene Mining District this is not the case as deep extensions of primarily silver mineralization are fault hosted, which has proved to support favorable host rocks and related silver mineralization to deep depths.

Contradictory age dates and lack of conclusive field evidence resulted in differing hypotheses as to the origin and timing of the ore deposits. One study suggests that zinc and lead-rich veins formed from stratiform Proterozoic deposits (1,500-900 ma) and that silver-rich veins were formed by a late cretaceous hydrothermal event (Bennett,1984). Field relationships and

laboratory age dating continue to underscore the complex nature of the ore bodies; however, most researchers favor the theory that the combination of faults, folds, fractures, and favorable host rocks created suitable conditions for all mineral deposition by a late Cretaceous hydrothermal origin that was possibly related to the formation of the Idaho Batholith.

For mining and exploration purposes, ore genesis can be considered academic and although it may affect regional exploration strategy, it does not change individual mine development and property-wide exploration strategy.

Structural studies augmented by geophysics and geochemistry have led many geologists to theorize that there was a 15-mile right-lateral, post-mineral displacement on the Osburn Fault. This is based upon fault displacement on the Gem and Dago Peak stocks, the postulated buried Atlas stock, fault displaced geochemical alteration haloes around the stocks, offset of structural blocks and stratigraphy, and location of the major mines.

Most of the district production has come from within a 15-mile long area from the Bunker Hill Mine to the Galena Mine. The Sunshine Mine is approximately in the center of this Bunker Hill-Galena Mine belt. Figure 7.1 shows the mineral belts within the Coeur d'Alene Mining District. The district is approximately 22 miles long in a west-northwest direction from the Page Mine on the west to the Carbonate Hill Mine on the east. However, reconstructing the geology to a pre-right-lateral faulting position (pre-post mineral faulting) allows the Star-Morning, Gold Hunter, and Lucky Friday Mines to be originally very close to the Bunker Hill Mine, putting nearly all of the district production within this 15-miles-long band.

7.2 Local and Property Geology

The Sunshine Mine is one of the most productive mines in the 22 plus-mile long Coeur d'Alene Mining District. Ore deposits are localized in the 600-foot thick St. Regis Formation and the underlying upper members of the 3,000-foot thick Revett Formation. The contact between the formations is indistinct and locally picked as the bottom of the lower-most distinct purple-colored interval in the St. Regis. Rock types include argillite, siltite, sericitic quartzite, and vitreous quartzite. Siltite and argillite dominate in the St. Regis Formation; while in the Revett, lithologies are gray to pale greenish-gray siltite and quartzite. Changes in lithologies are noted on the scale from a few inches to a few tens of feet. Detailed stratigraphy of the mine is poorly understood; geologic mapping by early workers focused on veins and alteration, facies changes, and subtleties between lithologies that complicate correlation and identification of rock units. The stratigraphic column in the mine is continually re-interpreted, and two apparent marker beds have been identified in the West Chance area. One of these argillaceous beds is thought to be a bentonite (ash tuff) unit, and may assist in correlations throughout the mine.

7.2.1 Faults

Four major west-northwest trending faults cut the mine area, and some have been mapped for several miles. The faults dip steeply to the south. The spatial relationship to the Osburn Fault suggests strike-slip movement, but studies of kinematics and rock fabrics in the mine show that most movement is dip-slip. The Polaris Fault has normal movement, but the Silver Syndicate, Chance, C, Chester, and Alhambra Faults have reverse movement. Offset is thought to be from 550 feet to up to 1500 feet in the vertical direction.

7.2.2 Folding

The principal fold in the Silver Belt is the Big Creek Anticline. Major ore deposits are localized on the anticline north limb, locally south of the Osburn Fault. Beds on the north limb are generally steeply dipping to overturned in the mine. Smaller sympathetic folds are notably present. On the hanging-wall side of the West Chance Vein, for instance, two folds with amplitudes of about 100 feet are noted. Bedding attitudes in some places suggest the major folds plunge to the west.

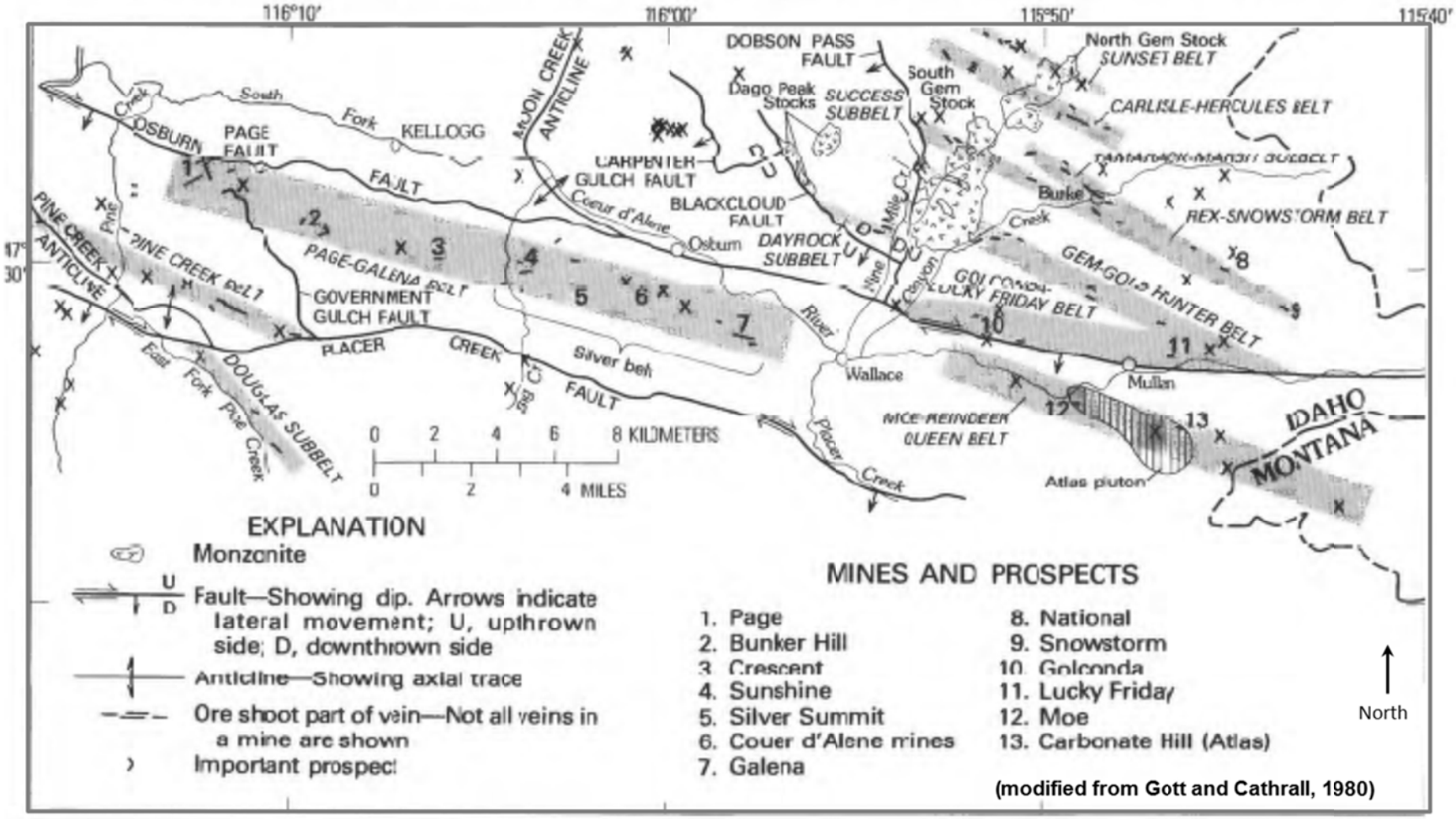


Figure 7.1 Mineral Belts of the Coeur d'Alene District, Idaho

7.2.3 Sunshine Mine Veins

The main productive vein systems in the mine include the Sunshine, Chester, Polaris, Copper, Yankee Girl, and West Chance. Mineralized silver veins are present within a zone approximately 12,500 feet long by 5,000 feet wide and over a vertical distance of 6,200 feet from the surface at 3,400 feet sea level to 2,800 feet below sea level. The mineralization is open at depth below the 5600 mine level.

Major veins strike east-west and typically dip 60° to 70° to the south. Vein strike lengths are up to 2,000 plus feet, with the down-dip length two to three times that of strike length and average between one to five feet thick. Ore minerals are principally tetrahedrite and galena with siderite and quartz as the main gangue minerals.

7.3 Mineralization

Over 30 veins have been named and mined at the Sunshine Mine. The Sunshine and Chester Veins have each produced over 90 million ounces of silver. The majority of veins strike east-west and dip about 65° to the south. Locally, dips range from 45° to 90°. Strike lengths locally exceed 2,000 feet and dip lengths are two to three times greater than the strike length. Major veins are located between the faults at an angle of 25° to the bounding faults. Veins vary in width from a few inches to over 30 feet, but are typically between one to five feet thick. Ore minerals include tetrahedrite and galena with siderite and quartz as the principal gangue minerals. Accessory minerals include bournonite, pyrargyrite, and magnetite.

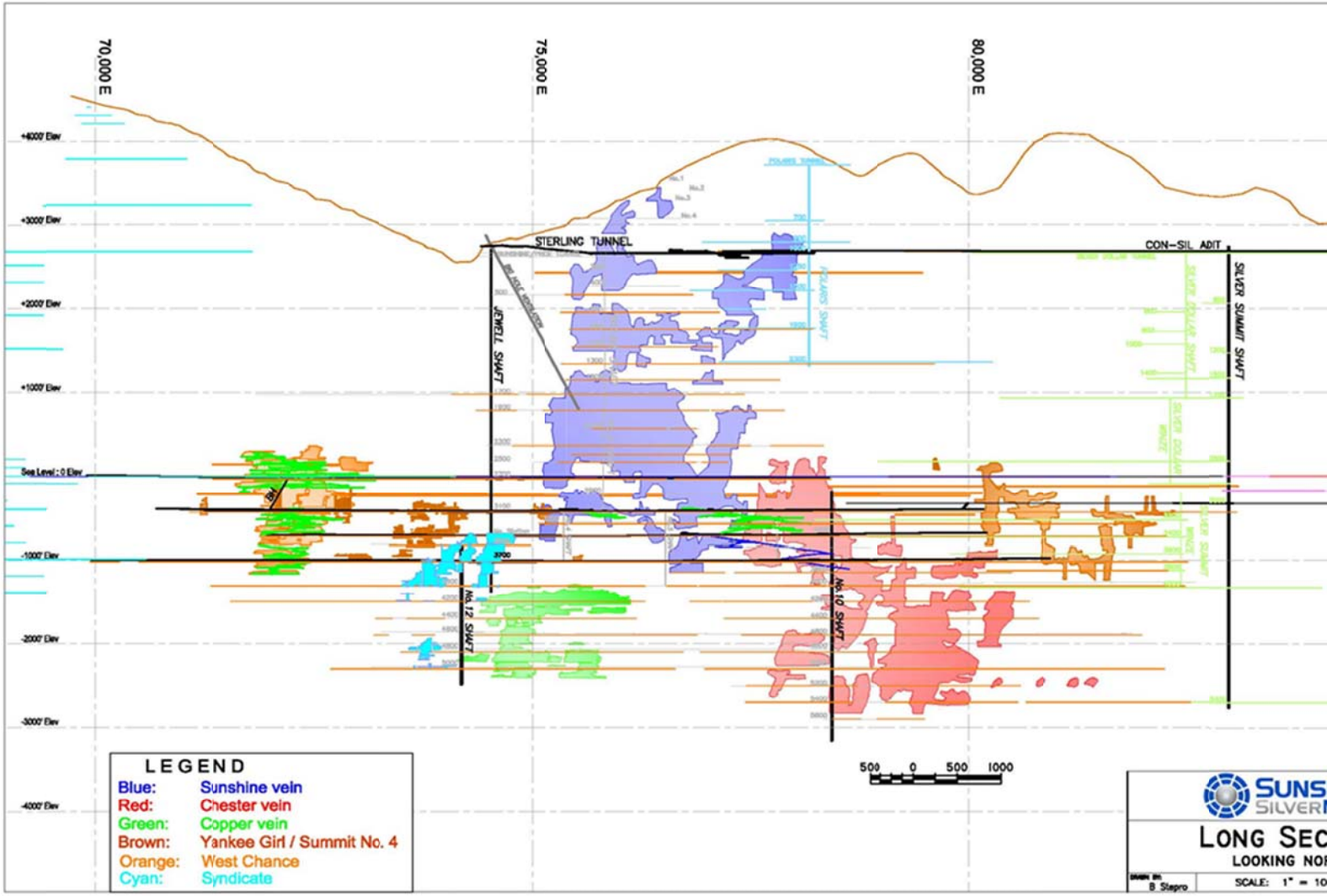
The silver content of the tetrahedrite varies and the silver to copper ratio in the ore ranges from 40:1 (ounce per ton silver:percent copper) up to 100:1. Tetrahedrite occurs as blebs, fracture fillings, or in veinlets. Grades on the veins vary from sub-ore to well over 1,000 ounces of silver per ton before mining dilution. Samples of over 2,000 ounces of silver per ton have been collected in the mine. Figure 7.2 presents a long section view, looking north, showing the related mined portions of major veins of the Sunshine Mine. Veins containing ore-grade material typically, but are not limited to, favorable stratigraphic host rocks of the Ravalli group. Overall the host rock assemblage is complexly folded and faulted. Lithology color and bed thicknesses are some of the key principal features used in stratigraphic interpretation. Alteration leaching haloes around the veins changes the pro-lithology color, complicating the task of stratigraphic correlation. The relatively recent district knowledge of stratigraphic control assisted the Sunshine Mine geologists in the discovery of the West Chance Vein. This method has continued to produce high-quality targets for future resource development programs.

7.4 Vein Mineralogy

Mineralogy is quite simple in the ores of the district and at the Sunshine Mine. Typically the Sunshine Mine ore consists principally of tetrahedrite, the high silver-content copper antimony sulfide ($3Cu_2S \bullet Sb_2S_3$). The silver content of the tetrahedrite varies considerably, and the silver-to-copper ratio in the ore ranges from 40:1 (ounce silver per ton: percent copper) to over 100:1. Tetrahedrite occurs as very fine grains in fracture fillings, veinlets, or discontinuous blebs in the vein-filled faults. This silver-bearing tetrahedrite is more properly called freibergite. Freibergite contains 3% to 30% silver substituting for the copper in the crystal structure. Gangue minerals are predominantly siderite ($FeCO_3$) with lesser amounts of quartz (SiO_2). Other sulfide minerals principally galena (PbS) and sphalerite (ZnS), are common in the district veins; however, at the Sunshine Mine, sphalerite is rare to absent. Galena is present in the West Chance Vein, where it is abundant enough to create an average lead grade of 2.17% in the “ore reserves.” In only

two other named veins at the Sunshine Mine is galena common (Silver Syndicate and Chester Hook). Other metallic minerals seen in the gangue are pyrite (FeS_2), arsenopyrite (FeAsS), and rarely boulangerite ($5PbS \bullet 2Sb_2S_3$), bournonite ($2PbS \bullet Cu_2S \bullet Sb_2S_3$), pyrargyrite ($3Ag_2S \bullet Sb_2S_3$), and magnetite ($Fe_3O_4$).

Figure 7.2 Long Section Primary Project Veins



8.0 DEPOSIT TYPES

The Sunshine Mine mineral deposits are narrow high-grade mesothermal stratbound vein deposits. Contradictory age dates and lack of conclusive field evidence resulted in differing hypotheses as to the origin and timing of the ore deposits. One study suggests that zinc and lead-rich veins formed from stratiform Proterozoic deposits (1,500-900 ma) and that silver-rich veins were formed by a late Cretaceous hydrothermal event (Bennett,1984). Field relationships and laboratory age dating continue to underscore the complex nature of the ore bodies; however, most researchers favor the theory that the combination of faults, folds, fractures, and favorable host rocks created suitable conditions for all mineral deposition by a late Cretaceous hydrothermal origin that was possibly related to the formation of the Idaho Batholith.

For mining and exploration purposes, ore genesis can be considered academic, and although it may affect regional exploration strategy, it does not change individual mine development and property-wide exploration strategy.

Veins characteristically strike east-west and dip steeply (average 65°) to the south. The combination of faults, folds, fractures, and favorable host rocks created suitable conditions for mineral emplacement by silver-rich and silver-base metal veins. Figure 8. shows the generalized relationship between the principal productive veins and the four major faults. Drilling from surface has been generally non-productive. Typically, underground drilling and drifting are historically the most productive exploration tools.

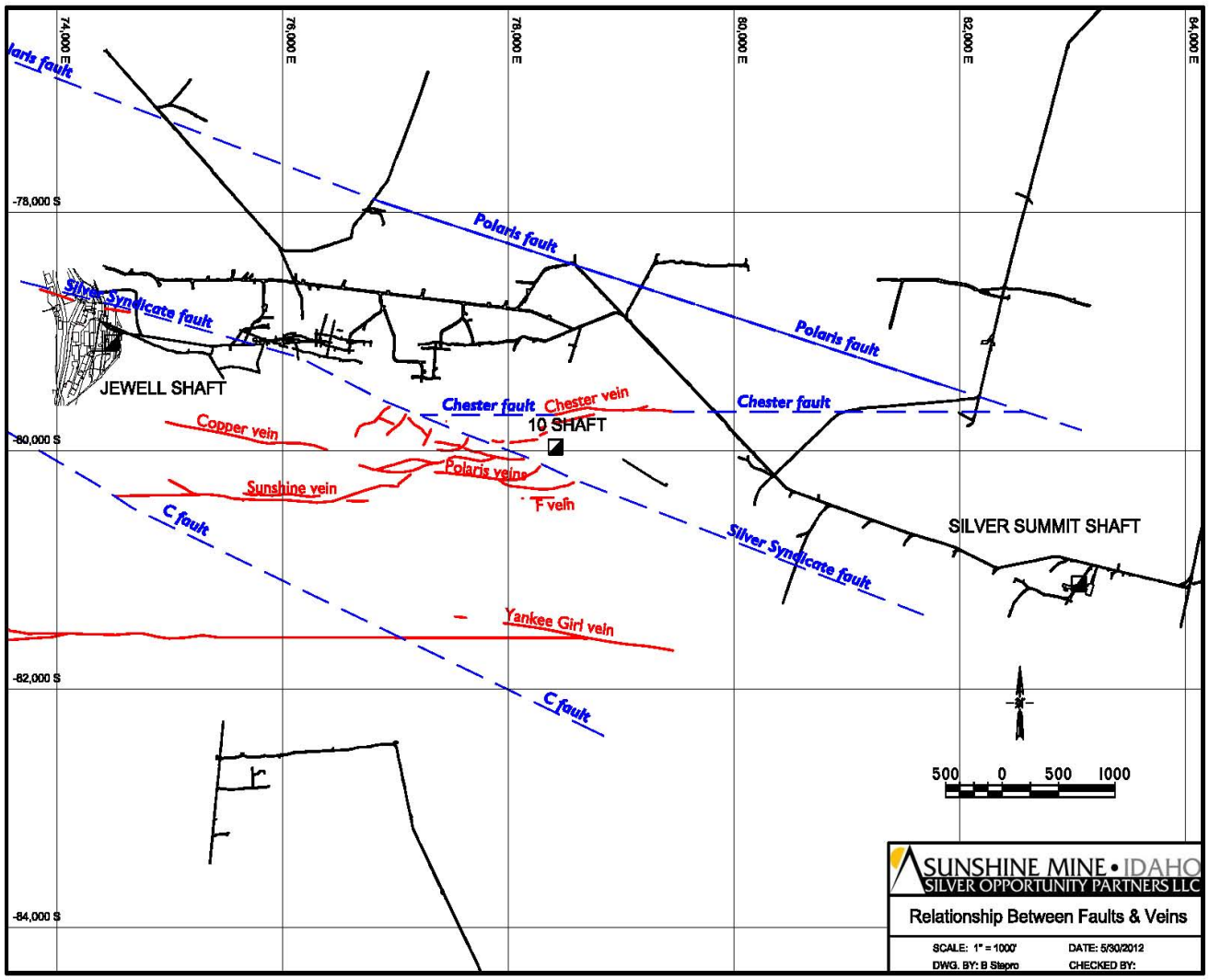


Figure 8.1 Relationship between Faults and Veins

9.0 EXPLORATION

The objectives of the current exploration program at the Sunshine Mine are to discover new high-grade veins and ore shoots in areas that already have nearby development, explore for new large veins in unexplored or under explored areas, and to systematically replace reserves as they are mined. All of the exploration work carried out at the Sunshine Mine created historic resources. It is necessary to describe this historic exploration work as it includes the methods utilized by prior owners/operators Sunshine Mining Company, Sterling, and current owner SOP. Exploration work presently underway is using the same methods as practiced by Sunshine Mining Company up through 2001, except that defined resource and reserve categories will now be classified in accordance with the CIM Definition Standards. Mine staff completed surveys and exploration work in the past and this practice continues incorporating new advancements in practice, method, and technology. A significant example of this, the Sterling Tunnel was driven to join the Sunshine and ConSil Tunnels and has allowed underground diamond drilling exploration to be resumed in the Upper Country mine area in conjunction with ramp and drift development. Normally, underground chip samples are collected for daily grade control and for resource estimation. Daily samples are collected underground from drift faces, stope faces, drift backs, drift ribs, and raise ribs. Samples are taken by collecting chips in a horizontal channel, left to right, across the mining face. Sampling protocol for channel samples is to collect separate samples of exposed wall rock on both sides of the apparent mineralized vein, and across the mineralized structure or vein samples are collected perpendicular to the mineralized structure. Multiple samples are taken across a face based upon changes in mineralization intensity or composition. Samples are a maximum of five feet in length. Each sample face has a referenced control distance to an established survey point. Drifts on the veins are generally sampled at four to six foot intervals. Both raises and stopes were sampled at regular intervals that typically vary based on advance cycles.

9.1 Prior Exploration

Beginning in August 2003, Sterling undertook a surface exploration program including induced polarization (IP), resistivity, and chargeability geophysics and geochemical sampling that yielded near surface zones of interest. The work was followed by a three-hole drilling program totaling 2,473 feet that was completed in 2004. While the first two holes increased stratigraphic information, the third hole drilled toward the Yankee Girl structure intercepted multiple veins in the Sunshine shear zone and the footwall of the Yankee Girl Vein. This information was a key driver in pursuing the Sterling Tunnel Project.

Sterling Tunnel underground contract drilling began in September 2006 and explored distant targets, such as the Silver Syndicate, Copper-Link, Hook, and Chester Veins, and the Polaris Fault to the north of the Sterling Tunnel as well as the Yankee Girl Vein to the south. The drill results yielded information essential to determining the nature of Upper Country structure and stratigraphy, providing a guide for additional drilling and exploration activities. In aggregate, total 2004-2008 exploration diamond drilling amounted to 46,570 feet.

9.2 Current SSMC Exploration Plans

SSMC has planned a very aggressive multi-year underground exploration program. The total footage to be drilled is estimated at a maximum of 360,000 feet. Large areas of the Sunshine Mine complex are largely unexplored and have received minor drilling over the past two decades. Important and prospective targets have been identified on the 2700, 3100, 3700, 3000, ConSil, and the Sterling Tunnel levels. Multiple areas of exploration will be tested within

all accessible underground levels. After minor needed repair work, these targets offer both rubber tire and track-level access.

Although the proposed multi-year program will explore eight primary areas of interest, including the Yankee Girl/Summit #4 Vein, Western Sunshine Vein, Metropolitan Vein, 101 Vein, up-dip extension of the West Chance Vein, Upper Sunshine/Polaris Veins, and the Silver Syndicate/Polaris Fault/Vein zones, the planned 2013 program is more limited in scope due to current access restraints.

Since initiating drilling in April 2011, SSMC has complete 41 drill holes totaling 49,836 feet (15,190 meters) as of June 4, 2012. The primary focus of the drilling from the Sterling Tunnel has been to follow up on open ground within the Sunshine Vein system, while also exploring the Yankee Girl Vein. During the process a new vein structure was identified west of the Sunshine Vein system, referenced as the West Chance Link Vein. The West Chance Link Vein is a linking structure between the Sunshine Vein and West Chance Vein. A summary of the Sterling Tunnel intercepts is presented in Table 10.0. The Sterling intercepts correlate with two recent deep surface intercepts, including a historical resource along the same dip and strike indicating that the mineral pay streak is continuous from the Sterling Tunnel elevation to the 1700 Level. The lateral extents of the new structure are not defined as drilling is ongoing to continue delineation. Additionally, in-fill drilling open ground above the Sterling Tunnel elevation, about the Sunshine Vein has also identified a new vein structure referenced as the South Yankee Boy Split. Caving sand ground conditions made diamond drill hole advance difficult. SSMC has completed driving an exploration incline drift to the drill vein intercepts. Approximately 180 feet of intermediate drifting was completed along vein strike. Select vein muck pile samples were collected in support of a bulk metallurgy sample to aid new mill process design criteria. A 16,000 lb (7,200 kg) sample was sent to G&T Metallurgical Services (G&T), a division of ALS Canada Ltd, in Kamloops, British Columbia. The sample is currently being analyzed.

Historic exploration was primarily constrained by legal property boundaries. Potential targets were also never explored due to limited availability of drills, lack of supporting infrastructure, lack of physical access to provide suitably located drill stations, and prohibitive depths from the surface. With the consolidation of the land position, long-known but previously unexplored target areas can now be explored, although some will require new drifting and new drill stations cut.

10.0 DRILLING

10.1 Historical Drilling

The current drill database contains approximately 3,536 underground drill holes. Additional drill holes are being identified and added to the master database. The longest underground hole is 3,000 feet. It is not uncommon for holes to be 1,500 to 2,000 feet long. Long underground exploration holes are required to locate structures and veins because the majority of historical development, except in the West Chance, has been on the vein structures themselves; thus, drilling platforms for shorter holes at appropriate angles to the targets have not been available.

The drilling was done by the prior companies with company-owned equipment and mainly by their corporate employees. All of the previous drilling was core. The drilling was done with the following equipment.

- Pneumatic percussion drills (CP 65s), 500 foot capability, typically obtained AQ core in the target zone
- Hagby drills for underground long hole exploration, typically obtained BQ or NQ core in target zones
- Longyear 38 for long range hole exploration to obtain NQ core in target zones

Historic core logs with appropriate descriptions exist with the exception of a single surface hole log book, which has been identified as “misplaced” during a past change of property ownership. The historical drill operators were competent and core recovery in the mineralized zone was generally 90% or higher. Given the fracturing and broken ground in the mineralized zones, core losses in some holes were significant. Sunshine began down hole surveying of its holes when capable equipment became available. The majority of drill holes have been down-hole surveyed. Shorter historical holes, typically less than 200 feet, are only denoted with a collar azimuth and dip. Historical core was not photographed. All recent drill core drilled is digitally photographed. Historically, mineralized core for assay analyses was split in half with hydraulic splitters. One-half was replaced in the core box and is stored onsite with skeleton core samples from select drill holes. The other half was utilized for assay analysis.

The historical purposes of the drilling to identify potential mineralized structure, has resulted in limited use of drill hole assay data incorporated in the historical Sunshine mineral resource and reserve estimates.

10.2 Current Drilling

The current drilling for exploration, delineation, and development conducted at SSMC has been performed with diamond core drills. Work is continuing with a local contract core drilling company, Dynamic Drilling Inc. from Osburn, Idaho. They currently operate one Hagby Onram-1000 diamond drill underground in the Sterling Tunnel. Down-hole surveys are attempted on all diamond drill holes. The primary survey tool is a Reflex EZ-AQ multi-shot down-hole survey camera. Core diameters range from 1.062 to 1.875 inches. Core recovery is generally very good, exceeding 90%. Core recovery can be difficult in certain faulted or sheared areas. The diamond drillers will change from wireline tools to conventional tools before encountering proven areas of loss, which significantly improves recovery. Recovery issues do not materially impact the reliability of the results.

All drill hole and sample information is stored in an Access® database for reporting purposes and in a Gemcom database for 3D evaluations in support of resource modeling. When drill hole samples are used for polygonal or accumulation methods of resource modeling, they are corrected back to true horizontal thickness. Diamond drill holes are typically designed to intersect mineralization as close to perpendicular as possible. Drill hole assaying is typically conducted to break out distinct mineralologies within single vein intercepts, however to be compatible with level and stope channel sampling assay data is composited across single vein intercepts. Down-hole directional surveys are conducted on all drill holes, since hole deviation is common. A Reflex EZ-AQ multi-shot down hole survey instrument is used for deviation surveys.

To date three new vein structures have been defined with drilling from the Sterling Tunnel elevation. Two new silver-copper veins have been defined in the immediate hanging wall of the historic Sunshine Vein and are named the West Chance Link Vein and the South Yankee Boy Split Vein. A total of 16 holes have been drilled by SSMC targeting the West Chance Link Vein and all have encountered silver mineralization. Additionally, a new lead-silver vein, named the "10 Vein" has been recently discovered 200 feet within the footwall of the Sunshine Vein. All veins carry economic silver-copper or lead-silver values. Drilling is continuing to define the vertical and lateral limits of the new vein structures.

Table 10.1 details drill intercepts completed in 2011 and 2012 that were included in this mineral resource estimate. Drilling subsequent to the cutoff date of September 15, 2012 that was not used in the calculation of this mineral resource estimate is included in Table 10.2. Drilling after the September 15, 2012 cutoff has been focused on underground drill station C, along the exploration decline from the Sterling Tunnel. Drill results indicate mineralization in the upward vertical extension of the Sunshine Vein and have assisted in the definition of the new "10 Vein" located approximately 200 feet in the footwall of the Sunshine Vein. Drilling continues along this portion of the decline to better define the nature and geometry of the two veins and for incorporation in future resource estimates.

Table 10.1 Drill Results 2011 and 2012 Drill Programs

Drill Hole ID	Drill Station	Azimuth	Dip	Total Depth	From (ft)	To (ft)	Downhole Length (ft)	Angle to Core Axis	True Thickness (ft)	Ag opt	Cu %	Pb %	Sb %	Zn %
ST2560	A	180	-13	542	499.8	500	0.2	80	0.2	58.90	1.70	31.50	0.10	0.02
ST2561	A	180	-30	1,720	439.1	443	3.9	80+	3.9	1.56	0.02	0.47	0.01	0
ST2563	A	190	-20	1,607	550.4	550.9	0.5	65	0.45	55.30	1.39	4.31	0.23	0.06
ST2564	A	200	-24	1,436	457.8	459.4	1.6	80	1.58	1.97	0.06	1.31	0.01	0.00
ST2568	A	200	-40	915	512.4	518.3	5.9	70	5.54	15.82	0.29	0.04	0.13	0.01
ST2568	A	200	-40	915	766	766.6	0.6	40	0.39	16.80	0.61	0.00	0.06	0.00
ST2570	A	210	-40	995	516.6	519	2.4	65	2.18	63.55	0.60	0.09	0.03	0.03
ST2570	A	210	-40	995	740.7	741.1	0.4	50	0.31	6.82	0.83	0.12	0.05	0.02
ST2572	A	180	-40	1,025	488.3	488.7	0.4	70	0.38	6.35	0.13	0.36	0.06	0.01
ST5273	A	180	-16	1,600	490.3	490.5	0.2	65	0.18	5.96	0.07	1.17	0.04	0.01
ST2577	A	190	-60	1,070	631.2	632.7	1.5	45	1.06	26.19	0.58	0.74	0.32	0.04
ST2577	A	190	-60	1,070	633.6	635.9	2.3	45	1.63	7.71	0.35	0.16	0.05	0.01
ST2578	A	200	-60	1,255	610.0	611.2	1.2	65	0.6	74.81	1.33	0.34	0.89	0.10
ST2579	A	206.5	-60	850	619.5	623.6	4.1	50	3.1	3.62	0.00	0.44	0.02	0.01
ST2580	A	211.2	-61	805	735.0	735.6	0.6	50	0.5	7.95	0.03	0.56	0.04	0.01
ST2587	A	200	1	450	436.5	439.5	3.0	80	3.0	20.65	1.17	0.17	0.31	0.04
ST2609	P	167	12	1,300	665.5	675.4	9.9	70/45	7.3	69.15	0.96	0.00	0.84	0.11
ST2611	P	170	4	740	681.0	682.5	1.5	75/65/50	1.4	41.20	0.63	0.00	0.51	0.05
Surf.2011-1	Surface	75	-68	1,480	1346.2	1346.7	0.5	55	0.4	1.52	0.24	1.35	0.03	0.01
Surf.2011-1	Surface	75	-68	1,480	1350.7	1355.9	5.2	55	4.3	28.03	0.33	18.87	0.29	0.29
Surf 2012-2	Surface	80	-75	1,793	1454.7	1455.2	0.5	65	0.4	16.00	0.29	2.78	0.01	0.03
Surf.2012-3A	Surface	80	-73	1,650	1445.6	1446.2	0.6	55	0.5	2.93	0.11	2.43	0.01	0.33
Surf 2012-4	Surface	82.5	-66.5	1,669	1593.0	1595.9	2.9	60	2.5	17.60	0.08	34.93	0.08	0.12
Surf 2012-4A	Surface	71	-66	1,770	1578.0	1579.7	1.7	58	1.4	5.53	0.06	1.20	0.03	0.02
Surf 2012-5	Surface	105	-74	1,910	1804.4	1805.0	0.6	58	0.5	11.20	0.10	6.49	0.07	0.02

Table 10.2 Recent Drill Hole Intersections Subsequent to the Mineral Resource Estimate

Drill Hole ID	Drill Station	Azimuth	Dip	From (ft)	To (ft)	Downhole Length (ft)	Angle to Core Axis	True Thickness (ft)	Ag opt	Cu %	Pb %	Sb %	Zn %	Vein
ST2624	C	228.4	-22	861.0	863.3	1.3	62.5	1.2	7.52	0.34	0.14	0.12	0.01	Sunshine
ST2625	C	357.7	-45.6	683.2	687.0	3.8	80	3.7	37.02	0.41	0.11	0.34	0.03	Sunshine
ST2625	C	357.7	-45.6	892.8	898.0	5.2	62	4.6	7.29	-	0.54	10.59	0.04	10
ST2627	C	341.4	-54.9	742.2	746.7	4.5	60	3.9	6.62	0.10	0.00	0.08	0.01	Sunshine
ST2627	C	341.4	-54.9	955.5	960.1	4.6	50	3.5	1.54	0.02	2.03	0.01	0.00	10
ST2627	C	341.4	-54.9	963.5	966.8	3.3	51	2.5	14.45	0.65	0.51	0.30	0.03	10
ST2628	C	340.2	-64.5	858.9	861.7	2.8	60	2.4	40.21	0.99	0.00	0.75	0.09	Sunshine
ST2628	C	340.2	-64.5	1090.3	1098.5	8.2	60	7.1	16.16	0.25	6.66	0.19	0.02	10
ST2629	C	341.5	-35.4	677.6	685.0	7.4	70	7.0	71.31	0.76	0.02	0.63	0.06	Sunshine
ST2629	C	341.5	-35.4	870.6	873.7	3.1	70	2.9	6.44	0.09	0.45	0.06	0.01	10
ST2630	C	340.1	-45.2	709.1	714.3	5.2	80/50	4.6	68.09	0.10	0.01	0.08	0.01	Sunshine
ST2630	C	340.1	-45.2	913.0	919.9	6.9	80	6.8	1.30	0.01	1.27	0.01	0.00	10
ST2631	C	358	-55	751.0	751.5	0.5	70	0.5	63.90	1.00	0.16	0.87	0.09	Sunshine
ST2631	C	358	-55	966.7	970.4	3.7	50	2.8	35.99	0.10	57.38	0.20	0.01	10
ST2632	C	358	-65	792.1	795.0	2.9	60	2.5	8.86	0.11	0.00	0.09	0.01	Sunshine
ST2634	C	343	-45.5	723.7	726.7	3.0	55	2.5	7.36	0.70	0.00	0.06	0.01	Sunshine
ST2635	C	343	-54.4	740.1	741.8	1.7	75	1.6	9.23	0.09	0.00	0.08	0.01	Sunshine
ST2635	C	343	-54.4	934.5	946.3	11.8	80	9.7	7.31	0.07	6.97	0.06	0.01	10
ST2636	C	345	-25	698.8	701.1	2.3	70	2.2	5.34	0.03	1.89	0.03	0.01	Sunshine
ST2637	C	345	-63.8	806.9	811.6	2.3	52	3.7	12.50	0.23	0.02	0.16	0.03	Sunshine
ST2637	C	345	-63.8	820.0	823.3	3.0	60	2.6	41.47	1.00	0.08	0.73	0.08	Sunshine FW
ST2637	C	345	-63.8	1025.3	1035	9.7	50	7.3	2.16	0.01	2.39	0.02	0.00	10

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The author finds the sample preparation, analyses, and security to be adequate and believes SSMC procedures are inline with common best practices. Both diamond drill samples and underground channel samples are used in resource estimation.

11.1 Channel Samples

Underground chip samples are collected for daily grade control and for resource estimation. Daily samples are collected underground from drift faces, stope faces, drift backs, drift ribs, and raise ribs. Samples are taken by collecting chips in a horizontal channel, left to right, across the mining face. Sampling protocol for channel samples is to collect separate samples of exposed wall rock on both sides of the apparent mineralized vein, and across the mineralized structure or vein samples are collected perpendicular to the mineralized structure. Multiple samples are taken across a face based upon changes in mineralization intensity or composition. Samples are a maximum of five feet in length. Each sample face has a referenced control distance to an established survey point. After samples are collected, the geologists carry them to the surface where they are secured, and inventoried, for transportation to the assay lab. On the sample ticket, the location is recorded, the sample is described, and a sketch of the vein and face is completed. A tear off tag on the main sample ticket is placed in each sample bag.

The analyses from the face samples taken during development and from samples taken as production mining proceeded are the primary sources of data that Sunshine uses to estimate its reserves. Those analyses are the basis for the estimates of resources in this report.

Drifts on the veins are generally sampled at four to six foot intervals. Both raises and stopes were sampled at regular intervals that typically vary based on advance cycles. The historical assay plan and long section paper data is currently being digitized and entered into an electronic database. Locations and analyses from the underground face samples, beginning in 1995, have been entered into the electronic database. Data from the face samples prior to 1995 have been digitized and entered into the database. The initial database system was TechBase®. Since 1996, Microsoft Access® and AutoCAD® were used, with all graphics in AutoCAD. Historic underground sampling assay results were plotted on paper and canvassed-back paper maps. A relatively complete set of historic sampling maps have been stored in the Sunshine Mine archives and vaults. The maps are quite detailed and document the results of extensive drift and stope sampling. SSMC is currently completing a thorough digitized scanning of these maps.

11.2 Diamond Drill Samples

Drilling done at the Sunshine Mine for resource estimation is done with diamond core drills. The core diameter is typically BQ (1.432” in diameter) or NQ2 (1.990”). Since 2000, the core has been detail logged and photographed in a dedicated surface facility. Core samples are collected through the vein or mineralized structure. Additional core on both sides of the mineralized zone are sampled to characterize waste dilution. No samples taken for assay are greater than five feet; large zones are broken into increments of five feet or less. When core is lost through a mineralized zone the total drill thickness of the zone is used for volume estimation. The portion of a diamond drill hole used to calculate the reserve for a given vein must be corrected to account for the true thickness of the vein at that point. The down hole length of the intercept is multiplied by the sine of the vein angle to the core axis. This is standard procedure prior to resource calculation.

11.3 Density Determinations

To date, the resource and reserve estimates for the Sunshine Mine were carried out utilizing historical bulk-density (specific gravity) and tonnage factor values that had been used for 100 years. Surviving documentation exists to verify the methodology. The Sunshine Mine has recently initiated a procedure to capture and document specific gravity for each vein type, waste rock type, and unique mineralized structure type, as applicable to deriving update tonnage factors for use in future resource calculations. The density of each sample is determined by weighing air-dried pieces of rock specimen or core with a high precision calibrated digital scale, via hanging rod, by submerging each specimen in water and comparing the submerged wet weight.

11.4 Analytical Facilities

Currently there is no sample preparation or laboratory facility at the Sunshine Mine except core splitting. Most Sunshine Silver samples are sent to American Analytical Services (AAS), of Osburn, Idaho (AAS; www.americananalytical.net). AAS conducts assaying on contract basis for SSMC and other clients including mining and exploration companies, and owns the laboratory building and the assay equipment. AAS is a business independent of SSMC.

The AAS laboratory is an ISO-17025 accredited laboratory (Similar to ISO-9000, but with an added level of quality management). AAS is also accredited by the State of Washington Department of Ecology, with registry number WA09-0799, for analytical capabilities in non-potable water, not in analysis of solids. Standard written procedures are used by AAS, and commercially prepared standard pulps are used.

No officer, director, or employee of SSMC is involved in AAS operations, sample preparation or assaying, after the samples arrive at the assay laboratory.

11.5 Sample Preparation

11.5.1 Historical

Previous authors have written several published reports and unpublished in-house reports on the Sunshine Mine. The authors were not able to see the sample preparation and analytical procedures during the site visits for these prior reports because the equipment had been largely dismantled and scrapped or the facilities were not in operation. The sample preparation and analytical procedures were described by Sterling in 2009 and confirmed by SSMC during Behre Dolbears's most recent site visit (Behre Dolbear 2011) and are described below. SSMC's procedures manual for sample preparation, analyses, security, core logging, and QA/QC has been reviewed by the author.

Core and underground samples were delivered to the sample preparation facility on-site by the geologist who logged the core or took the sample. The samples were crushed and ground and delivered to the laboratory for analyses. SSMC employees did all of the sample preparation, analyses, and posting of results on-site. Sample preparation includes discussion with supervisors about the interval, splitting of core, sampling, and delivery to the laboratory. This chain of custody maintained the sample integrity.

The exact crushing and grinding steps are specified and the protocol meets accepted industry standards. AAS has been contracted for drill hole and other assay services.

11.5.2 Current

The core samples, rock chip, channel, and select samples are placed in bags with identification tags and are tied closed at the sample site. The bags are marked externally with the same sample identification as the sample tag. The samples are placed in a designated location within the core logging facility until they are transported to the assay lab. The samples and a submittal sheet chain of custody are either transported to the lab by an SSMC employee, or are picked up by an AAS representative. The sample tags in the bags and the submittal sheet indicate a unique number for each sample and the chemical elements that are to be analyzed.

Upon arrival to the lab, samples are compared to the submittal sheet and placed in drying ovens to dry overnight at a temperature of approximately 65^0C. Samples are emptied from the sample bags into the jaw crusher, then run through a second time resulting in a size sample of approximately 1.2 inches. The sample is then run through a cone crusher reducing the size to about 50 percent passing a 10 mesh screen. The sample is then split using a Jones riffle splitter until a sample of approximately 200 grams is obtained. The rejected portion of the sample is returned to the original sample bag. The 200 gram sample is ring pulverized with an 8-inch bowl for 45 seconds, resulting in a 140 mesh passing pulp at approximately 90 percent. Approximately 125 grams of pulp is placed in a sample envelop and sent to the fire assay room. The ring pulverizer is cleaned between each sample with silica sand to prevent contamination. Barren rock is run through the crushers once a day and this sample is assayed as a sample blank. A split is made on one sample for every twenty that are prepared and this is assayed as a pulp duplicate.

11.6 Assaying

11.6.1 Historical

Historical assaying was undertaken at the mine site laboratory. Assaying of silver is currently conducted by fire assay with an atomic absorption (AA) finish by AAS.

The current and previous authors have not seen details of the historic analytical protocol, nor are they aware of any specific QA/QC procedures used by the laboratories. Such procedures are now standard practice but have only been practiced in most labs in the last 20 to 25 years. There is no QA/QC data from the Sunshine laboratory to verify the precision and accuracy of the results, and the quality of the results may have varied over time. The authors, however, do not regard the lack of such data as a significant reason to question the analytical results for the following reasons.

- The authors do not know of anything in the history of the mine to cause them to question the analytical results.
- The large number of analyses, over more than 50 years, makes any errors over a short period of time or on a relatively few samples insignificant in regards to the whole database.
- As reported by Sterling and SOP, the lack of questions by the smelter and refinery of the analyses of Sunshine's concentrates indicates that the Sunshine laboratory produced quality analyses.

Based on information gleaned from past Behre Dolbear site visits to the Sunshine Mine and the analytical data produced, the authors conclude that the historic Sunshine sample preparation

and analytical facilities produced acceptable analytical results. The authors accept those results as valid for use in estimating the Sunshine historic reserves and or the current estimation of resources.

11.6.2 Current

AAS samples are analyzed by atomic absorption (AA) and induced coupled plasma (ICP) techniques to determine silver, copper, lead, zinc and antimony content. AA silver values assaying over 40 opt Ag are also fire assayed for silver. The fire assay results are preferentially utilized in all calculations.

Fire assay at AAS involves one-half assay ton equivalent of drill core or channel sample is weighed into a 30 gram crucible with approximately 100 grams of standard flux mixture and a litharge cover. Twenty samples are fired at a time including a pulp duplicate and a control sample. Lead buttons are cupelled in either composite or bone ash cupels. Doré beads are weighed and then parted with (1 to 3) nitric acid, followed by decanting, subsequent washing with a weak ammonia solution, annealed, and then weighed. Subsequent to assay all pulps are boxed with proper identification and stored at the laboratory until they are released via chain of custody back to SSMC. The coarse rejects are routinely collected by SSMC personnel and stored at a secure dry location in support of generating future standards.

11.7 Security

Historically, the employees of the previous owner did all of the sample preparation, analyses, and posting of results on-site. This chain of custody maintained the sample integrity.

Currently, the coarse rejects and sample pulps are stored in a secure location in the core storage building for future use. All samples that remain on-site, prior to delivery to the laboratory (on-site or off-site) are kept in a secure location not accessible by anyone other than approved personnel.

11.8 Quality Assurance/Quality Control

As stated previously, the current and prior authors have little information on historical QA/QC data. No historical significant negative issues have been identified. For future exploration, SSMC recognizes that CIM Definition Standards best practices require mining companies to exercise due care with their exploration drilling, sampling, and assay procedures. The Sunshine Mine is complying with the CIM Definition Standards best practice requirements. The following QA/QC procedures have been established as the official SSMC protocol.

