National Instrument 43-101 Technical Report – Preliminary Economic Assessment

Commonwealth Silver and Gold Project

Cochise County, Arizona, USA

Report Date: April 30, 2014

Effective Date: November 30, 2013

Report Prepared for

Commonwealth Silver and Gold Mining Inc.

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1. Executive Summary

Introduction

Hard Rock Consulting, LLC ("HRC") was retained by Commonwealth Silver and Gold Mining Inc. ("CSGM" or the "Company") to complete a Preliminary Economic Assessment ("PEA") and associated National Instrument 43-101 ("NI 43-101") Technical Report for the Commonwealth Silver and Gold Project (the "Project" or the "Commonwealth Project") in Cochise County, Arizona. The PEA is preliminary in nature, and there is no certainty that the results set forth in the PEA will be realized. The mineral resource estimate included in this report includes inferred mineral resources which are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The inferred resources are not included in the economic model. Mineral resources that are not mineral reserves do not have demonstrated economic viability. This report presents the results of the PEA based on all available technical data and information as of November 30, 2013.

This report was prepared in accordance with the Canadian Securities Administrators ("CSA") NI 43-101 and in compliance with the disclosure and reporting requirements set forth in Companion Policy 43-101CP and Form 43-101F1 (June 2011). Mineral resources are classified in accordance with standards as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards - For Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 17, 2010.

Property Description and Ownership

The Commonwealth Silver and Gold Project is located in central Cochise County, Arizona, approximately 40km (25 miles) south of Willcox and 125 km (75 miles) southeast of Tucson. The Project area consists of 1,568 hectares covering portions of Sections 33 through 36, T17S, R25E, and Sections 1 through 8, T18S, R25E GSRBM; Gila and Salt River meridian. The Project area lies within the Basin and Range physiographic province, which extends from Nayarit state in Mexico to the south to northern Nevada in the United States to the north.

CSGM holds several varieties of mineral title which together comprise the Commonwealth Silver and Gold Project, including fee land, seven patented lode claims, one patented mill site, 133 unpatented lode claims on federal lands and 647.1 ha (1,599.44 acres) of Mineral Exploration Permits from the State of Arizona. The mining concessions are located in the Sulphur Springs Valley between the Chiricahua and Swisshelm mountains to the southeast and the Dragoon Mountains to the west. Elevation within the Project area ranges from 1,300 to 1,450 m above sea level, and local terrain is generally treeless with low growing, salt tolerant desert plants of the Chihuahuan desert and semi-desert grassland life-zones. Climate and terrain conditions generally allow for year-round exploration and mining activity.

Environmental

The property is not subject to any known environmental liabilities. As the area has a long history of mineral exploration, CSGM does not anticipate any barriers to access for work planned going forward. The preliminary mine permit assessment is based on the premise that the proposed Project mine facilities located in Pearce, Cochise County, Arizona, including probable mine features such as a pit, waste dumps, processing facilities and infrastructure are all located on private land. At this time, no federal Environmental Assessment or Impact Statement process is anticipated.

Geology and Mineralization

The Commonwealth Project lies near the northwestern limit of the Sierra Madre Occidental, a northwest trending volcanic plateau composed of thick accumulations of andesitic to rhyolitic volcanic rocks. The Project is situated within a highly mineralized belt that extends north from Mexico City (and possibly further south), along the Sierra Madre Occidental, and into southern Arizona, and is typical of Tertiary age, low sulfidation, epithermal precious metals systems found throughout northwestern Mexico and the southwestern United States.

The mineral deposits within the Commonwealth Project area are typical of silver dominant, low sulfidation, epithermal veins and stockworks emplaced in a near surface environment. The veins are best developed in the andesite to rhyolite units of the Pearce Volcanics. These volcanic rocks fracture well, are densely fractured to shattered, and host dense quartz stockworks, breccia zones and banded quartz veins. Cretaceous marine sediments of the Bisbee Group also host mineralization and are chemically favorable hosts. The calcareous sandstones and siltstones of the Bisbee Group are very similar to the “dirty carbonate” host rocks of many sediment hosted disseminated gold deposits in Nevada and in northern Mexico. These rocks seem to be especially favorable hosts for gold mineralization and there is a higher gold to silver ratio in the assay results from mineralized Bisbee Group samples as compared to the mineralized volcanic rocks at the Commonwealth Project. The Bisbee Group sediments are soft enough that they do not fracture well on faulting. Mineralization within the Bisbee Group sediments occurs as both vein type mineralization and some disseminated mineralization.

The conceptual geologic model that best applies to the Commonwealth Project is a variation of a rift, low sulfidation, epithermal chalcedony-ginguro model (Corbett, 2002). The characteristics of this model are mineralogy derived primarily from dilute, near neutral pH fluids, an extensional, dilatant structural setting, competent host rocks that fracture well and abundant banded chalcedonic quartz.

Status of Exploration

The Commonwealth deposit was discovered in 1895 by John Pearce, who, while driving cattle over Pearce Hill, picked up an unusually heavy rock and decided to have it assayed. Pearce’s rock ran 2,100 opt in silver (approximately 71,918 g/t), prompting him to locate six mining claims, which comprise the heart of the modern-day Commonwealth Project and are currently controlled by CSGM. Total

production from the mine was approximately 138,000 ounces of gold and 12 million ounces of silver from approximately 1,341,000 short tons (approximately 1,216,000 tonnes) of ore (Keith, 1973).

Between 1895 and 1927, extensive underground development totaling about 32 km (20 miles) of workings on 8 levels was completed at the Commonwealth Mine. Approximately 3 km (2 miles) of workings on 4 levels, mostly between C and D shafts, are accessible today. The mine extended down to the 8th level at an elevation of about 1,260 m (4133 ft). Detailed maps were completed for the 2nd, 3rd, 5th, 7th and 8th levels by Smith (1927) and of the 6th level by Howell (1977). These maps provide an excellent basis for modern exploration work. After 1927, the mine was worked intermittently by lessors until 1942 when precious metals mines across the United States were shut down by The War Powers Act. Exploration efforts on the Project resumed in the 1970's when the area was recognized as having bulk tonnage silver potential.

CSGM has conducted three drilling programs at the Project. The first 16 hole, 2,003 meter program began on April 1, 2011 and was completed on June 16, 2011. The second 35 hole, 5,033 meter program began on November 21, 2011 and was completed on April 2, 2012 and the third 7 hole, 657 meter program was begun September 17, 2012 and completed October 14, 2012. In the first program, CSGM completed 16 HQ-size core holes as part of a confirmation and infill drilling program used to support the calculation of the initial NI 43-101 compliant mineral resource estimate on the Project by SRK Consulting (U.S.), Inc. ("SRK") in October 2011. The second and third programs comprising 35 and 7 HQ-size core holes, respectively, were completed as part of infill and step out-drilling programs. In addition, the third program included 5 holes drilled specifically for metallurgical samples. Drilling is discussed in greater detail in Section 9 of this report.

Planned exploration in the main Project area, the area in the immediate vicinity of the old mine workings, will consist mainly of step-out drilling following the east plunging intersection of the Main and North Veins, exploration drilling in the footwall of the North Vein to define potential disseminated mineralization in the Bisbee Group sediments and step-out drilling to the west of Pearce Hill where the deposit is open along strike.

Mineral Resource Estimate

Zachary J. Black, SME-RM, a Resource Geologist with HRC is responsible for the estimation of the mineral resource herein. Mr. Black is a qualified person as defined by NI 43-101 and is independent of CSGM. HRC estimated the mineral resource for the Project from drill-hole data, using controls from the main rock types and a series of implicit grade shells with an Inverse Distance ("ID") algorithm.

The mineral resources presented in this report are classified under the categories of Measured, Indicated and Inferred according to CIM guidelines. Classification of the resources reflects the relative confidence of the grade estimates. Confidence with regard to the grade estimates is based on several factors, including but not limited to sample spacing relative to geological and geostatistical observations, the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, quality of the assay data, and other factors.

HRC created a three dimensional ("3D") block model in MicroModel mining software. The block model was rotated 20 degrees east of north to align the rotated easting along the strike of mineralization. The block model was created with individual block dimensions of 6x3x3 meters (xyz). All property and minerals within the block model extents are owned or claimed by CSGM. Each of the blocks was assigned attributes of gold, silver, and gold equivalent grade, resource classification, rock density, tonnage factor, lithology, and a grade domain classification.

The mineral resource at the Commonwealth Project was modeled by constructing a geologic block model from the CSGM geologic interpretation provided by CSGM. The drill data was geostatistically analyzed to define the parameters used to estimate gold and silver grades into the 3D block model. Leapfrog 3D® geological modeling software was used to create 3D stratigraphic and mineralized domain solids and MicroModel mining software was used to estimate gold and silver grades.

CSGM defined the structure and stratigraphy of the Commonwealth Project on electronic cross sections spaced 30m (100 ft) apart and oriented perpendicular to the strike of the vein system, to best account for orientation of the deposit. HRC combined the CSGM subsurface interpretations with the surface geology to create 3D stratigraphic and mineralization models.

The existing mine stopes were mapped by Harvest Gold Corporation ("Harvest Gold") and Atlas Precious Metals, Inc. ("Atlas") between 1994 and 1996. A polygon outlining the mapped stope on each accessible level was used create a 3D solid representing the mined out material between levels. The solid was provided to HRC and combined with the provided level plan solids to code the block model with mined out material.

The mineral resource estimate for the Commonwealth Silver and Gold Project is summarized in Table 1-1. This mineral resource estimate includes all drill data obtained as of June 10, 2013, and has been independently verified by HRC. Mineral resources are not mineral reserves and may be materially affected by environmental, permitting, legal, socio-economic, marketing, political, or other factors. In Table 1-1, mineral resources are reported above a 0.2 g/t gold equivalent ("AuEq") cut-off, assuming an average gold price of US$1,350 per ounce. This cut-off reflects the potential economic, marketing, and other issues relevant to an open pit mining scenario based on a Merrill-Crowe recovery process following cyanide heap leaching. HRC cautions that economic viability can only be demonstrated through prefeasibility or feasibility studies.

Table 1-1 Mineral Resource Statement for the Commonwealth Silver and Gold Project

Cochise County, Arizona, Hard Rock Consulting, LLC, December 31, 2013

Cutoff	Volume	Tonnage	Gold Equivalent		Gold		Silver	
(gpt)	cu. M	000 tonnes	gpt	t. oz.	gpt	t. oz.	gpt	t. oz.
Inverse Distance 2.5 Model In Pit Measured Resources								
0.4	1,662,900	4,069	1.380	180,800	0.57	74,800	48.6	6,357,700
0.3	1,841,200	4,504	1.280	185,700	0.53	77,200	45.0	6,516,900
0.2	**2,047,000**	**5,007**	**1.18**	**189,800**	**0.49**	**79,000**	**41.3**	**6,648,500**
Inverse Distance 2.5 Model In Pit Indicated Resources								
0.4	8,966,100	21,934	1.06	746,100	0.45	314,500	36.8	25,950,900
0.3	10,893,200	26,643	0.93	799,200	0.40	339,200	32.2	27,582,000
0.2	**12,522,400**	**30,623**	**0.85**	**832,000**	**0.36**	**354,400**	**29.1**	**28,650,600**
In Pit Measured and Indicated Resources								
0.4	10,629,100	26,003	1.11	926,900	0.47	389,300	38.6	32,308,700
0.3	12,734,400	31,147	0.98	984,900	0.42	416,400	34.1	34,098,900
0.2	**14,569,400**	**35,630**	**0.89**	**1,021,700**	**0.38**	**433,500**	**30.8**	**35,299,100**
Inverse Distance 2.5 Model Inferred Resources								
0.4	3,021,700	7,380	0.58	136,700	0.29	67,900	17.2	4,075,100
0.3	5,314,000	12,974	0.48	199,600	0.25	102,800	13.8	5,762,000
0.2	**7,672,600**	**18,733**	**0.41**	**245,400**	**0.21**	**127,600**	**11.6**	**6,998,200**

***Notes:**

(1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

(2) Measured and Indicated Mineral Resources captured within the pit shell meet the test of reasonable prospect for economic extraction and can be declared a Mineral Resource.

(3) Inferred Mineral Resources are that part of the Mineral Resource for which the quantity and grade or quality are estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity.

(4) All resources are stated above a 0.2 g/t gold equivalent ("AuEq") cut-off.

(5) Pit optimization is based on assumed gold and silver prices of US$1,350/oz. and US$22.50/oz., respectively and mining, processing and G&A costs of US$7.25 per tonne. Metallurgical recoveries for gold and silver were assigned by lithologic unit.

(6) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

(7) Gold Equivalent stated using a ratio of 60:1 and ounces calculated using the following conversion rate: 1 troy ounce = 31.1035 grams. Metallurgical recoveries are not accounted for in the gold equivalent calculation.

Economic Analysis

Certain statements made and information contained herein are considered "forward-looking" within the meaning of applicable Canadian securities laws. These statements address future events and conditions

and so involve inherent risks and uncertainties. Actual results could differ from those currently projected.

HRC envisions the Commonwealth Silver and Gold Project as an open pit, heap leach operation with Merrill-Crowe processing of leachates completely contained on patented or privately held surface lands, with an estimated resource of 31.2 million tonnes grading 0.39 gpt gold and 32 gpt silver. The operations are planned to run at a rate of 10,000 tonnes per day, 365 days per year, with all mineralized materials being crushed and agglomerated prior to stacking on the heap. Metal recoveries are expected to average around 79% for gold and about 34% for silver.

Economic analysis of the base case scenario for the Project uses a price of US$1350/oz for gold, which is approximately equal to both the five-year and the spot price at the end of October, 2013. The silver price for the base case, for all prices in the sensitivity calculations, and for the cut-off grade determination is at a factor of 1/60th of the gold price, which was the ratio at the end of October, 2013. The economic model shows an After-Tax Net Present Value @ 5% ("NPV-5") of $101.3 million using a 0.24 gpt gold equivalent (AuEq) mining cutoff grade, as well as an After-Tax Rate of Return ("IRR") of 58%. Table 1-2 summarizes the projected net present value, NPV-5; internal rate of return, IRR; years of positive cash flows to repay the negative cash flow ("Payback Period"); multiple of positive cash flows compared to the maximum negative cash flow ("Payback Multiple") for the Commonwealth Silver and Gold Project on both After Tax and Before Taxes bases.

Table 1-2 Commonwealth Economic Performance versus Gold Price

Au Price	Ag Price	NPV @ 5.0%, After Tax	Internal Rate of Return, After Tax	Payback Period, After Tax	Payback Multiple, After Tax	NPV @ 5.0%, Pre-Tax	Internal Rate of Return, Pre-Tax	Payback Period, Pre-Tax	Payback Multiple, Pre-Tax
$1,050	$17.50	$22.29	18.6%	5.20	1.83	$34.19	24.7%	2.60	2.20
$1,125	$18.75	$42.65	29.6%	2.48	2.45	$60.92	38.2%	2.16	3.02
$1,200	$20.00	$62.35	39.6%	2.11	3.04	$87.65	50.9%	1.81	3.85
$1,275	$21.25	$81.82	49.1%	1.81	3.63	$114.34	63.1%	1.54	4.68
$1,350	$22.50	$101.25	58.2%	1.59	4.22	$141.02	74.9%	1.26	5.53
$1,425	$23.75	$120.41	67.0%	1.37	4.80	$167.70	86.5%	1.04	6.39
$1,500	$25.00	$138.72	75.1%	1.28	5.34	$194.38	97.9%	0.95	7.24
$1,575	$26.25	$156.42	82.4%	1.14	5.85	$221.06	109.1%	0.79	8.09
$1,650	$27.50	$174.12	89.7%	1.02	6.36	$247.73	120.3%	0.68	8.95

The projected total lifespan of the project is 11 years: two years of pre-production and construction, and 9 years of full operations. Approximately 392,400 ounces of gold and 32.0 million ounces of silver are projected to be mined, with 311,500 ounces of gold and 10.9 million ounces of silver recovered and produced for sale. A total capital investment of $55.1 million, including contingency and sustaining

capital, is projected. Following the Gold Institute ("GI") guidelines, life-of-mine average base case cash operating cost is projected to be $1281 per ounce of gold sold, before credits for silver sales, and $492 per ounce of gold after silver credits. The GI life-of-mine average base case total cash cost (including royalties and production taxes) after credits would be $528 per ounce, and the GI life-of-mine average base case total production cost after credits is expected to be $708 per ounce. Table 22-2 details the GI life-of-mine average base case total production cost on an equivalent ounce per gold basis and on a cost/tonne of mineralized material basis.

Table 1-3 Commonwealth Total Production Cost/ounce Gold Equivalent

Operating Costs	$/oz AuEq	$/ton processed material	$/ton mined
Total Mining	$369.14	$5.85	$2.97
Total Processing	$386.72	$6.13	
Total Site General & Administration	$40.41	$0.64	
Property Taxes	$4.70	$0.07	
Cash Operating Costs	**$800.97**	**$12.69**	
Transportation and Refinery	$7.75	$0.12	
Royalties	$16.52	$0.26	
Severance Taxes	$5.99	$0.09	
Total Cash Costs	**$831.23**	**$13.17**	
Total Capital Costs	$113.86	$1.80	
Total Production Costs	**$945.09**	**$14.97**	

The gold price used in the economic evaluation (US$1,350/oz Au) is the five-year trailing average and approximate spot price as of the end of October 2013. The silver price for each case was determined by dividing the gold price by the end of October 2013 gold:silver price ratio (60:1).

Conclusions and Recommendations

Environmental

This preliminary mine permit assessment is based on the premise that the proposed Project mine facilities located in Pearce, Cochise County, Arizona including probable mine features including a pit, waste dumps, processing facilities, and infrastructure are all on private land. At this time, no federal Environmental Assessment or Impact Statement process is anticipated. The following areas will have to be investigated in reference to the Project:

- Cultural Resources;
- Endangered Species Act and other Biological Requirements;
- Aquifer Protection Permit;
- Clean Water Act, Section 402 (AZPDES);

- Clean Water Act, Section 404 (Dredge and Fill); and
- Air Permitting.

Environmental aspects that at this time may not apply to the Project include the Arizona Native Plant Law and NEPA since the Project would be permitted under state and local laws and regulations with no federal permits that would trigger NEPA.

Geology and Deposit Type

CSGM personnel have a thorough understanding of the Project geology and are applying the appropriate deposit model for exploration.

Exploration, Drilling, and Analytical

HRC is of the opinion that CSGM is conducting exploration activities, drilling, and analytical procedures in manner that meets or exceeds industry best practice.

Metallurgical

Overall, fine crushing may not be necessary based on the high metal extraction percentages from the bottle roll tests. No significant increase in metal extraction occurred from finely milling any of the composite material.

HRC agrees with CSGM's conclusion that, based on market conditions and comparing extensively tested metallurgical recovery rates associated with a lower capital cost heap leaching scenario to the preliminary results of metallurgical test work associated with a higher capital cost milling scenario, that while mining should remain open pit, the lower costs associated with heap leach processing increases the prospect for economic extraction of the Mineral Resources. HRC concludes that the metallurgical results presented in Table 1-4 are the most appropriate for the Commonwealth Project.

Table 1-4 Metallurgical Crush and Recovery Recommendations

Rock Type	Crush Size	Recoveries (%)	
		Au	Ag
Rhyolite	Minus 8	78	30
Vein	Minus 8	79	49
Lower Andesite	1/2"	81	33
Upper Andesite	1/2"	78	35
Bisbee	1/2"	80	23

Data Verification

HRC received original assay certificates in pdf format for all samples included in the current drill-hole database. A random manual check of 10% of the database against the original certificates was conducted focusing on the five primary metals (Au, Ag, Cu, Pb, and Zn), with occasional spot checks of secondary constituents. HRC also conducted a random check of at least 2% of the highest (5%) assay

values and continued to randomly spot check assays values throughout the modeling process. HRC is of the opinion that the data maintained within the database is acceptable for mineral resource estimation.

Resource

HRC finds that the density of data within the resource base is adequate for the Preliminary Economic Assessment. The mineral resource estimation is appropriate for the geology and assumed open pit mining method. Additional modeling should be conducted to define the alteration of the host rocks to support further metallurgical testing.

Significant Risks and Uncertainties

Metallurgical testing continues to be important to the Project going forward. Additional testing and analysis should be conducted to support the 1995 Atlas column leach program. Investigation of the factors retarding the silver recoveries could identify additional metallurgical categories to assist in the modeling of the resource.

Recommendations

HRC recommends that CSGM complete pre-feasibility and, dependent upon a positive outcome of the pre-feasibility, feasibility level studies necessary to confirm the results of exploration and of preliminary metallurgical, geotechnical, hydrological, and environmental studies.

Metallurgical Study

CSGM should complete a detailed metallurgical study conducted by rock type. This study should be designed to supplement the assumptions and conclusions in this PEA. The additional metallurgical studies should focus on improving silver recovery by the use of additional reagents such as lead nitrate and should develop an additional set of points on the crush size vs recovery curves at -3/8” for all rock types. Bottle roll tests using the additional reagents should be completed before initiating column tests. Additional column tests at -3/8” using the final selection of reagents run for a minimum of 120 days with the goal of simplifying the process flow sheet from the currently envisioned 2 final crush sizes to 1 uniform crush size.

- Gold and Silver Particle Size Analysis, deportment by rock-type;
- Acid-Base Accounting;
- Mineralogical Evaluation using QEMSCAN;
- Bond Abrasion and Bond Impact Tests;
- Bottle Roll Testing focusing on -3/8” crush size
- Agglomeration Testing (-1/2”, -3/8 and Minus 8); and
- Column Leach Tests focusing on the -3/8” crush size

HRC recommends that this work be completed on all rock types with an effort made to evaluate changes in mineralogy. Additionally, work should be completed on the basis of elevation as there are reports (Forrest, 1995) that suggest a metallurgical change with depth.

Geotechnical and Hydrogeological Study

HRC Recommends that CSGM complete an updated pit slope study analysis on the available data to evaluate the potential pit slope angles, identify any critical geotechnical areas, and to define a geotechnical exploration program to support the final design of an open pit.

Additionally, HRC recommends that CSGM conduct a preliminary hydrogeological study to support the future Project water needs and to define a critical path process to achieving the water needs for development. The hydrological study should address possible de-watering in the later stages of the pit, test wells in groundwater source areas and the completion of the monitor wells for acquisition of the baseline data for permitting.

Environmental Permitting

HRC recommends that CSGM continue to work towards meeting the requirements of the State of Arizona to permit a mine on private land. This should include in the short term continued work on:

- Cultural Resources Inventory;
- Endangered Species Act (ESA) and other biological requirements; and
- Collection of Environmental Baseline Data including the completion of monitor wells near the point of compliance.
- Complete and submit Aquifer Protection Permit Application to ADEQ.

Exploration Program

Continued exploration diamond core drilling should be targeted in four areas within the immediate mineral resource area:

- Step out drilling in the identified mineralization to the west of the Brockman Fault (4 holes);
- Step out drilling along both the Main and North Veins (5 holes);
- Infill and step out drilling along the footwall (Eisenhart Vein) mineralization (4 holes); and
- Step out drilling along the identified mineralization in the hanging wall block of the Main vein (7 holes).

The early open hole rotary/percussion holes drilled in the late 1970's and early 1980's by Bethex (P-76-6 to P-76-14) and Platoro (CS-1 – CS-5) should be twinned with diamond core holes and replaced in the database.

Underground surveys of the existing mine workings including 3D laser scanning where appropriate should be completed to improve the location of the workings in the project model to a level sufficient for final mine planning.

Additionally, leach pad and waste dump condemnation reverse circulation drilling should be conducted and historical barren holes (GH series holes) should have their collars surveyed and their data incorporated into the Project data base.

Budget

HRC's recommendations are intended to provide CSGM a path toward the development of the project and advancing the project to PFS level study is not contingent upon positive results from outlined work program. The engineering, permitting and environmental requirements necessary to bring a mine into development need to be assessed to understand any difficulties or costs that will impact the overall project economics at the PFS level. The anticipated costs for the recommended scope of work are presented in Table 1-5.

Table 1-5 Budget Based on Recommendations in 2013 Technical Report

Recommended Scope of Work	Expected Cost (US$)	
Preliminary Feasibility Study	$175,000*	$175,000*
Metallurgical Studies		
Qemscan Mineralogy	25,000	
Physical Testing (Impact and Abrasion)	10,000	
Column Testing (20 columns 120 days)	150,000	
Agglomeration and Percolation Testing	20,000	
Acid-Base Accounting	10,000	
Effluent analysis from column testing	15,000	
Further Bottle-Roll testing using additional reagents	50,000	280,000
Geotechnical and Hydrological Studies		
Geotechnical Report	5,000	
Drill 3 Monitor Wells	90,000	
Hydrological Report	5,000	
Drill 3 Water Source Identification Wells	90,000	190,000
Environmental Permitting		
Phase I and Phase III Cultural Resource Surveys	30,000	
Arizona Native Plant survey and update of ESA opinion	25,000	
Phase II Aquifer Protection Permit contractor fees	180,000	
Fees to ADEQ for APP application	200,000	435,000
Exploration		
Infill and step-out Drilling 3,000 meters (core)	750,000	
Condemnation Drilling 2,500 meters (reverse circulation)	240,000	
Percussion hole Re-Drilling 1500 m (reverse circulation)	180,000	1,170,000
Total Technical Budget for publication in NI 43-101 Technical Report	$2,250,000	$2,250,000

*Including pit and haul road design, preliminary design of heaps, ponds and waste dump, and proposed process flow sheet

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2. Introduction

Issuer and Terms of Reference

Commonwealth Silver and Gold Mining Inc. ("CSGM" or the "Company") is a private Canadian mineral exploration company headquartered in Toronto, Ontario. CSGM contracted Hard Rock Consulting, LLC ("HRC") to complete a Preliminary Economic Assessment and associated National Instrument 43-101 ("NI 43-101") Technical Report for the Commonwealth Silver and Gold Project (the "Project" or the "Commonwealth Project") in Cochise County, Arizona. This report has been prepared in accordance with the Canadian Securities Administrators ("CSA") NI 43-101 and in compliance with the disclosure and reporting requirements set forth in Companion Policy 43-101CP and Form 43-101F1 (June 2011). Resources are classified in accordance with standards as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards - For Mineral Resources and Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 17, 2010. The effective date of this report is November 30, 2013.

Sources of Information

HRC sourced information from referenced documents as cited in the text and summarized in Item 27 of this report. HRC previously completed a NI 43-101 Technical Report on Resources for the Commonwealth Project in September 2013:

Hard Rock Consulting, LLC, 2013. *NI 43-101 Technical Report on Resources, Commonwealth Silver and Gold* Project*, Cochise County, Arizona USA*, prepared for Commonwealth Silver and Gold Mining, Inc., September 5, 2013,

Another Technical Report compliant with current NI 43-101 standards was prepared in 2012:

SRK Consulting (U.S.), Inc. ("SRK") 2012. *NI 43-101 Technical Report on Resources, Commonwealth Silver and Gold Project,* Cochise *County, Arizona USA,* prepared for Commonwealth Silver and Gold Mining Inc., March 15, 2012, amended April 11, 2012.

As the Company is a private company and not a reporting issuer, this report was not filed on SEDAR or with the securities regulators. The report was however, reviewed and approved by the Ontario Securities Commission as part of a planned initial public offering by CSGM in 2012.

A portion of the background information and technical data for this report was obtained from the above reports. Additional information was requested from and provided by CSGM. With respect to Items 6, 9 through 13, 15, and 16 of this report, the authors have relied in part on historical information including exploration reports, technical papers, sample descriptions, assay results, computer data, maps and drill logs generated by previous operators and associated third party consultants. The authors cannot guarantee the quality, completeness, or accuracy of historical information, nor its preparation in

accordance with NI 43-101 standards. Historical documents and data sources used during the preparation of this report are cited in Item 27.

Qualified Persons

The Qualified Persons, as defined by NI 43-101, endorsing this report are Mr. Zachary Black, Ms. Jennifer J. Brown, P.G., and Mr. Jeff Choquette, P.E., all of HRC, and Mr. Deepak Malhotra, of RDi.

Mr. Black, SME-RM, has more than 8 years of experience working on structurally controlled gold and silver resources in the Sierra Madre Occidental of Mexico and the southern United States. Mr. Black completed the resource estimate for the Commonwealth Silver and Gold Project and is responsible for Items 1, 11, 12, 14, 25, and 26.

Ms. Brown, P.G., SME-RM, has more than 15 years of professional experience as a consulting geologist and has contributed to numerous mineral resource projects, including more than fifteen structurally controlled gold and silver resources throughout the southwestern United States and South America over the past five years. Ms. Brown is specifically responsible for Items 2 through 10, 23 and 27.

Mr. Choquette, P.E., is a professional mining engineer with more than 17 years of domestic and international experience in mine operations, mine engineering, project evaluation and financial analysis. Mr. Choquette has been involved in industrial minerals, base metals and precious metal mining projects around the world. Mr. Choquette is responsible for Items 15 through 24.

Deepak Malhotra is President of Resource Development Inc. (RDi) and has worked as a mineral process economist and metallurgical engineer for over 40 years. Mr. Malhotra is responsible for Item 13 of this report.

Details of Inspection

Ms. Brown visited the CSGM Project site on May 2 and 3, 2013. Accompanied by CSGM Senior Geologist Lora Chiehowski, Ms. Brown conducted general field reconnaissance, located and verified numerous modern and historic drill-hole collar locations, recorded a variety of field observations and measurements and became generally acquainted with the on-site facilities and technical staff. While on site, Ms. Brown reviewed sample collection, handling, transfer, and security procedures, observed on-site document and core storage, examined drill core, reviewed handwritten field logs, visually compared laboratory certificate assay values to corresponding core sample intervals and collected two (2) quarter-core samples for duplicate laboratory analysis. Ms. Brown also entered Level 3 of the underground mine workings and collected two channel samples, also for duplicate laboratory analysis. Based on observations in the field and conversation with CSGM staff during the site visit, it is Ms. Brown's opinion that CSGM field activities are carried out in general accordance with industry standard practices and that samples and data are handled with reasonable and appropriate care.

Units of Measure

Unless otherwise stated, all measurements reported herein are in metric units and currencies are expressed in US dollars ("US$"). Gold and silver values are reported in parts per million ("ppm") or in grams per tonne ("g/t"). Tonnage is reported as metric tonnes ("t"), unless otherwise specified.

3. Reliance on Other Experts

During preparation of this report, HRC fully relied upon information provided by CSGM regarding property ownership, mineral tenure, permitting, and environmental liabilities as described in Items 4, 5 and 20 of this report. HRC relied upon information on mining claims, claim status, and environmental liabilities from the following sources:

- CSGM for Land Tenure (Item 4)- Appendix A;
- CSGM for Royalties, Agreements and Encumbrances (Item 4) - Appendix B;
- Erwin & Thompson LLP, Title Opinion dated June 26, 2012 (Item 4); and
- Darling Environmental & Surveying, Ltd., 2011. Internal Company Memo Re: Preliminary Mine Permit Assessment (Item 4)

Information from the sources described above is used in Items 4, 5 and 20 of this report.

4. Property Description and Location

The Commonwealth Silver and Gold Project is located in central Cochise County Arizona (Figure 4-1). The historic Commonwealth Mine is located at the approximate center of the Project, at 31˚54'N latitude and 109˚49'W longitude, roughly 40 km (25 miles) south of Willcox and 125 km (75 miles) southeast of Tucson, Arizona. Access to the Project from Tucson is by Interstate 10 east 95 km (58 miles to the Dragoon Road exit, then east 20 km (13 miles) on a paved two lane road to U.S. Highway 191 ("US 191"), then south 16 km (10 miles) to Pearce, Arizona. The property is 5 km (3 miles) south of the community of Sunsites and 3 km (2 miles) south of the Pearce Post Office. The Project area consists of approximately 1,568 hectares (3,875 acres) covering portions of Section 35, T17S, R25E, and Sections 2 and 3, T18S, R25E SRBM; Gila and Salt River meridian (Figure 4-2).

Figure 4-1 Commonwealth Project Location

Mining Concessions

The Commonwealth Project includes several varieties of mineral title including fee land, seven patented lode claims, one patented mill site, 133 unpatented lode claims on Federal Lands and 647.1 ha (1,599.44 acres) of Mineral Exploration Permits from the State of Arizona.

The first parcel is 6.88 ha (17.34 acres) of fee land that includes surface and mineral ownership. CSGM owns 100% of this parcel following a purchase from J-Rod LLC in 2011.

CSGM also owns a 10% interest in fee lands (surface and mineral ownership) consisting of 22 town lots totaling 0.81 ha (2.133 acres), seven patented mining claims and one patented mill site claim totaling 57.46 ha (142.02 acres), which was purchased from John C.S. Breitner, Trustee, in 2011. CSGM also controls an 88% interest in the town's lots, patented mining claims and mill site discussed in the preceding sentence through a third party agreement with the Carl Thetford Family Trust and three other minority property owners.

CSGM owns 89 unpatented federal lode mining claims that include the J-Rod 1-12 and the CSGM 1-95 claims (non-inclusive) totaling 554.01 ha (1,369 acres). CSGM also controls 44 federal claims under third party agreements with the Carl Thetford Family Trust et al that include 10 Lyle/Pan claims, totaling 33.59 ha (83 acres) and Ralph M. Cartmell et al covering 34 claims (10 Blue Jeep, 18 San Ignacio and 6 Six Mile Hill) totaling 267.90 ha (662 acres). The total acreage of the CSGM owned and leased unpatented federal lode mining claims is 855.51 ha (2,114 acres).

CSGM holds three Mineral Exploration Permits issued by the State of Arizona. The three Mineral Exploration Permits pertain to 647.27 ha (1,599.44 acres) of mineral rights and provide for surface access. These permits are issued by the Arizona State Land Department.

All mineralized material included in the mineral resource estimate presented in this report is located on mining claims owned or leased with the option to purchase by CSGM. There is sufficient space for the process plant, tailings retention area, leach pads and mine dumps on land for which there are private surface rights.

Figure 4-2 shows land tenure of the Project. A list of CSGM fee land, patented claims, unpatented claims and State of Arizona Mineral Exploration Permits is provided in Appendix A.

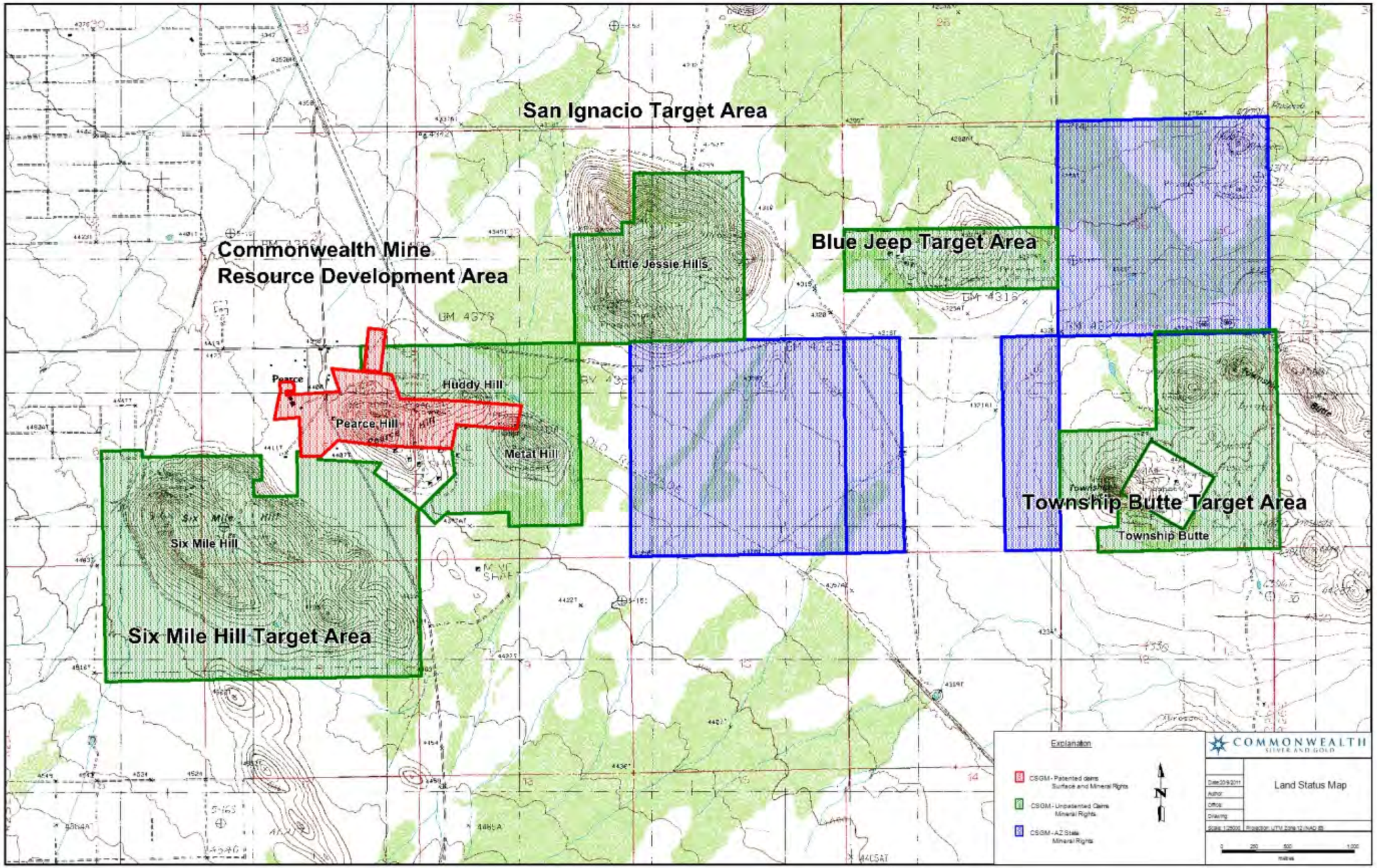


Figure 4-2 Commonwealth Silver and Gold Land Status Map

Royalties, Agreements, and Encumbrances

Lands owned by CSGM including fee lands, town lots, and patented and unpatented mining claims are not subject to any royalty burden. Unpatented mining claims currently carry no United States federal government royalty burden. Property taxes which are the responsibility of CSGM are paid to Cochise County, Arizona on fee lands, town lots, and patented mining claims. Lands controlled by CSGM under third party agreements are subject to royalty burdens and encumbrances as follows:

The Carl Thetford Family Trust et al Mining Lease and Option to Purchase Agreement ("Thetford Agreement") pertains to the fee lands, town lots and patented mining claims (88%) and ten unpatented mining claims (100%). The Thetford Agreement is a five year option to purchase agreement with the option payments and purchase price totaling US$4,500,000. It includes a 2% retained Net Smelter Return ("NSR") production royalty that can be bought down by CSGM to 1% for US$2,000,000, which can be paid in increments of US$1,000,000 per one-half of the 1% of the NSR. Payment of the US$4,500,000 represents an advance against the NSR. Accordingly, in the event that the property goes into production, the entire amount of the purchase price of the property will be recovered through a credit for pre-payment of the first US$4,500,000 of the NSR. Please see the Thetford Agreement summary in Appendix B for more details regarding the Thetford Agreement.

The Ralph M. Cartmell et al Mining Lease and Option to Purchase Agreement ("Cartmell Agreement") pertains to 34 unpatented mining claims as set forth above. The Cartmell Agreement is a five year option to purchase agreement with the option payments and purchase price totaling US$2,000,000. It includes a 2% retained NSR production royalty, which can be reduced by CSGM to 1% for US$1,000,000. Payment of the US$2,000,000 represents an advance against the NSR. Accordingly, in the event that the property goes into production, the entire amount of the purchase price of the property will be recovered through a credit for pre-payment of the first US$2,000,000 of the NSR. Please see the Cartmell Agreement summary in Appendix B for more details regarding the Cartmell Agreement.

The 133 unpatented federal lode mining claims owned or controlled by CSGM require an annual maintenance fee payment to the United States government Bureau of Land Management which is US$140 per claim. US$18,620 was paid in respect of these claims in August 2013 for the filing year September 1, 2013 to August 31, 2014.

The three State of Arizona Mineral Exploration Permits described above allow CSGM to conduct mineral exploration, but not mining, on the state lands totaling 647.27 ha (1,599.44 acres). The Mineral Exploration Permits were issued in 2011 and are renewable annually for up to five years. The rental is US$1.00 per acre, per year with the first two years rental paid in advance. Annual work expenditures of US$10.00 per acre in years one and two and US$20.00 per acre in years three through five are required. If a valuable mineral deposit is discovered, CSGM must apply for a mineral lease. State of Arizona Mineral Leases are issued for a maximum term of twenty years, which can be extended by production. A surface and mineral appraisal is required to determine lease rents and royalties. Most royalties are established at 5% to 6% gross royalty.

All fees, rents and taxes are current and paid in full. This was not independently verified by HRC.

Environmental Liabilities

The Project area has a long history of mineral exploration and CSGM does not anticipate barriers to work planned at the Project. In 1996, Harding Lawson and Associates ("HLA") completed a preliminary environmental overview for the Commonwealth Project. The HLA report identified the possibility of mercury contaminated soils in the area of the former location of the mill tailings from the old amalgam mill, but this was not considered an environmental liability at the time, nor is it now. The tailings were removed from the site in the 1970's and shipped off site for recovery of residual gold and silver values. The water well at the local school is tested twice a year and results have not exceeded applicable water quality standards. Soils in the area of the old tailings are highly impermeable and were developed from clay-rich volcanic rocks and the sediments from paleo-lake Cochise which covered the area during the Pleistocene. Because of this, groundwater contamination is considered unlikely.

Project Permitting

The Commonwealth Silver and Gold Project is located within a recognized historic mining area, with the mineral resource wholly located on patented mining claims and planned facilities located entirely on private ranch land. Because all of the lands potentially impacted by the project are private, the permit process should be limited to recognized and conventional permitting programs within the state of Arizona (no federal permitting required). Through the anticipated life of the operation, an Aquifer Protection Permit, Air Quality Permit, Mined Land Reclamation Permit and Stormwater Discharge Authorization will be required from the State of Arizona.

The major permits required to construct and initiate operations at the proposed mine are described in detail in Section 20 of this report.

Cultural Resources

In 1966, the National Historic Preservation Act ("NHPA") was passed so cultural resources would be protected from damage caused by the actions of federal agencies. As a result of the NHPA, the State of Arizona established the State Historic Preservation Act ("SHPA") in 1982 to protect cultural resources from the activities of state agencies. Cultural resources are identified as archaeological sites, historic buildings and structures, traditional cultural places, and other places or objects more than 50 years old that are considered important in Arizona state history. The State Land Department is responsible for managing cultural resources on State Trust Land to the extent necessary to comply with the SHPA.

Any development of portions of the Project located on State land permits triggers the SHPA. The first step in the process is a Class III cultural resource survey. If eligible sites are found, a cultural resource treatment plan is developed. Significant cultural resources are protected in place or recorded prior to their disturbance. It is unknown at this time whether the Project area contains eligible sites.

Endangered Species Act ("ESA") and other Biological Requirements

The ESA requires that the US Fish & Wildlife Service ("USFWS") identify species that are potentially at risk for extinction, evaluate available scientific information about the species, and (if warranted) list the species as either threatened or endangered. The USFWS is also required to designate "critical habitat" for listed species if prudent and determinable. A biological assessment/evaluation will need to be prepared for the Project in accordance with USFWS requirements. This report will address the likelihood of a listed species occurring on the Project area and the potential impact of the Project on those species.

If an endangered species is present and there is no federal nexus, a Habitat Conservation Plan must be developed by the Project proponent to list measures that will offset any harmful effects the Project may have on listed species. If there is a critical habitat designation that overlaps the Project area, there is no restriction on private development. Critical habitat only applies to federal actions. However, if a private action affects nearby critical habitat such as depleting surface water that is designated as critical habitat for an endangered fish, the federal government can restrict that activity in order to protect the habitat for the species.

National Environmental Policy Act ("NEPA")

The requirement to evaluate alternatives and impacts under NEPA will not apply unless a federal permit or federal funding is part of the Project. At this time it appears that the Project may be permitted under state and local laws and regulations with no federal permits that would trigger NEPA.

Other Significant Factors and Risks

HRC knows of no other significant factors or risks that may affect access, title, or the right or ability to perform work on the Commonwealth Project.

5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography

Topography, Elevation and Vegetation

The Commonwealth Project lies within the Basin and Range physiographic province that extends from approximately Nayarit state in Mexico in the south to northern Nevada in the United States in the north. The Basin and Range physiographic province is characterized by a series of broad alluvium filled valleys between relatively linear northwest-trending mountain ranges. The mountain ranges and valleys represent the horst and graben blocks, respectively, of a regional extensional fault system with large vertical displacement.

The Commonwealth Project is located in the Sulphur Springs Valley between the Chiricahua and Swisshelm mountains to the southeast and the Dragoon Mountains to the west. The valley is a closed basin characterized by internal drainage with no outlet to the sea. The lowest parts of the Sulphur Springs Valley, a few kilometers north of the Project area, are occupied by the completely barren Willcox playa, a dry lakebed with very saline soils. Areas in close proximity to the dry lake have relatively low plant diversity. Elevation within the Project area ranges from 1,300 to 1,450 m above sea level, and local terrain is generally treeless with low growing, salt tolerant desert plants of the Chihuahuan desert and semi-desert grassland life-zones. Characteristic plants include mesquite (Prosopis, Sp.), catclaw (Acacia gregii), ocotillo (Fouqueria splendens), creosote bush (Larrea tridentata), burroweed (Isocoma tenuisecta), a variety of yuccas (Yucca elata and others) and native grasses (Brown, D. ed., 1982).

Figure 5-1 Typical Physiography of the Project

Accessibility and Transportation to the Property

The Project area is readily accessible by paved State highways from Tucson, Willcox, and Douglas Arizona (Figure 5-5). A rail siding is available in Cochise, Arizona 19 km (12 miles) from the property. Access to the Project from Tucson is via Interstate 10 east 95 km (58 miles) to the Dragoon Road exit, then east 20 km (13 miles) on a paved two lane road to US 191, then south 16 km (10 miles) to Pearce, Arizona.

Local Resources and Infrastructure

Existing facilities at the Project site include a variety of portable trailers which have been effectively arranged to provide office space, core and equipment storage, a core layout and working area, and a segregated core cutting area. The trailers are arranged in two discrete but adjacent groups, one dedicated to core and activity associated with the main Project area and the other to house core from target exploration areas. The main group of trailers is surrounded by a chain link fence with locking gate, and the secondary group is secured by stout plywood walls and a locking door.

Figure 5-2 Core and Equipment Storage Facilities

Figure 5-3 Core Layout Area

Figure 5-4 Core Storage

Power

A 14.4 KvA powerline services the property with 60 amp service (Figure 5-5). Power for future development will require upgrading the powerline from the Apache Generating Station, which is located 19 km (11 miles) from the Project site. Sulfur Springs Valley Electric Cooperative (SSVEC) is planning on upgrading this line to 25 KvA in 2014, which will then provide ample power for the mine and site needs. It will be necessary to build approximately 1 mile of power line into the site.

Water

Water is currently obtained from a 6 inch diameter well located within 100 m of the north limit of the Ocean Wave Lode patented claim adjacent to the north property boundary of the Project (Figure 5-5). The well is 167 m (548 ft) deep. The standing water level in this well is reported as 98 m (321 feet) below the surface. The well is serviced by a 10 hp pump, and pump test results indicate available discharge of 25 gallons per minute at the depth of the pump. This water source is adequate for exploration and pre-feasibility Project activities. A larger water source will need to be developed for mining.

Mining Personnel

Trained personnel with experience in mining, heavy equipment operation, blasting, surveying, mill operation, etc. are readily available from local communities and throughout the greater regional area.

Climate and Length of Operating Season

The local climate in the vicinity of the Commonwealth Project is semi-arid with approximately 30 cm (12 inches) of precipitation annually. Most precipitation falls as heavy rain during the July to September monsoon season, with occasional light rain and snow in the winter months. Average annual snowfall is less than 5 cm (2 inches). Temperatures range from an average low of about 30˚ in the winter to summer highs above 38° C (100.4° F). The Project climate is ideal for a year round surface mining operation.

Sufficiency of Surface Rights

All mineralized material included in the mineral resource estimate presented in this report is located on mining claims controlled by CSGM. There is sufficient space for the process plant, leach pads and mine dumps on land for which there are private surface rights. Existing regional infrastructure in the vicinity of the Project area is depicted in Figure 5-5.

Figure 5-5 Existing Regional Project Infrastructure

6. History

The Commonwealth deposit was discovered in 1895 by John Pearce, who, while driving cattle over Pearce Hill, picked up an unusually heavy rock and decided to have it assayed. Pearce's rock ran 2,100 opt in silver (approximately 71,918 g/t), prompting him to locate six mining claims: the Ocean Wave, Commonwealth, One and All, Silver Crown, North Bell and Silver Wave. These claims comprise the heart of the modern-day Commonwealth Silver and Gold Project and are currently controlled by CSGM. Total production from the mine was approximately 138,000 ounces of gold and approximately 12 million ounces of silver from approximately 1,341,000 short tons (1,216,000 tonnes) of ore (Keith, 1973).

Between 1895 and 1927, extensive underground development totaling about 32 km (20 miles) of workings on 8 levels was completed at the Commonwealth Mine. Approximately 3 km (2 miles) of workings on 4 levels, mostly between C and D shafts, are accessible today. The mine extended down to the 8th level at an elevation of about 1,280 m (4,150 ft). Detailed maps were completed for the 2nd, 3rd, 5th, 6th, 7th and 8th levels by Smith (1927), and these maps provide an excellent basis for modern exploration work. After 1927, the mine was worked intermittently by lessors until 1942 when precious metals mines across the United States were shut down by The War Powers Act. There are no known production records for this period and tonnages are thought to be insignificant. By 1972, the Project was held by the Strong-Harris estate.

Prior Ownership

Carl Thetford and D. A. Corgill optioned the patented claims of the Commonwealth Project from the Strong-Harris estate in 1972, and exercised their option in 1977. Mr. Corgill and Mr. Thetford formed Cor-Ford, Inc. Corgill died in 1981 and Thetford exercised his option to buy out Corgill's interest. Mr. Thetford's heirs, by means of the Carl Thetford Family Trust, are the current underlying owners of a majority interest (80.5%) in the patented claims optioned to CSGM. A 5.0% interest is held by Mordecai Thetford, a 2.5% interest by The Spira Family Living Trust and 10.0% is owned by CSGM.

Previous Exploration and Results

From 1975 to 1976 Platoro Mines ("Platoro") held by Paul Eimon, et al., sought out the Commonwealth property for acquisition under the direction of Mr. Eimon, a noted silver expert. Platoro directed extensive exploration (including 5 percussion drillholes) and identified an eastern vein extension with indications of a possible bulk tonnage silver deposit. Limited metallurgical work showed favorable recoveries. Lacking funds, Platoro subleased the property to Bethlehem Copper Co. ("Bethex") in 1976. Bethex proceeded to drill 22 diamond core and percussion holes, mostly on the east side of Pearce Hill (Figure 6-1), in an attempt to identify a bulk silver target.

Bethex' sample results were poor and included some negative indications, causing them to lose confidence in the property and drop the sublease with Platoro. Subsequently, Platoro allowed the underlying lease to lapse. Western States Minerals ("WSM"), also seeking a bulk silver target, acquired the lease in 1977 and drilled 16 reverse circulation holes on the property between 1977 and 1978. WSM

located a zone with higher gold content on the west side of Pearce Hill, but allowed their lease to lapse after a plane crash killed several members of its exploration group and management in late 1978.

Cor-ford, Inc. held the Project between 1979 and 1982. When silver prices rose sharply in the early 1980's, Mr. Thetford's heirs shipped US$3.0 million in tailings to nearby smelters. Smelter records indicate that operations were quite profitable for many months. In 1981 Geo-Hendricks ("GH"), a successor to Platoro's remaining interest in an adjacent property, tried to locate the Commonwealth vein extensions as postulated by a mid-1970's Master's thesis. GH drilled 14 holes to shallow depths, only a few feet into bedrock, outside of the main Project area, but the holes were barren.

Alpine Resources leased the Commonwealth property in 1983. Alpine conducted geologic mapping and some sampling, but was unable to raise sufficient funding to maintain the lease beyond 1984.

Between 1985 and 1986 Santa Fe Pacific Mining, Inc. ("Santa Fe") drilled 5 holes, 4 of which were on the eastern extension of the North Vein, well away from the main mine area. The one hole drilled to the west was on an owner-dowsed site. Metallurgical work on high quartz, silver vein material was poor. Santa Fe was apparently testing a caldera rim model which failed to meet their specifications.

DRX Inc. ("DRX") conducted the first modern exploration work of real significance at the Project between 1988 and 1992. DRX drilled 55 holes to test the western high gold zone and central and eastern high silver areas. They reopened old workings, conducted extensive channel sampling, and completed some metallurgical work. They developed a (non-NI 43-101 compliant) mineral reserve estimate for gold and silver, and tried but failed to finance a small heap leach operation for the western gold zone.

DRX failed financially in 1992 after attempts to lease or option the Project to Placer Dome, Inc. ("Placer"), ASARCO, Inc. ("ASARCO"), Glamis Gold, Ltd. ("Glamis") and WSM. In 1990, Placer conducted a brief exploration and metallurgical testing program, but failed to follow through, likely due to the perceived limited size of the deposit and onerous terms from DRX. During 1991, ASARCO conducted a more extensive exploration drilling program with abundant check sampling that confirmed the results of previous sampling. The results of metallurgical testing conducted during ASARCO's exploration were generally positive, but the company opted not to continue after 1991.

Glamis drilled 13 holes and conducted metallurgical work in 1991. The majority of the Glamis drillholes were drilled into the Bisbee Group seeking a postulated, bedded Bisbee target. Drilling encountered widespread gold but did not reveal a continuously mineralized zone in the Bisbee Group. DRX's terms and the lack of well-defined, continuous mineralization in the Bisbee led Glamis to discontinue exploration efforts. In 1992, WSM reviewed the Project and again subleased the property from DRX. WSM completed 11 drillholes and conducted an extensive check assay program. Although meetings held with the local population reportedly encountered no resistance to mine development, assumptions about average grade and expected deposit size led WSM to terminate their involvement in the Project.

DRX disbanded in 1993, and one of the company officers formed Columbia Resources ("Columbia") and optioned the property from Mr. Thetford's heirs. Reports were compiled, but no significant work was completed. Later in 1993, Consolidated Nevada Goldfields, Corp ("CNG") took over the lease and option from Columbia, and completed more metallurgical work. CNG established that near surface mineralization was partially refractory to cyanidation, but that this could be mitigated by finer crushing. They were then forced to terminate operations due to a bankruptcy reorganization and management change. After CNG dropped out, Pegasus Gold ("Pegasus") optioned the Project from Columbia and drilled 5 reverse rotary holes targeting narrower widths of high grade gold and silver mineralization. Two holes were lost in old mine workings, and drilling revealed a lack of widespread low grade mineralization in the hanging wall, although recovery was sometimes poor.

Harvest Gold Corp. ("Harvest") optioned the property from Columbia in 1994. With a limited budget, they pursued data acquisition, a limited drilling program and a large metallurgical work program with Kappes Cassiday and Associates ("KCA"). The drilling program was terminated early due to poor performance by the drilling contractor. All data was stored within a TECHBASE database. Harvest then optioned the property to Atlas Precious Metals, Inc. ("Atlas") under a one year purchase of assets option agreement to be completed between 1995 and 1996.

Under the option agreement between Atlas and Harvest, Atlas was required to complete the following:

- Compile all previous works on 1:600 (1" = 50') scale maps after extensive resurveying;
- Conduct a successful check assay program;
- Re-habilitate the collar of the "D" shaft;
- Collect 565 ft (172 m) of surface and 2,110 ft (643 m) of underground channel samples, including new discovery sampling on the old 7th mine level;
- Conduct numerous bottle roll and 14 column leach tests with KCA (some to 180 days);
- Infill drilling to include 26 reverse rotary holes and 4 HQ core holes;
- Prepare a geotechnical report through Call & Nicholas, Inc.;
- Prepare a petrographic review of the mineralized material;
- Prepare an environmental overview report through Harding Lawson Associates, which included an environmental geochemistry program; and
- Prepare a Med System mine model of the deposit.

Atlas completed the prescribed work program and the results indicated a potentially viable mineral reserve (non-NI 43-101 compliant), but the required purchase price and Atlas' low stock price and inadequate cash position caused them to drop the option.

In 1996 the property was returned to Harvest who dropped the property after failing to raise sufficient funds to advance the Project. Between 1996 and 2010, the property was maintained in good standing by the Thetford Heirs, but no exploration or development work was carried out. A map of the historical drillhole locations is presented in Figure 6-1.

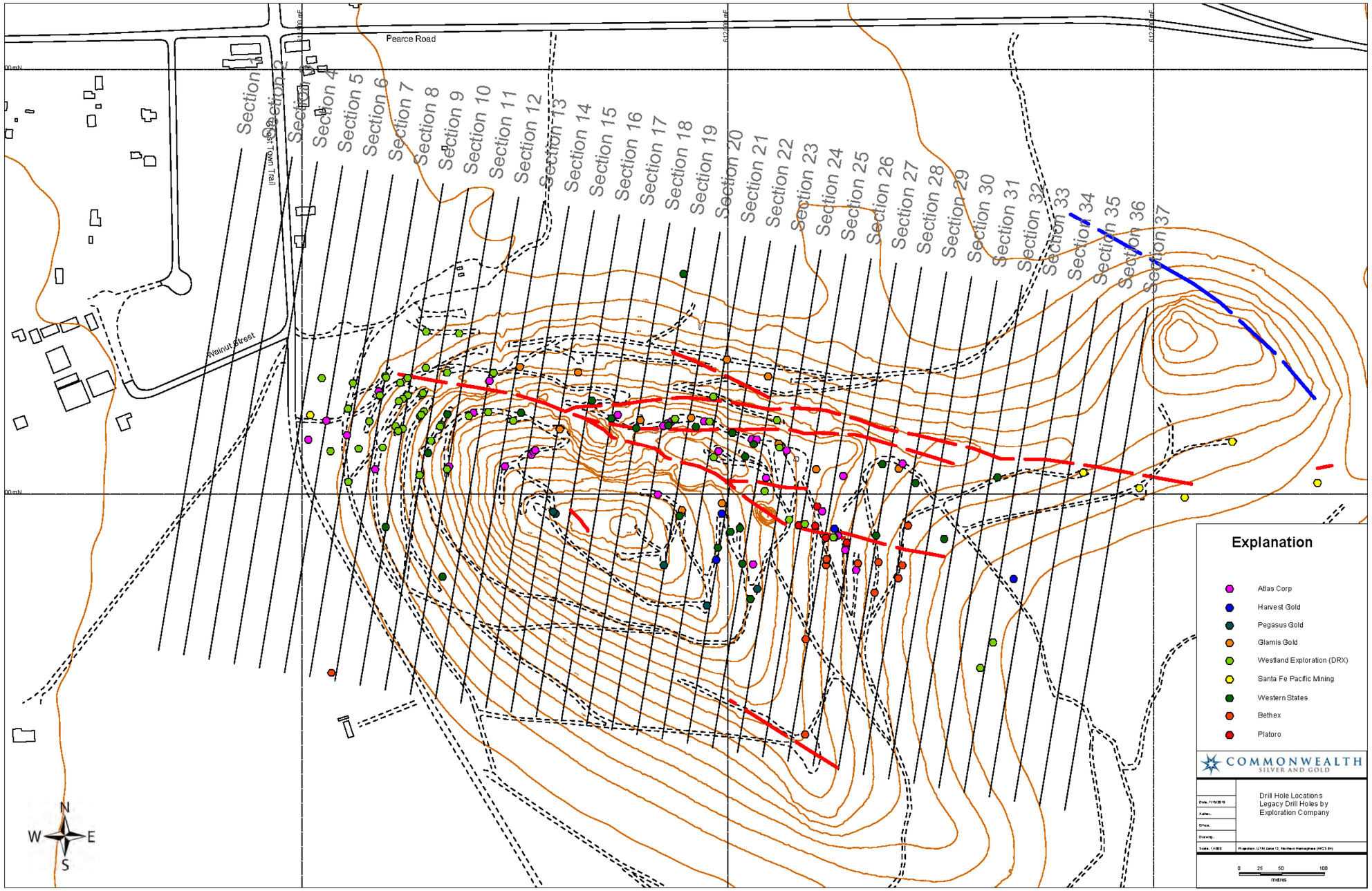


Figure 6-1 Historical Drillhole Location Map

Between 1989 and 1991, DRX completed 141 underground channel sample lines totaling 2182 meters and between 1995 and 1996 Atlas completed 68 surface and underground channel sample lines totaling 803 meters. The underground channel samples have been included in the resource estimate. These samples by range are listed in Table 6-1. Appendix C lists all available drillhole and channel sample information.

Table 6-1 Surface and Underground Channel Samples

Date	Hole Identification Range	Exploration Company	Type	Number	Reported/Actual Meters
1990-1991	CHN - series	DRX, Inc	UG Channel	141	2,180.77
1995-1996	SCHN - series	Atlas Precious Metals	Surface Channel	68	803.19

Historic Mineral Resource and Mineral Reserve Estimates

A number of historic, non-NI 43-101 compliant mineral resource estimates were completed in the 1990's. These resource estimates pre-date current NI 43-101 reporting standards, and the associated resource models, electronic or otherwise, are not available for verification. Although they were completed using industry best practices at the time, these mineral resources are not classified using current CIM definition standards, are not presented according to modern reporting codes, are not considered reliable, and are included here for historical completeness only.

Table 6-2 Historical Non-NI 43-101 Compliant Mineral Resource Estimates

Item	Nevada Goldfields 1992	Atlas Precious Metals 1995	Harvest Gold (Behre Dolbear) 1995	Geostat Systems Inc. 1997
Cut-off (opt) AuEq - 75:1	0.015	0.012	0.019	varies
Total (short) tons	6,200,000	9,523,071	7,549,952	16,002,000
Gold Grade opt	0.031 (AuEq)	0.023	0.026	0.025 (AuEq)
Silver Grade opt	n/a	1.54	1.85	n/a
Gold Equivalent oz	186,620	256,480	382,430	401,572
Method	Sectional	IDW	Kriging	Kriging
Strip Ratio	2.8:1	2.54:1	3.38:1	1.18:1
Computer modeling	---	MedSystem	Techbase	Geostat

These resource estimates were conducted prior to 2000 and pre-date NI 43-101 reporting standards. These estimated mineral resources are not classified using CIM Definition Standards – For Mineral Resources and Mineral Reserves and are not equivalent to current reporting codes. A Qualified Person has not completed sufficient work to classify these estimates, and CSGM is not treating, the historical estimate as the current mineral resource. The resource models electronic or otherwise are unavailable for validation and it cannot be confirmed that the estimates were conducted using industry best practices. The historic mineral resources are not considered current or reliable, and are not used in any capacity with regard to the mineral resource estimate presented in Section 14 of this report.

Historic Production

The Arizona Department of Mineral Resources reports production from the Pearce District from 1895 to 1942, virtually all of which is attributed to the Commonwealth Mine, at 1,341,000 short tons (1,216,000 tonnes) containing 12,020,000 ounces of silver and 138,409 ounces of gold (Keith, 1973).

7. Geological Setting and Mineralization

Regional Geology

The Commonwealth Project lies near the northwestern limit of the Sierra Madre Occidental, a northwest trending volcanic plateau composed of thick accumulations of andesitic to rhyolitic volcanic rocks. The rocks of the Sierra Madre Occidental are generally thought to reflect subduction-related continental arc magmatism that slowly migrated eastward during the early Tertiary and then retreated westward more quickly, reaching the western margin of the continent by the end of the Oligocene (Sedlock et al., 1993). The eastward migration is represented in the Sierra Madre Occidental by the Late Cretaceous-Early Tertiary lower volcanic series ("LVS") of calc-alkaline composition. Over 2,000 m of predominantly andesitic volcanic rocks with interlayered ash flows and occasional intrusions comprise the LVS.

The westward retreat of the subduction-related continental arc magmatism is represented by rhyolitic ignimbrites and flows, with subordinate andesite, dacite, and basalt, which were all deposited in conjunction with caldera activity throughout the Sierra Madre during the Eocene and Miocene. These volcanic rocks unconformably overlie the LVS andesitic rocks and constitute the "upper volcanic supergroup" of the Sierra Madre Occidental (Sedlock et al., 1993). The upper volcanic supergroup is also commonly referred to as the upper volcanic series ("UVS"). Gold and silver deposits in the Sierra Madre Occidental are commonly hosted in rocks of the LVS and the mineralizing systems are thought to be related to heat sources active during the ignimbrite flareup, which resulted in the deposition of the UVS (Sedlock et al., 1993: Camprubi, 2003). A geologic map of Cochise County, Arizona is presented in Figure 7-1.

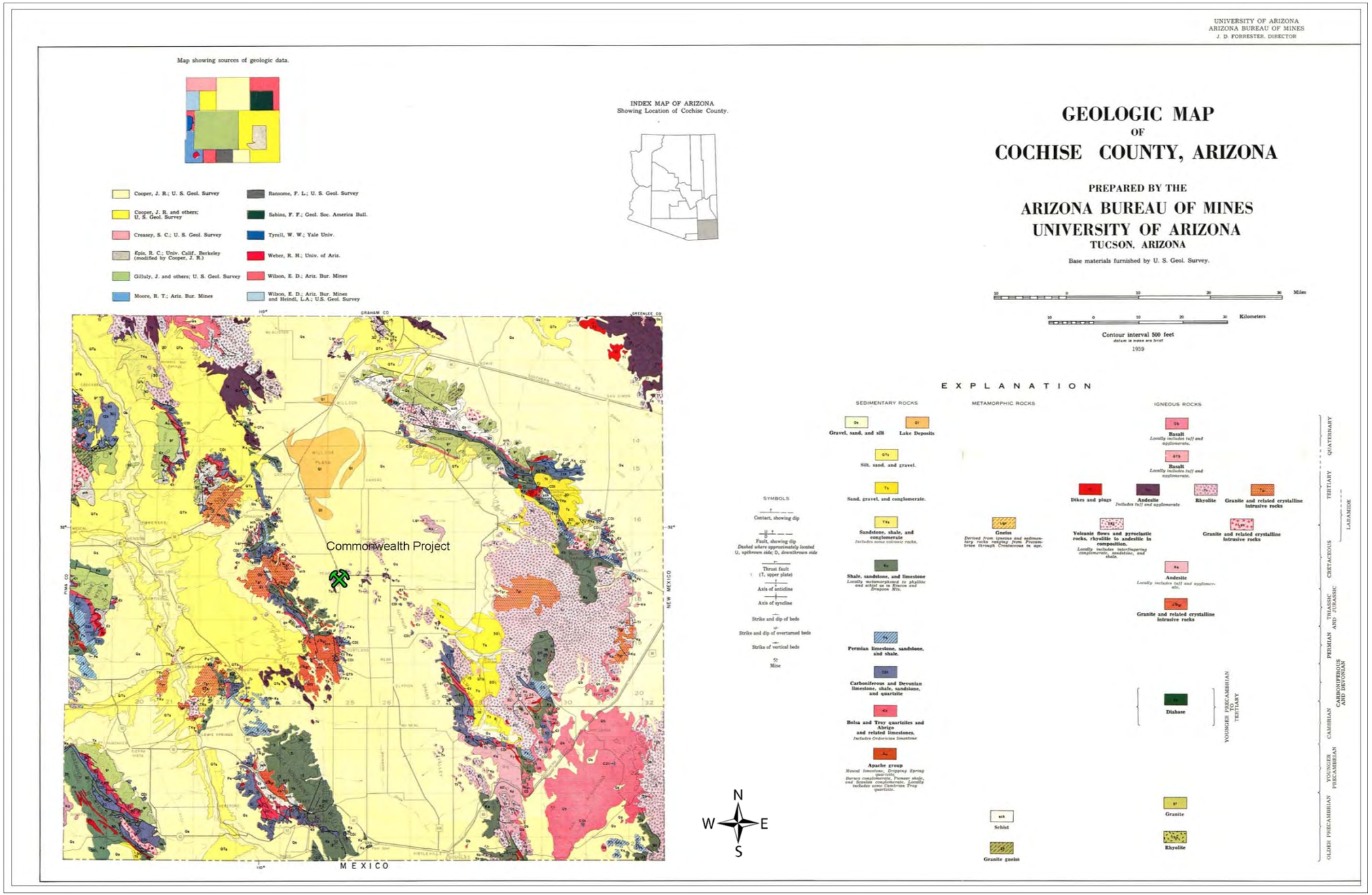


Figure 7-1 Commonwealth Silver and Gold Project Regional Geologic Map

Local Geology

The Pearce Hills and other low hills in the Sulphur Springs Valley lie south of the Willcox Playa between the Dragoon Mountains to the west and the Dos Cabezas, Chiricahua and Swisshelm Mountains to the east and southeast. The surrounding ranges are underlain by rocks as old as Proterozoic age (the Pinal Schist) as well as thick sections of Paleozoic sediments, Mesozoic sediments and volcanic rocks. In the Dos Cabezas Mountains there are large masses of Proterozoic igneous and metamorphic rocks. A large mass of Late Cretaceous volcanic breccia crops out in the core of the Dos Cabezas Range and there are also several quartz monzonite intrusions associated with the Laramide orogeny (Davis, et al., 1982). The Chiricahua Mountains are dominated by mid-Tertiary ignimbrites of rhyolite composition. These ignimbrites were erupted from the Turkey Creek caldera between 30 and 24 mya (Davis, et al., 1982). They are correlative in age with the UVS ignimbrites throughout the Sierra Madre Occidental of Mexico.

The large ranges of the Dragoon Mountains, Swisshelm Mountains and Chiricahua Mountains are bounded by northwest trending Basin and Range age faults. The down-dropped Sulphur Springs Valley between the ranges has been covered with alluvium eroded from the surrounding mountains. The altered and mineralized hills of the Pearce Mining District are topographic highs that extend through this alluvial cover. The Pearce Hills are composed of a complex of Tertiary volcanic flows, welded tuffs and pyroclastic rocks that were extruded over a platform of Cretaceous and older sediments. The volcanic units can be divided into two major groups tentatively correlated with the LVS and UVS, which make up the Mexican Sierra Madre Occidental.

Rocks of the lower portion of the Pearce stratigraphic section are similar to the LVS both in composition and timing of deposition. The lower portion of the Pearce stratigraphic section is composed of andesite flows and interbedded felsic volcanic breccia. These rocks likely correlate with the Rillito Formation near Tucson with an age of about 39 my, or with the Nipper Formation in the Chiricahua Mountains, at an age of 32 my (Davis, et al., 1982). This correlation is based on visual similarities of texture and composition as well as stratigraphic position.

A series of rhyolite flow domes and associated welded ash flow tuffs intrude and overlie the earlier rocks and are likely related to the Turkey Creek caldera located on the west flank of the Chiricahua Mountains. The flow domes may have served as the heat sources, which drove the convecting hydrothermal systems responsible for the mineral deposits in the Commonwealth Project area. The Faraway Ranch Formation in the Chiricahuas yields age dates between 29.7 and 28.3 my (Davis, et al., 1982). These felsic rocks are correlative in age, composition and style of extrusive volcanism with the rocks of the Mexican UVS. Contemporaneous with the emplacement of UVS rocks are low temperature, epithermal precious metals deposits formed throughout the Sierra Madre Occidental including the deposits in the Pearce Mining District.

Rock units throughout the Pearce Mining District are tilted fairly uniformly at 10˚ to 40° to the northeast. The hills are block faulted into a series of half-grabens, generally with the western side of the fault block down-dropped. Mineralized structures in the Pearce Mining District probably formed near

the onset of the extensional tectonic setting that formed the Basin and Range. However, most Basin and Range extension, which occurred from about 25 to 12 mya, appears to post-date mineralization in the Commonwealth Project area.

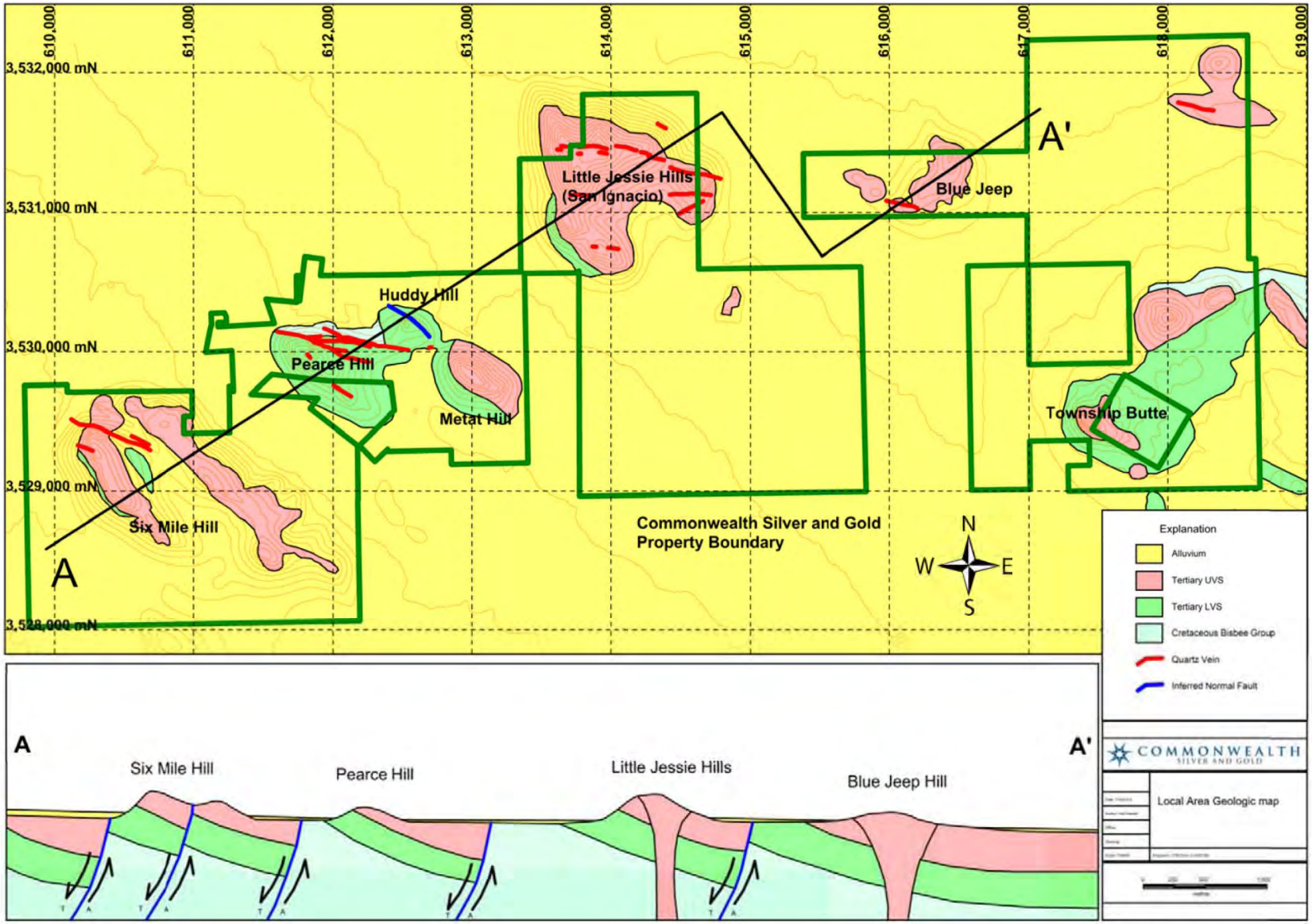


Figure 7-2 Local Geology

Property Geology

Host Rocks and Stratigraphy

Cretaceous

The oldest rocks in the Commonwealth Project area are sediments assigned to the Cretaceous Bisbee Group. These sediments were most likely deposited in a near shore marine/deltaic environment. The rocks observed in drill core and in outcrop are moderately well sorted and well-rounded calcareous sandstones and poorly sorted muddy calcareous siltstones. Thin beds of pebble conglomerate also occur within this unit. At Blue Jeep, two miles east of the main Project area, the Bisbee Group includes limestone beds. The Bisbee Group sediments are relatively soft and easily eroded forming slopes rather than resistant outcrops. Exposures of the Bisbee Group sediments are limited to outcrops on the north side of Pearce Hill in the footwall of the North Vein and outcrops at Blue Jeep and Township Butte. Rocks of the Bisbee Group are well exposed in the underground workings on and below the third level, and also in the drill core. Where unaltered, the sandstones are light grey to white and the siltstones and mudstones are a chloritic green colour, except where oxidation near veins has resulted in a rusty red hue.

Tertiary (Oligocene or older)

LVS rocks occur in the Pearce area as a series of andesite and rhyolite flows and pyroclastic units just a few hundred meters thick. Together, three discrete volcanic units (a lower andesite, a rhyolite breccia, and an upper andesite) and overlying volcaniclastic sandstone are known locally as the Pearce Volcanics. These mostly andesitic rocks are the principal hosts of mineralization on the Commonwealth property, and are tentatively correlated with either the Rillito andesite of the Tucson Basin or the Nipper Formation of the Chiricahua Mountains (Davis, et al., 1982).

Lower Andesite

The lower andesite (Tal), also referred to as Tf1 in Howell (1977) or earlier andesite in Smith (1927), lies unconformably over the Bisbee Group sediments and is estimated at about 50 meters thick. The unit crops out on the west side of Pearce Hill and on the west side of Huddy Hill (Figure 7-2). The lower andesite has a fine grained medium brownish grey groundmass with 5%, 0.5 to 3.0 mm plagioclase phenocrysts. The unit fractures well and is a favorable host for mineralization at the Project.

Rhyolite Breccia

The rhyolite breccia (Trb), also referred to as Ta1 in Howell (1977) or earlier breccia in Smith (1927), overlies the lower andesite and is locally separated from it by an unwelded, possibly waterlain tuff that ranges from 1 to 2 m in thickness. The rhyolite breccia is approximately 60 m thick and forms blocky ledges in outcrop high on Pearce Hill and at the top of Huddy Hill (Figure 7-2). The rhyolite breccia is a welded crystal-lithic ignimbrite with cognate fragments of similar rhyolite and accessory fragments of andesite (possibly derived from the lower andesite unit). The lithic fragments range in size from fine ash

to 10 cm. Phenocrysts of both quartz and clear sanidine feldspar are abundant. Eutaxitic compaction features are common in the upper part of the unit, while the lower part of the unit is welded but shows less compaction. The unit fractures well and broad zones of mineralized stockwork veining are well developed many meters away from principal veins.

Upper Andesite

The upper andesite (Tau), also referred to as Tf2 in Howell (1977) or Middle andesite in Smith (1927), overlies the rhyolite breccia. The upper andesite has the largest areal extent of all of the units exposed on Pearce Hill, covering the summit and eastern and southern slopes of the hill. The upper andesite is thought to be up to 150 m thick, but a complete section has not been observed. The unit forms rubbly outcrops where autoclastic breccia textures are observed. The upper andesite has a fine grained dark grey groundmass and 5%, 0.5 – 1.0 mm plagioclase phenocrysts and 2%, 0.5 – 1.5 mm hornblende phenocrysts. The unit is usually magnetic and this characteristic and the presence of hornblende serve to distinguish it from the lower andesite unit. Local stylolites are observed in the upper andesite, probably parallel to the direction of flow. A characteristic texture developed by weathering of the hornblende phenocrysts is locally known as “turkey track” for the molds left in the andesite once the hornblende is gone.

Volcaniclastic Sandstone

The volcaniclastic sandstone (Tss), also referred to as Tw3 in Howell (1977), is a coarse lithic arenite which varies in composition from conglomerate to coarse sandstone. It is generally weakly cemented with calcite and is easily distinguished from the Bisbee Group by its coarse grain size and weaker reaction to HCl. This unit crops out on the west flank of Metat Hill and on the south side of the San Ignacio hills (Figure 7-2), and separates the LVS from the UVS. The volcaniclastic sandstone serves as a marker bed at the top of the Pearce Volcanics and correlates with the basal volcaniclastic unit of the Faraway Ranch Formation in the Chiricahua Mountains. The Tss unit crops out in the central valley of Six Mile Hill and on the west flank of the Little Jessie Hills.

Mid-Tertiary (Oligocene – Miocene)

UVS rocks deposited during the mid-Tertiary ignimbrite flare-up overlie the Pearce Volcanics across the full extent of the Pearce mining district. Deposition of the UVS may have provided the heat to drive the hydrothermal systems responsible for mineralization in the main Commonwealth Project area as well as at Blue Jeep, Township Butte and Six Mile Hill (Figure 7-2). A rhyolite welded tuff in the Faraway Ranch Formation (unit 3) in the Chiricahua Mountains has been dated at 29.7 my, and an overlying rhyodacite (unit 7) in the same Faraway Ranch Formation has been dated at 28.3 my. Cochise County is intruded by a number of Tertiary stocks of similar age, including quartz monzonite in the Swisshelm Mountains (31 my), granodiorite in the northwestern Chiricahua Mountains (32 my), and the Ninemile granodiorite in the Dos Cabezas range (29 my) (Davis, et al., 1982).

Rhyolite Ash Flow Tuff

Ash flow tuff (Taf) is referred to by Howell (1977) as Ta3. This unit caps Metat Hill and the Little Jessie Hills (Figure 7-2) and is similar to rocks in the Six Mile Hill area. The ash flow tuff is similar to and probably correlates with unit 3 of the Faraway Ranch Formation. The rock is a dense welded rhyolite tuff with pumice fiamme, quartz phenocrysts and local vitrophyric textures. A well-developed eutaxitic compaction foliation is usually present. Secondary chalcedony is found in lithophysae.

Other Units

Basalt (Tb), also referred to as Tf3 by Howell (1977), occurs as fine grained, black, scoriaceous to amygdaloidal basalt flows, which cap the Taf unit. In the Pearce hills these flows are found on the east side of Metat Hill and are only a few feet (less than 2 meters) thick. At Six Mile Hill the basalt is over 61 m (200 ft) thick and has interbeds of obsidian. The basalt flow is similar to unit 4 of the Faraway Ranch Formation in the Chiracahua Mountains.

White rhyolite (Twr) is part of the rhyolite flow dome complex and a rhyolite dike unit crops out on the Blue Jeep hill and on the Little Jessie hills in the San Ignacio area.

Lake bed sediments (Qlb) in the form of caliche-rich sandstone and conglomerate with gypsum beds are locally present at Township Butte. These sediments were deposited in the paleo-lake Cochise. Figure 7-3 shows the Project Geology in plan view and Figures 7-4, 7-5 and 7-6 show cross sections through the deposit at Sections 11, 22 and 27 respectively.

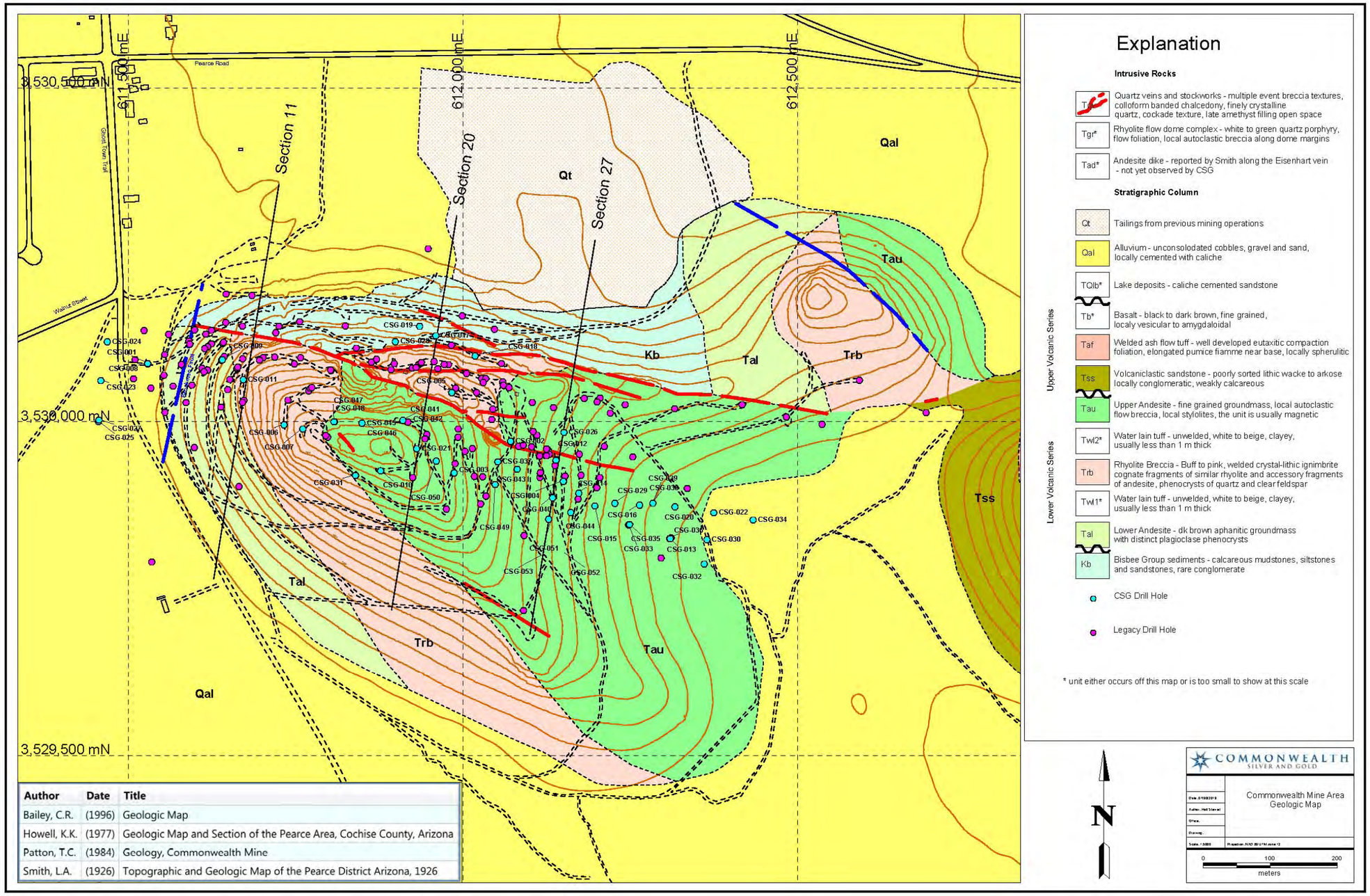


Figure 7-3 Commonwealth Silver and Gold Project Geology Map

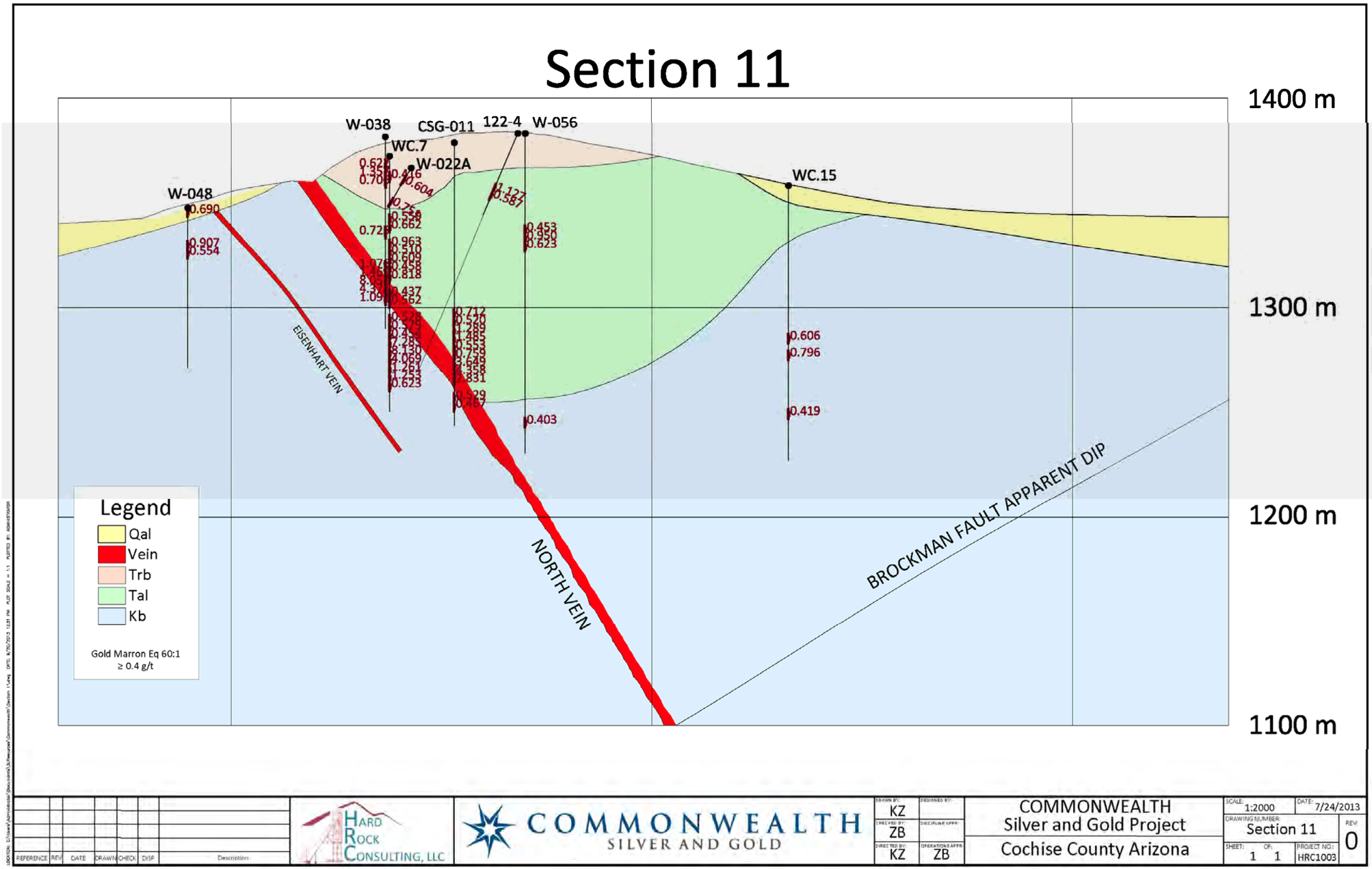


Figure 7-4 Section 11 Showing Lithology Interpretation and North Vein Near the Brockman Fault

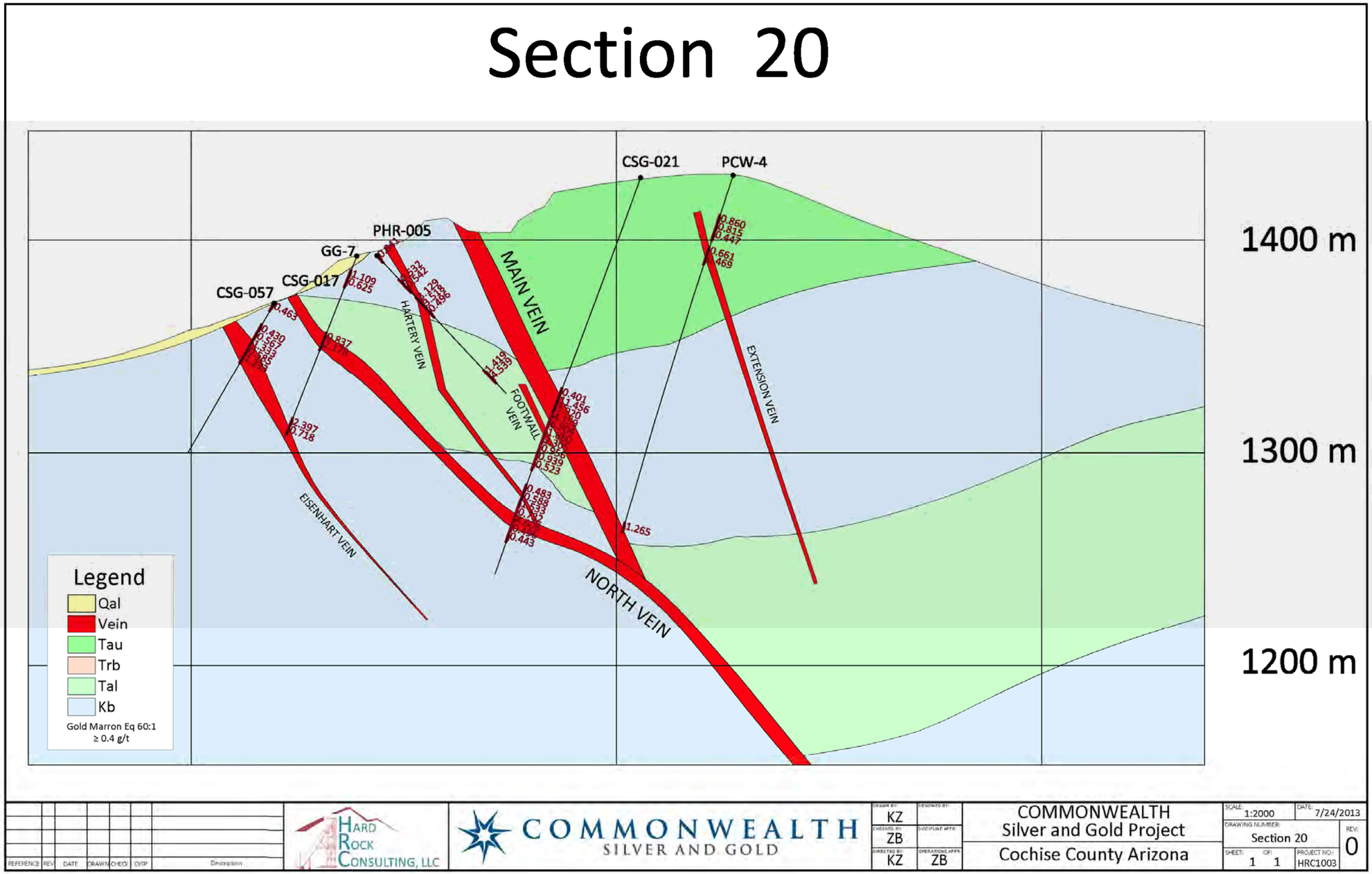


Figure 7-5 Section 20 Showing Geologic Interpretation and Mineralized Wedge between North and Main Veins

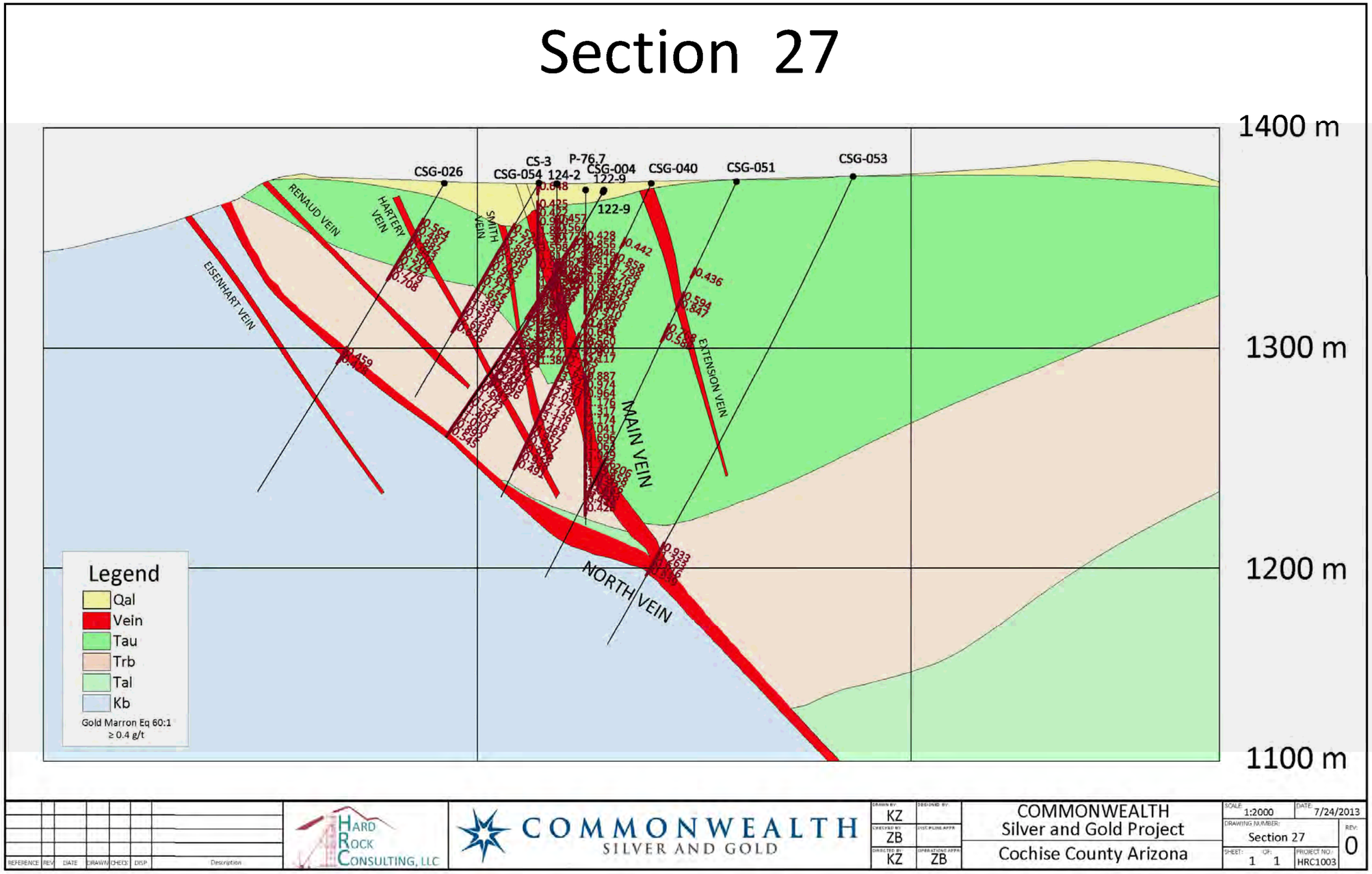


Figure 7-6 Section 27 Showing Geologic Interpretation and Multiple Vein Splays within the Mineralized Wedg

Structure

All of the significant structure in the Post-Cretaceous rocks is attributed to Basin and Range extension during the Miocene and Oligocene. The earliest structures in the Commonwealth Project area are represented by the N85°W striking, south dipping North Vein and the related hanging wall split or extension fracture of the N70°W striking, southwest dipping Main Vein. A number of other associated veins split off the Main and North Veins, and at least seven named veins were mapped during operation of the Commonwealth Mine (Smith, 1927). The veins represent normal faults with a component of right lateral displacement. Three-dimensional modeling of the stopes on the Main Vein reveals that this vein is emplaced along a major structural flexure related to right lateral movement. This flexure developed a dilatant zone favorable for deposition of vein minerals 250 m long and up to 20 m wide. It is likely that the structures that host the veins formed near the beginning of Basin and Range extension. Structures with similar orientations, widths and vein fill occur at Blue Jeep, San Ignacio and Six Mile Hill.

All pre-Quaternary rocks in the Project area have been faulted into blocks a few kilometers long by a kilometer or so wide oriented northwest/southeast. Rocks within each block have northwesterly strikes of N25°W to N55°W and dip 10˚ to 40° northeast. The block faulting that affected these rocks probably occurred after the Project area veins had been emplaced, rotating them into their present orientation. The character of the vein structures is dependent on the host rock lithology. The Main Vein, Smith Vein, Renaud Vein, and others hosted in the volcanic rocks have generally steep dips and intensely shattered margins extending several meters into the surrounding rock. The North Vein, which has Bisbee Group sediments in the footwall for the majority of its exposed strike, has more gouge developed in the footwall than the Main Vein, which occurs in the volcanic rocks. Stockwork veining is poorly developed in the Bisbee, but well developed on the hanging wall side of the North Vein in volcanic rocks.

The Main and North Veins diverge from their intersection near the west end of Pearce Hill. While the separation between the veins increases towards the surface, the veins still coalesce at depth along a shallow east plunging juncture or keel. A mineralized wedge of volcanic rocks between the North Vein and the Main vein is densely fractured to brecciated, and this crushed zone has served as a favorable locus for development of quartz veins and stockworks. The shattered wedge of rocks is triangular in cross-section and varies from a few meters wide at the intersection of the Main and North Veins to well over 100 m wide on the east side of Pearce Hill.

A number of post-mineralization faults with minor displacement formed during later Basin and Range tectonics. These faults include the Brockman fault to the west of the Pearce Hill and the Knox fault just east of D shaft. These post-mineral faults are exposed in the underground workings and have approximately north-south strike and vertical dip. The latest structures in the Project area, inferred from truncation of outcrops at a strike of roughly N40°E, pass between Huddy Hill and Metat Hill and downdrop the rocks on Metat Hill several tens of meters.

Alteration

Potassic alteration in the form of replacement of plagioclase feldspar with adularia is widespread, along with silicification in and within several meters of the mineralized veins. In drill core this is easily recognized in the Trb rhyolite breccia, but is difficult to identify in the andesitic rocks. Veinlet adularia is abundantly present in areas of early quartz and calcite veining as demonstrated by extensive studies of stained thin sections in a mid-1970's master's thesis (Howell,1977). Potassic zones grade laterally into sericitic material, and potassic stable envelopes range over 150 m (500 ft) vertically.

Moderate argillic alteration is present, generally between major veins in the underground workings. This argillic alteration is zoned from sericite in and immediately adjacent to the veins to yellow-green smectite distal to the veins and as late, cross cutting zones of clay. Argillization has been found between, above and below areas of previous mining. Argillic alteration evidenced by bleaching of the volcanic rocks is present over hundreds of square kilometers surrounding the Project area. This is observed in aerial photography and is considered an indication of a large, long-lived hydrothermal system. Weak regional propylitic alteration in the form of chloritzation and epidote rims on feldspars is also observed in areas distant from the veins.

Significant Mineralized Zones

The mineral deposits within the Commonwealth Project area are typical of silver dominant, low sulfidation, epithermal veins and stockworks emplaced in a near surface environment. The veins are best developed in the andesite to rhyolite units of the Pearce Volcanics. These volcanic rocks fracture well, are densely fractured to shattered, and host dense quartz stockworks, breccia zones and banded quartz veins. Cretaceous marine sediments of the Bisbee Group also host mineralization and are chemically favorable hosts. The calcareous sandstones and siltstones of the Bisbee Group are very similar to the "dirty carbonate" host rocks of many sediment hosted disseminated gold deposits in Nevada and in northern Mexico. These rocks seem to be especially favorable hosts for gold mineralization and there is a higher Au:Ag ratio in the assay results from mineralized Bisbee Group samples as compared to the mineralized volcanic rocks at the Commonwealth Project. The Bisbee Group sediments are soft enough that they do not fracture well on faulting. Mineralization within the Bisbee Group sediments occurs as both vein type mineralization and some disseminated mineralization.

At least seven major quartz veins were actively mined in the Commonwealth Mine between 1895 and 1927. The two most important veins are the Main Vein and the North Vein. The Main Vein strikes approximately N70°W and dips 65 to 85°to the southwest. The Main Vein is interpreted to be an extension fracture related to the North Vein, which has greater fault displacement. The Main Vein ranges from 3 m to over 8 m in true width and is composed of quartz cemented breccias, colloform banded very fine grained quartz, and chalcedony. Characteristic green chalcedonic quartz locally termed "talc quartz" is common in the Main Vein. This green color may be due to finely disseminated embolite (silver chloro-bromide). Late events within the Main Vein exhibit comb and cockade textures and drusy open space filling. Some of the late quartz is amethyst. The North Vein strikes N80°W to

N90°W and dips 45 to 65° to the southwest. The vein is up to 15 m wide and is characterized by abundant white colloform banded chalcedony and banded grey silica, which is almost opaline in texture.

Much of the mineralization on the Commonwealth Project occurs in the silicified and shattered structural wedge between the Main Vein and the North Vein. This mineralized zone is at least 550 m long and expands eastward from a point where the veins coalesce reaching a mineralized exposed width of 125 m before extending beneath alluvial cover. The covered extension of the mineralized wedge is intersected in holes CSG-022, CSG-030, CSG-032, and CSG-034 and is open to the east and below drilling.

Subsidiary veins in the mineralized wedge that were named by Smith (1927) are, from the footwall of the Main Vein and proceeding north, the Footwall Vein, the Fischer Vein, the Smith Vein, the Hartery Vein and the Renaud Vein. Each of these veins varies from 1 to 4 m in width, with the Renaud Vein being the widest. These veins are generally sheeted veins sub-parallel to the Main Vein. They may also be considered the thick, high fluid flow arteries within the stockwork zone between the Main and North Veins.

Figure 7-7 Surface exposure of the North Vein

Only one vein has been identified in the footwall of the North Vein. The Eisenhart Vein strikes 285° and dips 65 to 75° to the southwest. Smith (1927) recorded that the Eisenhart Vein is emplaced along the footwall of an andesite dike intruding into the Bisbee Group sediments, and can be observed both at ground surface and in underground workings. Two veins, each less than one meter wide, occur in the

hanging wall of the Main Vein, and work is planned to evaluate the continuity of and possible mineralization in these veins.

Ginguro bands are observed along vein margins and as bands within the Main Vein. All ginguro minerals have been oxidized to hematite, limonite and unidentified silver minerals. Most of the historic mined mineralized material from the Commonwealth Mine had abundant cerargerite (silver chloride) and embolite (silver chloro-bromide). Deep, post mineral oxidation of the veins has occurred and the sulfide zone has not yet been encountered in drilling. A number of sub-horizontal zones of gold and silver enrichment were recorded by Smith (1927), who correlated these enrichment zones with paleo-levels of Lake Cochise. Since the base of oxidation has not been reached, deeper zones of supergene precious metal enrichment may exist.

8. Deposit Types

The conceptual geologic model that best applies to the Commonwealth Silver and Gold Project is a variation of the rift, low sulfidation, epithermal chalcedony-ginguro model of Corbett (Corbett, 2002). The characteristics of this model are mineralogy derived primarily from dilute, near neutral pH fluids, an extensional, dilatant structural setting, competent host rocks that fracture well, and abundant banded chalcedonic quartz.

The Commonwealth deposit is situated within a highly mineralized belt that extends north from Mexico City (and possibly further south), along the Sierra Madre Occidental and into southern Arizona, and is typical of Tertiary age, low sulfidation, epithermal precious metals systems found throughout northwestern Mexico and the southwestern United States.

There are many similarities between the Commonwealth deposit and other silver dominant epithermal systems in Mexico such as Ocampo, Chihuahua, Palmarejo, Chihuahua, Fresnillo, Zacatecas and others. Age dates from the Commonwealth Project area are consistent with ages of mineralization at Ocampo, San Francisco del Oro, Bacis, Velardena, Guanajuato and other epithermal precious metals districts in Mexico. Other similarities include the host rock setting near the contact between the LVS and UVS, quartz textures including finely crystalline, low temperature quartz and chalcedony with remnant ginguro bands (oxidized at the Project to hematite), clays developed from near neutral fluids (illite-smectite) and an extensional structural setting where vein intersections and flexures in vein attitude are favorable loci for development of broad mineralized zones. A characteristic that distinguishes the immediate Project area from these analogs is extensive post mineral oxidation. A number of sub-horizontal zones of silver enrichment were recorded by (Smith, 1927), who correlated these zones with paleo-levels of Lake Cochise. Structural controls are shown in Figure 7-2 and Figures 8-1 through 8-3 illustrate the mineralized textures identified in core, which are consistent with low sulfidation, epithermal precious metal systems.

HRC is of the opinion that CSGM is applying an appropriate deposit model to the Project.

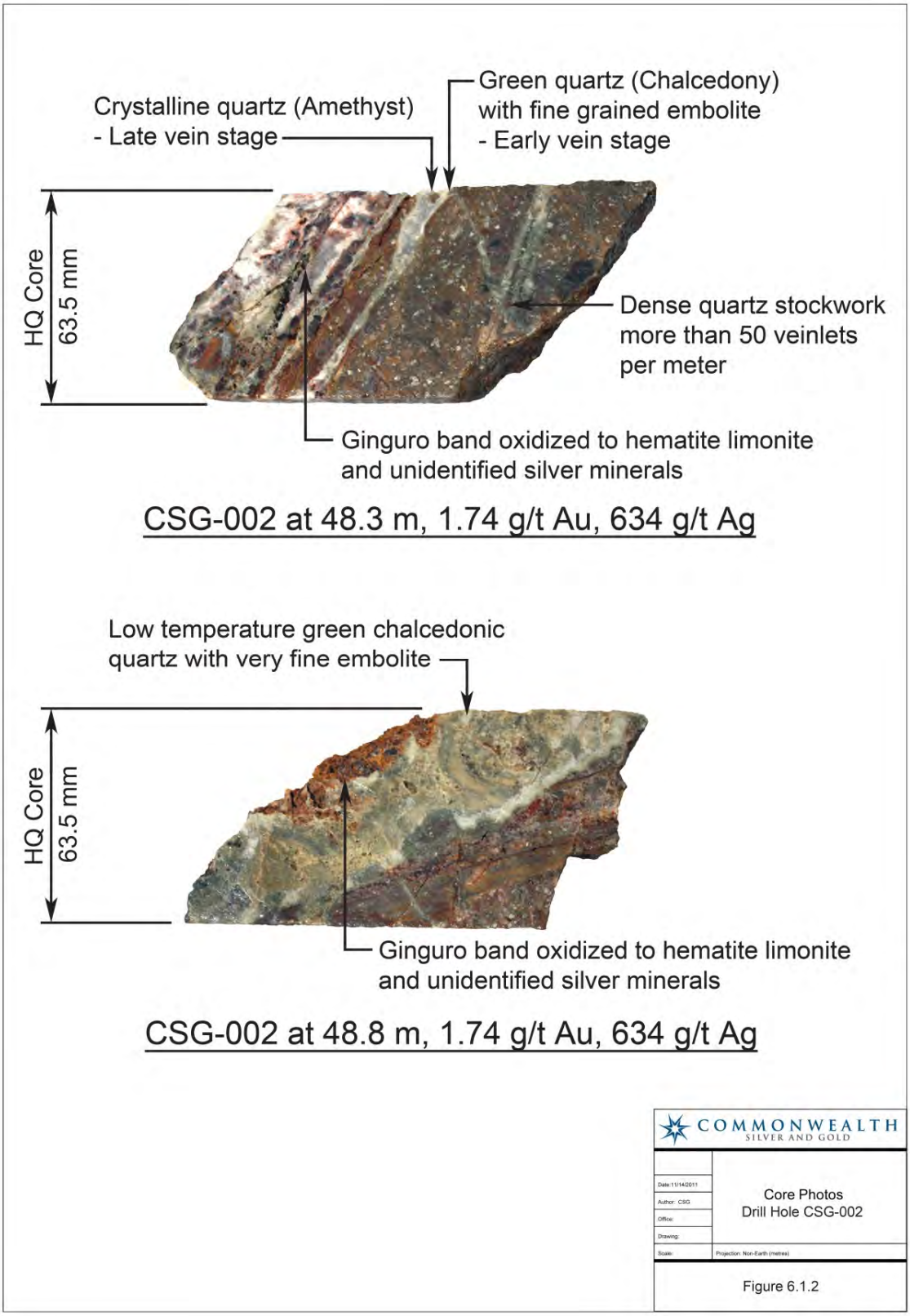


Figure 8-1 Mineralized Core Photos

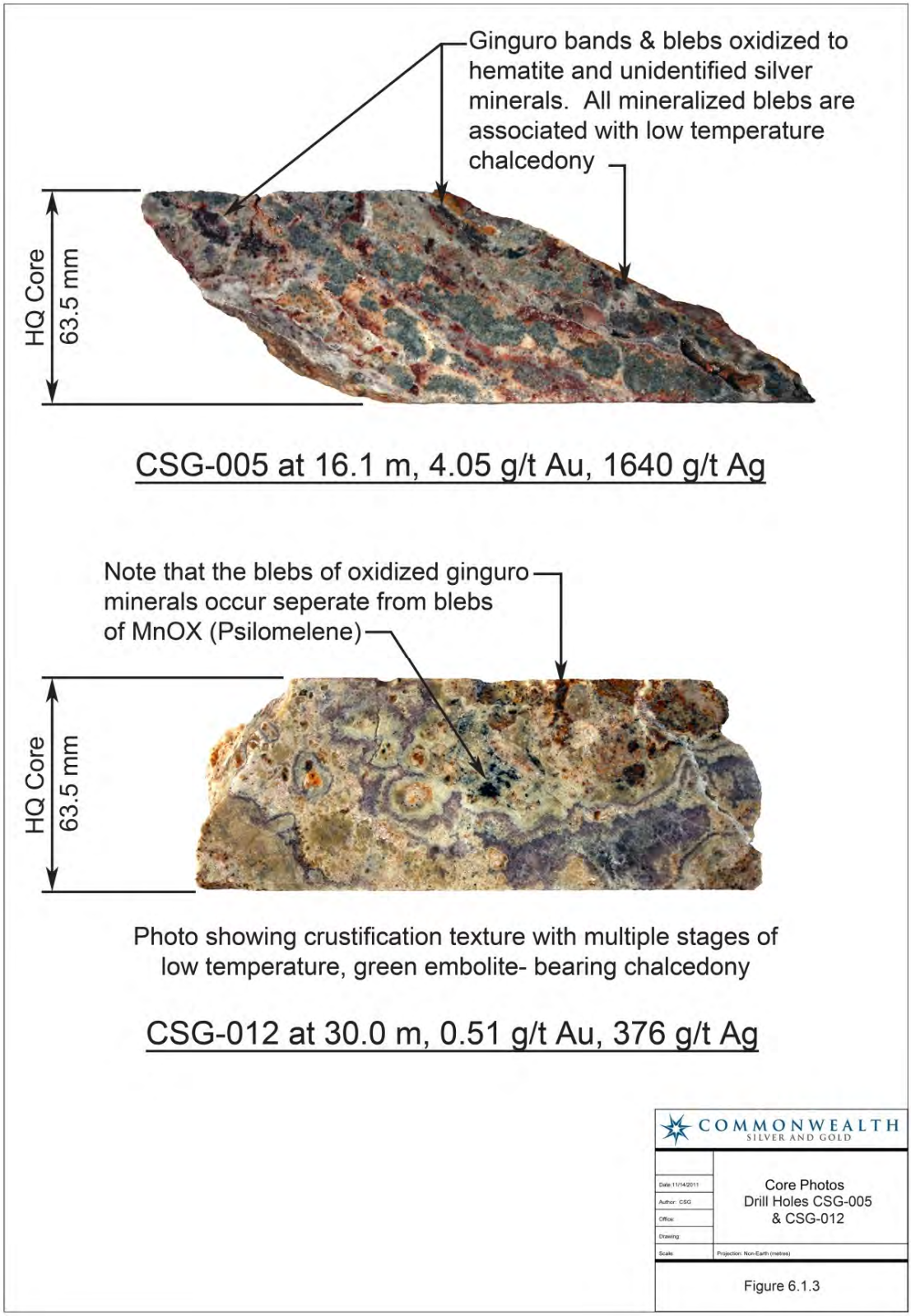


Figure 8-2 Mineralized Core Photos

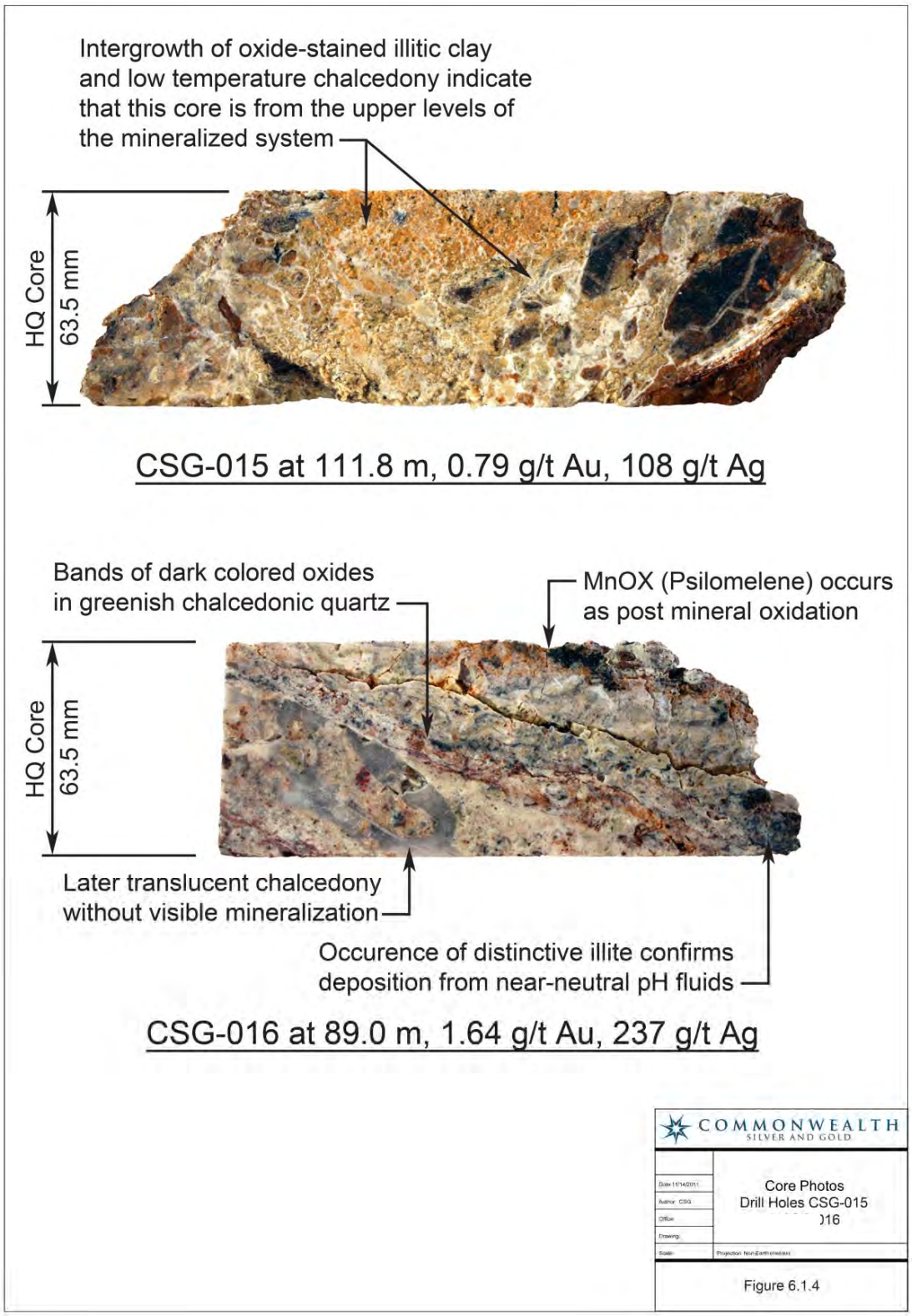


Figure 8-3 Mineralized Core Photos

9. Exploration

The Commonwealth Mine ceased commercial production in the late 1920's. Limited mining was conducted by lessors until 1942, when precious metals' mining in the United States was shut down by the Second War Powers Act. Exploration efforts on the Project resumed in the 1970's when the area was recognized as having bulk tonnage silver potential.