11.8.1 Procedures for Drill Core

It will be the drill contractor's responsibility, and the logging geologist's review responsibility, to ensure correct numbering of core boxes and maximizing drill core recovery. The core, coarse rejects, and pulps are locked in a secure location and stored. Access will only be available to approved personal. Drill holes will have collar and down-hole surveys.

11.8.2 Core Logging Procedures

All core is digitally photographed with a standard fixed mount camera base, having all core run intervals clearly marked on each box. A standardized paper logging form and standardized rock, mineral, and alteration color codes are utilized during core logging procedures. The core is

logged in detail, including lithology, structure, alteration, mineralization, and bedding forms. Core recovery and rock quality data (RQD) are included in the log. All structures are measured in relation to the core axis. Additional core samples are recovered, and isolated during logging in support of third party laboratory testing for uniaxial and triaxial rock strength parameters.

11.8.3 Sample Preparation and Analyses

Sunshine Mine exploration samples will be submitted to AAS. AAS performs internal laboratory checks that include duplicate assays for fire and AA assays and additional duplicate assay checks for additional base metals. Third party duplicate check assays are planned to be performed by Inspectorate America Corporation (Inspectorate). Every 30th sample pulp will be re-analyzed by a certified outside laboratory. A discrepancy in the secondary pulp analysis, by an outside certified laboratory with more than two standard deviations of the original assay, will result in a re-run of the coarse reject sample interval.

11.8.4 Certified Standards and Blanks

SSMC has purchased multiple silver, gold, and base metal standards from Shea Clark Smith. The five standards in use consist of two low grade standards, relative to average mine grade, two average grade standards, and one high-grade standard. There is a silver-gold-copper-lead-zinc standard at each grade level and a silver-copper-lead-zinc standard at low and average grades.

These standards are inserted into the sample stream. SSMC inserts a blind standard assay sample for each 30 samples. The blind sample is labeled with hole number and footage. The true sample interval will also be assayed with the pre-arranged sample identification.

Table 11.1 presents data regarding three Shea Clark Smith standards. The acceptable range is the certified mean plus or minus two standard deviations. AG-1 is a low grade silver standard of 7.24 opt Ag, AG-2 is regarded as an average grade standard of 8.72 opt Ag, and the third standard, AG-3, is a high grade value of 77.38 opt Ag.

Table 11.1 Silver-Gold-Copper-Lead-Zinc Standards

STANDARD	Ag opt	Cu%	Pb%	Zn%	Au opt
MEG-AG-1: Low grade Ag					
Certified Value	7.24	0.24	6.26	10.46	0.033
95% Confidence	6.92-7.56	0.220-.26	5.82-6.70	10.02-10.90	0.03-0.036
LABS AVG	7.24	0.24	6.26	10.46	0.033
MEG-AG-2: Average grade Ag					
Certified Value	8.72	0.25	6.50	11.24	0.031
95% Confidence	7.35-9.72	0.23-0.27	6.06-6.94	10.66-11.82	0.028-0.033
LABS AVG	8.68	0.25	6.50	11.24	0.031
MEG-AG-3: High grade Ag					
Certified Value	77.38	0.23	6.23	10.40	0.046
95% Confidence	68.48-88.08	0.21-0.25	5.81-6.65	9.84-10.96	0.040-0.052
LABS AVG	78.28	0.23	6.23	10.40	0.046

Figure 11.1 through Figure 11.3 are control charts of October 2011-June 2012 assay results from standards submitted to AAS. Standards are submitted as pulps, along with core or chip samples. The red lines represent the acceptable range of values for that standard, two standard deviations from the mean.

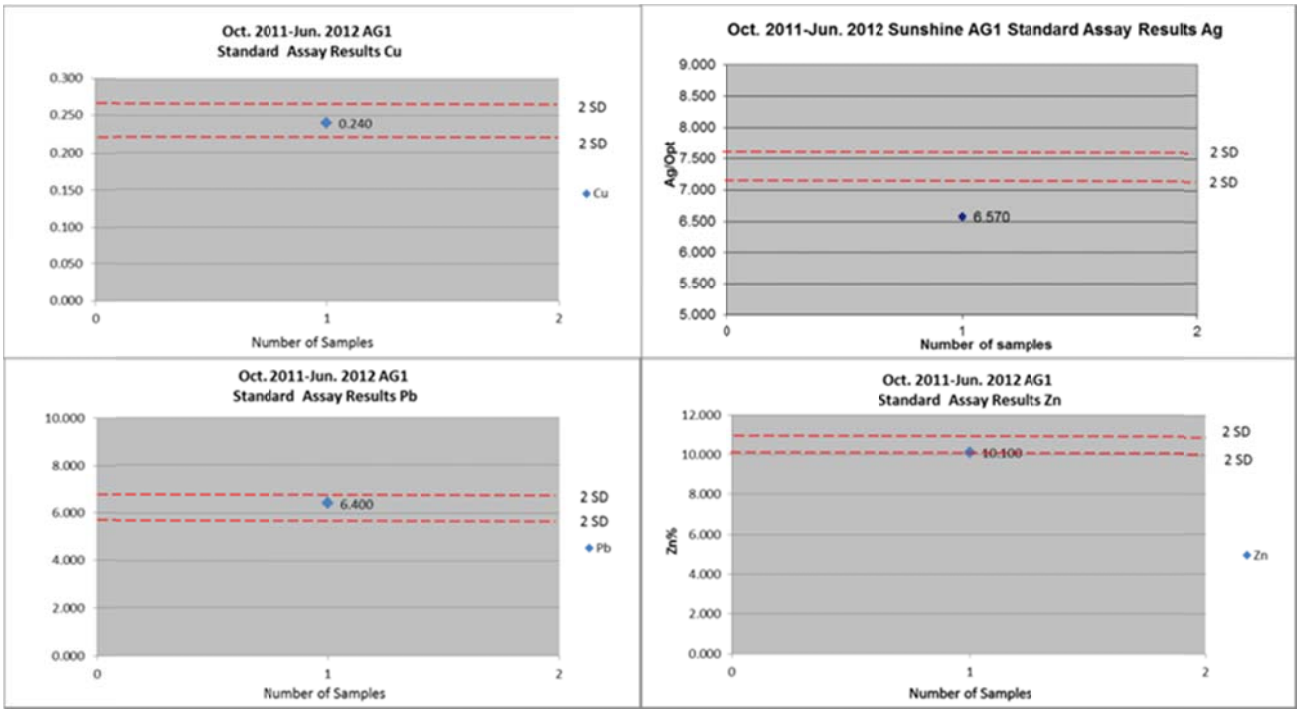


Figure 11.1 Results from Low Grade Silver Standard AG-1

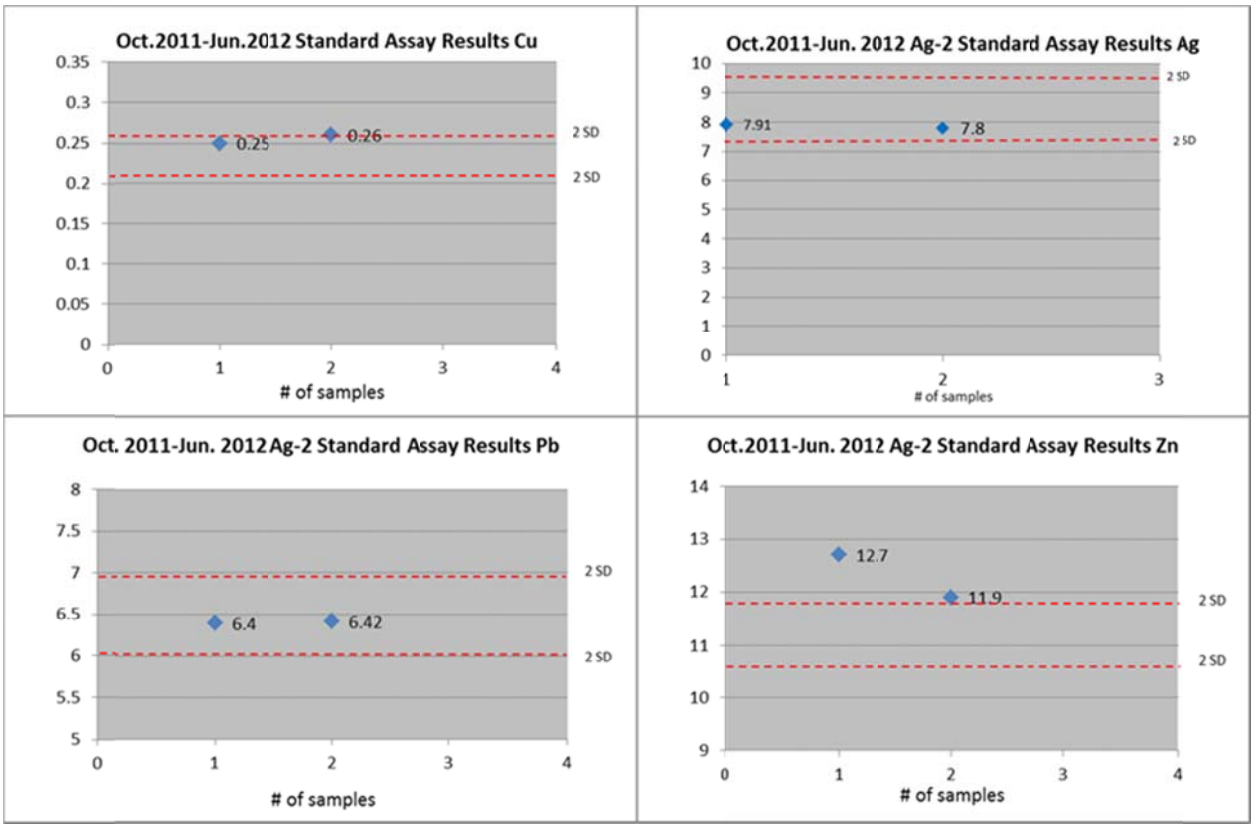


Figure 11.2 Results from Average Grade Silver Standard AG-2

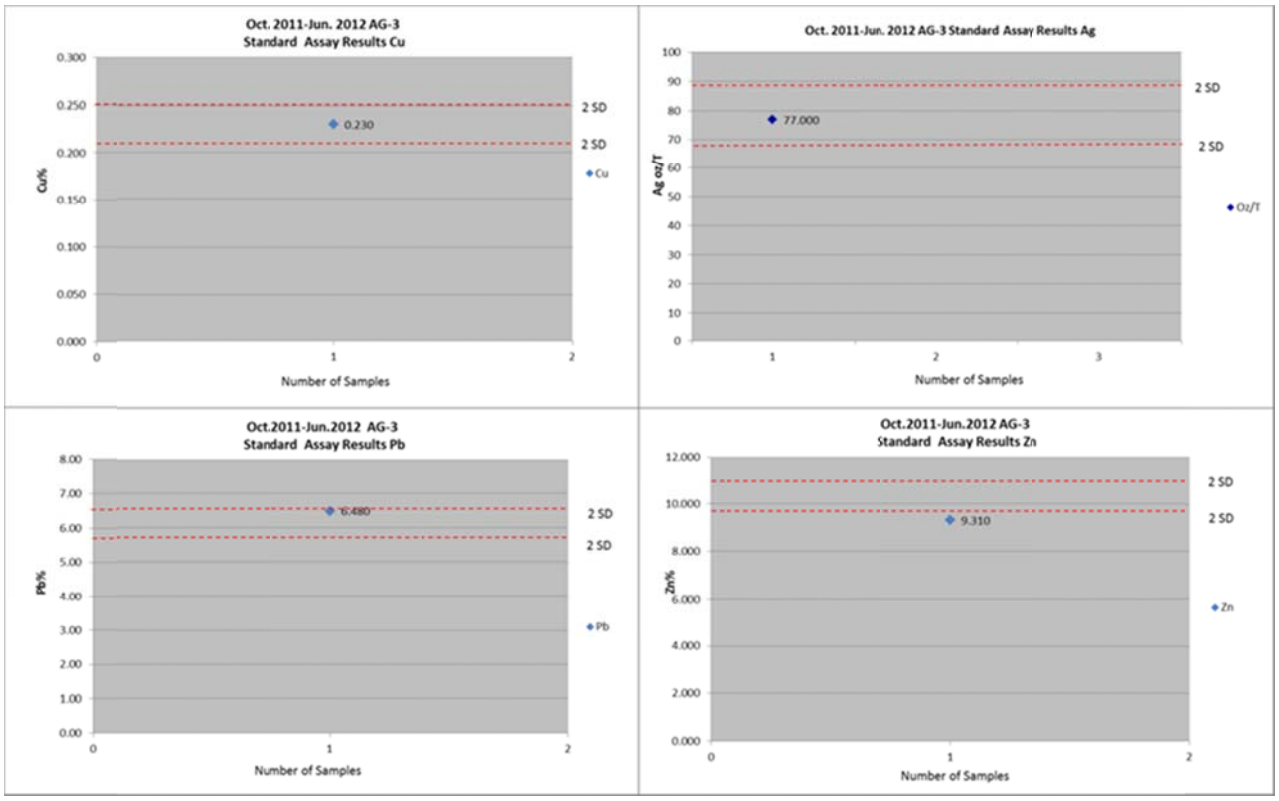


Figure 11.3 Results from High Grade Silver Standard AG-3

The correlation of the AAS mean results with the certified silver-gold-copper-lead-zinc standards is acceptable, with laboratory results averaging slightly lower than the standard. Laboratory result Zn values tend to drift slightly. The remaining results are within acceptable practice.

Table 11.2 presents baseline data for two Shea Clark Smith copper lab standards, CU-1 and CU-2. CU-2 standards have not been used at this time and are not shown in Figure 11.4. The acceptable range is the certified mean plus or minus two standard deviations. Plots for the Ag-Cu-Pb-Zn standards are shown below.

Table 11.2 Silver-Copper-Lead-Zinc Standards

STANDARD	Ag opt	Cu%	Pb%	Zn%
MEG-CU-1:Average copper				
Certified Value	0.80	0.48	0.10	2.53
95% Confidence	0.71-0.95	0.44-0.52	0.092-0.11	2.30-2.76
LABS AVG	0.80	0.48	0.10	2.53
MEG-CU-2: Low copper				
Certified Value	0.25	0.19	0.027	1.15
95% Confidence	0.22-0.27	0.17-0.21	0.024-0.030	1.03-1.27
LABS AVG	0.23	0.19	0.027	1.15

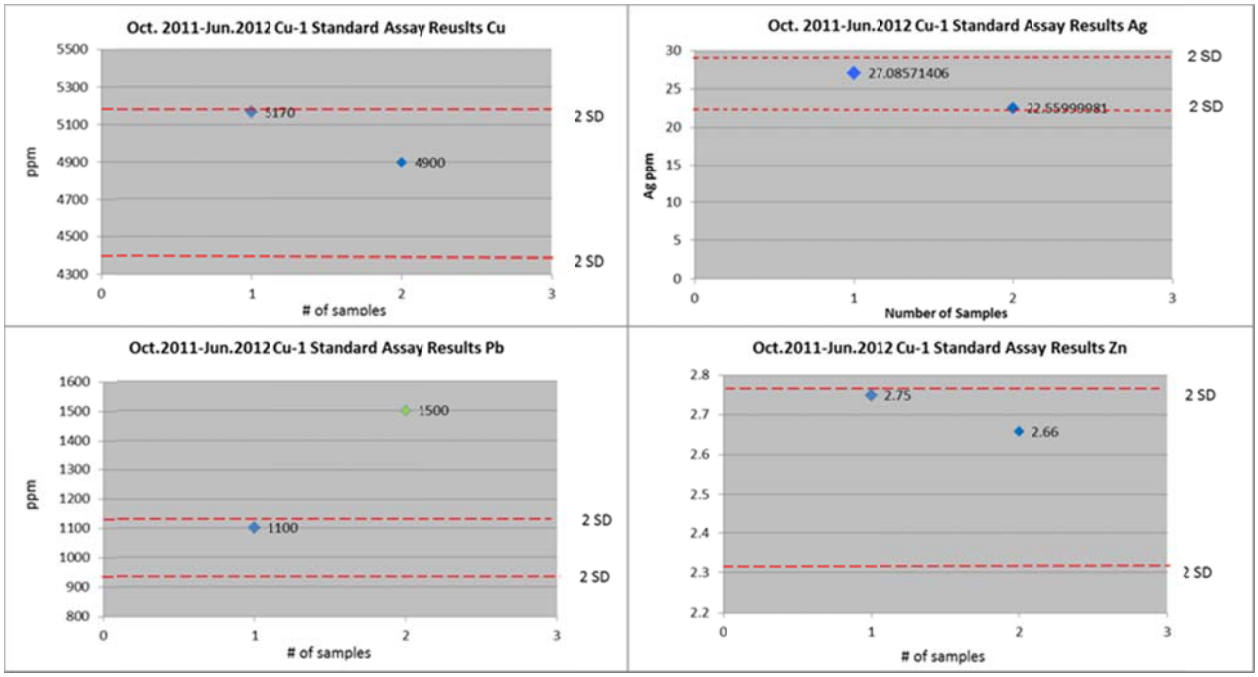


Figure 11.4 Average Grade Silver-Copper Standard Cu-1

The correlation of the AAS mean results with the certified high grade silver-copper standards is acceptable, with laboratory results averaging between two standard deviations. Laboratory result Pb values tend to drift slightly above average. Cu and Zn values are above the mean average and within acceptable practice.

11.8.5 Blank Samples

A blank sample, containing no detectable silver, gold, and base metal mineralization, is inserted at the rate of one blank for each 30 samples. Blank samples have been sourced from a third party supplier. The laboratory results are monitored by a designated person from the Sunshine Mine geology department. To date very minimal baseline data of standard and blank samples has been collected to present an accurate evaluation of the laboratory standards and blank samples. Figure 11.5 shows the results for the blank samples, with no major discrepancies above AAS's limit of analytical detection for Ag (0.100 opt Ag), Cu (0.01% Cu), Pb (0.1% Pb), and Zn (0.01% Zn).

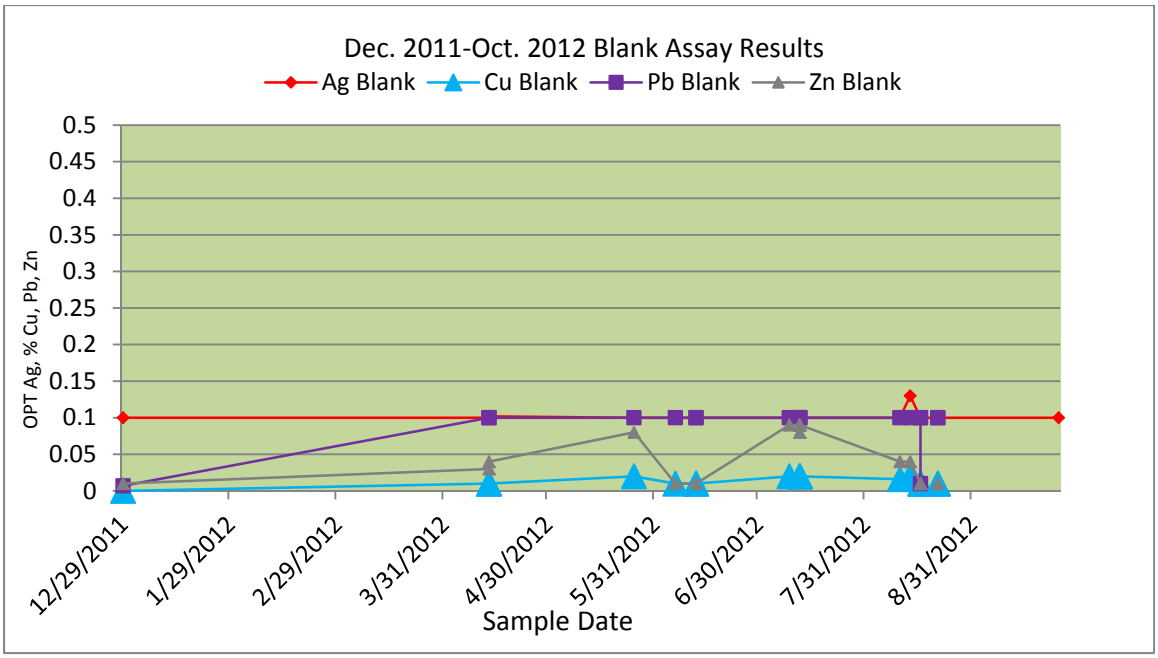


Figure 11.5 Laboratory Blank Assay Sample Results

SSMC site geological staff are continuously monitoring all on-going blank samples.

Shown below in Figure 11.6 is a scatterplot of the 2011-2012 check assays. The duplicates are rerun assays from AAS. A third party, Inspectorate, will be running further duplicate check assays as Sunshine continues its exploration program.

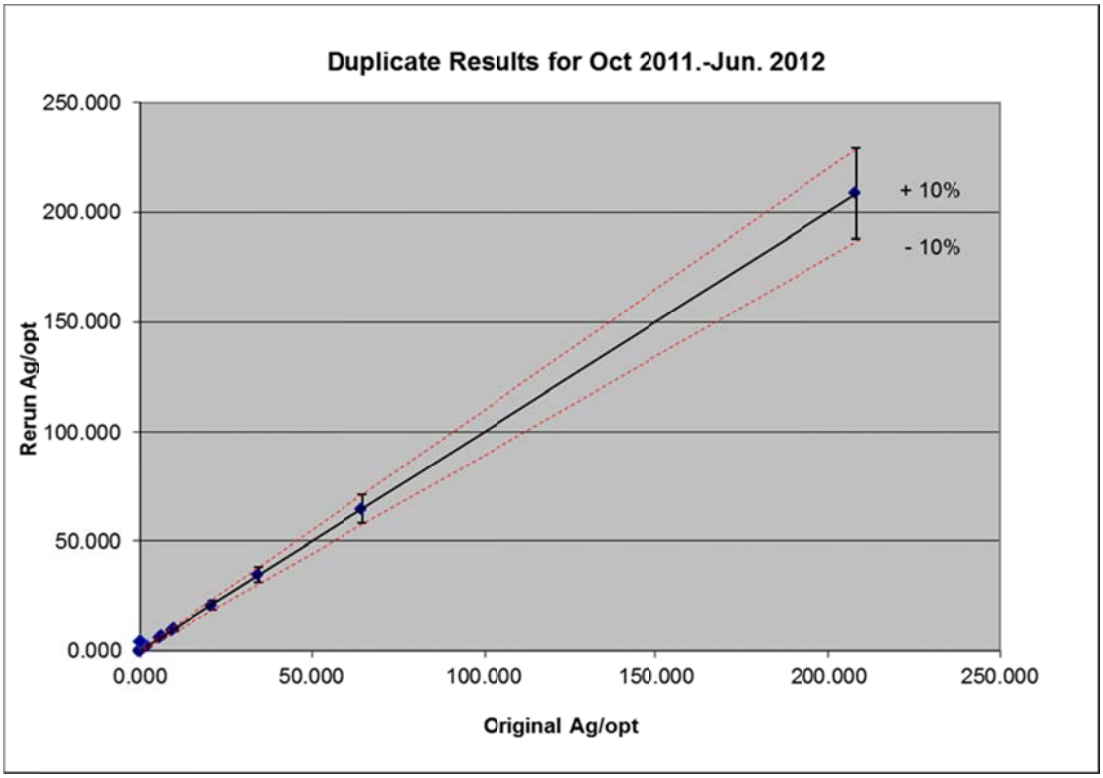


Figure 11.6 Duplicate Sample Results +/-10 % Confidence Intervals

Very little discrepancy was noted. Lower silver grades displayed are most variable. The lab is consistent within 10% of their duplicate samples.

11.8.6 Database

SSMC maintains a secure central database that contains all drill hole data. The database can only be accessed by approved personnel. The database is currently being expanded as new mining software is being initiated to support geology, modeling, and mine planning design software. A designated person from the Sunshine Mine geology department validates all assays prior to merging into the electronic database. Assays will only be accepted if established quality control parameters are met.

11.8.7 Adequacy

In the author's opinion the sample preparation, security, and analytical procedures followed by SSMC are adequate to support a mineral resource estimate.

12.0 DATA VERIFICATION

12.1 Historic Data Verification

The author has not been able to physically verify level channel assay samples, stope channel assay samples, or historic mine workings. Drilling and drifting conducted by SSMC in 2011 and 2012 in the Upper Country areas show similar thin and high-grade Ag mineralization as historically collected data. The author has reviewed level sampling maps, stope sampling maps, and mine working long section maps. Data used to develop stopes and collected while developing stopes has been considered verified

In the process of resource estimation, historic data sources have been checked against each other and show good correlation and reliability. For instance, level plan assays and stope assays show a good correlation when viewed simultaneously. Level plan triangulations show evidence of raises where long section mine working maps show raise locations. Mined out areas consistently correlate with the block model as areas of highest grade x thickness. Following the completion of resource estimation, a mined out reconciliation versus historic production was completed. The results show areas flagged in Tetra Tech's block model, reasonably account for the total ounces historically mined.

It is the author's opinion that all sources of data are internally consistent and have supported several decades of mining and are of reasonable reliability and suitable for use in resource estimation.

12.2 Assay Data Verification

Tetra Tech was provided with PDF files directly from AAS containing 257 analyzed assay intervals. Of the 257 assay intervals received from the laboratory 128 assay intervals were able to be compared to data provided to Tetra Tech by SSMC. The 128 intervals reflect all of the 6725 West and East Stope level drift assaying and a portion of the Upper Country underground and surface drilling. A comparison of the 128 intervals shows no major discrepancies. Minor issues with detection limits were observed. Assay results lower than detection limits were inserted into the SSMC database as "0.0". Tetra Tech would recommend detection limit assays be recorded as "<" and the numeric detection limit at the time of testing in the database. Tetra Tech reviewed several vein intersections in drill core during the site visit on February 9, 2012. It is the author's opinion that data provided by SSMC was represented accurately and is suitable for use in resource estimation.

12.3 Metallurgical and Process Verification

The Sunshine Mine in Kellogg, Idaho has operated for many decades with progressive development of several underground high grade veins containing silver, copper, lead, zinc, and iron sulfides. While complete records of the operation over those decades are not available, much of the historical data survives in the form of plant records, shift notes, concentrate shipments, umpire assay reports, plant analytical profiles, and some retained duplicate shipment samples. Analytical requirements, analytical capabilities, and environmental requirements have all changed over the intervening years.

SSMC management intends to continue mining on vein structures that have historically provided feed to the Sunshine mill when the mine reopens. Historically, a large volume of concentrates produced at the Sunshine mill have come from ore mined from the Sunshine vein structures

which have extended to the historic depths of the then operating mine. Several other vein structures also exist both laterally and at depth in the mine and have been mined lying parallel to the Sunshine veins. The Sunshine vein systems contained high grade silver ore in the form of freibergite, one of the minerals in a series comprising tetrahedrite as an end member. Silver is also contained in galena and in lesser amounts of other minerals in some of the vein structures contained within the Sunshine Mine and in other mines in the Coeur d'Alene district.

While some attempts at changing the concentrates produced at the Sunshine Mine have occurred over the years, historically Sunshine has produced a high grade silver-copper concentrate, and a lead concentrate containing lesser amounts of silver. While the Sunshine Mine has historically sold its lead concentrate with contained silver to various lead smelters, it has at various times treated the concentrate to reduce the content of antimony to render it saleable or to reduce the antimony penalty prior to smelter sale, or treated the whole concentrate to produce antimony, and to recover the silver and copper as refined metals. Information concerning some of these products is available from the historical record although not in a continuous fashion.

Suitable metallurgical test samples from the available upper levels of the mine have been obtained to determine a metallurgical process, including the production of two flotation concentrates as have been historically produced, and to confirm metallurgical recoveries. The samples collected were for optimizing the flotation schemes and to determine information for the design of the mill, such as thickening and filtering of concentrates.

Prior to the above samples being taken, samples were obtained from an existing ore stockpile and the fine ore bin to allow comminution testing. The author was involved in the selection of the samples and delivery of the samples under chain of custody to Phillips Enterprises LLC, Metallurgical & Testing Services (Phillips Enterprises), in Golden, Colorado, which conducted the required grinding tests to determine the energy requirements for milling and the variability amongst the samples (4) that were obtained and tested by the laboratory.

The results of grinding testing of the four samples at Phillip Enterprises were very close with Bond Work Index numbers of about 12.5 kWhr/ton, indicating a medium soft ore for grinding.

The numbers used in the MetSim model and for projection of mass flows for the PEA came from historical feed and concentrate data obtained from the Sunshine Mine historical information that was available. Preference in choosing data was given to years late in the 1990s as those years most closely reflect the most recent mill flowsheets and the ore blends available for milling. The historical data is not sufficient to specifically correspond the mine depth, stope, or vein information to specific mill results. This was because the feed to the mill on any given day was a blend of many stopes and mine elevations with no specific record of origin other than grade.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineral Processing

The Sunshine Mine Project will produce two flotation concentrates; one rich in silver and copper, the other in lead. These will be produced from the crushing, grinding, and flotation of freibergite-rich ore that has been mined on this site for more than a century within the Coeur d'Alene Mining District in northwestern Idaho. The economically important sulfide minerals in the ore are tetrahedrite (freibergite is a silver rich member of the tetrahedrite series of minerals) and galena with associated argentite, a silver mineral.

The anticipated flotation system will use a rougher and scavenger system with separate three-stage silver cleaning to produce a silver-copper concentrate and a lead concentrate. The process block flow diagram shown in Figure 13.1 serves as the basis for the following discussion. Ore from the ball mill is ground to 60% passing 200 mesh and proceeds through the rougher-scavenger flotation systems in series. All tailings depart from the third-stage scavenger and report to the tailings thickening, filtration, and disposal systems. Concentrate from the third scavenger is recycled to the front of the second scavenger. Concentrate from the first silver rougher is cleaned to produce a high grade silver-copper concentrate. The tails from the first silver cleaner and the concentrate from the second silver rougher/scavenger is screened with the oversize going to a regrind mill to increase the liberation of these recycled products.

Ninety-five percent of the undersize from the regrind circuit will pass a 325 mesh screen and proceed to the lead flotation circuit. The tails from the lead flotation is returned to the third silver scavenger to recover liberated silver mineralization with a second opportunity to go into a concentrate. The lead circuit concentrate can be filtered as a lead concentrate product if sufficiently high in lead. Alternatively, the lead circuit concentrate can be returned to the first silver rougher flotation cell to recover silver values if they are sufficiently high or returned to the second silver scavenger to help increase the lead values in the final lead concentrate

These provisions in the flotation circuit are necessary as the content of lead in the feed to the mills has historically been varible depending upon the stopes that are producing the feed to the mill.

The historic mill block flow diagram has been included in this report to provide a visual understanding of the flow of minerals in the process. Current metallurgical testwork may vary the flowsheet somewhat from the historical flowsheet described to avail the current project of improvements in reagents and flotation cell design. However, the essential issue for the process is to separate the silver rich freibergite mineral from the lower silver grade galena/argentite mineral assemblage.

Based on over 90 years of production history there are no known factors which should have a negative economic effect on recoveries.

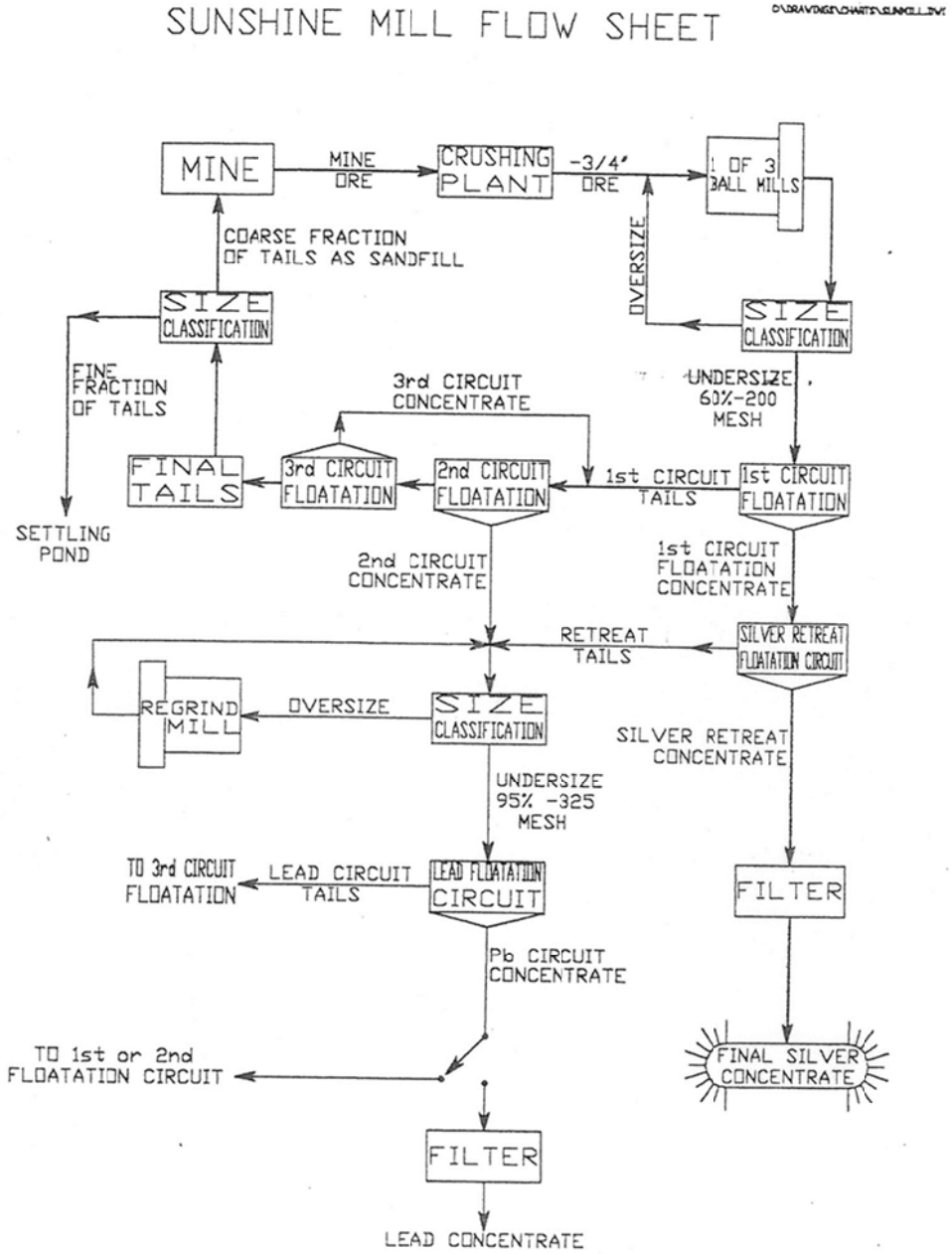


Figure 13.1 Historic Sunshine Process Block Flow Diagram

13.2 Metallurgical Testwork

Comminution testing has been completed by Phillip Enterprises and flotation testwork is currently underway by G&T.

The primary objectives of the metallurgical comminution testing and the results are detailed below:

- Sample preparation
- Crusher work index tests
- Ball Mill grindability tests
- Abrasion index tests

Four samples were crushed and screened appropriately for abrasion and ball mill tests. Abrasion test samples were screened to extract a ¾ inch by ½ inch size fraction and ball mill samples were prepared by stage crushing and screening to less than 6-mesh.

The crusher work index tests were conducted on natural rock samples according to test protocols and the results are summarized below Table 13.1.

Table 13.1 Crusher Work Index Results

Sample	CWi (kW-hr/st)	CWi (kW-hr/mt)
SR1 Sunshine Vein Mine Run	3.31	3.65
SR2 Decline Waste Rock	4.30	4.74
Ore Bin	5.21	5.74
Rock House Feed	4.98	5.49

Ball mill grindability tests were conducted by standard practice using 100-mesh (150 micron) closing screens. The ball mill work indices are shown below in Table 13.2.

Table 13.2 Ball Mill Work Indices

Sample	BM Wi (kW-hr/st)	BM Wi (kW-hr/mt)
SR1 Sunshine Vein Mine Run	12.47	13.75
SR2 Decline Waste Rock	12.58	13.87
Ore Bin	12.13	13.37
Rock House Feed	12.54	13.83

Abrasion index tests were conducted on a -¾ inch by ½ inch fraction of each sample according to the standard abrasion test protocol and the results are listed below in Table 13.3.

Table 13.3 Abrasion Index of Samples

Sample	Abrasion Index
SR1 Sunshine Vein Mine Run	0.0805
SR2 Decline Waste Rock	0.0504
Ore Bin	0.1187
Rock House Feed	0.0939

The vein sample material will be representative of the run of mine (ROM) mill feed with the variations coming in minor amounts of subsidiary minerals. The waste samples will be representative of the material found in 70% of the mine with the remaining mineralized material also being more quartzitic in context and therefore harder and more abrasive.

13.2.1 Metallurgical Testwork Objectives

The primary objectives of the G&T metallurgical testing are to determine the following:

- Ore hardness and metallurgical response of composite samples
- Grindability
- Flotation scheme and reagent optimization
- Process equipment selection data

To achieve these objectives, the test program is divided into two phases. Phase I will include bench scale testwork to investigate aspects of flowsheet assessment and development. This testing will be conducted on six composite samples and a selection of variability samples. Phase II will include pilot plant testwork on larger two tonne composite samples to produce the process stream mass required for vendor testwork.

13.3 Refining

Preliminary processing assumptions are based on a flowsheet that assumes a grinding and flotation circuit which will produce a high grade silver-copper concentrate. While silver grade and mineralogical characteristics are fairly constant, there are areas of the mine where lead is more prevalent and the amount of lead feed to the concentrator can be expected to be variable over time. Lead has previously been floated in a separate concentrate, containing relatively minor amounts of copper and variable amounts of silver, refined separately from the silver-copper concentrate.

For the initial year of plant operation, both the silver-copper and lead concentrates will be shipped to commercial refineries or smelters. Beginning in the second year of plant operation, the silver-copper concentrate will be refined at an onsite refinery to produce copper cathode and silver doré. The lead concentrate will continue to be shipped offsite for processing at a commercial smelter.

During the later part of historical production the Sunshine refinery was leaching the silver-copper rich freibergite ore to remove the antimony that is incorporated as part of the freibergite mineral. The antimony was treated to produce a commercial product in the later stages and was removed to reduce antimony penalites from the smelters in the earlier days of the operation. The antimony reduced residue was further processed in the Sunshine refinery to produce a high grade copper product and silver doré.

As the basis for the PEA, SSMC has opted to proceed directly to the refinery without an antimony leach step in order to produce a copper and silver product and an antimony rich residue.

Under this plan, the concentrate from the mill would be fed to batch autoclaves which would operate with Sunshine's historical nitrogen catalyzed leach process. This process plans to produce solubilized copper and silver and leave the antimony in the residue likely as a ferric antimonate. The residue, if sufficiently high in antimony, may be sold for further processing to another company.

The pregnant solution containing the silver and copper would proceed to a copper solvent extraction circuit where the copper would be removed from the solution and processed into copper cathodes by electrowinning. These copper cathodes would be directly marketable.

The solution containing the silver nitrate concentrations and other lesser value elements would proceed to the silver refinery where the silver would be precipitated from solution and then resolubilized in a more concentrated form in nitric acid. The silver would then be electrowon from the silver rich solution and ultimately cast into doré silver bars.

14.0 MINERAL RESOURCE ESTIMATES

This mineral resource estimation was completed by Tetra Tech in MicroMine® mining software utilizing data supplied by SSMC. This estimation is the first time a digital resource estimate has been calculated for the Sunshine Mine and is a direct result of an extensive data digitizing effort by SSMC. Tetra Tech was commissioned by SSMC to develop a digital 3D estimation that includes all available information and can be evaluated at various cutoffs.

Utilizing drill hole and channel assays data, Tetra Tech created best fit vein surfaces and estimated mineral resources along those vein surfaces. Ordinary Kriging was used to estimate 3D points along a string type block model for 38 veins. The results of the mineral resource estimate are tabulated in Table 14.1. This mineral resource has been diluted to a fixed mine width of 6.5 ft and is an insitu resource estimate. A base-case cutoff has been applied but no other economic parameters have been considered and all metal recoveries are assumed to be 100%. Royalties do exist in certain areas of the mine and have not been considered for this mineral resource estimate.

Table 14.1 Mineral Resource Estimate Sunshine Silver Mine

Resource Class	Cutoff Ag Opt Diluted	Tons Diluted	Grade Ag Opt Diluted	Ag Contained Ounces	Cu %	Pb %	Zinc %
Measured	10	1,215,000	24.6	29,890,000			
Indicated	10	1,960,000	21.6	42,410,000			
Measured + Indicated	10	3,175,000	22.8	72,300,000			
Inferred	10	9,115,000	24.4	222,050,000	0.26	0.35	0.02

14.1 Sources of Data

Assay data necessary to facilitate this resource estimate was derived from three sources: drill holes, level drift channel samples, and stope channel samples. Drill hole assays were provided by SSMC from a collated drill hole database, encompassing historic and modern drilling. Level drift channel sampling was digitized from historic level plan maps. Stope channel samples were digitized from historic stope production sheets. Convention of channel sampling in the Coeur d'Alene Mining District and discussions with SSMC staff indicate channel sampling was conducted across true thickness where true thickness can be determined. Because channel sampling was conducted across true thickness and drill holes were corrected for true thickness, all three data sources were able to be used simultaneously for resource estimation.

In addition to assay data sources, level plan vein sketches, development triangulations, and vein map long sections were essential for resource estimation.

14.1.1 Vein Sketches

To establish a digital representation of vein locations, SSMC geologists traced all veins on a level by level basis from large scale level plan scans. The vein sketch was then digitized into AutoCAD and assigned true elevation based on level. The vein sketches were essential for this resource estimate and provided a basis for spatial location of the veins and also a normalized nomenclature for veins throughout the project. Figure 14.1 shows the vein sketch color coded by vein name; due to the quantity of veins, colors have been used more than once to represent different veins.

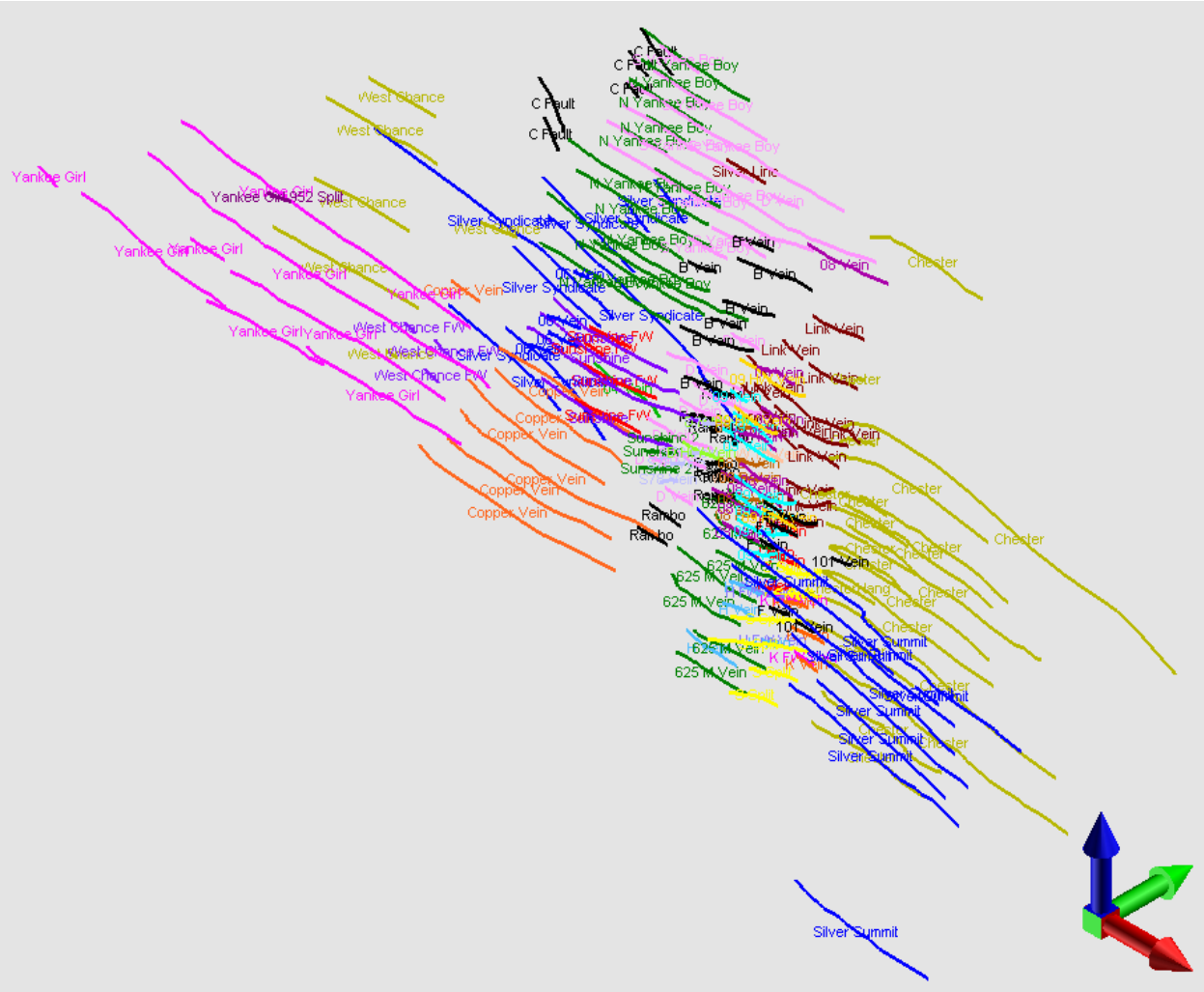


Figure 14.1 Vein Sketch, Looking NW from Above

14.1.2 Development Triangulations

Tetra Tech was also provided with levels, ramps, and shafts as 3D triangulations. Level triangulations were used by Tetra Tech to orient level samples in 3D space. Figure 14.2 shows level, ramp, and shaft triangulations. Historically levels were referenced to three different 0 levels: Sunshine, Consolidated Silver, and Polaris. Sunshine is the most widely used reference throughout the mine.