Planned exploration in the main Project area, the area in the immediate vicinity of the old mine workings, will consist mainly of step-out drilling following the east plunging intersection of the Main and North Veins, exploration drilling in the footwall of the North Vein to define potential disseminated mineralization in the Bisbee Group sediments, and step-out drilling to the west of Pearce Hill where the deposit is open along strike.

A number of geologists have completed geologic maps and have contributed to the geological understanding of the Project and surrounding region. Underground maps by Smith and Howell and surface maps by Eimon, Patton, Bailey and French were compiled by CSGM into a Mapinfo GIS database.

Relevant Exploration Work

CSGM has conducted three drilling programs at the Project. The first 16 hole, 2,003 meter program began on April 1, 2011 and was completed on June 16, 2011. The second 35 hole, 5,033 meter program began on November 21, 2011 and was completed on April 2, 2012, and the third 7 hole, 657 meter program was begun September 17, 2012 and completed October 14, 2012. During the first program, CSGM completed 16 HQ-size core holes as part of a confirmation and infill drilling program used to support the calculation of the initial NI 43-101 compliant mineral resource estimate on the Project by SRK Consulting (U.S.), Inc. ("SRK") in October 2011. The second and third programs comprising 35 and 7 HQ-size core holes, respectively, were completed as part of infill and step out-drilling programs. The third program included 5 holes drilled specifically for metallurgical samples.

Surveys and Investigations

To date, CSGM has compiled historic underground maps and drill collar locations and resurveyed drill-hole collars from previous drilling programs. CSGM also resurveyed the claim block to confirm claim corner locations and facilitate conversion of the existing Project grid into Universal Transverse Mercator ("UTM") coordinates, and conversion of all data into metric system units. CSGM acquired satellite imagery for the Project and conducted twin drill-hole and pulp comparison programs to verify the historical assay database. Results of the pulp comparison and twin drill-hole programs are discussed in detail Section 10 of this report.

Satellite Imagery

CSGM contracted Photosat Information, Ltd ("Photosat") of Vancouver, British Columbia, to generate a series of base maps for the Project. Using data from the Geoeye satellite, Photosat produced 100 km2 of digital imagery with a 0.5-meter pixel resolution as well as a 45 km2 of topographic coverage with a

10- cm vertical accuracy and 0.5 m x 0.5 m pixel size Digital Terrain Model ("DTM"). Contour maps with 1 m, 5 m and 10 m contour intervals were produced in AutoCAD drawing exchange (dxf) and Mapinfo formats. All Project work completed by CSGM following receipt of the digital products from Photosat was completed using the North American Datum ("NAD") 83, UTM zone 12 projection in meters.

Drill Collar Resurvey and Conversion of Grid and Project Units

CSGM contracted survey company Darling Environmental & Surveying, an Arizona Registered Land Surveying company out of Tucson, to resurvey the collar locations of all drill-holes on the Project. Darling located 99 of the 155 pre-existing drill-holes and confirmed claim locations. This survey was used during conversion of the original mine grid into UTM.

In order to convert the original, Arizona State Plane grid to UTM, Darling set up Trimble 5800 receivers on 4 aerial target points for 30 minutes each. The four target points were:

- The northeast corner of Section 4, Township 18 South, Range 25 East, G&SRM, (standard 3" GLO Brass Capped monument dated 1925);
- The Southwest comer of Section 4, Township 18 South, Range 25 East, G&SRM, (3" GLO Brass Capped monument dated 1925);
- Aerial Control Point 2001 (Darling point designation) near the saddle on the east side of Pearce Hill; and
- Aerial Control Point 2003 (Darling point designation) which was a USGS brass disk near the Border Patrol check point with the designation 22 BKG dated 1974.

The data files were processed through Online Position User Service ("OPUS") with a minimum accuracy of ±1 cm horizontally and ±4 cm vertically. Darling used the northwest and southwest corners of Section 4 to rotate the old grid into UTM NAD83, Zone 12 meters with elevations based on North American Vertical Datum ("NAVD") 88 (Darling, 2011).

Once the grid transformation was complete, Darling used a Trimble standard Real Time Kinematic ("RTK") methods calibrated to the 4 static positions listed above to complete surveys of the existing drill-holes (Darling, 2011).

10. Drilling

Type and Extent

Prior to CSGM's involvement in the Project, 155 holes were drilled in the main Project area by previous operators. Of these 155 historic holes, 148 were rotary or reverse circulation ("RC") and 7 were diamond drill-holes. Table 10-1 lists the drill-hole by series, type and company. Figure 10-1 shows the drill-hole locations.

Table 10-1 Commonwealth Silver and Gold Project Drill-hole Summary

Date	Hole Identification Range	Exploration Company	Drill-hole Type	No. of Holes Drilled	Reported/Actual Meters
1975	CS-1 - CS-5	Platoro Mines	Percussion	5	350.53
1976	DD-76.1 - DD-76.3	Bethlehem Exploration	Core	3	158.19
1976	P-76.6 – P-76.11, 9-76.13, P-76.14*	Bethlehem Exploration	Percussion	8	1,096.36
1977-1978	"WC.": 1A,2,3,4B,5,6,6B,6C,6D, 7,9,10,11B,12B,14,15	Western States Minerals	RC	16	1,441.71
1986	CM- series drillholes	Santa Fe Mining	RC	6	752.86
1988-1989	"W":1,1A,2A,3-5,7,8,11,12A,13-15,16A,16B,17A,17B,18A,19,20A,20B,21A, 21B,22A,22B,34-56 "E":1,1B,2,3,9,14,15	DRX, Inc.	RC	54	4,905.78
1991	GG-1 - GG-13	Glamis Gold	RC	13	1,350.26
1992	PHR-001 - PHR-011	Western States Minerals	RC	11	1,080.51
1993	PCW-1 - PCW-5	Pegasus Gold Corp	RC	5	981.46
1994	C-94-1 - C-94-4	Harvest Gold	RC	4	522.73
1995-1996	122-1 - 122-26	Atlas Precious Metals	RC	26	2,571.01
1995-1996	124-1 - 124-4	Atlas Precious Metals	Core	4	371.25
2011	CSG-001 - CSG-016	CSGM	Core	16	2,002.61
2012	CSG-017 – CSG-058	CSGM	Core	42	5,689.73

CSGM conducted a drilling program at the Project beginning April 1, 2011 and ending on June 16, 2011. Sixteen diamond drill-holes totaling 2,002.61 m were drilled during this time by Godbe Drilling, LLC ("Godbe") of Montrose, Colorado, using an Atlas Copco CS1000 and a Boart Longyear LF70 drill rig. All collar locations were surveyed and down-hole surveys were completed on 14 of the 16 holes. Diamond drill-holes were drilled using HQ and NQ tools.

CSGM conducted a second drilling program beginning on November 21, 2011 and ending on April 2, 2012. Thirty five additional diamond core drill-holes totaling 5,032.87 m were drilled by Godbe. All collar locations were surveyed and down-hole surveys were completed on 33 of 35 holes. Diamond drill-holes were drilled using HQ and NQ tools. During the 2012 drilling program CSGM was unable to complete three planned holes (CSG-024, CSG-025, and CSG-050). As a result no assay information was collected on the three holes. CSGM conducted a third drilling program using Brown Drilling ("Brown") of

Kingman, Arizona as contractor. Brown used a track mounted Boart Longyear LF70 with HQ and NQ tools. This third drilling program began on September 17, 2012 and was completed on October 14, 2012, and consisted of 7 diamond drill-holes totaling 656.86 meters. All collar locations were surveyed and down-hole surveys were completed on all holes. The last 5 holes drilled for CSGM by Brown were metallurgical holes (CSG-054 through CSG-058), that have not been used in the estimation of mineral resources for this report.

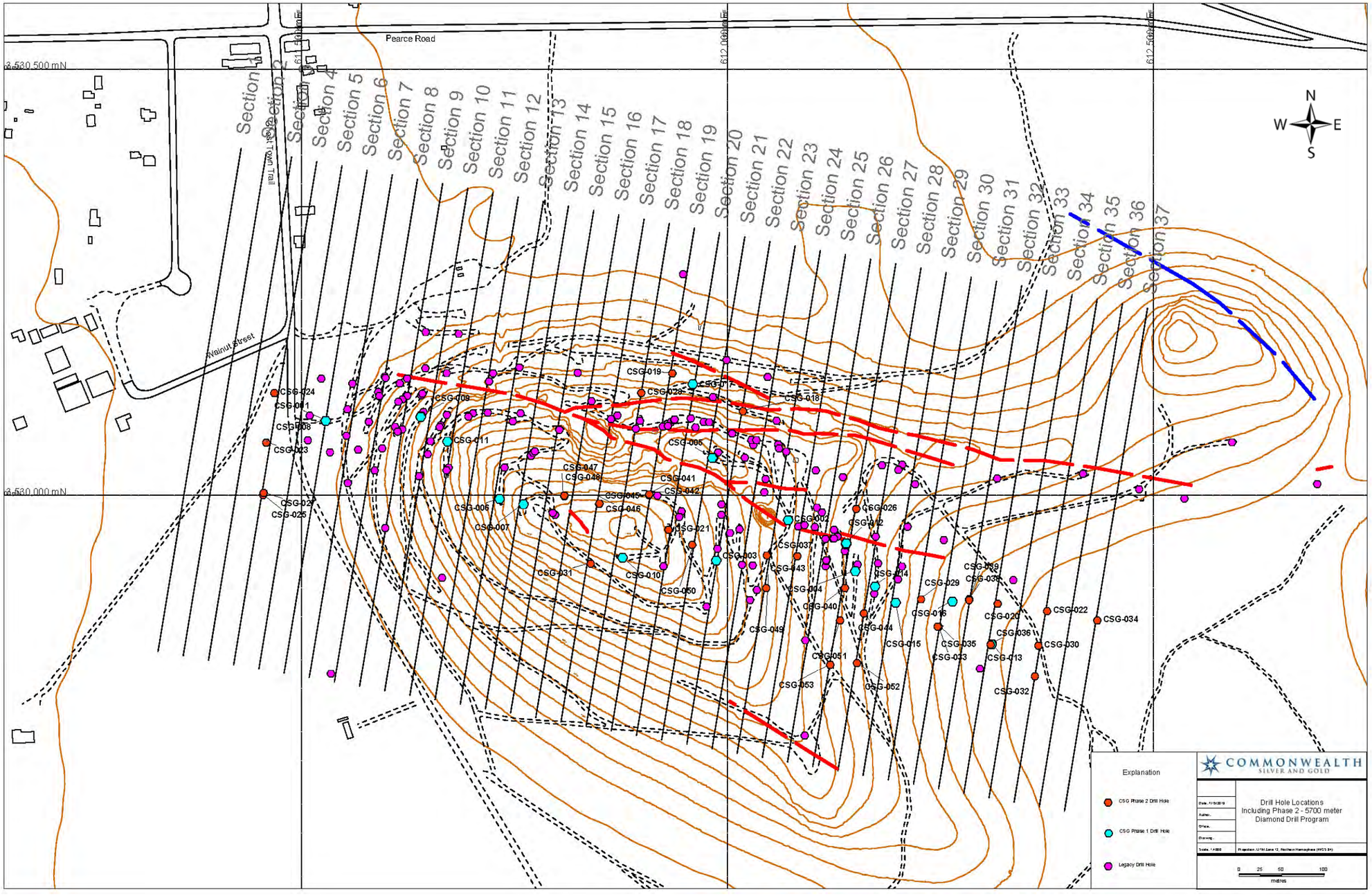


Figure 10-1 CSGM Drillhole Location Map

Procedures

RC campaigns conducted prior to CSGM's exploration programs used contemporary industry best practices. Hall Stewart, Vice President, Exploration for CSGM and Clive Bailey, CPG (former Senior Project Geologist for CSGM) worked at the Project during previous drilling campaigns and had supervisory control of much of the drilling conducted by previous owners. Drill sites for the 2011 and 2012 drilling programs were chosen to add diamond drill coverage along the entire strike length of the previously drilled resource area. The four specific goals of the drilling programs were:

- To provide diamond drilling support for the mostly RC historic drilling database;
- To twin 4 existing RC holes for comparison of RC versus diamond drilling sample results;
- To infill strategic areas in order to capture more of the mineral resource estimate in the higher confidence measured and indicated categories; and
- To complete limited step-out exploration.

The 2011 and 2012 drill sites were built using a Cat 988 front-end loader. This equipment proved to be very efficient at rehabilitating the network of existing roads, and in difficult areas, the loader was able to construct pads using fill material rather than having to cut into the hard, silicified bedrock.

The drill was aligned on the pads by pre-marking the drill-hole alignment with spray paint on the ground and then fine tuning the alignment using a Brunton compass when the drill was set up. The drill-hole inclinations were set using the inclinometer of the Brunton compass. A CSGM geologist visited the rig twice a day during day shift and once per night shift. CSGM geologists were responsible for determining when to terminate each drill-hole. CSGM's staff geologists supervised the set-up, drilling and abandonment of all drill-holes under ADWR requirements discussed in the following paragraph.

Down-hole surveys were completed on 54 of the 58 holes drilled by CSGM. Down-hole surveys were conducted using a Reflex EZTRAC digital down-hole survey tool. All drill-holes were abandoned according to ADWR requirements. All drill-holes were capped with a 6-m cement plug and marked with an aluminum block stamped with the drill-hole identification. Collar locations for all of CSGM's drill-holes and 99 of the pre-existing drill-holes were surveyed by Darling in UTM NAD83 zone 12 meters.

Drill core was collected once a day from the rig by CSGM geologists and brought to the secure core logging facility on the Project site. The facility is surrounded by a locked fence, and core storage and Project site offices are secured in locked Sea vans (cargo containers) within the fence. At the logging facility, each box of core was photographed by technicians who also logged core recovery and RQD data under the supervision of CSGM geologists. CSGM geologists then completed a geologic log of the core and marked intervals for sampling.

Sample intervals were nominally one meter each with sample lengths adjusted to break at lithologic and vein boundaries. In visually barren intervals, sample lengths were increased to a maximum of two

meters per sample. Sampling was extended outward from mineralized zones to include 20 m above and 20 m below the mineralized zone or to the bottom of the drill-hole. After the core was marked for sampling, the sampled intervals were sawn lengthwise using a 14 inch diamond saw and placed into plastic sample bags, then secured with heavy wire ties that require a winding tool to open or close in order to provide additional security against tampering once sealed.

Core and bagged samples were stored in steel shipping containers with shelving built for core boxes. Samples for the 2011 program were collected on site by Skyline Assayers and Laboratories ("Skyline") located in Tucson, Arizona, who transported the samples to the lab for analysis.

2012 Drilling Programs

CSGM completed a 35-hole, 5,032.87 m diamond drilling program at the Project between November 2011 and April 2012. The samples from the drilling program were shipped to ALS Minerals in Reno, NV ("ALS Reno") for assaying, and final results exhibit similar grades and mineralized widths to drill-holes already included in the current mineral resource estimate.

CSGM also completed a 749 m diamond drilling program in 7 holes at the Blue Jeep and San Ignacio properties. The Blue Jeep and San Ignacio Project is external to the Commonwealth Project and is not considered in this report. After compilation of these data and completion of the additional drilling programs recommended in Section 25, CSGM will complete an updated mineral resource estimate.

CSGM also conducted a 7-hole diamond drilling program in September and October 2012. This program was primarily to collect whole HQ size core for metallurgical studies. Two exploration holes and 5 metallurgical holes were drilled for a total of 656.86 m. The exploration portion of the program was 466.96 m in 2 holes and the metallurgy portion of the program was 189.90 m in 5 holes. Holes CSG-055 and CSG-056 were both twins of CSG-005. Hole CSG-055 was lost before completion and is not shown in the table. The metallurgical holes were all twins of exploration drill-holes and were planned to collect approximately 100 kg of mineralized core from each of the five main rock types with near average grades.

Relevant Results and Interpretation

Results

Relevant results from CSGM's drilling programs are listed in Table 10-2. The primary target of the drilling is a stockwork zone, and all intercepts listed are drilled intercepts and do not represent true thickness. Table 10-2 presents both the overall mineralized intercepts and the highest grade results from the drill-holes completed during the 2011 and 2012 drilling programs. These samples are not representative of the entire grade distribution of gold and silver mineralization.

Table 10-2 Significant Intercepts from CSGM Drilling Programs

Hole ID		Total Hole Depth (m)	From (m)	To (m)	Intercept Interval (m)	Au grade (g/t)	Ag grade (g/t)	At 60:1 AuEq grade (g/t)
Drilling Program #1: 16 Exploration Holes, 2,002.61 m – April 1, 2011 to June 16, 2011								
CSG-001		60.96	31.00	41.70	10.70	1.58	54.49	2.49
CSG-002		157.58	14.17	118.00	103.83	0.47	85.78	1.90
CSG-002	including		45.00	52.00	7.00	0.80	216.86	4.41
CSG-002	and including		69.00	82.00	13.00	1.71	197.15	5.00
CSG-002	and including		90.00	94.00	4.00	1.28	204.00	4.68
CSG-002	and		142.00	150.30	8.30	1.67	22.95	2.05
CSG-003		176.17	116.00	146.00	30.00	0.57	71.52	1.76
CSG-003	including		128.44	134.73	6.29	0.77	96.82	2.38
CSG-004		133.09	42.00	66.00	24.00	0.09	70.68	1.27
CSG-004	and		75.00	129.00	54.00	0.23	67.19	1.35
CSG-004	including		77.00	92.00	15.00	0.34	108.93	2.16
CSG-005		124.60	2.00	72.00	70.00	0.52	136.61	2.80
CSG-005	including		9.00	16.40	7.40	1.52	320.27	6.86
CSG-005	and including		35.00	58.00	23.00	0.58	150.53	3.09
CSG-005	and		87.30	111.00	23.70	0.40	42.64	1.11
CSG-006		151.06	136.00	142.00	6.00	0.74	22.17	1.11
CSG-007		169.55	82.00	90.00	8.00	0.22	69.00	1.37
CSG-007	and		134.00	148.00	14.00	0.73	51.13	1.58
CSG-008		74.79	31.00	35.00	4.00	1.68	8.00	1.81
CSG-008	and		45.00	49.00	4.00	1.41	14.50	1.65
CSG-008	and		55.00	59.00	4.00	0.65	15.25	0.90
CSG-009		74.99	4.00	23.00	19.00	0.89	50.37	1.73
CSG-009	and		29.00	35.00	6.00	0.71	28.83	1.19
CSG-009	and		48.00	59.00	11.00	0.42	29.68	0.91
CSG-009	and		62.00	74.99	12.99	0.49	31.34	1.01
CSG-010		158.66	144.00	158.66	14.66	0.59	89.84	2.09
CSG-011		135.18	86.00	115.00	29.00	1.27	50.11	2.11
CSG-011	including		104.00	115.00	11.00	2.30	101.55	3.99
CSG-012		112.47	22.00	74.00	52.00	0.20	82.36	1.57
CSG-012	including		45.00	49.56	4.56	0.61	122.06	2.64
CSG-014		99.67	67.00	99.67	32.67	0.08	60.47	1.09
CSG-015		140.21	83.52	140.21	56.69	0.11	54.80	1.02
CSG-016		105.16	79.00	105.16	26.16	0.82	136.50	3.10
CSG-016	including		84.00	90.00	6.00	3.05	426.67	10.16

Hole ID		Total Hole Depth (m)	From (m)	To (m)	Intercept Interval (m)	Au grade (g/t)	Ag grade (g/t)	At 60:1 AuEq grade (g/t)
Drilling Program #2: 35 Exploration Holes, 5,032.87 m – November 21, 2011 to April 2, 2012								
CSG-017		81.08	0.00	31.85	31.85	0.28	16.24	0.55
CSG-018		100.58	0.00	17.00	17.00	0.21	36.55	0.82
CSG-019		60.05	0.00	10.00	10.00	0.92	28.68	1.40
CSG-019	including		7.00	9.00	2.00	2.84	21.23	3.19
CSG-020		144.78	44.00	50.50	6.50	0.24	56.69	1.18
CSG-020	and		68.00	116.00	48.00	0.10	66.87	1.21
CSG-020	including		105.00	110.64	5.64	0.57	222.18	4.27
CSG-021		200.25	108.00	147.00	39.00	1.16	136.52	3.44
CSG-021	including		110.00	127.00	17.00	2.42	243.18	6.47
CSG-021	and		155.00	180.00	25.00	0.66	42.66	1.37
CSG-021	including		170.69	173.74	3.05	3.09	155.02	5.67
CSG-022		161.54	103.00	122.00	19.00	0.09	89.98	1.59
CSG-022	including		103.00	112.00	9.00	0.07	147.70	2.53
CSG-023		99.97	58.00	64.00	6.00	0.58	6.63	0.69
CSG-026		163.68	20.00	25.00	5.00	0.20	30.04	0.70
CSG-026	and		29.00	50.00	21.00	0.15	41.29	0.84
CSG-026	and		86.00	92.50	6.50	0.18	18.50	0.49
CSG-028		151.49	53.50	63.00	9.50	1.83	40.24	2.50
CSG-028	including		54.00	58.00	4.00	3.88	79.73	5.21
CSG-029		157.58	22.00	27.00	5.00	0.03	36.14	0.63
CSG-029	and		74.22	118.00	43.78	0.15	57.85	1.11
CSG-029	and		124.00	130.68	6.68	0.23	62.89	1.28
CSG-030		169.47	141.00	157.00	16.00	0.45	44.36	1.19
CSG-031		239.88	29.00	34.00	5.00	0.17	99.65	1.83
CSG-031	and		183.00	190.20	7.20	1.28	23.57	1.67
CSG-032		201.17	170.00	172.00	2.00	0.27	121.00	2.29
CSG-033		168.55	102.34	115.00	12.66	0.17	67.87	1.30
CSG-033	and		119.00	147.00	28.00	0.12	44.22	0.86
CSG-034		139.29	112.00	123.00	11.00	0.18	87.80	1.64
CSG-034	including		113.00	116.20	3.20	0.10	196.18	3.37
CSG-035		191.11	140.00	149.81	9.81	0.08	36.15	0.68
CSG-035	and		162.80	172.00	9.20	0.40	10.97	0.58
CSG-036		188.06	127.00	131.54	4.54	0.28	27.35	0.74
CSG-036	and		137.00	144.00	7.00	0.18	22.74	0.56
CSG-036	and		163.10	166.50	3.40	1.55	26.53	1.99

Hole ID		Total Hole Depth (m)	From (m)	To (m)	Intercept Interval (m)	Au grade (g/t)	Ag grade (g/t)	At 60:1 AuEq grade (g/t)
CSG-037		169.01	69.00	152.00	83.00	0.46	70.18	1.63
CSG-037	including		82.75	99.60	16.85	0.73	154.82	3.31
CSG-037	and		131.52	136.00	4.48	1.60	76.44	2.87
CSG-038		141.73	62.00	91.00	29.00	0.14	40.95	0.82
CSG-038	and		107.00	124.00	17.00	0.07	42.83	0.78
CSG-039		149.35	85.00	121.30	36.30	0.23	90.27	1.73
CSG-039	including		101.40	108.00	6.60	0.40	216.80	4.01
CSG-040		157.89	36.27	142.00	105.73	0.26	92.39	1.80
CSG-040	including		84.00	109.00	25.00	0.62	192.24	3.82
CSG-040	and including		112.36	121.00	8.64	0.68	133.33	2.90
CSG-042		151.18	62.00	77.30	15.30	0.68	352.24	6.55
CSG-042	and		89.36	95.08	5.72	0.29	28.95	0.77
CSG-043		56.10	34.00	41.76	7.76	0.02	46.19	0.79
CSG-044		195.07	29.26	41.60	12.34	0.20	126.77	2.31
CSG-044	and		72.00	94.00	22.00	0.01	61.31	1.03
CSG-044	and		104.00	116.00	12.00	0.05	45.93	0.82
CSG-044	and		121.00	149.00	28.00	0.32	67.90	1.45
CSG-044	and		181.60	185.93	4.33	1.27	20.15	1.61
CSG-045		178.92	123.00	166.00	43.00	1.20	72.39	2.41
CSG-045	including		126.05	131.00	4.95	3.07	125.73	5.17
CSG-045	and including		134.00	148.89	14.89	1.49	82.24	2.86
CSG-047		151.49	108.00	144.00	36.00	0.61	69.34	1.77
CSG-047	including		109.00	116.00	7.00	1.08	185.66	4.17
CSG-048		145.08	74.95	78.00	3.05	0.12	103.32	1.84
CSG-048	and		102.00	130.00	28.00	0.42	61.10	1.44
CSG-048	including		107.00	111.71	4.71	0.42	152.90	2.97
CSG-049		171.75	16.00	34.00	18.00	0.01	59.56	1.00
CSG-049	and		113.00	161.00	48.00	0.51	62.09	1.54
CSG-049	including		114.50	127.00	12.50	0.46	109.25	2.28
CSG-051		199.95	55.00	63.00	8.00	0.01	43.41	0.73
CSG-051	and		72.00	78.00	6.00	0.05	47.43	0.84
CSG-051	and		139.00	160.00	21.00	1.28	118.92	3.26
CSG-051	including		140.08	148.00	7.92	3.06	205.42	6.48

Hole ID		Total Hole Depth (m)	From (m)	To (m)	Intercept Interval (m)	Au grade (g/t)	Ag grade (g/t)	At 60:1 AuEq grade (g/t)
Drilling Program #3: 2 Exploration Holes, 5,032.87 m – November 21, 2011 to April 2, 2012								
CSG-052		224.95	170.00	177.00	7.00	0.02	30.23	0.52
CSG-052	and		185.83	191.00	5.17	0.60	41.54	1.29
CSG-053		242.01	187.00	204.00	17.00	0.65	20.12	0.98

The assay results from the 2011 and 2012 drilling programs, as well as assay results from historic drilling and channel sampling programs are representative and suitable for use in resource estimation. All results from the CSGM drilling programs as well as the results from the 155 historic holes in the resource area and 209 channel samples used in the mineral resource estimate are discussed in Section 14.

Interpretation

Ms. Brown of HRC visited the site and observed the core handling, logging and sampling procedures of CSGM and concludes the procedures meet current industry standards. Locations and elevations of all CSGM drill-holes have been surveyed and the locations and elevations of historical drill-holes have been resurveyed where available. HRC is of the opinion that CSGM meets or exceeds industry best practice in conducting its drilling and logging programs.

11. Sample Preparation, Analysis, and Security

All drill core is transported from drill sites by a representative of CSGM and stored in a secure storage area until the core can be logged. Sample security is controlled and supervised by CSGM personnel. CSGM observes industry best practice chain of custody.

Methods

CSGM used Skyline Labs of Tucson, Arizona for the 16-hole 2011 drilling program. Samples for the second 35-hole drilling program were shipped by United Parcel Service ("UPS") to ALS Minerals in Reno, Nevada ("ALS Reno") and samples for the third drilling program were further shipped by ALS Reno to ALS Minerals in Elko, Nevada ("ALS Elko"). All laboratories utilized for analytical testing are independent from CSGM.

Skyline Labs Methods

All CSGM samples were analyzed using a 30 g fire assay ("FA") with an atomic absorption spectroscopy ("AAS") finish for gold. This technique has a lower detection limit of 0.005 ppm and an upper detection limit of 3.00 ppm. Samples with greater than 3.00 ppm Au were re-analyzed using a 30 g, FA with a gravimetric finish. All CSGM samples were also analyzed using a 5 g sample with a four acid digestion for silver and multi-element analysis using an Inductively Coupled Plasma Optical Emission Spectroscopy ("ICP-OES") instrument. This technique has a lower detection limit of 1 ppm for silver and an upper detection limit of 150 ppm for silver. Samples with greater than 150 ppm Ag were re-analyzed using a 30 g, FA with a gravimetric finish.

ALS Minerals Methods

All CSGM samples were analyzed using a 30 g FA with an AAS finish for gold (ALS code AU-AA23). This technique has a lower detection limit of 0.005 ppm and an upper detection limit of 10.00 ppm. Samples with greater than 10.00 ppm Au were re-analyzed using a 30 g FA with a gravimetric finish (ALS code Au-GRA21). All CSGM samples were also analyzed using a 5 g sample with a four acid digestion for silver and multi-element analysis using an ICP-OES instrument (ALS code ME-ICP61). This technique has a lower detection limit of 0.5 ppm for silver and an upper detection limit of 100 ppm for silver. Samples with greater than 100 ppm Ag were re-analyzed using a 10 g sample with a four acid digestion for silver and an AA finish (ALS code AG-OG62). This technique has a lower detection limit of 1 ppm for silver and an upper detection limit of 1500 ppm for silver. Samples with greater than 1,500 ppm Ag were re-analyzed using a 30 g FA with a gravimetric finish (ALS code GRA-21). This technique has a lower detection limit of 5 ppm for silver and an upper detection limit of 10,000 ppm for silver.

CSGM received the core boxes at the core processing facility directly from drilling company personnel. Core was always kept locked inside steel storage containers with core shelving except during photography, logging and sample cutting. Samples were submitted to the assay lab in plastic bags secured with heavy wire ties that cannot be removed. Sample bags are cut open at the assay lab for sample preparation and analysis. Samples from the 2011 drilling program were picked up on site by

Skyline personnel and delivered directly to the lab in Tucson, Arizona, where they were kept in Skyline's secure facility until the samples were ready to be prepared and analyzed. Samples were shipped to ALS Minerals Reno by UPS. For the third drilling program, ALS Reno then shipped the samples to ALS Elko.

Sample Preparation

Skyline crushed the entire sample to 75% passing a -10 mesh and then split off 250 g for pulverization to 95% passing a -150 mesh. Cleaner sand was run through the crusher every 2 samples or at any colour change in the sample noticed by Skyline's lab technicians. Sand was run between every sample in the pulverizing step. Pulps were split again to separate a 30 g sample for FA/AA for gold and a 5 g sample for multi-acid digestion and ICP-OES for silver and multi-element analysis.

ALS Minerals crushed the entire sample to 75% passing a -6mm mesh and then split off 250 g for pulverization to 85% passing a -75 micron (200 mesh). Cleaner sand was run through the crusher every 5 samples or at any color change in the sample noticed by ALS's lab technicians. Sand was run between every sample in the pulverizing step. Pulps were split again to separate a 30 g sample for FA/AA for gold and a 5 g sample for multi-acid digestion and ICP-OES for silver and multi-element analysis. Further splits were taken from the same pulp if FA/GRAV was required for over-limit analyses of silver.

Laboratories

All CSGM samples for the 16-hole program (2,002.61 m) were analyzed at Skyline in Tucson, Arizona, discussed above. Skyline has ISO/IEC 17025:2005 certification for FA, AAS, ICP-OES and ICP-Mass Spectroscopy ("MS"). Laboratories used by previous operators for drilling are identified in Table 10.3.1.1 and for channel samples are identified in Table 10.3.1.2 in the SRK technical report dated March 15, 2012 and amended April 11, 2012. HRC has no information regarding relationships with analytical laboratories prior to 2011 and cannot comment. All samples from the CSGM drilling program as well as the historic drilling and channel samples were used in the mineral resource estimate.

For the drilling completed at the Project (5,032.87 m) and at Blue Jeep and San Ignacio (749 m) from November 21, 2011 to April 2, 2012 discussed above, the analyses for these programs were completed by ALS Reno. For the drilling completed at the Project (466.96 m) from September 17, 2012 to October 14, 2012 discussed above, the analyses were completed by ALS Elko. ALS Reno and ALS Elko have ISO 9001:2008 accreditation for quality management and ISO/IEC17025:2005 accreditation for gold assay methods.

Part of the Quality Assurance/Quality Control ("QA/QC") program discussed in the following sections was conducted to verify analytical results from the previous analytical programs.

Quality Assurance/Quality Control Procedures

QA/QC samples used by CSGM include blanks, standards and field duplicates. CSGM inserts QA/QC samples into the sample stream at the following frequencies:

- One blank approximately every 33 samples placed randomly in the sample stream and within apparently mineralized intercepts (3 samples per 100);
- One standard every 25 samples; and
- One core duplicate placed at sample 31, 61 and 91.

Blank material used was rhyolite collected from near the Project area and was submitted as a coarse preparation blank. The blank is uncertified but analysis showed that it was below detection limit for Au and at detection limit for Ag. Detection limit for Ag is significantly below the cut-off grade ("CoG") of the estimated mineral resource.

CSGM used only one standard for the 2011 drilling program. The standard used was a commercially available, certified standard, purchased from WCM Minerals Ltd., ("WCM") based in Burnaby, British Columbia. During the 2012 drilling program CSGM added 4 additional commercially available standards, purchased from CDN Resource Laboratories, Ltd. ("CDN").

CSGM used core duplicate samples for duplicate analyses. For duplicate samples, the remaining half of the core was submitted as a duplicate interval. No core was retained for these intervals which represent only 3% of the total core. During the recent drilling program, CSGM submitted coarse blanks, core duplicates, and multiple assay standards. Pulp duplicates and control samples representing 10% of all samples above 0.5 ppm gold equivalent, calculated at 60:1, were sent to a second laboratory after completion of the drilling program.

Blanks

Coarse blanks monitor the integrity of sample preparation and are used to detect contamination during crushing and grinding of samples. Blank failures can also occur during laboratory analysis or as the result of a sample mix-up. A blank analysis ≥5 times the detection limit is considered a blank failure. For gold this is 0.025 g/t and for silver this is 5 g/t.

CSGM submitted 49 coarse pulp blanks to Skyline to monitor sample preparation during the 2011 drilling program. Five blank samples submitted to Skyline were failures. Of these, two samples failed for gold, two samples failed for silver and one sample failed for both gold and silver. This represents a 10% failure rate for blank samples. Of the three gold failures, two were <0.06 g/t and the third failure was 0.346 g/t. The three silver failures were 7, 14 and 101 g/t. The failures with values of 0.346 g/t Au and 101 g/t Ag were for the same sample. In all cases where there was mineralized material adjacent to the blank failures, 5 samples either side of the blank were re-submitted and a new blank was inserted. Results were acceptable.

CSGM submitted 88 coarse blanks to ALS to monitor sample preparation during the second 2012 drilling program. Three blank samples submitted to ALS were failures. All of these samples failed for both gold and silver. This represents a 3.4% failure rate for blank samples. The blank assay failures reported in the previous report, and presented to HRC personnel, were resubmitted for re-analysis of both gold and

silver. All intervals were resubmitted with QA/QC samples and the results are acceptable. Gold and silver blank analyses are presented in graphical form in Figures 11-1 and 11-2, respectively.

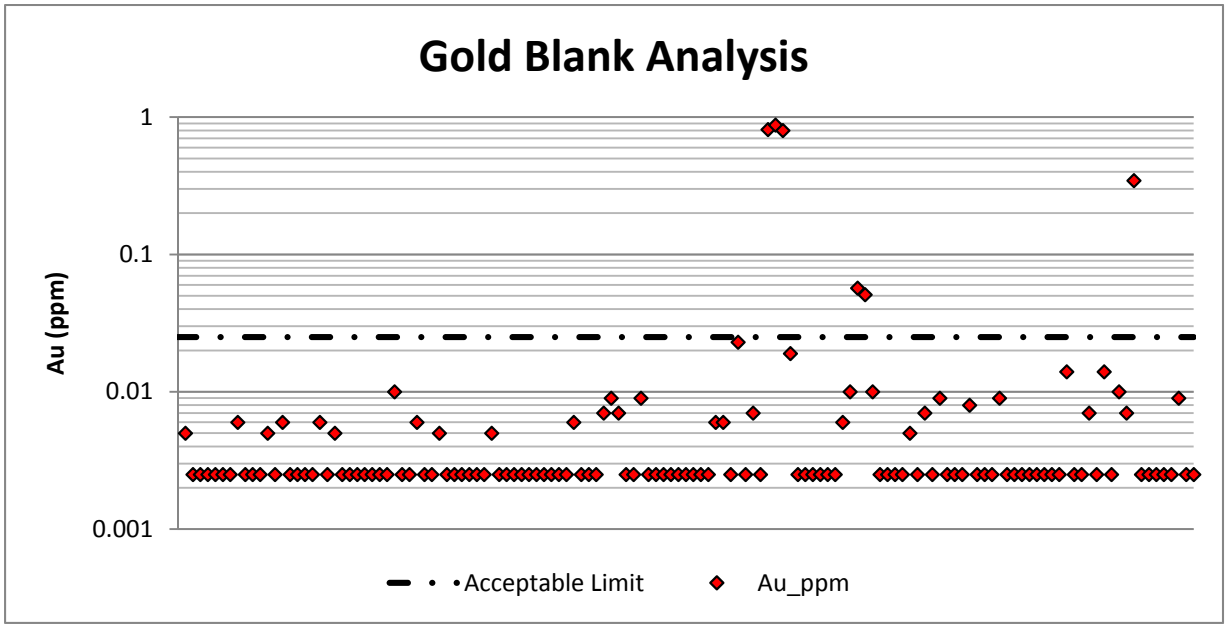


Figure 11-1 Gold Blank Analysis

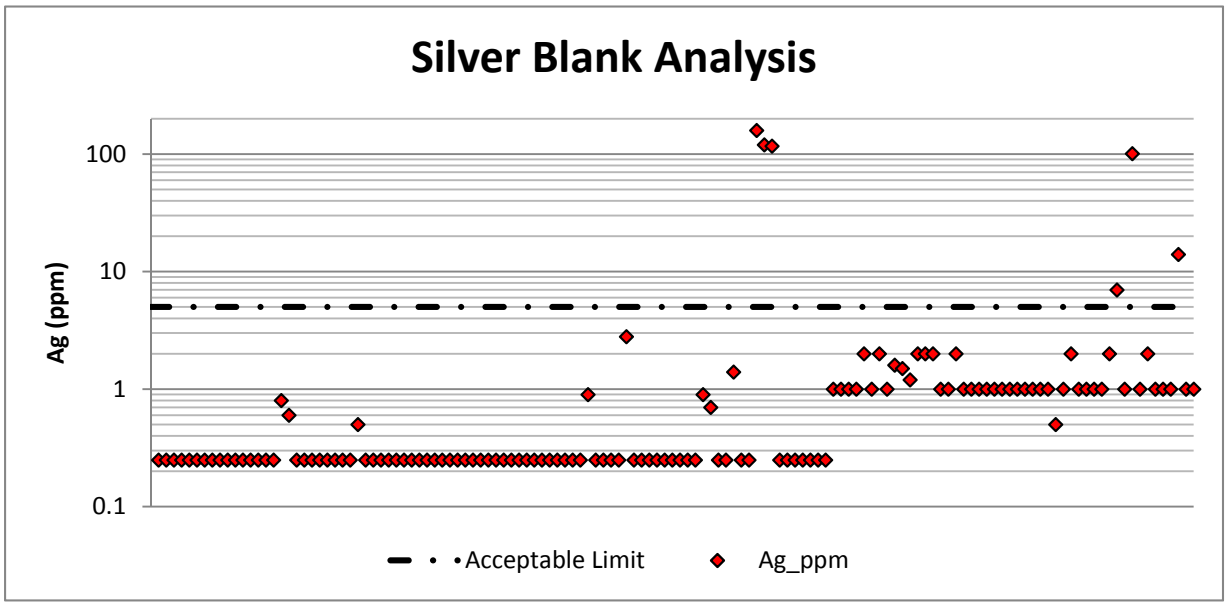


Figure 11-2 Silver Blank Analysis

Standards

Standards are used to monitor laboratory consistency and to identify sample mix-ups. They are usually submitted as a pulp and are either a Certified Reference Material ("CRM") or a site specific standard that may or may not be certified. A CRM has a performance range that is either specified by the certifying entity or direction is provided on how to determine a performance range. Generally, the performance range is approximately ±2 standard deviations ("stdev" or "σ") from the mean of the standard, and the standard is expected to perform within this range 95% of the time. The standard deviation is determined from analyses of the standard in a number of laboratories and can include more than 28 separate analyses. Standards are certified for a specific analytical technique. Within-set (samples run in sequence on the same day) shows homogeneity of the standard and the laboratory's ability to routinely reproduce the analytical method. Between-set considers the same factors as within-set, but includes bias between laboratories and bias in the subsets of samples sent to the participating laboratories.

CSGM used one standard for the 2011 drilling program and submitted 52 standard samples for analysis at Skyline. The WCM standard used was PM1138, which is certified for use with the following techniques:

- FA using a 30 g charge and AAS finish for gold; and
- AAS with a four acid digestion for silver and copper.

HRC reviewed the standards employed by CSGM to insure reliable assay information throughout the database. The individual standards were plotted against ±2 and ±3 standard deviations of the expected standard mean (Figure 11-3). The two types of failures can be identified by the red and orange colored symbols on Figure 11-3. CSGM at the request of HRC re-analyzed the failures within the standards following the protocol outlined for Type 1 and Type 2 failures:

- **Type 1 Failure** – Assays are outside ±3 standard deviations. If samples five meters before and five meters after the failed standard contain assays greater than or equal to 0.10 ppm gold or 10 ppm silver, then all five samples before and all five samples after the failed standard are re-assayed. A new standard should be submitted with each re-assay batch.
- **Type 2 Failure** – Two or more sequential standard assays are outside of ±2 standard deviations from their respective expected mean gold or silver grade and samples five meters before and five meters after each failed standard contain assays greater than 0.01 ppm gold or 10 ppm silver, then all five samples before and all five samples after the failed standards are re-assayed. A new standard should be submitted with each re-assay batch.

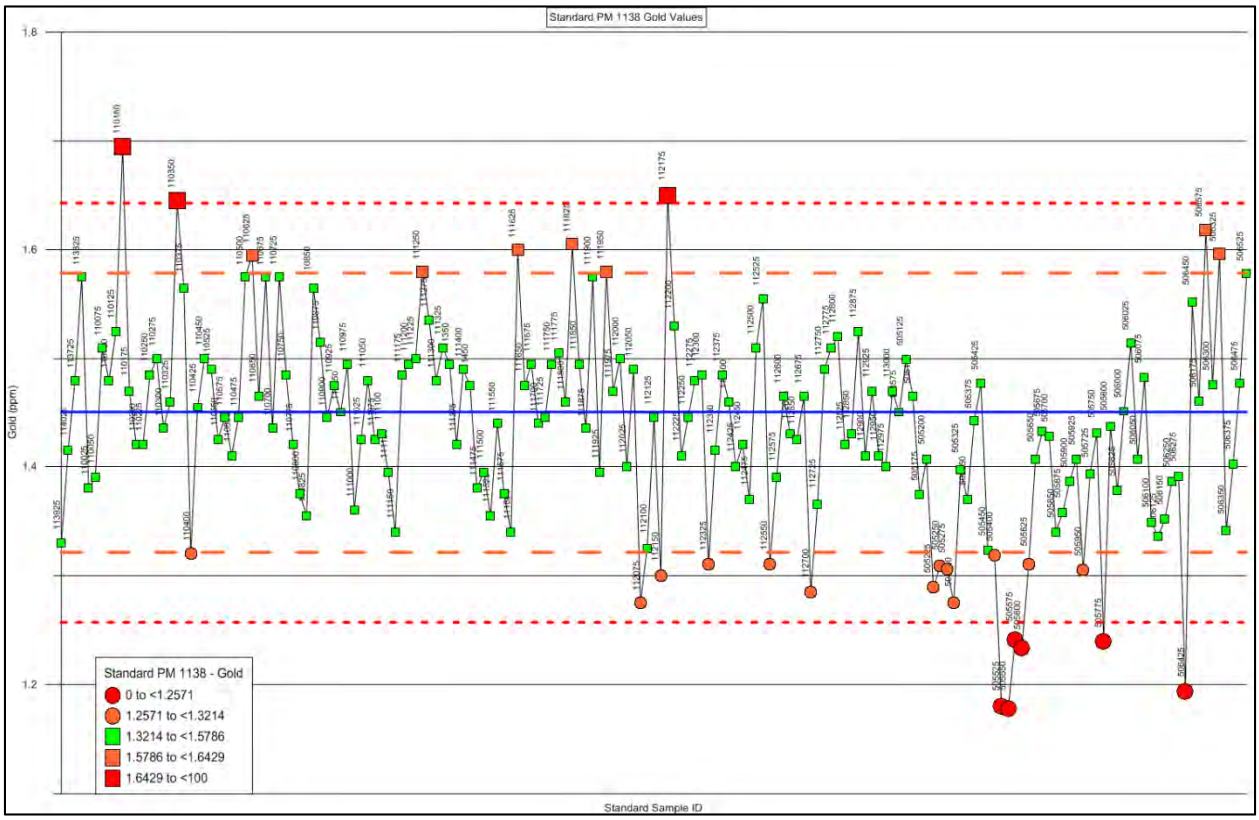


Figure 11-3 Gold Results for Standard PM1138

The standard assay failures reported in the previous technical report, and presented to HRC personnel have all been re-submitted for re-analysis of both gold and silver. All intervals were re-submitted with QA/QC samples and the results are acceptable.

At the recommendation of SRK, CSGM added four additional assay standards to represent a wider range of grades. These standards were introduced only in the September – October 2012 drilling program and not enough data are available for discussion of results. No apparent analytical failures have been identified.

Duplicates

Core Duplicates

Duplicates are used to monitor sample batches for sample mix-ups, data variability due to laboratory error and sample homogeneity at each step of preparation. Sample duplicates should be inserted at every sample split during sample preparation and they should not be placed in sequential order. When original and duplicates samples are plotted in a scatterplot, perfect analytical precision will plot on x=y (45°) slope. Core duplicates are expected to perform within ±30% of the x=y slope, coarse preparation duplicates should perform within ±20% of the x=y slope while pulp duplicates are expected to perform within ±10% of the x=y slope on a scatterplot.

CSGM used core duplicates during the first and second drilling programs. These were used to determine the correct sample size for analyses. In general, silver duplicates performed better than gold with 17 failures. There were 55 gold failures and in the majority of these the duplicate reported a lower result than the original sample. This represents a 12% failure rate for silver and a 40% failure rate for gold. Silver failures are divided above and below ±30% and relatively evenly clustered along the x=y axis. The original silver samples performed slightly higher than the duplicates. The majority of gold duplicates performed lower than the original sample. Scatterplots for gold and silver are shown in Figures 11-4 and 11-5, respectively.

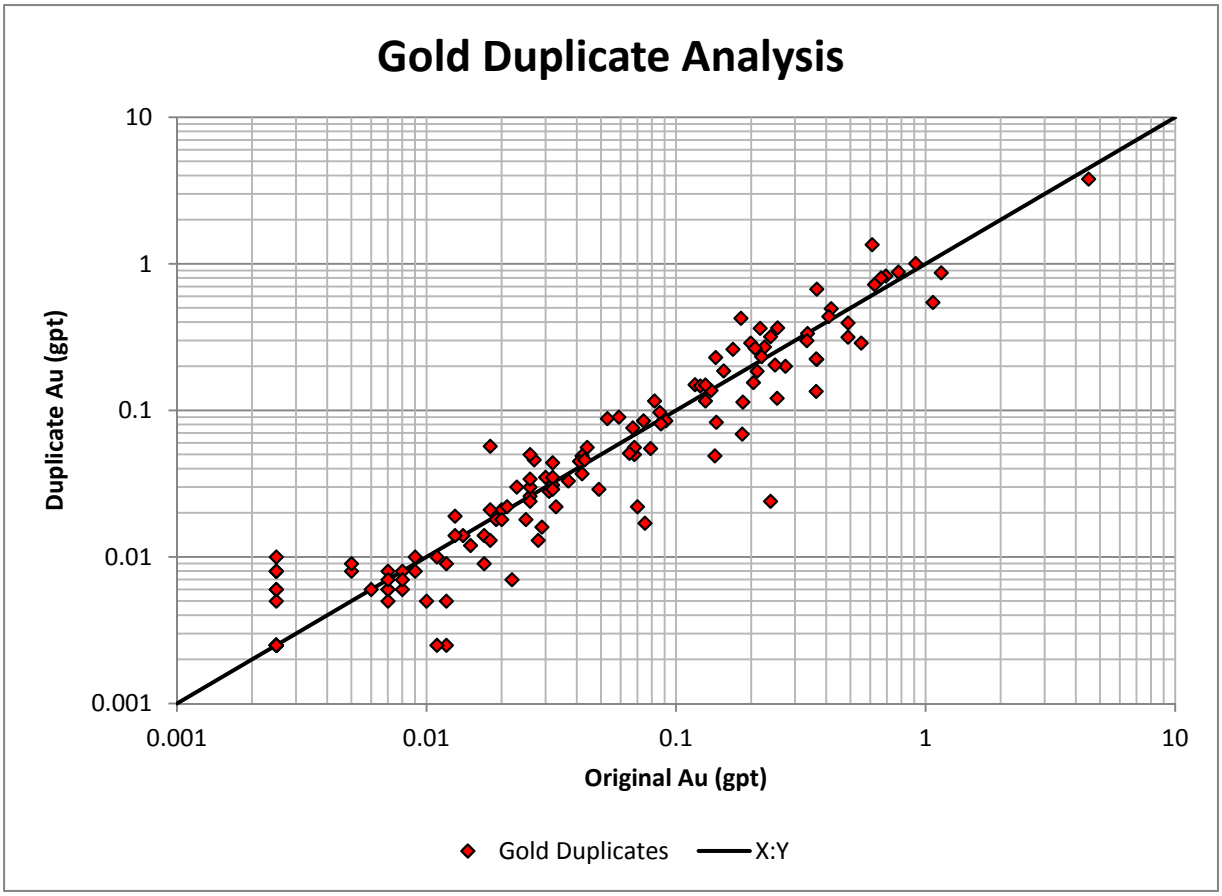


Figure 11-4 Gold Duplicate Analysis

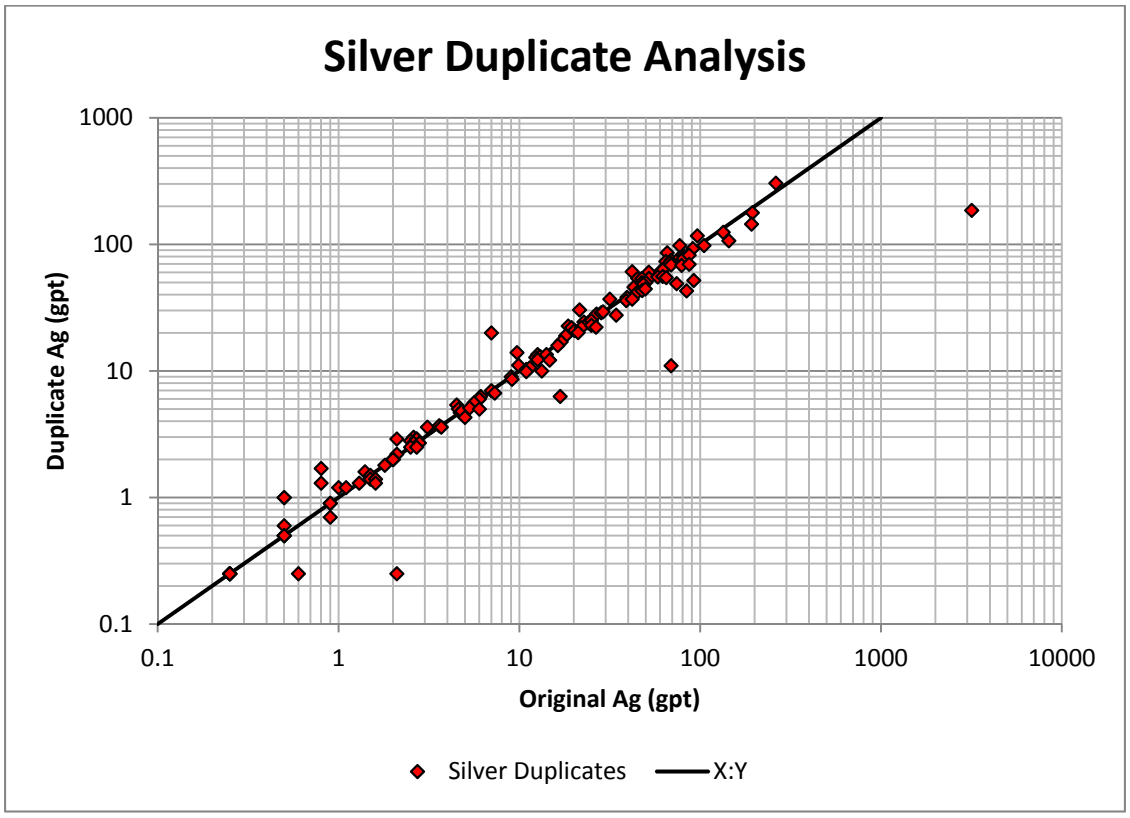


Figure 11-5 Silver Duplicate Analysis

The core duplicate performance suggests that the sample size is adequate for silver, but may be too small for gold. The failures are all in samples less than 1.5 g/t Au, which would suggest this, is not a nugget problem. Because of the standard performance, it may be too early to determine if this is a problem with the laboratory or with the samples.

Pulp Duplicates

Part of CSGM's ongoing QA/QC program includes re-analysis of pulps from the drilling programs as a check on the results from the original lab. CSGM selected 10% of all mineralized samples from CSGM's drilling programs for re-assay. Mineralized samples were defined as any sample with greater than or equal to 26 g/t AgEq calculated at 52:1 silver to gold ratio. This value was the cut-off grade used for reporting mineral resources in the April 2012 Technical Report. Fifty nine samples originally assayed at Skyline were re-submitted to ALS for re-assay and 101 samples originally assayed at ALS were re-submitted to Skyline for re-assay. Samples re-assayed ranged from less than detection limit gold to 16.7 g/t Au and from 3.8 g/t Ag to 560 g/t Ag. Samples were selected semi-randomly to cover all grade ranges and all mineralized drill-holes by sorting the assay table by silver equivalent ("AgEq") grade and then picking every 10th sample for re-assay. Results of the re-assay program are acceptable and are shown graphically in Figures 11-6 and 11-7, for gold and silver respectively.

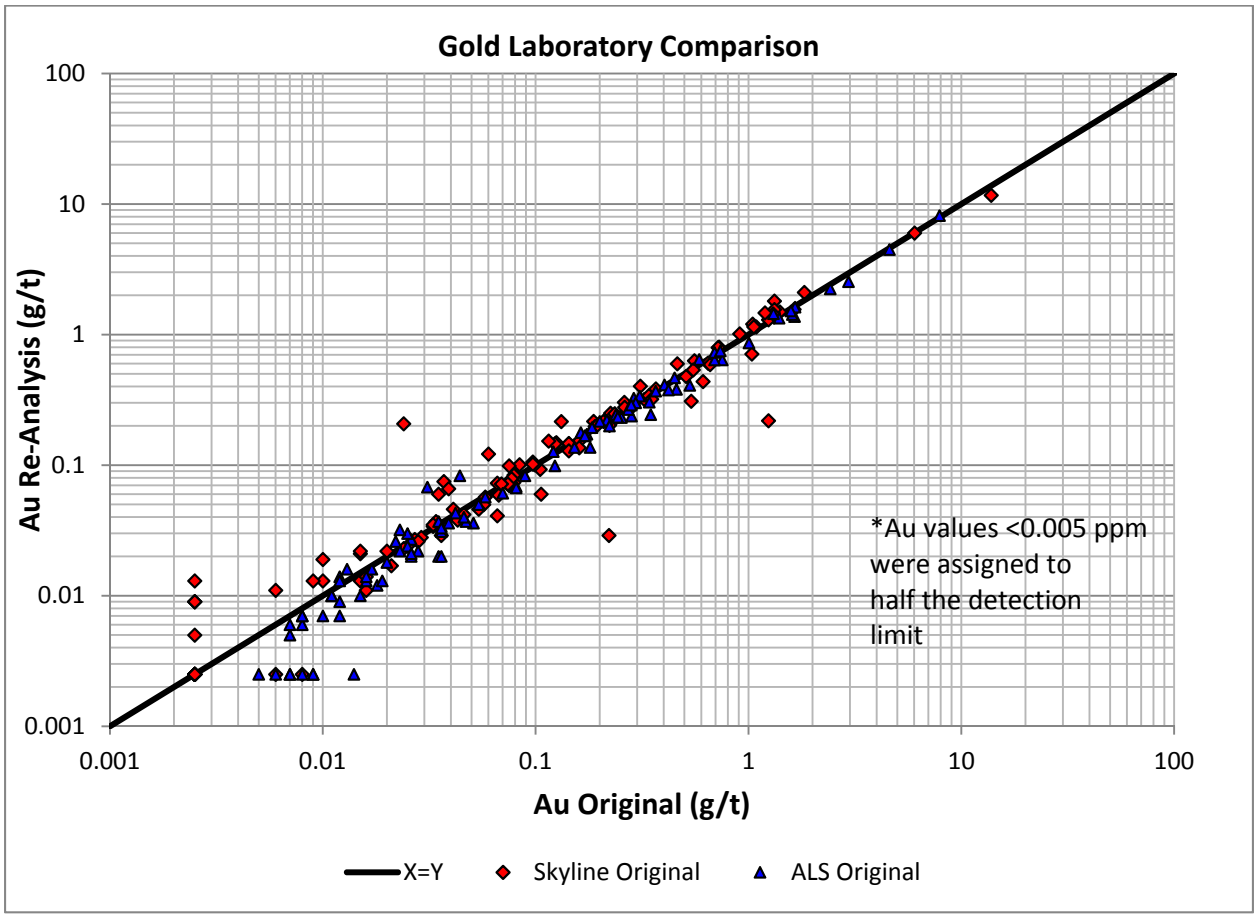


Figure 11-6 Gold Laboratory Comparison

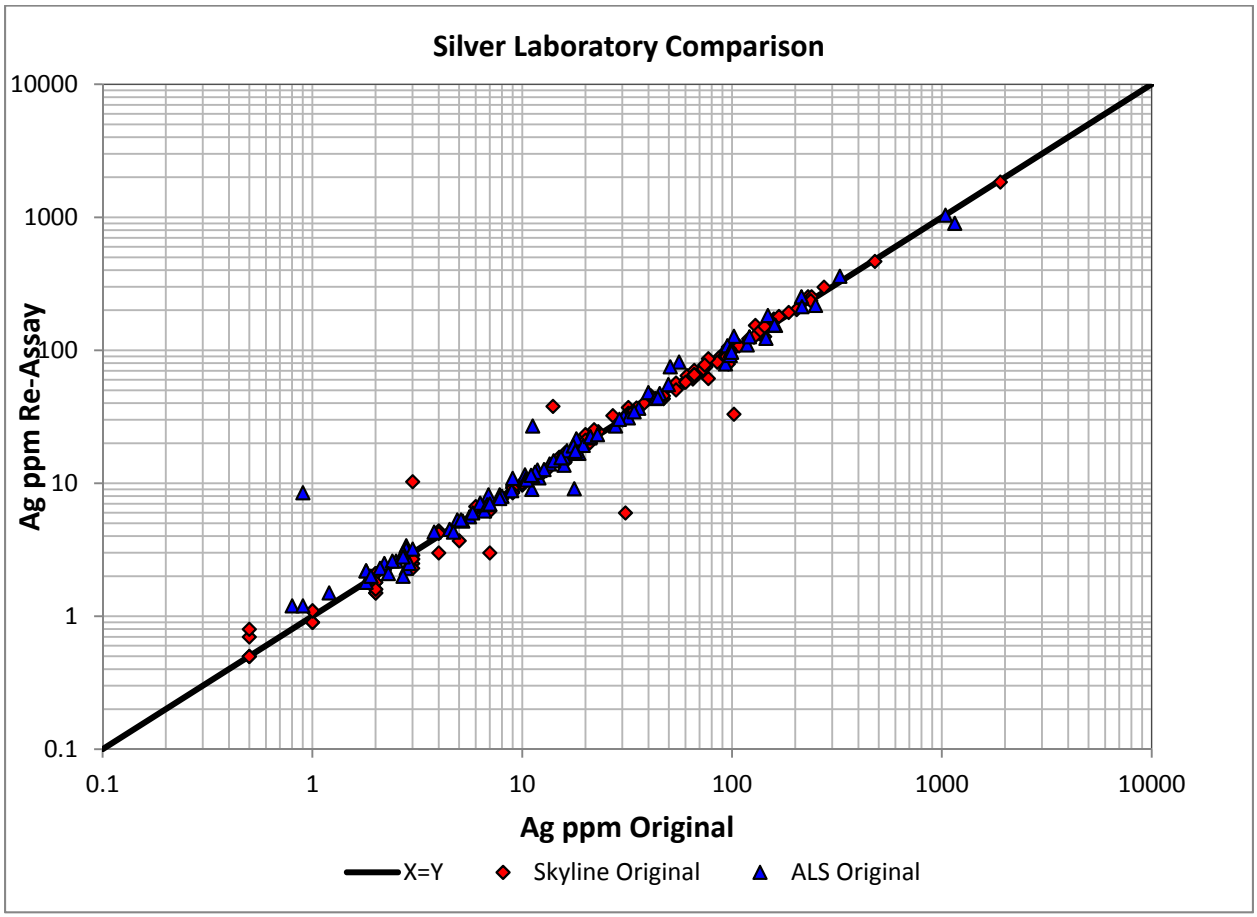


Figure 11-7 Silver Laboratory Comparison

Historic Pulp Re-analysis

The following section is taken directly from the NI 43-101 Technical Report on Resources for the Commonwealth Silver and Gold Project, Cochise County, Arizona, USA dated March 15, 2012, and amended April 11, 2012, prepared by SRK.

CSGM re-analyzed 97 underground channel sample pulps and 212 RC drilling pulps from historic sampling programs analyzed by Atlas to verify the historic analyses and add confirmation that the historic data in the database would be suitable for resource estimation. These samples were selected by SRK and were selected to represent the grade range observed at the Project and all portions of the resource area.

Overall, original silver analyses was lower than the duplicate analyses for both RC drilling and underground channel samples suggesting that the original samples are conservative. This was more pronounced in RC drill pulps than in the underground channel samples. Gold was more evenly distributed around the x=y axis.

The differences in silver analyses between the Atlas original and Skyline duplicate may be the result of differences in analytical techniques between Atlas and Skyline. Atlas performed analysis for silver using FA with a gravimetric finish while Skyline determined silver using ICP-OES.

The re-analysis for underground channel pulps displays more variability than the RC drilling pulps. This variability is most likely related to the grades. There are a number of higher grade gold and silver samples in the underground channel selection than in the RC drilling selection.

A review of historic pulp re-analyses data check programs conducted by previous operators showed similar results between all laboratories used during those programs. The samples were coarse reject material and the analyses showed good reproducibility between historic analyses at ALS Chemex, American Labs and the predecessor of the current Skyline also known as Skyline Laboratories.

HRC has independently reviewed the historical pulp duplicate program and agrees with the conclusions as present by SRK.

Twin Drillholes

The following section is taken directly from the NI 43-101 Technical Report on Resources for the Commonwealth Silver and Gold Project, Cochise County, Arizona, USA dated March 15, 2012 and amended April 11, 2012 prepared by SRK.

CSGM drilled 4 twin holes as part of sample verification. The twin holes were drilled using core tools to twin RC drill-holes. These drill-holes were distributed to test intercepts along the length and width of the Project mineralization. Twin holes are not a duplicate sample and are expected to intercept similar intervals at the same approximate drilled depths with similar magnitude of grade.

The four twin holes drilled performed well overall. Differences in magnitude of grade may be the result of the different drilling methods used (RC versus core drilling). The average percent difference between the RC and core drilling results was 30% for gold and 14% for silver.

HRC has independently reviewed the twin hole study program and agrees with the conclusions as presented by SRK.

QA/QC Actions

All QA/QC failures must be investigated to determine why the failure occurred. Should it be discovered that the failure is the result of an analytical failure, the failure plus several samples on either side should be re-analyzed. If there is more than one type of QA/QC failure in a given batch, the entire batch must be re-analyzed.

The QA/QC assay failures reported in the 2012 Technical Report, and presented to HRC personnel, were submitted for re-analysis of both gold and silver. All intervals were re-submitted with QA/QC samples and the results are acceptable.

Data Entry Validation Controls

CSGM input the drill-hole assay data into a Project specific Microsoft Access® database. Custom software created by Geomax, Inc. of Boulder, Colorado was used to merge digital assay files provided by the assay laboratory with a "from" and "to" interval file created by CSGM, with the sample number linking the two files.

Sample numbering errors were identified by the merging routine, while the Mapinfo® GIS software used in conjunction with Microsoft Access® identified overlapping sample intervals. CSGM uses these troubleshooting software routines to identify and repair any errors or inconsistencies.

Opinion on Adequacy

HRC concludes that the sample preparation, security and analytical procedures are correct and adequate for the purpose of this Technical Report. The sample methods and density are appropriate and the samples are of sufficient quality to comprise a representative, unbiased database.

12. Data Verification

The mineral resource estimate is based on the exploration database (Commonwealth.mdb) provided to HRC during the site investigation carried out on May 2nd and 3rd of 2013. Additional information was provided by CSGM in the form of electronic files.

HRC verified exploration data from the provided scanned files and visually inspected the paper logs maintained in CSGM's Tucson, Arizona office. The discussion contained herein is based on information that HRC considers reliable, including:

- Conversations with CSGM personnel;
- Personal investigation of the Commonwealth Project and Tucson office;
- Audit of exploration work conducted by CSGM; and
- Additional information obtained from historical reports and internal Company reports.

Received Data

HRC acquired the exploration drill-hole database during a site visit in May 2013. Drill-hole data, including collar coordinates, CSGM surveys, sample assay intervals, geologic logs, and QA/QC data were provided in a secure Microsoft Access database.

The present database has been updated to include 42 new core holes (CSG-017 – CSG-058) of which 5 were drilled as metallurgical holes (CSG-054 – CSG-058) and 5 certificates of legacy holes (CM-1 – CM-3, CM-5, and CM-6) which were completed or validated since the previous Technical Report on resources. The drill-hole database contains gold, silver and trace element assay analytical information for 15,449 sample intervals.

The current database was completely rebuilt since the 2012 Technical Report per SRK's recommendation. All assay data were entered directly from the original assay certificates by Geomax Information ("Geomax") of Boulder, Colorado. This database rebuild eliminated the errors identified by SRK that were due to three types of errors: inconsistent handling of less than detection limit samples, errors introduced by conversion form opt to ppm, and rounding errors.

Database Audit

HRC conducted a thorough audit of the current CSGM exploration drill-hole database. The following tasks were completed as part of the audit:

- Performed a mechanical audit of the database;
- Validated the geologic information compared to the paper logs;
- Validated the assay values contained in the CSGM database with assay certificates from CSGM; and
- Validated the assay values contained in the CSGM database by comparing with select, relevant historical assay certificates.

HRC limited the audit to the rock-type, assay, drill-hole collar, and survey data contained in the exploration drill-hole database (commonwealth.mdb).

Mechanical Audit

A mechanical audit of 4 of the tables from the database (Assays_ORIGINAL_MI, Collar_Query_for_MI, Survey_Query_for_MI, and Geology) was completed using Leapfrog mining software. The database was checked for overlaps, gaps, total drill-hole length inconsistencies, non-numeric assay values, and negative numbers. The following drill-holes were missing information:

- No Collar Data
  - W-002 (Hole excluded as essentially a twin of W-002A)
- No Assay Data
  - CHN-7153
  - CSG-024 (Not sampled)
  - CSG-025 (Not sampled)
  - CSG-050 (Not sampled)
  - CSG-054 (Metallurgical Hole)
  - CSG-055 (Metallurgical Hole)
  - CSG-056 (Metallurgical Hole)
  - CSG-057 (Metallurgical Hole)
  - CSG-058 (Metallurgical Hole)
- No Geology Data
  - CHN-7153
  - CM.4 (Database typo corrected to CM-4 to match collar, survey, and assay files)

A total of 213 drill-holes and 209 channel samples were imported into Leapfrog for validation.

Overlaps

A data entry error in drill-hole CSG-040 samples 111955 and 111956 was identified. The two intervals were corrected to match the CSGM sample ID paper logs in the Tucson office.

Gaps, Non-numeric Assay Values, and Negative Numbers

The software reported 191 missing intervals for each of the 37 elements analyzed. However, only 182 gaps in the data could be identified in the sheets, the majority of which are located at the collar of the drill-hole. Table 12-1 summarizes the gaps in the assay tables.

Table 12-1 Gaps in Database Audit

Hole ID	From	To	Hole ID	From	To	Hole ID	From	To
124-1	0.00	2.44	CSG-029	0.00	18.33	GG-1	27.43	32.00
124-2	0.00	3.05	CSG-030	0.00	2.44	GG-1	65.53	67.06
124-2	48.46	54.56	CSG-031	0.00	3.20	GG-12	0.00	1.52
124-3	0.00	3.05	CSG-031	130.00	148.00	GG-13	10.67	12.19
124-4	0.00	3.05	CSG-031	212.00	239.88	GG-13	38.10	39.62
3CHN-6669	7.62	10.67	CSG-032	0.00	140.00	GG-13	41.15	67.06
3CHN-6977	28.96	32.00	CSG-033	0.00	82.00	GG-3	25.91	28.96
3CHN-9	12.19	13.72	CSG-034	0.00	90.00	GG-6	12.19	13.72
C-94-2	85.34	88.39	CSG-035	0.00	106.00	GG-6	67.06	68.58
C-94-3	44.20	48.77	CSG-035	161.00	162.80	GG-7	0.00	7.62
C-94-3	51.82	54.86	CSG-036	0.00	98.00	GG-8	0.00	4.57
C-94-4	30.48	67.06	CSG-037	0.00	22.00	PCW-1	137.16	152.40
C-94-4	114.30	115.82	CSG-037	46.18	46.48	PCW-1	158.50	164.59
CM-2	0.00	6.10	CSG-037	50.14	50.44	PCW-1	179.83	181.36
CM-2	30.48	91.44	CSG-037	53.34	53.95	PCW-2	15.24	18.29
CM-3	0.00	7.62	CSG-037	56.69	57.00	PCW-2	67.06	68.58
CM-3	15.24	91.44	CSG-037	57.30	58.83	PCW-2	70.10	71.63
CM-4	0.00	15.24	CSG-037	61.87	63.09	PCW-2	80.77	83.82
CM-4	45.72	121.92	CSG-037	63.40	66.14	PCW-2	99.06	103.63
CM-5	0.00	60.96	CSG-037	69.95	71.93	PCW-2	144.78	156.97
CM-6	28.96	121.92	CSG-037	74.68	75.29	PCW-2	160.02	163.07
CSG-001	0.00	20.00	CSG-037	122.68	125.12	PCW-2	166.12	169.16
CSG-002	0.00	2.00	CSG-038	0.00	28.00	PCW-2	173.74	182.88
CSG-002	86.00	86.70	CSG-039	0.00	44.00	PCW-3	199.64	201.17
CSG-002	91.10	91.40	CSG-040	73.91	76.20	PCW-4	53.34	57.91
CSG-003	0.00	90.00	CSG-041	63.00	80.62	PCW-4	59.44	68.58
CSG-004	0.00	20.00	CSG-042	0.00	30.00	PCW-4	96.01	108.20
CSG-004	43.75	44.27	CSG-042	50.60	53.65	PCW-4	109.73	114.30
CSG-005	0.00	2.00	CSG-043	0.00	22.00	PCW-4	129.54	131.06
CSG-005	26.00	26.10	CSG-043	41.76	56.10	PCW-4	132.59	134.11
CSG-006	0.00	2.00	CSG-044	0.00	8.53	PCW-4	141.73	144.78
CSG-007	0.00	1.83	CSG-045	62.00	72.00	PCW-4	153.92	155.45
CSG-007	139.46	142.20	CSG-045	73.15	76.50	PCW-4	161.54	163.07
CSG-008	0.00	23.00	CSG-045	77.90	79.55	PCW-5	19.81	25.91
CSG-010	0.00	16.00	CSG-045	83.00	105.00	PCW-5	114.30	128.02
CSG-010	24.00	35.00	CSG-045	140.51	142.34	PCW-5	131.06	146.30
CSG-010	46.00	86.00	CSG-046	3.96	4.27	PHR-003	15.24	16.76

Hole ID	From	To	Hole ID	From	To	Hole ID	From	To
CSG-010	100.00	116.00	CSG-046	13.56	17.07	PHR-004	89.92	91.44
CSG-011	0.00	48.00	CSG-046	17.37	17.98	PHR-004	97.54	109.73
CSG-012	0.00	22.00	CSG-047	23.00	28.00	PHR-007	32.00	38.10
CSG-013	0.00	52.00	CSG-047	32.00	77.00	W-001A	99.06	100.58
CSG-013	52.50	58.00	CSG-047	110.19	112.47	W-012A	62.48	64.01
CSG-013	59.00	91.00	CSG-047	120.09	122.38	W-017A	27.43	28.96
CSG-013	100.00	108.00	CSG-048	0.00	46.00	W-017B	44.20	45.72
CSG-014	0.00	12.00	CSG-048	111.71	114.30	W-020A	3.05	4.57
CSG-014	71.74	75.90	CSG-049	50.00	106.00	W-021A	13.72	18.29
CSG-014	94.29	96.93	CSG-049	118.60	122.85	W-021B	36.58	42.67
CSG-015	0.00	18.00	CSG-052	0.00	2.44	W-022A	28.96	30.48
CSG-015	81.08	83.52	CSG-052	221.00	224.95	W-040	12.19	19.81
CSG-016	0.00	40.00	CSG-053	0.00	0.02	WC.10	59.44	111.25
CSG-017	2.13	12.50	CSG-053	224.00	242.01	WC.12B	0.00	25.91
CSG-020	0.00	22.00	E-1	30.48	41.15	WC.12B	35.05	115.82
CSG-021	0.00	90.00	E-1B	19.81	35.05	WC.14	70.10	71.63
CSG-021	198.12	200.25	E-2	24.38	27.43	WC.14	96.01	97.54
CSG-022	0.00	91.00	E-2	38.10	39.62	WC.15	0.00	67.06
CSG-023	0.00	46.00	E-2	73.15	82.30	WC.1A	0.00	1.52
CSG-026	0.00	7.00	E-3	36.58	42.67	WC.1A	53.34	54.86
CSG-026	112.00	163.68	E-3	79.25	83.82	WC.2	25.91	28.96
CSG-027	0.00	98.00	E-3	85.34	91.44	WC.4B	42.67	44.20
CSG-028	0.00	32.00	E-3	94.49	96.01	WC.5	39.62	41.15
CSG-028	150.49	151.49	---	---	---	WC.7	38.10	39.62

All of the non-positive numbers (-999) were assumed to be non-sampled intervals and were omitted from the dataset. No non-numeric assays were encountered in the audit. Table 12-2 below summarizes the number of intervals imported, the number of missing intervals, the number of non-positive values and the number of valid assays for each element.

Table 12-2 Database Import Summary

Element	Missing Interval	Non-Positive Values	Assay Values
CSG_Ag_ppm	191	1,261	14,188
CSG_Au_ppm	191	2,377	13,072
CSG_Cu_ppm	191	11,070	4,379
CSG_Pb_ppm	191	11,070	4,379
CSG_Zn_ppm	191	11,065	4,384
Al_pct	191	11,223	4,226
Ba_ppm	191	11,214	4,235
Be_ppm	191	11,236	4,213
Bi_ppm	191	11,129	4,320
Ca_pct	191	11,214	4,235
Cd_ppm	191	11,129	4,320
Co_ppm	191	11,129	4,320
Cr_ppm	191	11,214	4,235
Fe_pct	191	11,214	4,235
K_pct	191	11,223	4,226
Mg_pct	191	11,223	4,226
Mn_ppm	191	11,214	4,235
Mo_ppm	191	11,066	4,383
Na_pct	191	11,214	4,235
Ni_ppm	191	11,129	4,320
P_pct	191	11,223	4,226
Sr_ppm	191	11,214	4,235
Ti_pct	191	11,223	4,226
V_ppm	191	11,223	4,226
W_ppm	191	11,129	4,320
As_ppm	191	12,422	3,027
Ga_ppm	191	12,422	3,027
La_ppm	191	12,422	3,027
S_pct	191	12,422	3,027
Sb_ppm	191	12,422	3,027
Sc_ppm	191	12,422	3,027
Th_ppm	191	12,422	3,027
Tl_ppm	191	12,422	3,027
U_ppm	191	12,422	3,027

Survey Data

The collar coordinate elevations were compared to the corresponding elevation from the surface triangulation. Drill-hole WC.14 surveyed elevation was 1,364.89 m as compared to a topographical

elevation of 1,372.72 m resulting in a deviation of 7.83 m. HRC considers the topography (1m resolution) to be more precise and has moved the collar of drill-hole WC.14 to the surface. No other significant errors were identified in the collar survey file.

The 213 drill-holes audited in the database contained 151 single down-hole survey records at the collar and are assumed to have not been surveyed down-the-hole. Eighty-three of the unsurveyed holes were vertical and are expected to have very little down-hole deviation. The remaining 61 drill-holes have an average depth of 98.2 m with the longest drill-hole having a depth of 198.12 m. The unsurveyed drill-holes were evaluated on section and found to have similar locations for geologic and grade breaks as compared to the surrounding surveyed drill-holes, and therefore, are considered suitable for resource estimation.

Table Depth Consistency

The survey, assay, and geology tables maximum sample depth was checked as compared to the maximum depth reported in the collar table for each drill-hole. No intervals exceeded the reported drill-hole depths.

Certificates

HRC received original assay certificates in pdf format for all samples included in the current drill-hole database. A random manual check of 10% of the database against the original certificates was conducted, focusing on the five primary metals (Au, Ag, Cu, Pb, and Zn), with occasional spot checks of secondary constituents. HRC also conducted a random check of at least 2% of the highest (5%) assay values and continued to randomly spot check assays values throughout the modeling process.

Resource Estimation Data

Appendix C: Drill-hole and Underground Channel Sample Information Table summarizes the data received from CSGM that is pertinent to the estimation of mineral resources at the Commonwealth Project.

Check Samples

HRC independently collected two quarter-core samples and two channel samples from Level 3 of the underground mine workings for duplicate laboratory analysis. The duplicate samples were boxed and submitted to ALS Minerals via general post by HRC representative Jennifer J. Brown. Results of the duplicate analysis are summarized in Table 12-3, with the original sample assay results shown in bold. The assays of the selected quarter core and channel samples compare reasonably well to the original assays, with the largest deviations in the channel samples.

Table 12-3 HRC Check Sample Comparison

Sample ID	Sample Description	Gold (g/t)		Silver (g/t)	
		Original	Duplicate	Original	Duplicate
110356	Core	**0.489**	0.454	**108.0**	95.3
110396	Core	**2.56**	2.26	**149**	>100
112599	Core	**2.02**	2.95	**51.0**	52.6
3CHN-7100/10/15	Channel	**2.74**	1.99	**2.74**	5.70
3CHN-10/20/25	Channel	**3.42**	3.46	**39.4**	64.3

Adequacy of Data

HRC has reviewed CSGM's check assay programs and believes the programs provide adequate confidence in the data. Samples that are associated with the type 1 and 2 standard failures and the samples associated with erroneous blank samples have been re-analyzed prior to the completion of this Report and the results are acceptable.

All drill cores and cuttings from CSGM's drilling have been photographed. Drill logs have been digitally scanned and archived. The split core and cutting trays have been securely stored and are available for further checks.

13. Mineral Processing and Metallurgical Testing

Several metallurgical test work programs have been completed on the Project mineralization types by previous operators.

The results of the historical metallurgical testing are summarized in the report by R.A Forrest (1996). Forrest (1996) reports that approximately 200 cyanide bottle roll tests have been conducted by several of the previous operators of the property. The bottle roll tests were completed on various mineralization type composites and tailings material at different laboratories dating back to 1969. The majority of the bottle roll tests were completed on composites of the types of mineralized material and are not correlated to the current mineralization type nomenclature. As a result of the mineralized material type composites HRC has limited the metallurgical discussion to the 2012 CSGM and the 1995 and 1996 Atlas metallurgical testing programs. Table 13-1 below summarizes the Project metallurgical testing programs.

Table 13-1 Metallurgical Testing Summary

Company	Year	Test	Material
Basic Metals, Inc.	1969	Bottle Roll	Tailings
Platoro Mines, Inc.	1975	Bottle Roll	Tailings/UG
Santa Fe Mining, Inc.	1983	Bottle Roll	ROM Surface Material
Westland Exploration	1989	Bottle Roll	Drill Cuttings
Placer Dome	1990	Bottle Roll	Drill-hole Composites
ASARCO	1991	Bottle Roll	Composites
Chemgold Inc.	1991	Bottle Roll	Drill-hole Composites
Western States Mineral Corporation	1991	Bottle Roll	Drill-hole Composites
Westland Exploration	1992	Bottle Roll	Composites
Consolidated Nevada Goldfields, Inc.	1992	Bottle Roll	Composites
Harvest Gold	1994	Bottle Roll	UG/Surface Composites
Atlas Precious Metals, Inc.	1995	Bottle Roll	UG/Surface/Drill Core
Atlas Precious Metals, Inc.	1995	Column	Rock Type Composites
CSGM	2012	Bottle Roll, Grind Size, CN Strength, Roasting	Rock Type Composites

Testing and Procedures

In late 1995, Atlas Corporation, through its subsidiary, Atlas Precious Metals, Inc., submitted 12 bulk samples from surface and underground workings and two composites of drill core samples to KCA of Reno, Nevada for bottle roll testing. Four bulk samples were taken from the Rhyolite (0.891 tonnes), including two from surface. Four bulk samples and one core sample were taken from the Lower Andesite (0.814 tonnes). One core sample was taken from the Upper Andesite (0.079 tonnes). Two bulk samples were taken from the Bisbee (0.398 tonnes) and lastly, one bulk sample was taken from each of

the Main Vein and North Vein (0.352 tonnes). Bulk material was then separated into eight composite samples;

- Rhyolite low (0.206 g/t Au, 103.9 g/t Ag)
- Rhyolite medium (0.583 g/t Au, 117.6 g/t Ag)
- Rhyolite high (1.03 g/t Au, 119.7 g/t Ag)
- Lower Andesite (0.96 g/t Au, 79.4 g/t Ag)
- Lower Andesite (0.69 g/t Au, 53.1 g/t Ag)
- Vein (2.23 g/t Au, 140.2 g/t Ag)
- Bisbee (1.68 g/t Au, 17.8 g/t Ag)
- Upper Andesite (0.14 g/t Au, 128.9 g/t Ag)

Bottle Roll Tests

Three bottle roll tests were performed on each composite at -1/2 inch, -8 mesh and -100 mesh. Coarse samples were rolled intermittently to prevent attrition. Roll duration was 4 days. Table 13-2 below summarizes the results of the bottle roll tests:

Table 13-2 1995 Atlas Bottle Roll Test Summary

Sample ID	Rock Type	Crush Size	Head Grade (g/t)		Recoveries (%)		Reagents (kg/t)	
			Au	Ag	Au	Ag	NaCN	Lime
23302	Rhyolite	Minus 100	1.03	119.7	93	55	0.04	2.0
23302A	Rhyolite	Minus 8	1.03	119.7	76	31	0.08	1.0
23302B	Rhyolite	-1/2"	1.03	119.7	50	15	0.13	0.5
23305	Rhyolite	Minus 100	0.48	117.6	94	45	0.04	1.4
23305A	Rhyolite	Minus 8	0.48	117.6	64	25	0.23	1.0
23305B	Rhyolite	-1/2"	0.48	117.6	29	12	0.13	0.5
23316	Rhyolite	Minus 100	0.21	103.9	75	51	0.36	1.6
23316A	Rhyolite	Minus 8	0.21	103.9	67	24	0.18	1.1
23316B	Rhyolite	-1/2"	0.21	103.9	33	11	0.18	0.6
23317	Lower Andesite	Minus 100	0.96	79.5	93	52	0.46	1.6
23317A	Lower Andesite	Minus 8	0.96	79.5	83	35	0.18	1.2
23317B	Lower Andesite	-1/2"	0.96	79.5	67	28	0.23	0.7
23318	Veins	Minus 100	2.23	140.2	97	70	0.35	2.0
23318A	Veins	Minus 8	2.23	140.2	75	38	0.23	1.2
23318B	Veins	-1/2"	2.23	140.2	51	16	0.23	0.7
23329	Bisbee	Minus 100	1.68	17.8	95	25	0.35	2.8
23329A	Bisbee	Minus 8	1.68	17.8	91	18	0.18	1.9
23329B	Bisbee	-1/2"	1.68	17.8	84	13	0.18	1.6
23330	Lower Andesite	Minus 100	0.69	53.1	N/A	N/A	0.45	2.0
23330A	Lower Andesite	Minus 8	0.69	53.1	80	30	0.20	1.2
23330B	Lower Andesite	-1/2"	0.69	53.1	65	16	0.33	0.8
23601	Upper Andesite	Minus 100	0.14	128.9	N/A	N/A	0.46	2.4
23601A	Upper Andesite	Minus 8	0.14	128.9	40	36	0.73	1.4
23601B	Upper Andesite	-1/2"	0.14	128.92	40	32	0.83	1.3

Tail screen analysis of the bottle rolls showed that both gold and silver leach from various size fractions relatively equally. Both head and tail screen analysis comparisons show similar precious metal range distribution, suggesting almost uniform metal dissolution regardless of particle size.

Agglomeration Testing

Agglomeration tests were run on two-kilogram samples for each of the crush sizes using type II Portland cement, cured for 24 hours, and placed into 3-inch columns for 72 hours to simulate leaching conditions. Mineralized material height, agglomeration stability and percolation characteristics were recorded. All rock types preserved their stack height and maintained good percolation at a ½" crush size. The minus 8 mesh crush column test required the addition of 5 lbs of cement per short ton to maintain good percolation with no significant pellet breakdown observed on any of the agglomeration tests.

Column Testing

Twenty-two columns on minus ½" and minus 8 mesh were assembled from the eight composites; Rhyolite (5), Lower Andesite (3), Vein (1), Bisbee (1) and Upper Andesite (1). The columns were continuously drained drip leach tests for the first 90 days. The flow rate was calibrated from 0.004 to 0.006 gallons per minute per square foot of column surface to mimic production heap leaching conditions. Leach solution and "barren" solution was monitored for pH, NaCN, Au, Ag, and occasionally copper. Activated carbon was used to extract gold and silver from solution. Additional NaCN was added to maintain solution target levels. All minus 8 mesh columns were agglomerated using 4 pounds of cement and cured for 72 hours. Sixteen columns were terminated at 91 days. The remaining 6 columns, one with Lower Andesite and two with Rhyolite, were subjected to weekly leaching and draining to mimic an actual heap operation from day 92 through day 183. The results of the column tests for minus ½" and minus 8 mesh are presented in Tables 13-3 and 13-4, respectively.

Table 13-3 Minus 1/2" Feed Size Column Test Results

Sample ID	Rock Type	Days Leached	Head Grade (g/t)		Recoveries* (%)	
			Au	Ag	Au	Ag
23331	Rhyolite	91 (1)	1.03	119.66	56	18
23334	Rhyolite	143 (2)	1.03	119.66	62	21
23343	Rhyolite	91 (1)	0.48	117.60	61	16
23349	Rhyolite	91 (1)	0.21	103.88	54	15
23352	Rhyolite	143 (2)	0.21	103.88	61	17
23361	Lower Andesite	91 (1)	0.96	79.54	80	34
23364	Lower Andesite	143 (2)	0.96	79.54	75	34
23373	Vein	91 (1)	2.23	140.23	70	24
23384	Bisbee	91 (1)	1.68	17.83	90	23
23391	Lower Andesite	91 (1)	0.69	50.40	87	27
23602	Upper Andesite	90 (1)	0.21	128.91	50	35

(*) Recovery results are based on the daily solution assays vs. the average head grade
(1) Test ended
(2) Leach for 180 days

Table 13-4 Minus 8 Mesh Feed Size Column Test Results

Sample ID	Rock Type	Days Leached	Head Grade (g/t)		Recoveries* (%)	
			Au	Ag	Au	Ag
23337	Rhyolite	143 (2)	1.03	119.66	75	39
23340	Rhyolite	91 (3)	1.03	119.66	74	68
23346	Rhyolite	91 (1)	0.48	117.60	80	32
23355	Rhyolite	91 (1)	0.21	103.88	57	31
23358	Rhyolite	143 (2)	0.21	103.88	94	32
23367	Lower Andesite	91 (4)	0.96	79.54	87	43
23370	Lower Andesite	143 (2)	0.96	79.54	86	43
23376	Vein	91 (1)	2.23	140.23	86	50
23387	Bisbee	91 (4)	1.68	17.83	90	23
23394	Lower Andesite	91 (1)	0.69	50.40	85	34
23605	Upper Andesite	90 (1)	0.21	128.91	93	46

(*) Recovery results are based on the daily solution assays vs. the average head grade
(1) Test ended
(2) Leach for 180 days
(3) Detox with H_2O_2
(4) Fresh water rinse

CSGM Milling Study

In August 2012 CSGM commissioned KCA to perform metallurgical testing to optimize gold and silver recovery by conventional milling. Studies of cyanide strength vs. recovery and grind size vs. recovery were performed, and to further improve the silver recoveries, CSGM also had KCA do a limited study on precious metals recovery after roasting the mineralized material in a reducing gas atmosphere. Results of these studies were only marginally better than the column test recoveries as tested by KCA for Atlas in 1996 and KCA concluded that, "Overall, fine crushing may not be necessary based on the high metal extraction percentages from the bottle roll tests. No significant increase in metal extraction occurred from finely milling any of the composite material."

After completing the study with KCA, CSGM determined that the lower capital cost heap leach alternative would likely be the best processing route for the Commonwealth Project.

Summary of Recent Metallurgical Test Work

Composite samples were generated from coarse reject material from CSGM's 2011 and 2012 drilling programs. Composites were selected to represent the 5 rock types across a range of mineralized gold and silver grades. A total of 25 sample composites were generated, 5 for each rock type. The sample composites were then utilized to generate a total of 5 rock type composites. A portion from each rock type composite was then utilized to generate a single Master Composite. Portions from each composite were prepared and utilized for metallurgical test work. All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures.

Sample Preparation

Upon receipt, each sample was individually weighed and grouped according to compositing and rock type information provided by CSGM. Samples from each group were combined and blended to generate a total of 25 sample composites. Each sample composite was assigned a unique sample number (KCA Sample Nos. 65601 through 65625). Portions from each sample composite were then prepared and utilized for head analyses and cyanide bottle roll leach test work.

Each of the 25 sample composites were then grouped by one of 5 rock types: Kb, QV, Tal, Tau and Trb.

A portion of material from each sample composite group was then split out, combined and blended to generate a single rock type composite. Each rock type composite was assigned a unique composite number (KCA Composite Nos. 65633 through 65637). Portions of material from each rock composite were then prepared and utilized for head analyses for gold and silver, and cyanide bottle roll leach test work.

A portion of material from each of the 5 rock type composites was split out, combined and blended to generate a single Master Composite based on CSGM specifications. The master composite was blended on rock type percentages from the Atlas study, and overweights the Tau unit and underweights the QV and Tal units as compared to the current CSGM block model. The Master Composite was assigned a unique sample number (KCA Sample No. 65638). Portions of the Master Composite were then prepared and utilized for head analyses for gold and silver, and cyanide bottle roll leach test work.

Head Analyses

Portions of head material from the sample composites, rock type composites, and the Master Composite were ring and puck pulverized and analyzed for gold and silver by standard fire assay and wet chemistry methods. A hot cyanide shake test was also conducted on a portion of the pulverized head material from each of the 25 sample composites. The results of the head analysis and hot cyanide shake tests are presented below in Table 13-5:

Table 13-5 Sample Composite Head Analysis and Hot Cyanide Shake Tests

KCA Sample No.	Description	Head Grade (g/t)		pH	Recoveries (%)	
		Au	Ag		Au	Ag
65601	Bisbee, Kb	1.361	19.76	10.2	93%	31%
65602	Lower Andesite, Tal	0.309	49.90	10.1	78%	32%
65603	Rhyolite, Trb	0.135	46.70	10.3	74%	36%
65604	Vein, QV	0.519	116.01	10.2	92%	86%
65605	Upper Andesite, Tau	0.041	55.22	10.1	97%	32%
65606	Rhyolite, Trb	0.310	52.90	10.2	90%	41%
65607	Vein, QV	0.271	77.55	10.0	89%	77%
65608	Lower Andesite, Tal	0.511	42.70	10.0	94%	41%
65609	Rhyolite, Trb	0.573	77.91	10.2	87%	88%
65610	Rhyolite, Trb	0.569	179.61	10.1	91%	90%
65611	Vein, QV	0.459	117.82	10.2	96%	88%
65612	Bisbee, Kb	0.667	13.05	10.1	96%	42%
65613	Lower Andesite, Tal	0.495	31.30	9.9	93%	52%
65614	Lower Andesite, Tal	1.042	58.11	10.1	92%	87%
65615	Rhyolite, Trb	0.761	99.51	10.2	81%	88%
65616	Lower Andesite, Tal	1.959	96.91	10.1	91%	93%
65617	Upper Andesite, Tau	0.252	83.21	10.0	87%	91%
65618	Upper Andesite, Tau	0.179	82.30	10.0	89%	52%
65619	Bisbee, Kb	0.504	24.31	10.0	91%	56%
65620	Vein, QV	4.065	506.97	10.3	86%	84%
65621	Bisbee, Kb	1.953	45.81	10.2	96%	89%
65622	Upper Andesite, Tau	0.118	81.00	10.0	85%	63%
65623	Upper Andesite, Tau	0.033	72.91	10.1	61%	58%
65624	Bisbee, Kb	0.297	48.70	10.1	87%	73%
65625	Vein, QV	1.788	166.87	10.3	95%	91%
	Average -	0.767	89.88		89%	66%

Bottle Roll Leach Test Work

For each of the 25 sample composites, a 1,000 gram portion of head material was ring and puck pulverized to a target size of 80% passing 0.075 mm. The pulverized material was then utilized for a 96 hour bottle roll leach test. Each test was conducted at a NaCN concentration of 1.0 grams NaCN per liter of solution with sampling conducted for gold and silver. The results of the testing are presented in Tables 13-6 and 13-7 for gold and silver, respectively.

Table 13-6 Gold Summary of Cyanide Bottle Roll Test Results

KCA Sample No.	KCA Test No.	Description	Rock Type	Target p80 Size, mm	Target NaCN, gpL	Head Average, (g/t)	Calculated Head, (g/t)	Extracted, (g/t)	Avg. Tails, (g/t)	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65601	65626 A	CSG-COMP-001	Kb	0.075	1.0	1.361	1.386	1.350	0.036	97%	96	0.12	1.50
65602	65626 B	CSG-COMP-002	Tal	0.075	1.0	0.309	0.333	0.312	0.021	94%	96	0.07	2.00
65603	65626 C	CSG-COMP-003	Trb	0.075	1.0	0.135	0.134	0.114	0.021	85%	96	0.11	1.00
65604	65626 D	CSG-COMP-004	QV	0.075	1.0	0.519	0.343	0.307	0.036	89%	96	0.11	1.00
65605	65627 A	CSG-COMP-005	Tau	0.075	1.0	0.041	0.058	0.040	0.019	68%	96	0.04	1.50
65606	65627 B	CSG-COMP-006	Trb	0.075	1.0	0.310	0.296	0.274	0.022	92%	96	0.04	1.00
65607	65627 C	CSG-COMP-007	QV	0.075	1.0	0.271	0.256	0.234	0.022	91%	96	0.04	2.00
65608	65627 D	CSG-COMP-008	Tal	0.075	1.0	0.511	0.495	0.459	0.036	93%	96	0.04	2.00
65609	65628 A	CSG-COMP-009	Trb	0.075	1.0	0.573	0.541	0.524	0.017	97%	96	0.11	1.00
65610	65628 B	CSG-COMP-010	Trb	0.075	1.0	0.569	0.517	0.481	0.036	93%	96	0.07	1.50
65611	65628 C	CSG-COMP-011	QV	0.075	1.0	0.459	0.406	0.381	0.025	94%	96	0.09	1.00
65612	65628 D	CSG-COMP-012	Kb	0.075	1.0	0.667	0.636	0.579	0.057	91%	96	0.07	1.50
65613	65629 A	CSG-COMP-013	Tal	0.075	1.0	0.495	0.496	0.467	0.029	94%	96	0.11	2.50
65614	65629 B	CSG-COMP-014	Tal	0.075	1.0	1.042	0.866	0.841	0.026	97%	96	0.04	1.50
65615	65629 C	CSG-COMP-015	Trb	0.075	1.0	0.761	0.576	0.561	0.015	97%	96	0.04	1.50
65616	65629 D	CSG-COMP-016	Tal	0.075	1.0	1.959	1.655	1.529	0.127	92%	96	0.14	1.50
65617	65630 A	CSG-COMP-017	Tau	0.075	1.0	0.252	0.268	0.232	0.036	87%	96	0.04	1.50
65618	65630 B	CSG-COMP-018	Tau	0.075	1.0	0.179	0.170	0.151	0.019	89%	96	0.07	2.00
65619	65630 C	CSG-COMP-019	Kb	0.075	1.0	0.504	0.504	0.471	0.033	94%	96	0.07	1.00
65620	65630 D	CSG-COMP-020	QV	0.075	1.0	4.065	1.517	1.407	0.110	93%	96	2.38	1.00
65621	65631 A	CSG-COMP-021	Kb	0.075	1.0	1.953	1.561	1.520	0.041	97%	96	0.07	1.50
65622	65631 B	CSG-COMP-022	Tau	0.075	1.0	0.118	0.164	0.151	0.014	92%	96	0.07	2.00
65623	65631 C	CSG-COMP-023	Tau	0.075	1.0	0.033	0.088	0.074	0.014	84%	96	0.04	1.50
65624	65631 D	CSG-COMP-024	Kb	0.075	1.0	0.297	0.300	0.266	0.034	89%	96	0.07	1.50
65625	65631 E	CSG-COMP-025	QV	0.075	1.0	1.788	1.255	1.109	0.146	88%	96	0.09	1.00
Overall Average:						**0.767**	**0.593**	**0.553**	**0.040**	**91%**	**96**	**0.17**	**1.48**
Kb Average:						**0.956**	**0.877**	**0.837**	**0.040**	**94%**	**96**	**0.080**	**1.40**
QV Average:						**1.420**	**0.755**	**0.687**	**0.068**	**91%**	**96**	**0.54**	**1.20**
Tal Average:						**0.863**	**0.769**	**0.721**	**0.048**	**94%**	**96**	**0.08**	**1.90**
Tau Average:						**0.125**	**0.150**	**0.130**	**0.020**	**84%**	**96**	**0.05**	**1.70**
Trb Average:						**0.470**	**0.413**	**0.391**	**0.022**	**93%**	**96**	**0.07**	**1.20**

Table 13-7 Silver Summary of Cyanide Bottle Roll Test Results

KCA Sample No.	KCA Test No.	Description	Rock Type	Target p80 Size, mm	Target NaCN, gpL	Head Average, (g/t)	Calculated Head, (g/t)	Extracted, (g/t)	Avg. Tails, (g/t)	Ag Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65601	65626 A	CSG-COMP-001	Kb	0.075	1.0	19.757	17.546	4.295	13.251	24%	96	0.12	1.50
65602	65626 B	CSG-COMP-002	Tal	0.075	1.0	49.903	36.404	12.095	24.309	33%	96	0.07	2.00
65603	65626 C	CSG-COMP-003	Trb	0.075	1.0	46.697	45.534	15.534	30.000	34%	96	0.11	1.00
65604	65626 D	CSG-COMP-004	QV	0.075	1.0	116.006	133.196	68.790	64.406	52%	96	0.11	1.00
65605	65627 A	CSG-COMP-005	Tau	0.075	1.0	55.217	51.974	13.574	38.400	26%	96	0.04	1.50
65606	65627 B	CSG-COMP-006	Trb	0.075	1.0	52.903	49.024	13.024	36.000	27%	96	0.04	1.00
65607	65627 C	CSG-COMP-007	QV	0.075	1.0	77.554	78.026	30.720	47.306	39%	96	0.04	2.00
65608	65627 D	CSG-COMP-008	Tal	0.075	1.0	42.703	38.302	14.508	23.794	38%	96	0.04	2.00
65609	65628 A	CSG-COMP-009	Trb	0.075	1.0	77.914	81.450	34.753	46.697	43%	96	0.11	1.00
65610	65628 B	CSG-COMP-010	Trb	0.075	1.0	179.606	166.013	42.996	123.017	26%	96	0.07	1.50
65611	65628 C	CSG-COMP-011	QV	0.075	1.0	117.823	109.048	45.037	64.011	41%	96	0.09	1.00
65612	65628 D	CSG-COMP-012	Kb	0.075	1.0	13.046	11.785	3.925	7.860	33%	96	0.07	1.50
65613	65629 A	CSG-COMP-013	Tal	0.075	1.0	31.303	29.486	11.280	18.206	38%	96	0.11	2.50
65614	65629 B	CSG-COMP-014	Tal	0.075	1.0	58.114	50.918	36.518	14.400	72%	96	0.04	1.50
65615	65629 C	CSG-COMP-015	Trb	0.075	1.0	99.514	98.056	24.256	73.800	25%	96	0.04	1.50
65616	65629 D	CSG-COMP-016	Tal	0.075	1.0	96.909	88.805	64.505	24.300	73%	96	0.14	1.50
65617	65630 A	CSG-COMP-017	Tau	0.075	1.0	83.211	66.821	34.318	32.503	51%	96	0.04	1.50
65618	65630 B	CSG-COMP-018	Tau	0.075	1.0	82.303	76.905	21.396	55.509	28%	96	0.07	2.00
65619	65630 C	CSG-COMP-019	Kb	0.075	1.0	24.309	21.774	8.677	13.097	40%	96	0.07	1.00
65620	65630 D	CSG-COMP-020	QV	0.075	1.0	506.966	544.009	443.398	100.611	82%	96	2.38	1.00
65621	65631 A	CSG-COMP-021	Kb	0.075	1.0	45.806	47.090	34.730	12.360	74%	96	0.07	1.50
65622	65631 B	CSG-COMP-022	Tau	0.075	1.0	81.000	67.460	14.162	53.297	21%	96	0.07	2.00
65623	65631 C	CSG-COMP-023	Tau	0.075	1.0	72.909	66.957	9.957	57.000	15%	96	0.04	1.50
65624	65631 D	CSG-COMP-024	Kb	0.075	1.0	48.703	44.662	7.153	37.509	16%	96	0.07	1.50
65625	65631 E	CSG-COMP-025	QV	0.075	1.0	166.869	152.350	65.650	86.700	43%	96	0.09	1.00
Overall Average:						**89.882**	**86.944**	**43.010**	**43.934**	**40%**	**96**	**0.17**	**1.48**
Kb Average:						**30.324**	**28.571**	**11.756**	**16.815**	**37%**	**96**	**0.080**	**1.40**
QV Average:						**197.043**	**203.326**	**130.719**	**72.607**	**51%**	**96**	**0.54**	**1.20**
Tal Average:						**55.786**	**48.783**	**27.781**	**21.002**	**51%**	**96**	**0.08**	**1.90**
Tau Average:						**74.928**	**66.023**	**18.682**	**47.342**	**28%**	**96**	**0.05**	**1.70**
Trb Average:						**91.327**	**88.015**	**26.113**	**61.903**	**31%**	**96**	**0.07**	**1.20**

For each of the 5 rock type composites, two 1,000 gram portions of head material were milled in a laboratory rod mill to the target sizes of 80% passing 0.053 and 0.045 mm. The milled slurry was then utilized for a 96 hour bottle roll leach test. Each test was conducted at a NaCN concentration of 2.0 grams NaCN per liter of solution with sampling conducted for gold and silver. The results of the testing are presented in Tables 13-8 and 13-9 for gold and silver, respectively.

Table 13-8 Gold Summary of Cyanide Bottle Roll Mill Study

KCA Composite No.	KCA Test No.	Rock Type	Target p80 Size, mm	Calculated p80 Size, mm	Target NaCN, gpL	Au Calculated Head, (g/t)	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65633	65639 A	Bisbee	0.053	0.051	2.0	0.814	94%	96	1.63	1.00
65633	65639 B	Bisbee	0.045	0.044	2.0	0.850	94%	96	1.56	1.00
65634	65639 C	Vein	0.053	0.057	2.0	1.572	97%	96	2.81	1.00
65634	63639 D	Vein	0.045	0.045	2.0	1.508	96%	96	3.50	1.00
65635	65640 A	Lower Andesite	0.053	0.051	2.0	0.863	94%	96	0.58	1.50
65635	65640 B	Lower Andesite	0.045	0.042	2.0	0.821	94%	96	1.06	1.50
65636	65640 C	Upper Andesite	0.053	0.054	2.0	0.187	89%	96	0.93	1.00
65636	65640 D	Upper Andesite	0.045	0.047	2.0	0.160	85%	96	0.96	1.50
65637	65641 A	Rhyolite	0.053	0.051	2.0	0.345	92%	96	0.46	1.00
65637	65641 B	Rhyolite	0.045	0.045	2.0	0.397	94%	96	0.83	1.00

Table 13-9 Silver Summary of Cyanide Bottle Roll Mill Study

KCA Composite No.	KCA Test No.	Rock Type	Target p80 Size, mm	Calculated p80 Size, mm	Target NaCN, gpL	Ag Calculated Head, (g/t)	Ag Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65633	65639 A	Bisbee	0.053	0.051	2.0	28.24	36%	96	1.63	1.00
65633	65639 B	Bisbee	0.045	0.044	2.0	27.94	37%	96	1.56	1.00
65634	65639 C	Vein	0.053	0.057	2.0	206.57	64%	96	2.81	1.00
65634	63639 D	Vein	0.045	0.045	2.0	234.48	69%	96	3.50	1.00
65635	65640 A	Lower Andesite	0.053	0.051	2.0	55.44	63%	96	0.58	1.50
65635	65640 B	Lower Andesite	0.045	0.042	2.0	55.21	64%	96	1.06	1.50
65636	65640 C	Upper Andesite	0.053	0.054	2.0	77.36	34%	96	0.93	1.00
65636	65640 D	Upper Andesite	0.045	0.047	2.0	76.39	34%	96	0.96	1.50
65637	65641 A	Rhyolite	0.053	0.051	2.0	89.18	31%	96	0.46	1.00
65637	65641 B	Rhyolite	0.045	0.045	2.0	89.46	32%	96	0.83	1.00

For the Master Composite, a total of 20 bottle roll leach tests were conducted. Individual portions of head material were milled and utilized for 10 direct bottle roll leach tests, 2 agitated cyanide leach tests and 8 agitated cyanide leach tests utilizing roasted feed material.

For each direct bottle roll leach test conducted on Master Composite material, a 1,000 gram portion of head material was utilized. The direct bottle roll tests were conducted utilizing material milled in a laboratory rod mill to the target sizes of 80% passing 0.150, 0.106, 0.075, 0.053 and 0.045 mm. For each particle size, two 1,000 gram portions were milled and utilized for individual bottle roll leach tests conducted at a NaCN concentration of either 2.0 or 5.0 grams NaCN per liter of solution.

Each of the 5 direct bottle roll leach test conducted utilizing a NaCN concentration of 2.0 grams NaCN per liter of solution was run for a leach period of 96 hours, with sampling conducted for gold and silver content.

Each of the 5 direct bottle roll leach test conducted utilizing a NaCN concentration of 5.0 grams NaCN per liter of solution was run for a leach period of 24 hours, with sampling conducted for gold and silver content.

The test results of the Master Composite bottle roll tests are presented in Tables 13-10 and 13-11 for gold and silver, respectively:

Table 13-10 Gold Master Composite Cyanide Bottle Roll Test

KCA Sample No.	KCA Test No.	Target p80 Size, mm	Calculated p80 Size, mm	Target NaCN, gpL	Calculated Head, (g/t)	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65638	65642 A	0.150	0.145	2.0	0.623	93%	96	0.18	1.00
65638	65642 B	0.106	0.103	2.0	0.691	95%	96	0.46	1.00
65638	65642 C	0.075	0.053	2.0	0.657	92%	96	0.65	1.00
65638	65641 C	0.053	0.042	2.0	0.674	97%	96	0.66	1.00
65638	65641 D	0.045	0.032	2.0	0.585	97%	96	0.64	1.00
65638	65642 D	0.150	0.137	5.0	0.795	91%	24	1.16	1.00
65638	65643 A	0.106	0.108	5.0	0.712	95%	24	1.66	0.50
65638	65643 B	0.075	0.054	5.0	0.727	94%	24	1.80	0.50
65638	65643 C	0.053	0.041	5.0	0.753	96%	24	2.26	0.50
65638	65643 D	0.045	0.033	5.0	0.707	97%	24	2.24	0.50

Table 13-11 Silver Master Composite Cyanide Bottle Roll Test

KCA Sample No.	KCA Test No.	Target p80 Size, mm	Calculated p80 Size, mm	Target NaCN, gpL	Calculated Head, (g/t)	Ag Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65638	65642 A	0.150	0.145	2.0	72.21	41%	96	0.18	1.00
65638	65642 B	0.106	0.103	2.0	75.92	46%	96	0.46	1.00
65638	65642 C	0.075	0.053	2.0	68.81	44%	96	0.65	1.00
65638	65641 C	0.053	0.042	2.0	72.89	46%	96	0.66	1.00
65638	65641 D	0.045	0.032	2.0	67.97	43%	96	0.64	1.00
65638	65642 D	0.150	0.137	5.0	74.79	42%	24	1.16	1.00
65638	65643 A	0.106	0.108	5.0	73.98	43%	24	1.66	0.50
65638	65643 B	0.075	0.054	5.0	69.50	41%	24	1.80	0.50
65638	65643 C	0.053	0.041	5.0	67.39	42%	24	2.26	0.50
65638	65643 D	0.045	0.033	5.0	69.94	46%	24	2.24	0.50

A total of 2 agitated cyanide leach tests were conducted on portions of head material from the Master Composite. Each test was conducted utilizing a 1,000 gram portion of material milled in a laboratory rod mill to the target size of 80% passing 0.045 mm. Tests were continuously sparged with oxygen to maintain an oxygen concentration of approximately 15 ppm. Each test was run with differing concentrations of NaCN (10.0 and 2.0 grams NaCN per liter of solution) for a leach period of 48 hours, with sampling conducted for gold and silver content.

A total of 8 agitated cyanide leach tests were conducted on portions of roasted head material from the Master Composite. For each test, a 100 gram portion of nominal 1.70 mm material was milled in a laboratory rod mill to the target size of 80% passing 0.045 mm. The milled material was then roasted at a single temperature of 300°C, 400°C, 500°C or 600°C for 4 hours in a controlled (reducing) gas environment. The roasted material was then utilized for a 48 hour agitated cyanide leach test, with sampling conducted for gold and silver content.