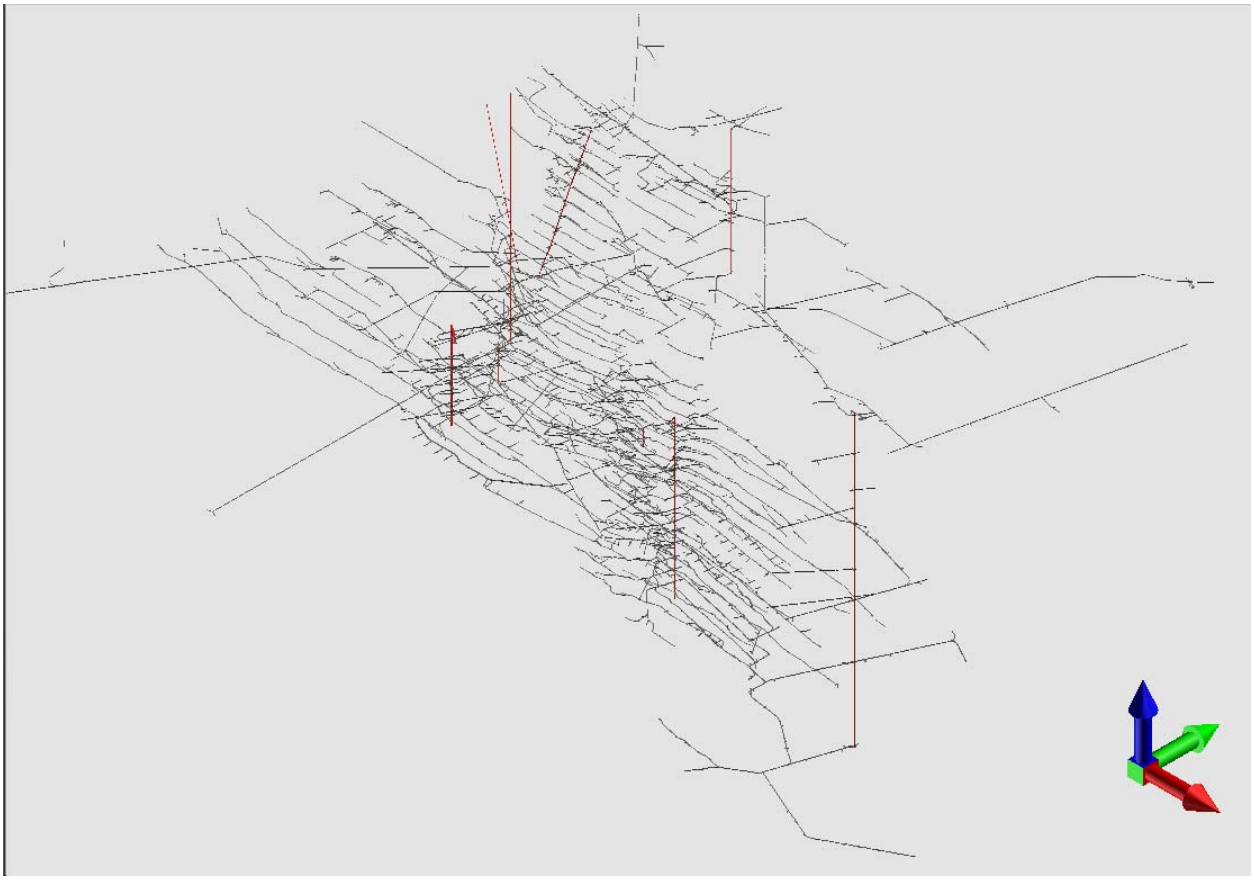


Figure 14.2 Development Triangulations, Looking NW from Above

14.1.3 Vein Map Long Sections

Tetra Tech received AutoCAD® files containing 2D vein map long sections. The long sections included location references, drill hole pierces, mined out outlines, raise locations, and areas of potential resources identified prior to this resource estimation. Historically the long sections represent a graphical record keeping of all necessary information on a vein by vein basis. Boundaries representing mined out areas were constructed from smaller scale, more detailed stope production maps. Long section maps were the primary source of accounting for mined out areas. The long section maps were scaled and brought into the resource estimation software MicroMine. The majority of the vein long section maps are oriented looking north but many have slight bearings off of east-west. Veins oriented more than 10 degrees off of east west are often represented in a best fit vein strike.

Figure 14.3 shows the West Chance Vein long section map. The green and black areas represent mined out areas. The red areas represent previously identified resource locations. Level references and easting locations are also shown in Figure 14.3.

Tetra Tech has not physically confirmed the vein map long sections underground, but the weight of evidence indicates they are a reasonably reliable source for accounting for mined out material.

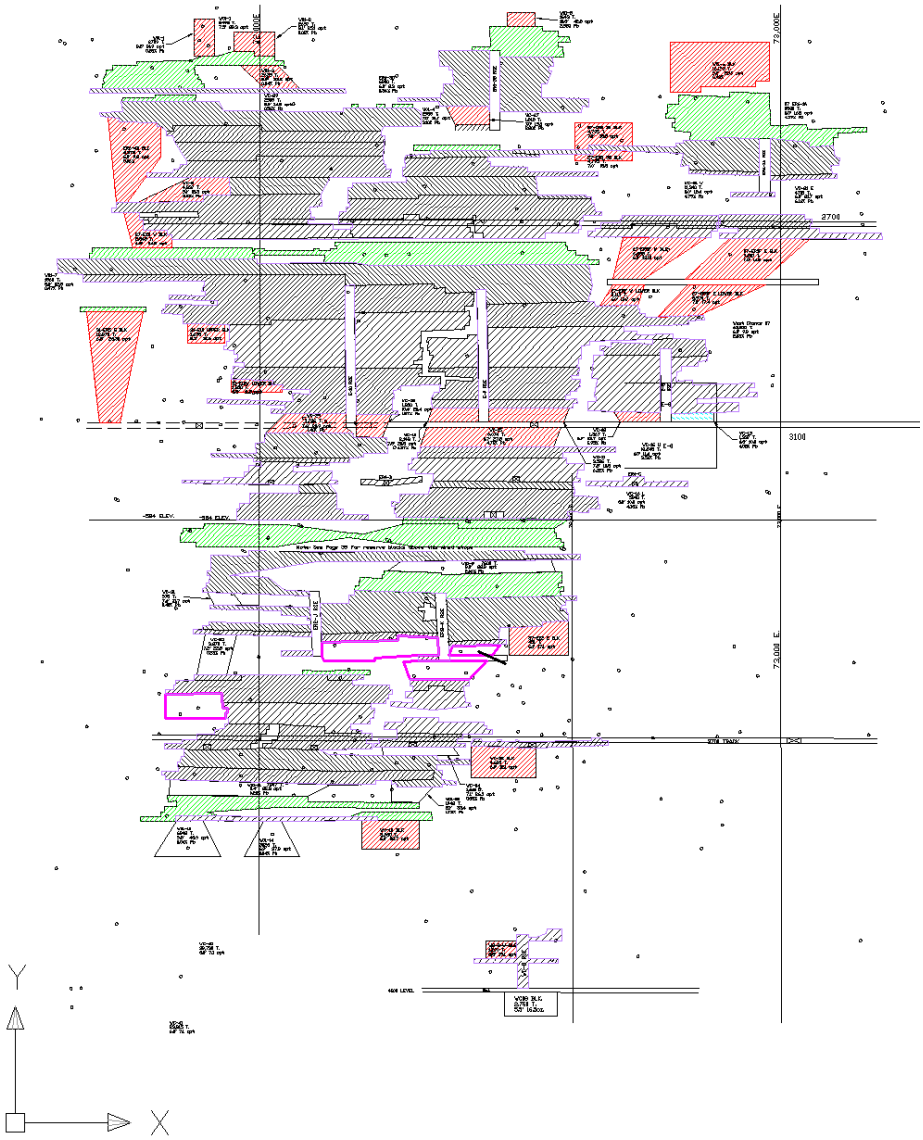


Figure 14.3 West Chance Long Section AutoCAD® Map, Looking North

14.1.4 Drill Hole Database

A collated drill hole database, encompassing historic and modern drilling was provided by SSMC. This database included all available drilling results. However, drill holes outside the project area or without accompanying survey data were not used in computing this resource estimate, leaving usable data based on 2,400 drill holes. The drill holes used contained 9,315 assay samples for 15,860 ft of sampling with an average sample length of 1.7 ft. The assay file contains assays for Ag, Pb, Cu, As, Sb, Zn, and Fe, but primarily intervals were assayed for Ag and often Cu and Pb. Assays for As, Sb, Zn, and Fe are far less prevalent.

As received, assay intervals in the drill hole database were not composited across veins or flagged/associated with veins. Tetra Tech used the vein sketches to flag assay intervals with

corresponding vein intersections. Figure 14.4 shows the drill holes and assays looking west. Figure 14.5 shows the drill holes from plan view.

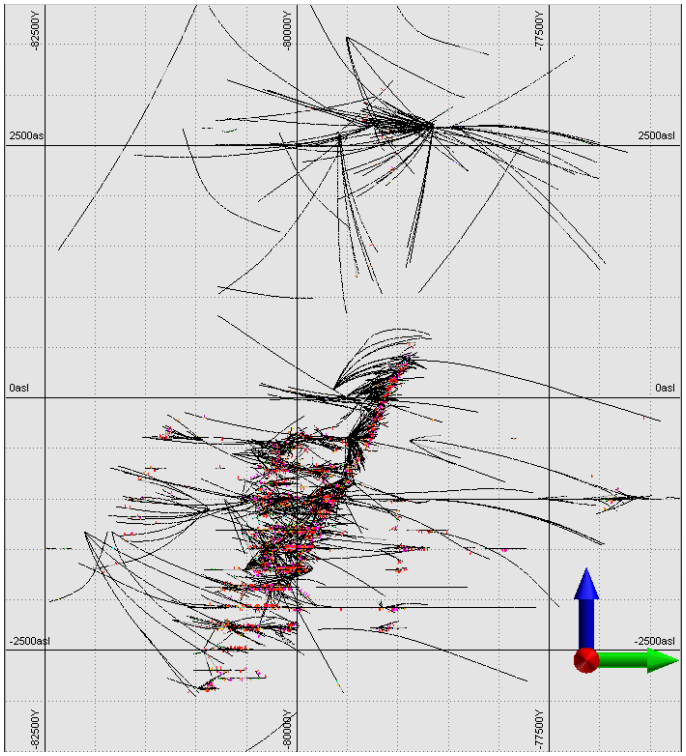


Figure 14.4 Drill Holes and Ag Assays, Looking West

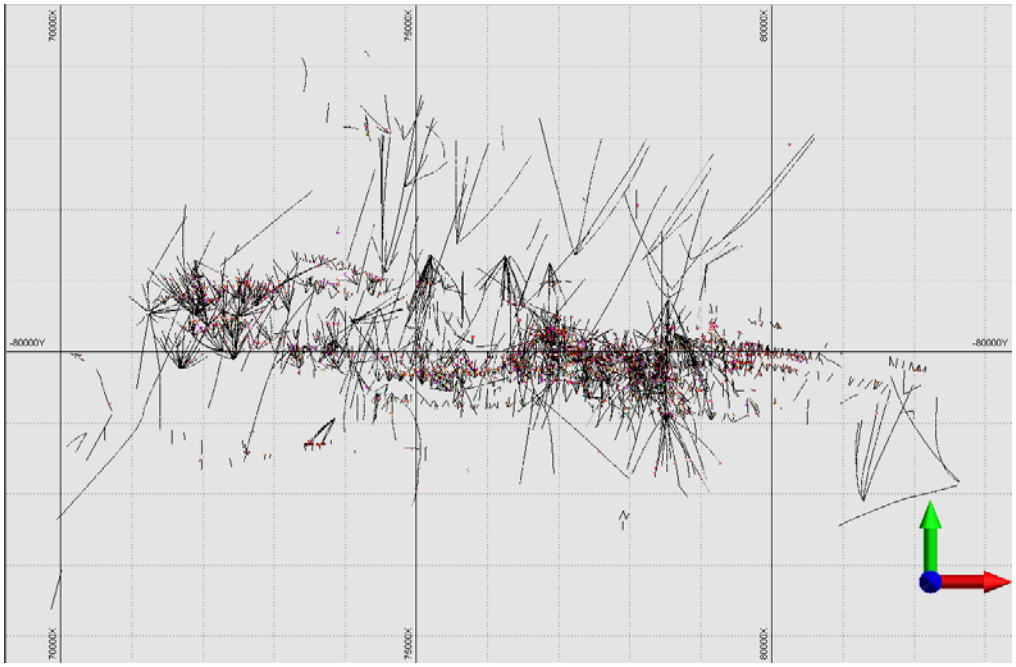


Figure 14.5 Drill Holes and Ag Assays, Plan View

14.1.5 Level Drift Channel Assay Samples

Level drift channel assay samples were digitized from historic large format level plan maps by MDA and Tetra Tech. Level plan maps contain illustrations of level drifting, grid markers, level labels, and sample locations. Sample locations were annotated with true thickness, Ag opt, and often original sample numbers and Cu% or Pb%. Figure 14.6 shows a level plan map annotated with sampling information.

Figure 14.6 Sunshine Vein Level Plan Map, Levels 3100 and 3250

Digitized level samples by MDA and Tetra Tech were combined into a single database containing a digitized sample identification number, location (easting, northing, and elevation), level number, level reference, width, and Ag, Cu, Pb, and Zn content. Several historic level plans overlap each other, resulting in duplicated data entry. Tetra Tech has identified such instances by proximity search and labeled samples as duplicate and isolated them from resource estimation. The level sample database contains 26,804 records, 5,924 of which have been identified as duplications. Figure 14.7 shows the level plan assay database in 3D.

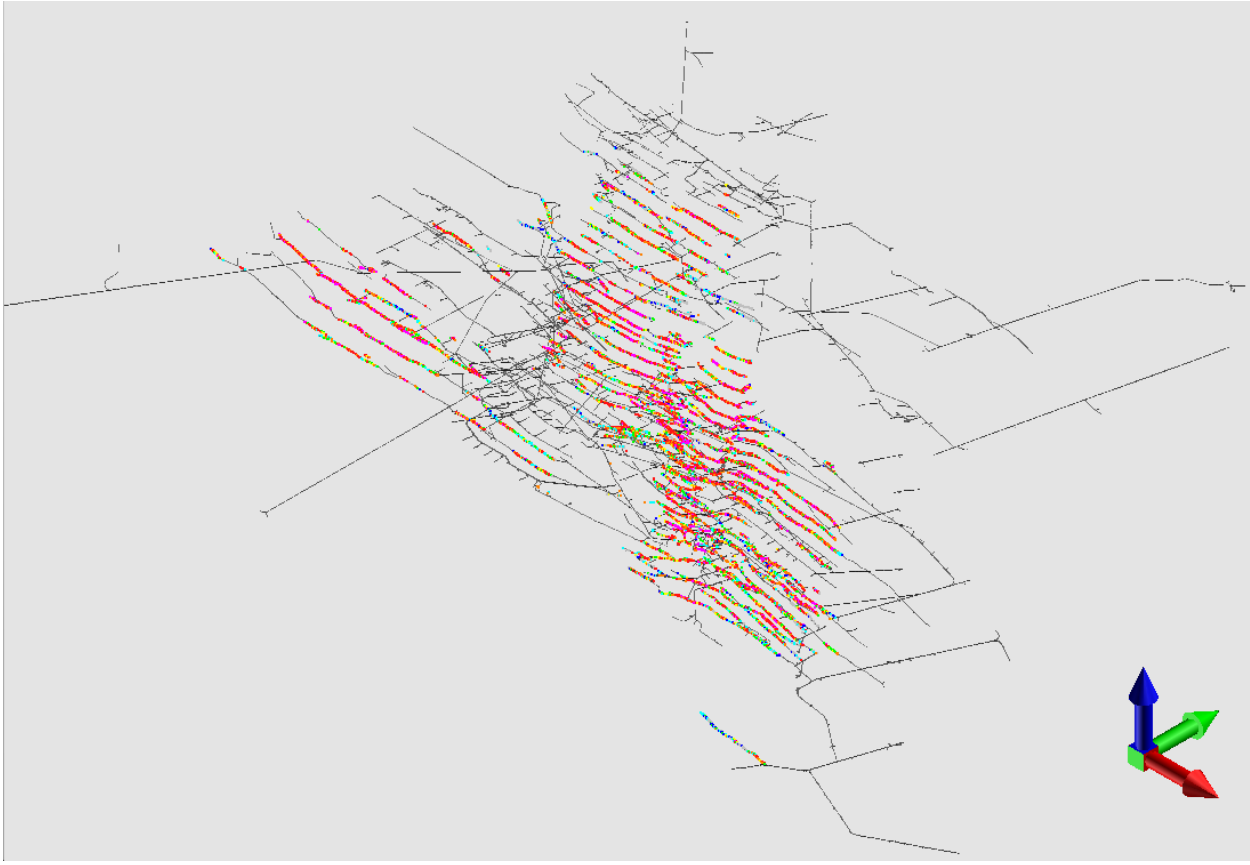


Figure 14.7 Level Drift Channel Ag Assay Samples and Level Triangulations, Looking NW from Above

14.1.6 Stope Channel Assay Samples

Stope channel assay samples were digitized from historic stope production books by MDA. The stope books were scanned and digitized page by page. It is important to note that not all stope books and sheets were located and digitized. Figure 14.8 shows a stope sheet from the Chester Vein.

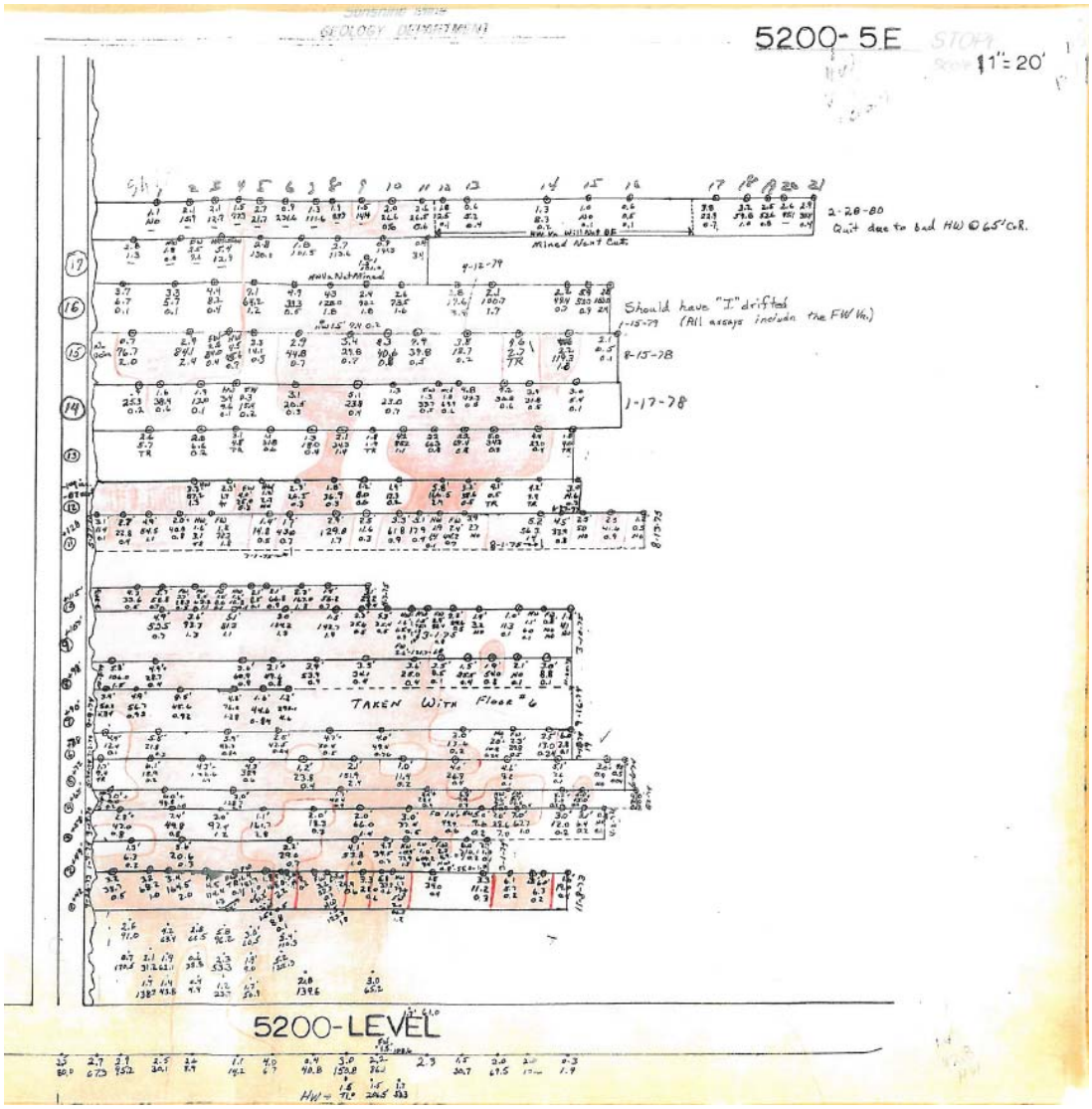


Figure 14.8 Chester Vein Stope Sheet, Level 5200

Stope sheets are annotated with true thickness, Ag opt, and often Cu% or Pb%. The stope channel assay sample database contains 92,287 records. In some instances mining occurred locally in the hanging wall or footwall of a main vein stope following small split or splays veins. These samples are flagged by a hanging wall or footwall with a numeric offset and have not been considered while estimating the main veins. In a few instances, stope sheets overlap other stope sheets or have been entered twice. These samples have been flagged as duplicates; 2,863 samples have been so identified. Only a limited amount of stope sheets could be located and digitized. An exact percentage of digitized stopes versus total stopes has not been calculated, but by visual inspection can be estimated at 30-50%. Various factors contribute to the availability of stope sheets, the biggest being time period of when the stope was mined. Figure 14.9 shows the stope channel assay database in 3D. Figure 14.9 shows data gaps in the nearer surface or Upper Country areas that were mined in the first half of the 20th century. Many

stope channel assay sample data gaps are covered by level drift channel assay samples, as shown in the Upper Country in Figure 14.7 above.

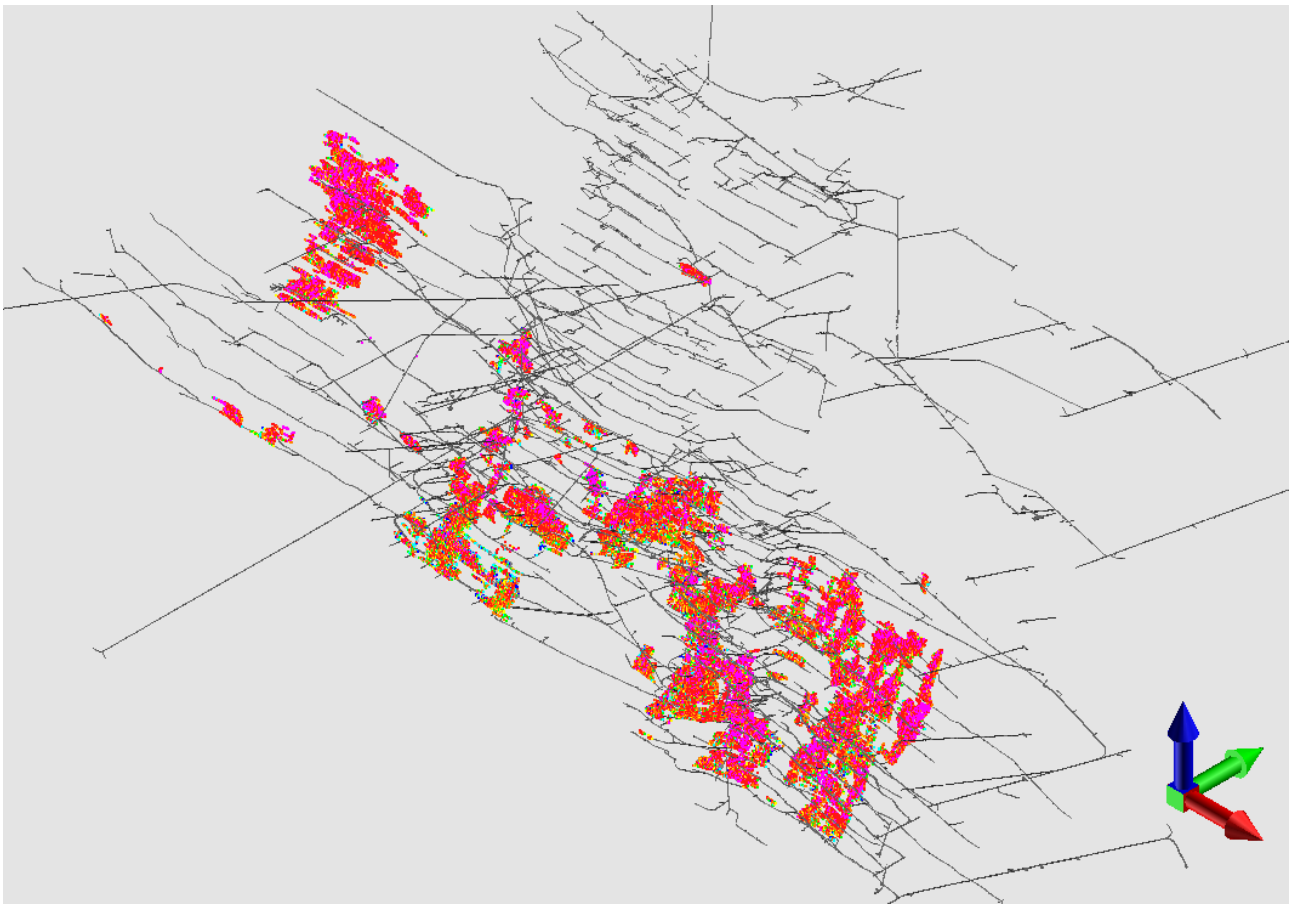


Figure 14.9 Stope Channel Ag Assay Samples and Level Triangulations, Looking NW from Above

14.2 Vein Modeling

This mineral resource estimation was conducted on a vein by vein basis. Vein assignments emanated from the vein sketches. Using the vein sketches as a guide Tetra Tech assigned drill hole assay intervals with vein names. Using the drill hole assays and drift triangulations that coincide with the vein sketches, points were placed on the hanging wall and footwall of the drill hole assay intervals and drift triangulations. The hanging wall and footwall points were then brought into MapInfo® GIS software where approximate vein centerline surfaces were created using a minimum curvature gridding method. The centerline grids were then brought back into MicroMine, converted to triangulations, and verified with the vein sketches, drifts and assay intervals. The vein centerline triangulations are an approximation of vein centerlines and were not “snapped” to the drilling, and, in effect, deviate from exact drift and drill hole intersections. Due to the large number of points it was not possible to force the surfaces to respect the points exactly. The vein centerline triangulations were then clipped by a 500 foot buffer, based on a convex hull created from the points used to create the surface. Following clipping by the buffer, all vein centerline triangulations were then compared to their neighboring vein centerline triangulations and clipped where overlaps were present. In a few instances where veins cross-

cut each other, such instances where verified by SSMC staff interpretations. Figure 14.10 shows vein surface triangulations.

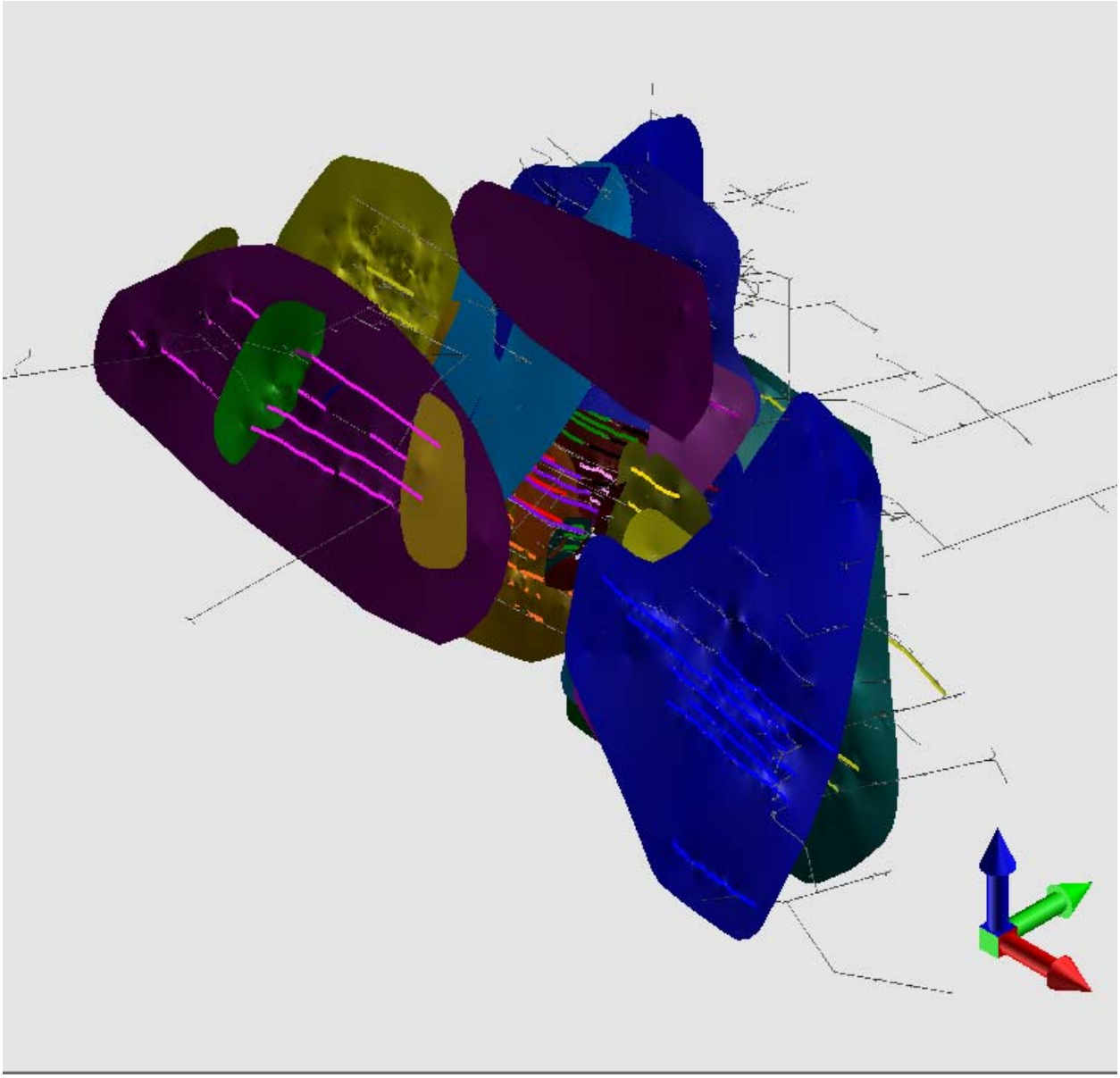


Figure 14.10 Vein Centerline Triangulations and Vein Sketches, Looking NW from Above

14.3 Block Model Structure and Preparation for Estimation

A single block model was established for each vein. Block size was determined by the best estimation of a single mining unit, being 10 ft in the elevation dimension and 10 ft in the east dimension. The thickness of each block was assigned a temporary value of one foot, later to be assigned a variable thickness based on undiluted vein width during the Kriging process. Each block model has only one row in the northing direction. Block models were draped to their corresponding vein centerline surface in a northing orientation.

14.3.1 String Model Setup

Due to the dip and strike variability of the vein surfaces, an ideal block representation would conform to dip and strike locally with the vein surface. To achieve this Tetra Tech used a string model. After draping, each block centroid was evaluated by its orientation to the next block centroid in the string along the string models columns and levels. If two consecutive centroids sharing the same easting are separated by a line with a dip other than 90 degrees, the centroids will be further apart than the minimum distance of 10 ft. The adjusted separation distance was assigned as the new block length in the elevation direction. If two consecutive centroids sharing the same elevation are separated by a line with a strike other than 90 degrees, the centroids will be further apart than the minimum distance of 10 ft. The adjusted separation distance was assigned as the new block length in the easting direction. Between 78,100 easting and 78,500 easting, many of the vein structures change strike from east west to northeast. Due to this shift, block model string lengths in the east direction are much larger; therefore represent more volume than the majority of the blocks that have a near east west strike and an average dip of 60 degrees. Figure 14.11 is a plan view section at -1025 ft elevation showing the 09 Vein changing strike to the northeast at 78,100 easting. The vein sketch is shown in cyan. Vein centerline is shown in black and the block model centroids are shown as cyan dots along the vein centerline.

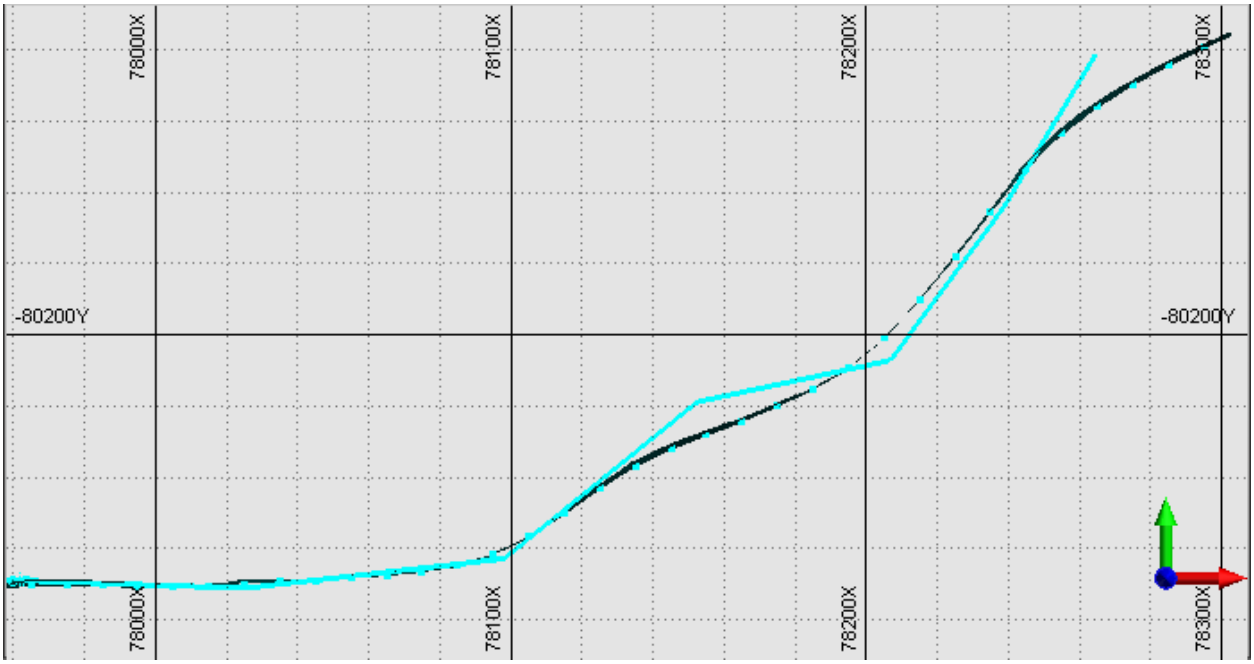


Figure 14.11 Vein Sketch, Vein Centerline, and Block Model Centroids Illustrating Changing Strike

14.3.2 Assigning Mined Out Areas

Each block model was flagged for mined out areas using the 2D vein map long sections. The 2D vein map long sections were converted into active polygons and used to assign a value of "1" to the block model column "mined out." Figure 14.12 shows the West Chance Vein 2D vein map long section and the block model centroids, with filtering out of the "mined out" code.

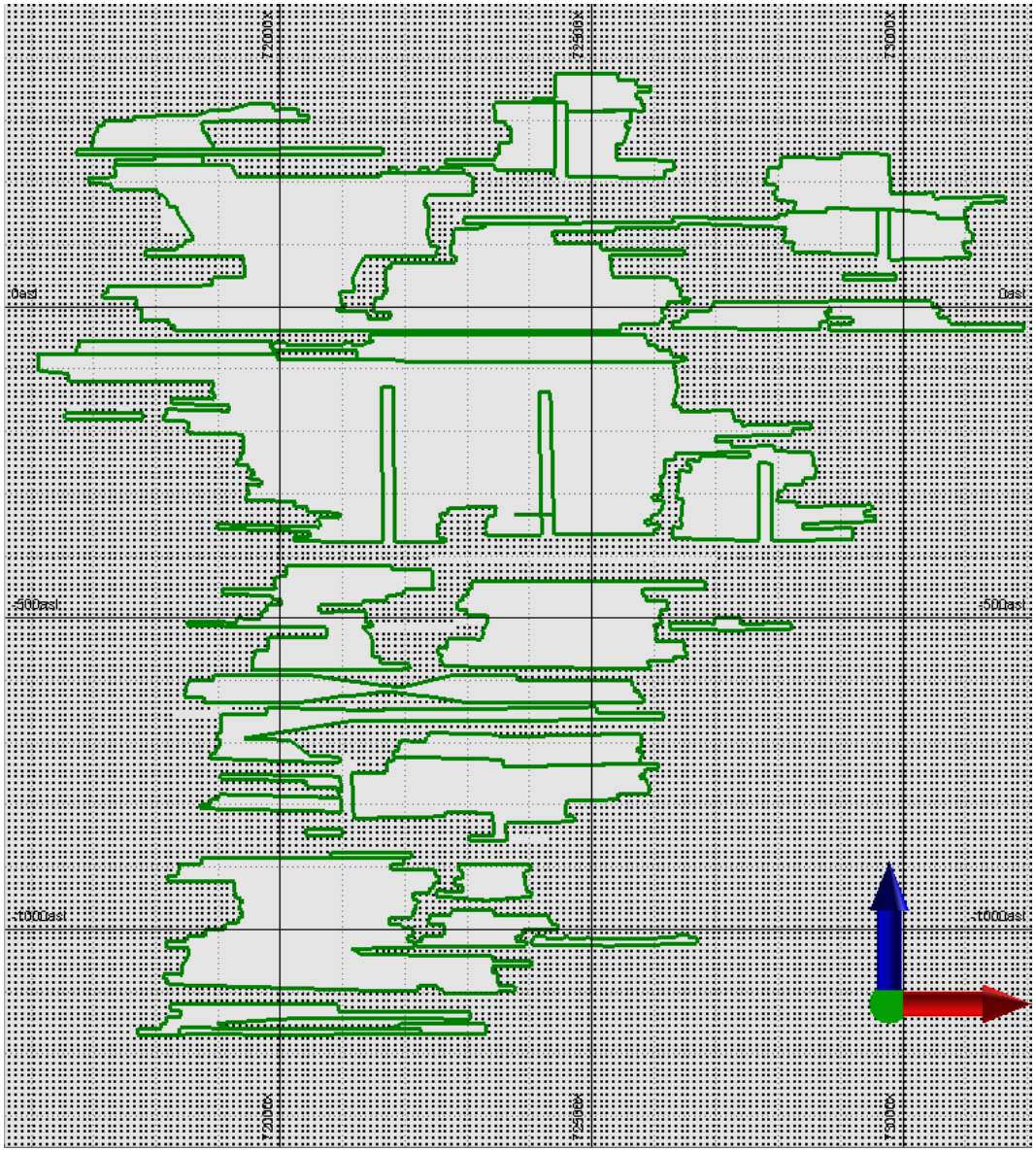


Figure 14.12 Block Model Centroids Filtered By Mined Out Code, West Chance Vein

14.3.3 Combining Assay Data Sources

Assay data used for estimation was derived from three sources: drill holes, level drift channel samples, and stope channel samples.

Based on assay interval vein assignments, drill hole assays were composited across vein intervals and then corrected for true thickness based on the orientation of the vein centerline surfaces. True thickness was assigned to the “width” column. The location (easting, northing, elevation) of the center of the composite interval was also generated.

Level drift channel assay samples were digitized in true easting and northing and true thickness and the elevation was generated from level drift triangulations.

Stope channel assay samples were digitized relative to the 2D vein map long section raise locations and were brought into MicroMine® along with the 2D vein map long sections. Once proper easting and elevation positions were determined, the stope channel assay samples were draped to the vein centerline surfaces to best approximate their true location.

Following draping of stope channel assay samples, all three assay data sources were in true location (easting, northing, elevation), referenced to true thickness, and could be used as one consolidated assay data set.

A combined assay file was generated for each vein surface with the columns: Data Type, Assay ID, East, North, Rl, Width, Ag, Cu, Pb, and Zn.

After combining the data sources, blanks or null values were present in the Ag, Cu, Pb, and Zn columns. Blanks in the Ag column were rare and were assigned 0.05 opt, and Cu blanks were assigned 0.025 %, Pb blanks were assigned 0.05 %, and Zn blanks were assigned 0.05 %. Blank values were determined by a review of the project-wide detection limits for the three data sources. Different detection limits have existed depending on the time period of the assay; the default values chosen represent estimated values. Assay values were top cut on a vein by vein basis.

14.4 Resource Estimation

Estimations of grade and thickness were completed by multi-pass inverse distance weighting and Ordinary Kriging using MicroMine®. Block volumes, tons, and contained metal values and summations were determined using SQL calculation queries in Microsoft Access®.

14.4.1 Search Ellipse Orientation and Variography

Search ellipse orientations were determined using 2D vein map long section maps and fit to the orientations of mined out shapes. Search ellipse anisotropy was also generalized by the mined out shapes. Search ellipse anisotropy ranged from a maximum of 1:0.5 for the initial inferred pass to 1:0.25 to 1:0.4 for select measured and indicated passes. Figure 14.13 shows the West Chance Vein with three nest search ellipse passes, inferred shown in blue, indicated in green, and measured in red, all oriented with the mined out shapes. Figure 14.13 also shows the 2D vein map long section, along with Ag assay points combined from all three assay data sources.

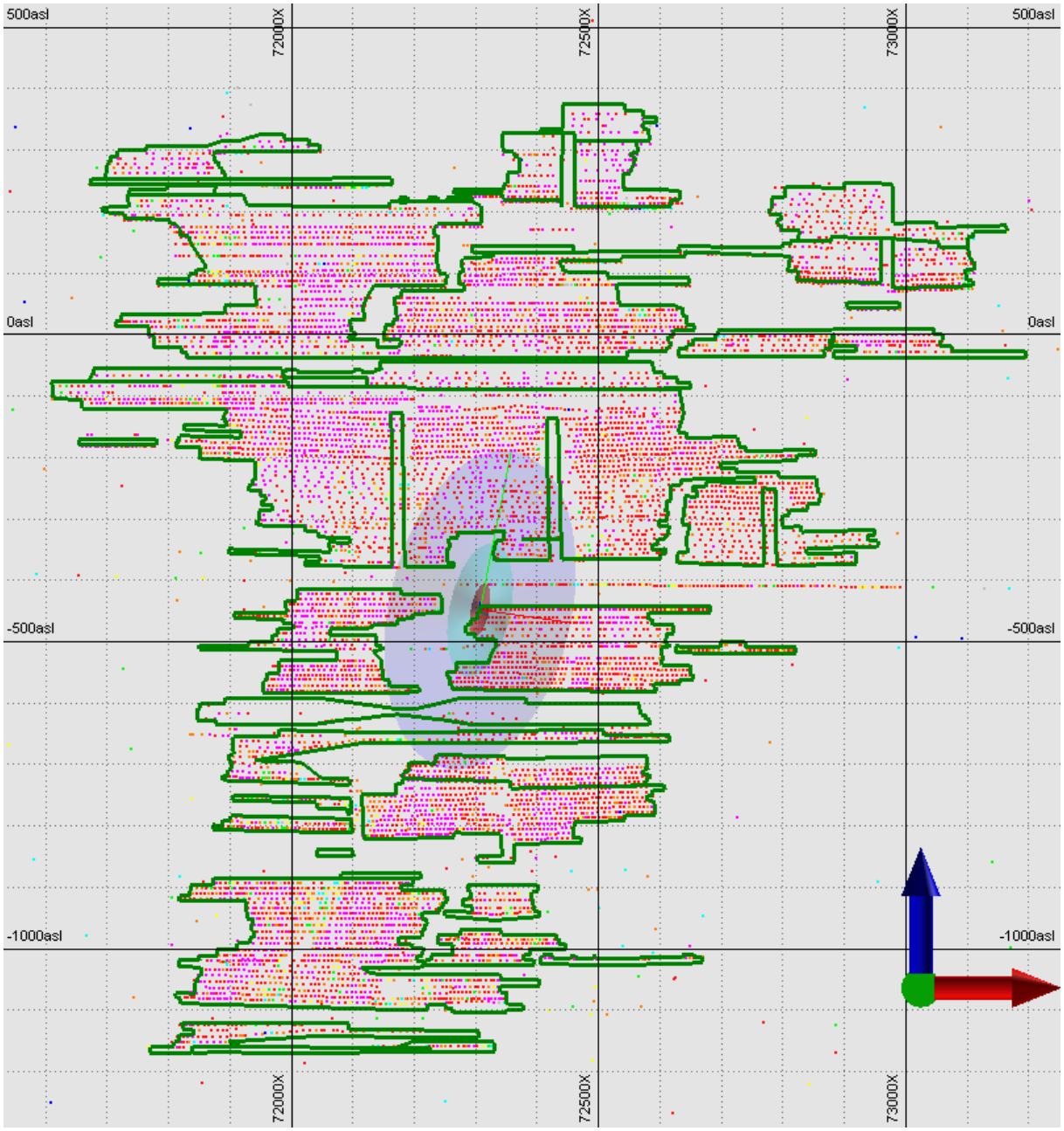


Figure 14.13 Nested Search Ellipses Orientation and Combined Ag Assay Data, West Chance Vein

Omni-directional variography was calculated on a vein by vein basis from combined assay files for Ag. For instance, only assays labeled West Chance Vein were used to calculate the variogram for the West Chance Vein. Search distance parameters determined by Ag variograms were applied to Cu, Pb, Zn, and width.

The results of the omni-direction on a vein by vein basis indicate project wide a nugget between 40 and 60% of the sill and sill ranges at 90 to 150 ft. To correct for a relatively high nugget, a

nested variogram from 0 to 20 ft was assumed to have a nugget of 0. From 20 ft to the sill the nugget determined by the omni-directional variography was used.

Inferred long axis search distance radii were between 250 and 300 ft, generalized as twice the distance of the sill. Indicated long axis search distance radii were between 90 and 150 ft. Measured long axis search distance radii were between 25 and 50 ft. Additional parameters for indicated and measured classification were added and are described in the following paragraphs. Figure 14.14 shows omni-directional variography for the West Chance Vein.