Carbon in leach ("CIL") was utilized for select tests (KCA Test Nos. 65650 B and 65650 D). For each test utilizing CIL, granulated activated carbon ("GAC") was added to the slurry at the start of each test.

It should be noted that for select tests (KCA Test Nos. 65650 C and 65650 D), the nominal 1.70 mm test feed material was first roasted and then milled to the target size of 80% passing 0.045 mm.

The test results of the milled agitated cyanide leach tests and reduction roast tests are presented in Tables 13-12 and 13-13 for gold and silver, respectively:

Table 13-12 Gold Milled Agitated Cyanide Leach Tests, Reduction Roast Tests

KCA Sample No.	KCA Test No.	Leach Feed Material	Target p80 Size, mm	Calc. p80 Size, mm	Target NaCN, gpL	Roast Time, Hours	Roast Temp., ºC	Leach Type	Au Head Average, (g/t)	Au Calculated Head, (g/t)	Au Extracted, (g/t)	Au Tails, (g/t)	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65638	65648 A	Milled	0.045	--	10.0	0	none	Direct	0.621	0.684	0.658	0.026	96%	48	2.56	0.49
65638	65648 B	Milled	0.045	--	2.0	0	none	Direct	0.621	0.677	0.649	0.027	96%	48	0.88	0.49
65638	65649 A	Milled/Roast	0.045	--	5.0	4	300	Direct	0.621	0.701	0.679	0.022	97%	48	0.10	1.00
65638	65649 B	Milled/Roast	0.045	--	5.0	4	400	Direct	0.621	0.613	0.381	0.232	62%	48	0.32	1.00
65638	65649 C	Milled/Roast	0.045	--	5.0	4	500	Direct	0.621	0.697	0.575	0.122	83%	48	0.92	1.00
65638	65649 D	Milled/Roast	0.045	--	5.0	4	600	Direct	0.621	0.740	0.635	0.106	86%	48	1.42	1.00
65638	65650 A	Milled/Roast	0.045	0.034	5.0	4	600	Direct	0.621	0.789	0.629	0.161	80%	48	1.31	1.00
65638	65650 B	Milled/Roast	0.045	0.034	5.0	4	600	CIL	0.621	0.995	0.806	0.190	81%	48	3.33	1.00
65638	65650 C	Roast/Milled	0.045	0.027	5.0	4	600	Direct	0.621	0.766	0.608	0.158	79%	48	2.32	1.00
65638	65650 D	Roast/Milled	0.045	0.024	5.0	4	600	CIL	0.621	1.034	0.828	0.206	80%	48	4.97	1.00

Note: Elevated dissolved oxygen utilized during leach (± 15 mg/L) for all tests.
Note: Target p80 size generated from milling study conducted on non-roasted head material.

Table 13-13 Silver Milled Agitated Cyanide Leach Tests, Reduction Roast Tests

KCA Sample No.	KCA Test No.	Leach Feed Material	Target p80 Size, mm	Calc. p80 Size, mm	Target NaCN, gpL	Roast Time, Hours	Roast Temp., °C	Leach Type	Ag Head Average, (g/t)	Ag Calculated Head, (g/t)	Ag Extracted, (g/t)	Ag Tails, (g/t)	Ag Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
65638	65648 A	Milled	0.045	--	10.0	0	none	Direct	67.82	68.54	29.29	39.26	43%	48	2.56	0.49
65638	65648 B	Milled	0.045	--	2.0	0	none	Direct	67.82	72.88	34.17	38.71	47%	48	0.88	0.49
65638	65649 A	Milled/Roast	0.045	--	5.0	4	300	Direct	67.82	75.35	37.22	38.14	49%	48	0.10	1.00
65638	65649 B	Milled/Roast	0.045	--	5.0	4	400	Direct	67.82	72.82	33.77	39.04	46%	48	0.32	1.00
65638	65649 C	Milled/Roast	0.045	--	5.0	4	500	Direct	67.82	74.23	37.32	36.91	50%	48	0.92	1.00
65638	65649 D	Milled/Roast	0.045	--	5.0	4	600	Direct	67.82	72.18	45.19	26.98	63%	48	1.42	1.00
65638	65650 A	Milled/Roast	0.045	0.034	5.0	4	600	Direct	67.82	74.97	42.44	32.53	57%	48	1.31	1.00
65638	65650 B	Milled/Roast	0.045	0.034	5.0	4	600	CIL	67.82	67.22	37.68	29.54	56%	48	3.33	1.00
65638	65650 C	Roast/Milled	0.045	0.027	5.0	4	600	Direct	67.82	68.37	33.54	34.83	49%	48	2.32	1.00
65638	65650 D	Roast/Milled	0.045	0.024	5.0	4	600	CIL	67.82	62.69	28.48	34.21	45%	48	4.97	1.00

Note: Elevated dissolved oxygen utilized during leach (± 15 mg/L) for all tests.
Note: Target p80 size generated from milling study conducted on non-roasted head material.

Discussion

For each of the 25 sample composites, the head analyses and bottle roll calculated heads for gold and silver compared well for the majority of the samples tested.

Silver recoveries from the hot cyanide shake tests were significantly higher than recoveries from the subsequent test work. Four of the tests returned 90% or greater silver recovery. Additional test work is required to determine whether process scale methods can achieve recoveries similar to those from the hot cyanide shake tests.

For the 5 rock type composites, the bottle roll extraction results for gold and silver did not show a significant increase for smaller particle size.

For the Master Composite direct bottle roll leach tests, extraction results for gold and silver did not show a significant increase for smaller particle size or greater NaCN concentration.

For the Master Composite agitated cyanide leach tests, extraction results for gold and silver did not show a significant increase for greater NaCN concentration. Tests on roasted material utilizing CIL showed greater overall extractions for gold when compared to those without CIL.

Two of the tests on roasted material at 600°C showed significantly improved silver recoveries (57% and 63%). Further test work may be justified to determine if process scale methods can achieve high silver recoveries.

The calculated grind size for roasted material was significantly lower than the target grind size (80% passing 0.045 mm). This is most likely due to a changing mineral composition during roasting.

Overall, fine crushing may not be necessary based on the high metal extraction percentages from the bottle roll tests. No significant increase in metal extraction occurred from finely milling any of the composite material.

Conclusions

HRC agrees with CSGM's conclusion that, based on market conditions and comparing extensively tested metallurgical recovery rates associated with a lower capital cost heap leaching scenario to the preliminary results of metallurgical test work associated with a higher capital cost milling scenario, that while mining should remain open pit, the lower costs associated with heap leach processing increases the prospect for economic extraction of the mineral resources. HRC concludes that the metallurgical results presented in Table 13-14 indicate the most appropriate approach for the CSGM Project.

Table 13-14 Crush and Recovery Recommendations

Rock Type	Crush Size	Recoveries (%)	
		Au	Ag
Rhyolite	Minus 8 Mesh	78	35
Vein	Minus 8 Mesh	79	49
Lower Andesite	1/2"	81	33
Upper Andesite	1/2"	78	35
Bisbee	1/2"	80	23

This approach requires two different crush sizes, which can be accomplished with three-stage crushing, and campaign crushing to the finer size.

Significant Factors

HRC knows of no other significant factors that might affect the recovery of gold and silver on the Commonwealth Silver and Gold Project.

14. Mineral Resource Estimates

Zachary J. Black, SME-RM, a Resource Geologist with HRC is responsible for the mineral resource estimate herein. Mr. Black is a qualified person as defined by NI 43-101 and is independent of CSGM. HRC estimated the mineral resource for the Project from drill-hole data, using controls from the main rock types and a series of implicit grade shells with an Inverse Distance ("ID") algorithm.

The mineral resources presented this Technical Report are classified under the categories of Measured, Indicated and Inferred in accordance with the standards defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards - For Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 17, 2010. These resource classifications reflect the relative confidence of the grade estimates. HRC knows of no environmental, permitting, legal, socio-economic, marketing, political, or other factors that may materially affect the mineral resource estimate.

Block Model Physical Limits

HRC created a three dimensional ("3D") block model in MicroModel mining software. The block model was rotated 20 degrees east of north to align the rotated easting along the strike of mineralization. The block model was created with individual block dimensions of 6 x 3 x 3 m (xyz). The model origin is located at 611,235 east, 3,529,885 north, and at an elevation of 1,100 m above sea level ("masl"). The block model extends 1,302 m (217 blocks) in the rotated easting direction, 702 m (234 blocks) in the rotated northing direction, and vertically 402 m (134 blocks) to an elevation of 1,502 masl. All of the block model coordinates are stored as UTM NAD83, Zone 12 meters with elevations based on North American Vertical Datum ("NAVD") 88 (Darling, 2011). All property and minerals within the block model extents are owned or claimed by CSGM. Each of the blocks was assigned attributes of gold, silver, and gold equivalent grade, resource classification, rock density, tonnage factor, lithology, and a grade domain classification (Table 14-1).

Table 14-1 Block Model Labels

Block Model Label	Definition
Rock	Zone Code
Lith	Lithology Code
EqCode	Au Equivalent 60 Grade Shell Code
Au	Gold Block Grade
Ag	Sliver Block Grade
AuEq	Gold Equivalent Block Grade
TF	Tonnage Factor
CCat	Resource Classification Category
NNau	Nearest Neighbor Gold Block Grade
NNag	Nearest Neighbor Silver Block Grade

NNDist	Distance to Nearest Neighbor

Data Used for the Grade Estimation

HRC acquired the exploration drill-hole database during a site visit in May 2013. Drill-hole data, including collar coordinates, CSGM surveys, sample assay intervals, geologic logs, and QA/QC data were provided in a secure Microsoft Access database.

The present database has been updated to include 37 new core holes (CSG-017 – CSG-053), 5 certificates of legacy holes (CM-1 – CM-3, CM-5, and CM-6), and 5 metallurgical holes, which were completed or validated since the previous Technical Report on resources. The drill-hole database contains gold, silver, and trace element assay analytical information for 15,449 sample intervals.

Geologic Model

The mineral resource at the Commonwealth Project was modeled by constructing a geologic block model from the CSGM geologic interpretation provided by CSGM. The drill data was geostatistically analyzed to define the parameters used to estimate gold and silver grades into the 3D block model. Leapfrog 3D® geological modeling software was used to create 3D stratigraphic and mineralized domain solids, and MicroModel mining software was used to estimate gold and silver grades.

CSGM defined the structure and stratigraphy of the Commonwealth Project on electronic cross sections spaced 30 m apart and oriented perpendicular to the strike of the vein system, to best account for orientation of the deposit. HRC combined the CSGM subsurface interpretations with the surface geology to create 3D stratigraphic and mineralization models.

Visual evaluation of the assay data in the cross-sections revealed that while the majority of the mineralization is restricted to the veins and stockworks, related contact mineralization occurs in the footwall Bisbee group sediments and along other sub-parallel structures in both the hanging wall and footwall areas (Figure 14-1). HRC utilized a gold equivalent ("AuEq"), calculated at 60:1 gold to silver, to evaluate the mineralization along the structures and lithologic contacts within the Commonwealth Project. HRC found that a +0.15 g/t AuEq grade population represented a continuous zone of mineralization related to the distal alteration zone. A higher grade + 1.0 g/t AuEq grade population represented a continuous zone of higher grade material possibly related to silica flooding within the stockworks zone and the veins. Grade breaks were added at + 0.3 and + 0.5 g/t AuEq in order to better model the gradational boundaries of the structurally controlled areas. These grade breaks were used to construct grade domain boundaries representative of the lithology, alteration, and grade of the zone being modeled. The grade domains were used as both soft and hard boundaries designed to replicate the gradational changes identified in the drill-hole assay data.

The existing mine stopes were mapped by Harvest Gold Corporation and Atlas Precious Metals, Inc. between 1994 and 1996. A polygon outlining the mapped stope on each accessible level was used create a 3D solid representing the mined out material between levels. The solid was provided to HRC and combined with the provided level plan solids to code the block model with mined out material.

Bulk Density

Density tests were performed with core samples from exploration core holes. The tests were performed both by CSGM at the Pearce, Arizona field office and by commercial laboratories. Table 14-2 below summarizes the results.

Table 14-2 Rock Type Density Summary

All Data	KCA Density Data		Commonwealth Density Data		ALS Minerals Density Data	
Rock Type	Number of Samples	Avg Density (g/cm3)	Number of Samples	Average Density (g/cm3)	Number of Samples	Average Density (g/cm3)
Bisbee (Kb)	10	2.37	10	2.47	3	2.44
Vein	12	2.44	10	2.46	3	2.47
Lower Andesite (Tal)	31	2.43	20	2.42	3	2.46
Upper Andesite (Tau)	17	2.36	10	2.41	3	2.46
Rhyolite (Trb)	12	2.48	20	2.40	3	2.42
Sandstone (Tss)	N/A	N/A	5	2.36	3	2.28

HRC chose to use a weighted average of the three density results from Table 14-2 to populate the block model. Each block was assigned the density corresponding to the block rock code as populated by the lithologic model. A density of 0.00 was applied to the areas of mined out material. Table 14-3 presents the block model densities applied by lithology.

Table 14-3 Block Model Densities

Lithology	Lith Code	Density
Qal	1	2.25
Bisbee (Kb)	4	2.42
Vein	50	2.45
Lower Andesite (Tal)	3	2.43
Upper Andesite (Tau)	5	2.39
Rhyolite (Trb)	2	2.43
Sandstone (Tss)	14	2.33
Mine Workings (MW)	6	0.00

Estimation Domains

In order to accommodate statistical search parameters appropriate for individual mineralization styles and structural orientations, the block model was divided into two zones. The zones were delineated based on the Brockman Fault. The Brockman Fault resides on the western margin of the block model and is the only post mineralization structure affecting the known mineralization. The two zones of the Project area, Brockman and Main, were the starting demarcations for building the domains. Each of these zones was then divided into a domain based on the individual characteristics of the area (Figure 14-1).

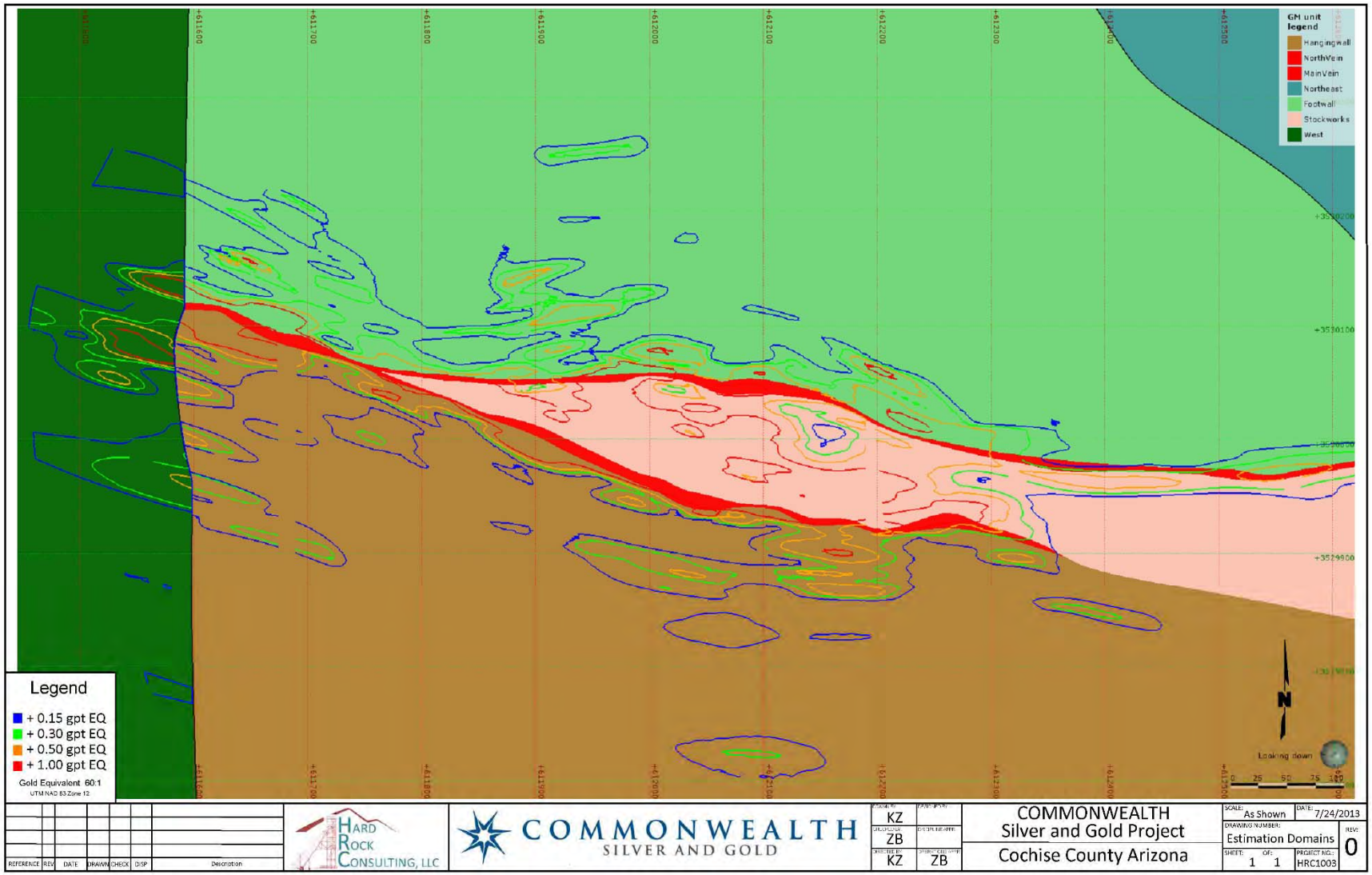


Figure 14-1 Estimation Domains

West of the Brockman Fault

This zone is dominated by a structurally controlled gold dominant vein postulated to be the continuance of the North Vein on the western (upthrown) side of the Brockman Fault (dark green area on Figure 14-1). This zone occurs primarily within the Bisbee Group sediments, and dips approximately 40 degrees to the South.

Main Zone

This zone was the main focus of historical mining activities and is defined by the two most important veins, the Main Vein and the North Vein (solid red area on Figure 14-1). Much of the mineralization on the Commonwealth Project occurs in the silicified and shattered structural wedge between the Main Vein and the North Vein (pink area on Figure 14-1). This mineralized zone is at least 1,350 m long and expands eastward from a point where the veins coalesce to a well mineralized exposed width of 125 m.

Subsidiary veins in the mineralized wedge that were named by Smith (1927) are, from the footwall of the Main Vein and proceeding north, the Footwall Vein, the Fischer Vein, the Smith Vein, the Hartery Vein and the Renaud Vein. Each of these veins varies from 1 to 4 m in width, with the Renaud Vein being the widest. These veins are generally sheeted veins sub-parallel to the Main Vein. They may also be considered the thick, high fluid flow arteries within the stockwork zone between the Main and North Veins (pink area on Figure 14-1).

Along the footwall (light green area on Figure 14-1) of the North Vein cretaceous marine sediments of the Bisbee Group also host mineralization and are chemically favorable hosts for gold. The Bisbee Group sediments are soft enough that they do not fracture well on faulting. Mineralization within the Bisbee Group sediments occurs as both vein type mineralization and some disseminated mineralization.

Only one vein has been identified in the footwall of the North Vein. The Eisenhart Vein strikes N70°W and dips 65 to 75° to the southwest. Smith (1927) recorded that the Eisenhart Vein is emplaced along the footwall of an andesite dike intruding into the Bisbee Group sediments, and can be observed both at ground surface and in underground workings.

Two veins, each less than one meter wide, occur in the hanging wall of the Main Vein (brown area on Figure 14-1).

Grade Shell Estimation

Leapfrog mining software was used to generate grade boundaries using a Radial Basis Function ("RBF") in conjunction with a dual kriging algorithm. Leapfrog implicitly defined the areas of the Project at cut-offs established by HRC at 0.15, 0.3, 0.5, and 1.0 g/t AuEq based on 4-foot composited intervals (Figure 14-1).

The grade boundaries have been used to define each of the estimation domains. The grade boundaries were used to code blocks and drill-hole assay samples residing within the individual grade boundary solids. Table 14-4 below defines the integer codes stored in the block model as "EqCode".

Table 14-4 Block Model Domain Codes

File Name	Zone	Grade (g/t AuEq)	Code
Eq15Main	Main	+0.15	100
Eq25Main	Main	+0.30	200
Eq50Main	Main	+0.50	300
Eq100Main	Main	+1.00	400
Eq15BMFlt	Brockman	+0.15	101
Eq30BMFlt	Brockman	+0.30	201
Eq50BMFlt	Brockman	+0.50	301
Eq100BMFlt	Brockman	+1.00	401

Blocks codes were restricted to the grade boundaries on either side of the block being estimated, i.e. blocks within the + 0.3 g/t AuEq grade boundary (200) used the closest samples from within the + 0.15 g/t AuEq (100), + 0.3 g/t AuEq (200), and + 0.5 g/t AuEq (300) boundary for grade estimation. The "grade" codes used as soft boundaries are generalized in Table 14-5.

Table 14-5 Soft Boundary Estimation Domains

Grade Boundary	Code	Soft Boundary Domains
Outside Grade Shells	0	Not Estimated
0.15 g/t AuEq	100	0, 200
0.3 g/t AuEq	200	100, 300
0.5 g/t AuEq	300	200, 400
1.0 g/t AuEq	400	300

Exploratory Data Analysis

Statistics are calculated for each of the grade shell domains listed in Table 14-4 for gold and silver, as shown in Tables 14-6 and 14-7, respectively.

Table 14-6 Gold Descriptive Statistics

Gold Descriptive Statistics							
Domain		Count	Min	Max	Mean	Std. Dev.	COV
Name	Code	n	g/t Au	g/t Au	g/t Au	g/t Au	
Eq15Main	100	1804	0.002	1.627	0.094	0.127	1.35
Eq15BMFlt	101	132	0.008	1.079	0.158	0.165	1.05
Eq30Main	200	1653	0.002	13.014	0.168	0.389	2.32
Eq30BMFlt	201	90	0.001	1.610	0.224	0.280	1.25
Eq50Main	300	2481	0.002	13.014	0.282	0.499	1.77
Eq50BMFlt	301	120	0.001	2.911	0.414	0.520	1.26
Eq100Main	400	3575	0.002	43.185	0.867	1.864	2.15
Eq100BMFlt	401	56	0.016	14.760	1.945	2.782	1.43
Rock	9999	62	0.005	0.230	0.038	0.052	1.37

Table 14-7 Silver Descriptive Statistics

Silver Descriptive Statistics							
Domain		Count	Min	Max	Mean	Std. Dev.	COV
Name	Code	n	g/t Ag	g/t Ag	g/t Ag	g/t Ag	
Eq15Main	100	2032	0.1	67.8	6.5	7.1	1.10
Eq15BMFlt	101	110	0.2	16.8	3.2	3.1	0.98
Eq30Main	200	1825	0.2	150.0	13.1	13.2	1.01
Eq30BMFlt	201	94	0.2	27.4	5.8	5.7	0.99
Eq50Main	300	2639	0.2	301.0	26.8	24.4	0.91
Eq50BMFlt	301	120	0.2	67.5	10.0	12.7	1.27
Eq100Main	400	3648	0.2	3170.0	77.0	102.9	1.34
Eq100BMFlt	401	56	1.4	273.0	38.3	58.4	1.52
Rock	999	62	0.1	16.0	2.0	2.9	1.46

HRC statistically compared the channel samples to each of the drilling methods implemented at the Project. Of the 607 channel samples, 465 (77%) reside within the +1.00 g/t AuEq grade shell and display similar statistical characteristics. Combining the channel samples with the drill-hole samples resulted in a 21% increase in the mean and a minimal increase in the coefficient of variation. This increase in the mean is warranted as the channel samples are taken from within underground workings and represent the best approximation of the remaining in situ grade surrounding the mine workings.

Capping

Grade capping is the practice for replacing any statistical outliers with a maximum value from the assumed sampled distribution. This is done statistically to better understand the true mean of the sample population. The estimation of highly skewed grade distribution can be sensitive to the presence of even a few extreme values. HRC utilized a log scale cumulative Frequency Plot ("CFP") of the assay

data for both gold and silver to identify the presence of statistical outliers (Figures 14-2 and 14-3, respectively). From these plots, it was determined gold samples should be capped at 10 g/t and silver samples should be capped at 1,000 g/t. The final dataset for grade estimate in the block model consists of 4 m down-hole composites capped at 10 g/t Au and 1,000 g/t Ag.

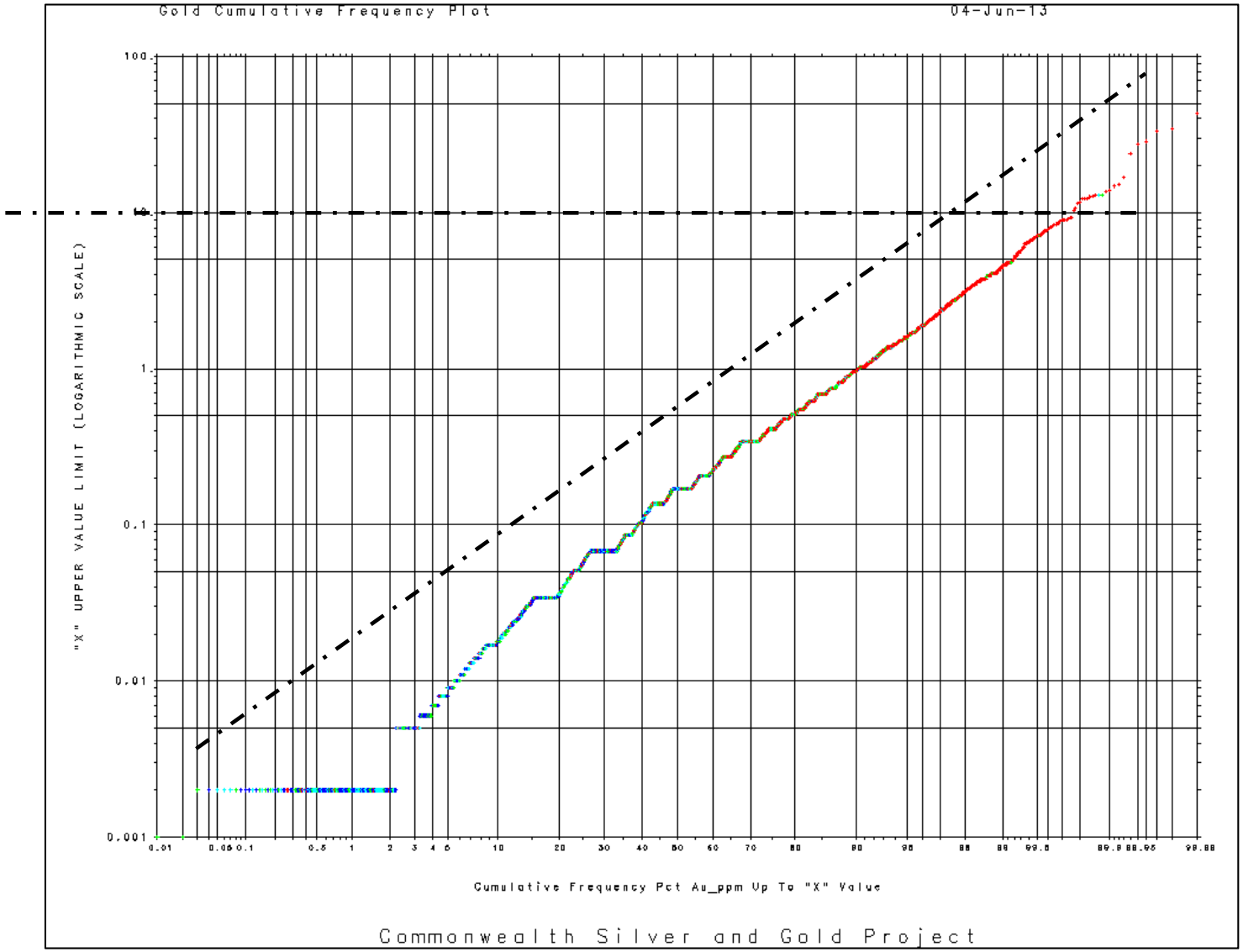


Figure 14-2 Gold Cumulative Frequency Plot

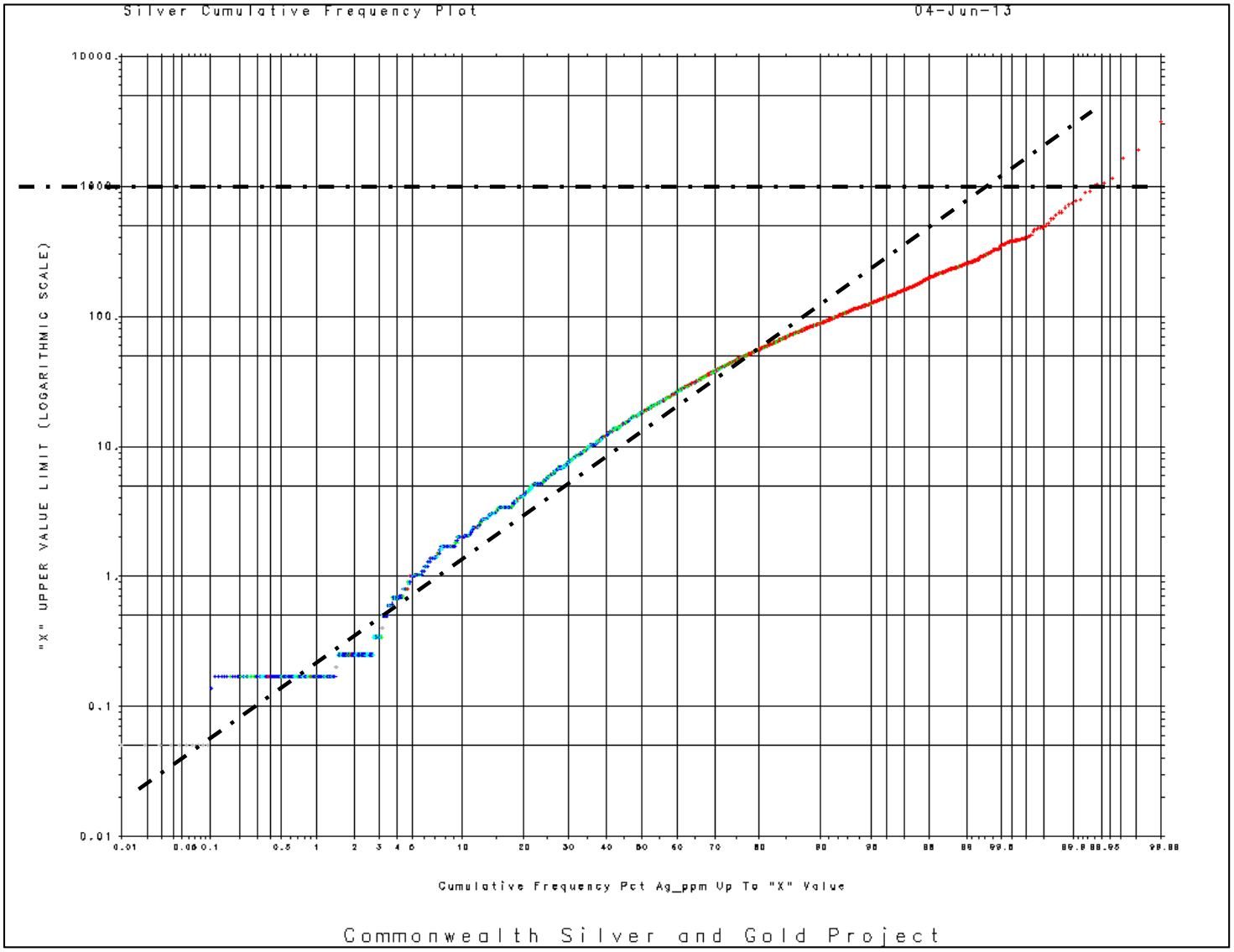


Figure 14-3 Silver Cumulative Frequency Plot

Compositing

HRC used down-hole compositing to standardize the drill-hole and channel gold and silver assay data set. An analysis of different composite lengths ranging from 1 to 15 m in length revealed that larger composites (>5m) begin to dilute the statistics and overestimate the mean of the sample population. HRC selected a 4 m down-hole composite as it is larger in length than the longest sample intervals and represents data that are not averaging mixed population samples down-hole (Figures 14-4 and 14-5 for gold and silver, respectively). The composites were broken at the boundary of each gold equivalent grade shell with a minimum acceptable composite length of 2 m and maximum of 6 m. The descriptive statistics for gold and silver composited data are presented in Tables 14-8 and 14-9, respectively.

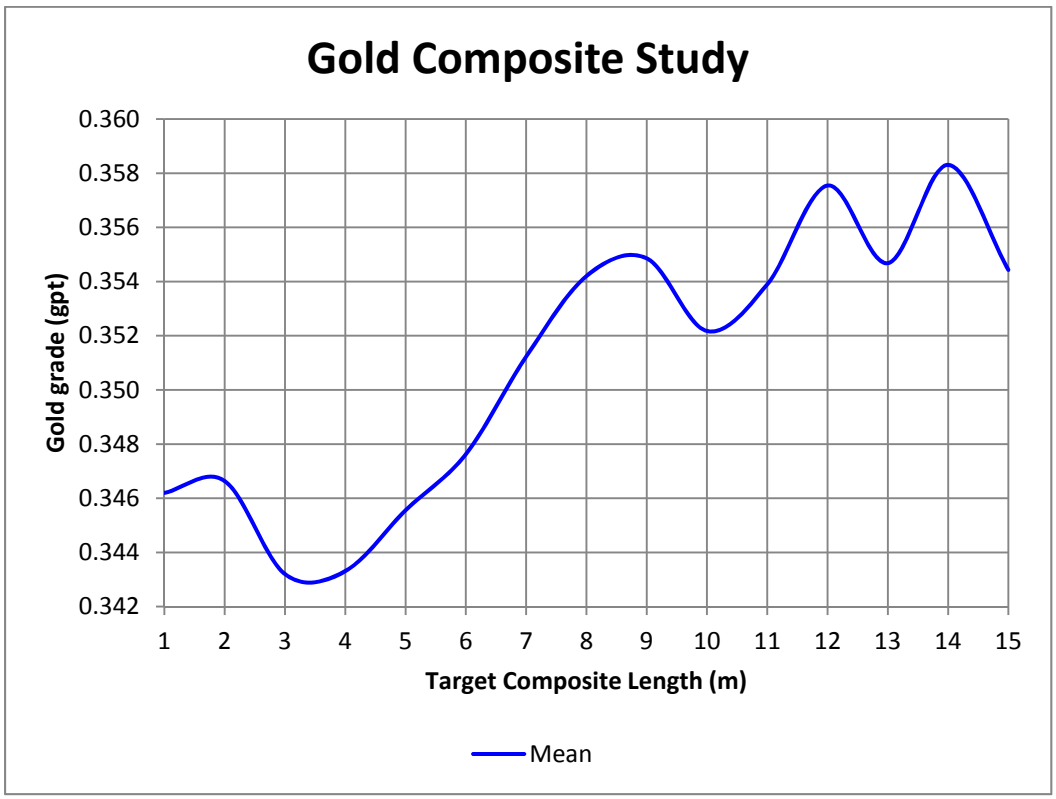


Figure 14-4 Composite Study of Mean Gold Grades

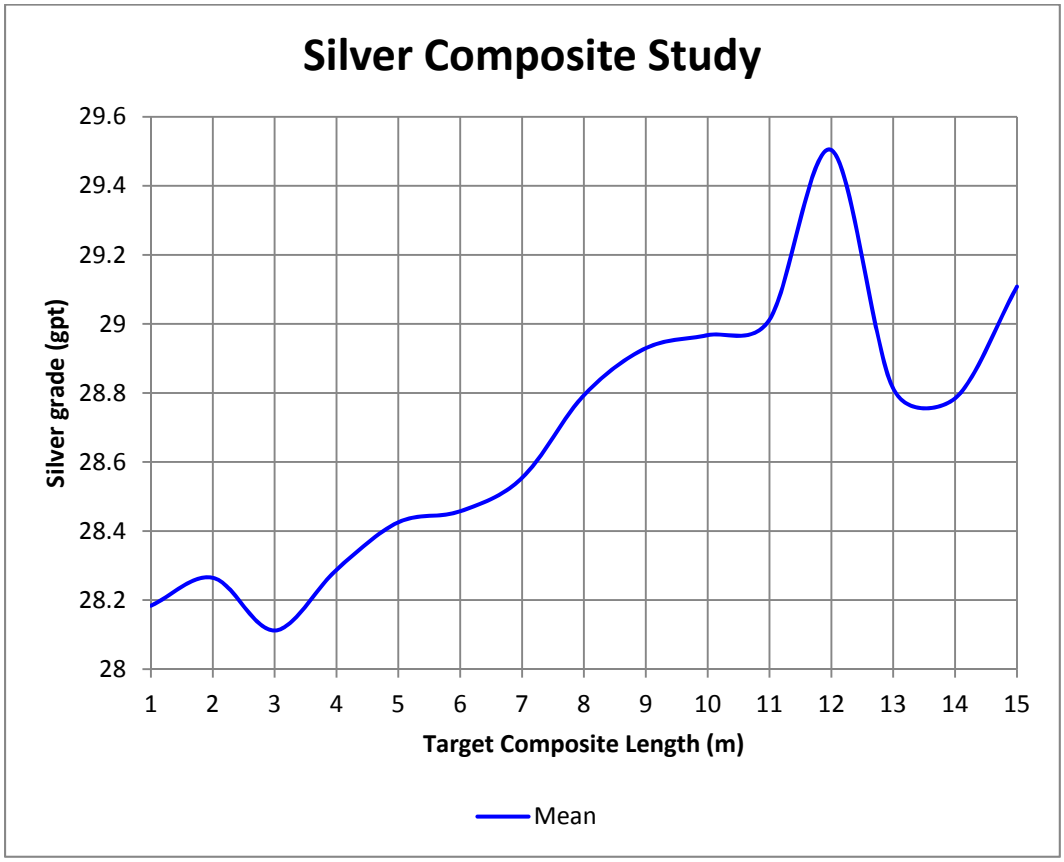


Figure 14-5 Composite Study of Mean Silver Grades

Table 14-8 Gold Composite Descriptive Statistics

Gold Descriptive Statistics							
Domain		Count	Min	Max	Mean	Std. Dev.	COV
Name	Code	n	g/t Au	g/t Au	g/t Au	g/t Au	
Eq15Main	100	658	0.002	0.800	0.100	0.100	0.99
Eq15BMFlt	101	44	0.034	0.608	0.170	0.122	0.72
Eq30Main	200	598	0.002	6.074	0.178	0.297	1.67
Eq30BMFlt	201	34	0.006	0.710	0.236	0.236	1.00
Eq50Main	300	884	0.002	7.356	0.287	0.367	1.28
Eq50BMFlt	301	42	0.003	1.677	0.445	0.403	0.91
Eq100Main	400	1278	0.002	9.844	0.834	1.014	1.22
Eq100BMFlt	401	21	0.047	4.595	1.653	1.117	0.68
Rock	9999	25	0.005	0.355	0.051	0.075	1.48

Table 14-9 Silver Composite Descriptive Statistics

Silver Descriptive Statistics							
Domain		Count	Min	Max	Mean	Std. Dev.	COV
Name	Code	n	g/t Ag	g/t Ag	g/t Ag	g/t Ag	
Eq15Main	100	740	0.2	41.8	6.6	6.0	0.91
Eq15BMFlt	101	36	0.2	9.4	2.9	2.2	0.76
Eq30Main	200	660	0.2	131.3	13.4	12.1	0.90
Eq30BMFlt	201	36	0.2	15.5	5.7	4.6	0.80
Eq50Main	300	948	0.2	171.9	26.7	20.0	0.75
Eq50BMFlt	301	41	0.2	31.7	9.4	9.2	0.98
Eq100Main	400	1301	0.2	888.7	74.6	64.9	0.87
Eq100BMFlt	401	21	3.1	134.7	35.1	36.1	1.03
Rock	9999	25	0.1	22.0	2.8	4.8	1.69

Variograms

A variography analysis was completed to establish spatial variability of gold and silver values in the deposit. Variography establishes the appropriate contribution that any specific composite should have when estimating a block volume value within a model. This is performed by comparing the orientation and distance used in the estimation to the variability of other samples of similar relative direction and distance.

Variograms were created for horizontal and vertical orientations in increments of 30° horizontally and 15° vertically. Search ellipsoid axis orientations were based on the results of the analysis. The sill and nugget values were taken from the omnidirectional and down-hole variograms, respectively. Tables 14-10 and 14-11 summarize the variogram parameters used for the analysis for gold and silver, respectively. The resultant variograms were used to define the search ellipsoid responsible for the sample selection in the estimation of each block (Table 14-12). An example directional spherical gold variogram is shown in Figure 14-6.

Table 14-10 Summary of Gold Variogram Parameters

Nugget (C0)	C1	C2
0.230	0.520	0.250

Axis	Range (meters)	Azimuth	Dip
Z	9/65	189	28
Y'	8/80	39	59
X'	10/160	106	-13

Modeling Criteria
Minimum number pairs required: 15
Sample variogram points weighted by # pairs

Table 14-11 Summary of Silver Variogram Parameters

Nugget (C0)	C1	C2
0.146	**0.466**	**0.389**

Axis	Range (meters)	Azimuth	Dip
Z	5/60	189	27
Y'	10/120	39	59
X'	45/160	106	-13

Modeling Criteria
Minimum number pairs required: 15
Sample variogram points weighted by # pairs

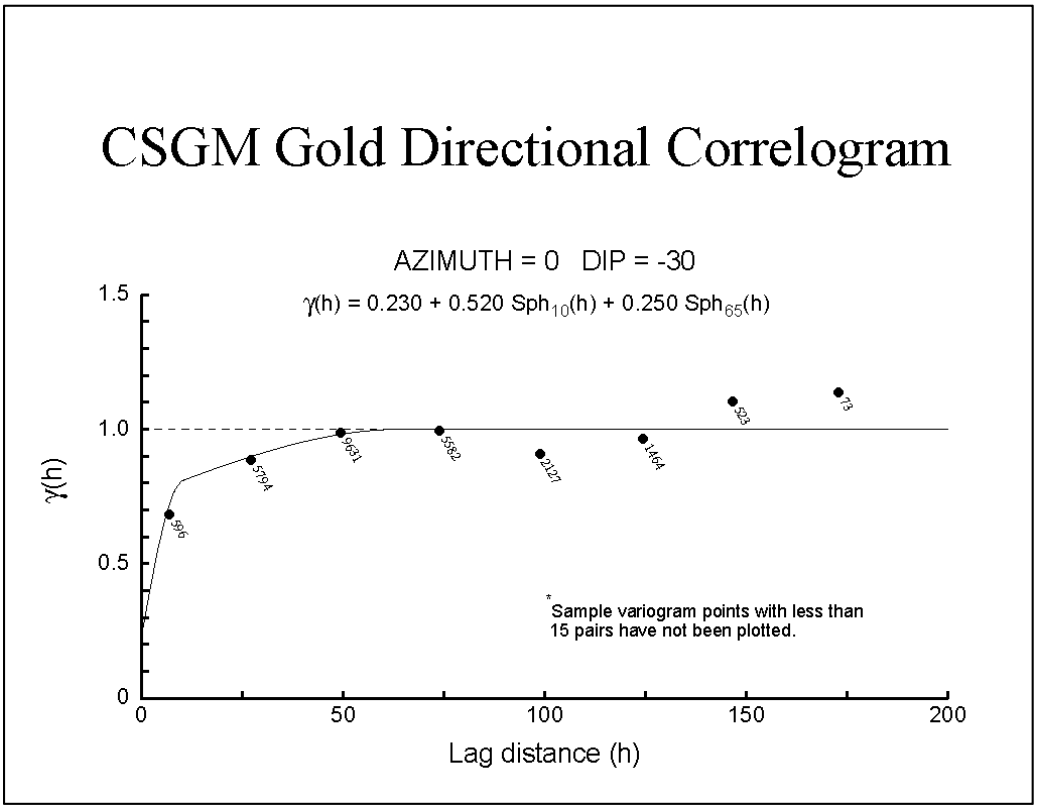


Figure 14-6 Spherical Gold Directional Variogram

In grade modeling, the variograms were used to establish search distances. Comparisons were made with ordinary kriging ("OK") and inverse distance-squared ($ID^{2.5}$) methods. The $ID^{2.5}$ method was selected for reporting due to better fit with drill-hole data throughout the model. The search ellipse parameters used for estimation are shown in Table 14-12 below. These parameters feature a major axis orientation striking 106 degrees and dipping 59 degrees to the southeast.

Estimation Methodology

Gold grades were estimated in each domain by using incremental search ellipses oriented in the direction of maximum continuity to provide an estimation of the gold and silver grade within every block inside the grade shells. Grades outside of the defined domains were not estimated. The estimation of each block was based on a factor of the distance in an anisotropic direction as established by the second structure range (Tables 14-10 and 14-11 for gold and silver, respectively) from the variogram model for the domain being estimated.

Inverse Distance to the power of 2.5 was used to estimate grade for all domains. Estimation parameters for each of the domains are presented in Table 14-12.

Table 14-12 Estimation Parameters

Metal	Gold			Silver		
# of composites	1st Pass	2nd Pass	3rd Pass	1st Pass	2nd Pass	3rd Pass
Min	3	3	2	3	3	2
Max	8	8	8	8	8	8
Max per Hole	2	2	2	2	2	2
Search Ellipsoid Distance						
Primary	80	160	240	80	160	240
Secondary	60	120	180	60	120	180
Tertiary	30	60	90	30	60	90

Mineral Resource Classification

HRC used a ratio of the calculated standard error to the distance to the closest sample of each block to classify mineral resources. The standard error of each block was calculated by dividing the kriging variance of each block by the total number of blocks estimated and multiplied by 1000 for scale. The scatter plot comparing the standard error to the distance of the nearest composite was evaluated and blocks were classified as Measured, Indicated, or Inferred based on the relationships highlighted in Figure 14-7. Measured resources are those blocks with a standard error less than 0.0011 (0 to 20 m from the closest composite interval). Indicated resources are those blocks with a standard error less than 0.0016 (1 to 180 m from the closest composite interval). Inferred resources are those blocks greater than or equal to 0.0016 standard error (4 to 240 m from the closest composite interval).

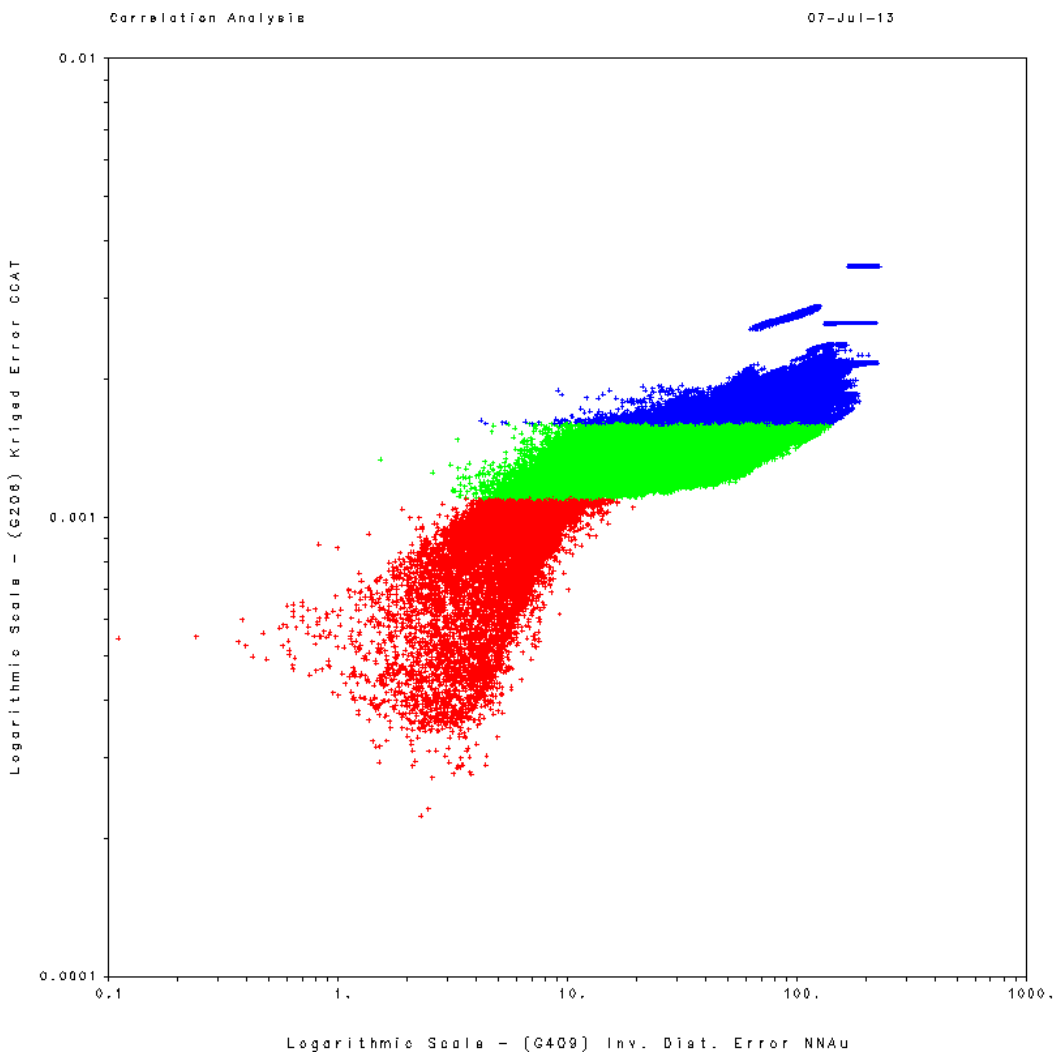


Figure 14-7 Standard Error (y-axis) versus Distance to Closest Sample (x-axis)
(Red – Measured, Green – Indicated, Blue – Inferred)

Model Validation

Overall, HRC utilized several methods to validate the results of the $ID^{2.5}$ method. The combined evidence from these validation methods validate the $ID^{2.5}$ method estimation model results.

Comparison with Ordinary Krige and Nearest Neighbor Models

Ordinary Krige ("OK") and Nearest Neighbor ("NN") models were run to serve as comparison with the estimated results from the $ID^{2.5}$ method. Descriptive statistics for the $ID^{2.5}$ method along with those for the OK, polygonal, NN, and drill-hole composites for gold and silver are shown in Tables 14-13 and 14-14, respectively.

Table 14-13 Gold Model Descriptive Statistical Comparison

Gold Model Descriptive Statistics						
Domain	Count	Min	Max	Mean	Std. Dev.	COV
Name	n	g/t Au	g/t Au	g/t Au	g/t Au	
Composite	3559	0.002	9.844	0.438	0.730	1.66
ID2.5	458918	0.002	8.533	0.304	0.349	1.15
OK	458918	0.004	6.529	0.300	0.315	1.05
NN	458918	0.002	9.844	0.322	0.528	1.64

Table 14-14 Silver Model Descriptive Statistical Comparison

Silver Model Descriptive Statistics						
Domain	Count	Min	Max	Mean	Std. Dev.	COV
Name	n	g/t Ag	g/t Ag	g/t Ag	g/t Ag	
Composite	3783	0.171	888.7	36.3	49.1	1.35
ID2.5	459911	0.177	748.1	23.0	28.7	1.25
OK	459859	0.074	620.3	22.8	26.9	1.18
NN	459911	0.171	888.7	23.1	37.1	1.61

The overall reduction of the maximum, mean, standard deviation, and coefficient of variation within the OK and $ID^{2.5}$ models represent an appropriate amount of smoothing to account for the point to block volume variance relationship. This is confirmed in Figure 14-8, comparing the cumulative frequency plots of each of the models and drill-hole composites.

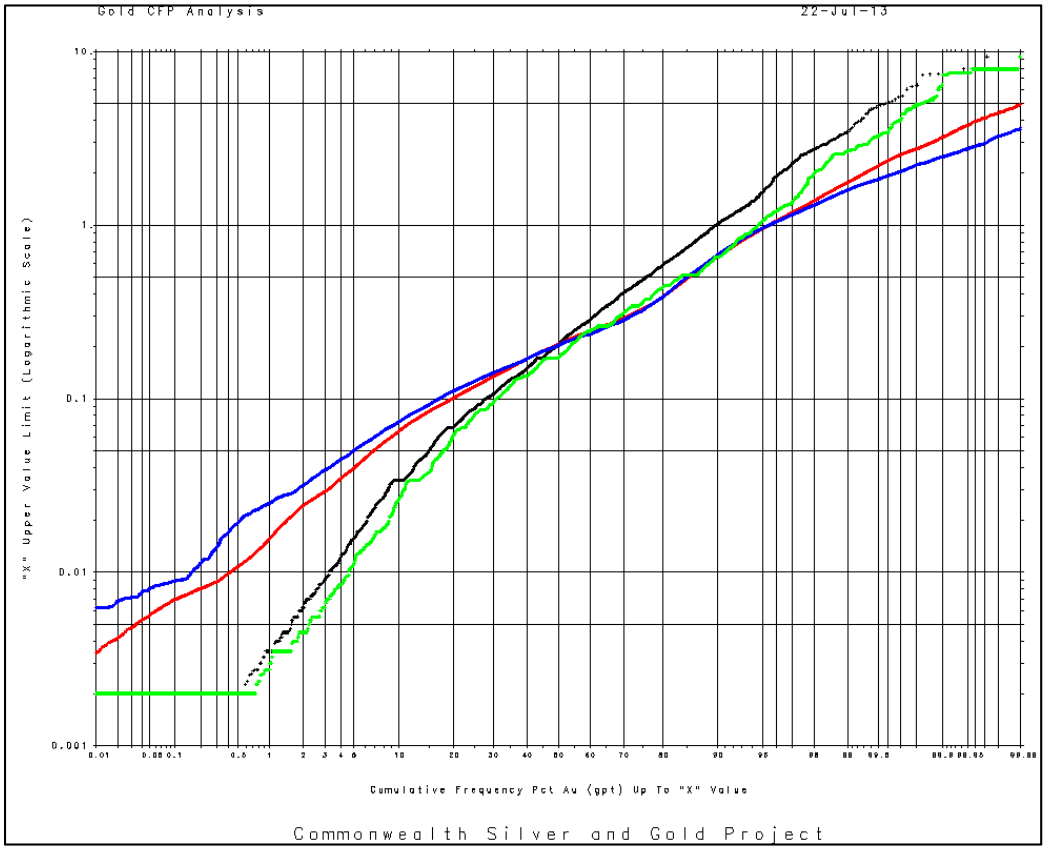


Figure 14-8 Cumulative Frequency Plot - Model Comparison
(Composites – Black, NN – Green, ID – Red, OK – Blue)

Swath Plots

Swath plots were generated to compare average gold and silver grade in the composite samples, estimated gold and silver grade from $ID^{2.5}$ method and the two validation model methods (OK and NN). The results from the $ID^{2.5}$ model method, plus those for the validation OK model method are compared using the swath plot to the distribution derived from the NN model method and the composites used in the estimation.

For comparison purposes, assay data from the 4-meter composite samples are included in the swath plots along with the model results.

Six swath plots were generated:

- Figure 14-9 shows average gold grade from west to east (rotated);
- Figure 14-10 shows average gold grade from south to north (rotated);
- Figure14-11 shows average gold grade in the 3-meter benches, from bottom to top;
- Figure 14-12 shows average silver grade from west to east (rotated);
- Figure 14-13 shows average silver grade from south to north (rotated); and
- Figure14-14 shows average silver grade in the 3-meter benches, from bottom to top.