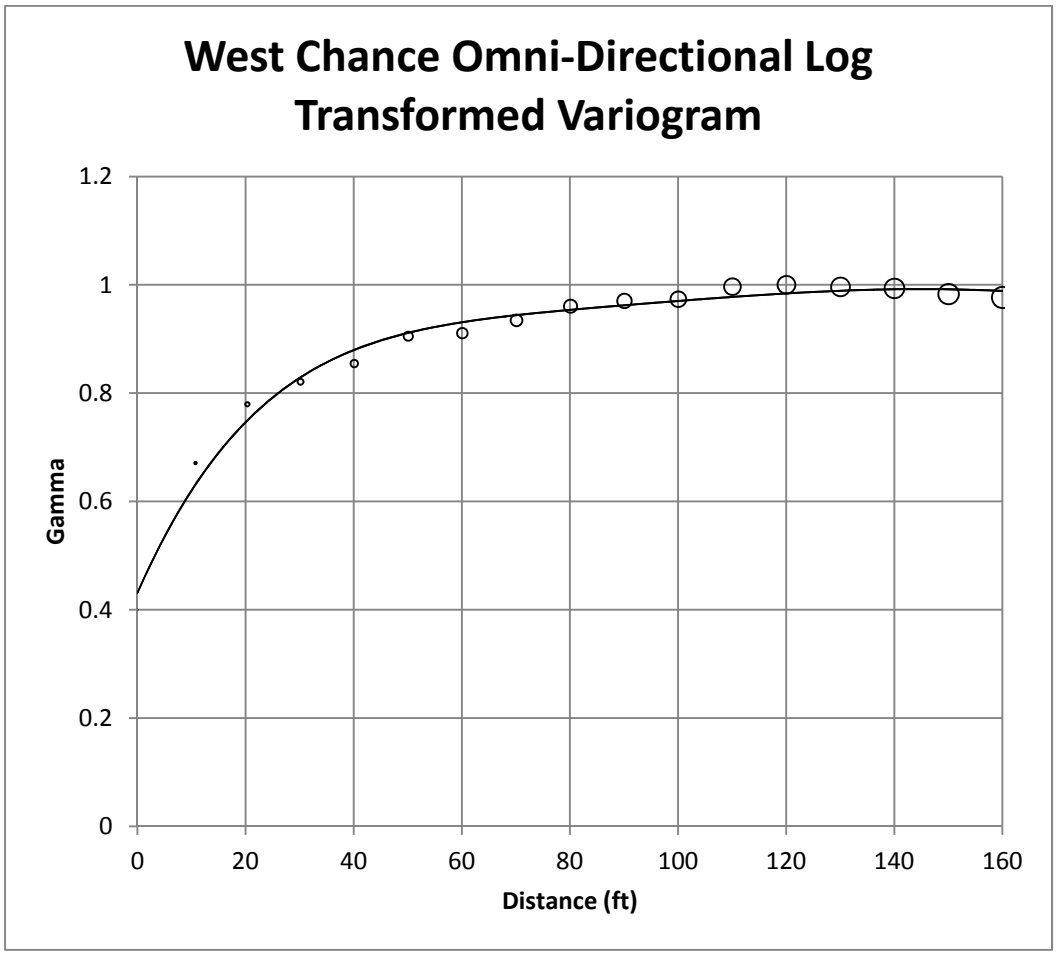


Figure 14.14 West Chance Omni-Directional Log Transformed Variogram

14.4.2 Estimation Passes and Block Classification

Estimation of Ag, Cu, Pb, Zn, and width were completed in three passes for each attribute. The first pass had a maximum range between 250 and 300 ft and employed an inverse distance to the third power weighted estimate using the nearest five data points. All block estimates from the first pass were assigned an inferred classification.

The second pass had a maximum range between 90 and 150 ft and performed a two-sector Kriged estimate with a maximum of 10 data points per sector. All block estimates from the second pass were initially assigned inferred and then reclassified as indicated if a certain number of data points were used for estimation. Any block estimated in the second pass over

wrote the same block estimated in the first pass. Minimum data points required for reclassification to indicated was subject to data type available on a vein by vein basis. Veins with a large density of stope samples required more samples to achieve indicated and veins with only level data points required fewer data points. Classification by sample count was difficult to establish vein by vein due to the varying combination of data sources causing different sample location geometries. A single standard for indicated classification could not be determined. Minimum sample counts were calibrated to previous ranges for measured and indicated classification made by SSMC resource geologists in previous resource calculations.

The third pass had a maximum range between 25 and 50 ft and performed a one sector Kriged estimate with a maximum of 10 data points and a minimum of three data points. Minimum data points required for measured was adjusted in the same way as for the indicated classification. If the third pass did not meet the minimum requirement the second pass block value was used.

In addition to the classification scheme described above, each vein was further reclassified using bounding strings as a final pass to limit the extension of measured and indicated along strike from known mined out areas. Bounding strings were drawn close to mined out stopes along strike and limited measured and indicated from passes two and three to only a few blocks along strike outside known stope boundaries. Up and down dip the bounding strings were drawn further away from mined out stopes and were less limiting to the measured and indicated blocks from passes two and three.

In select cases bounding strings were also used to reclassify blocks failing to meet the requirements of indicated classification in pass two, but were assigned as indicated by SSMC resource geologists in previous resource calculations. In many instances the author has relied on SSMC resource geologists to establish areas for measured and indicated, in light of their extensive knowledge of the deposit.

At this time Cu, Pb, and Zn report only to the inferred class. Historically, assaying for Cu, Pb, and Zn were limited to locations where visual inspection indicated the potential for high grade values. For this reason, there is inconsistent data coverage and an incomplete population currently exists compared to Ag assays. The author suggests that future resource estimates explore the possibilities of a vein by vein regression for Cu, Pb, and Zn.

Figure 14.15 shows block classifications for the West Chance Vein. Measured blocks centroids are in red, indicated are in green, and inferred are in blue. Black dots represent assay data points. The blue line represents the inferred reclassification bounding string; the lime green line represents the indicated reclassification bounding string.

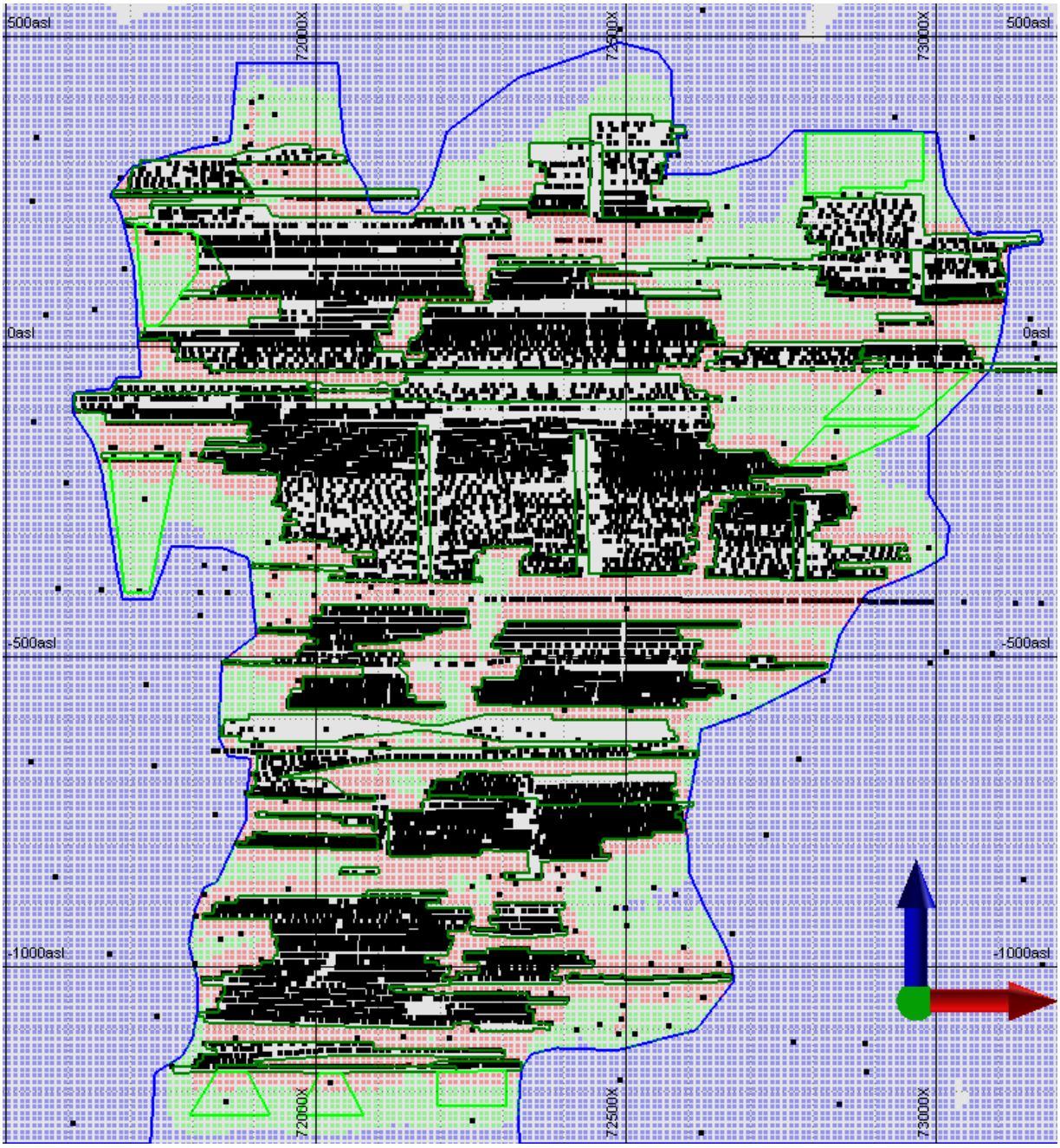


Figure 14.15 West Chance Block Resource Classification and Reclassification Bounding Strings

14.4.3 Block Calculations

Block volumes, tons, and contained metal values and summations were determined using SQL calculation queries in Microsoft Access®. Block volumes were calculated in cubic feet (cu ft) by multiplying east dimension by elevation dimension by width. Tons were calculated by dividing volume by a density of 10 cu ft/ton. A density of 10 cu ft/ton was used for both vein material and

diluted material and represents an average density throughout the history of the mine of both vein material and diluted material. A density of 9.5 cu ft/ton was used for the West Chance veins due to the abundance of Pb. Contained Ag ounces per block were determined using tons multiplied by Ag opt.

Diluted tons were calculated assuming a minimum mine thickness of 6.5 ft. If vein width was less than 4.5 ft wide a diluted thickness of 6.5 ft was assigned. If vein width was greater than or equal to 4.5 ft, 2 ft of dilution was also added. Dilution material was assumed at 0 Ag opt. Diluted grade was then calculated from diluted tons and contained Ag ounces. Resource tabulations were completed using diluted grade Ag opt, diluted tons, and contained metal.

14.4.4 Mineral Resource Estimates by Vein

Estimated mineral resources were tabulated on a vein by vein basis. Table 14.2 is a tabulation of estimated measured resources, Table 14.3 is a tabulation of estimated indicated resources, and Table 14.4 is a tabulation of estimated inferred resources.

Table 14.2 Estimated Measured Mineral Resources by Vein

Vein	Cutoff Ag Opt Diluted	Resource Class	Tons Diluted	Grade Ag Opt Diluted	Ag Contained Ounces
06Vein	10	1-Measured	12,000	45.9	550,000
08BVein	10	1-Measured	6,000	28.4	170,000
08Vein-DHWVein	10	1-Measured	91,000	19.3	1,760,000
09HWVein	10	1-Measured	35,000	19.6	680,000
09Vein	10	1-Measured	17,000	19.7	330,000
625MVein	10	1-Measured	17,000	25.7	430,000
BVein	10	1-Measured	5,000	19.0	100,000
CFault	10	1-Measured	22,000	22.2	490,000
Chester	10	1-Measured	284,000	27.0	7,660,000
ChesterHang	10	1-Measured	31,000	24.4	760,000
CopperVein	10	1-Measured	36,000	21.9	800,000
DVein	10	1-Measured	22,000	26.7	600,000
FVein	10	1-Measured	12,000	24.9	300,000
HVein	10	1-Measured	10,000	28.9	300,000
KFWVein	10	1-Measured	1,000	21.3	10,000
KVein	10	1-Measured	5,000	23.7	120,000
NYankeeBoySunshine	10	1-Measured	76,000	23.9	1,820,000
S78Vein	10	1-Measured	15,000	20.7	310,000
SilverLine	10	1-Measured	3,000	13.4	40,000
SilverSummitNo4	10	1-Measured	53,000	21.0	1,120,000
SilverSyndicateLink	10	1-Measured	79,000	30.8	2,440,000
Sunshine2	10	1-Measured	3,000	17.8	50,000
SunshineFW	10	1-Measured	5,000	23.0	110,000
SYankeeBoy	10	1-Measured	136,000	23.2	3,160,000
W16Vein	10	1-Measured	1,000	18.4	10,000
WestChance	10	1-Measured	185,000	25.0	4,610,000
WestChanceFW	10	1-Measured	9,000	31.9	290,000
YankeeGirl	10	1-Measured	44,000	19.8	860,000

Table 14.3 Estimated Indicated Mineral Resources by Vein

Vein	Cutoff Ag Opt Diluted	Resource Class	Tons Diluted	Grade Ag Opt Diluted	Ag Ounces
06Vein	10	2-Indicated	30,000	38.9	1,170,000
08BVein	10	2-Indicated	10,000	24.2	240,000
08Vein-DHWVein	10	2-Indicated	67,000	18.5	1,230,000
09HWVein	10	2-Indicated	31,000	18.0	550,000
09Vein	10	2-Indicated	60,000	21.2	1,280,000
101Vein	10	2-Indicated	9,000	14.9	140,000
625MVein	10	2-Indicated	74,000	22.3	1,640,000
BVein	10	2-Indicated	7,000	20.5	140,000
CFault	10	2-Indicated	58,000	21.5	1,250,000
Chester	10	2-Indicated	323,000	22.2	7,170,000
ChesterHang	10	2-Indicated	53,000	21.3	1,130,000
CopperVein	10	2-Indicated	64,000	20.1	1,300,000
DVein	10	2-Indicated	48,000	27.0	1,300,000
FVein	10	2-Indicated	11,000	19.4	220,000
HFWVein	10	2-Indicated	3,000	25.2	90,000
HVein	10	2-Indicated	14,000	21.4	290,000
KFWVein	10	2-Indicated	5,000	24.0	110,000
KVein	10	2-Indicated	33,000	19.0	620,000
NYankeeBoySunshine	10	2-Indicated	114,000	20.9	2,380,000
S78Vein	10	2-Indicated	6,000	18.0	110,000
SilverLine	10	2-Indicated	15,000	18.5	270,000
SilverSummitNo4	10	2-Indicated	77,000	19.1	1,480,000
SilverSyndicateLink	10	2-Indicated	200,000	24.6	4,920,000
Sunshine2	10	2-Indicated	6,000	15.0	100,000
SunshineFW	10	2-Indicated	8,000	18.2	140,000
SYankeeBoy	10	2-Indicated	280,000	20.9	5,850,000
W16Vein	10	2-Indicated	2,000	70.3	110,000
WestChance	10	2-Indicated	163,000	19.7	3,200,000
WestChanceFW	10	2-Indicated	11,000	35.7	400,000
YankeeGirl	10	2-Indicated	179,000	20.1	3,600,000

Table 14.4 Estimated Inferred Mineral Resources by Vein

Vein	Cutoff Ag Opt Diluted	Resource Class	Tons Diluted	Grade Ag Opt Diluted	Ag Ounces	Cu %	Pb %	Zinc %
06Vein	10	3-Inferred	96,000	32.9	3,170,000	0.07	0.03	0.02
08BVein	10	3-Inferred	113,000	22.7	2,560,000	0.23	0.02	0.02
08Vein-DHWVein	10	3-Inferred	145,000	26.4	3,840,000	0.08	0.07	0.04
09HWVein	10	3-Inferred	128,000	20.5	2,620,000	0.19	0.04	0.02
09Vein	10	3-Inferred	107,000	24.9	2,670,000	0.06	0.02	0.03
101Vein	10	3-Inferred	126,000	31.1	3,930,000	0.1	0.02	0.02
625MVein	10	3-Inferred	281,000	20.5	5,770,000	0.32	0.02	0.02
BVein	10	3-Inferred	42,000	25.3	1,070,000	0.02	0.05	0.05
CFault	10	3-Inferred	163,000	24.2	3,950,000	0.26	1.38	0.05
Chester	10	3-Inferred	619,000	22.9	14,170,000	0.3	0.31	0
ChesterHang	10	3-Inferred	268,000	23.0	6,160,000	0.11	1.45	0
ChesterHWSplit	10	3-Inferred	439,000	30.9	13,550,000	0.22	0.05	0
CopperVein	10	3-Inferred	324,000	29.2	9,460,000	0.08	0.25	0.03
DVein	10	3-Inferred	362,000	27.1	9,820,000	0.03	0.04	0.04
FVein	10	3-Inferred	124,000	20.9	2,580,000	0.09	0.11	0
HFWVein	10	3-Inferred	72,000	16.5	1,190,000	0.04	0.01	0.02
HVein	10	3-Inferred	109,000	36.7	3,980,000	0.21	0.06	0.03
KFWVein	10	3-Inferred	120,000	19.8	2,370,000	0.39	0.01	0.01
KVein	10	3-Inferred	64,000	21.9	1,400,000	0.36	0.06	0
NYankeeBoySunshine	10	3-Inferred	493,000	24.3	12,000,000	0.05	0.04	0.04
S78Vein	10	3-Inferred	53,000	16.7	890,000	0.09	0.02	0.02
SilverLine	10	3-Inferred	78,000	16.4	1,290,000	0.19	2.29	0.02
SilverSummitNo3	10	3-Inferred	314,000	26.8	8,410,000	0.64	0.03	0.03
SilverSummitNo4	10	3-Inferred	1,313,000	25.6	33,650,000	0.85	0.03	0.03
SilverSyndicateLink	10	3-Inferred	715,000	25.0	17,830,000	0.13	1.06	0.05
Sunshine2	10	3-Inferred	40,000	18.2	730,000	0.05	0.01	0.01
SunshineFW	10	3-Inferred	72,000	18.9	1,370,000	0.07	0.01	0.01
SYankeeBoy	10	3-Inferred	682,000	20.8	14,210,000	0.02	0.04	0.04
W16Vein	10	3-Inferred	29,000	57.8	1,690,000	0.15	0.02	0.02
WestChance	10	3-Inferred	432,000	24.8	10,730,000	0.25	2.61	0
WestChanceFW	10	3-Inferred	107,000	19.0	2,020,000	0.01	0.01	0
WestChanceFWWest	10	3-Inferred	7,000	12.1	80,000	0.07	0	0
YankeeGirl	10	3-Inferred	828,000	20.6	17,020,000	0.07	0.02	0.02
YankeeGirl952Split	10	3-Inferred	61,000	12.8	770,000	0	0.01	0.01
YankeeGirlFW	10	3-Inferred	119,000	32.1	3,820,000	0.01	0.03	0.03
YankeeGirlHW	10	3-Inferred	70,000	18.0	1,260,000	0.01	0.02	0.02

14.4.5 Mineral Resource Estimate Grade Tonnage Relationships

Figure 14.16 shows the grade tonnage relationship of measured and indicated resources within all veins at a range of cutoff grades. Figure 14.7 shows the grade tonnage relationship of inferred resources within all veins at a range of cutoff grades.

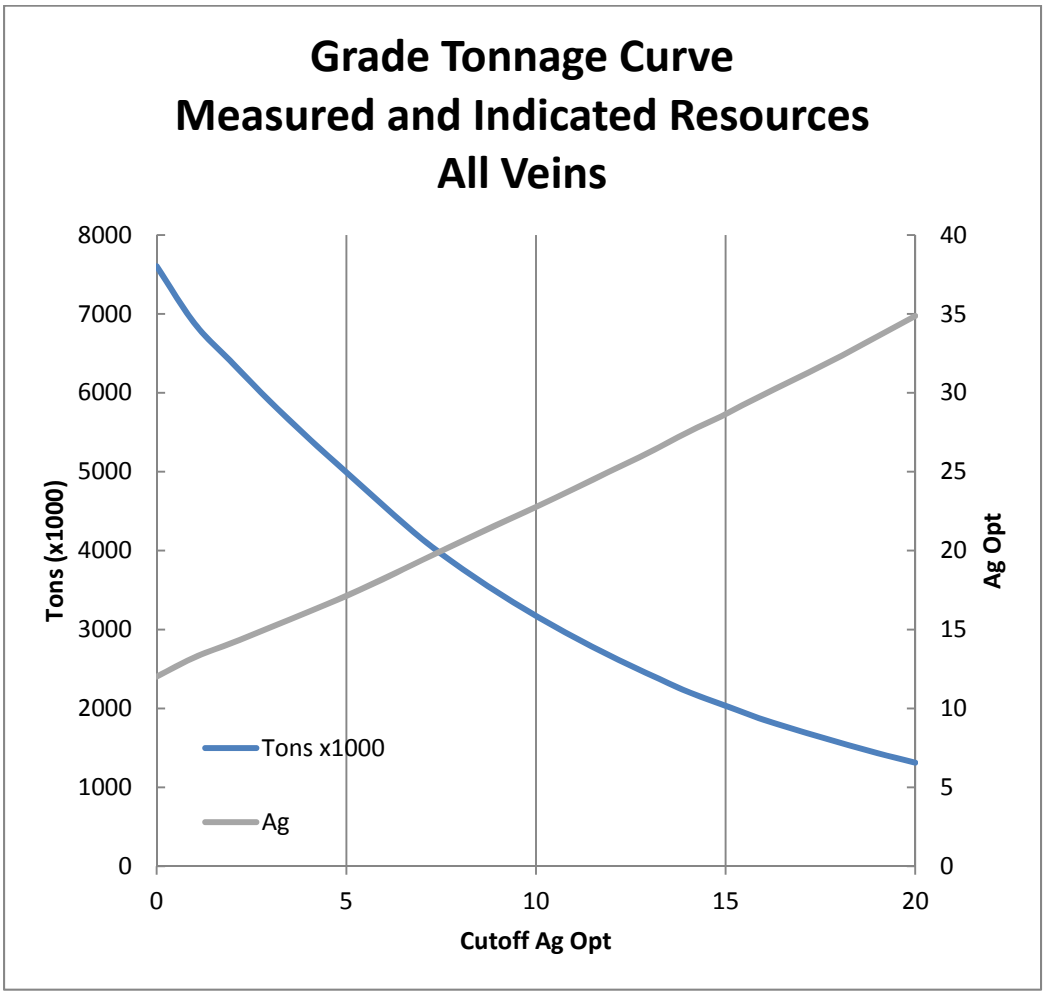


Figure 14.16 Grade Tonnage Curve Measured and Indicated Resources All Veins

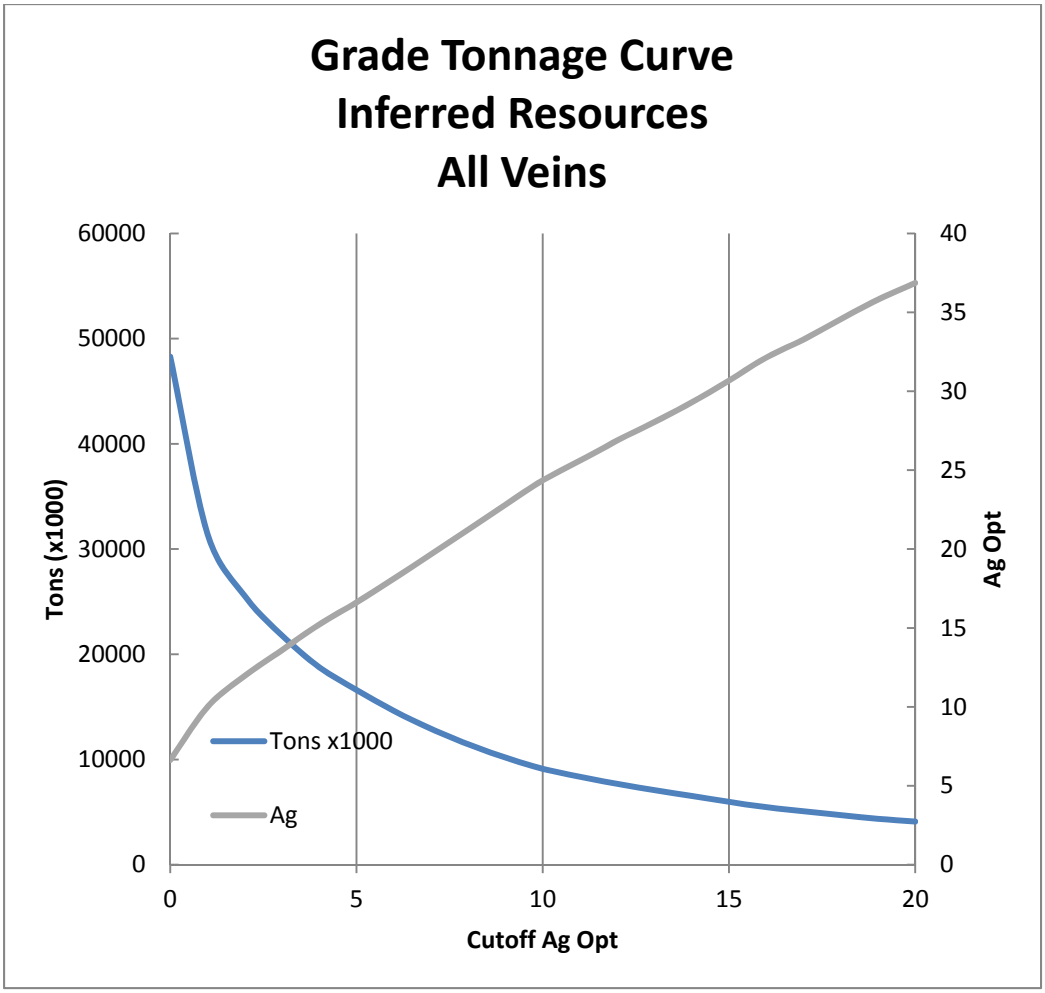


Figure 14.17 Grade Tonnage Curve Inferred Resources All Veins

14.4.6 Validation and Visual Review

The author was provided with yearly production records for the Sunshine Mine from 1884 to 2001 and was able to compare the sum of the blocks flagged as mined out in this resource estimate to the historic total production.

The records are based on annual mandatory reporting and contain yearly totals of tons and contained silver ounces. The annual totals are not broken out on a vein by vein basis. There are few data gaps, specifically, contained ounces from 1927 to 1928 are missing, but recovered ounces have been provided. These records are believed to be a reliable accounting for total mined material.

A direct comparison between this mineral resource estimate and the historic production is limited for two reasons. The first being production from the Rambo area veins was not considered as part of this mineral resource estimate, and second the true diluted width of each mined out block is not known. A diluted width of 6.5 ft has been used as an approximation. Table 14.5 shows the production record totals compared to the mined out totals estimated by

this mineral resource estimate. Results of historic production are not necessarily indicative of the remaining mineral resources. Comparisons of historic production records to estimations of mined material by this model contribute to the reliability of the estimation of remaining resources but do not necessarily indicate that mineral resource estimations of the remaining material will be successful.

Table 14.5 Historic Production Comparison

	Total Mined Tons	Grade Ag opt	Contained Silver Ounces
Production Records 1884 to 2001	13,177,664	28.0	369,423,312
	Total Mined Tons	**Grade Ag opt**	**Contained Silver Ounces**
Production Estimated By Block Model	12,314,573	28.6	351,656,990

Each estimate was visually reviewed on a vein by vein basis. Assay grades were compared to estimated block grades. In addition estimated grade x thickness values were compared within and outside mined out boundaries. Figure 14.18 shows Ag assays compared to estimated block grade for all resource classes. Figure 14.19 shows block grade x thickness flagged in the block model as mined out and Figure 14.20 shows block grade x thickness of remaining resources.

Grade x thickness was used as a proxy for contained ounces and therefore used to identify areas most likely to have been mined. Visual comparison shows that on average areas of higher grade x thickness were mined out.

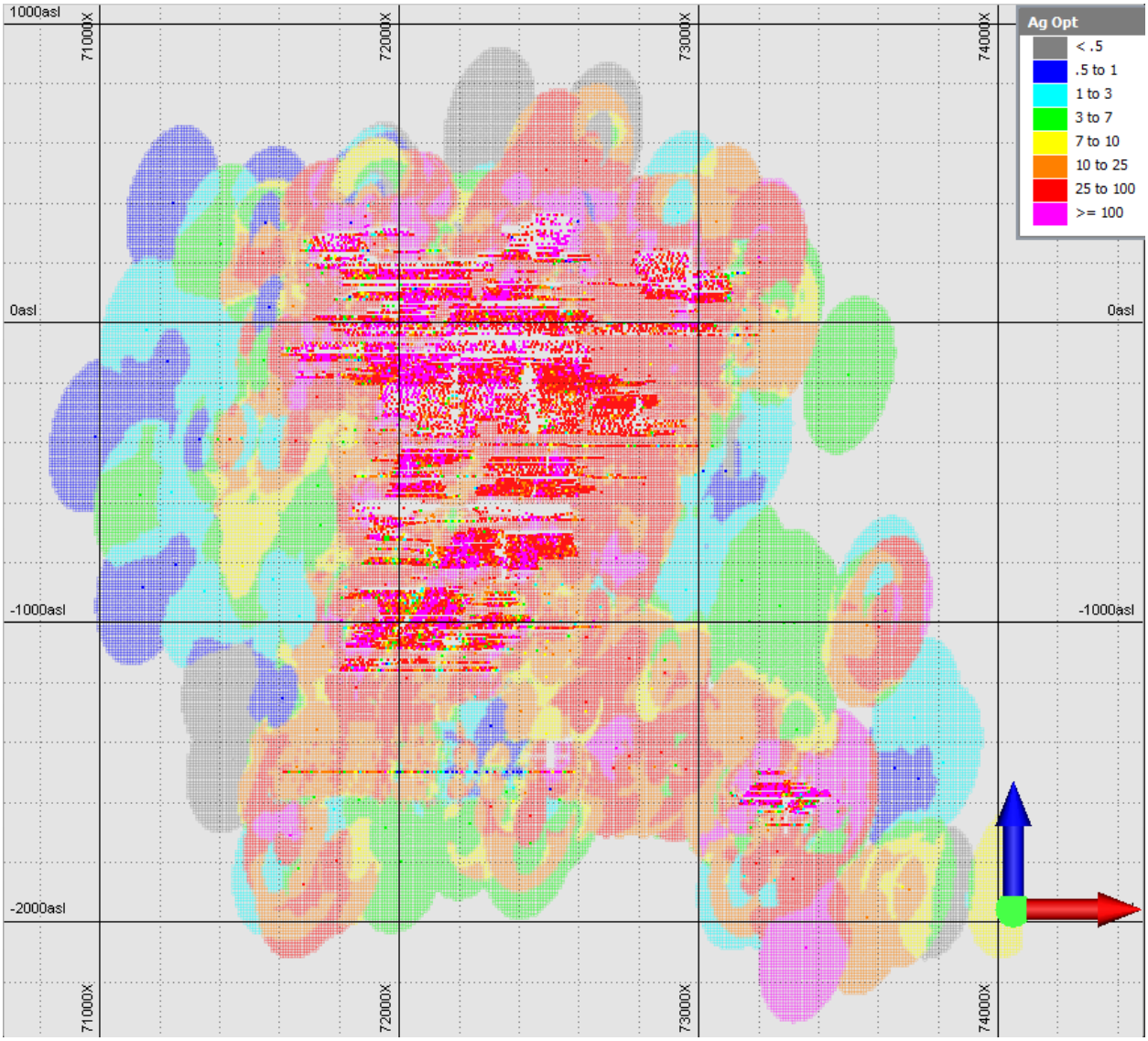


Figure 14.18 Visual Comparison of Assays Compared To Block Ag Grades, West Chance Vein

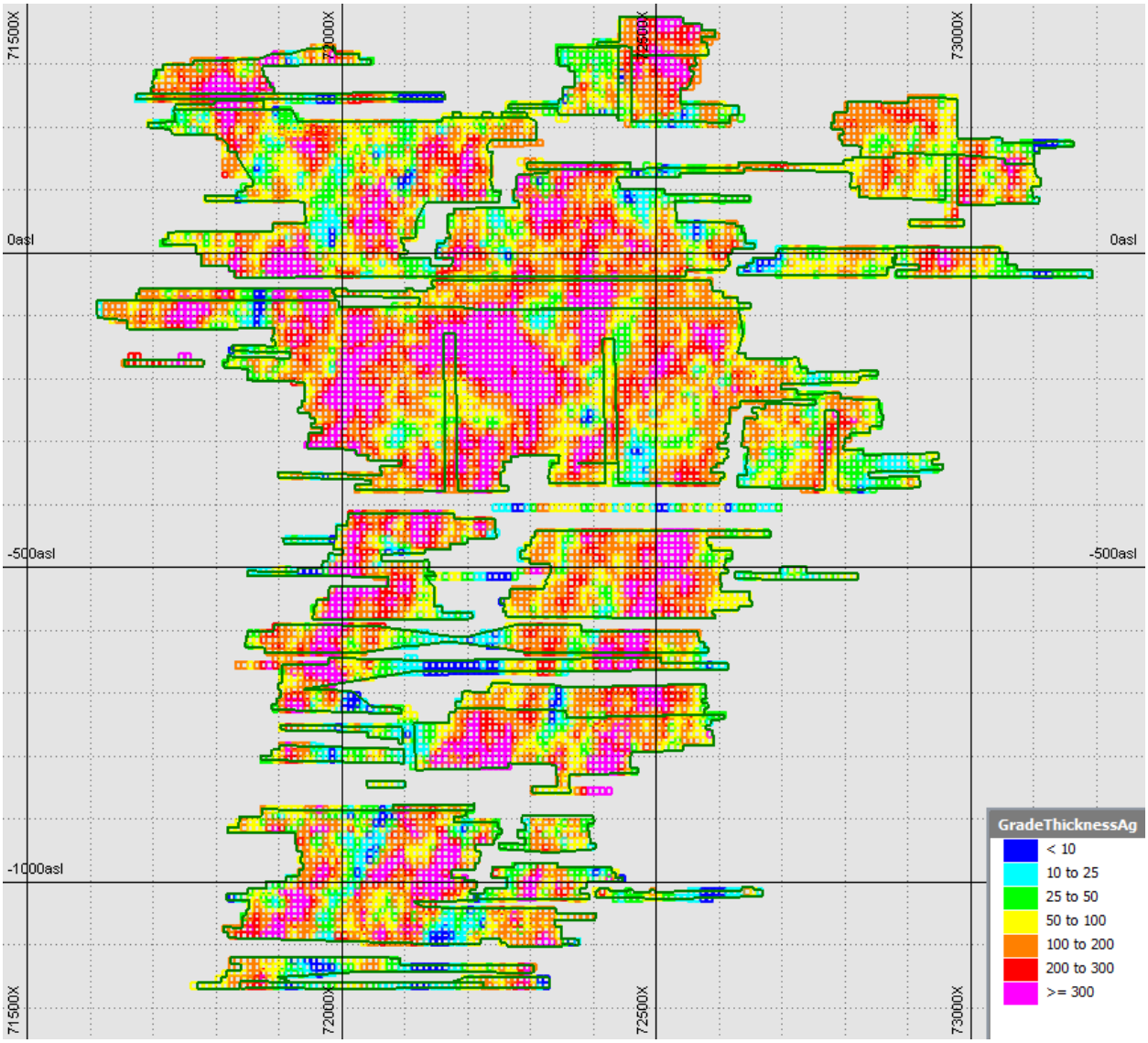


Figure 14.19 Visual Review of Grade x Thickness Ag Ft and Mined Out Areas, West Chance Vein

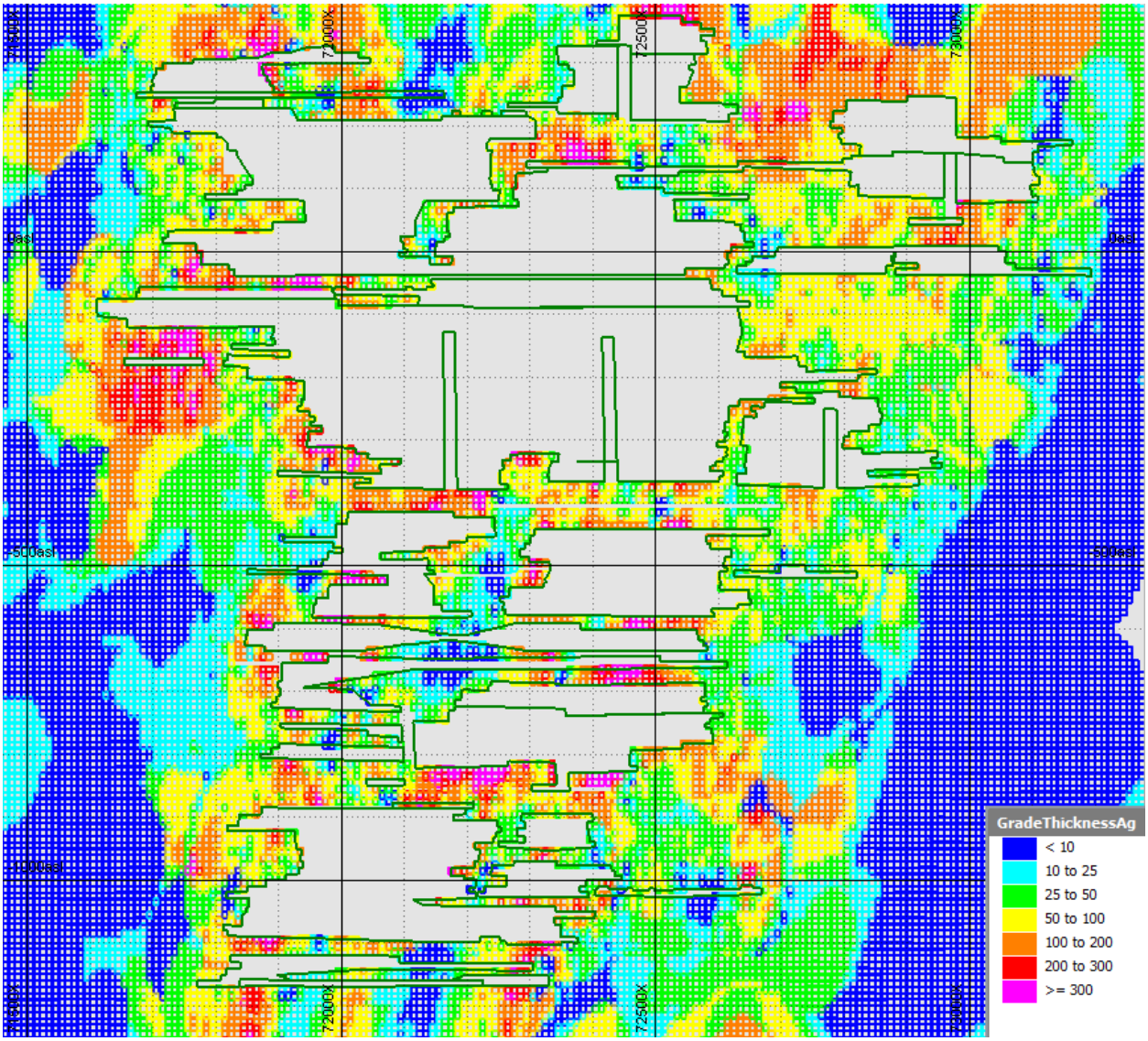


Figure 14.20 Visual Review of Grade x Thickness Ag Ft and Remaining Resources, West Chance Vein

Estimating grade and thickness independently was compared to estimating grade x thickness as a single entity for the H vein. The investigation confirmed that estimating grade and thickness independently provided satisfactory results.

Estimations were completed using both methods and the resulting grade x thickness histograms and cumulative frequencies were compared. Figure 14.21 shows grade x thickness calculated from grade and thickness estimated independently in red, and grade x thickness estimated as a single entity in blue. The resulting histograms and cumulative frequencies show no significant bias and validate the practice of estimating grade and thickness independently.

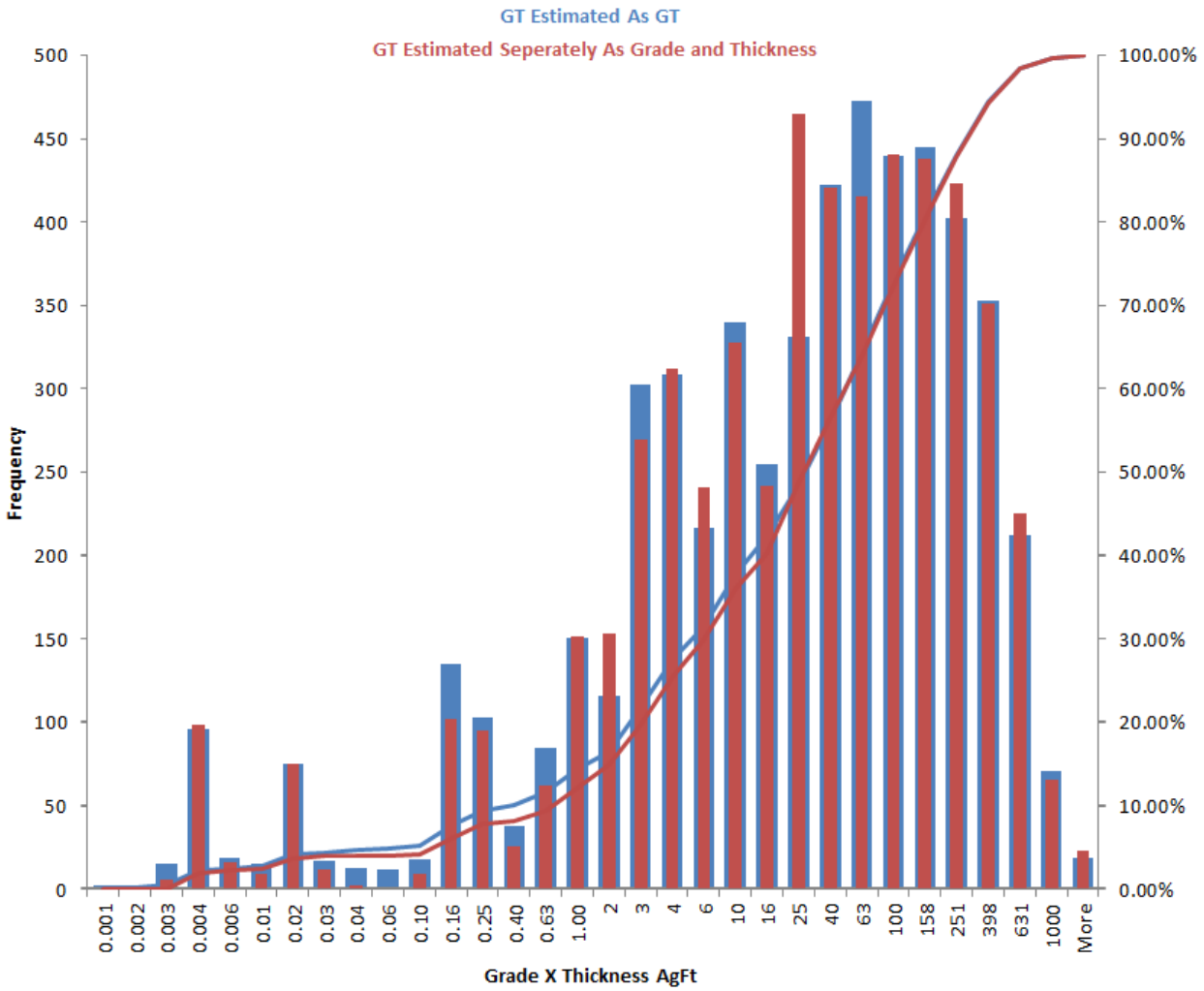


Figure 14.21 Comparison of Populations of Grade x Thickness Estimated Independently and as a Single Entity, H Vein

14.4.7 Relevant Factors

As described in Item 4.0 of this report the property is subject to NSR agreements on various claims. The author is unaware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect this mineral resource estimate.

15.0 MINERAL RESERVE ESTIMATES

The Sunshine Mine property does not currently contain mineral reserves so this Section does not apply.

16.0 MINING METHODS

16.1 Historical Mining Methods

16.1.1 Mine Material Movement Infrastructure

Rehabilitation work on the existing infrastructure started in 2012, but a fire in the mine on the 3100 Level delayed the rehabilitation work. The fire has been extinguished for about six months and rehabilitation work is scheduled to re-commence early in 2013.

Primary access to the lower levels of the Sunshine mine is through the four-compartment Jewell Shaft. The shaft extends vertically downward 4,000 feet with primary haulage ways connecting on the 3100 and 3700 Levels. The Jewell Shaft has a nominal hoisting capacity of 1,300 tpd of ore and waste. An existing ramp system will be expanded to allow truck haulage from most stopes to the haulage level used to load the Jewell Shaft skip pockets on the 2300, 2700, 3100, or 3700 Levels. A new ore pass and loading pocket system will be installed to allow efficient skip loading. Prior to using the Jewell Shaft for ore production, about a year of rehabilitation work is required.

The Silver Summit Tunnel and vertical shaft (winze) provide a secondary escapeway and could be used to augment the haulage of ore and waste from the mine. The Silver Summit Shaft needs to be rehabilitated as well. This rehabilitation work is expected to re-start in the second quarter of 2013 with completion about a year after the re-start. Additional work would be required to include hoisting capability to the shaft.

The #10 Shaft that crosses the Chester Vein will be abandoned because of poor ground conditions and many planned stopes are located close to this shaft. Access to areas of the mine served by the #10 Shaft will be replaced by load, haul dump (LHD) ramps descending from the east end of the 3700 Level (CSR) existing ramp system. An additional ramp system (existing with new portions) will be used to mine the West Chester deposit. No rehabilitation is planned for the #12 Shaft for this study. All material between the bottom of the Jewell Shaft and the 4600 Level will be transported by a new ramp system.

Main haulage levels are spaced on 150 to 300 foot intervals. The 3100 Level will require a new drift around the area of the 2012 fire. A loading pocket on the 1900 Level for the Jewell Shaft will be installed prior to production. The cost of basic rehabilitation for other levels and keeping the mine dewatered is included in the capital cost estimate. Most levels will use rail haulage to move material delivered to the level and needing movement to the Jewell Shaft. Ramp and raise systems will provide for truck movement of material from the stopes to the levels.