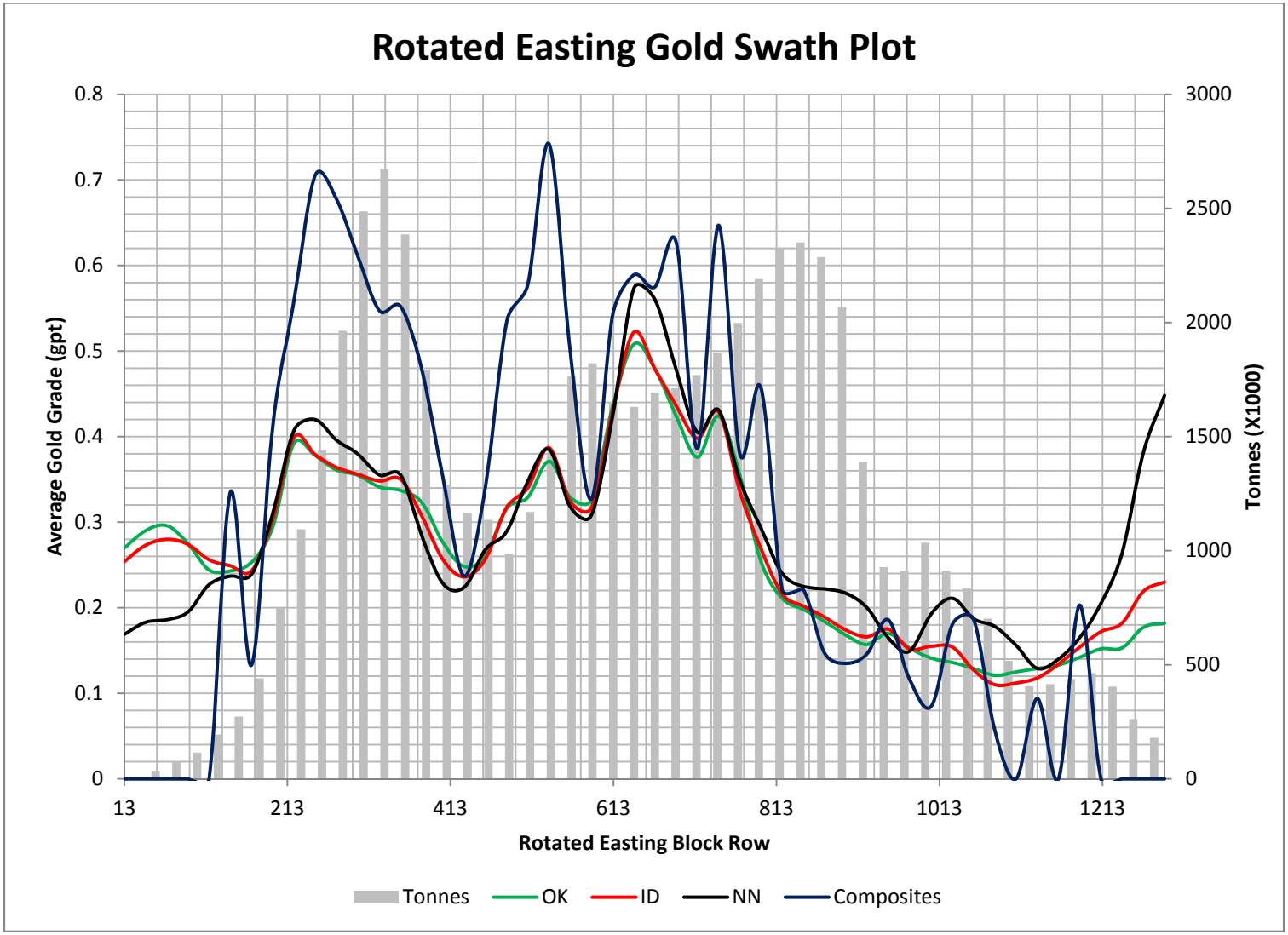


Figure 14-9 Rotated Easting Gold Swath Plot

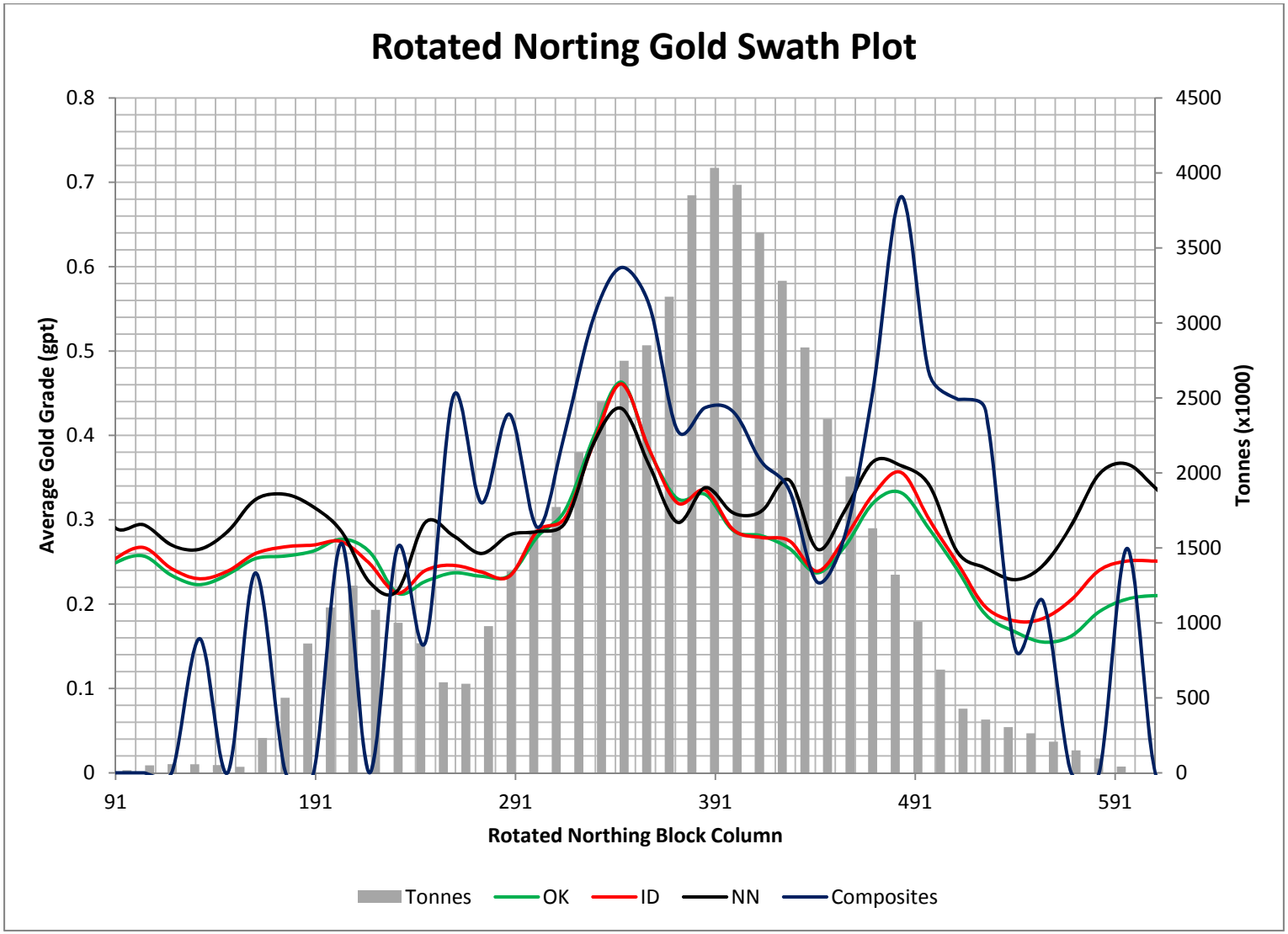


Figure 14-10 Rotated Northing Gold Swath Plot

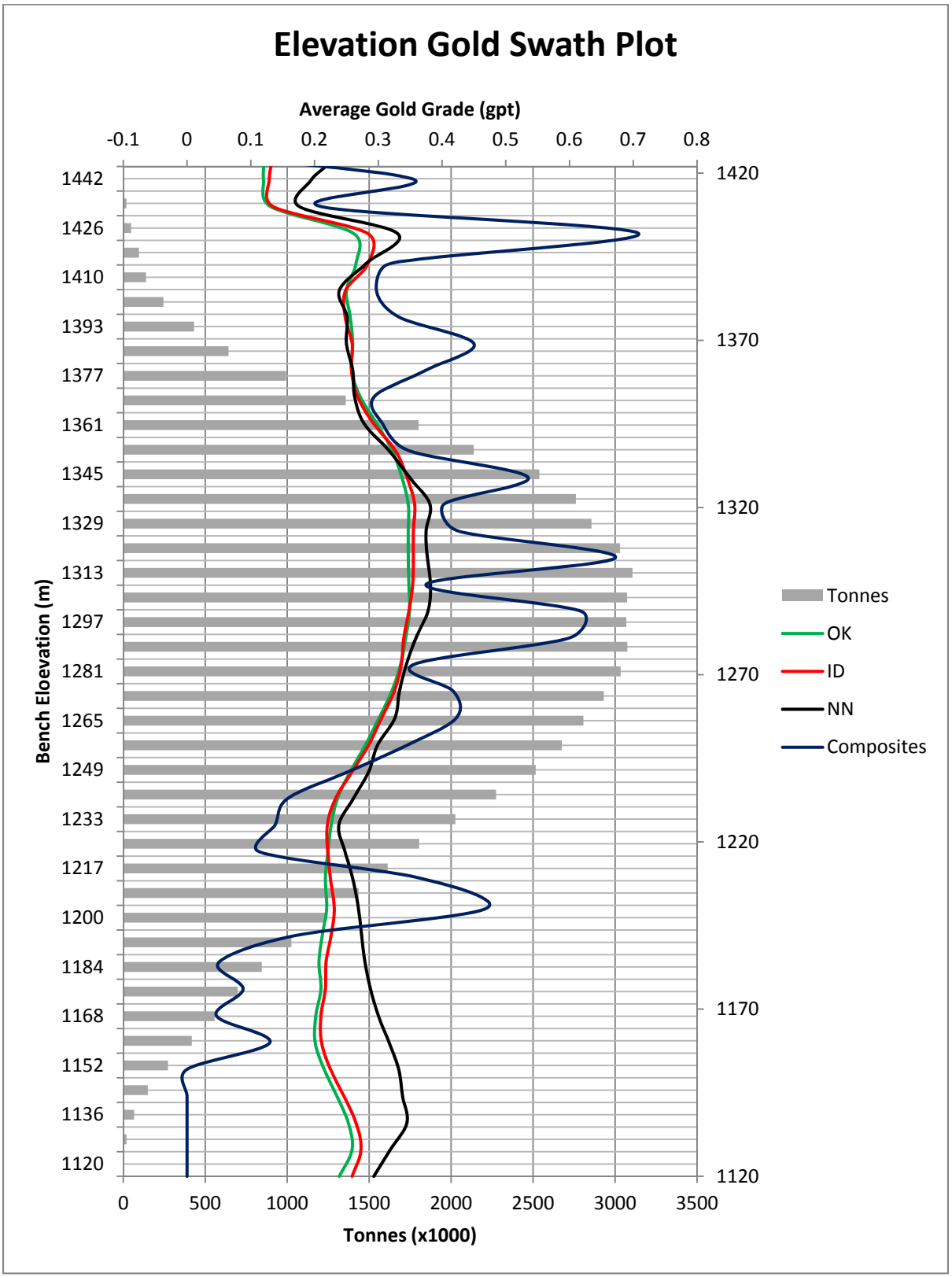


Figure 14-11 Elevation Gold Swath Plot

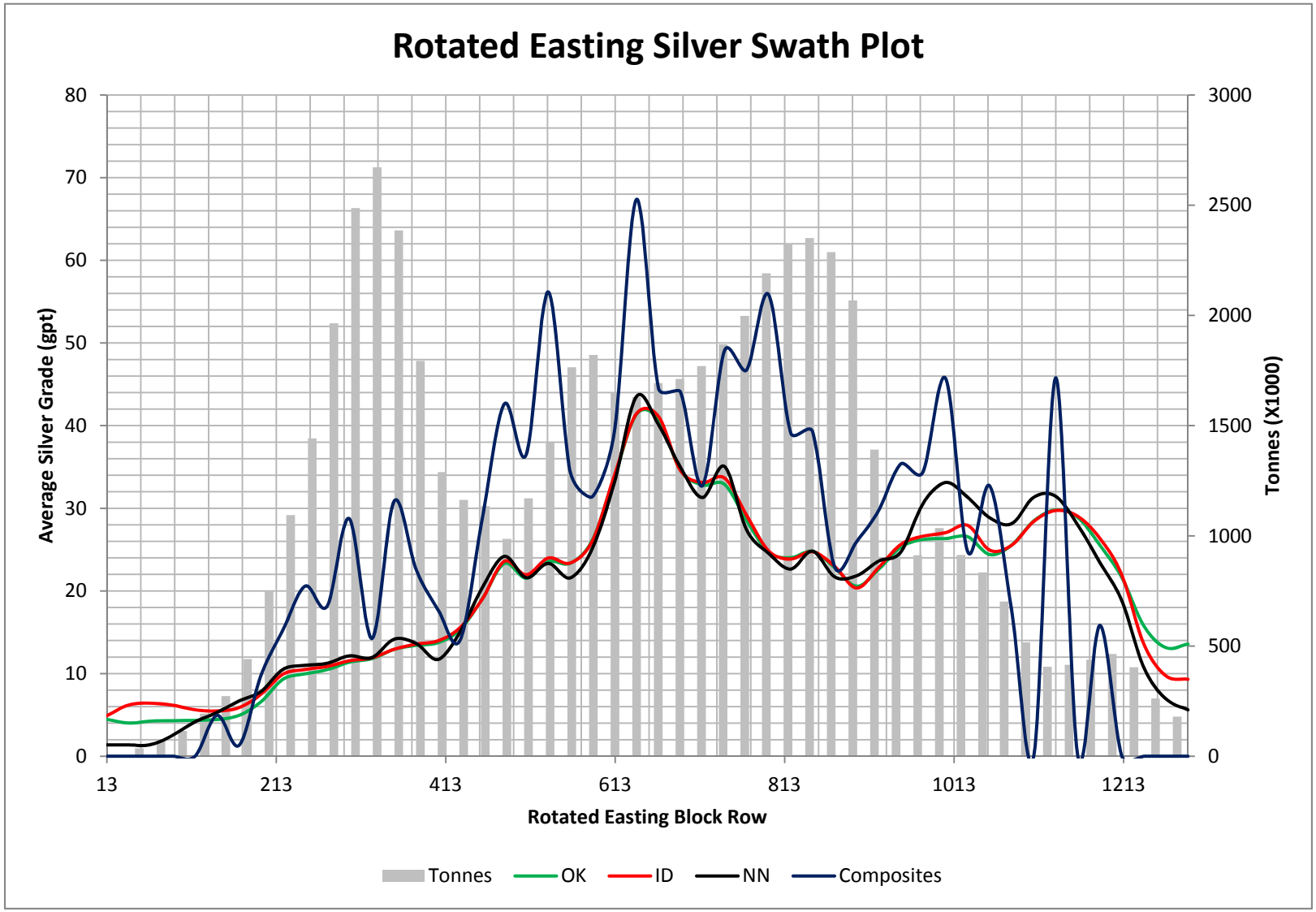


Figure 14-12 Rotated Easting Silver Swath Plot

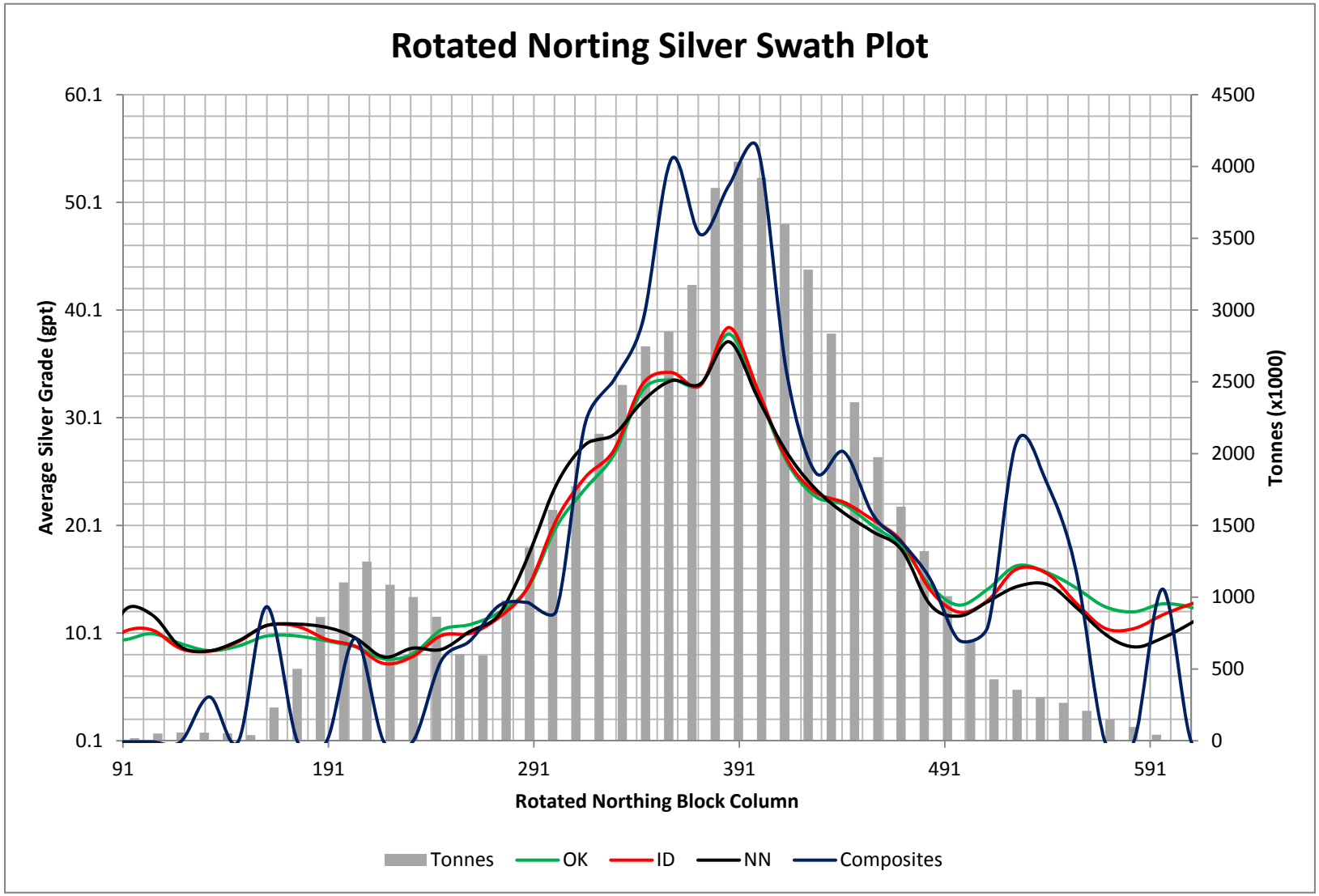


Figure 14-13 Rotated Northing Silver Swath Plot

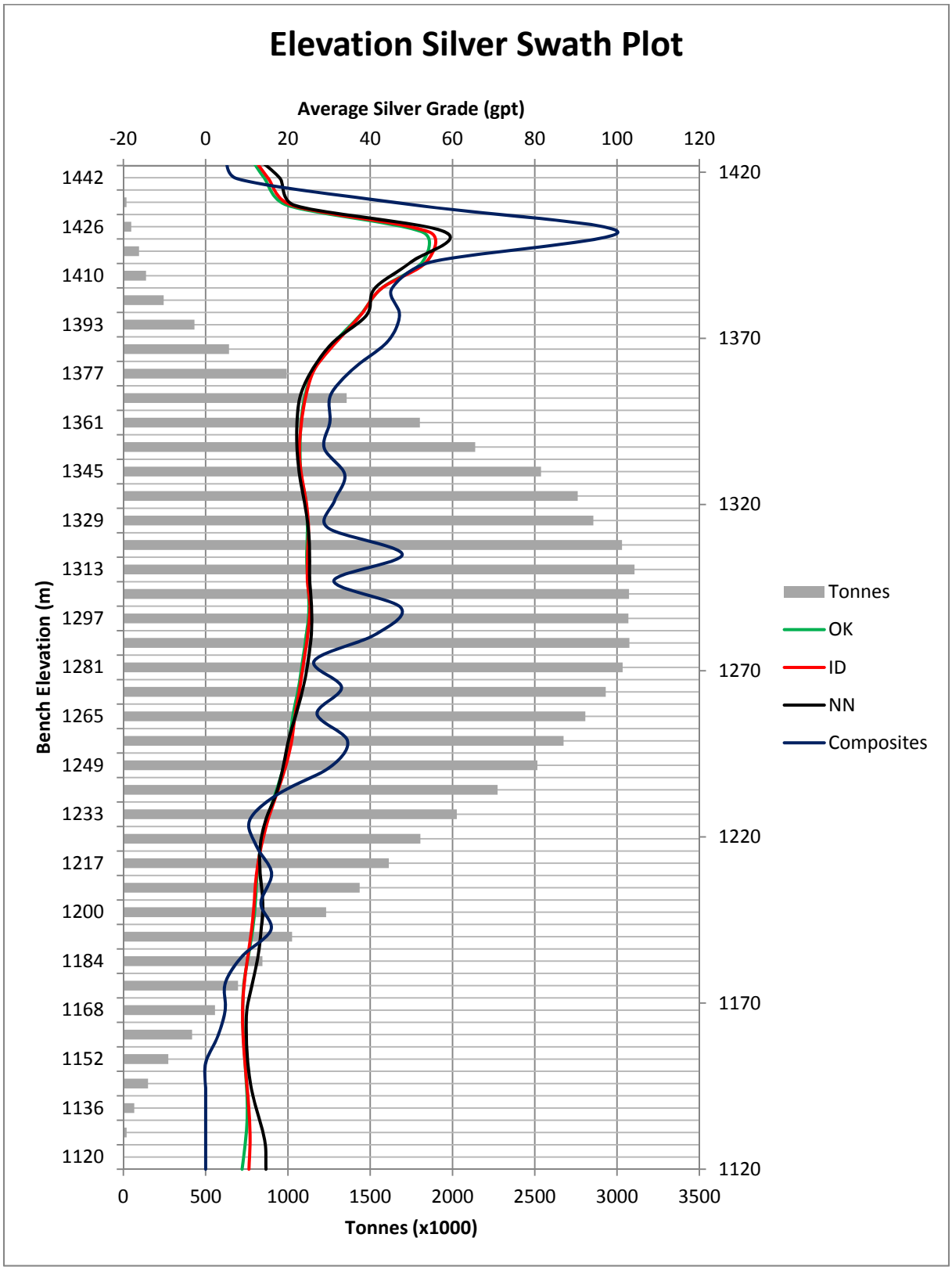


Figure 14-14 Elevation Silver Swath Plot

On a local scale, the NN model method does not provide a reliable estimate of grade, but on a much larger scale, it represents an unbiased estimation of the grade distribution based on the total data set. Therefore, if the $ID^{2.5}$ model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the distribution of grade from the NN.

Overall, there is good correlation between the grade models and the composite data, although deviations occur near the edges of the deposit and in areas where the density of drilling is less and material is classified as Inferred resources.

Sectional Inspection

Bench plans, cross-sections, and long sections comparing modeled grades to the 4-meter composites are shown in Figures 14-15 through 14-20. The figures show good agreement between modeled grades and the composite grades. In addition, the modeled blocks display continuity of grades along strike and down dip.

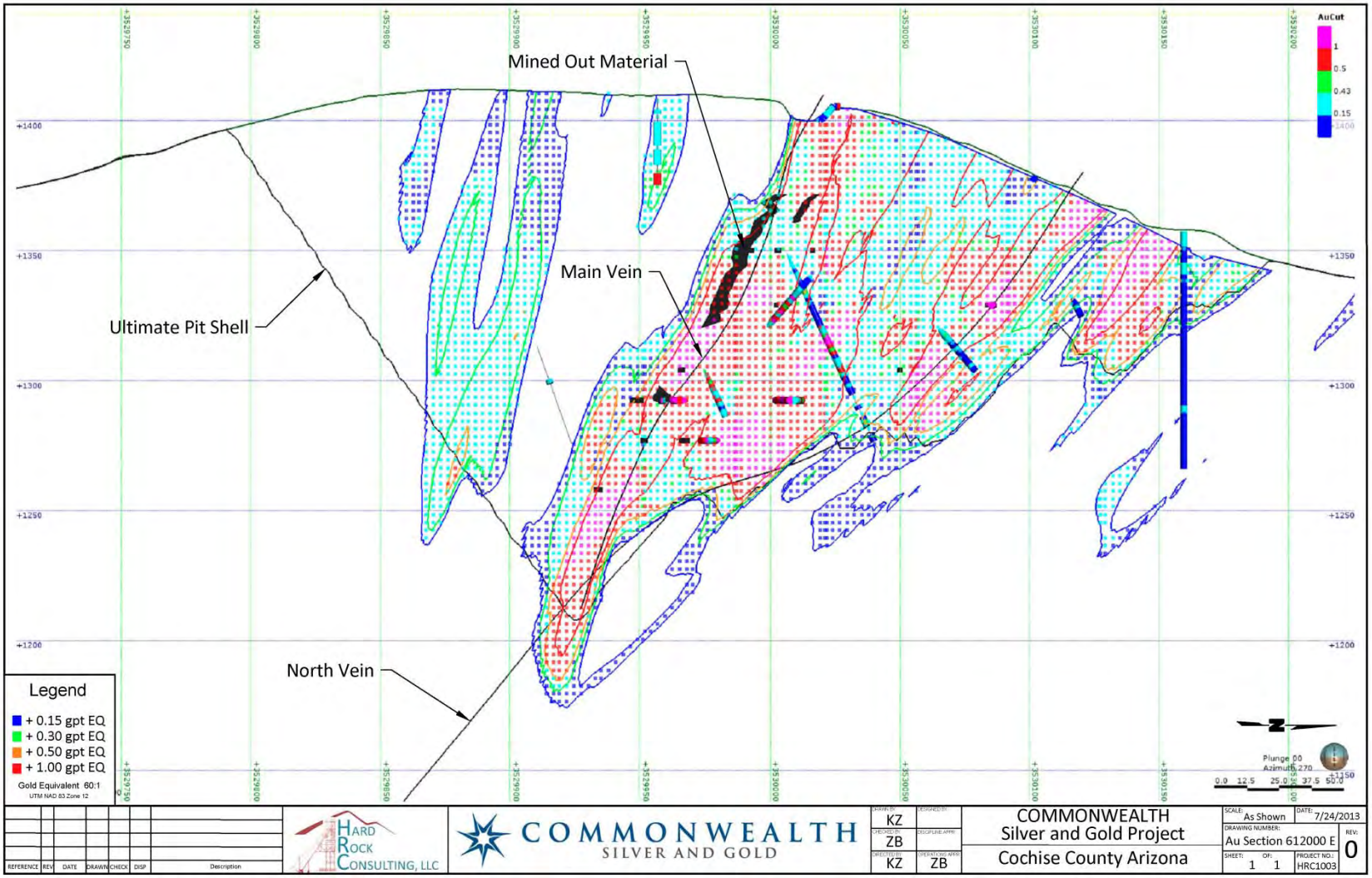


Figure 14-15 North-South Cross Section 612,000E, Showing Block and Composite Gold Grades, Gold Equivalent Solids, Resource Pit, Mine Workings, and Major Structures

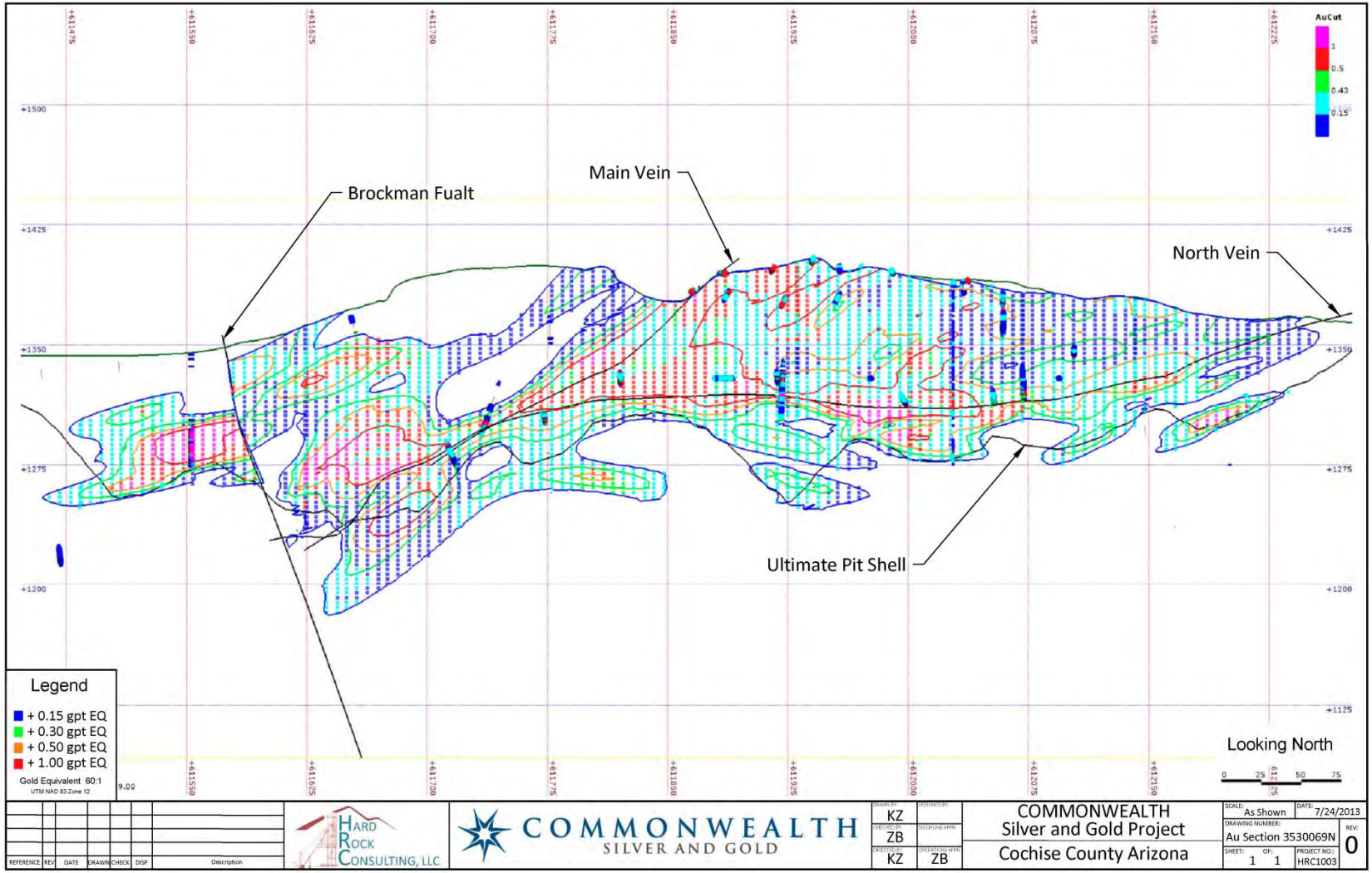


Figure 14-16 West-East Cross Section 3,530,069N, Showing Block and Composite Gold Grades, Gold Equivalent Solids, Resource Pit, Mine Workings, and Major Structures

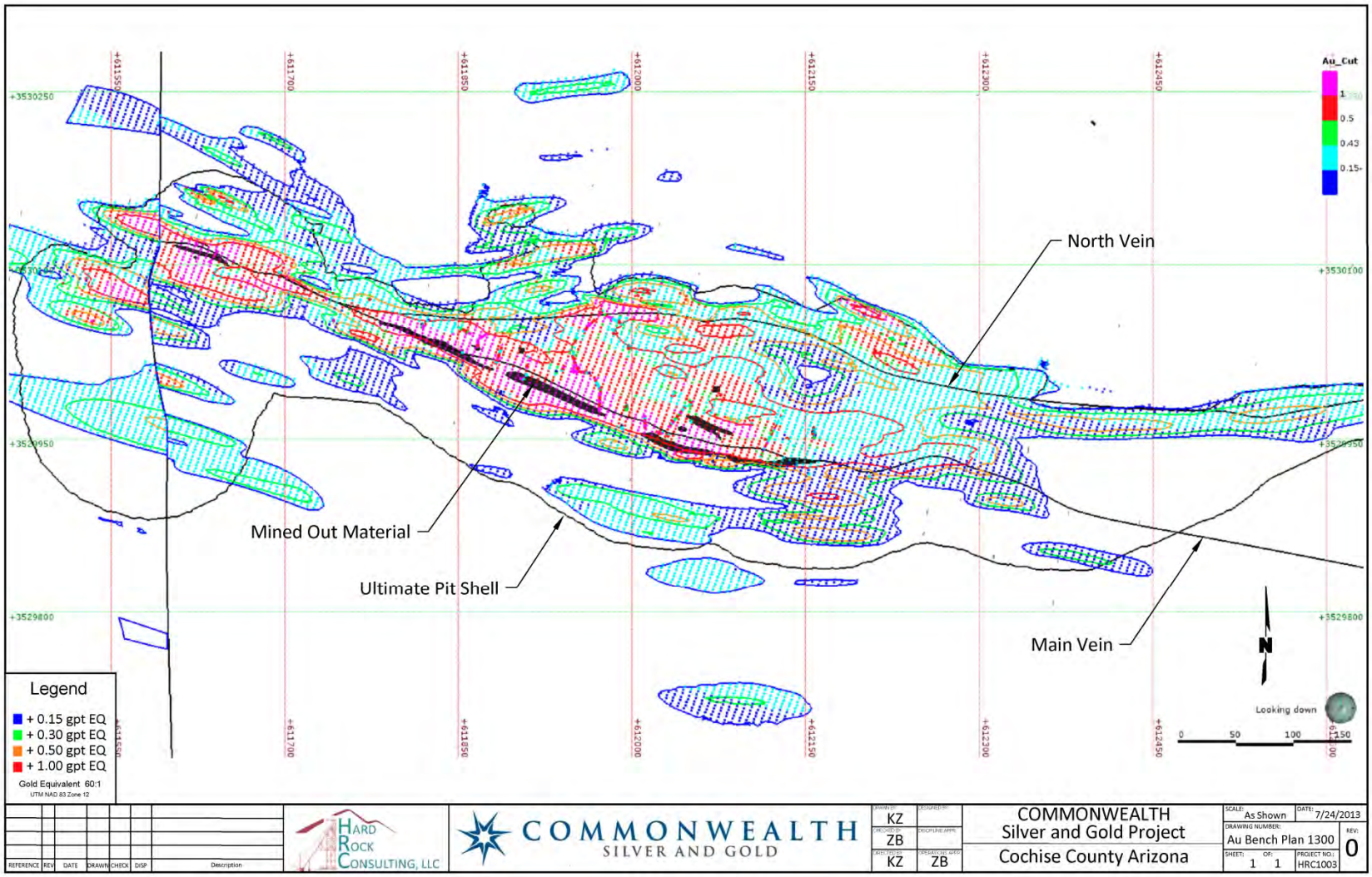


Figure 14-17 Bench Plan Elevation 1,300, Showing Block and Composite Gold Grades, Gold Equivalent Solids, Resource Pit, Mine Workings, and Major Structures

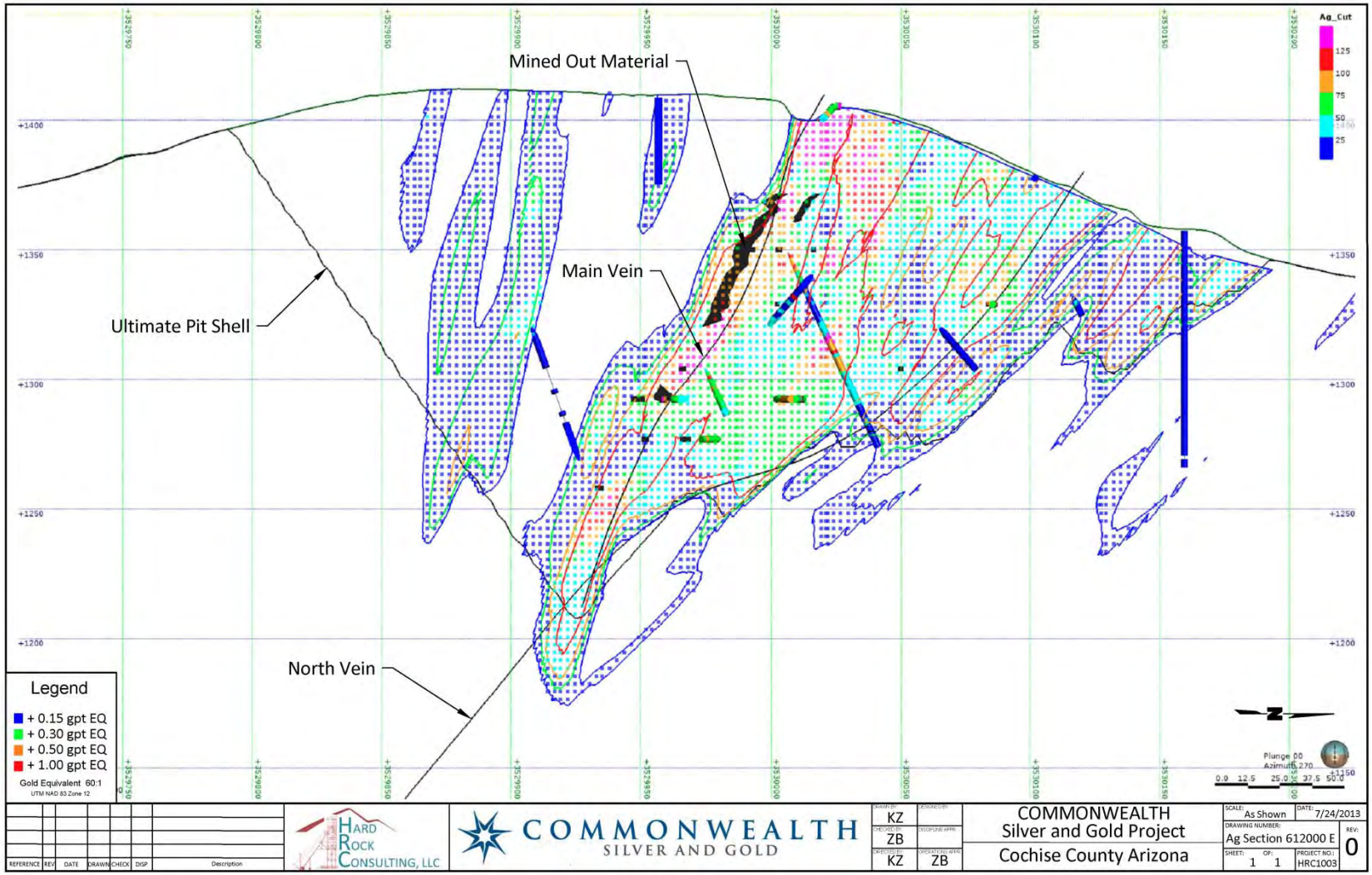


Figure 14-18 North-South Cross Section 612,000E, Showing Block and Composite Silver Grades, Gold Equivalent Solids, Resource Pit, Mine Workings, and Major Structures

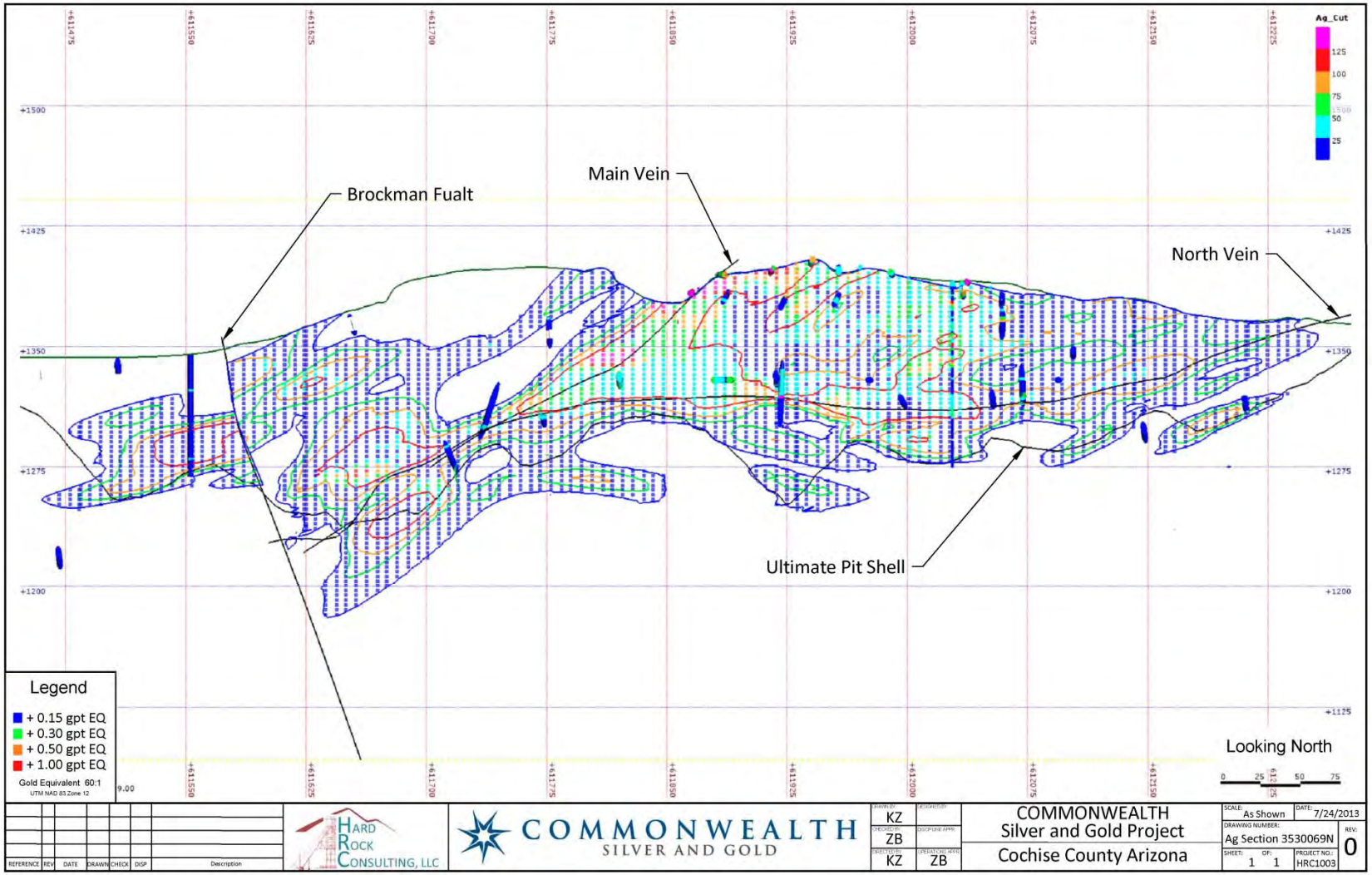


Figure 14-19 West-East Cross Section 3,530,069N, Showing Block and Composite Silver Grades, Gold Equivalent Solids, Resource Pit, Mine Workings, and Major Structures

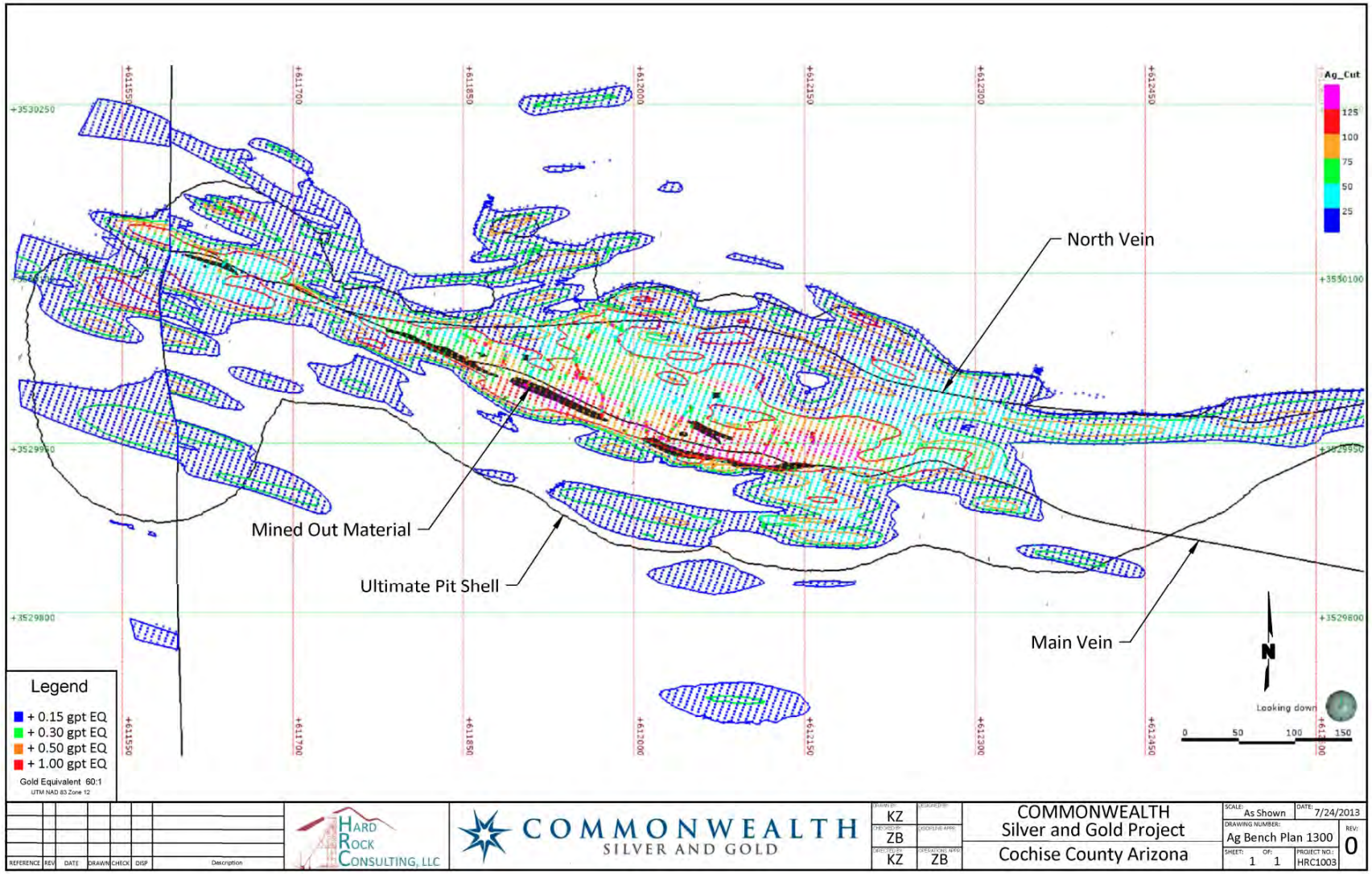


Figure 14-20 Bench Plan Elevation 1,300, Showing Block and Composite Silver Grades, Gold Equivalent Solids, Resource Pit, Mine Workings, and Major Structures

Mineral Resources

The mineral resource estimate for the Commonwealth Silver and Gold Project is summarized in Table 14-15. This mineral resource estimate includes all drill data obtained as of June 10, 2013, and has been independently verified by HRC. Mineral resources are not mineral reserves and may be materially affected by environmental, permitting, legal, socio-economic, marketing, political, or other factors. In Table 14-15, mineral resources are reported above a 0.2 g/t gold equivalent ("AuEq") cut-off, assuming an average gold price of US$1,350 per ounce. This cut-off reflects the potential economic, marketing, and other issues relevant to an open pit mining scenario based on a Merrill-Crowe recovery process following cyanide heap leaching. HRC cautions that economic viability can only be demonstrated through prefeasibility or feasibility studies.

Table 14-15 Mineral Resource Statement for the Commonwealth Silver and Gold Project
Cochise County, Arizona, Hard Rock Consulting, LLC, December 31, 2013

Cutoff	Volume	Tonnage	Gold Equivalent		Gold		Silver	
(gpt)	cu. M	000 tonnes	gpt	t. oz.	gpt	t. oz.	gpt	t. oz.
Inverse Distance 2.5 Model In Pit Measured Resources								
0.4	1,662,900	4,069	1.380	180,800	0.57	74,800	48.6	6,357,700
0.3	1,841,200	4,504	1.280	185,700	0.53	77,200	45.0	6,516,900
0.2	**2,047,000**	**5,007**	**1.18**	**189,800**	**0.49**	**79,000**	**41.3**	**6,648,500**
Inverse Distance 2.5 Model In Pit Indicated Resources								
0.4	8,966,100	21,934	1.06	746,100	0.45	314,500	36.8	25,950,900
0.3	10,893,200	26,643	0.93	799,200	0.40	339,200	32.2	27,582,000
0.2	**12,522,400**	**30,623**	**0.85**	**832,000**	**0.36**	**354,400**	**29.1**	**28,650,600**
In Pit Measured and Indicated Resources								
0.4	10,629,100	26,003	1.11	926,900	0.47	389,300	38.6	32,308,700
0.3	12,734,400	31,147	0.98	984,900	0.42	416,400	34.1	34,098,900
0.2	**14,569,400**	**35,630**	**0.89**	**1,021,700**	**0.38**	**433,500**	**30.8**	**35,299,100**
Inverse Distance 2.5 Model Inferred Resources								
0.4	3,021,700	7,380	0.58	136,700	0.29	67,900	17.2	4,075,100
0.3	5,314,000	12,974	0.48	199,600	0.25	102,800	13.8	5,762,000
0.2	**7,672,600**	**18,733**	**0.41**	**245,400**	**0.21**	**127,600**	**11.6**	**6,998,200**

***Notes:**

(1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

(2) Measured and Indicated Mineral Resources captured within the pit shell meet the test of reasonable prospect for economic extraction and can be declared a Mineral Resource.

(3) Inferred Mineral Resources are that part of the Mineral Resource for which the quantity and grade or quality are estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity.

(4) All resources are stated above a 0.2 g/t gold equivalent ("AuEq") cut-off.

(5) Pit optimization is based on assumed gold and silver prices of US$1,350/oz. and US$22.50/oz., respectively and mining, processing and G&A costs of US$7.25 per tonne. Metallurgical recoveries for gold and silver were assigned by lithologic unit.

(6) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

(7) Gold Equivalent stated using a ratio of 60:1 and ounces calculated using the following conversion rate: 1 troy ounce = 31.1035 grams. Metallurgical recoveries are not accounted for in the gold equivalent calculation.

15. Mineral Reserve Estimates

This section is not required for the Preliminary Economic Analysis.

16. Mining Methods

Open Pit Mine Plan

The Commonwealth Project contains mineralization at or near the surface that is ideal for open pit mining methods. The method of material transport evaluated for this study is open pit mining by a mining contractor using two 12.2-m^3 front end loaders as the main loading units with a third 12.2-m^3 front end loader as a backup loading unit and feeding the mineralized material stockpile when required. The mineralized material will be loaded into 90-tonne haul trucks and transported to the primary jaw crusher, which will be set up at the south end toe of the waste dump. The plan assumes that the contract mining company owns, operates, and maintains all equipment. The general site layout, including pits, waste dumps, the crusher site, infrastructure, and heap leach pad, is shown on Figure 16-1.

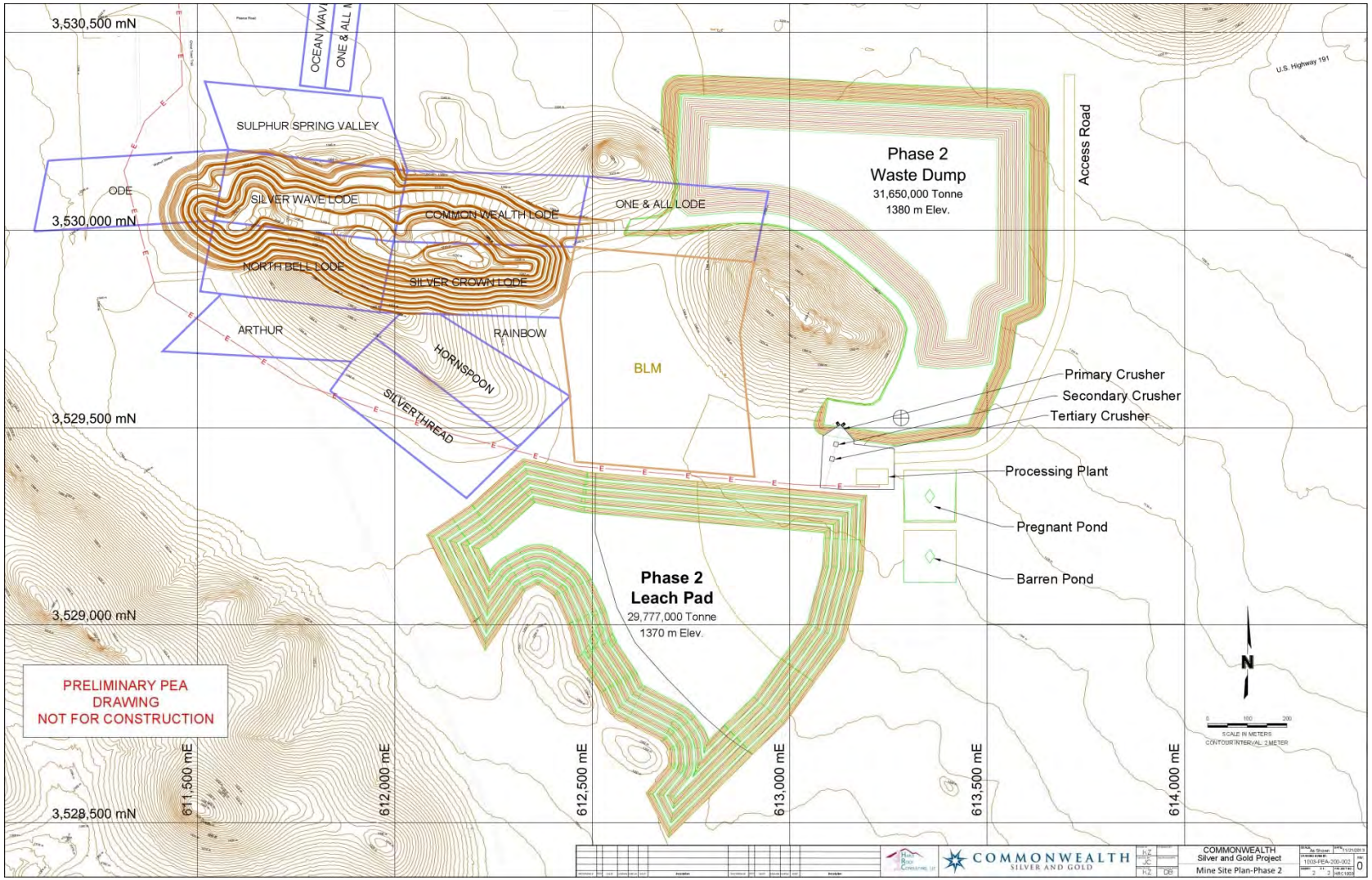


Figure 16-1 General Site Layout

Production of mineralized material is planned at a nominal rate of 10,000 tonnes per day (tpd), equivalent to 3.65 million tonnes per annum with an 8.5 year mine life. Mining is planned on a 7 day per week schedule, with two 12 hour shifts per day. Other mining schedules may prove to be more effective, but are not expected to significantly change project economics. Peak mineralized material and waste production is estimated at 33,000 tpd during year five with an average rate of 20,000 tpd. The average life of mine stripping ratio is 1:1 waste-to-ore, using a 0.30 g/t AuEq cutoff. Lower grade material is stockpiled using 0.24 g/t AuEq cutoff in order to improve project economics, which results in a maximum low grade stockpile of 2.4 million tonnes. The stockpile is used to balance the feed of mineralized material during the stripping of the phase 2 pit, and the reaming is crushed and placed on the heap at the end of the mine life. Other cutoff scenarios using 0.15, 0.18, 0.21, 0.24, 0.27 and 0.30 g/t AuEq were evaluated during the study but the chosen scenario resulted in the best IRR and NPV. The mine schedule is based on measured and indicated material only. The mine schedule does not include inferred material, and the mine plan and subsequent metallurgical flow sheet are preliminary in nature and as a result no material is categorized as mineral reserves. Table 16-1 lists the resources used in the mine production plan at a 0.24 g/t AuEq cutoff.

Table 16-1 Resources Inside Pit Design

Pit Phase	Resource Category	Tonnes ('000)	Au Grade (g/t)	Ag Grade (g/t)	AuEq Grade (g/t)	Contained Metal (Ounces)		
						Au	Ag	AuEq
Phase 1	*Measured*	3,253	0.46	43.07	1.18	48,034	4,504,874	123,115
Phase 1	*Indicated*	12,227	0.36	34.57	0.94	142,539	13,590,165	369,042
Phase 1	***Measured + Indicated***	**15,480**	**0.38**	**36.36**	**0.99**	**190,573**	**18,095,039**	**492,157**
Phase 2	*Measured*	1,633	0.56	38.17	1.19	29,212	2,003,915	62,610
Phase 2	*Indicated*	14,040	0.38	26.37	0.82	172,644	11,902,972	371,027
Phase 2	***Measured + Indicated***	**15,673**	**0.40**	**27.60**	**0.86**	**201,856**	**13,906,886**	**433,637**
Total All Phases	*Measured*	4,886	0.49	41.43	1.18	77,245	6,508,788	185,725
Total All Phases	*Indicated*	26,267	0.37	30.19	0.88	315,184	25,493,137	740,069
Total All Phases	***Measured + Indicated***	**31,154**	**0.39**	**31.95**	**0.92**	**392,429**	**32,001,925**	**925,794**

1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

2) Prepared by Jeff Choquette, P.E., Mining Engineer, an independent Qualified Person within the meaning of NI43-101, using a reporting cut-off grade of 0.24 g/t AuEq.

3) Gold Equivalent stated using a ratio of 60:1 and ounces calculated using the following conversion rate: 1 troy ounce = 31.1035 grams. Metallurgical recoveries are not accounted for in the gold equivalent calculation.

Pit Optimization

The mineral resources for the Project were determined using Datamine's MaxiPit™ Lerchs Grossman pit optimizer to generate optimized pit shells. Pit shells were generated based on varying metal prices with a base gold price of $1350/oz and base silver price of $22.50/oz. A total of 124 pit shells were generated to determine optimal break points in the pit phases and the final pit phase.

Parameters for the shells were $2.25/tonne of material moved for mining, $4.15/tonne for processing the mineralized material requiring a ½" crush size, and $4.45/tonne for processing the material requiring an 8 mesh crush size. General and administrative costs were estimated at 0.75/tonne of material processed. Gold and Silver recoveries were assigned by rock type as described in Section 17 of this PEA. Interramp slope angles were varied by sector and rock type in the pit based on recommendations from Call and Nicholas Inc. in a 1996 report as presented in Table 16-2. The values presented in Table 16-2 are the "Best Estimate" which was the middle range of estimates provided in the report and used for final design. The "Worst Case" values which were 5 to 8 degrees shallower were used in the optimization in order to allow room for insertion of haul roads in the design.

Table 16-2 Recommended Pit Slope Angles

Rock Type	Block Flag	Design Sector	Bench Ht. (m)	Interramp Pit Slope Angle (°)	Bench Width (m)	Batter angle
Qal-(Ovb)	1		18	32	8.50	41.55
Trb	2	2,3	18	54	8.50	75.73
Tal	3	2,3	18	54	8.50	75.73
Kb	4	1,6,7	18	50	8.50	69.85
Tau	5	2,3	18	54	8.50	75.73
Tss	14	5	18	55	8.50	77.16
Veins	50-57	2,3	18	54	8.50	75.73

Pit Design

The Commonwealth open pit is planned in two pit phases utilizing the optimized pit shells as a basis for the design. The final design was based on a pit shell that utilized a 0.8 metal price factor which equates to a $1,080/oz Au price and an $18.00/oz Ag price. The reduced metal price pit was chosen to minimize the risk of additional stripping as the pits beyond this showed marginal increase in value.

Pit slopes were designed based on the recommendations by Call and Nicholas as presented in Table 16-2. Haul roads are designed at a width of 29 meters, which provides a safe truck width (6.5 meters) to running surface width ratio of 1:3.5, including a 6.1 meter width for a berm on the edge of the road. Maximum grade of the haul roads is 10%, except for the lowermost few benches where the grade is increased to 14% and the ramp width is narrowed to 15 meters to minimize excessive waste stripping. The pit design criteria are presented in Table 16-3.

Table 16-3 Pit Design Criteria

Mine Design Criteria	
Pit Design Criteria	**Parameter**
Inter Ramp Angles	See Table 16-2
Face Angles	See Table 16-2
Catch Bench Berm	8.5 m
Catch Bench Vertical Spacing	18 m
Minimum Turning Radius	25 m
Road Widths	29 m
Road Grade	10%
Road Widths Pit Bottom	15 m
Road Grade Pit Bottom	14%

The pits were designed in two phases in order to balance the required stripping throughout the mine life. The pit designs phases are shown in Figures 16-2 and 16-3.

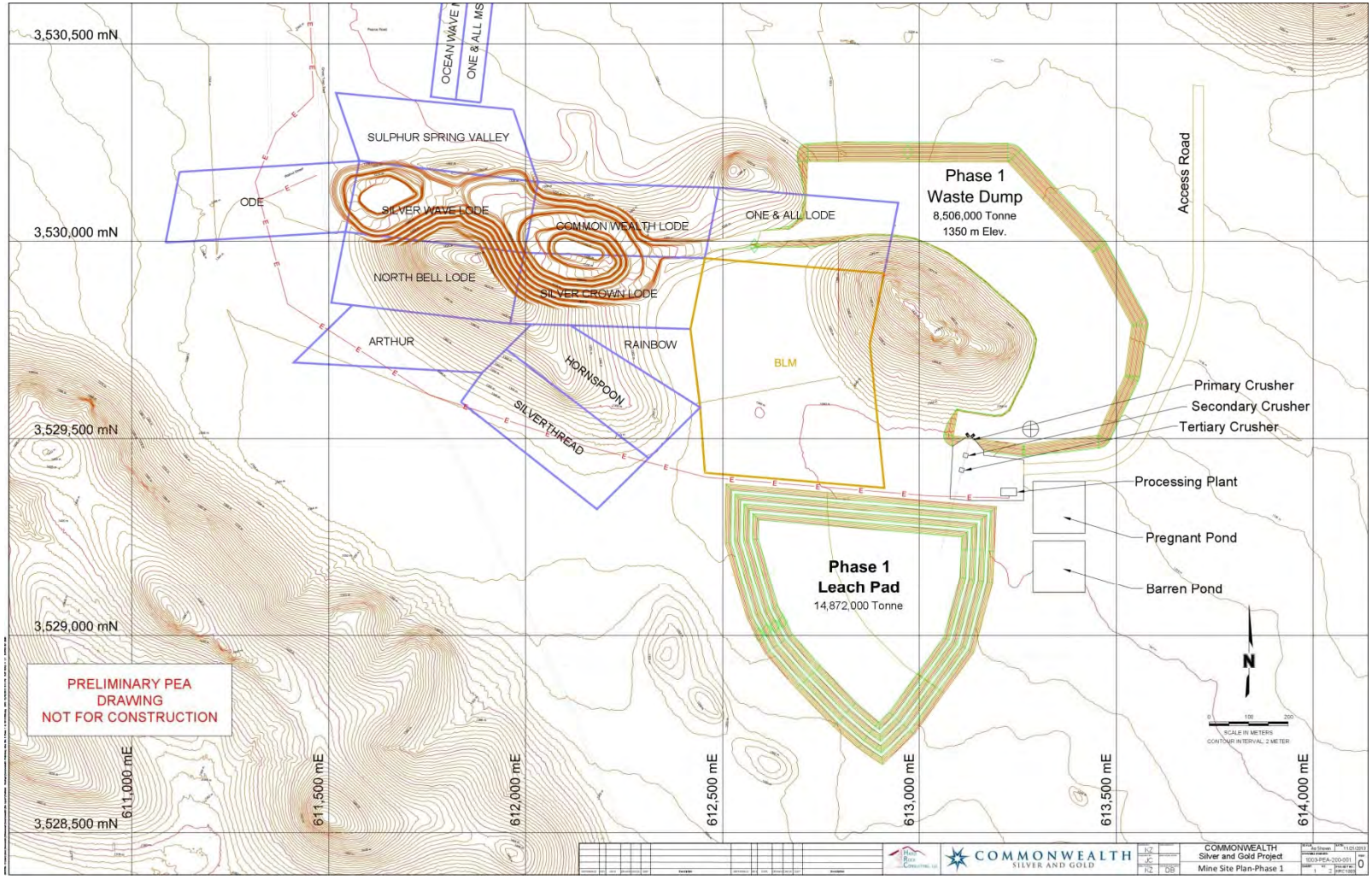


Figure 16-2 Phase 1 Pit Design

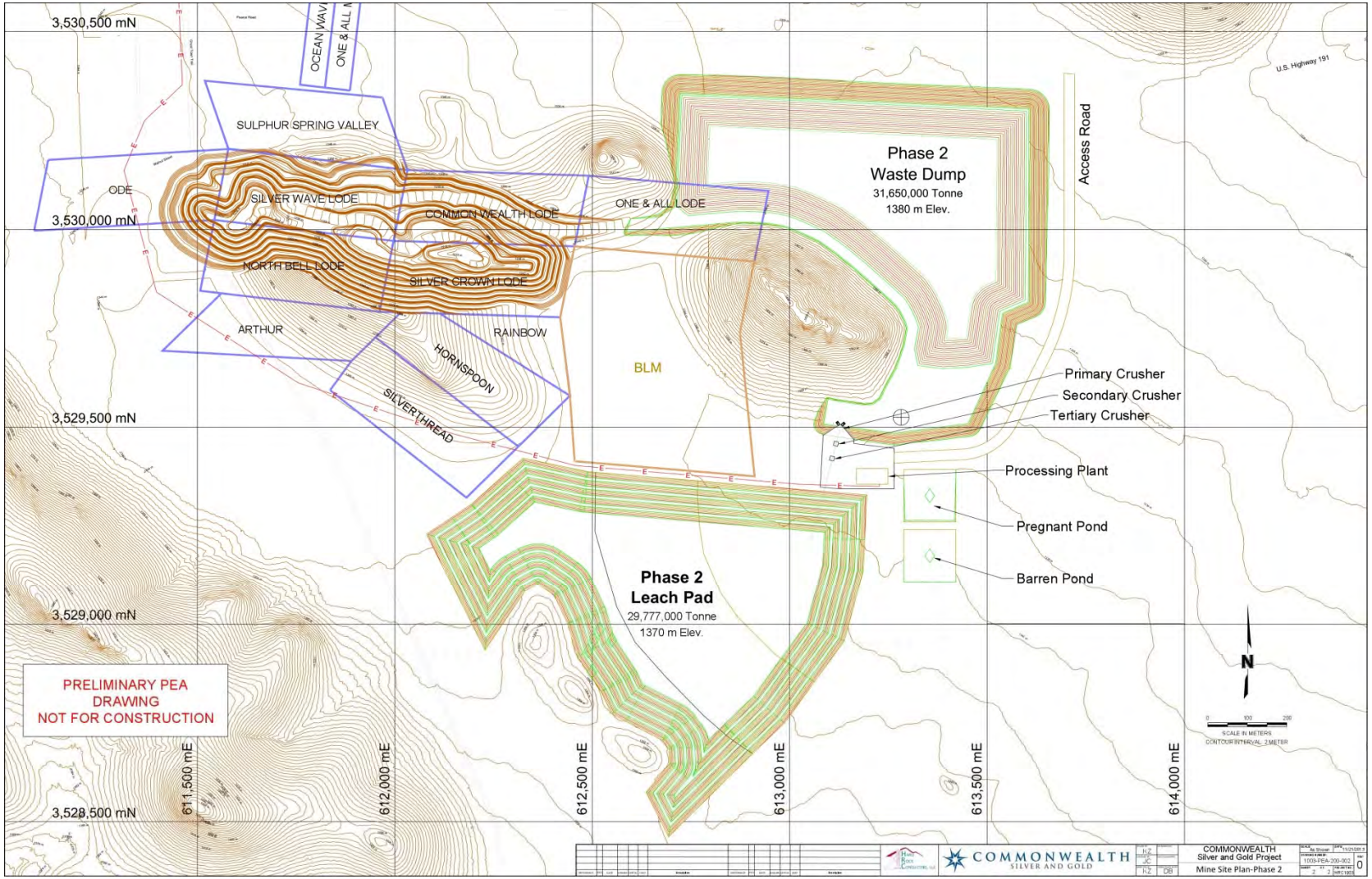


Figure 16-3 Phase 2 Pit Design

Waste Dump Design

The waste dump was designed near the pit ramp exit just north east of the pit design. The dump was designed with a maximum slope angle of 2.5:1. The waste dump is designed with a capacity of 32 million tonnes to accommodate the mine plan. The location of the waste dump in relation to the other facilities is shown in Figure 16-3.

Preproduction Development

The preproduction requirements at the Project are minimal given the presence of mineable mineralization near the surface. An estimated allowance of $1.0M has been included in the initial capital to cover the costs of the initial road construction and any clearing, or grubbing that may take place

Production Schedule

The yearly mine production schedule is presented in Table 16-4. The production schedule is driven by the nominal production rate of 10,000 tpd. The production schedule has been calculated on a monthly basis for the first three years and then yearly for the remaining life of the mine. The schedule shows material being delivered to the pad during the first month of mining at a rate of 1,025 tpd with the mine being able to provide the required crushing and stacking rate of 10,000 tpd starting in month four. Peak mineralized material and waste production is estimated at 33,000 tpd during the stripping of the phase two pit in year four with a life of mine average rate of 20,000 tpd. The average life of mine stripping ratio is 1:1 waste-to-mineralized material.

The mine schedule is based on measured and indicated material only, and although the mine schedule does not include inferred material, the mine plan and subsequent metallurgical testing is preliminary in nature and as a result no material is categorized as mineral reserves. Thus, there is no certainty that the production profile concluded in the PEA will be realized. Actual results may vary.

The total mineralized material mined is 28.3 M tonnes with an additional 2.8 M tonnes of low grade material which is used to balance the mineralized material feed during the stripping of the phase 2 pit and the reaming is crushed and placed on the heap at the end of the mine life.

Table 16-4 Yearly Production Schedule

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Life-of-Mine
MINE PRODUCTION										
Tonnes Ore Mined	3,085,175	3,650,000	3,650,000	4,679,004	2,242,597	3,611,970	3,650,000	3,650,000	87,982	28,306,729
Ag, g/tn	49.0	31.2	31.7	40.5	27.2	28.9	32.4	31.8	16.1	34.32
Au, g/tn	0.34	0.38	0.43	0.39	0.36	0.47	0.51	0.43	0.27	0.42
Tonnes LG Ore Mined	309,112	429,290	333,494	414,928	275,208	495,376	315,196	250,635	23,606	2,846,846
Ag, g/tn	9.7	8.8	7.5	8.6	8.9	7.1	8.0	9.0	8.3	8.34
Au, g/tn	0.11	0.12	0.15	0.13	0.12	0.15	0.14	0.12	0.13	0.13
Waste	3,300,966	2,836,281	1,337,211	6,953,099	7,583,538	4,940,971	2,277,946	976,800	66,388	30,273,199
Total Tonnes Mined	**6,695,253**	**6,915,571**	**5,320,705**	**12,047,031**	**10,101,343**	**9,048,317**	**6,243,142**	**4,877,436**	**177,976**	**61,426,774**
SR	**1.0**	**0.7**	**0.3**	**1.4**	**3.0**	**1.2**	**0.6**	**0.3**	**0.6**	**1.0**
Other Tonnes	240,000	240,000	240,000	240,000	1,382,194	240,000	240,000	240,000	2,591,806	5,654,000
Total Tonnes Moved	**6,935,253**	**7,155,571**	**5,560,705**	**12,287,031**	**11,483,538**	**9,288,317**	**6,483,142**	**5,117,436**	**2,769,782**	**67,080,775**
Stockpile Balance	264,107	693,397	1,026,891	2,470,823	1,328,628	1,785,974	2,101,171	2,351,806	0	0
Ag, g/tn	9.6	9.1	8.6	21.9	8.6	8.2	8.2	8.2	0.0	0.0
Au, g/tn	0.11	0.12	0.13	0.24	0.13	0.13	0.13	0.13	0.00	0.00
Stockpile Change	264,107	429,290	333,494	1,443,932	(1,142,194)	457,346	315,196	250,635	(2,351,806)	-
PROCESS PRODUCTION										
Tonnes Ore Processed	**3,130,181**	**3,650,000**	**3,650,000**	**3,650,000**	**3,660,000**	**3,650,000**	**3,650,000**	**3,650,000**	**2,463,394**	**31,153,575**
Ag, g/tn	48.4	31.2	31.7	40.5	29.0	28.7	32.4	31.8	8.5	31.95
Au, g/tn	0.34	0.38	0.43	0.39	0.34	0.47	0.51	0.43	0.14	0.39

Production Schedule Parameters

The mine production schedule is based on a 7 day per week schedule, with two 12 hour shifts per day. There are four crews planned to cover the rotating schedule. Each 12 hour shift contains a half hour down for blasting and miscellaneous delays, a half hour for shift start up and shutdown and an hour for lunch and breaks for a total of 10 effective working hours. Table 16-5 below shows typical yearly schedule parameters and hours scheduled.

Table 16-5 Mine Schedule Parameters

Mine Schedule	
Crews	4
Shifts/day	2
Hours/shift	12 hr.
Lunch, Breaks, etc.	1 hr.
Blasting, Misc.	0.5 hr.
Startup & Shutdown	0.5 hr.
Days/Year	365 days
Scheduled Hours/Year	**8,760**

The amount of equipment required to meet the scheduled tonnages is calculated based on the mine schedule, equipment availabilities, usages and haul and loading times for the equipment. Equipment

mechanical physical availabilities start at 92% for the trucks, drills and loading units. For each year of production, the mechanical physical availabilities decrease by one percent. The use of availability for all of the equipment is calculated at 83% based on the breaks and down time in the schedule parameters. An additional 85% efficiency factor is applied to all of the equipment for calculating the total units of equipment required. Equipment availability parameters are presented in Table 16-6.

Table 16-6 Equipment Availabilities

Equipment Availabilities	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Physical Availability	92%	91%	90%	89%	88%	87%	86%	85%	84%
Use of Availability	83%	83%	83%	83%	83%	83%	83%	83%	83%
Efficiency	85%	85%	85%	85%	85%	85%	85%	85%	85%

Drill and Blast Parameters

The design parameter used to define drill and blast requirements are based on a 171 mm blast hole on a 4.9-m by 4.3-m pattern in the mineralized zones and a 5.2-m by 4.6-m pattern in the waste zones. Benches will be blasted and mined on 6-m levels with 1.0 m of sub-drill. Buffer rows and pre-shear are planned to allow for controlled blasting and minimize damage to the highwalls. The number of blast holes and blast hole drills required each month or year is calculated based on the parameters presented in Table 16-7 and used in calculating the operating costs. The first three years of the mine life only requires one rotary production drill with a second required during the phase two pit stripping in year four, a smaller track mounted drill is planned for pre-shear drilling and some of the buffer drilling.

Table 16-7 Drill and Blast Parameters

Production & Wall Control Blast Pattern Data		Production Pattern				
DRILLING & BLASTING PARAMETERS		Ore	Waste	Wall Control Pattern		
	Units	Rock	Rock	Buffer	Buffer	Preshear
Tonnage Factor	dmt/cubic meter	2.42	2.40	2.40	2.40	2.40
Blast Pattern Details						
Bench Height	meters	6.00	6.00	6.00	6.00	6.00
Sub Drill	meters	1.00	1.00	1.00	0.60	0.00
Diameter of Hole	mm	171.00	171.00	171.00	171.00	101.00
Staggered Pattern Spacing	meters	4.30	4.60	4.30	3.70	1.40
Staggered Pattern Burden	meters	4.90	5.20	3.70	3.00	1.40
Drill Equivalent Square Pattern	meters	4.60	4.90	4.00	3.35	1.40
Hole Depth	meters	7.00	7.00	7.00	6.60	6.00
Height of Stemming or Unloaded Length	meters	3.60	3.80	4.00	4.50	
Material Quantity						
Volume Blasted/Hole	cubic meters	127	144	96	67	12
Tonnes Blasted/Hole	tonnes	307	346	230	162	28
Powder Factor						
Percent Emulsion		30%	30%	30%	30%	30%
Percent Anfro		70%	70%	70%	70%	70%
Density of Powder	g/cc	0.96	0.96	0.96	0.96	0.96
Loading Density	kg/m	21.93	21.93	21.93	21.93	7.65
Powder/hole	kg	74.57	70.18	65.80	46.06	2.54
Powder Factor	kg/tn	0.243	0.203	0.286	0.285	0.090
Powder Factor	kg/bcm	**0.587**	**0.487**	**0.685**	**0.684**	**0.216**
Drill Productivities						
Penetration Rate	M/hr	42.00	42.00	42.00	40.00	30.00
Penetration Rate	M/min	**0.70**	**0.70**	**0.70**	**0.67**	**0.50**
Cycle Time Estimate						
Drilling Time	minutes	10.00	10.00	10.00	9.90	12.00
Steel Handling Time	minutes	0.00	0.00	0.00	0.00	0.50
Set up Time	minutes	3.30	3.30	3.30	3.30	2.00
Add Steel	minutes	0.00	0.00	0.00	0.00	2.00
Pull Rods	minutes	0.50	0.50	0.50	0.50	2.00
Total	**minutes**	**13.80**	**13.80**	**13.80**	**13.70**	**18.50**
Drilling Factors for Wall Control						
Buffer Holes - 2 Rows						
Wall Control Drill Holes Required	***Perimeter Blast***					
Buffer Holes - 2 Rows	holes/meter		**0.55**			
Material to Remove from Production Blast	tonnes/meter		**126.00**			

Load and Haul Parameters

The design parameters used to define the loading and hauling requirements are shown in Table 16-8. The method of material transport evaluated for this study is open pit mining using two 12.2-m^3 front end loaders as the main loading units with a third 12.2-m^3 front end loader as a backup loading unit and feeding the mineralized material stockpile when required. The main hauling units are 90 tonne haul trucks, and the loaders will require 5 passes to load the trucks. A shovel that requires four pass loading was evaluated but the lower capital and operating cost of the 5-pass loaders showed better economics. Haulage profiles for the mineralized material and waste material from each pit phase were generated and used to calculate the truck cycle times, which were then used in the equipment requirement

calculations. The selected contract miner may use different equipment, however for this stage of the project, the equipment and mining parameters are representative of anticipated operations.

Table 16-8 Load and Haul Parameters

Loading & Truck Match Calculation		Loader 90mt
Bucket Capacity (heaped)	cm	12.20
Bank Material Weight Dry	kg/bcm dry	2410
Bank Material Weight Wet	kg/bcm wet	2470
Bulk Factor (Swell Factor)		1.40
Loose Material Weight Dry	kg/lcm dry	1,721.4
% Moisture		2.5%
Bucket Fill Factor		0.90
Effective Bucket Capacity	cm	10.98
Wet Material Weight (LCM)	wmt/lcm	1.76
Dry Material Weight (LCM)	dmt/lcm	1.72
Tonnes/Pass	wmt	19.37
Truck Size Capacity (volume)	cubic m heaped	60.0
Truck Size Capacity (tonnes)	wmt	90.3
Theoretical Passes (volume)	passes	5.46
Theoretical Passes (tonnes)	passes	4.66
Actual Passes	passes	**5.0**
Truck Load - Volume (volume)	cm	54.9
Truck Load - Volume (tonnes)	wmt	96.9
Truck Load for Productivity	dmt	**94.5**
Truck Capacity Utilized (tonnes)	by weight	**107.3%**
Truck Capacity Utilized (volume)	by volume	91.5%
Average Cycle Time	sec	50
Truck Spot Time	sec	45
Load Time per Truck	sec	295
Load Time per Truck	minutes	4.92
Maximum Productivity	trucks/hr	12.2
Insitu Volume/Hour	bcm/hr	478.5
Tonnes/Hour	dmt/hr	1,153.3

Mining Equipment

Mining equipment will be supplied by the mining contractor. The initial mine production equipment is expected to include three 12.2-m^3 front end loaders, two for pit production and a third will function as a backup loading unit and to feed the mineralized material stockpile when required. Initially, three 90-tonne haul trucks are required to meet the production schedule. Two trucks will need to be added during year four to meet production requirements, for a total of five trucks. One production drill will be required initially, with a second production drill purchased during year five. A pre-shear drill will also be required for wall control purposes and backup drilling during the initial years. Table 16-9 lists the initial and total equipment requirements.

Table 16-9 Mine Production Equipment

Description	# Initial Units	# Total Units
12 m3 Loader	2	3
Production Drill	1	2
PreShear Drill	1	1
Haul Truck - 90t	3	5

Support equipment will consist of one Cat D8 and two D9 dozers. A 16' road grader will service the haul roads along with a 10,000 gallon water truck. A 0.9 m^3 excavator will be purchased for scaling highwalls and other miscellaneous projects around the mine site. Five mobile light plants will be required for lighting the working areas during nighttime production. A maintenance service truck with a mobile crane will be needed for field maintenance and a self-contained fuel lube truck will be needed for infield fueling. Anticipated mine support equipment is listed in Table 16-10.

Table 16-10 Mine Support Equipment

Description	# Initial Units	# Total Units
16' Grader	1	1
Water Truck	1	1
448hp Dozer	2	2
347hp Dozer	1	1
Lube/Fuel/Service	3	3
Light Plants	5	5
Small Excavator 148 hp	1	1
50 ton Crane	1	1
IT Loader	1	1

Staffing

The manpower required for the mine department is calculated based on the equipment required to meet the production schedule. The average yearly manpower requirements for the contract miner are shown in Table 16-11. Required mine department personnel including both Commonwealth and contract mining staff is expected to average 89 people.

Table 16-11 Mine Department Manpower

Manpower Summary	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mining G&A									
Mine Superintendant	1	1	1	1	1	1	1	1	1
Mine Foreman	4	4	4	4	4	4	4	4	4
Blasting Foreman	1	1	1	1	1	1	1	1	1
Maintenance Superintendent	1	1	1	1	1	1	1	1	1
Maintenance Foreman	4	4	4	4	4	4	4	4	4
Mine Salaried	11	11	11	11	11	11	11	11	11
Drilling and Blasting									
Driller	4	4	4	8	8	8	4	4	4
Blaster	2	2	2	2	2	2	2	2	2
Blaster Helper	2	2	2	2	2	2	2	2	2
Drilling and Blasting	8	8	8	12	12	12	8	8	8
Loading									
Loader Operator	7	7	4	8	8	8	8	4	8
Loading	7	7	4	8	8	8	8	4	8
Hauling									
Truck Driver	12	12	10	20	20	16	16	12	8
Hauling	12	12	10	20	20	16	16	12	8
Roads and Dumps									
Dozer Operator	8	8	8	8	8	8	8	8	8
Grader Operator	4	4	4	4	4	4	4	4	4
Utility Operator	4	4	4	4	4	4	4	4	4
Support	16	16	16	16	16	16	16	16	16
Mine Maintenance									
Lead Mechanic	4	4	4	4	4	4	4	4	4
Heavy Equipment Mechanic	4	4	4	4	4	4	4	4	4
Light Vehicle Mechanic	1	2	2	2	2	2	2	2	2
Welder/Mechanic	4	6	8	8	8	8	8	8	8
Apprentice/Fueler	4	4	4	4	4	4	4	4	4
Planner/Clerk	1	2	2	2	2	2	2	2	2
Electrician	1	1	1	1	1	1	1	1	1
Total Mine Maintenance	19	23	25	25	25	25	25	25	25
Engineering									
Sr Mining Engineer	1	1	1	1	1	1	1	1	1
Jr Mining Engineer	1	1	1	1	1	1	1	1	1
Chief Surveyor	1	1	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1
Engineering	4	4	4	4	4	4	4	4	4
Geology & Grade Control									
Sr Geologist	1	1	1	1	1	1	1	1	1
Ore Control Geologist	1	1	1	1	1	1	1	1	1
Sampler	2	2	2	2	2	2	2	2	2
Geology	4	4	4	4	4	4	4	4	4
Total Mine OP Eng Geo	8	8	8	8	8	8	8	8	8
Total Mine Department	82	85	82	100	100	96	92	84	84

17. Recovery Methods

Based on the large amount of metallurgical test work accomplished to date, CSGM and HRC selected a crush, heap leach, Merrill-Crowe processing scheme. Due to the recovery sensitivity of some of the mineralized material to crush size, we have elected to go to 3-stage crushing, resulting in two different size products. The Lower Andesite, Upper Andesite, and Bisbee mineralized materials will be crushed to 80% passing ½", whereas the Rhyolite and Vein materials will be crushed to 80% passing 1/8". The crushing circuit will consist of a primary jaw crusher, a secondary cone crusher, and two tertiary cone crushers. The mineralized materials will be campaigned through the crushing circuit such that the ½" crush will be accomplished in three stages, and when the finer crush materials are processed, an alternate tertiary crusher will be used which is set to a finer crush. It is anticipated that the volume of material requiring the finer crush will be less than 200 tph after screening. Blending will occur downstream of the tertiary crusher, in order to improve the kinetics of the heap. Agglomeration will take place on the conveyors, with lime and cement being added after tertiary crushing.

Heap leaching will take place using wobblers and a sodium cyanide solution added on the heap. Collection of the pregnant leachate will be to a pregnant solution pond, from which the pregnant solution will be pumped to the Merrill-Crowe plant.

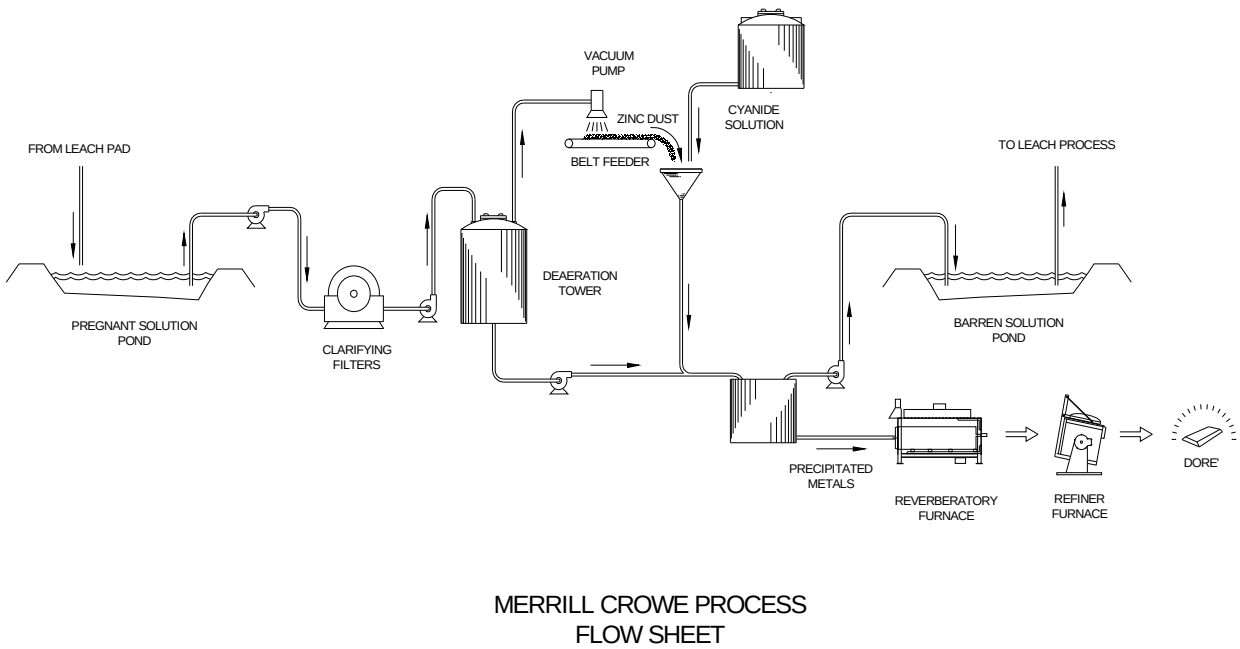


Figure 17-1 Typical Merrill-Crowe Process Flowsheet

Final precipitate from the Merrill-Crowe plant will be shipped to a refiner such as Johnson Mathey or Metalor for final metal production and sale.

Leach Pad

The heap leach pad will be loaded with 5-meter lifts, and will allow for 60-days of leaching at a minimum. The beginning leach pad will consist of approximately 140,000 m2 which will provide for approximately one and a half years of production. The leach pad will be expanded in the second year of production, and again in the 6th year of production. The crushed material will be blended through placement on the pad such that the smaller crushed mineralized material is spread and blended with the larger crushed mineralized material. This blending can be accomplished through the campaigning approach, and the use of the telestacker to spread the finer crushed material on the leach pad.

Leaching

Sodium Cyanide will be added to the heap leach material at the rate of .004 gal / sf, or 0.0014 liters per square meter. The planned production rate, and the planned application rate result in a planned flow into the Merrill-Crowe plant of 3,000 gal per minute, or 11,356 liters / minute. The size of the Merrill-Crowe and the pregnant and barren pond requirements have been factored into the capital cost accordingly.

Reagents

From the extensive test work, it was apparent that approximately 1 lb of Sodium Cyanide was required per tonne of mineralized material, and similarly from test work, 4 lb of cement and 1 lb of lime would be required for agglomeration of each tonne of material.

Recoveries

Due to the different sensitivities of the mineralized material types to the size of crush and resulting recoveries, the plan for operation is for two different sizes of crushed material. The Rhyolite and Vein material will be crushed to 80% passing 1/8”, and the Lower Andesite, Upper Andesite, and Bisbee material will be crushed to 80% passing ½”. The planned gold and silver recoveries at these crush sizes which resulted from the extensive test work are presented in Table 17-1.

Table 17-1 Gold and Silver Recoveries

Rock Type	Crush Size	Recoveries (%)	
		Au	Ag
Rhyolite	Minus 8 Mesh	78	35
Vein	Minus 8 Mesh	79	49
Lower Andesite	1/2"	81	33
Upper Andesite	1/2"	78	35
Bisbee	1/2"	80	23

18. Project Infrastructure

The Commonwealth Project is conveniently located with robust infrastructure. The Project area is located in central Cochise County, approximately 40km (25 miles) south of Willcox and 125 km (75 miles) southeast of Tucson, and is readily accessible by paved State highways from both cities. Interstate 10, approximately 20 miles north of the property, connects with Arizona State Highway 191, which runs adjacent to the property. A rail siding is available in Cochise, Arizona 19 km (12 miles) from the property.

The existing access road may need to be upgraded for approximately 3 miles to provide access from Highway 191 to the plant and offices.

Figure 18-1 Access road and 14.5 KvA Power Line

Buildings

Given the size of the operation, there will be a limited number of buildings required. The average temperatures are moderate year round, with some high temperatures in the summer. Buildings anticipated for the project include:

Truck shop	10,000 sf
Administration building	3,000 sf
Warehouse	2,000 sf
Laboratory	2,000 sf (self-contained mobile lab)

These facilities are minimal, typical of a low-budget operation. Facilities may be added or expanded during operation as operating profits allow.

Power

A 14.47kV powerline currently services the property (Figure 18-2), and is planned to be upgraded to 25kV in 2014. Power for future development will be provided from this upgraded power line to the Project site.

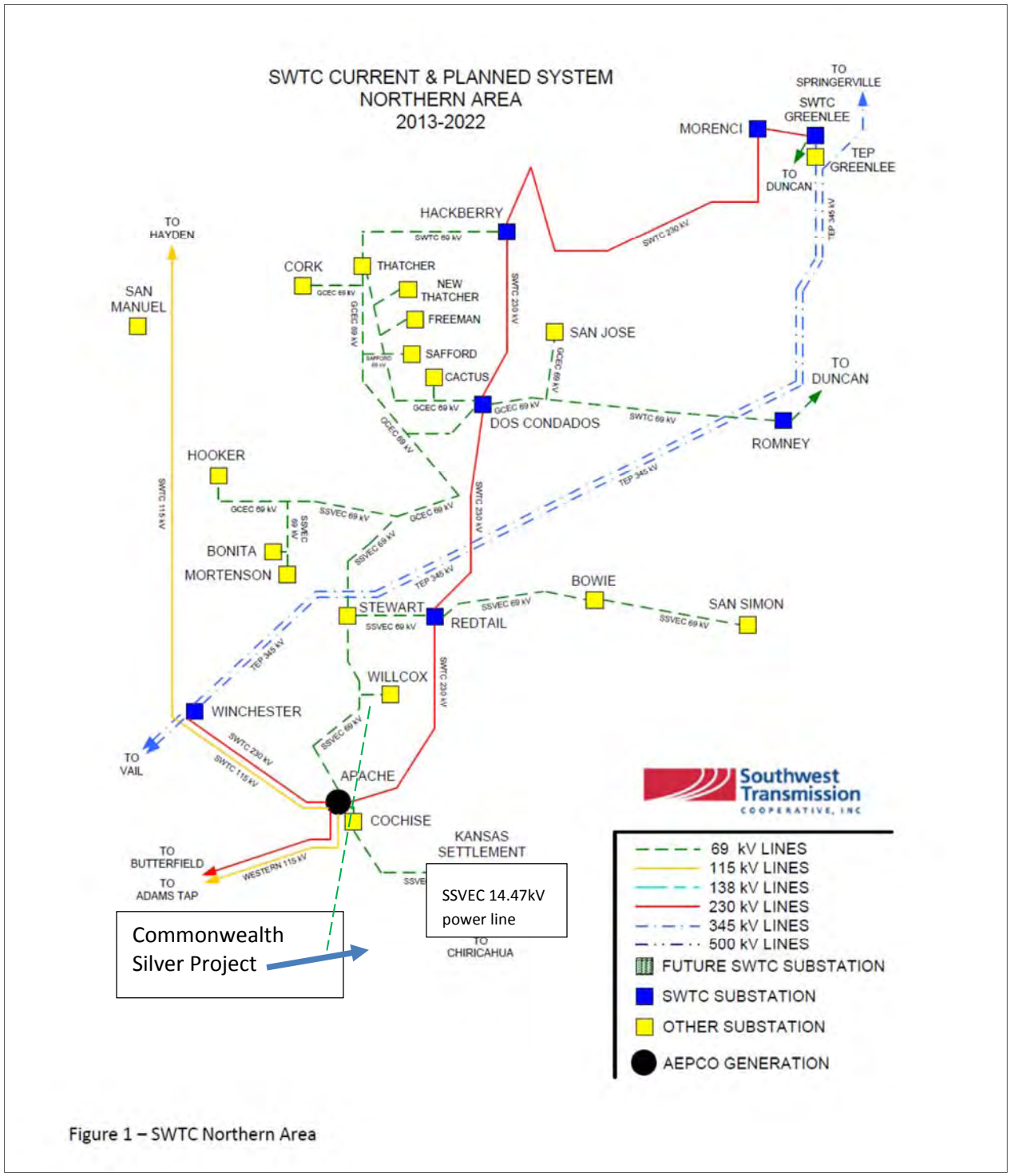


Figure 1 – SWTC Northern Area

Figure 18-2 Existing Power Infrastructure

The 14.47kV line runs adjacent to the project site and right to the entrance gate. Capital costs to access power will be minimal. The power line is scheduled for an upgrade to 25 KV in 2014. The power line will need to be extended an estimated 1 mile, and installation of the line should be in the neighborhood of $350,000.

Water

Water is currently obtained from a 6 inch diameter well located within 100 m of the north limit of the Ocean Wave Lode patented claim adjacent to the north property boundary of the Project. The well is 167 m (548 ft) deep. The standing water level in this well is reported as 98 m (321 feet) below the surface. The well is serviced by a 10 hp pump, and pump test results indicate available discharge of 25 gallons per minute at the depth of the pump. This water source is adequate for exploration and pre-feasibility Project activities. A larger water source will need to be developed for mining.

Water for operations will come from production wells on site, located south of the pit in the Douglas hydrologic basin.

Initially, production water will need to be provided from production wells which will be drilled on the property. Water in this area is fairly deep – 400 feet or more. It is anticipated that 2 water production wells will be required, and will cost approximately $20,000 each. Water usage has been estimated at 231 gpm (Table 18-1).

Table 18-1 Estimated Water Usage

	Required Makeup GPM
Leach Pad	50
Roads	7
Conveyor	8
Agglomeration	166
Total	**231**

The Commonwealth Silver and Gold Project is located on a drainage and hydrological divide. To the north is the Willcox Basin, and to the south is the Douglas Basin (Figure 18-3). Given the proximity of landowners to the north, it is anticipated that production water wells will be sited on the south side of the property in the Douglas Basin.

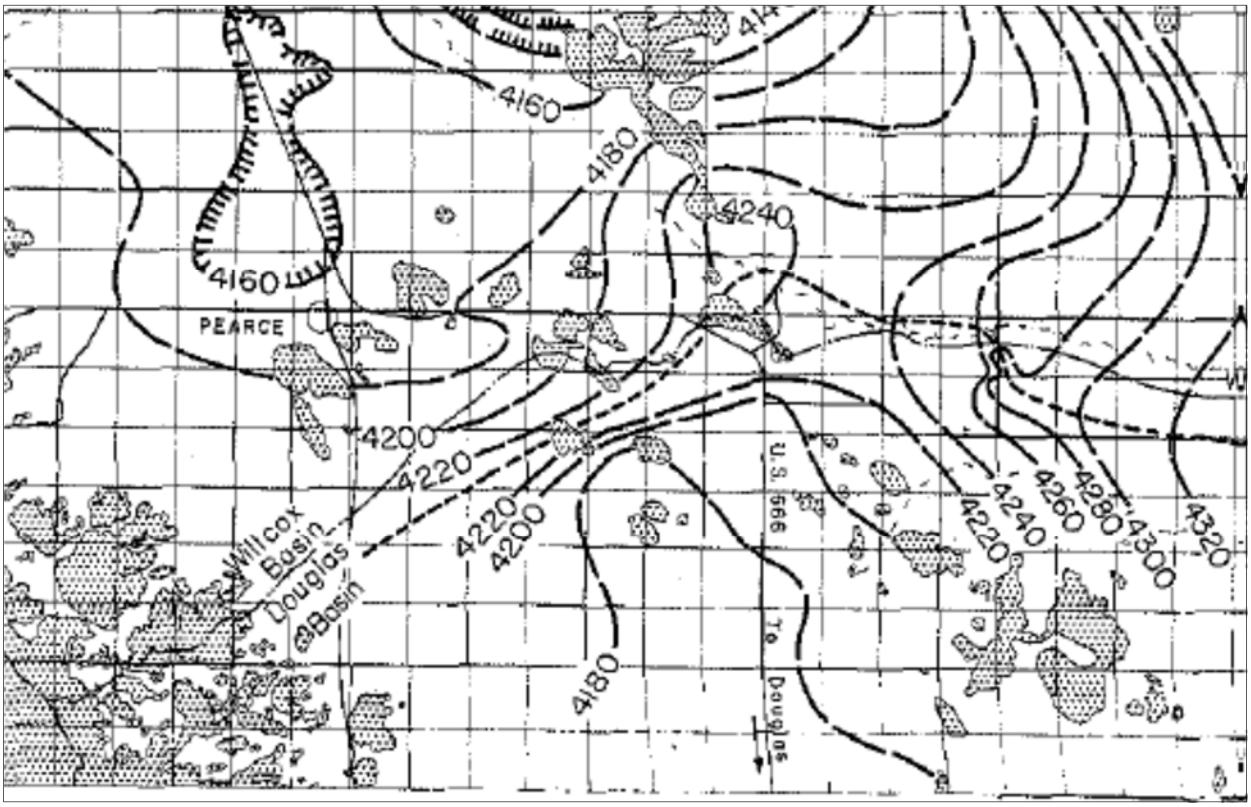


Figure 18-3 Regional Groundwater Isopach

Montgomery & Associates prepared a shortened well inventory including wells within 1 mile of the mine site. Well uses other than exploration include domestic water production wells. The Pearce School well is the only well designated as a public water supply. A few wells are designated for stock or irrigation uses.

Limited water quality data are available for the Pearce Elementary School well located to the west of the proposed mine site. The school well exceeds the drinking water maximum contaminant level for arsenic (naturally occurring), but employs point-of-use arsenic treatment systems to meet the required limit. In 2010, analytical results from water quality samples indicated that the arsenic level was exceeded. No additional water quality data are available for other wells near the proposed mine site at this time.

Wells investigated within one mile radius of the mine site indicate that water levels are approximately 350 ft below ground level. Long term water level measurements have indicated that the water levels are dropping, mostly due to agricultural uses.

Labor Availability

The cities of Sunsites, Pearce, Willcox, and Benson are reasonably close to the Project and should provide a ready source of mining personnel. There are experienced mining personnel associated with

active mining in the area at Safford and Johnson Camp who would be available for the Commonwealth operation. Trained personnel with experience in mining, heavy equipment operation, blasting, surveying, mill operation, etc. are available from the local communities and throughout the greater regional area. We anticipate approximately 150 personnel required for operations at Commonwealth.

Leach Pad and Ponds

The leach pad will be constructed in phases. Phase I will require an area of approximately 139,355 square meters (1.5 million square feet), which will provide for at least 1.25 years of production (4 lifts). At the end of the first year, the leach pad will be expanded, allowing for construction to be completed prior to the requirement of additional pad space. The leach pad will be expanded periodically throughout the mine life.

Barren and pregnant ponds will be constructed next to the Merrill-Crowe plant, with sufficient volume for managing the leach solution, and sized for storm events. Diversion canals will be constructed around the pit, ponds, leach pad, and plant areas to handle the design storm events.

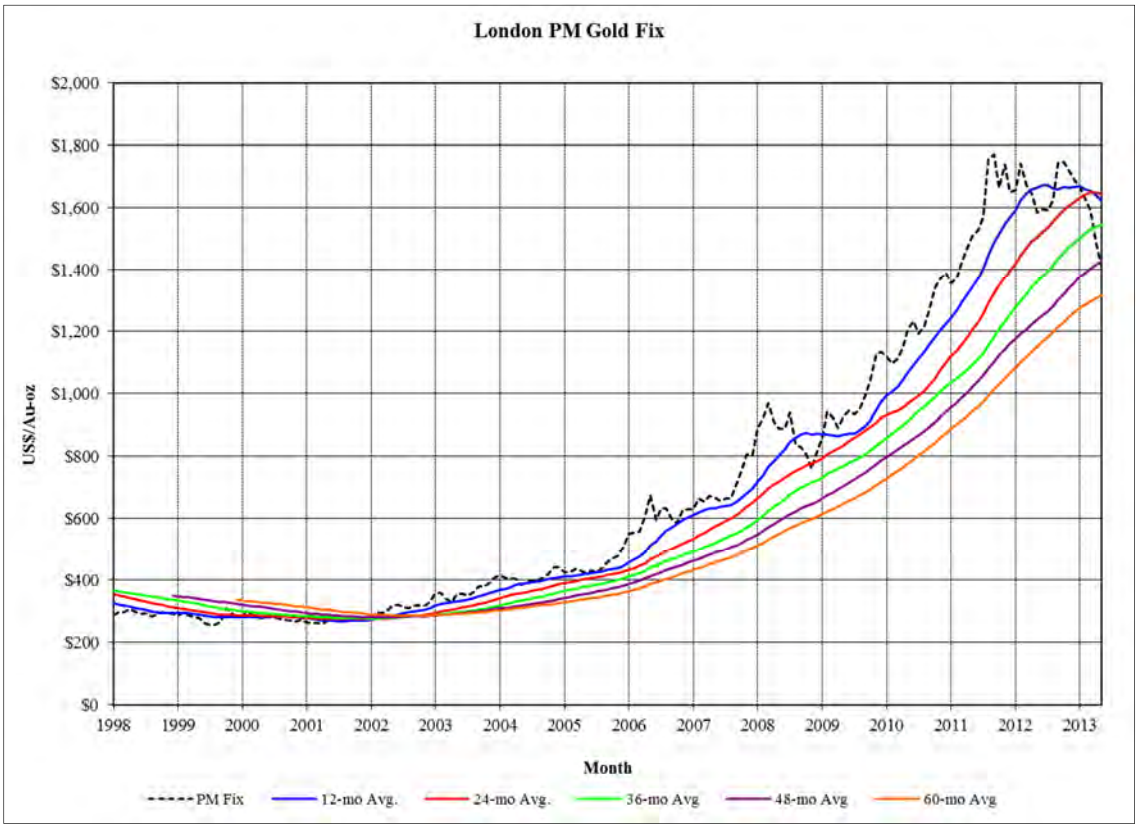
19. Market Studies and Contracts

Gold and silver markets are stable, transparent, global markets serviced by well-known smelters and refiners located throughout the world. Silver and gold will be refined to .9999 or .99999 purity in the refinery, and, as such, are fungible commodities bought and sold universally. Therefore no contracts have been negotiated at this stage of the Commonwealth Silver and Gold Project. CSGM does not have any forward sales of silver or gold, nor does it have any hedging programs in place at this time.

For the economic analysis, CSGM selected silver and gold prices that approximate the five-year trailing average prices of silver and gold at the date of this report (silver - $22.50; gold - $1,350). The trailing average prices as of the date of the report were $24.64 per ounce of silver and $1,354 per ounce of gold. With the recent volatility in the prices of silver and gold and given that the spot prices at the date of the report are within 3% of the prices selected. HRC's QP has reviewed Commonwealth's commodity price projections and the prices selected by CSGM adequately represent that of the current market. The results thereof support the assumptions in this technical report

As shown in Figures 19-1 and 19-2, silver and gold prices have been on a general upward trend for around 10 years. There is no method to predict future sales prices of silver and gold. However, a five year trailing average is a reasonable basis for the economic analysis of the project, given that the mine life is expected to exceed 10 years and averaging prices over this time period smoothes out the fluctuations up and down.

Figure 19-1 Silver Price Average Trends



Notes:
Gold and silver daily London PM fix prices are averaged get the monthly averages, which are entered into the gold and silver data sheets. Daily price source: http://www.kitco.com/gold.londonfix.html

Figure 19-2 Gold Price Average Trends

The economic analysis further addresses the basis of the silver and gold price assumptions by preparing sensitivity analyses for various price levels above and below the selected prices.

20. Environmental Studies, Permitting and Social or Community Impact

Summary Results of Environmental Studies

CSGM is currently conducting the baseline resource studies necessary to obtain the permits and authorizations required to reopen the Commonwealth (Pearce) Mine.

Darling Environmental & Surveying, Ltd. completed a daytime roosting bat survey at the mine site in 2012. The following paragraphs are excerpted from their report, 'Bat Roost Survey of the Commonwealth Silver Mine, Pearce, Arizona' dated September 2012:

"The nature of the activities associated with the new mining operation will inherently destroy all of the underground mine features including shafts, declines, adits and stopes that have been excavated into Pearce Hill. In preparation for this eventuality, at the recommendation of Tim Snow, Arizona Game and Fish Department Bat Biologist, bat exclusion devices were installed during the winter of 2011/2012 on the exterior of all openings that provide access to underground mine features. Exclusion devices consist of netting, steel mesh and solid wooden or metal covers that are anchored to the rock surface.

Commonwealth Silver plans to leave the exclusion devices in-place as the site is developed into an open pit mine operation. The intention is to exclude bats from the mine, especially the endangered lesser long-nosed bat (LLNB: *Leptonycteris yerbabuenae*) to prevent impacts to bats during the feasibility and construction phases of the project. Furthermore, by excluding all bats from the mine, conflicts with state and federal wildlife laws will be avoided.

The roost survey consisted of two separate site visits to the Pearce Mine to search for the presence or absence of bats inside the underground mine. The survey was conducted systematically throughout the underground workings by walking slowly and thoroughly scanning all safely accessible tunnels and looking into crevices, cracks and other features where roosting bats may occur. No roosting bats or sign of bat usage such as bone material from dead bats, insectivorous bat fecal material (guano), nectar bat fecal material (yellow splatter), insect debris below roost sites, and staining on walls or ceilings from body oils of bats were observed inside the Pearce Mine during the two surveys.

No bat roosts were detected. As long as the bat exclusion devices are maintained, no further bat surveys or other mitigation measures are recommended for the Commonwealth site."

Known Environmental Issues

HRC knows of no known environmental issues that could materially affect the issuer's ability to extract the mineral resources.

Waste and Tailings Disposal, Site Monitoring, Water Management and Post Mine Closure

The wastes generated at the Commonwealth Silver and Gold Project will be disposed of in several different ways. The unmineralized native rock from the pit will be placed in stockpiles on the mine site. Mineralized rock, after having gone through the leaching process, will remain on the heap leach pad into mine closure and post closure. There are no tailings associated with the Project.

There will be a series of groundwater monitoring wells on site, which will be used to monitor impacts to groundwater to comply with APP regulations. There will be no wastewater discharged from any of the facilities. The site will run as a zero discharge facility. As a zero discharge facility all wastewater will be recycled back into the mine operations. Stormwater falling directly onto the heap leach pad and process ponds will not be discharged but incorporated into the mine operations. Stormwater run-on will be diverted around the mine facilities via stormwater ditches. Stormwater that is impacted by other mining activities will be controlled in accordance with the Stormwater Pollution Prevention Plan that will be developed for the site. In the case of a storm event, there is also the option to install additional water evaporators or an evaporation pond used specifically for excess stormwater. After the mine has been closed, facilities will be reclaimed so that any discharges from the facilities will have negligible impacts on any receiving waters. The reclaimed facilities will be graded and provided with a cover to provide a compatible surface for post mining use and erosion protection.

Permitting Requirements and Status of Permit Applications

The Commonwealth Silver and Gold Project is located within a recognized historic mining area, with the mineral resource wholly located on patented mining claims and planned facilities located entirely on private land. Because all of the lands potentially impacted by the project are private, the permit process should be limited to recognized and conventional permitting programs within the state of Arizona (no federal permitting required).

CSGM holds three Mineral Exploration Permits issued by the state of Arizona. The three Mineral Exploration Permits pertain to 647.27 ha (1,599.44 acres) of mineral rights and provide for surface access. These permits are issued by the Arizona State Land Department. Through the anticipated life of the operation, an Aquifer Protection Permit, Air Quality Permit, Mined Land Reclamation Permit and Stormwater Discharge Authorization will be required from the state of Arizona.

Aquifer Protection Permit

An Aquifer Protection Program (APP) permit will be required to be issued through the Arizona Department of Environmental Quality (ADEQ). This permit is established to minimize affects to groundwater quality in Arizona where a reasonable probability exists that pollutants may reach an aquifer. The Arizona Administrative Code (A.A.C.) R18-9-A202(A)(5) requires that an application for an APP include a description of the Best Available Demonstrated Control Technology (BADCT) to be

employed at a specific mining facility. There are five demonstrations required for obtaining an APP permit:

- The facility will be designed, constructed, and operated in accordance with BADCT requirements;
- The facility will not cause or contribute to an exceedance of Aquifer Water Quality Standards
- (AWQS) at the point of compliance or, if an AWQS for a pollutant has been exceeded in an aquifer, that no additional degradation will occur (A.A.C. R18-9-A202(A)(8)(a and b));
- The person applying for the APP is technically capable of carrying out the conditions of the permit (A.A.C. R18-9-A202(B));
- The person applying for the APP is financially capable of constructing, operating, closing, and assuring proper post-closure care of the facility (A.A.C. R18-9-A203); and
- The facility complies with applicable municipal or county zoning ordinances and regulations (A.A.C. R18-9-A201(A)(2)(c)).

In the case of the Commonwealth Project, APP coverage will be required to address aquifer protection in relation to any proposed onsite open pit mines, process solution ponds, waste rock dumps, tailings facilities, and/or leaching operations. Development of the draft permit application, including implementation of a hydro-geologic characterization study, will need to be completed for the APP permitting phase of the Project.

A permittee or applicant is required to propose an applicable point of compliance (i.e. monitoring well) or multiple points of compliance (depending on the operation) to monitor impacts from the operations on groundwater and to ensure that BADCT provisions are effective. Typically, the monitoring is conducted for eight consecutive quarterly observations to establish baseline conditions during the early stages of mine facility development. Alert levels are then established based on this monitoring to signal when impacts may threaten groundwater quality and intervention may be required.

Financial assurance is required prior to issuance of an APP permit. The law requires that the permitting process be limited to 329 days under the Arizona licensing timeframes rule. The completion time may be extended if the proponent submits incomplete application materials and the ADEQ requests additional information to complete their review. The permit process typically takes twelve to eighteen months, depending on the complexity of the hydrogeology and mining operations as well as the workload/budget restrictions in place at the regional ADEQ office. An expedited APP process is available through ADEQ if all necessary information (design drawings etc.) is properly prepared.

Air Quality Permit

Air quality is regulated at the federal level by the EPA under the Clean Air Act ("CAA"). National Ambient Air Quality Standards ("NAAQS") have been established for each of the criteria pollutants of ozone,

carbon monoxide, nitrogen dioxide, sulfur dioxide, particulate matter less than 2.5 microns and less than 10 microns aerodynamic diameter, and lead. Authority for air quality permitting has been delegated by the EPA to the ADEQ.

Air emissions are regulated under the CAA in the context of the NAAQS. The law and regulations differentiate between mobile and stationary sources, as well as between new and existing facilities. New or modified existing stationary sources must meet performance standards, referred to as New Source Performance Standards ("NSPS"), established by the EPA for certain categories of sources. The standard of performance for a particular facility is based on the application of the best available system of emission reduction, taking into consideration cost. New major sources are subject to preconstruction review, with different standards and levels of review applied to facilities proposed within attainment areas ("Prevention of Significant Deterioration" requirements) and nonattainment or non-classifiable areas ("New Source Review" requirements).

Emissions of "hazardous air pollutants" ("HAPs") are also regulated under the CAA. The EPA sets standards for HAPs for both specific pollutants and families of pollutants that are not emitted by a sufficient number of sources to justify development of a NAAQS for that pollutant but that can have serious health implications for humans. The CAA requires identification of major sources of HAPs as well as area sources (sources below the volumetric thresholds for major sources). Sources are required to obtain permits for emitting any of the HAPs, again with variance between new and existing source standards.

The permitting components of the CAA for stationary sources are described in Title V of the CAA; thus, air emission operating permits are commonly referred to as Title V permits. These permits comprehensively address all relevant air emissions limitations, monitoring and reporting requirements, HAPs, and NSPS. The ADEQ has established three other classes of permits. Class I permits are required for major sources, solid waste incineration units, affected sources (a defined term) and any source in a category designated by the EPA Administrator and adopted by the ADEQ Director. Mining operations qualify as Class I major sources. Class II permits are required for construction or modification of sources that otherwise do not qualify for Class I permits but that emit pollutants above certain thresholds or for sources that are certain types of facilities. Finally, General Permits are pre-approved permits available for a specific class of sources, such as common types of facilities like gasoline stations. Depending on the process, CSGM may incorporate facilities covered under 40 CFR 60.380 Subpart LL, Title V permitting and NSPS review may apply.

Metallic Mineral Processing Plants are covered under 40 CFR 60.380 Subpart LL and are specific to operations from mining through concentrating. Included are all material transfer and storage operations that precede those operations that produce refined metals from metallic mineral concentrates.

In addition to Subpart LL, Subpart Kb for petroleum storage will also apply to the facility. Petroleum storage is specific to fuel and reagent tank storage, and would not apply to "flow through" process tanks.

In the arid southwest, fugitive emissions are a problem if not properly controlled. In an effort to conserve water and protect watershed areas, alternative forms of dust control should be investigated. A combination of dust suppressants, water, and cover or hooding can be used to manage fugitive emissions from process areas. Capping, seeding, and land management techniques will be used on waste rock piles and storage areas. In addition, captured water from operations and storm water will be used when and where appropriate to control dust to conserve groundwater resources. Management techniques for operations such as speed control, cleanup, and road maintenance can also be used to conserve resources and manage the potential to create fugitive emissions.

Mined Land Reclamation Permit

A Mined Land Reclamation Permit in Arizona is issued through the Arizona Mine Inspector's office. An applicant is required, through the application process, to identify:

- The nature of the operations,
- Anticipated impacts and mitigation measures,
- Anticipated post mining land use, and
- Reclamation measures required to achieve the post mining land use. Reclamation typically involves those measures necessary to stabilize reclaimed lands (e.g. rock armor or revegetation) and provide public safety protection (e.g. reduce high walls or fence open pits).