While the shaft rehabilitation work is ongoing, a new ramp system will be installed from the Silver Summit Tunnel to about the 2300 Level. This ramp system will facilitate development of a new ventilation system for the mine and mining and exploration for the upper areas of the mine. Work on this new ramp is expected to start during the second quarter of 2013. Exploration during the past year has developed new areas of mineralization in the upper areas of the mine. Sunshine calls this the Upper Country program.

16.1.2 Historic Mining Methods

Historically, mining was initially completed by timbered overhand open stopes and fill mining. These methods gave way in the 1930s to overhand cut and fill mining with raise access to the stopes. In the 1960s hydraulic sand fill was added and used as a floor for stope mining to proceed. It was not until the 1990s that two ramp systems were developed to provide mechanical access to some of the stopes. The mine has been developed to the 5800 Level. Since the introduction of mechanical access the mining methods have been divided into conventional slusher stopes that are less than six feet wide and mechanical stopes which are greater than six feet wide.

16.1.3 Conventional (Slusher) Stopes

In the “conventional” or slusher mining method, a stope of ore was developed by extending an underground opening (drift) to the bottom of the block. The drifts were on levels in the mine and were typically spaced 200 vertical feet apart. Once accessed, the ore block was mined in a series of slices, or cuts, starting at the bottom. The cuts were about nine feet high, and after each cut was mined, the resultant void was filled with hydraulically-placed classified mill tailings (sandfill). Ground control was facilitated by the fact that there was never more than one cut open in the stope at any given time.

An opening was maintained from the level below and up through the filled cuts to the current cut. This opening, called a “raise”, was boxed off from the sandfill on both sides and provided access to the work area. The raise has three compartments — two for broken ore, and a ladder way for man access.

The stope was advanced by drilling the end of the stope (face) and blasting the rock. The broken rock, a mixture of ore and waste, was pulled to the raise using a slusher. The rock was loaded into rail cars at the bottom of the raise, and hauled to the main production shafts.

Typically, slusher stopes were less than six feet wide, or too narrow for mechanical equipment, but at least 4.5 feet wide. Figure 16.1 shows a typical historic overhand cut and fill stope.

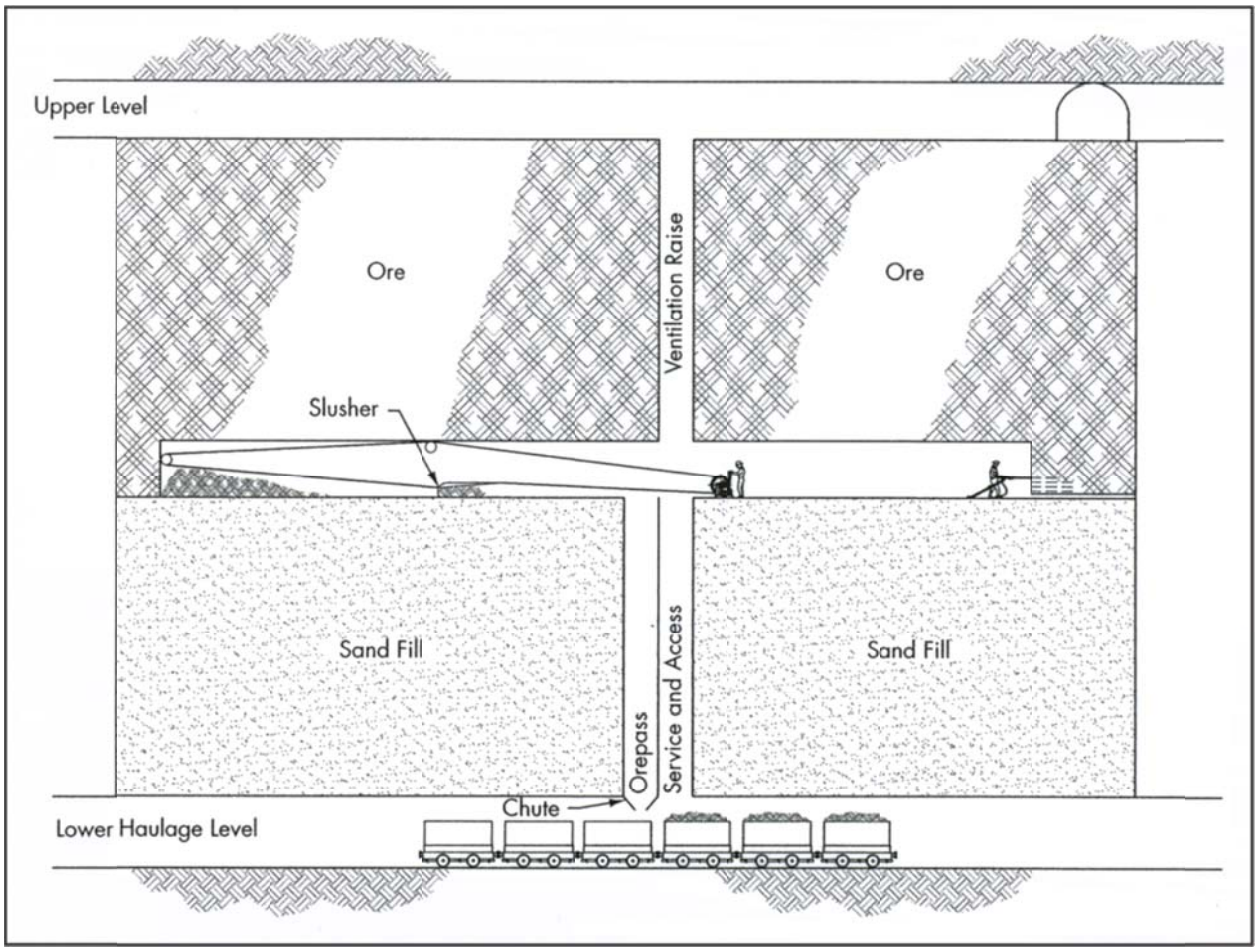


Figure 16.1 Typical Historic Overhand Cut and Fill Stope – Longsection View

16.1.4 Mechanical Stopes

Beginning in the 1990s mechanical methods were used to mine stopes over six feet wide. These wider stopes allowed access inside the stope for diesel equipment. Access to each stope was provided by an attack ramp that was driven to the lowest level of the stope. As the stope was advanced in an overhand or upward method, the attack ramp was raised one cut by breasting down material dozed and used as a base for the next cut of the stope. Figures 16.2 and 16.3 show typical historic cut and fill mining method utilizing diesel equipment. Figure 16.4 shows a drawing of the existing mine workings in longsection view. Figure 16.5 shows a typical Alimak raise setup for a stope.

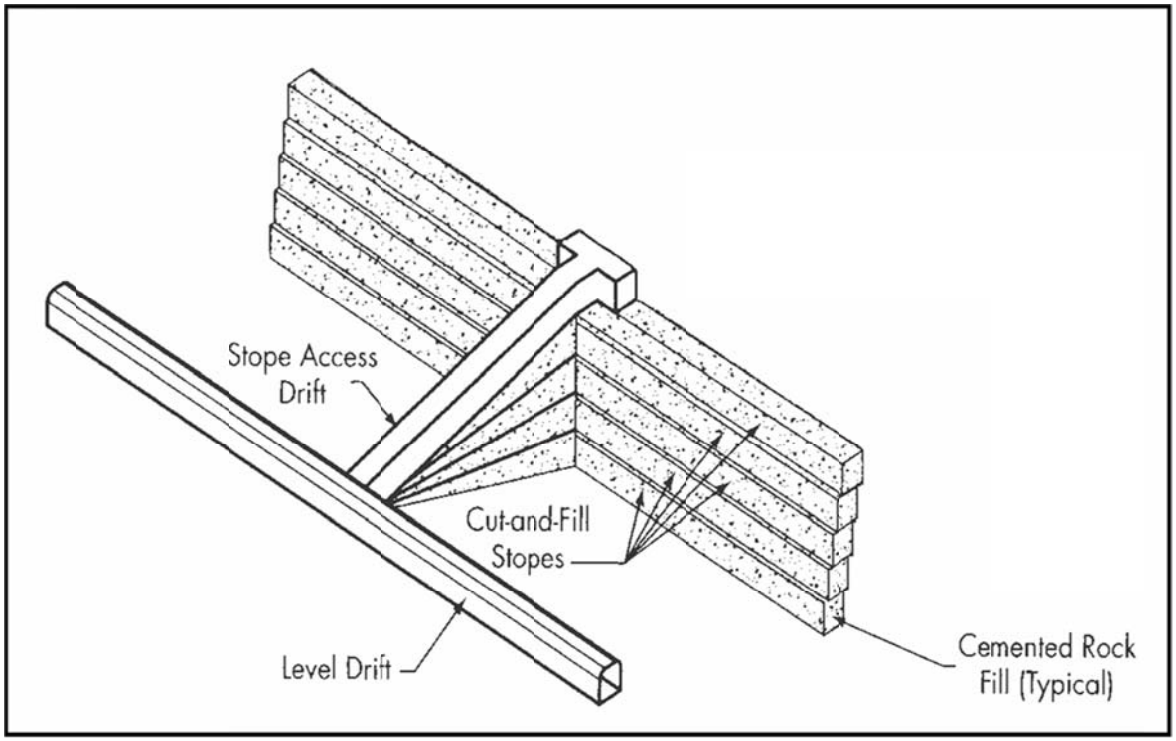


Figure 16.2 Typical Historic Overhand Mechanical Cut and Fill Stope – Isometric View

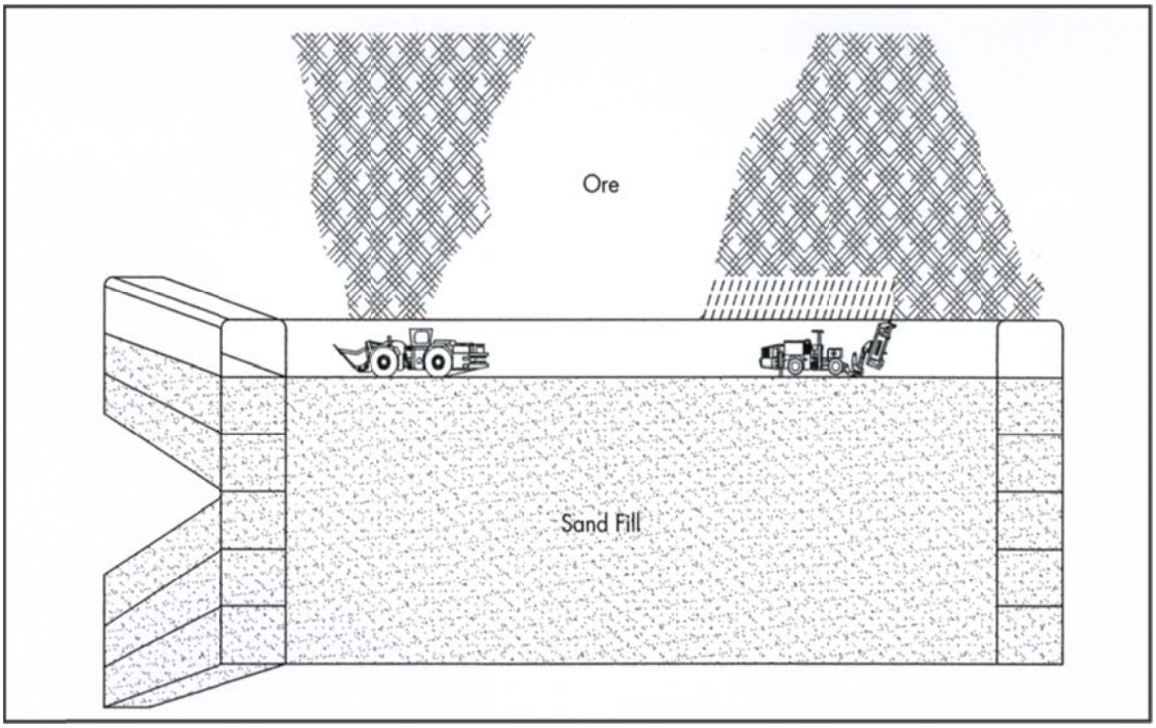


Figure 16.3 Typical Historic Mechanical Cut and Fill Mining – Longsection View

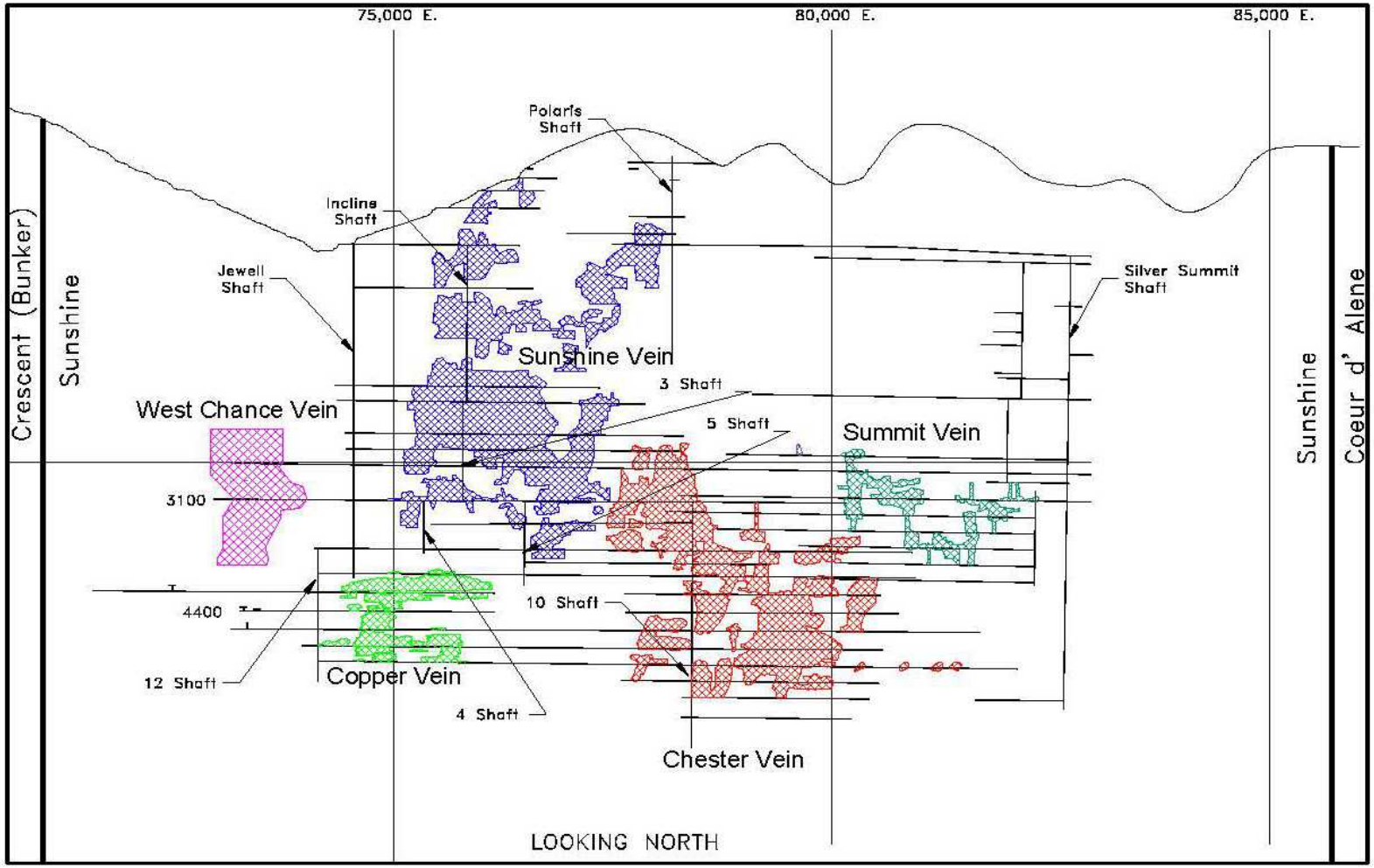


Figure 16.4 Existing Sunshine Mine Workings – Longsection View

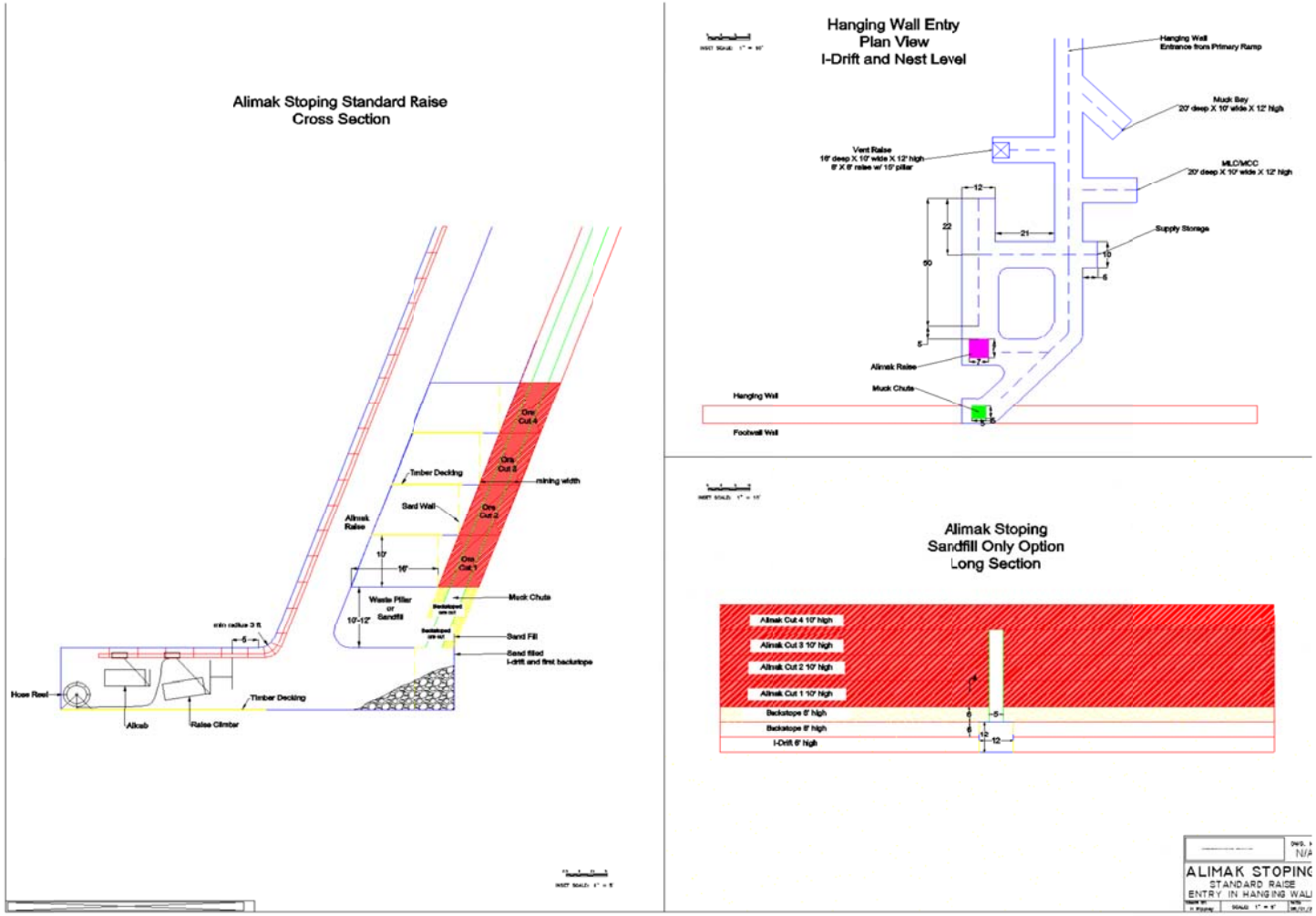


Figure 16.5 Typical Alimak Raise Setup for a Stope

16.2 Current Mine Condition and 2012 Mine Fire

The Sunshine Mine was in the process of level rehabilitation and was about to start dewatering (the mine has been flooded below the 3400 Level for a number of years) when a mine fire started just below the 3100 Level in February 2012. The fire caused some damage to the 3100 Level and will require some drifting around damaged areas. The fire has been extinguished and the rehabilitation work has resumed. Access to the mine was closed through the Jewell Shaft between February and the end of June 2012.

Current plans for drifting around the fire damaged area on the 3100 Level have been included as a capital cost in this study. This bypass drift will be utilized to obtain the required air flows for proper ventilation of the mine. Basic rehabilitation on other levels is included in sustaining capital.

This study will include an updated ventilation system for the mine as well as a refrigeration system for areas on and below the 3700 Level. The measured and indicated resources are fairly scattered below the 4600 Level and are not included in the PEA mine plan.

16.3 Upper Country Mine Development Project

A 12 by 12 foot ramp will be driven from the Sterling Tunnel to the 2300 Level of the mine. This ramp will have several purposes:

- To provide drill stations for continued exploration;
- To potentially mine resources prior to the planned PEA study mill start-up date; and
- To provide stations to continue advancing required raise development for the new ventilation system.

This project is expected to commence during the second quarter of 2013. This project is called the Upper Country Project by SSMC. This project is intended to result in some preproduction from the mine.

To reduce mine development, Alimak raise mining is proposed for the stopes that are not wide enough for mechanical cut and fill mining. It should be noted that a detailed cost comparison should be completed for typical stopes. Most of the mine drill hole vein intersections are less than three feet wide and have been diluted to five feet wide. Most of the vein intersections can be diluted to more than three feet wide and still be ore-grade. The cost of Alimak slusher mining and processing a vein diluted to five feet should be compared to the cost of mechanically mining and processing a vein diluted to six or six and one half feet, including required development.

16.4 Geotechnical Considerations

A memo report completed on June 23, 2012 by Mark Board of Itasca Consulting Group summarized the general geotechnical conditions of the mine and mining methods. With his familiarity with the Silver Valley mines, their respective ground conditions relative to mining, and his significant career experience with seismic issues, he reviewed the locations, sizes, shapes, and orientation of stopes to mined out areas. This allowed him to recommend the best mining and backfill method for the stopes reviewed, including destressing when appropriate. This level of assessment was considered to be adequate for a PEA. More detailed modeling of stresses using the ultimate mine design was recommended for the next phase of development. MDA

incorporated Mark Board's recommendations in the sequencing of the stopes. Excerpts of this report are incorporated in this section below:

…a preliminary, empirical assessment of the geotechnical assessment of the mining methods and risk assessment of seismic potential based on reserve blocks defined by SSM staff and Mine Development Associates (MDA), Reno, Nevada. In this memo a sampling of the general types of mineable ore reserve blocks (supplied by SSM) (sic) from the prominent veins at the mine are examined from a geotechnical perspective and the following empirical assessments made:

- *estimate of the type of mining method that could be used to mine the block of ground*
- *estimate of the seismic risk associated with removal of this block of ground*

Unfortunately, detailed geological information on each block of ground is not available at this time to assist in prediction of other types of ground problems such as dilution and general squeezing ground issues. Therefore, only the general seismic risk is assessed based on experience in pillar bursting issues from Coeur d'Alene mines. Additionally, recommendations are given for studies to be performed during the feasibility-level studies and when underground access is re-established.

16.4.1 Geotechnical Assessment of Mining Approach

16.4.1.1 General

The Sunshine Mine ore resources are found within the St. Regis and Revette formation which consists of interbedded quartzites and argillites that vary from vitreous, strong and brittle quartzites, siltite, argillaceous quartzites and argillites. The beds vary in thickness from thin units (less than 1 ft) to several feet in thickness and dip steeply (typically 65° or greater) to overturned in some cases. Over 30 veins have been named and mined at the Sunshine Mine. Principal vein systems in the mine include the Sunshine, Chester, Copper, Yankee girl and West Chance. These veins strike east-west and dip about 65° to the south. Locally, dips range from 45° to 90°. Strike lengths exceed 2000 feet and dip lengths are two to three times greater than the strike length. Major veins are located between major faults at an angle of about 25° to the bounding faults. Veins vary in width from a few inches to over 30 feet, but are generally 1 to 5 feet thick. Ore minerals include tetrahedrite and galena with siderite and quartz as the principal gangue minerals. Other minerals include pyrite and arsenopyrite with minor to trace amounts of chalcopyrite, sphalerite, boulangerite, bourmonite, pyrargyrite, and magnetite.[1] Of importance from a geotechnical perspective is that the ore minerals tend to be strong, silicified and brittle in most cases and may be coupled with strong and brittle quartzitic wall rocks.

The in situ stress state has been measured at numerous mines in the Coeur d'Alene district, indicating a northwesterly bearing of the major horizontal stress component. Breakouts observed in the 10 and 12 shafts at Sunshine Mine are consistent with the northwest orientation of the major stress. Measurements at the Lucky Friday mine indicate a ratio of the major horizontal to vertical stress of about 1.5.

[1] *Geologic description taken from Wikipedia: Sunshine Mine.*

Seismicity at the Sunshine Mine has been documented by Scott, et al., (1997)[2], Whyatt, et al., (2002)[3] and Blake and Hedley (2003)[4]. Whyatt, et al. (2002) summarized basic rockburst mechanisms in the Coeur d'Alene district which indicates that the Sunshine was one of the more seismically active mines with most damaging seismicity related to pillar or face bursting. At Sunshine Mine, the overhand mining method created numerous small pillars that were subjected to high stress concentrations. The relatively brittle silicified ore and wall rocks of the Revette formation were subject to localized sudden failures of these brittle rocks. Some instance of fault-slip seismicity has also been noted. Scott, et al. (1997) document a fault-slip related seismic event between the 4400 and 4600 Levels of the Chance Vein, but it appears that fault-slip related events are less prevalent here than at other district mines.

SSMC staff and MDA are currently creating an inventory of the ore resources on the various vein structures at Sunshine based on the historical geologic and mine development maps and sampling assays conducted by Sunshine Mine. The maps have been digitized and a three-dimensional model created from which dimensions, tonnage and grade have been determined. Reserve blocks have been identified on over 25 veins with depth ranging from about 500 to over 4000' in depth. Major reserves exist at relatively shallow depth (500 to 2000' depth) and below 2700' (with significant reserves from 3400' and below). Since few geotechnical issues are expected at shallow depth, the deep reserves, below 2700', on the 09, Sunshine, West Chance and Yankee Girl veins were selected for a scoping-level geotechnical assessment of potential mining methods and approximate seismic risk.

16.4.1.2 Geotechnical Classification of Resources Blocks

…For the pillar case, the pillar will likely be in a highly-stressed state when mining is initiated. Due to the generally strong and brittle ore in the Sunshine lower levels, and the highly-confined nature of the pillars due to narrow vein thickness, there is no reason to assume that these pillars have naturally yielded and destressed since completion of the surrounding mining. Therefore, these pillar cases are considered to be of high potential seismic risk. The recommended mining method in these cases is underhand stoping with paste fill with potential use of destress blasting in advance of mining to precondition the ground.

In the case of blocks bounded on one side by mining, it can be assumed that the block will be subjected to some additional mining-induced stress, but not to the extent of a typical pillar condition. The seismic potential for this case is classified as moderate. Typically, these blocks would be mined by retreating from the adjacent mined face toward the solid abutment. Depending on the particular geometry, these blocks can be mined by either underhand mining (retreating downward away from a mined block above), by overhand mining (retreating upward from a block mined below), or by narrow-vein blasthole to endslice a block toward an abutment. In all cases, paste fill would be used with varying cement content depending on the mining method.

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[2] *Scott, D.F, T.J. Williams, and M.J. Friedel. (1997) "Investigation of a Rockburst Site, Sunshine Mine, Kellogg, Idaho," in Proceedings of the 4th International Symposium on Rockbursts and Seismicity in Mine (Krakow, Poland), pp 311–315.*

[3] *Whyatt, J., Blake, W., Williams, T. and B. White. (2002). "60 Years of Rockbursting in the Coeur d'Alene District of Northern Idaho, USA; Lessons Learned and Remaining Issues," in 109th Annual Exhibit and Meeting, Society for Mining, Metallurgy, and Exploration (Phoenix, Arizona, February 25-27).*

[4] *Blake, W. and D.G.F. Hedley. (2003) Rockbursts: Case Studies from North American Hard Rock Mines. Society of Mining Engineers.*

Finally, in the case of a block of ground isolated from existing stopes, any of the above mining methods are possible. Here, the stopes will be subjected to the in situ stress state and the seismic potential is classified as low. The geotechnical classification is summarized in Table 16.1.

Table 16.1 Classification Method for Mining Blocks

Geometry of Block and Surrounding Mining	Suggested Mining Methods	Seismic Potential
Pillar surrounded on two or more sides by existing stopes - full impact of stress concentration from previous mining	Underhand cut-and-fill with paste	High
Block against one existing stope face - impact of stress concentration from previous mining	Underhand cut-and-fill or end-slicing with narrow-vein blasthole (if appropriate) with paste fill	Moderate
Block surrounded by solid abutments - under in situ stress conditions only	Underhand cut-and-fill, overhand cut-and-fill, narrow-vein blasthole (all with paste fill)	Low

…it is suggested that about 50% of the tonnage in these lower level ore blocks be mined by underhand mining with paste fill, about 10% by overhand mining and the remaining 40% by one of the other methods, depending on the specific geometry of the ore block and its relation to adjacent mining. The approximate seismic potential of these lower blocks, based on the percentage of the total tonnage, indicates about 70-75% would be classified as having low potential, 20% moderate, and 5-10% in pillars with high potential.

The result of this classification of the ore reserves at depth (below 2700') indicates that the majority of identified mining blocks are not contained in high-risk pillar situations, but in isolated blocks, or blocks bounded on only one side by previous mining. This situation will provide Sunshine Mine with significant flexibility in scheduling production to optimize mining method and to manage geotechnical risk.

Itasca recommended that geotechnical work continue, as summarized in the conclusions and recommendations section. Future studies should include a stope-by-stope review of seismicity risks to evaluate potential mining methods.

16.5 PEA Mining Methods

Most of the ore and waste is planned to be hauled to the Jewell Shaft ore and waste pockets and hoisted to the surface up the Jewell Shaft. The exception will be material above the 1700 Level, most of which is planned be hauled up the upper mine ramp system to the surface.

The Jewell Shaft and hoist is planned to be rehabilitated and equipped with new lighter bottom dump skips, raising the hoisting capacity to 1,300 tons per day. The ore is planned to be unloaded into the concentrator's coarse ore bin, while the first 500,000 tons of waste is planned to be placed in the existing WRSF located near the Jewell Shaft area and the remaining waste will be hauled to the ConSil waste rock storage facility. Some waste can be used as backfill in longhole and overhand stopes.

Potential mining methods include underhand and overhand cut and fill mining generally using paste backfill, longhole stoping, and breast-down of exposed ore-grade material with minimal vertical extent.

16.5.1 Stope Design

MDA received 38 models of the veins present at the Sunshine deposit from Tetra Tech. The models contained information on the silver, copper, lead, and zinc grades for each block within the model, although many areas did not contain information on these grades which were modeled as half the detectable assay limits for these metals. The model contained a vein width and a diluted vein width. The diluted vein width was based on a diluted width of five feet for veins less than three feet and adding two feet of dilution to veins greater than three feet.

Each of the veins was plotted in section view to determine where mining had taken place (coded in the vein models) and where the remaining mineralization was. A cutoff grade of 10 oz Ag/ton for the diluted vein was used to outline stopes. A minimum 10 foot pillar was left between mined stopes and new stopes. Two mining methods were planned for the stopes. Alimak slusher mining will be used for areas where the stopes were generally less than 6.5 feet, and mechanical cut and fill mining will be used for areas where the stopes were generally greater than 6.5 feet. In general, underhand mining would be employed below 3100 feet and overhand above 3100 feet. More work is required to determine the correct areas to mine by overhand or underhand methods. Rock-bursts have occurred at the mine in the past, and for that reason, most of the material was assigned to be mined by underhand methods to minimize the chance of rock-bursts. Underhand stopes start at the top of a stope block and each new cut proceeds downward after the working cut has been backfilled with a cemented paste made from filtered tailings. Eight to ten percent cement is estimated to be added to the paste to provide a stable back to work under for the next stope cut.

Table 16.2 shows a summary of the measured, indicated, and inferred stopes that were outlined. About 19% of the stope tons are contained in measured and indicated resources, while 77% are in inferred resources, and about 4% of the tonnage in stopes is included as waste.

Table 16.2 Measured, Indicated & Inferred Resources Considered for PEA Mine Plan

Location	Ore	Ag	Cu	Pb	Zn	Diluted Width
	tons	*oz/t*	*%*	*%*	*%*	*ft*
06Vein	114,632	38.53	0.08	0.02	0.01	5.0
08BVein	110,936	24.77	0.23	0.02	0.02	5.2
08VeinDHWVein	235,021	22.89	0.05	0.04	0.03	5.4
09HWVein	156,681	22.11	0.16	0.04	0.02	5.1
09Vein	179,822	23.30	0.04	0.01	0.02	5.2
101Vein	117,719	35.81	0.13	0.03	0.02	5.0
625MVein	293,924	23.55	0.27	0.02	0.02	5.1
BVein	31,468	24.50	0.02	0.04	0.04	7.3
CFault	244,242	23.93	0.18	0.99	0.04	6.3
Chester	866,671	23.63	0.14	0.15	0.00	5.4
ChesterHang	308,291	23.21	0.09	1.10	0.00	6.0
ChesterHWSplit	346,707	32.90	0.22	0.05	0.00	6.1
CopperVein	394,806	27.89	0.07	0.24	0.02	5.6
DVein	386,179	27.78	0.02	0.03	0.03	6.9
FVein	131,377	22.83	0.08	0.13	0.00	5.1
GVein	14,891	12.59	0.11	0.01	0.01	5.0
HFWVein	86,808	17.33	0.05	0.02	0.02	5.2
HVein	111,187	37.84	0.16	0.05	0.03	5.6
KFWVein	108,449	23.20	0.44	0.01	0.01	5.0
KVein	96,009	21.59	0.22	0.03	0.00	5.4
NYankeeBoySunshine	480,310	24.60	0.05	0.04	0.03	5.6
S78Vein	54,450	18.24	0.06	0.02	0.02	5.2
SilverLine	132,845	14.17	0.12	1.49	0.01	6.4
SilverSummitNo3	338,910	24.85	0.54	0.03	0.03	5.5
SilverSummitNo4	1,096,581	26.10	0.84	0.03	0.03	5.8
SilverSyndicateLink	1,052,978	21.74	0.09	0.70	0.04	5.6
Sunshine2	27,259	20.85	0.01	0.02	0.02	5.0
SunshineFW	74,333	19.83	0.06	0.01	0.01	5.1
SYankeeBoy	970,570	22.04	0.02	0.03	0.03	6.1
W16Vein	18,351	66.14	0.09	0.01	0.01	5.2
WestChance	601,141	26.06	0.17	1.46	0.00	5.9
WestChanceFW	140,222	19.59	0.02	0.01	0.00	6.4
WestChanceFWWest	12,007	11.37	0.07	0.00	0.00	6.5
YankeeGirl	974,619	22.54	0.07	0.02	0.01	5.2
YankeeGirl952Split	83,388	13.95	0.00	0.01	0.01	5.0
YankeeGirlFW	106,415	36.39	0.02	0.03	0.03	5.8
YankeeGirlHW	61,736	20.86	0.01	0.03	0.03	5.5
YankeeGirlWestFlare	71,980	11.69	0.01	0.01	0.01	5.0
Total / Average	**10,633,915**	**24.37**	**0.19**	**0.27**	**0.02**	**5.7**

Once the possible stopes were drawn, they were investigated to determine which stopes were potentially minable. Small isolated stopes were omitted from the potential stopes in this study. Table 16.3 shows a summary of the potentially minable stopes by vein.

Table 16.3 Potentially Minable Stopes – Summary by Vein

Location	Ore	Ag	Cu	Pb	Zn	Diluted Width
	tons	*oz/t*	*%*	*%*	*%*	*ft*
06Vein	109,968	39.11	0.08	0.02	0.01	5.04
08BVein	104,962	24.98	0.23	0.02	0.02	5.47
08VeinDHWVein	181,653	23.09	0.05	0.03	0.03	5.38
09HWVein	123,046	22.31	0.18	0.04	0.02	5.14
09Vein	136,697	21.70	0.04	0.01	0.02	5.15
101Vein	90,444	40.05	0.16	0.03	0.02	5.00
625MVein	115,809	26.23	0.45	0.01	0.01	5.00
BVein	0	0.00	0.00	0.00	0.00	0.00
CFault	202,504	24.72	0.17	1.13	0.04	6.41
Chester	401,742	25.96	0.18	0.15	0.00	5.32
ChesterHang	279,107	23.87	0.09	1.20	0.00	6.14
ChesterHWSplit	99,442	40.37	0.15	0.05	0.00	5.76
CopperVein	213,922	38.01	0.04	0.40	0.02	5.29
DVein	351,413	28.20	0.03	0.03	0.03	7.02
FVein	114,926	22.67	0.09	0.14	0.00	5.09
GVein	0	0.00	0.00	0.00	0.00	0.00
HFWVein	9,412	18.40	0.10	0.00	0.02	5.01
HVein	59,972	56.58	0.25	0.06	0.03	5.63
KFWVein	24,490	29.31	0.52	0.01	0.01	5.00
KVein	12,238	30.97	0.27	0.01	0.00	6.87
NYankeeBoySunshine	416,444	24.64	0.06	0.04	0.03	5.90
S78Vein	50,763	18.01	0.06	0.02	0.02	5.23
SilverLine	132,845	14.17	0.12	1.49	0.01	6.52
SilverSummitNo3	283,089	26.44	0.55	0.03	0.03	5.54
SilverSummitNo4	940,295	26.19	0.83	0.03	0.03	5.92
SilverSyndicateLink	810,437	23.32	0.05	0.81	0.04	6.65
Sunshine2	22,343	20.17	0.01	0.02	0.02	5.00
SunshineFW	68,460	19.49	0.05	0.01	0.01	5.12
SYankeeBoy	838,996	22.39	0.02	0.03	0.03	6.15
W16Vein	17,603	67.88	0.09	0.01	0.01	5.20
WestChance	465,514	21.62	0.21	1.64	0.00	6.61
WestChanceFW	104,707	21.30	0.01	0.01	0.00	6.51
WestChanceFWWest	10,894	11.64	0.08	0.00	0.00	6.50
YankeeGirl	944,209	22.75	0.07	0.02	0.01	5.21
YankeeGirl952Split	77,353	14.11	0.00	0.01	0.01	5.00
YankeeGirlFW	106,415	36.39	0.02	0.03	0.03	5.79
YankeeGirlHW	61,736	20.86	0.01	0.03	0.03	5.50
YankeeGirlWestFlare	71,980	11.69	0.01	0.01	0.01	5.00
Total	**8,055,830**	**25.04**	**0.19**	**0.31**	**0.02**	**5.87**

The potential stopes below the 4600 Level were only added to the production schedule after mining all the material above the 4600 Level. The total material contained in the stopes below the 4600 Level is about 1.5 million tons.

16.6 Mine Development

16.6.1 Mine Rehabilitation and Upgrading

The mine has operated over a considerable period and requires upgrading or rehabilitation in a number of areas.

16.6.1.1 Jewell Shaft Hoist and Headframe Rehabilitation

The Jewell Shaft repair activities recommenced in June 2012. The activities planned for 2012 and 2013 are:

- Detailed shaft cleandown, relieving all sets of accumulated debris, and any bolting or timber replacement that is found necessary as the cleandown is carried out. This is planned double shift seven days per week and is expected to take 12 months, or between May 2013 and May 2014. Any bearing sets deemed necessary will be installed during this period. In addition, a new 12.3kV power cable and pump column will be installed following the completion of the rehabilitation work.

- Repair and refurbishment of the Jewell headframe, as commented on by Tiley in their hoist system audit, is planned to be completed in 2013.

- Repair and rebuild of the current skip loading facilities on 2780 and 3180 Levels. The rebuilds are planned to be temporary in anticipation of a complete refurbishment of the muck transfer system in 2013. The PEA includes costs in 2013 for the replacement of the Jewell skips and cages, so any temporary modifications to the current skip loading facilities will need to take into account the future conveyances.

16.6.1.2 Silver Summit Shaft Rehabilitation

As of the end of July 2012, the Silver Summit headframe replacement is complete, and in a condition ready to commence sub-collar repair to the 3000 Level Silver Summit Station. All equipment necessary for this repair has been received, with the exception of a small stripping deck and a Cryderman tugger. The timber for the repair has been ordered and received and is being properly stored on the ConSil waste rock stockpile. The remaining repair has been deferred until May 2013. The current plan estimates completion of this repair 12 months from the start.

16.6.1.3 Level Rehabilitation

The 2700 Level and part of the 3100 Level will be rehabilitated as a sunk cost. Levels 1700, 1900, 2300, 2900, 3400, 3700, 4000, 4200, 4400, 4600, and 4800 will be rehabilitated over the life of the mine. Level rehabilitation includes clearing loose muck on the tracks, replacing utility lines, track, and installing rock bolts and support as necessary. An allowance of $400 per foot rehabilitated has been included for mining through and supporting any caved areas on the levels, with the exception of the 3100 Level. A new drift is planned around the fire damaged area and will be used for mine ventilation as well.

16.6.1.4 Muck Handling Improvements

An upgrade to the Jewell Shaft muck transfer system is planned in the first two years of production, which is based on the scope shown below. The outcomes of implementing these upgrades will be:

- Improvement to the safety of workers when loading muck skips,
- Centralization of skip loading activities in the shaft at the 3180 Level, improving overall shaft efficiency,
- Increased underground muck storage.

These outcomes will improve the overall productive capacity of the system.

The upgrade will include:

- A waste pass from the 1900 Level to the 3800 Level,
- A cascaded ore pass system from the 1900 Level to the 3800 Level,
- Grizzly stations on the 2700, 3100, and 3700 Levels,
- Camel back car dump system on levels,
- Access drift to a new skip loading facility from the 3700 Level to the 3850 Level, and
- Skip loading station infrastructure.

16.6.1.5 Mine Dewatering

The mine is currently flooded below the 3540 Level. The current pumping system is capable of pumping about 650 gpm from the mine and consists of:

- One 75 hp submersible pump in the Jewell Shaft bottom (4000 Level),
- One 200 hp two-stage centrifugal pump (3100 Level),
- Two 400 hp two-stage centrifugal pumps (2700 Level), one operating and one spare, and
- Two 350 hp eight-stage centrifugal pumps (1700 Level), one operating and one spare.

Mine dewatering below the 4000 Level is by a series of portable pumps. Improvements planned for the mine dewatering system include rebuilding all of the stationary pumps with new motor control centers, a new 8 inch pump column for the Jewell Shaft collar to the 3100 Level, and enlarging the 1700 Level pump room.

16.6.1.6 Mine Compressed Air

The current compressed air system has two Atlas Copco 700 hp screw compressors with a combined capacity of 6,000 cfm. The mine and mill will require 10,000 cfm to accommodate the PEA plan. Two additional Atlas Copco 700 compressors are planned to be added to supplement

the current system, including a cooling package upgrade, which will eliminate the need to use water from Big Creek for cooling.

16.6.1.7 Mine Power

A new power feeder line has been included down the Jewell Shaft. New electrical substations have also been included.

16.6.1.8 Mine Ventilation and Refrigeration

The mine ventilation system will be completely revamped. Mark Butterworth from BBE Consulting (South Africa) completed conceptual studies for mine ventilation. BBE wrote in the executive summary of their report, Sunshine Mine Concept Study: Ventilation and Cooling Requirements {Rev 1} of May 2012:

The objective of the work is to determine technical and cost viability and relates to engineering analyses, general design and conceptual descriptions at concept level of detail. The ventilation design principles used are proven and recognized internationally. The ventilation design is based on achieving an average stope wet-bulb temperature of 80.0°F at depths where the virgin rock temperature will approach 107°F for the deepest mining level when mining at 4800'.

A total airflow of 450 kcfm is required and will be provided by main fan stations located at the exit to the Sterling Tunnel and at the top of a new RBH located on surface in the vicinity of Silver Summit shaft. Refrigerated air will be required when mining below 3700L [cooling horizon]; this will be provided by a 1000 ton underground refrigeration plant located on 3100L. Cold water will be distributed in closed circuit to spot coolers located on or below 3700L.

Existing shaft infrastructure cannot provide 450 kcfm ventilation and a number of additional RBHs are required, specifically:

- *Downcast from surface to 1900L [adjacent to Jewell Shaft]*
- *Downcast from 1900L to the cooling horizon [3700L]*
- *Upcast from 3000L to 1750L [adjacent to Silver Summit Shaft]*
- *Upcast from 1750L to surface/Silver Dollar tunnel*

Figure 16.6 shows a longsection view of the proposed new ventilation concept.

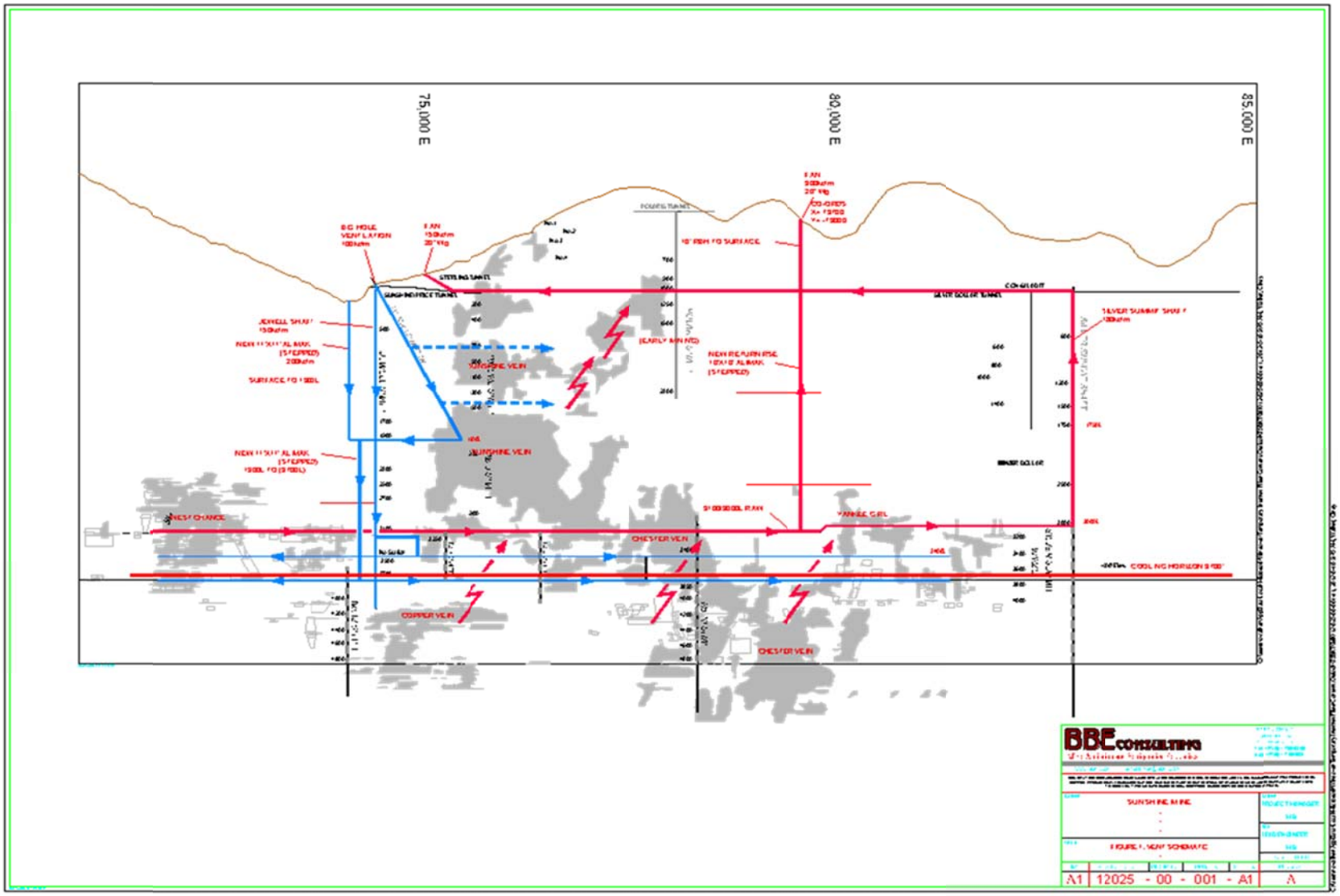

Figure 16.6 New Sunshine Ventilation Concept

16.6.1.9 Backfill Distribution

A paste backfill system was conceptually designed by Fred Brackebusch of Mine System Design, Inc. from Kellogg, Idaho in May 2012. In Mr. Brackebusch's report, entitled Design Rationale and Paste Distribution System Conceptual Backfill System Design, he reported:

The planned mining rate generates about 312 cubic meters of void space to be filled per day. It was decided by consensus that a rate of 50 stph of tailings would be established for this study with the possibility of increasing to 75 stph. Generally, the entire tailings stream would be sent to backfill when the backfill plant operates, but a processing option of the paste plant would allow classification of the tailings stream with about 67% recovery to underflow or 33.5 stph of tailings to be used for paste backfill. This range of rates of tailings usage together with 5% binder corresponds to a range of filling rate from about 22 cubic meters per hour to 33 cubic meters per hour. If it is assumed that the average filling rate is mid-range, the daily filling rate would be 660 cubic meters per day which would fill 636 cubic meters of stope volume assuming 3.6% bleed. Thus the backfill plant utilization would be …49%, assuming all stopes are filled with paste. The plant utilization could be increased, if needed in the future, to about 85% thus providing paste backfill for a total of 1,700 stpd of production.

A paste pump is required due to need to distribute paste a significant distance horizontally before entering the borehole or shaft pipeline. Maximum pump pressures should not exceed 300 psi. In most cases the pump will be metering paste to the surface borehole or shaft pipeline with only minimal pressure, say 125 psi. Other issues including operating procedures, emergency procedures, flushing, and pipe wear will be addressed in the detailed engineering phase.

…duplicate inclined boreholes would be drilled from the surface in the concentrator area to intersect [or nearly] mine workings on the 3100 level. All boreholes at the Sunshine mine, including intermediate boreholes between mine workings, must be cased with steel casing because of ground conditions and fault zones. The intermediate boreholes would be cased to 4-inches internal diameter without an internal pipe liner, but the main boreholes would be cased with approximately 6-inch internal diameter steel casing. In the main boreholes an internal liner consisting of 4-inch pipe with 0.337 inch wall thickness and hardened to 600 brinnel would be installed. The annulus would be protected from corrosion by painting and lubricant so that the internal pipe can be removed and replaced.

Assuming a near full pipe, flowing condition the maximum pressures occur either at the bottom of the shaft pipeline or the bottom of the main boreholes, depending on which option is chosen. In the shaft option the pressures are slightly greater than 1,200 psi, and in the borehole option the pressures are about 600 psi.

The permanent pipeline would be 4-inch [1OOrnrn] diameter rated for 1,500 psi [dependent upon coupling types]. …the orebody extends to a significant height above 3100 level, so the paste distribution system will have to be modified by taking off from either the shaft pipeline or the main boreholes at a higher elevation. … a crosscut would be driven to intercept the boreholes on an upper level.

A new process plant with a paste backfill facility is expected to start-up in July 2014. Prior to the startup of the new paste plant, a contractor will provide a cemented crushed rock slurry to be used as backfill of the preproduction stopes.

16.6.2 Development Schedule

A schedule was prepared for the mine development. Two items determined the duration of the development program. The Silver Summit Shaft must be rehabilitated to provide a second escapeway for the mine. This is planned to commence during the second quarter of 2013 and be completed about a year later. The second item is to complete the new ventilation system prior to commencing mine production. This requires two new vent raises and a bypass drift on the 3100Level around the fire-damaged area.

It is expected that development will commence on the Upper Country mine exploration program during 2013, parts of which will be used to develop the mine's new western ventilation raise. Completion of the new ventilation system is not expected until the end of 2016. A new 1,000 tpd plant is expected to be constructed and operational by mid-year 2014. Stopes planned in the Upper Country are scheduled to provide plant feed until the 3100 Level bypass and new ventilation system are completed in early 2017.

A mine contractor is planned to be used for all development and mining for the first two years of development. All vent raises are also planned to be completed by a contractor. The 3100 Level bypass is over 6,000 feet in length and is planned to be completed by crews working on both sides of the bypass from the Jewell and Silver Summit shafts. All mining and development after the first two years is planned to be completed by the Owner, with the exception of the ventilation raises.

Table 16.4 shows estimated mine development unit costs and Table 16.5 shows the planned development for the mine. Figure 16.7 shows the planned mine development in longsection view.

Table 16.4 Mine Development Unit Costs ($/ft)

Development Type	Owner - Cost ($/ft)	Contractor - Cost ($/ft)
12 x 12 Ramp	1,030	1,440
10 x 10 Drift	860	1,200
8 x 10 Attack Ramp	740	1,030
10 x 10 Alimak Vent Raise		1,360

Table 16.5 Mine Development Schedule

Period	Development, ft										
	Rehab	ALI	VAC	NVT	DRF	Bypass 3,100	Bypass	MKB	RMP	ATK	Totals
-2	0	466	319	349	0		0	18	300	0	**1,452**
-1	1,952	4,907	0	804	577		0	256	4,260	0	**12,756**
1	1,597	5,226	138	0	3,678	4,243	0	269	4,482	0	**19,634**
2	3,841	1,184	162	2,368	4,379	2,015	1,942	340	5,659	0	**21,890**
3	5,840	4,249	0	0	4,256		3,650	234	3,898	0	**22,127**
4	5,840	6,090	535	2,252	3,225		1,005	321	5,356	1,306	**25,930**
5	5,840	2,086	0	0	5,555		32	60	1,006	2,199	**16,779**
6	5,856	2,715	0	0	3,599		433	170	2,829	135	**15,737**
7	5,840	8,134	0	0	3,636		0	153	2,542	2,017	**22,322**
8	5,840	5,840	57	602	3,650		0	219	3,650	2,186	**22,044**
9	4,624	5,856	0	0	3,198		0	219	3,650	2,201	**19,748**
10	0	5,840	0	0	3,660		0	220	3,660	2,494	**15,874**
11	0	5,840	0	0	3,650		0	219	3,650	1,372	**14,731**
12	0	5,840	0	0	3,480		0	219	3,650	0	**13,189**
13	0	4,385	0	0	3,118		0	219	3,650	130	**11,502**
14	781		0	0	2,359		0	220	3,660	4,353	**11,373**
15	2,091	1,272	0	0	1,613		0	219	3,650	394	**9,238**
16	9,256		0	0	5,704		130	198	3,301	3,001	**21,592**
17	9,256	174	0	0	5,704		130	198	3,301	3,001	**21,765**
18	9,256	499	0	0	5,704		130	198	3,301	3,001	**22,091**
19	9,256	1,419	0	0	5,704		130	198	3,301	3,001	**23,011**
20	9,256	1,532	100	400	5,704		130	198	3,301	3,001	**23,624**
21	5,785		100	400	3,565		81	124	2,063	1,876	**13,995**
22	5,785	3,953	100	400	3,565		81	124	2,063	1,876	**17,948**
23	9,305	5,261	0	0	5,513		0	205	3,418	2,126	**25,828**
24	9,305	7,859	0	0	5,513		0		3,418	2,126	**28,221**
25	9,305	7,840	0	0	5,513		0		3,418	2,126	**28,202**
26	9,305		0	0	5,513		0		3,418	2,126	**20,362**
Totals	**145,015**	**98,467**	**1,511**	**7,575**	**111,338**	**6,258**	**7,877**	**5,016**	**93,854**	**46,049**	**522,960**

ALI - Alimak Raise; VAC - Vent Access; NVT - Vent Raise; DRF - 3100 Vent Drift; RMP - Access Ramps; ATK - Stope Attack Ramps

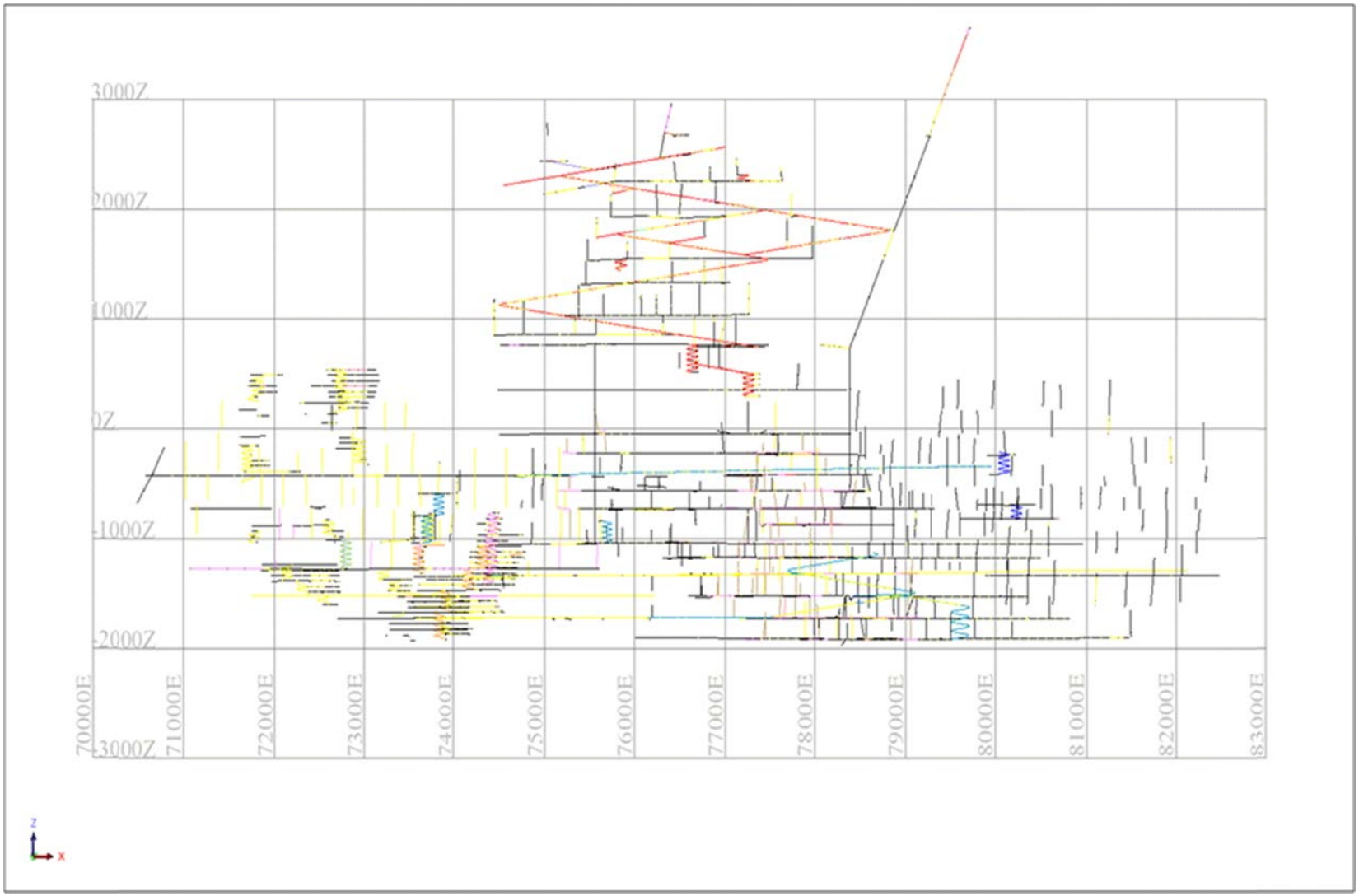

Figure 16.7 Sunshine Mine Development

16.6.3 Production Schedule

Mine ore production is scheduled to start during the second quarter of 2014 however, this material will be stockpiled until the mill start-up scheduled for the third quarter of 2014. Production from the mine is limited to the Upper Country stopes until the ventilation system and secondary escapeway are completed at the end of 2016. Mine production is expected to increase when the stopes are available to sustain a 1,000 tpd operation, currently scheduled to occur in the second half of 2016. The production was scheduled in detail for the first 15 years, with the remaining production added first from material above the 4600 elevation and second from below the 4600 Level.

Table 16.6 summarizes the mine production schedule.

Table 16.6 Mine Production Schedule

Year	Ore Tons 000's	Ag oz/ton	Cu %	Pb %	Zn %	Diluted Width Avg
(-1)	13.7	29.07	0.46	0.86	0.04	5.58
1	155.8	24.24	0.14	0.10	0.03	5.83
2	175.8	24.42	0.11	0.06	0.03	6.03
3	365.0	24.37	0.18	0.13	0.02	6.05
4	365.0	23.50	0.19	0.03	0.02	5.95
5	365.0	26.39	0.16	0.03	0.03	6.11
6	366.0	24.42	0.12	0.14	0.03	6.12
7	365.0	22.63	0.13	0.19	0.02	5.96
8	365.0	24.61	0.14	0.22	0.02	5.98
9	365.0	24.06	0.18	0.21	0.02	5.90
10	366.0	23.64	0.25	0.25	0.02	5.81
11	365.0	23.99	0.24	0.28	0.02	5.71
12	365.0	24.57	0.17	0.20	0.02	5.68
13	365.0	24.12	0.16	0.19	0.02	5.68
14	366.0	26.57	0.19	0.24	0.02	5.76
15	365.0	26.65	0.21	0.33	0.02	5.73
16	365.0	25.72	0.22	0.52	0.02	5.82
17	365.0	25.72	0.22	0.52	0.02	5.82
18	365.0	25.72	0.22	0.52	0.02	5.82
19	365.0	25.72	0.22	0.52	0.02	5.82
20	365.0	25.72	0.22	0.52	0.02	5.82
21	365.0	25.72	0.22	0.52	0.02	5.82
22	365.0	25.72	0.22	0.52	0.02	5.82
23	365.0	25.72	0.22	0.52	0.02	5.82
24	365.0	23.27	0.05	0.16	0.16	5.71
25	365.0	22.74	0.01	0.08	0.19	5.69
26	365.0	22.74	0.01	0.08	0.19	5.69
27	365.0	22.74	0.01	0.08	0.19	5.69
28	179.3	22.74	0.01	0.08	0.19	5.69
Total	**9,652.5**	**24.63**	**0.16**	**0.27**	**0.05**	**5.83**

17.0 RECOVERY METHODS

17.1 Introduction

The Sunshine Mine's processing facility will receive ROM ore delivered by an existing hoist in the Jewell Shaft on the Sunshine mine and processing plant site. ROM ore will be delivered by hoist to the ROM ore bin and then fed to the primary crushing circuit. Material from the Sterling Tunnel will be delivered to a truck dump and fed to the primary crushing circuit.

The processing facility is expected to consist of a comminution circuit followed by a silver and copper flotation circuit and a subsequent lead flotation circuit to produce two concentrates. The two concentrates are planned to be thickened and filtered for load out to bulk bags. Concentrate in bulk bags would be stored on site for shipment to appropriate metal recovery facilities. The concentrate storage facility would accommodate enough concentrate for at least five days production.

The silver, copper, and lead metals will all be included within the two concentrates produced in the flotation system. In total, approximately 97% of the silver, 94% of the copper, and 85% of the lead in the ore is expected to be contained in the two concentrates, based on historical estimates of recovery. This recovery may improve once metallurgical testwork has been completed and the grinding and flotation process has been adjusted to optimize recovery of all three metals.

The lead concentrate will also contain varying amounts of silver depending upon the origin of the ROM plant feed in the mine. Historically the Coeur d'Alene district has produced silver rich galena concentrates as well as the high silver-copper tetrahedrite containing concentrates. Stopes outside of the main Sunshine zone area of the mine contain increasing amounts of galena bearing ore. Depending upon the production stopes feeding the mill, variable amounts of lead bearing ore will be processed in the mill. The lead concentrates with the galena-argentite mineral association have considerably lower contents of silver compared to the silver-copper tetrahedrite concentrates. The lead-silver concentrate will be sold to a lead toll smelter such as Teck Metals Ltd.'s smelter at Trail, BC, Canada.

The silver-copper concentrate contains most of the silver and is included with the copper in a tetrahedrite matrix. Antimony, a penalty metal in this concentrate, would be retained in the residue after the silver and copper have been leached.

The high-grade silver-copper concentrate may be further refined by one of the following methods:

- Transport the concentrate to a hydro-metallurgical refinery that would take the high-antimony concentrate for recovery of silver, copper, and an antimony-rich residue.
- Ship the concentrate to a refinery where silver and copper metal would be leached from the concentrate and the remaining antimony-rich residue would be shipped to US Antimony Corp. in Thompson Falls, Montana.
- Leach antimony to an acceptable penalty level and sell the high-grade silver-copper concentrate to a smelter. This approach would require an autoclave-based leaching operation.

- Leach silver, copper, and antimony, recovering all the metals. This approach would also require an autoclave-based leaching process plant.
- Sell the concentrate to a smelter, as is, accepting any antimony penalties included. This option may not be viable based on the potentially high antimony content.

The final processing methodology will be determined in the next phase of project development, after metallurgical testing results are available. For the basis of this Technical Report, the silver-copper concentrate is planned to initially be processed at a commercial refinery the first year of plant operation and at an onsite refinery commencing in the second year of mine operation.

The concentrator estimated process power requirement is 3,775 kW (5,062 connected hp). This includes crushing, grinding, flotation, pumping, concentrate handling and storage, reagent mixing and storage, process utilities, and tailings handling and pumping, using as much tailings as possible as a 50% cemented paste mine backfill.

Based on typical demand and load factors, the maximum demand is estimated to be 3,330 kW and the average demand is estimated to be 3,000 kW.

18.0 PROJECT INFRASTRUCTURE

18.1 Site Plan

The Sunshine Mine is located in a constrained, topographically-challenged area approximately two and one-half miles south of I-90 on Big Creek Road. The mine site is divided into an east and west side by Big Creek. Big Creek Road passes through the property on the west side of Big Creek and the majority of the existing mine site, dating back to the early 1900s, is situated on the east side of Big Creek.

Due to the age of the existing infrastructure, the majority of the buildings will be demolished and new buildings are planned to be constructed for operations. Numerous options for facility locations were identified and sequencing of the demolition will be evaluated in the feasibility stage. Potential locations for the process plant and ancillary facilities were based on the following considerations:

- Utilization of the existing topography to minimize site development and mass earthworks;
- Incorporation of vertical facilities to the extent possible due to limited space

For purposes of this PEA, one possible site plan of those identified is shown below in Figure 18.1. Further evaluations of all potential site plans will be completed during the next phase of project development.

Ancillary facilities, including the existing TSF and existing WRSF, are located in close proximity to the mine. The TSF is located approximately one mile north of the mine site on the west side of Big Creek and Big Creek Road. The TSF was permitted for seven lifts and it is currently halfway into lift number five. The WRSF is located approximately one-quarter mile north of the mine site on the east side of Big Creek Road. It currently has the capacity to handle the waste from the Sterling Tunnel and some of the existing waste rock will be used for development of the remaining lifts for the TSF. SSMC is also permitted to store waste rock in the ConSil WRSF located approximately four miles east of the mine site.

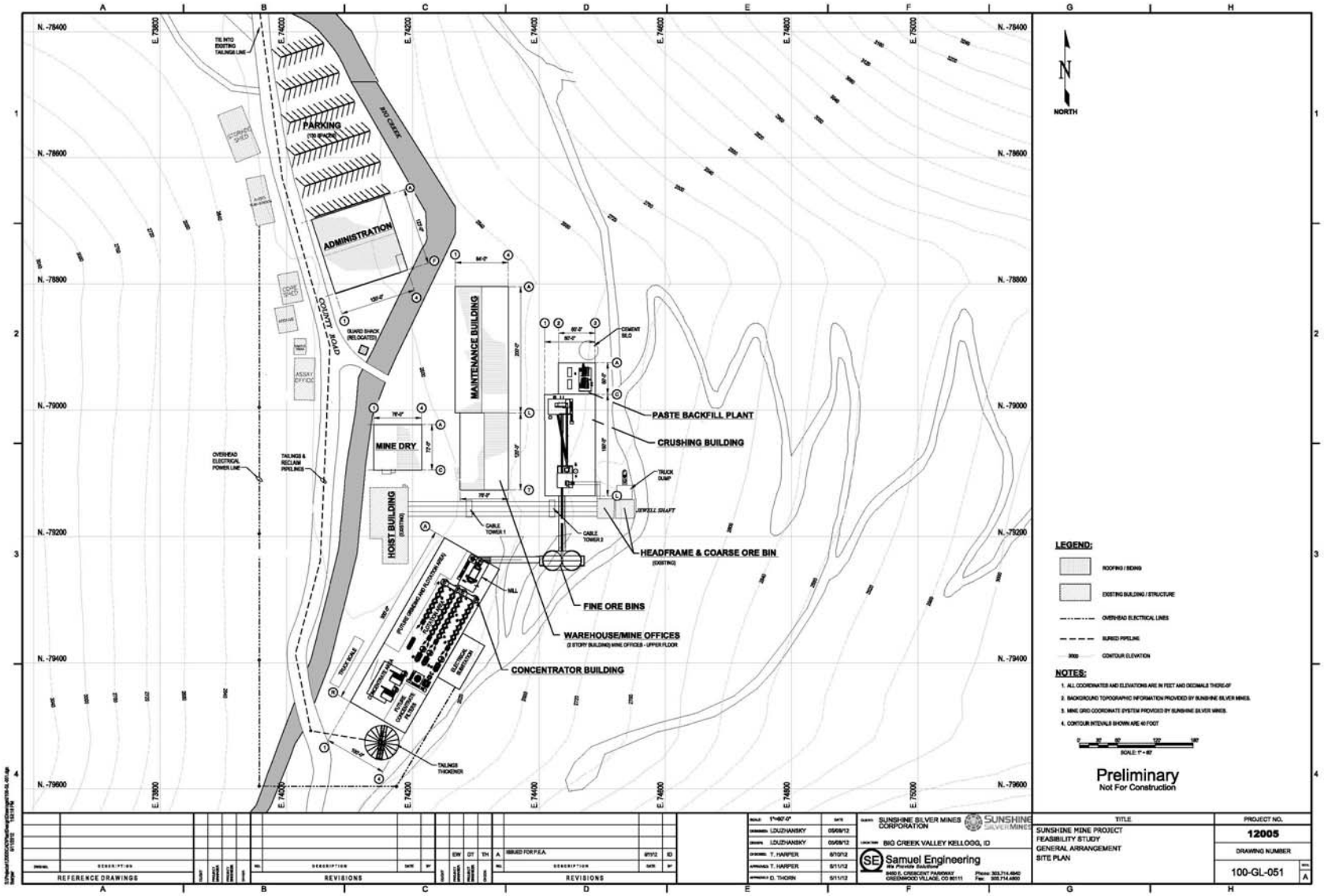

Figure 18.1 Conceptual Site Plan

18.2 Plant Site Grading

Due to the location of the mine site and the challenging topography, a substantial amount of earthworks and site development will need to be completed for newly constructed facilities. A geotechnical investigation was completed to evaluate the subsurface soil, bedrock, and groundwater conditions within the proposed construction areas. Nine geotechnical penetration boreholes were drilled in the plant site area and results from the drilling concluded that existing foundations and fill within the mine site area should be excavated and removed to enhance the support for the new facilities. It is recommended that foundations be over excavated and structural fill placed before new facilities can be constructed. In some areas, retaining walls will need to be installed for ground support.

Groundwater was encountered in several of the borings at depths ranging from 7 to 28 feet below the surface. As new construction commences, the site will need to be graded so storm run-off is diverted away from buildings to the drainage collection system.

18.3 Water Supply

Water supply is abundant as Big Creek passes directly through the mine site and SSMC currently has four water rights licenses, three surface water licenses from Big Creek and one groundwater well. Water from Big Creek is drawn from intake stations located south of the mine and is used for water supply, including process make-up, non-contact cooling, fire protection, and other non-potable uses. The combined water volume available is approximately nine cubic feet per second. Water storage is not an issue for the mine due to the abundance of water rights.

Preliminary water requirements for the project were developed by Golder Associates. For purposes of the PEA, the make-up water requirement (200 m^3/hr) can be taken from Big Creek via the pump station located one mile south of the mine site. The Sunshine Mine will not require any storage facility to provide water through the dry season.

Potable water is obtained from a water line that runs up Big Creek Road to the mine and is owned and maintained by the municipal water district, Central Shoshone County Water.

18.4 Main Access Road

The Sunshine Mine is located approximately 37 miles east of Coeur d'Alene, Idaho along I-90, and four and one-half miles southeast of the town of Kellogg, Idaho at the Big Creek exit. The main access road to the project is Big Creek Road, which is paved and well maintained year round. Roads to all plant facilities currently exist, so no new roads are expected to be constructed.

With the mine site just off of I-90, there are neither logistical issues for equipment and supply deliveries nor shipment of concentrates for refining. From I-90, concentrate can be trucked to smelters and refineries located in Idaho, Montana, or Canada, or transported to overseas or eastern Canadian smelters.

18.5 Security

Access to the main plant site is via an SSMC-owned bridge across Big Creek. A guard house is located at this entrance and is manned 24/7 to monitor access to the site. Security personnel

maintain perimeter security, authorizing access to incoming personnel, and performing roving patrols around the site area. Natural barriers, including Big Creek and rugged terrain, are augmented with wire fencing to enhance security.

18.6 Power Supply

The local utility company, AVISTA Utilities, owns and maintains the 13.2kV line which parallels Big Creek Road to the mine site. This power line is dedicated to the Sunshine Mine and it terminates at the AVISTA substation located on the north end of the property. Conceptual designs in support of this PEA have considered replacing as many as a dozen older, existing substations throughout existing facilities with several modern, more efficient substations located adjacent to the major load centers envisioned as part of the new processing facilities. As the project advances, a power study will be conducted to ensure critical areas of operation maintain power at all times. Back-up generators will be in place to maintain power to critical systems during power outages.

18.7 Water Treatment and Mine site Sewage

Precipitation and surface flows will be diverted around processing and mining facilities, thereby negating the need for any form of treatment. To the extent possible, process water will be retained within process circuits for reuse, reducing the amount of freshwater make-up required from the water supply source. Water from mine dewatering is pumped into the TSF, treated to the necessary water quality standards, and discharged to the South Fork per the mine's discharge permit.

A sewage leach field is located to the south of the current administration building under the parking lot. The leach field supports the administration building. Effluent from processing and ancillary facilities east of Big Creek is discharged to the TSF. In an ongoing effort to improve the overall infrastructure of the Big Creek transportation corridor, Shoshone County and the South Fork Sewer District have identified the need to enlarge and extend the current municipal sewer system to include the Sunshine Mine. This improvement is scheduled to occur within three years.

18.8 Employee Transportation

With the mine site being located near a well populated region, SSMC does not provide company housing or transportation. Movement of workers about the project site is by company-supplied vehicles.

For management, supervisory, and technical staff subject to call-out during off-hours, company vehicles are assigned for transportation between local communities and work locations.

18.9 Fire Protection

An existing firewater loop circles the site and water is drawn from Big Creek by a 1500 GPM (at 130 psi vertical head) diesel driven fire pump. The fire pump is maintained in the manual start position with the local fire department having a start key that will start the pump from remote locations. The pump charges all nine fire hydrants and supplies water to the 40,000 gallon firewater tank located above the mine. A 30 hp pump maintains tank level automatically by level control instrumentation. In the event of failure, a 50 hp pump is available as a backup.

Approximately 60% of the mine site is protected by automatic sprinklers. There are nine valve stations in which all the valves were replaced in 2006 by modern dry pipe valves. Nine separate compressors are available to provide air to the dry side of each system. In the event of fire, the sprinkler heads will break seal and air pressure will drop, allowing the valve to open. Each valve station is equipped with water flow detection alarms which alert mine personnel to contact the local authorities upon flow detection.

Portable fire extinguishers are located throughout the mine site facilities in accordance with fire codes and standards and MSHA Standards. All extinguishers are inspected monthly and tested annually.

The Kellogg Fire Department is located approximately five miles from the mine site and responds in a timely manner. Thus, the mine does not maintain a fire brigade or fire fighting hoses.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

An independent marketing consultant, Takefumi Maene, conducted a marketing study for the Sunshine Mine Project. The study was initially divided into two phases with the first phase completed in the second quarter of 2012. The first phase included the review of historical metallurgical data and provided early marketing information regarding the products, including penalty elements that could affect the conceptual flowsheet and unfavorable payment terms. The second phase includes a review of preliminary metallurgical test results and completion of the marketing study, smelter selection, providing information on the marketability of the products, current market pricing for payable metals, treatment charges, refining charges, penalty elements, and freight charges. The second phase of the study is planned to commence once the metallurgical testwork program is completed, currently scheduled for the fourth quarter 2012.

For the initial year of plant operation, both the silver-copper and lead concentrates are planned to be shipped to commercial refineries or smelters. Pricing was obtained from Teck Metals Ltd.'s facility in Trail, BC to process the lead concentrate and pricing for processing the silver-copper concentrate was received from Xstrata's Horne Smelter in Quebec. The terms and conditions received from these facilities form the basis for the terms used in the economic model and are shown in the Table 19.1 below.

Table 19.1 Concentrate Processing Charges

Description	Rate *USD*	Unit
Year 1 - Silver-Copper Concentrate		
Payable: Silver @ 90%		
Copper @ 100%		
Treatment Charges ($195/DMT)	176.87	/dst
Refining Charges - Silver Con		
Refining Charge - Silver	0.35	/Toz
Refining Charge - Copper	0.20	/lb
Copper Price Participation	0.25	/lb
Penalties - Silver Con		
Lead: $2.00/DMT/1.0% > 0.5% Pb	1.81	/dst
Arsenic: $8.00/DMT/0.1% > 0.5% As	7.26	/dst
Antimony: $2.75/DMT/0.1% > 10.0% Sb	2.49	/dst
plus $100.00/DMT base penalty	90.70	/dst
Year 2 - LOM Silver-Copper Concentrate		
Payable: Silver @ 100%		
Lead @ 100%		
Refining Charges - Silver Dore *	0.50	/Toz
LOM - Lead Concentrate		
Payable: Silver @ 95%		
Lead @ 95%		
Treatment Charges - Lead Con	362.88	/dst
Refining Charges - Lead Con	1.50	/Toz
Penalties - Lead Con		
Arsenic: (> 0.3%)	3.00	/dst

**Year 2 - LOM after onsite refining*

The following inputs were considered for establishing base metals prices for economic evaluation purposes:

- Silver, copper, and lead market prices were drawn from consensus pricing commodity decks provided by Morgan Stanley (last updated November 5-28, 2012). Consensus pricing is accomplished by periodically surveying major commodity forecasters and reporting their results for yearly and long-term forecasts.

- Kitco pricing and forecasts for the three past years of actual prices plus two years forward forecast

Based on the above information, the basis for the price figures in the forecast was to use the median future price forecasts provided by Morgan Stanley for 2015-17 for each metal for the first three years of production, and use the long term median future price forecasts beginning in year 2018 for year four through the end of mine life.

Neil Prenn, PE, has reviewed the above information, and as author and Qualified Person of this section, has reviewed the analyses and confirmed that the results support the assumptions used in the economic analysis in this Technical Report.

19.2 Contracts

SSMC has two drilling contracts in place for surface and underground exploration. Negotiations are underway with contractors, including the contractor demobilized in September 2012, for development and mining in the upper portion of the Sunshine Mine and rehabilitation of the Silver Summit Shaft. SSMC has not entered into any contract negotiations for smelting, refining, transportation, handling, sales or hedging, and forward sales. As FS activities recommended in Item 26.0 commence, contracts will be put in place for the indicated services.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL/COMMUNITY IMPACT

This section of the PEA summarizes the environmental studies that have occurred and are planned for the future, the project permitting requirements and status, and potential social or community issues associated with reopening the Sunshine Mine.

The Sunshine Mine has been in operation intermittently between 1921 and 2001, with numerous changes of ownership and operating rates. The current property holdings consist of 9,637 acres, including surface and mineral properties owned in fee simple, and patented and unpatented mining claims. The primary components of the mine around which environmental monitoring and permitting are associated include: the Sunshine Mine and Mill area; the Sunshine WRSF, adjacent laydown yard and storage buildings; the Sunshine TSF (referred to as Tailing Pond #2; and the adjacent Consolidated Silver Mine (referred to as Silver Summit and ConSil) and the ConSil mill, waste rock, and tailings areas.

The Sunshine Mine and related facilities are located in the Big Creek watershed. Big Creek flows into the South Fork. The ConSil facilities are located in the Rosebud Creek watershed. Rosebud Creek flows toward the South Fork, however surface water flow in the creek appears to infiltrate below the mine facilities before reaching the South Fork; no surface discharge to the South Fork has been identified or observed. The South Fork is included on the State of Idaho's Clean Water Act (CWA) Section 303(d) list as "impaired" (exceeding water quality standards) for suspended sediments, cadmium, lead, and zinc. This is due to historic mine operations located in the Silver Valley that discharged mine wastes and tailings into the South Fork and tributaries before environmental protection laws were enacted.

The Sunshine and ConSil Mines are located within the upper basin of the South Fork, idenfitied as part of Operable Unit 3 (OU3) of the Bunker Hill Mining and Metallurgical Complex Superfund Site, originally listed in 1983 under the Comprehensive Response, Compensation, and Liability Act (CERCLA). OU3 is also referred to as the Coeur d'Alene Basin, or "the Basin" and is outside of the original site boundary centered around the Bunker Hill site. EPA Region 10 has conducted numerous investigations and studies for the CERCLA site and in July 2010, issued a Proposed Plan for remediation of certain parts of the Basin, specifically the Upper Basin, which includes areas along the South Fork above its confluence with the North Fork of the Coeur d'Alene River. In August 2012, the EPA finalized a cleanup plan for the Upper Basin by issuing an Interim Record of Decision Amendment.

20.1 Environmental Baseline

Environmental baseline studies and monitoring data related to the Sunshine and ConSil Mine areas are available from several sources, including:

- monitoring and studies associated with past mine operations;
- monitoring and studies performed as required under permits for past mine operations; and
- monitoring and studies used for the Superfund documents prepared by the EPA and the Idaho Department of Environmental Quality (IDEQ)

This existing data provides some information to characterize the environmental baseline. However, upon acquiring the Sunshine Mine property in 2010, SSMC determined that more comprehensive data would be beneficial to implement the detailed mine and facility development plans and permits for reopening the Sunshine Mine. In 2012, SSMC initiated such environmental studies, to provide the following information:

- surface water quality and hydrology;
- groundwater hydrogeology; and
- waste rock geochemical characteristics.

The subsections below summarize the current environmental monitoring program.

20.1.1 Surface Water

A surface water sampling program was initiated in March 2012 to evaluate the water quality around the Sunshine Mine and ConSil Mine facilities. Sample locations were selected based on the location of existing and proposed facilities associated with the Sunshine Mine. Two rounds of sampling were conducted at 11 locations in Big Creek, the South Fork, and Rosebud Creek. In addition, a seep discovered between the Sunshine TSF and Big Creek was sampled. Samples were analyzed for field parameters, major ions, and a full suite of metals.

20.1.2 Hydrology

Measurements of surface water flow have been collected in Big Creek, the South Fork, and in Rosebud Creek at the same locations identified for water quality analysis. The USGS maintains several stream flow stations on the South Fork that provide historical and real time data that can also be used to characterize hydrology around the site.

20.1.3 Hydrogeology

Hydrogeology baseline studies included the collection of groundwater from the alluvial (shallow) groundwater system and collection of water exiting the mine portals and seeps. Groundwater was sampled at 17 locations around the Sunshine WRSF and mill area and the ConSil WRSF. This preliminary groundwater characterization effort was conducted at temporary monitoring locations established through direct-push and air-rotary drilling methods. The establishment of a permanent network of groundwater monitoring wells is currently being considered, based on the results of these initial investigations.

Samples were also collected from five mine-related discharge points and surface seeps, including: Old Sunshine Mine Portal, Silver Summit Mine Portal, Lower Crane Portal, Gullickson Seep, and Mill Seep.

Piezometers in place at the Sunshine TSF have been evaluated to determine if they could be redeveloped to provide information on water levels.

20.1.4 Wetlands

Sunshine has no plans to construct project facilities in wetland areas therefore no wetlands surveys are planned at this time.

20.1.5 Meteorology and Air Quality

There are no plans at this time to install a meteorological monitoring station. A final determination as to whether baseline meteorological or ambient air quality data is required will be based on the outcome of an Idaho air quality regulatory review, and if existing nearby meteorological data is available. The Sunshine Mine is in an area of Idaho which is in attainment of the national ambient air quality standards.

20.1.6 Rock and Mine Waste Characterization

Sunshine plans to place waste rock (host rock mined during development work or rock of insufficient grade to warrant processing) on the WRSF, and is considering storage areas adjacent to the current WRSF. There is limited geochemical data available for the WRSF; therefore, a characterization program was conducted in order to evaluate the potential for acid generation and metal leaching from waste rock and to evaluate geotechnical properties. Samples representing the range of rock types in the Sunshine and ConSil WRSFs were collected and subjected to the following test procedures:

- Mineralogical and elemental analyses;
- Static geochemical testing and acid-base accounting using standard geochemical techniques (sulfur analysis, paste pH, acid neutralization potential, and acid generating potential) to determine acid rock drainage potential; and
- Meteoric Water Mobility Procedure (MWMP) analyses to evaluate metals leaching potential.

The results of the waste rock characterization program indicate that the waste rock is characterized as non-acid generating under several different evaluation criteria.

Tailings will be disposed in the existing Sunshine TSF. Characterization of the tailings is not planned since processing methods are expected to be substantially similar to the methods most recently used (flotation) and therefore, the chemistry of water coming into contact with the tailings is expected to be similar to the current discharges from the TSF. The discharge chemistry has been monitored for years as a requirement of the current NPDES permit.

The TSF was constructed in the 1980s and currently covers approximately 33 acres. Additional dam lifts have been approved to provide capacity for tailings from the project. A TSF expansion evaluation was completed to evaluate the applicability of the existing tailings facility design to current permitting standards, evaluate the geotechnical stability of the facility, and develop capacity of the existing facility and planned lifts to accommodate tailings from the reopened Sunshine Mine.

20.2 Material Environmental Issues

There are no environmental issues that are anticipated to materially impact the ability to reopen the Sunshine Mine. This conclusion is based on a review of the studies completed to date and planned for the immediate future and review of the permits and approvals needed for the project and associated regulatory requirements.

20.3 Permitting

Numerous federal and state permits, plans, and approvals will be required for this project. The permits are described in the Permit Handbook for the Sunshine Mine, produced by Tetra Tech and summarized in Table 20.1.

In certain situations, issuance of a Federal permit requires compliance with the National Environmental Policy Act (NEPA) and development of an Environmental Impact Statement (EIS) or Environmental Assessment (EA). Based on the current proposed operating plan, reopening of the Sunshine Mine will not require development of an EIS or an EA. The NPDES permit will be a reissuance of an existing permit. Clean Water Act Section 404 actions, if any, would most likely be authorized under a nationwide permit, which offers a streamlined permitting process for specific categories of activities that does not require the development of an EIS or EA.

Table 20.1 Sunshine Mine Activities and Permits

Activity	Permit, Approval, Certification Requirement	Responsible Agency
Building Demolition	Asbestos Removal	USEPA NESHAP
	Institutional Controls Permit	Panhandle Health District
	Site Disturbance permit	Shoshone County Planning and Zoning
	Contaminated soil investigations and cleanup	Idaho Department of Environmental Quality (IDEQ)
Storm water runoff that discharges to waters of the U.S. during construction and operations	EPA Multi-sector general permit (2008 MSGP) and SWPPP	USEPA Region 10
Point source discharges of wastewater to waters of the U.S.	NPDES	USEPA Region 10
Point source discharges of wastewater to waters of the U.S.	State CWA 401 certification	IDEQ
Building construction	Building and Site Disturbance Permit	Shoshone County Planning and Zoning Department
Tailings Impoundment modifications, if beyond current design capacity	Form 1721	Idaho Department of Water Resources (IDWR)
Tailings dam modifications; if beyond current design capacity	Form 1710	IDWR
	CWA 404 permit for dredge and fill if in waters of the U.S.	USACE
	401 certification of the 404 permit, if necessary	IDEQ
Tailings Dam Operation	Idaho Dam Emergency Action Plan	IDWR
Petroleum storage	SPCC	USEPA Region 10
Facility construction and operation	Air Quality Permit	IDEQ
Groundwater Protection	Point of Compliance	IDEQ
Stream Channel alterations associated with construction activities	Joint Stream Channel Alteration Permit	IDWR
	CWA 404 permit	USACE

Activity	Permit, Approval, Certification Requirement	Responsible Agency
	CWA 401 certification of 404 permit	IDEQ
Metal contaminated soils removal	ICP permit	Panhandle Health District
Waste rock facility expansion, if in waters of the U.S.	CWA 404 permit	USACE
	CWA 401 certification of the 404 permit	IDEQ
Repair or maintenance of outfalls, if in waters of the U.S.	CWA 404 permit	USACE
	CWA 401 certification	IDEQ

20.4 Status of Permit Applications

There have been no permit applications submitted for project construction at this time. SSMC is collecting data, consolidating information to be used for the permit applications, and has prepared a schedule for integrated permit application and development.

20.4.1 NPDES Permit Renewal

SSMC currently holds an NPDES permit that was issued by EPA Region 10 on August 8, 1991. The permit term expired on September 9, 1996, but has been administratively extended and is in effect until EPA reissues the permit. The NPDES permit authorizes the discharge of wastewater from the Sunshine TSF to the South Fork through Outfall 001. The permit also authorized discharge from the silver/antimony refinery to Big Creek (Outfall 002) and discharge from the Price Tunnel to Big Creek (Outfall 003). Outfalls 002 and 003 are no longer discharging. Sunshine plans to submit an NPDES permit application to EPA for reissuance of the Sunshine Mine NDPES permit for Outfall 001. The current NPDES permit includes effluent limits for metals, monitoring of effluent and the receiving water, and other requirements.

The permit application and issuance process for the NPDES permit renewal is expected to be relatively straightforward, although additional data will be necessary to prepare a complete application and sufficient time should be allowed to accommodate the required review by the EPA, which administers the NPDES permit process on behalf of the State of Idaho.

Storm water discharged from the Sunshine Mine activities is authorized under the Multi-sector General Permit (MSGP), issued by EPA. The substantive requirement of the MSGP is preparation and implementation of a SWPPP. The current SWPPP for the site will be modified to incorporate changes based on plans for demolition and construction.

SSMC also holds an NPDES permit for discharges from the ConSil adit and TSF to the South Fork. This permit may need to be reissued if proposed mine operations impact the ConSil area.

A certification pursuant to CWA Section 401 from the IDEQ is required to ensure that the discharges, as authorized under the NPDES permit, will comply with state water quality standards. The certification application and issuance process occurs concurrently with the NPDES permitting process.

20.4.2 CERCLA Issues

EPA has proposed cleanup actions that may affect some aspects of SSMC's plans for reopening the Sunshine Mine. Likewise it is important to ensure that proposed mine activities do not adversely affect the cleanup activities. According to EPA's Proposed Plan "USEPA intends to manage its Superfund responsibilities in the Upper Basin in a manner that will allow for responsible mining and mineral processing activities as well as exploration and development."

In 2001, the former owners of the Sunshine Mine (Sunshine Mining and Refining Company and Sunshine Precious Metals, Inc.) entered into a consent decree with the United States and Coeur d'Alene Tribe that settled Sunshine's CERCLA liability and federal natural resource damage claims. SSMC should continue to consult with EPA and IDEQ to ensure that reopening the Sunshine Mine will not impact cleanup activities and that cleanup activities will not affect proposed mine operations. Any contaminated soils or other materials from historic operations encountered during demolition or construction must be managed and disposed under the Institutional Controls Program (ICP) run by the Panhandle Health District.

20.5 Social or Community Related Requirements

The Sunshine Mine is located approximately 37 miles from Coeur d'Alene, Idaho in Shoshone County. The project occurs three miles north of the U.S. Forest Service, Coeur d'Alene National Forest boundary (Idaho Panhandle National Forests) exclusively on private lands.

The Sunshine Mine is located approximately four and one-half miles outside the city limits of Kellogg, Idaho. Kellogg had a population of approximately 2,000 in 2010, which is down from 6,000 in 1980, after the closure of the Bunker Hill Mine. The town has since developed a ski resort community at the base of the Silver Mountain Resort. It is expected that the 331 workers the mine would employ would live in Kellogg or in the surrounding communities of Pinehurst, Osburn, Silverton, or Wallace, or would commute from Coeur d'Alene. The local residents of Kellogg and the surrounding communities will be important stakeholders in the region and to the Project.

20.6 Mine Closure and Reclamation Costs

The State of Idaho does not require a reclamation and closure plan for underground mining operations. The State, through the Idaho Department of Water Resources (IDWR) does require financial assurance for abandonment and closure of TSFs. A conceptual abandonment plan for the Sunshine TSF is available and includes: dewatering the facility, capping the facility, and construction of a spillway to route precipitation around the dam to a detention pond. The cost to implement this closure plan is approximately $275,000 and that is the basis for the current financial assurance requirements for the mine based on State of Idaho regulations. These regulations require that a surety bond, or equivalent, must be on file with the IDWR during the active life of the TSF.

IDWR requires that updates to the surety bond be submitted when an additional, previously approved, stage of the TSF is constructed. Details of TSF closure should continue to be developed so that the surety bond can be updated for the additional tailings embankment lifts required for the Project.

The TSF closure plan will be included in a Reclamation and Closure Plan for the entire project, including the underground mine, WRSFs, mill facility, and wastewater management. Costs for reclamation and closure can be established based on the Reclamation and Closure Plan.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

21.1.1 Summary

The capital cost estimate prepared for the Sunshine Mine Project PEA assumes a brownfield silver project capable of processing a nominal 365,000 dry short tons per annum of ore.

The key objectives of the capital cost estimate are to:

- Support the preliminary economic evaluation of the project;
- Support the identification and assessment of the processes and facilities that will provide the most favorable return on investment; and
- Provide guidance and direction for project financing and execution.

The total estimated initial cost to design, procure, construct, and commission the facilities described in this Technical Report is $130.3 million. Table 21.1 summarizes the initial capital costs by major area.

This PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that these inferred resources will ever be upgraded or that this PEA will be realized. Mineral resources that are not mineral reserves have no demonstrated economic viability.

21.1.2 Exclusions and Clarifications

The estimate is expressed in third quarter 2012 United States dollars and the following items are not included in the capital estimate:

- Sunk costs that are expected to be incurred prior to completion of a positive Feasibility Study;
- All SSMC's taxes (excluding sales/use taxes), which are included in the financial analysis;
- Reclamation costs, which are included in the financial analysis;
- Working capital and sustaining capital, which are included in the financial analysis;
- Interest and financing costs;
- Escalation beyond third quarter 2012; and
- Risk due to political upheaval, government policy changes, labor disputes, permitting delays, weather delays, or any other force majeure occurrences.

Table 21.1 Summary of Initial Capital Costs

Description	Cost	Total
Mine	21,830,000	
Crushing / Ore Handling	4,463,700	
Concentrator	15,518,500	
Tailings	1,709,400	
Tailings Paste Backfill (Surface Facility)	2,488,300	
Reagents	554,400	
Utilities	4,714,300	
General & Infrastructure	5,244,600	
Total Contracted Directs		**56,523,100**
Process Facilities Contractor Indirects	1,924,600	
Construction Equipment	684,200	
Freight and Duties	801,300	
EPCM - Process Facilities	5,204,000	
Commissioning Support	255,000	
Third Party Testing Services	240,000	
Vendor Representatives	353,300	
Spare Parts and Initial Fills	2,600,300	
Total Contracted Indirects		**12,062,600**
Mining and Ancillary Equipment	13,164,000	
Preproduction Mine Development	17,768,200	
Mobile Equipment & Light Vehicles	770,000	
Other (equipment, furniture, software, etc)	150,000	
Temporary Facilities	200,000	
Medical, Security and Safety	25,000	
Total Owner Direct Cost		**32,077,200**
Client Management	2,377,700	
Preproduction Employment & Training	7,010,000	
Utilities/Site Overhead Expenses	2,660,000	
Insurance	2,025,000	
Outside Services (Legal & Accounting)	400,000	
Corporate Overhead (Travel and Expenses)	2,400,000	
Total Owner Indirect Cost		**16,872,700**
Subtotal Project Cost		117,535,600
Contingency	12,805,700	
Total Initial Capital Costs (USD)		**$ 130,341,300**

21.1.3 Estimating Methodology

The capital cost estimate addresses the proposed engineering, procurement, construction, and start-up of a process plant and its ancillary facilities. Samuel Engineering (SE), MDA, SSMC, and MTB developed the capital cost estimate and it is built up by area cost centers as defined by the project Work Breakdown Structure (WBS).

The estimate is based on the assumption that new equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages for lump

sum or unit rate contracts. Various sources for pricing were used, including budgetary quotations, in-house historical data, published databases, factors, and estimators' judgment.

21.1.4 Contingency

A contingency of $12.8 million has been included in the initial capital cost. This contingency is based on the level of definition that was used to prepare the estimate.

Contingency is an allowance to cover unforeseeable costs that may arise during the project execution, which reside within the scope-of-work but cannot be explicitly defined or described at the time of the estimate, due to lack of information. However, it does not cover scope changes or project exclusions. For the purposes of the financial analysis, it is assumed that the contingency will be spent.

21.1.5 Accuracy

The captial cost estimate included in this PEA has been developed to a level sufficient to assess/evaluate the project concept, various development options, and the potential overall project viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy in the range of minus 35 percent plus 35 percent. This is based on the level of contingency applied, the confidence level of SE, MDA, SSMC, and MTB on the estimate accuracy, and an assessment comparing the PEA estimate to standard accuracy levels on scoping study estimates. The Qualified Person for this section has reviewed and approved the capital cost estimates for inclusion in this Technical Report.

21.2 Operating Cost Estimate

21.2.1 Summary

Operating costs have been estimated according to the main project areas identified as mining, processing, refining, G&A, and mine reclamation and closure. Methodologies and further details are summarized in sections that follow. Table 21.2 shows the LOM operating cost summarized by area. Average annual costs and LOM costs per ton of ore were calculated using estimated mine life of 27.5 years and 9.65 million tons LOM throughput, respectively. These same factors were used in calculating area-specific costs in the tables that follow for each area of operation.

Total operating personnel are estimated to include 250 in mining, 55 in processing, and 26 in G&A, for a total initial workforce of 331 employees.

Table 21.2 LOM Operating Cost Summary – All Areas

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore
Mining	1,287,050,608	46,801,840	133.34
Processing	238,809,316	8,683,975	24.74
Refining	380,902,297	13,850,993	39.46
General & Administration	209,552,915	7,620,106	21.71
Mine Reclamation & Closure Cost	1,350,000	**	0.14
Total Operating Cost (USD)	**$ 2,117,665,135**	**$ 76,956,914**	**$ 219.39**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*

*** No Average Annual Cost indicated as cost is considered incurred after LOM in Year 28 and after.*

21.2.2 Mining

Mine operating costs are based on mining by SSMC beginning in the first year of production. Contractor mining will be used to develop all the Alimak ventilation raises and for all development and mining during the 1.25 years of preproduction mine development. Table 21.3 shows the LOM operating cost summary for mining.

Table 21.3 LOM Operating Costs Summary - Mining

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore
Mining	1,217,872,085	44,286,258	126.17
Power (Above Ground)	34,963,665	1,271,406	3.62
Paste Backfill Cement	34,214,858	1,244,177	3.54
Total Owner Mining Cost (USD)	**$ 1,287,050,608**	**$ 46,801,840**	**$ 133.34**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*

Mine operating costs were estimated using the production schedule tonnages and unit rates developed for each mining method. A 20% contingency allowance, or $203 million has been included in estimated mine operating costs as shown below in Table 21.4.

Table 21.4 LOM Operating Cost – Mining by Method

Year	Tons	Mechanical $/ton	Alimak $/ton	Total $/ton	Subtotal $ 000's	Contingency 20%	Total $ 000's
*-1 **	*13,650*		*153.34*	*153.34*	*2,093.1*	*418.6*	*2,511.7*
1	155,776		109.53	109.75	17,096.3	3,419.3	20,515.5
2	175,751		109.53	109.53	19,250.1	3,850.0	23,100.1
3	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
4	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
5	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
6	366,000	56.22	109.53	101.53	37,161.3	7,432.3	44,593.5
7	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
8	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
9	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
10	366,000	56.22	109.53	101.53	37,161.3	7,432.3	44,593.5
11	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
12	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
13	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
14	366,000	56.22	109.53	101.53	37,161.3	7,432.3	44,593.5
15	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
16	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
17	365,000	56.22	109.53	101.53	37,059.7	7,411.9	44,471.7
18	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
19	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
20	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
21	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
22	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
23	365,000	56.22	109.53	109.53	39,978.4	7,995.7	47,974.1
24	365,000	56.22	109.53	109.53	39,978.5	7,995.7	47,974.1
25	365,000	56.22	109.53	109.53	39,978.5	7,995.7	47,974.1
26	365,000	56.22	109.53	109.53	39,978.5	7,995.7	47,974.1
27	365,000	56.22	109.53	109.53	39,978.5	7,995.7	47,974.1
28	179,342	56.22	109.53	109.53	19,643.4	3,928.7	23,572.0
LOM	**9,638,870**	**$ 56.22**	**$ 109.53**	**$ 105.22**	**$ 1,014,893.4**	**$ 202,978.7**	**$ 1,217,872.1**

**Charged to Initial Capital - Mining by Contractor - Preproduction mining*

Note: Production from year -1 and year 1 processed during year 1

21.2.3 Processing

Table 21.5 provides a summary of LOM operating costs for processing.

Table 21.5 LOM Operating Cost Summary – Processing

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore
Supervision and Labor	134,327,524	4,884,637	13.92
Power	33,011,617	1,200,422	3.42
Crushing and Grinding Steel	15,154,456	551,071	1.57
Reagent Chemicals	31,467,214	1,144,262	3.26
Water Treatment Chemicals	615,175	22,370	0.06
Maintenance Supplies & Materials	18,806,590	683,876	1.95
Operations Supplies, Oil & Lube, Misc.	2,820,978	102,581	0.29
Mobile Equipment	2,605,763	94,755	0.27
Total Processing Cost (USD)	**$ 238,809,316**	**$ 8,683,975**	**$ 24.74**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*

SE completed detailed estimates for input to process operating costs. Methodologies for the major criteria are shown below:

- Labor – Burdened US labor costs for hourly and salaried personnel were applied to a detailed staffing schedule. Burdened salary and hourly rates were supplied by SSMC.
- Power – Annual average power consumption was based on the mechanical equipment outlined in the PEA capital expenditure estimate and the operating time of the different circuits. The annual costs were based on unit rates for two tiers of usage currently being charged to SSMC.
- Process Consumables - Consumables were estimated on an order-of-magnitude anticipated annual consumption rate and unit cost rates per ton were applied to each consumable commodity amount.

21.2.4 Refining

Table 21.6 provides a summary of LOM operating costs for refining. Beginning in year two, a unit rate of $1.94 per ounce of silver refined (including processing cost for Cu cathode) was applied to each ounce of silver estimated to be produced to determine total refining cost. This unit rate was developed by SE based on estimated labor, consumables, and associated costs. Dividing this total by the years of mine life and tons of ore to be processed returns the average annual cost and cost per ton shown.

Table 21.6 LOM Operating Cost Summary – Refining

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore	LOM Cost per Oz Silver
Total Refining Cost (USD)	**$ 380,902,297**	**$ 13,850,993**	**$ 39.46**	**$ 1.91**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*
LOM Ag Con Ag Oz: 199,846,678

21.2.5 General and Administration

The G&A operating costs are described below and are shown in Table 21.7.

Table 21.7 LOM Operating Cost Summary – G&A

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore
Supervision and Labor	71,800,740	2,610,936	7.44
Insurance	75,877,175	2,759,170	7.86
Corporate Overhead	19,800,000	720,000	2.05
Corporate Outside Services (Legal & Accounting)	5,500,000	200,000	0.57
Site Overhead (Communications, Power, Supplies)	36,575,000	1,330,000	3.79
Total General & Administration Cost (USD)	**$ 209,552,915**	**$ 7,620,106**	**$ 21.71**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*

A staffing organization chart was developed to identify all G&A staff and labor positions. Burdened labor rates were provided by SSMC.

G&A expenses were estimated using historical data from recent projects.

21.2.6 Mine Reclamation and Closure Cost

Table 21.8 provides a summary of LOM operating costs for estimated mine reclamation and closure costs. No average annual cost is indicated as cost is considered incurred after LOM in year 28 and after. Any costs associated with concurrent reclamation during LOM operations are included in operating costs.

Table 21.8 LOM Operating Costs – Mine Reclamation and Closure

Description	Total Life of Mine Cost	Average Annual Cost	LOM Cost per Ton Ore
Total Mine Reclamation & Closure Cost (USD)	**$ 1,350,000**	******	**$ 0.14**

LOM = 27.5 years *LOM Tons of Ore: 9,652,520*

*** No Average Annual Cost indicated as cost is considered incurred after LOM in Year 28 and after.*

21.2.7 Net Cash Costs

Net cash costs were calculated using LOM totals for the following line items from the Sunshine economic model's cash flow forecast:

- Operating costs
- Freight, treatment and refining charges, and penalties
- Royalties
- Copper revenue (credit)
- Lead revenue (credit)

Total costs net of credits for secondary products were then divided by the estimated volume of payable silver ounces over the LOM. Table 21.9 shows the result of the calculation of LOM net cash costs to be $11.80 per payable ounce of silver.

Annual net cash costs were also estimated by summing the yearly values of the above cost and revenue items before dividing by the payable silver ounces in the corresponding year. Figure 21.1 is a graphical representation of the annual net cash costs per payable ounce of silver for each year in the life of the mine.

Table 21.9 LOM Net Cash Costs Including Copper and Lead Credits

Description	LOM Amount	Cumulative Amount
Operating Costs	$2,117,665,135	
Freight, Treatment/Refining Costs, Penalties	253,484,217	
Royalties	421,420,284	
Subtotal Cash Cost w/o Credits		2,792,569,636
Copper Revenue (Credit)	(73,778,831)	
Lead Revenue (Credit)	(17,292,590)	
Subtotal Cash Cost w/ Credits		$2,701,498,215
Total Payable Silver	*228,878,370*	*Ounces*
LOM Net Cash Cost	**$11.80**	**per Payable Ounce of Silver**

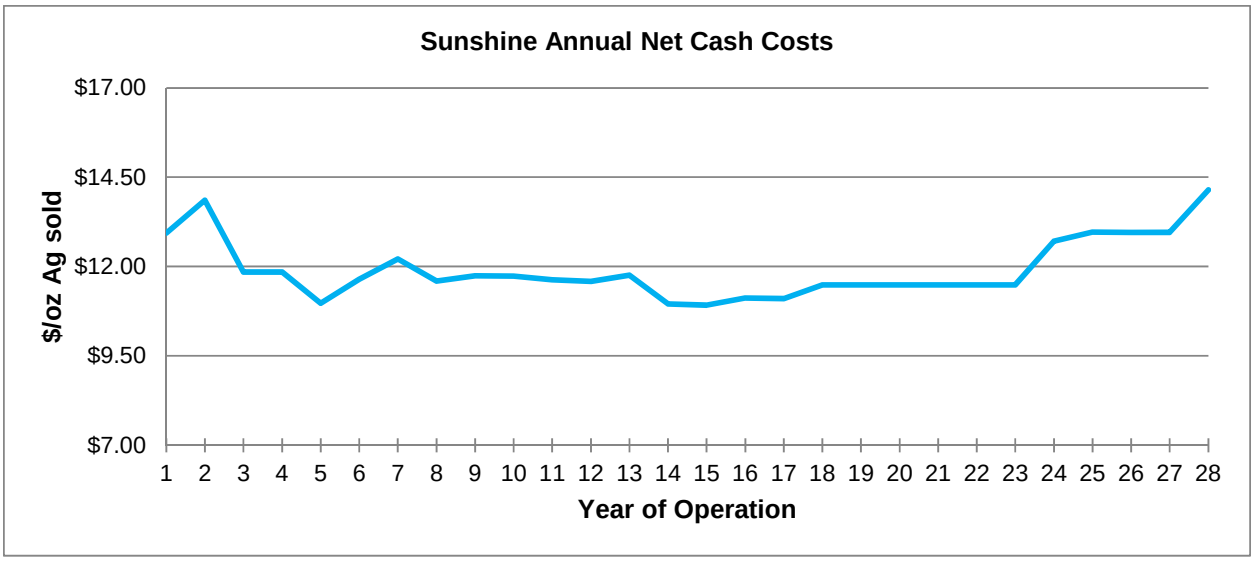

Figure 21.1 Annual Net Cash Costs Including Copper and Lead Credits

22.0 ECONOMIC ANALYSIS

22.1 General Criteria

MTB, under the supervision of the Qualified Person for this section, has completed a scoping level preliminary economic evaluation of the Sunshine Mine Project in Idaho for SSMC. Key objectives of developing the preliminary economic model were to:

- Gather information from various professionals in related disciplines including mine development, engineering, and metallurgy, among others;
- Identify and balance components in the model to determine the most favorable return on investment;
- On a high level, simulate operation over the expected life of the project;
- Allow for assessment of the project's potential economic viability;
- Support management in the financial decision-making process; and
- Provide a foundation for the next phase of project advancement.

Methodology involved in developing the preliminary economic model is explained in the following sections and technical parameters are provided as applicable. Summations of key project data are presented in tables extracted from the model. A listing of select model inputs is given in Table 22.1.

This PEA is preliminary in nature, it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA results described herein will be realized. Mineral resources that are not mineral reserves have no demonstrated viability.

Table 22.1 Economic Model Inputs

Description	Values			
Construction Period	15 months			
Preproduction Period	1.25 years			
Mine Life (after Preproduction)	27.5 years			
LOM Ore (ton)	9,652,519			
LOM Silver Concentrate (tons)	131,997			
LOM Lead Concentrate (tons)	232,309			
LOM Silver Con Grade (% Cu)	10.10%			
LOM Silver Con Grade (% Pb)	8.69%			
LOM Lead Con Grade (% Cu)	0.61%			
LOM Lead Con Grade (% Pb)	4.60%			
Avg. Annual Process Production Silver (troy ozs)	8,391,830			
Avg. Annual Process Production Copper (lbs)	1,072,549			
Avg. Annual Process Production Lead (lbs)	1,611,990			
Market Price	***Year 1***	***Year 2***	***Year 3***	***Year 4 >***
Silver Price ($/Toz Ag)	*32.50*	*29.91*	*26.15*	*25.00*
Copper Price ($/lb Cu)	*3.40*	*3.10*	*2.75*	*2.75*
Lead Price ($/lb Pb)	*0.94*	*0.95*	*0.85*	*0.85*
Silver Price (Average LOM $/Toz Ag)	25.27			
Copper Price (Average LOM $/lb Cu)	2.77			
Lead Price (Average LOM $/lb Pb)	0.85			
Cost and Tax Criteria				
Estimate Basis	3rd Qtr 2012 USD			
Inflation/Currency Fluctuation	None			
Leverage	100% Equity			
Income Tax - Federal, Idaho state, and local *(effective tax rate including depletion)*	27.7%			
Depreciation	Straight Line			
Royalties				
Coeur d'Alene Tribe	7.0%			
Chester Mining Co*	4.0%			
Hecla Mining Co*	4.0%			
Metropolitan Mines Corp* **on select revenue*	4.0%			
Transportation Charges				
Truck/Rail Freight - Silver Con to Horne ($/wst)	181.40			
Road Freight - Lead Con (to Trail, British Columbia) ($/wst)	53.70			
Freight and Insurance - Silver Dore to Utah ($/Toz)	0.02			
Road Freight - Copper Cathode to Missouri ($/lb)	0.14			
Payment Terms				
Provisional upon Bill of Lading	90%			
Settlement within 3 months after Bill of Lading	10%			

22.2 Production Summary

At the foundation of the economic model, data was drawn from the mine production and process production schedules, which were produced by MDA and SE, respectively and are summarized in Table 22.2. The Qualified Person for this section has reviewed and approved the production schedules for inclusion into the Technical Report.

Table 22.2 Process Production Schedule

Year	Total Ore Processed *tons*	Ag Con Produced *tons*	Pb Con Produced *tons*	Silver ozs *000's*	Copper lbs *000's*	Lead lbs *000's*
1	169,426	2,317	4,078	4,048	541	461
2	175,751	2,403	4,230	4,332	363	179
3	365,000	4,991	8,785	8,628	1,235	807
4	365,000	4,991	8,785	8,320	1,304	186
5	365,000	4,991	8,785	9,343	1,098	186
6	366,000	5,005	8,809	8,670	826	871
7	365,000	4,991	8,785	8,012	892	1,179
8	365,000	4,991	8,785	8,713	961	1,365
9	365,000	4,991	8,785	8,518	1,235	1,303
10	366,000	5,005	8,809	8,393	1,720	1,556
11	365,000	4,991	8,785	8,494	1,647	1,737
12	365,000	4,991	8,785	8,699	1,167	1,241
13	365,000	4,991	8,785	8,540	1,098	1,179
14	366,000	5,005	8,809	9,433	1,307	1,493
15	365,000	4,991	8,785	9,435	1,441	2,048
16	365,000	4,991	8,785	9,106	1,510	3,227
17	365,000	4,991	8,785	9,106	1,510	3,227
18	365,000	4,991	8,785	9,106	1,510	3,227
19	365,000	4,991	8,785	9,106	1,510	3,227
20	365,000	4,991	8,785	9,106	1,510	3,227
21	365,000	4,991	8,785	9,106	1,510	3,227
22	365,000	4,991	8,785	9,106	1,510	3,227
23	365,000	4,991	8,785	9,106	1,510	3,227
24	365,000	4,991	8,785	8,239	343	993
25	365,000	4,991	8,785	8,051	69	496
26	365,000	4,991	8,785	8,051	69	496
27	365,000	4,991	8,785	8,051	69	496
28	179,342	2,452	4,316	3,956	34	244
LOM	**9,652,519**	**131,997**	**232,309**	**230,775**	**29,495**	**44,330**

22.3 Gross Revenue from Mining

For purposes of the economic model, silver, copper, and lead market prices were drawn from consensus pricing commodity decks provided by Morgan Stanley. Consensus pricing is accomplished by periodically surveying major commodity forecasters and reporting their results

for yearly and long-term forecasts. Median values were then determined from all of the forecasts reported for each metal. Forecasts were last updated November 5-28, 2012. Table 22.3 shows the median consensus pricing values to be used in the model as market prices for silver, copper and lead.

Table 22.3 Median Consensus Prices for Silver, Copper, and Lead

	Year 1	Year 2	Year 3	Year 4 & after
Silver $/oz Ag	32.50	29.91	26.15	25.00
Copper $/lb Cu	3.40	3.10	2.75	2.75
Lead $/lb Pb	0.94	0.95	0.85	0.85

Year one market price for silver is applied to corresponding recovered silver ounces in silver-copper concentrate, payable at 90%, based upon preliminary smelter terms. Assuming the operation of the onsite refinery in the second year of production, the market prices for silver are applied to recovered silver ounces in silver concentrate from year two through the end of mine life. For recovered silver ounces in the lead concentrate, market prices are applied and payable at 95%, based upon preliminary smelter terms.

Market prices for copper are applied to all pounds of recovered copper in silver concentrate and payable at 100% for LOM. Because copper is considered as a penalty in lead concentrate, there is no calculation of revenue on those copper pounds recovered from lead concentrate.

The same is true for lead in the silver concentrate. Because it is considered a penalty, there is no calculation of revenue on lead pounds recovered from silver concentrate. As for the lead concentrate, market prices for lead are applied to all pounds of recovered lead, payable at 95% according to preliminary smelter terms.

22.4 Transportation

Transportation charges for trucking silver-copper concentrate to a rail head in Montana and then rail shipment to Xstrata's Horne Smelter in Quebec are based on current pricing in effect at a neighboring mine. Charges for transporting and insuring silver doré from the mine to the refiner in Salt Lake City, Utah were provided by the refiner.

Lead concentrate is expected to be trucked to Teck Metals Ltd.'s facility in Trail, B.C. for processing. A local trucking company provided a written quote for transportation charges based on the estimated quantities and characteristics of the lead concentrate.

Transportation charges for transporting copper cathodes to a buyer in Missouri were provided by a trucking company.

Table 22.4 summarizes the transportation types and costs.

Table 22.4 Transportation Types and Costs

Transportation Type	Purpose	Rate	Unit
Truck/Rail Freight	Silver Con to Horne	$181.40	Wst
Road Freight	Lead Con to Trail, BC	$53.70	Wst
Freight and Insurance	Silver Dore to Utah	$0.02	Toz
Road Freight	Copper Cathode to Missouri	$0.14	Lb

22.5 TCs, RCs, and Penalties

An independent marketing consultant conducted a marketing study for the project. Characteristics of the concentrates, as well as expected quantities that would be produced, were provided for use in obtaining early marketing information including payable metals, treatment charges, refining charges, penalty elements, and freight charges.

For the initial years of plant operation, both the silver-copper and lead concentrates are planned to be shipped to commercial refineries or smelters. Pricing was obtained from Teck Metals to process the lead concentrate and pricing for processing the silver-copper concentrate was received from the Horne Smelter. The terms and conditions received from these facilities form the basis for the terms used in the economic model and are shown in the Table 19.1 in Section 19.1. After the first year of plant operation, the silver-copper concentrate is planned to be processed at an onsite refinery.

22.6 Royalties

Many parts of the Sunshine Mine property are subject to royalties that are payable to parties from whom mineral rights are leased or to others who have a right to royalties on certain areas of the property. Certain of these agreements have royalty payments that are triggered when SSMC begins producing and selling metal-bearing concentrate. These royalties are based upon proceeds paid by smelters less certain costs, including costs incurred to transport the concentrates to the smelters, or NSR, for ore produced in the property area subject to the royalties.

Royalties have been previously described in greater detail under Item 4.2.

22.7 Operating Costs

Operating costs as previously described in Item 21.2 of this PEA served as input to the economic model.

22.8 Mine Development Costs

Mine development during preproduction will be performed by a contractor, as will construction of a bypass drift on the 3100 Level and all vertical development during production years. All other development during production will be performed by SSMC employees. LOM mine development is detailed in Table 16.5.

Estimated preproduction mine development costs ($17.8 million) are included within initial capital costs detailed in Table 21.1 and depreciated accordingly. A 5% contingency allowance has been included in estimated preproduction mine development costs. Estimated production mine development costs totaling $456.5 million for production years 1-26 are expensed in the

year in which the related ore is mined at an average rate of approximately $17 million per year. A 10% contingency allowance, or $40.6 million has been included in estimated mine development costs. The total estimated development costs are the product of the development quantities shown in Table 16.5 and the unit costs shown below in Table 22.5. The unit costs shown for contractor responsibilities are based on recent contractor work at site.

Table 22.5 Mine Development Unit Costs

Development Type	Owner - Cost ($/ft)	Contractor - Cost ($/ft)
12 x 12 Ramp	1,030	1,440
10 x 10 Drift	860	1,200
8 x 10 Attack Ramp	740	1,030
10 x 10 Alimak Vent Raise		1,360

22.9 Depreciation

In calculating depreciation, all initial and sustaining capital costs were assigned asset classes as defined under the Internal Revenue Service (IRS) Modified Accelerated Cost Recovery System (MACRS) using the General Depreciation System (GDS). Asset types are itemized in Table 22.6 according to asset class. Although MACRS defines nine classifications of property under GDS, with class lives ranging from three to 27.5 years, only those asset classes pertaining to the project are shown.

Table 22.6 Depreciation

Property Description	Asset Class	Annual Depreciation
Vehicles, IT Equipment, Copiers	5 year	20%
Office Furniture & Fixtures	7 year	14%
All Other Mining Assets	7 year	14%
Roads and Fences	15 year	7%

22.10 Income Taxes

Income taxes are provided based on the US federal and Idaho state and local statutory rates after allowable deductions including percentage depletion equal to 15% of silver and copper sales and 22% of lead sales.

22.11 Initial Capital Costs

Initial capital cost estimates as previously described in Item 21.1 of this Technical Report served as input to the preliminary economic model.

Of the total, just over $2.8 million is identified as spare parts, consumables, and initial fills. Because the cost of these items is recaptured at the end of mine life in year 28, their value is represented as a separate line item in the cash flow after being deducted from initial capital costs.

The remaining capital costs are then apportioned 40% and 60% to preproduction years (-2) and (-1) respectively.

22.12 Sustaining Capital Costs

Acquiring additional assets, increasing facility capacities, or replacing assets are considered sustaining capital expenses over the life of the project. Such expenditures fall into four main categories for the Sunshine Mine Project: mining, surface facilities, surface equipment, and TSF. The largest is mining equipment estimated at approximately $124.1 million. Determining each piece of equipment's useful life upon acquisition allows for its replacement capital cost to be scheduled in the last year of its useful life. Additional mining equipment and primary and ancillary equipment will be needed as well as miscellaneous items (e.g., mine software, and shop equipment). Sustaining capital costs are estimated to total $215.4 million over the LOM and are summarized in Table 22.7.

Table 22.7 Sustaining Capital Cost Summary

Description	Life of Mine Cost	
Underground Shop - Excavation & Equipment	900,000	
Silver Summit Shaft Rehab & Muck Hoisting Upgrade	8,250,000	
Mine Refrigeration System	5,000,000	
Muck Transfer Infrastructure	3,550,000	
Jewell Shaft Ore Pass System	6,938,400	
Jewell Shaft Pump Column Replacement	1,000,000	
Jewell Shaft Headframe Upgrade/Repairs & Pocket Rehab	8,000,000	
UG Backfill Distribution	3,500,000	
Substations (UG Power)	1,920,000	
Mine Dewatering and Fans	1,990,000	
Explosive Storage	250,000	
General	6,600,000	
Mine Equipment and Communications	124,111,000	
Refuge Chamber	450,000	
Mine Spare Parts	2,558,100	
Total Mining		**175,017,500**
Silver & Copper Refinery	28,623,000	
Administration Building	2,019,500	
Warehouse Building	2,303,200	
Maintenance Shop Building	2,163,500	
Total Surface Facilities		**35,109,100**
Pickups	1,040,000	
966 Front-end Loader	1,170,000	
Skid Steer	245,000	
JLG Manlift	180,000	
Fork Lift	75,000	
Mini Hoe	150,000	
Total Surface Equipment		**2,860,000**
Stage 6 Raise - Placement/Engineering/Inspection	150,000	
Stage 7 Raise - Placement/Engineering/Inspection	150,000	
Dry Stack Tailings Expansion	1,500,000	
Ongoing 3rd Party Inspection	625,000	
Total Tailings Storage Facility		**2,425,000**
Total Sustaining Capital Costs		**$ 215,411,600**

22.13 Working Capital

Defined as the highest amount of funding needed during the initial operating period, working capital is used to cover expenses prior to the cumulative revenue exceeding the cumulative expenses, or the point at which the operation becomes self-sustaining in its cash flow. Considering production schedule ramp-up, revenue was calculated on a weekly basis.

Projected revenue receipt was based upon shipments every week, allowing for an initial lag of four weeks leading up to first shipment. Estimating four weeks production would ship in week five, weekly shipments would occur thereafter. Assuming receipt of 90% of funds one week

after issuance of the shipping bill of lading, the 10% balance of funds were considered received 12 weeks after shipping, allowing for delivery, assaying, and accounts payable functions.

Weekly expenditure rates were calculated from the operating and development costs estimated for year 1.

The largest deficit of funds is expected to occur in week 4, in the amount of $4.5 million. This working capital cost was recorded in the cash flow model in year 1, with recovery at the end of mine life in year 28.

22.14 Base Case Analysis

The results of this PEA estimate payback to occur late in the fourth year of mine life, approximately 3.9 years after start of production.

The base case financial model was developed from information described in this section. Based upon this information, the Sunshine Project is estimated to have an after-tax IRR of 28.6%. Assuming a discount rate of five percent over an estimated mine life of 27.5 years, the after-tax NPV is estimated to be approximately $732 million. Base-case NPVs at various discount rates are presented in Table 22.8.

Table 22.8 NPV at Various Discount Rates

Discount Rate	0%	2.5%	5%	7.5%
NPV ($M)	1,648.8	1,083.4	732.2	505.7

22.15 Sensitivity Analysis

Table 22.9 reflects the sensitivities for IRR and NPV in 5% increments of negative and positive deviation from the base case for silver and copper prices and operating and capital costs. Metallurgical recovery variances are shown in one percent increments and silver grade per ton of ore are shown in 2.5 ounces per ton of ore increments of negative and positive deviation from their respective base cases.

Graphical representations follow of the sensitivities of NPV and IRR to the incremental changes in silver and copper prices in Figures 22.1 and 22.2 and capital costs versus operating costs in Figures 22.3 and 22.4, respectively. In addition, NPV and IRR sensitivities to varying metallurgical recovery rates and silver grade per ton of ore are illustrated in Figures 22.5, 22.6, 22.7 and 22.8, respectively.

Table 22.9 Sensitivity Analysis of IRR and NPV

Base Case Variance	*-20%*	*-15%*	*-10%*	*-5%*	*Base*	*+5%*	*+10%*	*+15%*	*+20%*	*+25%*	*+30%*
$/Toz Silver (Ag) *	20.21	21.48	22.74	24.00	25.27	26.53	27.79	29.06	30.32	31.58	32.85
IRR	17.4%	20.3%	23.2%	25.9%	28.6%	31.2%	33.8%	36.3%	38.8%	41.3%	43.7%
NPV @ 5% ($M)	347.1	443.4	539.6	635.9	732.2	828.4	924.7	1,021.0	1,117.3	1,213.5	1,309.8
$/lb Copper (Cu)	2.21	2.35	2.49	2.63	2.77	2.90	3.04	3.18	3.32	3.46	3.60
IRR	28.5%	28.5%	28.5%	28.6%	28.6%	28.6%	28.7%	28.7%	28.7%	28.8%	28.8%
NPV @ 5% ($M)	727.1	728.4	729.6	730.9	732.2	733.5	734.7	736.0	737.3	738.6	739.8
$/t Unit Operating Cost	175.51	186.48	197.45	208.42	219.39	230.36	241.33	252.30	263.27	274.24	285.21
LOM Operating Cost ($M)	1,694.1	1,800.0	1,905.9	2,011.8	2,117.7	2,223.5	2,329.4	2,435.3	2,541.2	2,647.1	2,753.0
IRR	32.7%	31.7%	30.6%	29.6%	28.6%	27.6%	26.5%	25.4%	24.4%	23.3%	22.2%
NPV @ 5% ($M)	884.4	846.3	808.3	770.2	732.2	694.1	656.1	618.0	580.0	541.9	503.9
Capital Cost ($M)	104.3	110.8	117.3	123.8	130.3	136.9	143.4	149.9	156.4	162.9	169.4
IRR	32.8%	31.6%	30.5%	29.5%	28.6%	27.7%	26.9%	26.1%	25.4%	24.7%	24.1%
NPV @ 5% ($M)	757.3	751.0	744.7	738.5	732.2	725.9	719.6	713.3	707.0	700.8	694.5
Base Case Variance	*- 4%*	*- 3%*	*- 2%*	*- 1%*	*Base*	*+ 1%*	*+ 2%*				
Silver Recovery	93.1%	94.1%	95.1%	96.0%	97.0%	98.0%	98.9%				
IRR	26.5%	27.0%	27.6%	28.1%	28.6%	29.1%	29.6%				
NPV @ 5% ($M)	657.1	675.9	694.7	713.4	732.2	750.9	769.7				
Copper Recovery	90.2%	91.2%	92.1%	93.1%	94.0%	94.9%	95.9%				
IRR	28.6%	28.6%	28.6%	28.6%	28.6%	28.6%	28.6%				
NPV @ 5% ($M)	731.2	731.5	731.7	731.9	732.2	732.4	732.7				
Lead Recovery	81.6%	82.5%	83.3%	84.2%	85.0%	85.9%	86.7%				
IRR	28.6%	28.6%	28.6%	28.6%	28.6%	28.6%	28.6%				
NPV @ 5% ($M)	732.0	732.0	732.1	732.1	732.2	732.2	732.3				
Base Case Variance			*- 5.0 opt*	*- 2.5 opt*	*Base*	*+ 2.5 opt*	*+ 5.0 opt*				
Silver Tozs/Ore T			19.63	22.13	24.63	27.13	29.63				
IRR			17.5%	23.2%	28.6%	33.8%	38.8%				
NPV @ 5% ($M)			351.3	541.7	732.2	922.6	1,113.1				

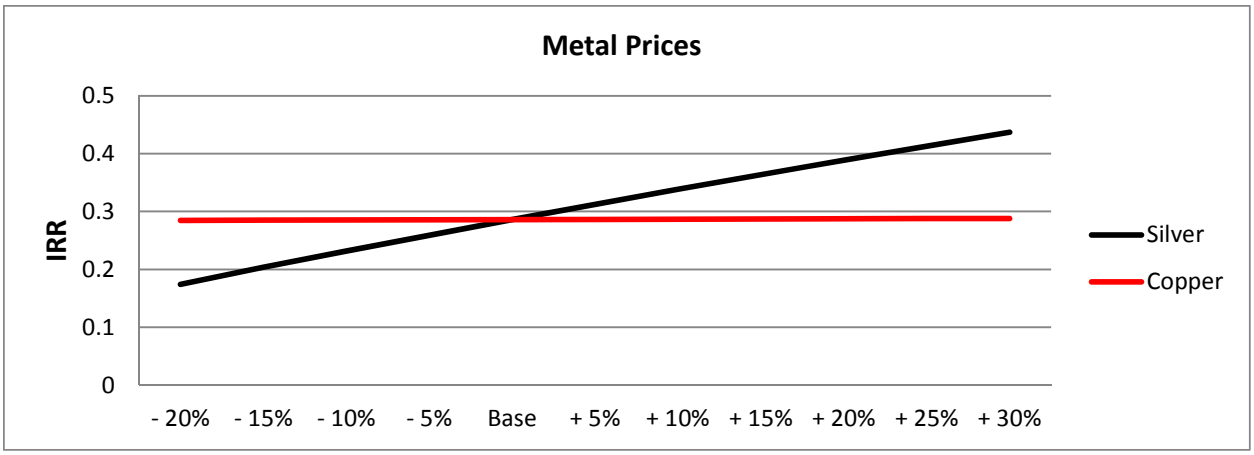

Figure 22.1 IRR Sensitivity Analysis – Metal Prices

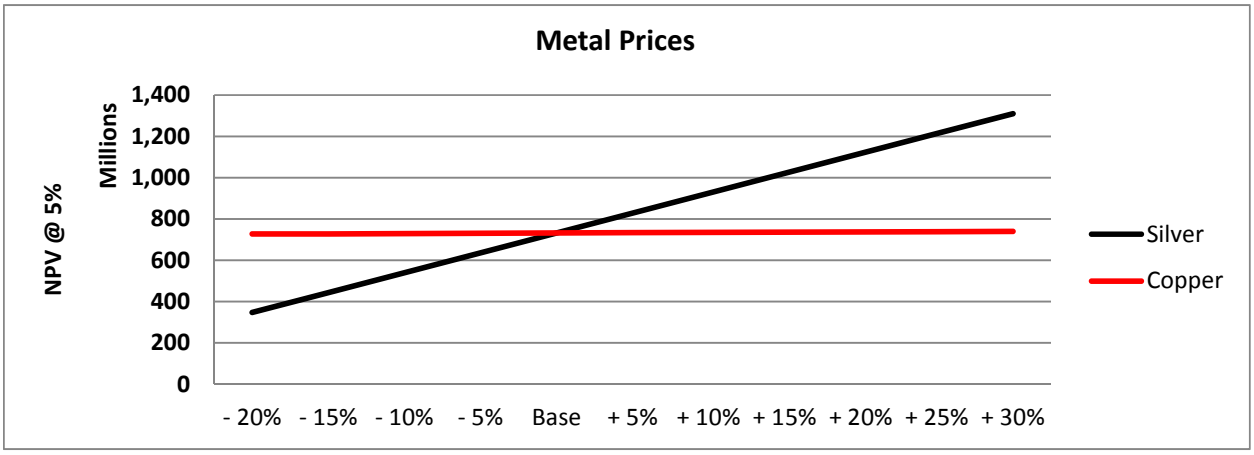

Figure 22.2 NPV Sensitivity Analysis – Metal Prices

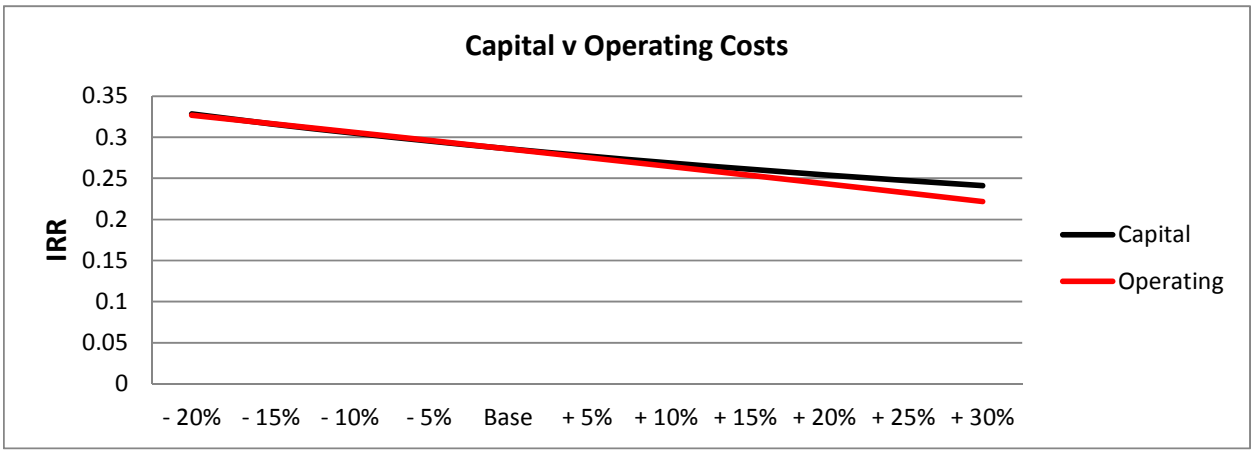

Figure 22.3 IRR Sensitivity Analysis – Capital v Operating Costs

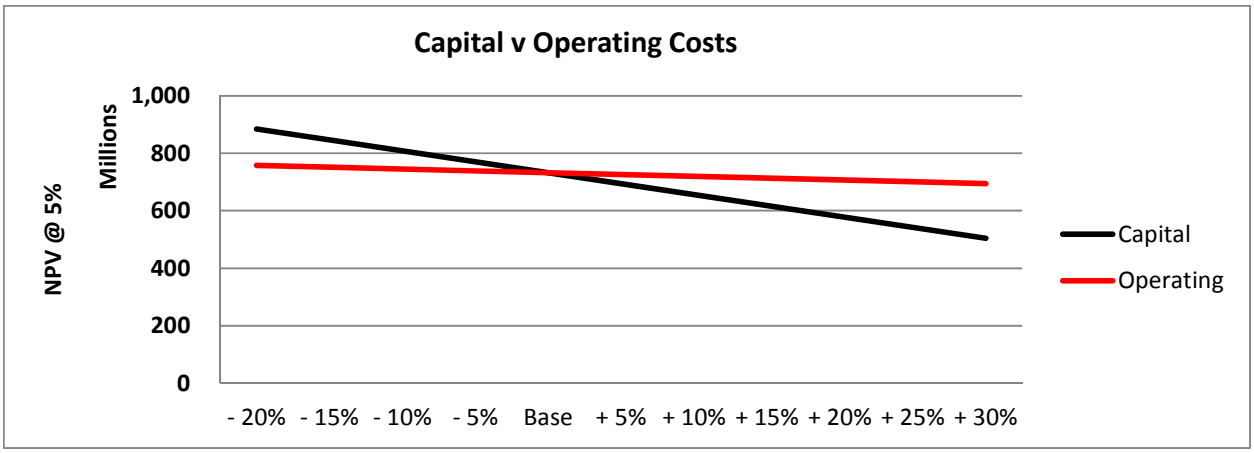

Figure 22.4 NPV Sensitivity Analysis – Capital v Operating Costs

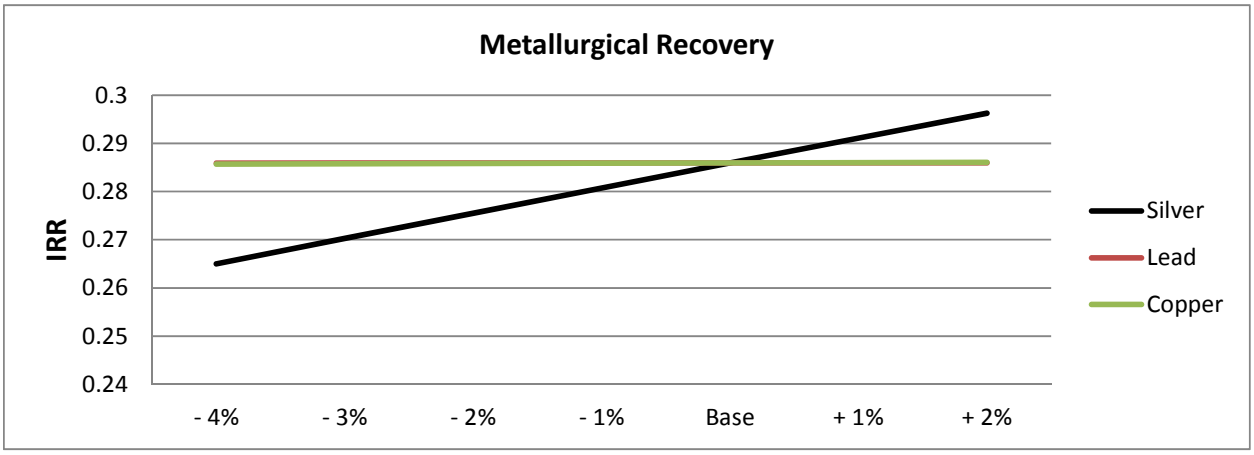

Figure 22.5 IRR Sensitivity Analysis – Recovery

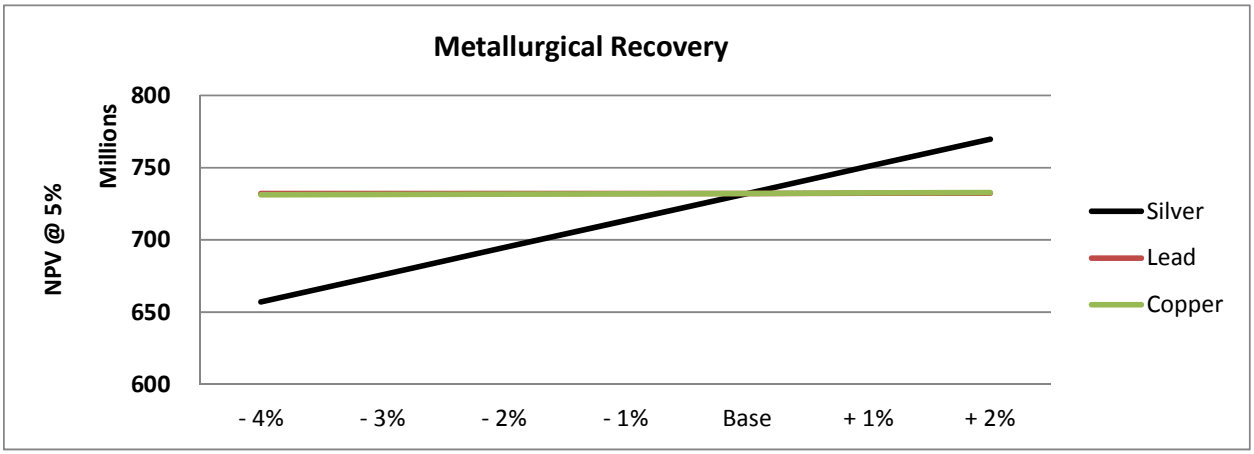

Figure 22.6 NPV Sensitivity Analysis – Recovery

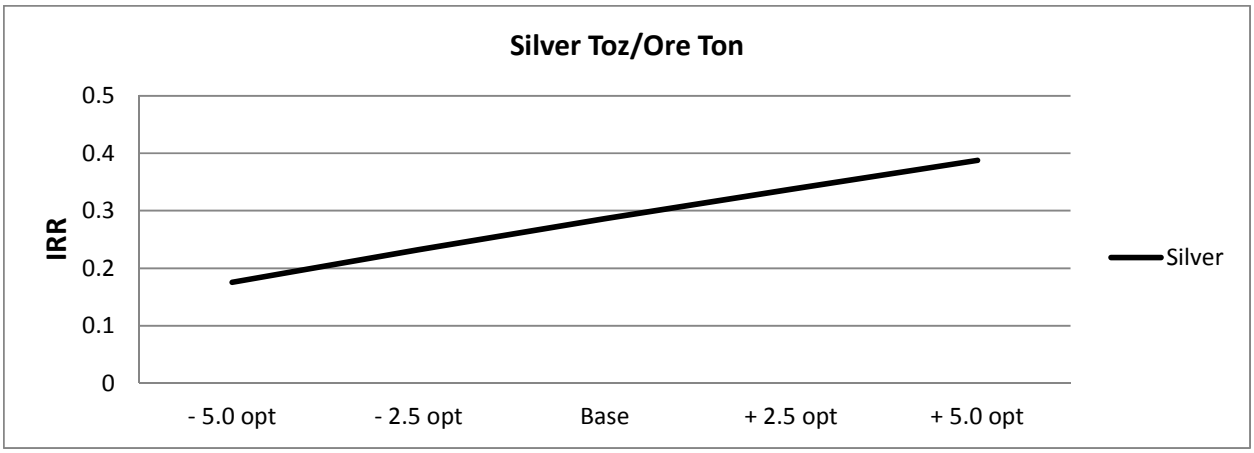

Figure 22.7 IRR Sensitivity Analysis – Silver Ounces per Ton of Ore

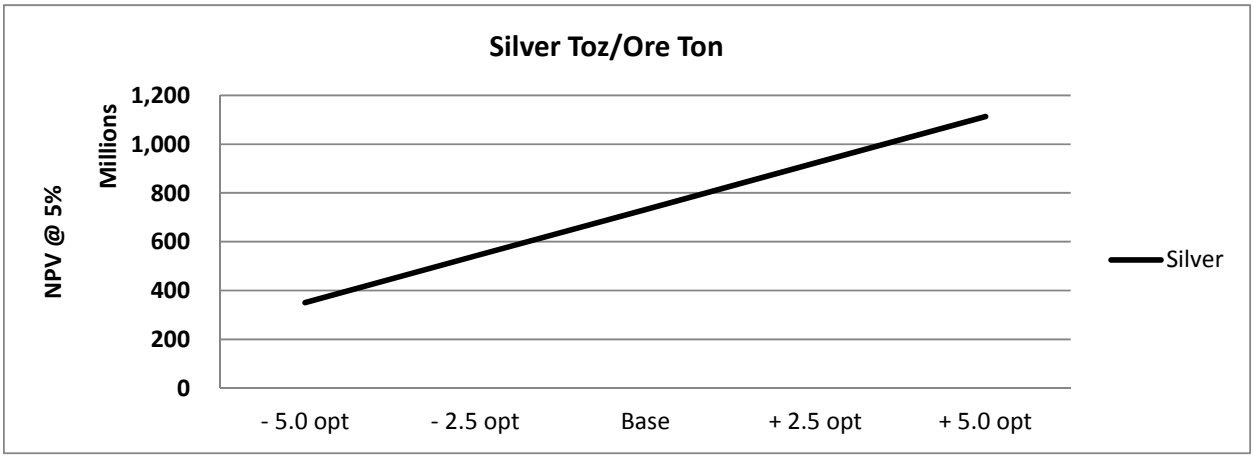

Figure 22.8 NPV Sensitivity Analysis – Silver Ounces per Ton of Ore

22.16 Financial Outputs

Multiple financial output results are grouped together and shown for varying silver prices in Table 22.10 below. The effective income tax rate varies from 23.1% ($20 per ounce silver price) to 31.6% ($40 per ounce silver price) reflecting percentage depletion equal to 15% of silver and copper sales and 22% of lead sales.

Table 22.10 Financial Outputs

Silver Price	Average Annual Free Cash Flow	NPV			IRR	Payback Period
USD	*(after tax)*	**0.0%**	**5.0%**	**7.5%**		*Production*
$/oz	*$M*	*$M*	*$M*	*$M*		*Year*
20.00	37.6	902.8	343.2	207.0	16.0%	6.6
22.50	50.6	1,259.9	522.7	342.0	21.0%	5.1
25.00	63.3	1,609.7	697.9	473.5	25.5%	4.4
* **25.27**	**64.7**	**1,648.8**	**732.2**	**505.7**	**28.6%**	**3.9**
27.50	75.8	1,954.4	870.1	602.6	29.8%	3.9
30.00	88.4	2,301.7	1,043.4	732.4	34.0%	3.5
32.50	100.9	2,644.5	1,214.2	860.3	37.9%	3.2
35.00	113.6	2,992.8	1,387.7	990.1	41.8%	3.0
37.50	126.1	3,337.0	1,559.1	1,118.4	45.5%	2.8
40.00	138.6	3,682.3	1,730.9	1,246.9	49.1%	2.6

** Base Case - Consensus Curve Price*

Cumulative free cash flow for varying silver prices is shown below in Figure 22.9.

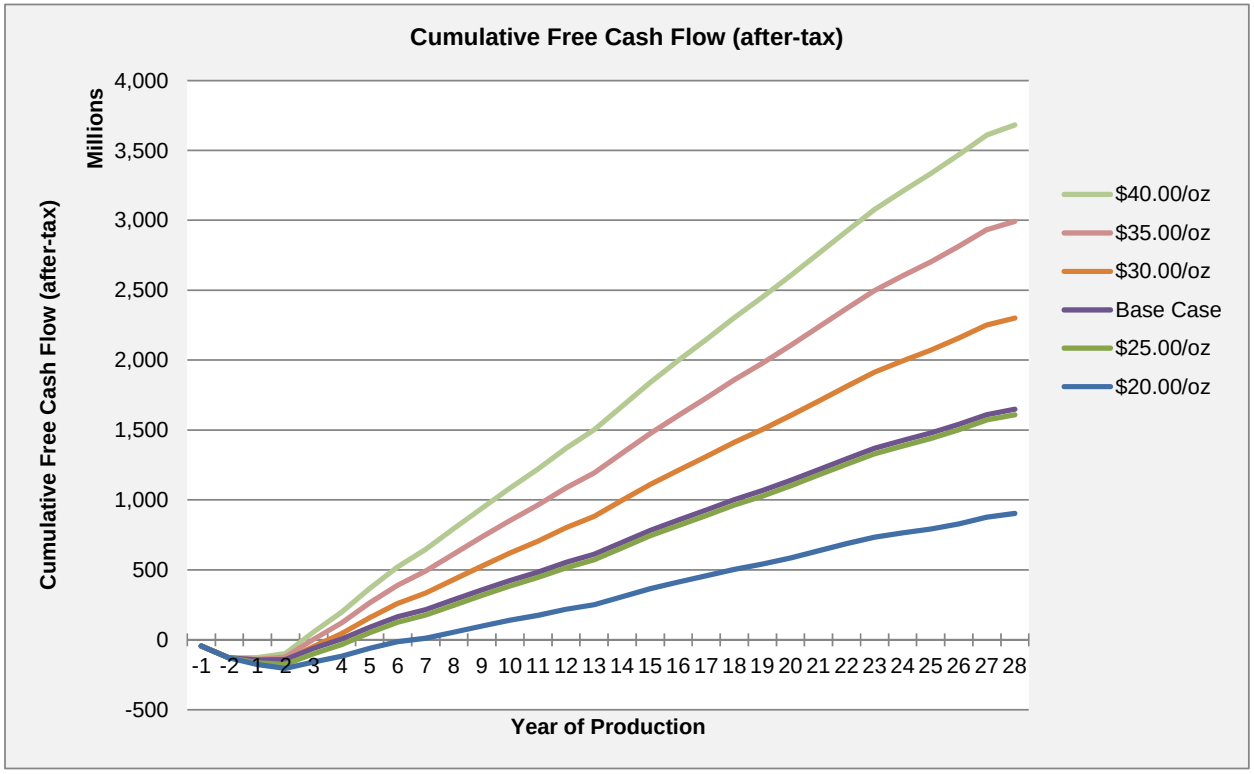

Figure 22.9 Cumulative Free Cash Flow

22.17 Economic Model

Figure 22.10 is a graphical representation of key components of the cash flow model. Immediately following is the complete cash flow model in Table 22.11.

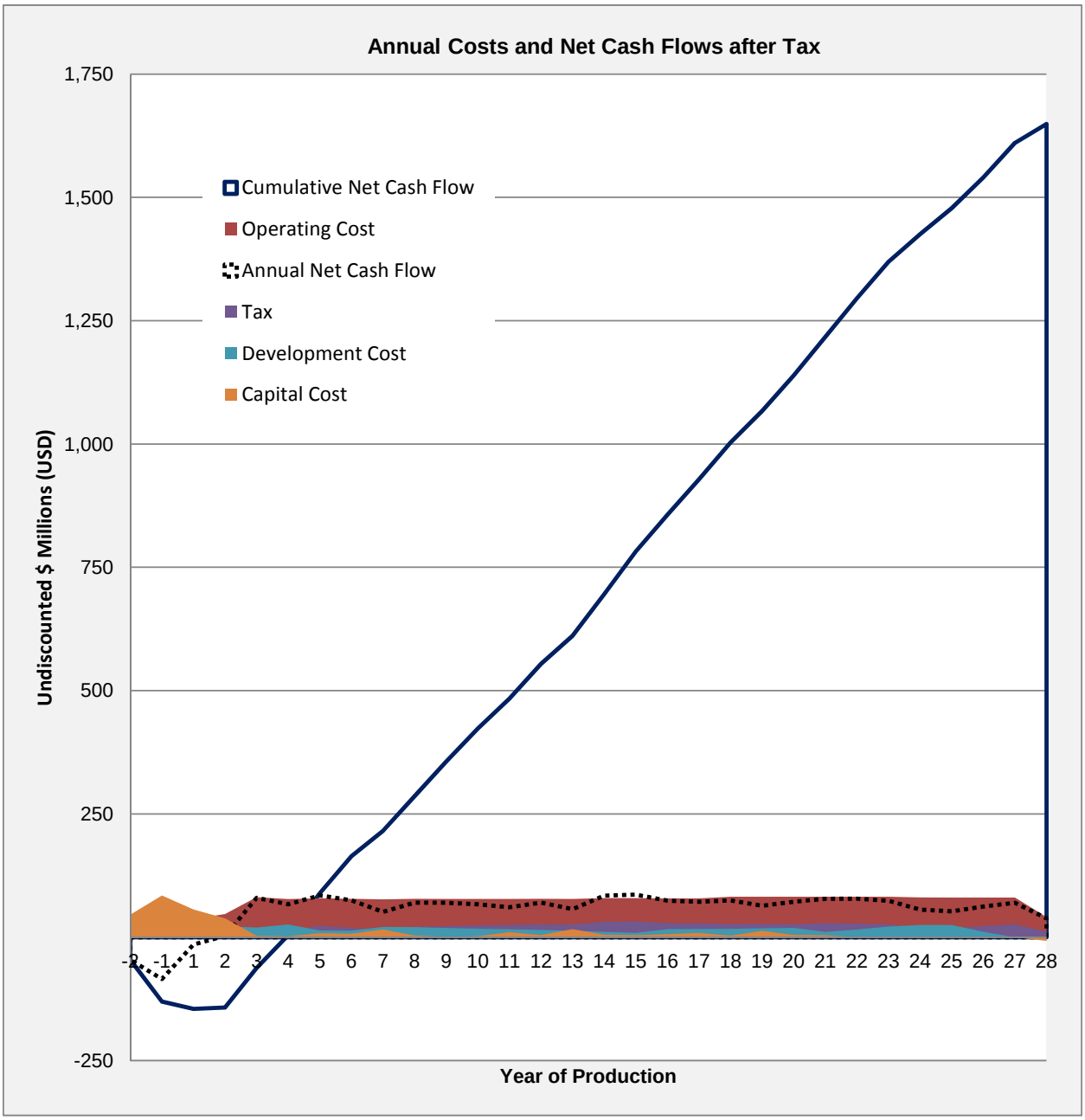

Figure 22.10 Annual Costs and Net Cash Flows after Tax

Table 22.11 Cash Flow Model

Cash Flow Forecast
Sunshine Silver Mines Corp
Sunshine Silver Mine

					Preproduction														
				Total LOM or Average	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
PRODUCTION SUMMARY																			
Ore Processed			***tons***	***9,652,519***			***169,426***	***175,751***	***365,000***	***365,000***	***365,000***	***366,000***	***365,000***	***365,000***	***365,000***	***366,000***	***365,000***	***365,000***	***365,000***
Silver Concentrate			**tons**	**131,997**			**2,317**	**2,403**	**4,991**	**4,991**	**4,991**	**5,005**	**4,991**	**4,991**	**4,991**	**5,005**	**4,991**	**4,991**	**4,991**
1.00	Recovered Ag to Ag Con		troy ozs	199,846,678			3,505,288	3,751,300	7,471,842	7,205,100	8,091,174	7,507,685	6,938,358	7,545,426	7,376,796	7,267,882	7,355,334	7,533,162	7,395,192
1.00	Recovered Cu to Ag Con		lbs	26,671,104			489,641	328,654	1,116,900	1,178,950	992,800	746,640	806,650	868,700	1,116,900	1,555,500	1,489,200	1,054,850	992,800
1.00	Recovered Pb to Ag Con		lbs	22,947,149			238,552	92,797	417,580	96,360	96,360	450,912	610,280	706,640	674,520	805,200	899,360	642,400	610,280
Lead Concentrate			**tons**	**232,309**			**4,078**	**4,230**	**8,785**	**8,785**	**8,785**	**8,809**	**8,785**	**8,785**	**8,785**	**8,809**	**8,785**	**8,785**	**8,785**
Recovered Ag to Pb Con			troy ozs	30,928,653			542,485	580,558	1,156,357	1,115,075	1,252,206	1,161,904	1,073,794	1,167,745	1,141,647	1,124,791	1,138,326	1,165,847	1,144,494
Recovered Cu to Pb Con			lbs	2,823,999			51,844	34,799	118,260	124,830	105,120	79,056	85,410	91,980	118,260	164,700	157,680	111,690	105,120
Recovered Pb to Pb Con			lbs	21,382,571			222,287	86,469	389,090	89,790	89,790	420,168	568,670	658,460	628,530	750,300	838,040	598,600	568,670
GROSS INCOME FROM MINING																			
Market Price																			
1.00	Silver		$/oz	25.27			32.50	29.91	26.15	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00
1.00	Copper		$/lb	2.77			3.40	3.10	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
	Lead		$/lb	0.79			0.94	0.95	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Payable Metals																			
Silver Concentrate																			
Silver			90%	$ 102,529,686			102,529,686												
Copper			100%	1,664,780			1,664,780												
Payable Metals																			
Silver Bullion			100%	4,935,527,493				112,182,616	195,388,668	180,127,500	202,279,350	187,692,120	173,458,950	188,635,650	184,419,900	181,697,040	183,883,350	188,329,050	184,879,800
Copper Cathode			100%	72,114,051				1,018,829	3,071,475	3,242,113	2,730,200	2,053,260	2,218,288	2,388,925	3,071,475	4,277,625	4,095,300	2,900,838	2,730,200
Lead Concentrate																			
Silver Payfor			95%	742,389,282			16,749,228	16,493,516	28,726,786	26,483,031	29,739,881	27,595,211	25,502,596	27,733,932	27,114,116	26,713,791	27,035,231	27,688,854	27,181,733
Lead Payfor			95%	17,292,590			197,446	78,039	314,190	72,505	72,505	339,286	459,201	531,706	507,538	605,867	676,717	483,370	459,201
Total Gross Revenue				**$ 5,871,517,881**			**121,141,140**	**129,772,999**	**227,501,120**	**209,925,149**	**234,821,936**	**217,679,876**	**201,639,034**	**219,290,213**	**215,113,029**	**213,294,323**	**215,690,598**	**219,402,111**	**215,250,934**
NSR CALCULATION																			
Total Freight, TC/RC, and Penalties				(253,484,217)			(5,644,832)	(4,654,920)	(9,491,101)	(9,299,979)	(9,937,749)	(9,474,508)	(9,045,881)	(9,510,319)	(9,419,794)	(9,411,878)	(9,457,673)	(9,528,108)	(9,415,602)
NSR				**$ 5,618,033,664**			**115,496,308**	**125,118,079**	**218,010,019**	**200,625,170**	**224,884,187**	**208,205,368**	**192,593,153**	**209,779,894**	**205,693,235**	**203,882,445**	**206,232,925**	**209,874,003**	**205,835,331**
ROYALTY																			
Total Royalties				(421,420,284)			(6,768,749)	(8,927,321)	(13,989,340)	(14,500,112)	(15,525,908)	(15,009,224)	(13,886,918)	(15,676,975)	(15,715,501)	(15,546,907)	(15,659,081)	(15,909,092)	(15,724,403)
Gross Income from Mining				**$ 5,196,613,380**			**108,727,559**	**116,190,758**	**204,020,679**	**186,125,058**	**209,358,278**	**193,196,144**	**178,706,235**	**194,102,920**	**189,977,734**	**188,335,539**	**190,573,844**	**193,964,911**	**190,110,928**
OPERATING MARGIN																			
Unit Operating Costs																			
Mining		$	/t ore	126.17			203.33	131.44	131.44	121.84	121.84	121.84	121.84	121.84	121.84	121.84	121.84	121.84	121.84
Power (Above Ground)		$	/t ore	3.62			12.60	7.23	3.48	3.48	3.48	3.47	3.48	3.48	3.48	3.47	3.48	3.48	3.48
Paste Backfill Cement		$	/t ore	3.54			3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57
Processing		$	/t ore	24.74			42.40	41.17	24.10	24.10	24.10	24.06	24.10	24.10	24.10	24.06	24.10	24.10	24.10
Refining (per payable ozs in Ag Con refined)		*$*	*/Toz Ag*	*1.94*			-	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*
Refining (Ag Con)		$	/t ore	39.46			-	41.41	39.71	38.30	43.01	39.79	36.88	40.10	39.21	38.52	39.09	40.04	39.31
General & Administration		$	/t ore	21.71			44.98	43.36	20.88	20.88	20.88	20.82	20.88	20.88	20.88	20.82	20.88	20.88	20.88
Mine Reclamation & Closure Cost		$	/t ore	0.14															
Total Unit Operating Cost per Ton Mined		**$**	**/t ore**	**$ 219.39**			**218.09**	**268.18**	**223.18**	**212.17**	**216.88**	**213.56**	**210.75**	**213.98**	**213.08**	**212.29**	**212.97**	**213.91**	**213.18**
Total Unit Operating Cost per Payable Ounce		**$**	**/Toz Ag**	**$ 9.25**			**10.07**	**10.95**	**9.50**	**9.37**	**8.53**	**9.08**	**9.67**	**9.02**	**9.19**	**9.32**	**9.21**	**9.04**	**9.17**
Total UOC with By-product Credit and Royalties per Payable Ounce		$	/Toz Ag	$ 10.70			11.40	14.25	12.16	12.18	11.17	11.81	12.44	11.80	12.04	12.17	12.04	11.85	12.02
Net Cash Cost per Payable Ounce		$	/Toz Ag	$ 11.80			12.94	13.86	11.85	11.85	10.97	11.64	12.21	11.60	11.74	11.73	11.63	11.59	11.76
Operating Costs																			
Mining				(1,217,872,085)			(20,515,504)	(23,100,061)	(47,974,140)	(44,471,673)	(44,471,673)	(44,593,513)	(44,471,673)	(44,471,673)	(44,471,673)	(44,593,513)	(44,471,673)	(44,471,673)	(44,471,673)
Power (Above Ground)				(34,963,665)			(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)
Paste Backfill Cement				(34,214,858)			(360,213)	(627,432)	(1,303,050)	(1,303,050)	(1,303,050)	(1,306,620)	(1,303,050)	(1,303,050)	(1,303,050)	(1,306,620)	(1,303,050)	(1,303,050)	(1,303,050)
Processing				(238,809,316)			(7,183,490)	(7,235,676)	(8,796,977)	(8,796,977)	(8,796,977)	(8,805,227)	(8,796,977)	(8,796,977)	(8,796,977)	(8,805,227)	(8,796,977)	(8,796,977)	(8,796,977)
Refining				(380,902,297)			-	(7,277,521)	(14,495,373)	(13,977,894)	(15,696,878)	(14,564,909)	(13,460,415)	(14,638,126)	(14,310,984)	(14,099,690)	(14,269,348)	(14,614,334)	(14,346,672)
General & Administration				(209,552,915)			(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)
Mine Reclamation & Closure Cost				(1,350,000)															
Total Operating Costs				**$ (2,117,665,135)**			**(36,950,720)**	**(47,132,203)**	**(81,461,052)**	**(77,441,106)**	**(79,160,090)**	**(78,161,781)**	**(76,923,627)**	**(78,101,338)**	**(77,774,196)**	**(77,696,563)**	**(77,732,560)**	**(78,077,546)**	**(77,809,884)**
Mine Development Costs				(456,544,797)			(21,708,739)	(22,284,319)	(19,959,041)	(25,958,866)	(13,733,762)	(13,545,502)	(20,909,745)	(20,653,267)	(18,722,895)	(17,382,675)	(16,388,549)	(15,073,231)	(13,038,839)
Net Profit Before Depreciation				**$ 2,622,403,448**			**50,068,100**	**46,774,237**	**102,600,586**	**82,725,086**	**116,464,426**	**101,488,861**	**80,872,864**	**95,348,315**	**93,480,643**	**93,256,301**	**96,452,735**	**100,814,134**	**99,262,205**
Depreciation				(345,752,917)			(19,464,892)	(26,874,466)	(32,367,409)	(32,819,480)	(33,114,980)	(31,238,412)	(32,244,983)	(18,175,433)	(11,298,288)	(5,892,631)	(5,744,846)	(6,964,060)	(6,400,488)
Net Profit Before Taxes				2,276,650,531			**30,603,208**	**19,899,771**	**70,233,177**	**49,905,606**	**83,349,446**	**70,250,449**	**48,627,881**	**77,172,881**	**82,182,354**	**87,363,670**	**90,707,890**	**93,850,074**	**92,861,716**
1.00	State and Federal Income Tax Factor			27.7%	(630,632,197)			(8,477,089)	(5,512,237)	(19,454,590)	(13,823,853)	(23,087,797)	(19,459,374)	(13,469,923)	(21,376,888)	(22,764,512)	(24,199,737)	(25,126,085)	(25,996,471)	(25,722,695)
Net Profit After Taxes				**$ 1,646,018,334**			**22,126,119**	**14,387,534**	**50,778,587**	**36,081,753**	**60,261,649**	**50,791,075**	**35,157,958**	**55,795,993**	**59,417,842**	**63,163,934**	**65,581,804**	**67,853,604**	**67,139,021**
Add-Back Depreciation				345,752,917			19,464,892	26,874,466	32,367,409	32,819,480	33,114,980	31,238,412	32,244,983	18,175,433	11,298,288	5,892,631	5,744,846	6,964,060	6,400,488
Capital																			
Total Capital Costs (less Spare Parts/Initial Fills)				(108,896,711)	(43,558,684)	(65,338,027)													
Preproduction Mine Development				(18,656,640)	(2,126,889)	(16,529,751)													
Spare Parts/Initial Fills				(2,787,949)	-	(2,787,949)													
1.00	**Total Initial Capital Costs**			**$ (130,341,300)**	**(45,685,573)**	**(84,655,727)**	-	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital				(4,512,266)			(4,512,266)												
Sustaining Capital				(215,411,615)			(51,835,015)	(38,450,600)	(3,132,500)	(2,036,500)	(8,465,000)	(7,158,000)	(15,868,000)	(3,695,000)	(691,000)	(2,130,000)	(10,459,000)	(4,520,000)	(16,460,000)
Recapture of Spare Parts/Initial Fills				2,787,949															
Working Capital Recapture				4,512,266															
ANNUAL CASH FLOW				**$ 1,648,806,285**	**(45,685,573)**	**(84,655,727)**	**(14,756,269)**	**2,811,400**	**80,013,496**	**66,864,733**	**84,911,630**	**74,871,486**	**51,534,941**	**70,276,426**	**70,025,131**	**66,926,565**	**60,867,650**	**70,297,664**	**57,079,509**
CUMULATIVE CASH FLOW					**(45,685,573)**	**(130,341,300)**	**(145,097,569)**	**(142,286,169)**	**(62,272,673)**	**4,592,060**	**89,503,690**	**164,375,176**	**215,910,117**	**286,186,543**	**356,211,674**	**423,138,238**	**484,005,888**	**554,303,552**	**611,383,061**

IRR	**28.6%**
Disc	**NPV**
0.0%	**$ 1,648,806,285**
2.5%	**$ 1,083,435,262**
5.0%	**$ 732,176,299**
7.5%	**$ 505,695,638**

Table 22.11 Cash Flow Model (continued)

Cash Flow Forecast
Sunshine Silver Mines Corp
Sunshine Silver Mine

			Total LOM or Average	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28
PRODUCTION SUMMARY																		
Ore Processed		***tons***	***9,652,519***	***366,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***365,000***	***179,342***
Silver Concentrate		**tons**	**131,997**	**5,005**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**4,991**	**2,452**
1.00 Recovered Ag to Ag Con		troy ozs	199,846,678	8,168,681	8,170,890	7,885,752	7,885,752	7,885,752	7,885,752	7,885,752	7,885,752	7,885,752	7,885,752	7,134,582	6,972,084	6,972,084	6,972,084	3,425,719
1.00 Recovered Cu to Ag Con		lbs	26,671,104	1,182,180	1,303,050	1,365,100	1,365,100	1,365,100	1,365,100	1,365,100	1,365,100	1,365,100	1,365,100	310,250	62,050	62,050	62,050	30,488
1.00 Recovered Pb to Ag Con		lbs	22,947,149	772,992	1,059,960	1,670,240	1,670,240	1,670,240	1,670,240	1,670,240	1,670,240	1,670,240	1,670,240	513,920	256,960	256,960	256,960	126,257
Lead Concentrate		**tons**	**232,309**	**8,809**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**8,785**	**4,316**
Recovered Ag to Pb Con		troy ozs	30,928,653	1,264,201	1,264,543	1,220,414	1,220,414	1,220,414	1,220,414	1,220,414	1,220,414	1,220,414	1,220,414	1,104,162	1,079,013	1,079,013	1,079,013	530,171
Recovered Cu to Pb Con		lbs	2,823,999	125,172	137,970	144,540	144,540	144,540	144,540	144,540	144,540	144,540	144,540	32,850	6,570	6,570	6,570	3,228
Recovered Pb to Pb Con		lbs	21,382,571	720,288	987,690	1,556,360	1,556,360	1,556,360	1,556,360	1,556,360	1,556,360	1,556,360	1,556,360	478,880	239,440	239,440	239,440	117,648
GROSS INCOME FROM MINING																		
Market Price																		
1.00 Silver		$/oz	25.27	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00
1.00 Copper		$/lb	2.77	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
Lead		$/lb	0.79	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Payable Metals																		
Silver Concentrate																		
Silver		90%	$ 102,529,686															
Copper		100%	1,664,780															
Payable Metals																		
Silver Bullion		100%	4,935,527,493	204,217,020	204,272,250	197,143,800	197,143,800	197,143,800	197,143,800	197,143,800	197,143,800	197,143,800	197,143,800	178,364,550	174,302,100	174,302,100	174,302,100	85,642,979
Copper Cathode		100%	72,114,051	3,250,995	3,583,388	3,754,025	3,754,025	3,754,025	3,754,025	3,754,025	3,754,025	3,754,025	3,754,025	853,188	170,638	170,638	170,638	83,842
Lead Concentrate																		
Silver Payfor		95%	742,389,282	30,024,764	30,032,884	28,984,833	28,984,833	28,984,833	28,984,833	28,984,833	28,984,833	28,984,833	28,984,833	26,223,836	25,626,559	25,626,559	25,626,559	12,591,557
Lead Payfor		95%	17,292,590	581,633	797,560	1,256,761	1,256,761	1,256,761	1,256,761	1,256,761	1,256,761	1,256,761	1,256,761	386,696	193,348	193,348	193,348	95,001
Total Gross Revenue			**$ 5,871,517,881**	**238,074,412**	**238,686,082**	**231,139,418**	**231,139,418**	**231,139,418**	**231,139,418**	**231,139,418**	**231,139,418**	**231,139,418**	**231,139,418**	**205,828,269**	**200,292,644**	**200,292,644**	**200,292,644**	**98,413,379**
NSR CALCULATION																		
Total Freight, TC/RC, and Penalties			(253,484,217)	(10,033,557)	(10,042,538)	(9,837,616)	(9,837,616)	(9,837,616)	(9,837,616)	(9,837,616)	(9,837,616)	(9,837,616)	(9,837,616)	(9,121,121)	(8,963,223)	(8,963,223)	(8,963,223)	(4,404,061)
NSR			**$ 5,618,033,664**	**228,040,854**	**228,643,543**	**221,301,803**	**221,301,803**	**221,301,803**	**221,301,803**	**221,301,803**	**221,301,803**	**221,301,803**	**221,301,803**	**196,707,148**	**191,329,421**	**191,329,421**	**191,329,421**	**94,009,318**
ROYALTY																		
Total Royalties			(421,420,284)	(16,989,159)	(17,389,957)	(16,964,114)	(16,845,314)	(16,845,314)	(16,845,314)	(16,845,314)	(16,845,314)	(16,845,314)	(16,845,314)	(15,358,228)	(14,627,948)	(14,539,354)	(14,539,354)	(10,255,444)
Gross Income from Mining			**$ 5,196,613,380**	**211,051,695**	**211,253,586**	**204,337,689**	**204,456,489**	**204,456,489**	**204,456,489**	**204,456,489**	**204,456,489**	**204,456,489**	**204,456,489**	**181,348,920**	**176,701,473**	**176,790,067**	**176,790,067**	**83,753,873**
OPERATING MARGIN																		
Unit Operating Costs																		
Mining	$	/t ore	126.17	121.84	121.84	121.84	121.84	131.44	131.44	131.44	131.44	131.44	131.44	131.44	131.44	131.44	131.44	131.44
Power (Above Ground)	$	/t ore	3.62	3.47	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.48	3.54
Paste Backfill Cement	$	/t ore	3.54	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57	3.57
Processing	$	/t ore	24.74	24.06	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.10	24.76
Refining (per payable ozs in Ag Con refined)	*$*	*/Toz Ag*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*	*1.94*
Refining (Ag Con)	$	/t ore	39.46	43.30	43.43	41.91	41.91	41.91	41.91	41.91	41.91	41.91	41.91	37.92	37.06	37.06	37.06	37.06
General & Administration	$	/t ore	21.71	20.82	20.88	20.88	20.88	20.88	20.88	20.88	20.88	20.88	20.88	20.88	20.88	20.88	20.88	21.24
Mine Reclamation & Closure Cost	$	/t ore	0.14															
Total Unit Operating Cost per Ton Mined	**$**	**/t ore**	**$ 219.39**	**217.06**	**217.30**	**215.79**	**215.79**	**225.38**	**225.38**	**225.38**	**225.38**	**225.38**	**225.38**	**221.39**	**220.52**	**220.52**	**220.52**	**229.14**
Total Unit Operating Cost per Payable Ounce	**$**	**/Toz Ag**	**$ 9.25**	**8.48**	**8.46**	**8.71**	**8.71**	**9.09**	**9.09**	**9.09**	**9.09**	**9.09**	**9.09**	**9.87**	**10.07**	**10.07**	**10.07**	**10.46**
Total UOC with By-product Credit and Royalties per Payable Ounce	$	/Toz Ag	$ 10.70	11.21	11.21	11.44	11.43	11.85	11.85	11.85	11.85	11.85	11.85	12.82	13.00	12.99	12.99	14.29
Net Cash Cost per Payable Ounce	$	/Toz Ag	$ 11.80	10.95	10.92	11.12	11.10	11.49	11.49	11.49	11.49	11.49	11.49	12.71	12.97	12.96	12.96	14.14
Operating Costs																		
Mining			(1,217,872,085)	(44,593,513)	(44,471,673)	(44,471,673)	(44,471,673)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(47,974,140)	(23,572,037)
Power (Above Ground)			(34,963,665)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(1,271,406)	(635,703)
Paste Backfill Cement			(34,214,858)	(1,306,620)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(1,303,050)	(640,252)
Processing			(238,809,316)	(8,805,227)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(8,796,977)	(4,440,974)
Refining			(380,902,297)	(15,847,241)	(15,851,527)	(15,298,359)	(15,298,359)	(15,298,359)	(15,298,359)	(15,298,359)	(15,298,359)	(15,298,359)	(15,298,359)	(13,841,089)	(13,525,843)	(13,525,843)	(13,525,843)	(6,645,895)
General & Administration			(209,552,915)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(7,620,106)	(3,810,053)
Mine Reclamation & Closure Cost			(1,350,000)															(1,350,000)
1.00 **Total Operating Costs**			**$ (2,117,665,135)**	**(79,444,113)**	**(79,314,739)**	**(78,761,571)**	**(78,761,571)**	**(82,264,038)**	**(82,264,038)**	**(82,264,038)**	**(82,264,038)**	**(82,264,038)**	**(82,264,038)**	**(80,806,768)**	**(80,491,522)**	**(80,491,522)**	**(80,491,522)**	**(41,094,915)**
Mine Development Costs			(456,544,797)	(10,885,065)	(8,960,970)	(16,485,868)	(16,696,070)	(17,090,031)	(18,202,916)	(19,065,772)	(11,029,261)	(15,812,143)	(21,962,914)	(24,873,856)	(24,850,627)	(11,269,876)	-	-
Net Profit Before Depreciation			**$ 2,622,403,448**	**120,722,517**	**122,977,878**	**109,090,250**	**108,998,849**	**105,102,420**	**103,989,535**	**103,126,679**	**111,163,190**	**106,380,308**	**100,229,537**	**75,668,296**	**71,359,325**	**85,028,669**	**96,298,545**	**42,658,959**
Depreciation			(345,752,917)	(7,717,917)	(6,170,917)	(6,214,143)	(7,019,143)	(7,963,429)	(6,957,143)	(8,213,000)	(6,539,143)	(6,385,571)	(5,858,571)	(4,969,857)	(3,724,857)	(3,236,000)	(1,349,429)	(833,429)
Net Profit Before Taxes			2,276,650,531	**113,004,600**	**116,806,961**	**102,876,107**	**101,979,706**	**97,138,992**	**97,032,392**	**94,913,679**	**104,624,047**	**99,994,737**	**94,370,966**	**70,698,439**	**67,634,468**	**81,792,669**	**94,949,116**	**41,825,530**
1.00 State and Federal Income Tax Factor		27.7%	(630,632,197)	(31,302,274)	(32,355,528)	(28,496,682)	(28,248,378)	(26,907,501)	(26,877,973)	(26,291,089)	(28,980,861)	(27,698,542)	(26,140,758)	(19,583,468)	(18,734,748)	(22,656,569)	(26,300,905)	(11,585,672)
Net Profit After Taxes			**$ 1,646,018,334**	**81,702,326**	**84,451,433**	**74,379,425**	**73,731,327**	**70,231,491**	**70,154,419**	**68,622,590**	**75,643,186**	**72,296,195**	**68,230,208**	**51,114,971**	**48,899,720**	**59,136,099**	**68,648,211**	**30,239,858**
Add-Back Depreciation			345,752,917	7,717,917	6,170,917	6,214,143	7,019,143	7,963,429	6,957,143	8,213,000	6,539,143	6,385,571	5,858,571	4,969,857	3,724,857	3,236,000	1,349,429	833,429
Capital																		
Total Capital Costs (less Spare Parts/Initial Fills)			(108,896,711)															
Preproduction Mine Development			(18,656,640)															
Spare Parts/Initial Fills			(2,787,949)															
1.00 **Total Initial Capital Costs**			**$ (130,341,300)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital			(4,512,266)															
Sustaining Capital			(215,411,615)	(5,071,000)	(4,120,000)	(6,326,000)	(8,820,000)	(3,447,000)	(13,231,000)	(4,775,000)	(4,108,000)	(463,000)	(25,000)	(25,000)	(25,000)	(25,000)	(25,000)	(25,000)
Recapture of Spare Parts/Initial Fills			2,787,949															2,787,949
Working Capital Recapture			4,512,266															4,512,266
ANNUAL CASH FLOW			**$ 1,648,806,285**	**84,349,242**	**86,502,350**	**74,267,568**	**71,930,470**	**74,747,920**	**63,880,562**	**72,060,590**	**78,074,329**	**78,218,766**	**74,063,780**	**56,059,828**	**52,599,577**	**62,347,099**	**69,972,640**	**38,348,502**
CUMULATIVE CASH FLOW				**695,732,304**	**782,234,653**	**856,502,221**	**928,432,691**	**1,003,180,611**	**1,067,061,173**	**1,139,121,764**	**1,217,196,093**	**1,295,414,859**	**1,369,478,638**	**1,425,538,467**	**1,478,138,044**	**1,540,485,144**	**1,610,457,783**	**1,648,806,285**

IRR	**28.6%**
Disc	**NPV**
0.0%	**$ 1,648,806,285**
2.5%	**$ 1,083,435,262**
5.0%	**$ 732,176,299**
7.5%	**$ 505,695,638**

23.0 ADJACENT PROPERTIES

The Sunshine Mine property is joined on the west side by United Silver Corp.'s Crescent Mine and to the east by U.S. Silver & Gold Inc.'s properties, which includes the Coeur d'Alene Mine, Coeur Mine, Galena Mine, and the Caladay Mine. Hecla Mining Company's Lucky Friday Mine is located to the east of the Sunshine mine property.

The Crescent Mine property covers 902 acres of ground and is currently being redeveloped by a new surface decline, reopening of the No. 4 Level, and the Hooper Tunnel. Mineralization lies along two parallel systems, the South Vein and the Alhambra Fault. The South Vein is the western extension on the Yankee Girl Vein and the Alhambra Fault is the western extension of the West Chance Vein at the Sunshine Mine. Estimated resources for the Crescent Mine are shown below in Table 23.1 and have been sourced from United Silver Corp.'s 2011 Technical Report (*Amended NI 43-101 Technical Report on Resources United Silver Corp Crescent Mine Kellogg*, ID SRK Consulting, 2011) by SRK.

U.S. Silver & Gold Inc.'s Galena Mine property covers 14,000 acres of ground. U.S. Silver Corp. is currently mining the Galena Mine and is working on reopening the Coeur Mine. The Coeur d'Alene Mine has been shut down for many years and the Caladay Mine is used only for exhaust for the Galena Mine at this time. The Galena Mine currently produces about 2.3 million ounces of silver per year. Estimated resources for the Galena Mine are shown below in 23.1 and have been sourced from U.S. Silver Corp.'s 2012 Technical Report (*Technical Report Galena Project Shoshone County, Idaho*, Chlumsky, Armbrust & Meyer, LLC, 2012) by Chlumsky, Armburst and Meyer. All of the mining has occurred in either the hanging wall or footwall of the major geological structure known as the Polaris Fault. The Polaris Fault intersects the Chester Fault on the eastern side of the Consolidated Silver Mine and also traverses all of the Sunshine Mine property as well.

Hecla Mining Company's Lucky Friday Mine estimated resources are also included in Table 23.1 below and have been sourced from a public news release by Hecla Mining Company from February of 2012.

This information is reproduced from the public disclosure of United Silver Corp. and U.S. Silver & Gold Inc., and Hecla Mining Company, respectively, and the author has been unable to verify the information. Assay results, mineral resources, and reserves from adjacent properties are not necessarily indicative of assay results, mineral resources, and reserves of the subject property.

Table 23.1 Adjacent Properties Estimated Resources

Classification	Cutoff (opt)	Short Tons	Silver Grade (opt)	Silver Ounces
Crescent Mine - United Silver Corp.				
Measured and Indicated	11.0	324,000	18.7	6,058,800
Inferred	11.0	211,000	19.5	4,114,500
Galena - U.S. Silver Corporation				
Measured and Indicated[1]	8.0	2,495,554	14.1	35,135,237
Inferred	8.0	974,139	13.5	13,141,135
Lucky Friday - Hecla Mining Company				
Measured and Indicated[1]	Not Provided	22,548,600	7.5	168,133,350
Inferred	Not Provided	4,396,900	9.7	42,649,930

Note 1: Includes Proven and Probable Reserves

24.0 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information has been included in the above sections.

25.0 INTERPRETATIONS AND CONCLUSIONS

This section presents the conclusions of the authors and Qualified Persons for the Sunshine Mine Project, as addressed in this Technical Report.

25.1 Interpretations and Conclusions

- Overall, the results of the PEA indicate that the Sunshine Mine Project is a robust silver project at this stage of development and warrants further work toward the next stage of development. The exploration program continues to demonstrate the potential for future growth of the resource. Risks, as well as significant opportunities (identified below in Item 25.2), can be evaluated in the feasibility stage of the project.
- All sources of historic data are internally consistent, have supported several decades of mining, and are suitable for use in resource estimation.
- The Sunshine Mine is complying with CIM Definition Standards best practice requirements for sample handling QA/QC.
- The sample preparation, security, and procedures followed by SSMC are adequate to support a mineral resource estimate.
- Assay data provided by SSMC were represented accurately and are suitable for use in resource estimation.
- Based on over 90 years of production history, there are no known factors which should have a negative economic effect on metallurgical recoveries.
- As the operation progresses and reclamation or environmental legislation/regulation requirements evolve, SSMC will be required to maintain, renew existing, or possibly acquire new approvals and permits. However, at this time, all environmental permits, agreements, and approvals necessary to commence surface and subsurface operations are in place.
- There are no environmental issues existing or anticipated that could materially impact the ability to reopen the Sunshine Mine.
- There are no known factors related to metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect the mineral resource estimates.
- This PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that these inferred resources will ever be upgraded or that this PEA will be realized. Mineral resources that are not mineral reserves have no demonstrated economic viability.

25.2 Risks and Opportunities

A number of risks and opportunities have been identified for the Sunshine Mine Project. Potential risks that could affect the performance of the project include:

- Long term depressed metals pricing, particularly for silver.
- Political changes affecting regulatory requirements or the general business climate.
- Shortage of skilled labor due to competing demands from the mining industry in general, and other mines in the Silver Valley in particular.
- Increased inflation and substantial escalation of project equipment, bulk materials, and consumables costs.

- Failure to obtain or maintain necessary permits and approvals of government authorities.

Substantial opportunities for improving the Sunshine Silver Mine Project's performance exist. They include:

- Higher metals pricing, particularly for silver, than used as long term forecast in the financial model.
- Continued expansion of the mineral resources.
- Improved performance in the new mineral processing plant, resulting in reduction of concentrate penalty elements and more economic distribution of payable metals between concentrates.
- Significantly improved copper and lead grades than currently modeled. Model is limited by lack of available data.
- More cost effective development with more detailed information.
- More cost effective assignment of mining methods with more detailed information.

During the next phase of project development, feasibility, a number of risks will be investigated further and possibly reduced or eliminated. Similarly, further investigation and determination of some opportunities may allow them to be incorporated in the project.

26.0 RECOMMENDATIONS

Based on results of this PEA, the authors recommend that SSMC complete a FS to further define the Sunshine Mine Project in order to: more accurately assess its economic viability; support permitting activities; and, ultimately, support project financing should the FS results be positive.

An estimate of the costs to complete the FS is summarized below in Table 26.1.

Table 26.1 Recommended Future Work

Task	Estimated Cost US$000
1. Upgrade Mineral Resource Classification	
a) Infill drilling, including sample preparation, assaying, and all related site support activities	600
b) Update resource model/estimate with infill drilling results and additional historical data	100
Subtotal Resource Estimate	**700**
2. Complete Feasibility Study	
a) study, backfill design, and preparation of cost estimates	330
b) Metallurgical testing, including peer review	465
c) Hydrological, hydrogeological, and geotechnical field investigations	175
d) Process, geotechnical and infrastructure design, including preparation of cost estimates	1,170
e) Environmental and permitting activities	170
f) Marketing Study	15
g) Study management and coordination, execution planning and scheduling, owner's cost estimating, and economic evaluation	285
Subtotal Feasibility Study	**2,610**
Total Estimated Cost for Recommended Future Work	**$ 3,310**

27.0 REFERENCES

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28.0 ILLUSTRATIONS

All figures, tables and illustrations have been included in their respective sections.