The Reclamation Permit requires financial assurance to ensure that the costs for reclamation will be available if the permittee becomes insolvent. The amount of the financial assurance required will be adjusted if there is any overlap between the costs of reclamation and the costs for APP closure. The review of a permit application typically takes approximately four months, including a public comment period.

Stormwater Discharge Authorization

Either an individual National Pollutant Discharge Elimination System (NPDES) or a Multi-Sector General Permit (MSGP) is required for mining operations in Arizona, depending on the individual operation. The MSGP requires preparation of a Stormwater Pollution Prevention Plan (SWPPP). The MSGP coverage also requires the establishment of discharge outfalls and regular analytical monitoring of storm water discharges. The Project will require coverage under the MSGP program. Depending upon the nature of discharges from the Project area, individual AZPDES coverage may be required as well. CSGM will need to refine a geochemical management strategy such that materials are excavated and placed to minimize the potential to generate acid or alkaline rock drainage. Mineralized materials can be encapsulated, to

the extent practicable, in low-grade waste deposition areas, thus reducing storm water contact and potential metals leaching.

Reclamation Bonding Requirements

Financial assurance for reclamation is required by the ADEQ under the APP program and by the Arizona Mine Inspector for the Mined Land Reclamation Permit. ADEQ requires bonding for closure of the APP-regulated facilities prior to issuing the permit. The leach pad and process ponds would fall under the ADEQ requirements. The Arizona Mine Inspector requires bonding for reclamation of mining facilities not covered under the APP. This would include staging areas, crushing pads, process buildings, haul roads, pits, and native rock or mineralized rock storage areas.

Social or Community Impact

HRC knows of no existing, anticipated, or reasonably potential social or community impact that might materially affect the advancement of the Commonwealth Silver and Gold Project.

Remediation and Reclamation Requirements and Costs

The closure of the Commonwealth Silver and Gold Project includes both permitted and other facilities on the mine site. In accordance with the APP, a closure and post-closure plan will be drafted and submitted to the ADEQ for approval within ninety (90) days of notifying ADEQ of the intent to permanently cease operations. The closure plan strategy will eliminate, to the greatest extent practicable, any reasonable probability of further discharge from the facility and of exceeding Aquifer Water Quality Standards (AWQS) at the applicable point of compliance. The closure plan will outline management strategies for the facilities and those strategies may include:

- Prior to closure, leach pad mineralized rock will continue to be leached until concentrations of gold in the leach solutions fall below economical levels. At that time the leach pad will be contoured and covered with clean mine native rock and the eastern pit will be benched and contoured.. The leach pad will then be allowed to drain down prior to removal of other facilities.
- Process ponds will be drained and contained solutions removed or evaporated. Any solid residues on the upper liner will be removed and disposed appropriately. The lower liner and underlying soils will be inspected for visual signs of liner damage, defects, or leakage through the liner. If visual signs of leakage are found, additional investigation and soil remediation may be required. Once the underlying soil is determined to be clean, the liner can be placed back into the excavation and the area backfilled. If removed, the liner will be disposed off site. The area will be graded to drain surface runoff and minimize precipitation infiltration and capped with clean borrow or rock.

21. Capital and Operating Costs

Capital Costs

The capital costs for developing the Commonwealth mine are estimated from other recent mine development capital cost history, as well as quotes for some of the larger components. The current concept for the project operation is contract mining and crushing, and therefore capital cost for purchase of the mining fleet has been avoided. Table 21-1 presents the capital cost to develop the mine and processing facilities, as anticipated by HRC.

Table 21-1 Mine and Processing Facility Capital Costs

Capital Category	Estimated Cost
Mobile Equipment	$3,203,000
Plant & Facilities	$13,712,000
General & Infrastructure	$3,820,000
Contingency & CM	$5,168,600
Subtotal - Initial Capital	$25,903,600
Operating Capital & Startup Costs	$18,239,483
Total Development Capital	$44,233,083

Mining Fleet

The mining fleet will be the responsibility of the contract mining company, and therefore does not require capital investment by CSGM. The only capital requirements for the mining fleet are the building of haul roads and a truck shop. Totals of $1 million for haul road development and $2.045 million for construction of the truck shop are included in the capital cost estimate.

Crushing and Conveying

The Commonwealth Silver and Gold Project will require two stages of crushing for 60% of the mineralized material, and three stages of crushing for the remainder of the materials. It will not be possible to campaign the mineralized material, therefore all three crushing stages will need to be built and operated. The crushing and conveying system will consist of:

- Primary Crusher (jaw)
- Conveying to secondary and tertiary crushing
- Secondary crusher (cone)
- Screening after secondary crushing
- Tertiary crusher (cone)
- Grass Hopper conveyors to heap
- Telestacker conveyor on heap

Lime and cement will be added directly onto the overland conveyor, and agglomeration will occur as the mineralized is moved through the conveying system.

Since the contract mining concept includes mining and crushing and conveying, the need for this capital has been avoided, and is therefore not included in the initial capital estimate.

Buildings

Buildings associated with the Commonwealth Silver and Gold Project will consist of a truck shop (included above), warehouse, laboratory (included in the plant and facilities below) and a small administration building. The total estimated cost for all buildings is $1,400,000.

Infrastructure

Required infrastructure for the Project will include power supply, water supply, access roads, communication equipment, emergency vehicles, and rescue supplies. The total estimated cost for general infrastructure is $3.8 million, as shown in Table 21-2.

Table 21-2 Project Infrastructure Capital Cost

G&A Capital, Description	Initial Capital Cost, with Sales Tax
General & Infrastructure	
Site Access Road	$250,000
Relocate County Road	125,000
Diesel Fuel Purchase	20,000
Electric Power Purchase	50,000
Potable Water Purchase	5,000
Site Environmental Cost	41,000
Misc. Construction Materials	11,000
Site Construction Insurance	200,000
Vendor Assistance	50,000
Site Buildings	1,431,000
Mine Training, including Salaries	290,000
Site Laboratory Operations	90,000
Site Environmental Training Program	13,000
Site Safety/Security Costs	24,000
Recruitment	100,000
Site G&A Operating Cost	110,000
Rescue Supplies	80,000
Computers Software	80,000
Safety Supplies	80,000
General Communications	106,000
Small Tool Purchases	53,000
Site Ambulance	42,000
Power Line - 69kv, 2 miles	358,000
Power - Emer Generator, 1 Mw	211,000
Total	$3,820,000

Plant and Facilities

The throughput anticipated for the Commonwealth mine will require a Merrill-Crowe plant capable of 3000 gpm. Based upon other recent mining installations and CostMine information, we expect the capital cost for purchasing and installing the Merrill-Crowe plant and associated facilities to be approximately $13.712 million, as shown in Table 21-3.

Table 21-3 Estimated Plant & Facilities Capital Cost

Plant & Facilities	Initial Capital Cost, with Sales Tax
Conveying Equipment	$4,752,000
Plant General	603,000
Precipitation and Filtration	1,869,000
Leach Pad - Est Total	3,944,000
Leach Pad Overliner	304,000
Ponds - Est Total	669,000
Assay Laboratory	1,022,000
Fills - Chem/Catalysts/Fuels	549,000
Sub-total Plant & Facilities	$13,712,000

Heap Leach Pads and Ponds

The heap leach pad will initially be constructed at approximately 140,000 square meters in size. This size will allow for loading of mineralized material on the pad for 1 ¼ years. The construction of the next phase of leach pad will begin after one year of operation. The second phase of pad construction is anticipated to be an additional 232,000 square meters, constructed in year 2, and the final expansion will add another 204,000 square meters, which will be constructed in year 6. The cost of initial leach pad construction is expected to be $3.9 million, and is included in Table 21-3.

Pregnant and barren ponds will be constructed to handle the 3000 gpm flow of leachate. The cost of pond construction is estimated at $669,000 as shown above.

Construction Management & Contingency

The costs for construction management and operations staff during construction were estimated at approximately $0.45 million. Monitoring wells required for the APP permit are estimated at $506k. Contingency for capital cost was estimated at $4.2 million, and an additional $15k for contingency on mine equipment. The components of the construction management and support cost estimate are presented in Table 21-4.

Table 21-4 Estimated Construction Management and Support Capital Costs

Capital Indirects & Contingency, Description	Initial Capital Cost, with Sales Tax
General & Infrastructure	
Construction Management - CSGM	$250,000
Operations Staff During Construction	200,000
Monitor Well & App Permit	506,000
Contingency Mine Equip (@ 10%)	15,800
Contingency (@ 20%)	4,196,800
Sustaining Contingency (@ 20%)	0
Total	$5,168,600

Working Capital

Four months of operating costs were included in the initial capital cost estimate for losses during startup, and working capital has also been included at a level of approximately $7 million, as shown in Table 21-5. This will allow time for initial operations to begin, the leaching to start, silver and gold to report to the Merrill-Crowe plant prior to producing dore at the mine, and receipt of payment from the offtake organizations.

Table 21-5 Working Capital

Other Capital Concepts, Description	Initial Capital Cost, with Sales Tax
Participation by NPI Owners	-$518,073
Working Capital	7,048,000
Land Purchase	1,250,000
Losses during Startup	10,487,758
Total	$18,267,685

Project Operating Costs

Operating costs for the Project were developed from known similar operations and historical information. The total operating costs per tonne of mineralized material for the life of mine are shown in Table 21-6.

Table 21-6 Estimated Project Operating Costs

Operating Costs	$/oz AuEq	$/tn min mat'l.	$/tn mined
Total Mining	$369.14	$5.85	$2.97
Total Processing	$386.72	$6.13	
Total Site General & Administration	$40.41	$0.64	
Property Taxes	$4.70	$0.07	
Cash Operating Costs	**$800.97**	**$12.69**	
Transportation and Refinery	$7.75	$0.12	
Royalties	$16.52	$0.26	
Severance Taxes	$5.92	$0.09	
Total Cash Costs	**$831.16**	**$13.17**	

Manpower

The total manpower for the project is expected to reach 73, not including the contract miner staffing. The anticipated manpower breakdown is presented in Table 21-7.

Table 21-7 Operating Manpower

Staffing area	No of People
Mining G&A	5
Drilling and Blasting	0
Loading	0
Hauling	0
Roads and Dumps	0
Mine Maintenance	0
Engineering	4
Geology & Grade Control	4
Process Plant and Pad	44
General and Administrative	16
Total Staffing	**73**

Mining

Mining will be accomplished with a mining contractor, using 90-tonne trucks and a shovel. Expected mining operating costs for a typical year (year 2) are summarized in Table 21-8.

Table 21-8 Mining Operating Costs (typical year)

Mining Total Mined tonnage	Operating Cost	Cost / tonne mined 6,931,165
Contract Mining	14,878,984	$ 2.15
Diesel Fuel	4,088,741	$ 0.59
Drilling	incl	incl
Blasting	incl	incl
Loading	incl	incl
Hauling	incl	incl
Roads & Dumps	5,835	$ 0.00
Dewatering	incl	incl
Mine Maint.	incl	incl
Engineering	402,390	$ 0.06
Geology	443,071	$ 0.06
Factor, @ 2.5%	495,476	$ 0.07
Total Mining	**20,314,497**	**$ 2.93**

Note that these operating costs are slightly different than the life-of-mine figures included in the summary presented in Table 21-6, because the costs listed in Table 21-8 are only indicative of year 2 operating costs.

Operating costs for the contract miner were developed from quoted pricing from two contract mining companies, as well as recently experienced costs for contract mining in the area.

Processing

Processing consists of crushing, conveying, leaching, and Merrill-Crowe operation. Estimated operating costs for the processing portion of the Commonwealth Mine for a typical year (year 2) are shown in Table 21-9.

Table 21-9 Plant Operating Costs (typical year)

Plant Operating Total Processed Tonnage	Operating Costs	Cost/tonne processed 3,650,000
Plant G&A	$279,638	$0.08
Crushing	$7,539,777	$2.07
Conveying	$1,866,425	$0.51
Plant	$4,572,745	$1.25
Leaching	$6,934,354	$1.90
Plant Maintenance	$693,153	$0.19
Assay Lab	$379,513	$0.10
Factor, @ 2.5%	$556,640	$0.15
Total Yearly	$22,822,245	$6.25

Again, note that these operating costs are slightly different than the life-of-mine figures included in the summary presented in Table 21-6, since the costs listed in Table 21-9 are only indicative of year 2 operating costs.

Site General and Administrative

General and administrative operating costs include staffing for the mine site operations, such as administration, human relations, security and safety, accounting, environmental, etc.

Estimated operating costs for general and administrative staff for a typical year (year 2) are shown in Table 21-10.

Table 21-10 G&A Operating Costs

Site G&A Total min mat'l. tonnage	Operating Cost	Cost / tonne mined 3,650,000
Administration	958,962	$ 0.26
Human Relations	186,900	$ 0.05
Security & Safety	349,498	$ 0.10
Accounting	298,244	$ 0.08
Purchasing	271,296	$ 0.07
Environmental	325,020	$ 0.09
Factor, @ 2.5%	59.748	$ 0.02
Total G&A	**2,449,668**	**$ 0.67**

22. Economic Analysis

Summary

Certain statements made and information contained herein are considered "forward-looking" within the meaning of applicable Canadian securities laws. These statements address future events and conditions and so involve inherent risks and uncertainties. Actual results could differ from those currently projected.

HRC envisions the Commonwealth Silver and Gold Project as an open pit, heap leach operation with Merrill-Crowe processing of leachates completely contained on patented or privately held surface lands, with an estimated resource of 31.2 million tonnes grading 0.39 gpt gold and 32 gpt silver. The operations are planned to run at a rate of 10,000 tonnes per day, 365 days per year, with all mineralized material being crushed and agglomerated prior to stacking on the heap. Metal recoveries are expected to average around 79% for gold and about 34% for silver.

Economic analysis of the base case scenario for the Project uses a price of US$1350/oz for gold, which is approximately equal to both the five-year and the spot price at the end of October, 2013. The silver price for the base case, for all prices in the sensitivity calculations, and for the cut-off grade determination is at a factor of 1/60th of the gold price, which was the ratio at the end of October, 2013. The economic model shows an After-Tax Net Present Value @ 5% ("NPV-5") of $101.3 million using a 0.24 gpt gold equivalent (AuEq) mining cutoff grade, as well as an After-Tax Rate of Return ("IRR") of 58%. Table 22-1 summarizes the projected net present value, NPV-5; internal rate of return, IRR; years of positive cash flows to repay the negative cash flow ("Payback Period"); multiple of positive cash flows compared to the maximum negative cash flow ("Payback Multiple") for the Commonwealth Silver and Gold Project on both After Tax and Before Taxes bases.

Table 22-1 Commonwealth Economic Performance versus Gold Price

Au Price	Ag Price	NPV @ 5.0%, After Tax	Internal Rate of Return, After Tax	Payback Period, After Tax	Payback Multiple, After Tax	NPV @ 5.0%, Pre-Tax	Internal Rate of Return, Pre-Tax	Payback Period, Pre-Tax	Payback Multiple, Pre-Tax
$1,050	$17.50	$22.29	18.6%	5.20	1.83	$34.19	24.7%	2.60	2.20
$1,125	$18.75	$42.65	29.6%	2.48	2.45	$60.92	38.2%	2.16	3.02
$1,200	$20.00	$62.35	39.6%	2.11	3.04	$87.65	50.9%	1.81	3.85
$1,275	$21.25	$81.82	49.1%	1.81	3.63	$114.34	63.1%	1.54	4.68
$1,350	$22.50	$101.25	58.2%	1.59	4.22	$141.02	74.9%	1.26	5.53
$1,425	$23.75	$120.41	67.0%	1.37	4.80	$167.70	86.5%	1.04	6.39
$1,500	$25.00	$138.72	75.1%	1.28	5.34	$194.38	97.9%	0.95	7.24
$1,575	$26.25	$156.42	82.4%	1.14	5.85	$221.06	109.1%	0.79	8.09
$1,650	$27.50	$174.12	89.7%	1.02	6.36	$247.73	120.3%	0.68	8.95

Table 22-2 summarizes the projected production schedule and cash flows. The economic evaluation and schedule excludes inferred resources and are is based on measured and indicated resources only. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources estimated will be converted into mineral reserves.

Table 22-2 Commonwealth Schedule and Cash flow

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Life-of-Mine
Mine Production												
Tonnes Ore Mined			2,978,401	3,650,000	3,650,000	4,620,370	2,310,932	3,602,269	3,650,000	3,650,000	194,757	28,306,729
Tonnes LG Ore Mined			298,247	429,898	333,730	415,351	269,727	500,948	318,149	232,513	48,282	2,846,846
Waste			3,222,740	2,851,267	1,379,984	6,758,657	7,578,189	5,080,885	2,318,400	981,697	101,379	30,273,199
Total Tonnes Mined			**6,499,388**	**6,931,165**	**5,363,714**	**11,794,378**	**10,158,848**	**9,184,103**	**6,286,549**	**4,864,211**	**344,419**	**61,426,774**
SR			**1.0**	**0.7**	**0.3**	**1.3**	**2.9**	**1.2**	**0.6**	**0.3**	**0.4**	**0.97**
Other Tonnes			240,000	240,000	240,000	240,000	1,319,999	240,000	240,000	240,000	2,357,162	5,357,161
Total Tonnes Moved			**6,739,388**	**7,171,165**	**5,603,714**	**12,034,378**	**11,478,847**	**9,424,103**	**6,526,549**	**5,104,211**	**2,701,580**	**66,783,935**
Process Production												
Tonnes Ore Processed			**3,043,673**	**3,650,000**	**3,650,000**	**3,650,000**	**3,660,000**	**3,650,000**	**3,650,000**	**3,650,000**	**2,549,902**	**31,153,575**
Au, g/tn			0.3	0.4	0.4	0.4	0.3	0.5	0.5	0.4	0.1	0.39
Ag, g/tn			48.7	31.4	31.4	40.8	29.0	28.6	32.3	32.3	9.0	31.95
Net Revenue	$ (400,000)	$ (3,650,000)	$ 67,929,989	$ 71,067,367	$ 85,320,366	$ 84,754,778	$ 66,408,010	$ 81,338,995	$ 89,936,472	$ 81,460,984	$ 17,018,268	$ 641,185,230
Cash Operating Costs	0	0	42,546,561	45,209,021	41,767,594	57,347,659	53,577,669	50,668,266	44,018,778	40,514,187	14,700,027	390,349,761
Total Capital & Working Capital	2,328,314	21,869,830	9,528,384	2,673,394	2,778,503	3,268,503	2,786,115	3,268,503	2,778,503	2,778,503	1,024,216	55,082,766
Beginning Cash	**0**	**-2,728,314**	**-28,248,144**	**-16,552,852**	**1,789,173**	**31,475,276**	**50,469,680**	**58,678,410**	**80,333,173**	**112,170,637**	**141,011,675**	
Period Net Cash Flow	**-2,728,314**	**-25,519,830**	**11,695,292**	**18,342,024**	**29,686,103**	**18,994,404**	**8,208,731**	**21,654,763**	**31,837,464**	**28,841,038**	**1,294,025**	**142,305,700**
Ending Cash	**-2,728,314**	**-28,248,144**	**-16,552,852**	**1,789,173**	**31,475,276**	**50,469,680**	**58,678,410**	**80,333,173**	**112,170,637**	**141,011,675**	**142,305,700**	**142,305,700**

The projected total lifespan of the project is 11 years: two years of pre-production and construction, and 9 years of full operations. Approximately 392,400 ounces of gold and 32.0 million ounces of silver are projected to be mined, with 311,500 ounces of gold and 10.9 million ounces of silver recovered and produced for sale. A total capital investment of $55.1 million, including contingency and sustaining capital, is projected. Following the Gold Institute ("GI") guidelines, life-of-mine average base case cash operating cost is projected to be $1281 per ounce of gold sold, before credits for silver sales, and $492 per ounce of gold after silver credits. The GI life-of-mine average base case total cash cost (including royalties and production taxes) after credits would be $528 per ounce, and the GI life-of-mine average base case total production cost after credits is expected to be $708 per ounce. Table 22-2 details the GI life-of-mine average base case total production cost on a per ounce gold equivalent basis and on a cost/tonne of mineralized material basis.

Table 22-3 Commonwealth Total Production Cost/ounce Gold Equivalent

Operating Costs	$/oz AuEq	$/tn min mat'l.	$/tn mined
Total Mining	$369.14	$5.85	$2.97
Total Processing	$386.72	$6.13	
Total Site General & Administration	$40.41	$0.64	
Property Taxes	$4.70	$0.07	
Cash Operating Costs	**$800.97**	**$12.69**	
Transportation and Refinery	$7.75	$0.12	
Royalties	$16.52	$0.26	
Severance Taxes	$5.99	$0.09	
Total Cash Costs	**$831.23**	**$13.17**	
Total Capital Costs	$113.86	$1.80	
Total Production Costs	**$945.09**	**$14.97**	

As previously mentioned, the gold price used in the economic evaluation (US$1,350/oz Au) is the five-year trailing average and approximate spot price as of the end of October 2013. The silver price for each case was determined by dividing the gold price by the end of October 2013 gold:silver price ratio (60:1).

Taxes

State, local, and federal taxes, including income taxes, have been considered in this preliminary economic assessment, and are included in the economic analysis.

Royalties

A 2 percent royalty, calculated on the gross proceeds less transportation and refining costs, has been included for all of the metal produced, as required by underlying agreements, until a $2 million payment is made to buy down the royalty to 1%, which is expected to occur in the first quarter of year 2.

Corporate Income Taxes

United States and State corporate taxes have been considered in the economic analysis.

Economic Model

Basis of Evaluation

Mineral resources were incorporated in the model only if classified as Measured or Indicated Resources according to CIM definitions. Any Inferred mineral resource tonnes projected to be mined were categorized as waste, which are planned to be sent to the waste dump. A throughput of 10,000 tons per day is considered to be the optimum processing rate, and operations and capital factors were developed from this basis. Recoveries for gold are expected to range from 78% up to 81%, depending on the rock type being mined, and are expected to average 79% for the life of the mine. A range of recovery from 23% to 49%, and averaging 34% over the life of the mine, is expected for silver. Construction of the facilities is projected to begin in the middle of year -2, concluding in the first month of year 1. Operations are expected to commence at approximately 1,000 tonnes per day, ramping up to 10,000 tonnes per day during the first five months, and then remaining at that rate for the succeeding 8 years.

After-Tax cash flows were calculated on a monthly basis for the year of start-up, and on a quarterly basis for the remainder of the life of the mine. Federal, state and local taxes were considered for this evaluation.

Sensitivity Analysis

Price

The Commonwealth Silver and Gold Project, like almost all precious metals projects, is very responsive to changes in the price of its chief commodity, gold. From the base case, a change in the average gold price of US$50/oz Au (assuming that the gold price maintains its current average to the silver price) would change the NPV-5 by 12%, or approximately $12.6 million (Figure 22-1).

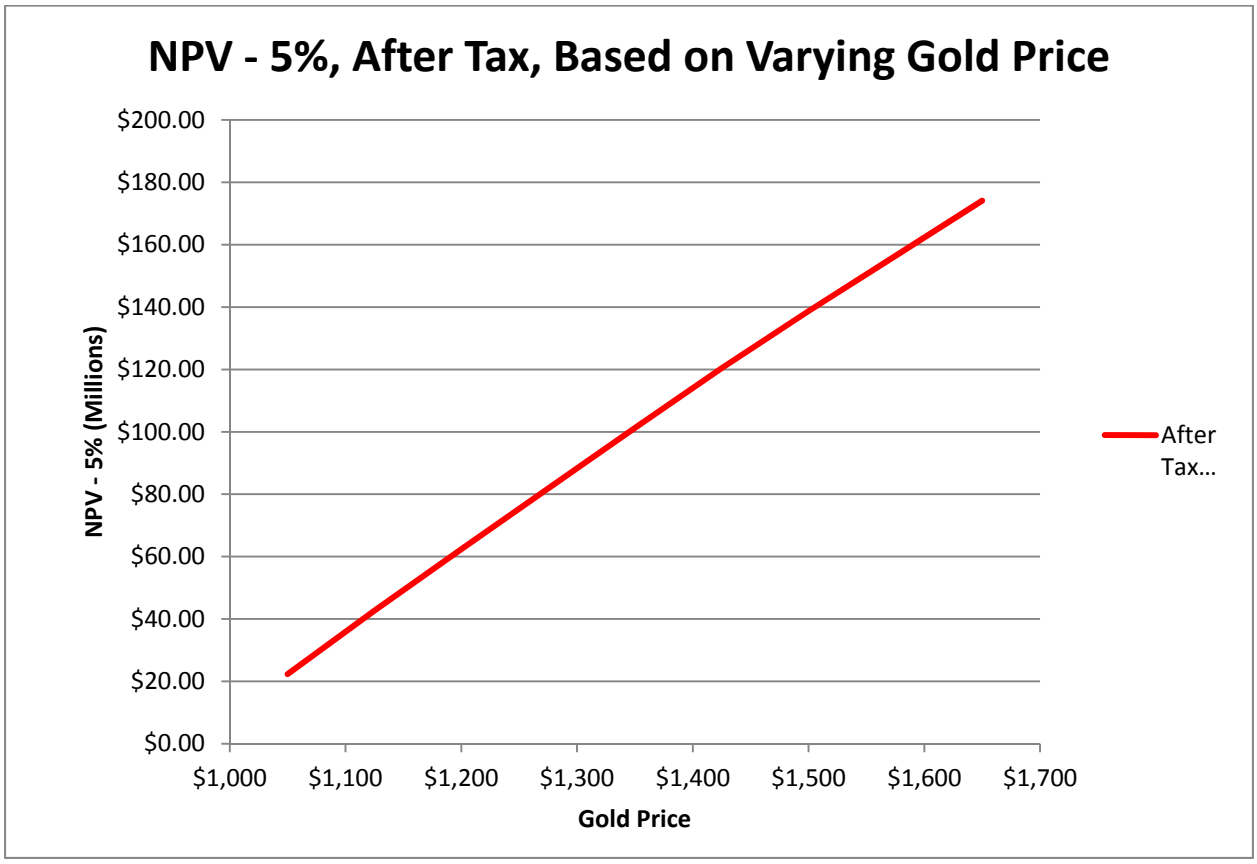


Figure 22-1 Commonwealth Project Gold Price Sensitivity Analysis

Cost and Recovery

The Project is quite sensitive to the cost of operations, incurring an approximately 20% decline in the NPV-5 for each increase of 10% in the operating costs. The Project is less sensitive to variances in the cost of capital, experiencing about 3.0% in decline in the NPV-5 for each increase of 10% in the capital costs, as shown in Figure 22-2.

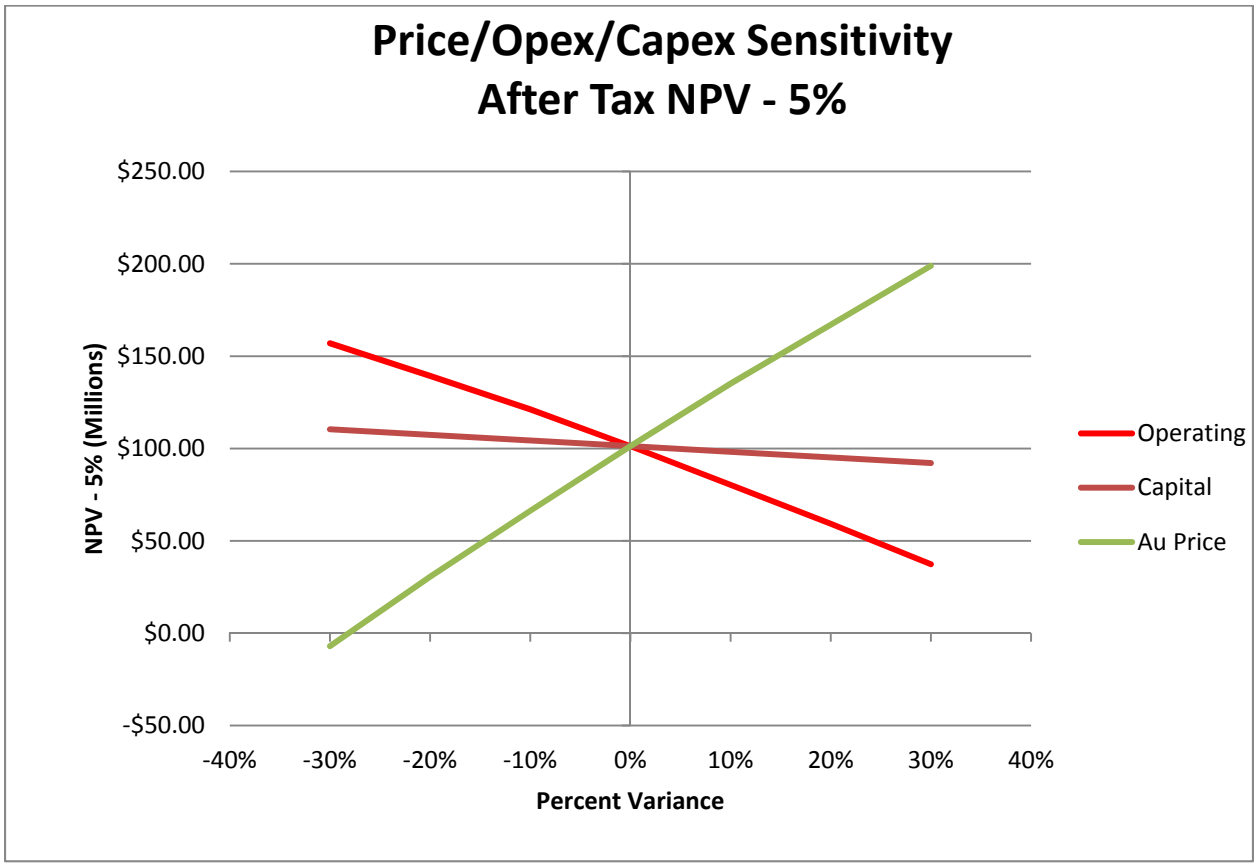


Figure 22-2 Commonwealth Project Operating Cost & Capital Cost Sensitivity Analysis

Conclusion

The Commonwealth Silver and Gold Project would be economically viable based on the parameters considered in this study. The base case scenario produces approximately 312,000 salable ounces of gold and 10.9 million salable ounces of silver over a 9 year period. The Project is most sensitive to the gold price and to operating costs, but not greatly sensitive to capital costs due to the design that a contractor will perform the mining and crushing.

The base case economic analysis of the Project at a gold price of US$1,350/oz shows an After-Tax NPV-5 of $101.3 million using a 10,000 tonne/day crushing/agglomeration/heap leach plant and Merrill-Crowe processing.

23. Adjacent Properties

Immediately adjacent to the Commonwealth Project are the Blue Jeep, San Ignacio and Six Mile Hill properties, also controlled by CSGM. These properties have not been considered in this Technical Report and do not currently have NI 43-101 compliant mineral resource estimates.

Immediately adjacent to the south of the Commonwealth pit are 4 patented claims that belong to a private owner. All facilities have been designed to avoid these claims and they do not materially affect the development of the Commonwealth Project.

HRC knows of no other immediately adjacent properties that might materially affect the Commonwealth Silver and Gold Project.

24. Other Relevant Data and Information

HRC knows of no additional relevant data that might materially impact the interpretations and conclusions presented in this Technical Report.

25. Interpretation and Conclusions

Results

Environmental

The Commonwealth Silver and Gold Project is not subject to any known environmental liabilities. As the area has a long history of mineral exploration, CSGM does not anticipate any barriers to access for work planned going forward. The preliminary mine permit assessment is based on the premise that the proposed Project mine facilities located in Pearce, Cochise County, Arizona, including probable mine features such as a pit, waste dumps, processing facilities and infrastructure are all located on private land. At this time, no federal Environmental Assessment or Impact Statement process is anticipated.

Geology and Deposit Type

HRC is of the opinion that CSGM has thorough understanding of the Project geology and are applying the appropriate deposit model for exploration.

Brockman Fault

This zone is dominated by a structurally controlled gold dominant vein postulated to be the continuance of the North Vein on the western (upthrown) side of the Brockman Fault. Mineralization within this zone occurs primarily within the Bisbee Group sediments, and dips approximately 40 degrees to the South.

Main Zone

This zone was the main focus of historical mining activities and is defined by the two most important veins, the Main Vein and the North Vein. Much of the mineralization on the Commonwealth Project occurs in the silicified and shattered structural wedge between the Main Vein and the North Vein. This mineralized zone is at least 650 m long and expands eastward from a point where the veins coalesce to a well mineralized exposed width of 125 m.

Subsidiary veins in the mineralized wedge that were named by Smith (1927) are, from the footwall of the Main Vein and proceeding north, the Footwall Vein, the Fischer Vein, the Smith Vein, the Hartery Vein and the Renaud Vein. Each of these veins varies from 1 to 4 m in width, with the Renaud Vein being the widest. These veins are generally sheeted veins sub-parallel to the Main Vein. They may also be considered the thick, high fluid flow arteries within the stockwork zone between the Main and North Veins.

Along the footwall of the North Vein cretaceous marine sediments of the Bisbee Group also host mineralization and are chemically favorable hosts for gold. The Bisbee Group sediments are soft enough that they do not fracture well on faulting. Mineralization within the Bisbee Group sediments occurs as both vein type mineralization and some disseminated mineralization.

Only one vein has been identified in the footwall of the North Vein. The Eisenhart Vein strikes N70°W and dips 65 to 75° to the southwest. Smith (1927) recorded that the Eisenhart Vein is emplaced along the footwall of an andesite dike intruding into the Bisbee Group sediments, and can be observed both at ground surface and in underground workings.

Two veins, each less than one meter wide, occur in the hanging wall of the Main Vein.

Exploration, Drilling, and Analytical

HRC is of the opinion that CSGM is conducting exploration activities, drilling, and analytical procedures in manner that meets or exceeds industry best practice.

Quality Assurance/Quality Control

HRC has reviewed CSGM's QA/QC assay programs and believes the programs provide adequate confidence in the data. Samples that are associated with the type 1 and 2 standard failures and the samples associated with erroneous blank samples have been reanalyzed prior to the completion of this Report and the results are acceptable.

All drill cores and cuttings from CSGM's drilling have been photographed. Drill logs have been digitally scanned and archived. The split core and cutting trays have been securely stored and are available for further checks.

Metallurgical

Overall, fine crushing may not be necessary based on the high metal extraction percentages from the bottle roll tests. No significant increase in metal extraction occurred from finely milling any of the composite material.

HRC agrees with CSGM's conclusion that, based on market conditions and comparing extensively tested metallurgical recovery rates associated with a lower capital cost heap leaching scenario to the preliminary results of metallurgical test work associated with a higher capital cost milling scenario, that while mining should remain open pit, the lower costs associated with heap leach processing increases the prospect for economic extraction of the Mineral Resources. HRC concludes that the metallurgical results presented in Table 25-1 are the most appropriate for the CSGM Project.

Table 25-1 Metallurgical Crush and Recovery Recommendations

Rock Type	Crush Size	Recoveries (%)	
		Au	Ag
Rhyolite	Minus 8	78	30
Vein	Minus 8	79	49
Lower Andesite	1/2"	81	33
Upper Andesite	1/2"	78	35
Bisbee	1/2"	80	23

Data Verification

HRC received original assay certificates in pdf format for all samples included in the current drill-hole database. A random manual check of 10% of the database against the original certificates was conducted, focusing on the five primary metals (Au, Ag, Cu, Pb, and Zn), with occasional spot checks of secondary constituents. HRC also conducted a random check of at least 2% of the highest (5%) assay values and continued to randomly spot check assays values throughout the modeling process. HRC is of the opinion that the data maintained within the database is acceptable for mineral resource estimation.

Resource Estimation Data

Appendix C – Drillhole and Underground Channel Sample Information Table, summarizes the data received from CSGM that is pertinent to the estimation of mineral resources at the Commonwealth Silver and Gold Project.

Check Samples

HRC independently collected two quarter-core samples and two channel samples from Level 3 of the underground mine workings for duplicate laboratory analysis. The assays of the selected quarter core and channel samples compare reasonably well to the original assays (Table 12-3). Based on the results of the check sample program, and in conjunction with the results of the database audit, HRC considers the data included in the database to be sound and sufficient for use in estimating the mineral resources at the Commonwealth Project.

Resource

HRC finds that the density of data within the resource base is adequate for the Preliminary Feasibility Study. The mineral resource estimation is appropriate for the geology and assumed open pit mining method. Additional modeling should be conducted to define the alteration of the host rocks to support further metallurgical testing.

Significant Risks and Uncertainties

Metallurgical testing continues to be important to the Project going forward. Additional testing and analysis should be conducted to support the current metallurgical concepts. Improvement of silver recoveries can likely be accomplished, however additional test work should provide concepts for achieving this improvement in recoveries. Investigation of the deleterious constituents retarding the silver recoveries could identify additional metallurgical categories to assist in the modeling of the resource.

26. Recommendations

Metallurgical Study

CSGM should complete a detailed metallurgical study conducted by rock type. This study should be designed to supplement the assumptions and conclusions in this PEA. The additional metallurgical studies should focus on improving silver recovery by the use of additional reagents such as lead nitrate and should develop an additional set of points on the crush size vs recovery curves at -3/8" for all rock types. Bottle roll tests using the additional reagents should be completed before initiating column tests. Additional column tests at -3/8" using the final selection of reagents run for a minimum of 120 days with the goal of simplifying the process flow sheet from the currently envisioned 2 final crush sizes to 1 uniform crush size.

- Gold and Silver Particle Size Analysis, deportment by rock-type;
- Acid-Base Accounting;
- Mineralogical Evaluation using QEMSCAN;
- Bond Abrasion and Bond Impact Tests;
- Bottle Roll Testing focusing on -3/8" crush size
- Agglomeration Testing (-1/2", -3/8 and Minus 8); and
- Column Leach Tests focusing on the -3/8" crush size

HRC recommends that this work be completed on all rock types with an effort made to evaluate changes in mineralogy. Additionally, work should be completed on the basis of elevation as there are reports (Forrest, 1995) that suggest a metallurgical change with depth.

Geotechnical and Hydrogeological Study

HRC Recommends that CSGM complete an updated pit slope study analysis on the available data to evaluate the potential pit slope angles, identify any critical geotechnical areas, and to define a geotechnical exploration program to support the final design of an open pit.

Additionally, HRC recommends that CSGM conduct a preliminary hydrogeological study to support the future Project water needs and to define a critical path process to achieving the water needs for development. The hydrological study should address possible de-watering in the later stages of the pit, test wells in groundwater source areas and the completion of the monitor wells for acquisition of the baseline data for permitting.

Environmental Permitting

HRC recommends that CSGM continue to work towards meeting the requirements of the State of Arizona to permit a mine on private land. This should include in the short term continued work on:

- Cultural Resources Inventory;
- Endangered Species Act (ESA) and other biological requirements; and

- Collection of Environmental Baseline Data including the completion of monitor wells near the point of compliance.
- Complete and submit Aquifer Protection Permit Application to ADEQ.

Exploration Program

Continued exploration diamond core drilling should be targeted in four areas within the immediate mineral resource area:

- Step out drilling in the identified mineralization to the west of the Brockman Fault (4 holes);
- Step out drilling along both the Main and North Veins (5 holes);
- Infill and step drilling along the footwall (Eisenhart Vein) mineralization (4 holes); and
- Step out drilling along the identified mineralization in the hanging wall block of the Main vein (7 holes).

The early open hole rotary/percussion holes drilled in the late 1970's and early 1980's by Bethex (P-76-6 to P-76-14) and Platoro (CS-1 – CS-5) should be twinned with diamond core holes and replaced in the database.

Underground surveys of the existing mine workings including 3D laser scanning where appropriate should be completed to improve the location of the workings in the project model to a level sufficient for final mine planning.

Additionally, leach pad and waste dump condemnation reverse circulation drilling should be conducted and historical barren holes (GH series holes) should have their collars surveyed and their data incorporated into the Project data base.

HRC recommends that CSGM continue to insert the high grade and low grade standards, duplicates and blanks in ensuing drilling and sampling programs.

Budget

HRC's recommendations are intended to provide CSGM a path toward the development of the project and advancing the project to PFS level study is not contingent upon positive results from outlined work program. The engineering, permitting and environmental requirements necessary to bring a mine into development need to be assessed to understand any difficulties or costs that will impact the overall project economics at the PFS level. The anticipated costs for the recommended scope of work are presented in Table 26-1.

Table 26-1 Budget Based on Recommendations in 2013 Technical Report

Recommended Scope of Work	Expected Cost (US$)	
Preliminary Feasibility Study	$175,000*	$175,000*
Metallurgical Studies		
Qemscan Mineralogy	25,000	
Physical Testing (Impact and Abrasion)	10,000	
Column Testing (20 columns 120 days)	150,000	
Agglomeration and Percolation Testing	20,000	
Acid-Base Accounting	10,000	
Effluent analysis from column testing	15,000	
Further Bottle-Roll testing using additional reagents	50,000	280,000
Geotechnical and Hydrological Studies		
Geotechnical Report	5,000	
Drill 3 Monitor Wells	90,000	
Hydrological Report	5,000	
Drill 3 Water Source Identification Wells	90,000	190,000
Environmental Permitting		
Phase I and Phase III Cultural Resource Surveys	30,000	
Arizona Native Plant survey and update of ESA opinion	25,000	
Phase II Aquifer Protection Permit contractor fees	180,000	
Fees to ADEQ for APP application	200,000	435,000
Exploration		
Infill and step-out Drilling 3,000 meters (core)	750,000	
Condemnation Drilling 2,500 meters (reverse circulation)	240,000	
Percussion hole Re-Drilling 1500 m (reverse circulation)	180,000	1,170,000
Total Technical Budget for publication in NI 43-101 Technical Report	$2,250,000	$2,250,000

*Including pit and haul road design, preliminary design of heaps, ponds and waste dump, and proposed process flow sheet

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******All referenced unpublished reports are in the possession of Commonwealth Silver and Gold Corp., a wholly-owned subsidiary of Commonwealth Silver and Gold Mining Inc.***

28. Glossary

Mineral Resources

The mineral resources and mineral reserves have been classified in accordance with standards as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards - For Mineral Resources and Mineral Reserves" prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 17, 2010. Accordingly, the Resources have been classified as Measured, Indicated or Inferred and the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below. The Commonwealth Project has Mineral Resource estimates but there not currently any Mineral Reserve estimates.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill-holes that are spaced closely enough to confirm both geological and grade continuity.

Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

The following general mining terms may be used in this report.

Table 28-1 Definition of Terms

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure:	All other expenditures not classified as operating costs.
Composite:	Combining more than one sample assay to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the mineralized material.
Crushing:	Initial process of reducing mineralized material particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold equivalent content by further concentration.
Dike:	A tabular body of igneous rock that cuts across the structure of adjacent rocks or cuts massive rocks.
Dilution:	Waste, which is unavoidably mined with mineralized material.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Felsic	A rock type that is composed predominantly of silicic and potassic rock-forming silicate minerals. Contrasted with mafic.

Term	Definition
Footwall:	The underlying side of a fault, orebody, or stope.
Gangue:	Non-valuable components of the mineralized material.
Grade:	The measure of concentration of gold, silver and gold equivalent within mineralized rock.
Hanging wall:	The overlying side of a fault, orebody, or slope.
Haulage:	A horizontal underground excavation which is used to transport mined mineralized material.
Hydrocyclone:	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous:	Primary crystalline rock formed by the solidification of magma or lava.
Kriging:	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
LoM Plans:	Life-of-Mine plans.
LRP:	Long Range Plan.
Material Properties:	Mine properties.
Mafic:	A rock type that is composed predominantly of magnesian rock-forming silicate minerals; subsilicic. Contrasted with felsic.
Milling:	A general term used to describe the process in which the mineralized material is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease:	A lease area for which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
Ongoing Capital:	Capital estimates of a routine nature, which is necessary for sustaining operations.
Pillar:	Rock left behind to help support the excavations in an underground mine.
RoM:	Run-of-Mine.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft:	An opening cut downwards from the surface for transporting personnel, equipment, supplies, mineralized material and waste.
Sill:	An intrusive body of relatively thin and tabular igneous rock which has been emplaced parallel to the bedding or schistosity of the intruded unit
Smelting:	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope:	Underground void created by mining.

Term	Definition
Stratigraphy:	The study of stratified rocks in terms of time and space.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide:	A metallic, sulfur bearing mineral.
Tailings:	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening:	The process of concentrating solid particles in suspension.
Total Expenditure:	All expenditures including those of an operating and capital nature.
Variogram:	A statistical representation of the characteristics (usually grade).

The following abbreviations may be used in this report.

Table 28-2 Abbreviations

Abbreviation	Unit or Term
A	ampere
AA	atomic absorption
A/m^2	amperes per square meter
ANFO	ammonium nitrate fuel oil
Ag	Silver
APMA	Annual Placer Mining Application
Au	gold
AuEq	gold equivalent
As	arsenic
AST	above ground storage tank
BSE	backscatter electron
°C	degrees Centigrade
CCD	counter-current decantation
CIL	carbon-in-leach
CoG	cut-off grade
Cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
Cfm	cubic feet per minute
ConfC	confidence code

Abbreviation	Unit or Term
CRec	core recovery
CSS	closed-side setting
CTW	calculated true width
Cu	copper
°	degree (degrees)
dia.	Diameter
EDX	energy dispersive x-ray
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
FA	fire assay
Ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
G	gram
Gal	gallon
g/L	gram per liter
g-mol	gram-mole
gpm	gallons per minute
g/t	grams per tonne, equivalent to 1 part per million
ha	hectares
HDPE	Height Density Polyethylene
Hp	Horsepower
HQ	drill core diameter of ~63.5 mm
HTW	horizontal true width
ICP-AES	inductively coupled plasma atomic emission spectroscopy
ICP-MS	inductively coupled plasma mass spectrometry
ID2	inverse-distance squared
ID2.5	inverse-distance 2.5
ILS	Intermediate Leach Solution
ITH	International Tower Hill
kA	kiloamperes
kg	kilograms
km	kilometer
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter

Abbreviation	Unit or Term
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	Long-Haul Dump truck
LLDDP	Linear Low Density Polyethylene Plastic
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
Masl	meters above sea level
Ma	millions of years before present
MDA	Mine Development Associates
mg/L	milligrams/liter
MLA	mineral liberation analysis
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering
Moz	million troy ounces
Mt	million tonnes
MTW	measured true width
MW	million watts
mya	million years ago
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NQ	drill core diameter of ~47.5 mm
opt	troy ounce per ton
OSC	Ontario Securities Commission
oz	troy ounce
%	Percent
Pb	lead
PGM	Pilot Gold Mill
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million, equivalent to 1 gram / tonne
QA/QC	Quality Assurance/Quality Control
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
Sb	antimony

Abbreviation	Unit or Term
SEC	U.S. Securities & Exchange Commission
sec	second
SEM	Scanning Electron Microscope
SG	specific gravity
SPCC	Spill Prevention, Control, and Countermeasure
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
t/d	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
µm	micron or microns
V	volts
VFD	variable frequency drive
W	tungsten
XRD	x-ray diffraction
XRF	x-ray fluorescence
Y	Year
Zn	zinc

Appendices

Appendix A: Land Status

Table A-1: CSGM Fee Land, Patented Claims

CSGM Fee (Patented) Lands as of September 5, 2013				
Commonwealth Owned Fee Lands				
Parcel/Town Lots	**Description**	**Surface and/or Minerals**	**Gross Acres**	**Note**
Purchased by CSGM from J-Rod LLC PIN#11317003	A portion of the amended map of the Townsite of Pearce located in the NW1/4 of Section 5, Township 18 South, Range 25 East, G&SBM, Cochise County, AZ (see deed for meters and bounds description)	Surface and Mineral rights	17.34	CSGM Interest 100%
Lots 1-12, inclusive, Block 17 Pearce Townsite PIN#11323004	Lots 1-12, inclusive, Block 17, a portion of abandoned alley in said Block 17, and a portion of abandoned Walnut Street, all in Pearce Townsite (see Commowealth/ Thetford Mining Lease and Option to Purchase Agreement, dated January 25, 2011, for detailed property description)	Surface and Mineral Rights	2.133	Acquired from John C.S. Breitner CSGM Interest 10%
Lots 32-41, inclusive, Block 17 Pearce Townsite PIN#11323007	Lots 32-41, inclusive, Block 17, a portion of abandoned alley in said Block 17, and a portion of abandoned Fifth Street, all in Pearce Townsite (see Commonwealth/Thetford Mining Lease and Option to Purchase Agreeement, dated January 25, 2011, for detailed property description)	Surface and Mineral Rights		Acquired from John C.S. Breitner CSGM Interest 10%
Patented Claim Name	**Patent #**	**Mineral Survey Number**	**Gross Acres**	**Note**
Sulphur Springs Valley	35979	1391	142.02 Acquired from John C.S. Breitner	CSGM Interest 10%
Silver Wave Lode	29026	1249A		
North Bell Lode	29026	1249A		
Common Wealth Lode	29026	1249A		
Silver Crown Lode	29026	1249A		
One and All Lode	29026	1249A		
Ocean Wave	29026	1249A		
One and All Millsite	29026	1249B		
PIN#'s 60607001A and				
60607001B				

Note: Patented Claims include surface and mineral ownership.

CSGM fee lands held by third party Agreement				
Parcel/Town Lots	**Description**	**Surface and/or Minerals**	**Gross Acres**	**Owner(s)**
Lots 1-12, inclusive, Block 17 Pearce Townsite PIN#11323004	Lots 1-12, inclusive, Block 17, a portion of abandoned alley in said Block 17, and a portion of abandoned Walnut Street, all in Pearce Townsite (see Commowealth/Thetford Mining Lease and Option to Purchase Agreement, dated January 25, 2011, for detailed property description)	Surface and Mineral Rights	2.133	Carl Thetford Family Trust, M. Thetford and Spira Family Trust 88% Ownership Interest
Lots 32-41, inclusive, Block 17 Pearce Townsite PIN#11323007	Lots 32-41, inclusive, Block 17, a portion of abandoned alley in said Block 17, and a portion of abandoned Fifth Street, all in Pearce Townsite (see Commonwealth/Thetford Mining Lease and Option to Purchase Agreeement, dated January 25, 2011, for detailed property description)	Surface and Mineral Rights		Carl Thetford Family Trust, M. Thetford and Spira Family Trust 88% Ownership Interest
Patented Claim Name	**Patent #**	**Mineral Survey Number**	**Gross Acres**	**Owner**
Sulphur Springs Valley	35979	1391	142.02	Carl Thetford Family Trust, M. Thetford and Spira Family Trust 88% Ownership Interest
Silver Wave Lode	29026	1249A		
North Bell Lode	29026	1249A		
Common Wealth Lode	29026	1249A		
Silver Crown Lode	29026	1249A		
One and All Lode	29026	1249A		
Ocean Wave	29026	1249A		
One and All Millsite	none	1249B		
PIN#'s 60607001A and				
60607001B				

Note: Patented Claims include surface and mineral ownership. See CSGM/Thetford Mining Lease and Option to Purchase Agreement, dated January 25, 2011, for detailed property description.

Table A-2: Unpatented Claims

CSGM Properties as of September 5, 2013					
Unpatented Mining Claims - Total 133 claims					
CSGM owned upatented mining claims					
Claim Group Name	**AMC Numbers**	**County Recording Information**		**Interest**	**Acres**
J-Rod-1 through J-Rod-10	404924-404933	2010-24861 thru 2010-24870		100%	J-Rod Grp
J-Rod-11	394700	2008-24088		100%	154
J-Rod-12	394698	2008-24086		100%	
CWSG #1-CWSG #35	405744-405778	2011-04289 thru 2011-04323		100%	CWSG Grp
CWSG #38-CWSG #39	405779-405780	2011-04324 thru2011-04325		100%	1215
CWSG #50-CWSG #59	405781-405790	2011-04326 thru 2011-04335		100%	
CWSG#66-CWSG #95	405791-405820	2011-04336 thru 2011-04365		100%	
CSGM unpatented mining claims held by a third party Agreement.					
Claim Group Name	**AMC Numbers**	**County Recording Information**	**Owner**	**Interest**	**Acres**
Lyle 1-2	364081-364082	2005-00377 thru 2005-00378	Carl Thetford Family Trust, M. Thetford and Spira Family Trust	100%	Pan Grp
Pan 8-15	364083-364090	2005-00379 thru 2005-00386	Carl Thetford Family Trust, M. Thetford and Spira Family Trust	100%	83
Brindle Steer	67975	Bk 119 Pg 414	Cartmell's	100%	Blue Jeep
Blue Jeep	67976	Bk 119 Pg 366	Cartmell's	100%	Grp 183
Blue Jeep #2-Blue Jeep #4	67977-67979	Bk 386 Pages 195-97	Cartmell's	100%	
Blue Jeep #5	67980	Bk 817 Pg 300	Cartmell's	100%	
Blue Jeep #6	67981	Bk 817 Pg 303	Cartmell's	100%	
Blue Jeep #7	67982	Bk 817 Pg 306	Cartmell's	100%	
Blue Jeep #8	67983	Bk 817 Pg 309	Cartmell's	100%	
Blue Jeep #9	103278	Bk 1418 Pg 176	Cartmell's	100%	
San Ignacio #1-18	75620-75637	Bk 983 Pg 239 thru Bk 983 Pg 256	Cartmell's	100%	355
San Ramon #1-6	253698-253703	86-0510172 thru 86-0510177	Cartmell's	100%	124
				Total	2,114

Note: County recording information for mining claim amendments is not included in this list which is limited to county recording information for original location notices.

TABLE A-3: State of Arizona Mineral Exploration Permits

CSGM State of Arizona Mineral Exploration Permits as of September 5, 2013					
Permit Number	**Effective Date**	**Description**	**Surface and/or Minerals**	**Gross Acres**	**Interest**
08-115457	8-Apr-11 5 year term	T18S - R 25 E, Section 2 Lot 1; SE1/NE1/4; E1/2SE1/4 Lot 4; SW1/4NW1/4; and W1/2SW1/4	Minerals with surface access	320.00	100%
08-115458	8-Apr-11 5 year term	T18S - R 25 E, Section 3 Lots 2, 3, and 4 Lot 1; S1/2N1/2; S1/2	Minerals with surface access	639.44	100%
08-115844	20-Oct-11 5 year term	T17S - R 25 E, Section 36 All	Minerals with surface access	640.00	100%
				1,599.44	Total

Appendix B: Royalties, Agreements and Encumbrances

CSGM - Thetford Agreement Summary

AGREEMENT

NAME: Carl Thetford Family Trust et al Mining Lease and Option To Purchase Agreement

PROJECT: Commonwealth

RECORDER: Cochise County, Arizona

FILE NO:

OWNER: Carl Thetford Family Trust
c/o Vicky Carol Klekar
22219 Cimarron Parkway
Katy, TX 77450
Telephone number: 281-414-7360 vklekar@msn.com

Mordecai Thetford
c/o The Lightship Group
8249 Parkline Blvd., Suite 200
Orlando, FL 32809

Fred Spira and Marva Spira
Assigned June 20, 2013 to:
The Spira Family Living Trust
c/o Frederick A. Spira
2712 North Cloverland Avenue
Tucson, AZ 85712

COMPANY: Commonwealth Silver and Gold Corp., an Arizona corporation
5210 E. Williams Circle, Suite 730
Tucson, AZ 85711
Telephone: 520-790-1909

cc: Commonwealth Silver and Gold Mining Inc.
10 King Street East, Suite 801
Toronto, ON, Canada M5C 1C3
Attention: Michael Farrant

DATE: January 25, 2011

TERM: 5 years

PROPERTY:

List of Patented Claims

Claim Name	Patent No.	Mineral Survey Number
Sulphur Springs Valley	35979	1391
Silver Wave Lode	29026	1249A
North Bell Lode	29026	1249A
Common Wealth Lode	29026	1249A
Silver Crown Lode	29026	1249A
One and All Lode	29026	1249A
Ocean Wave	29026	1249A
One and All Millsite	None	1249B

List of Unpatented claims

Lyle #1-2, AMC 364081-364082

Pan #8-15, AMC 364083-364090

MINIMUM PAYMENTS:

Due Date of Payment	**Amount (US$)**
On signing letter of intent:	$10,000
Date of Execution:	$40,000
July 25, 2011	$50,000
January 25, 2012	$50,000
July 25, 2012	$100,000
January 25, 2013	$100,000
July 25, 2013	$100,000
January 25, 2014	$200,000
July 25, 2014	$200,000
January 25, 2015	$200,000
July 25, 2015	$200,000
January 25, 2016	$3,250,000

The minimum payments shall be credited against the Purchase Price on CSGM's exercise of the Option and shall be advance payments of the royalty payable by CSGM on the commencement of commercial production of minerals from the property. **Note: The minimum payments are to be divided among three parties as set forth in Exhibit C of the Thetford Agreement.**

CSGM may not mine until it exercises the option except for tests and samples, including bulk samples up to 10,000 tonnes.

PURCHASE PRICE: US$4,500,000, less minimum payments previously made to Owner by CSGM

WORK COMMITMENTS: None

ASSIGNMENT BY CSGM: Assignable with Owner's consent which Owner may not delay or withhold unreasonably. If Owner does not respond to CSGM's request for consent to assignment within 10 business days following Owner's receipt of the request, Owner shall have deemed to have consented to CSGM's assignment of the Agreement.

ASSIGNMENT BY OWNER: Freely assignable

TERMINATION BY CSGM: Freely terminable with 30 days notice. CSGM must provide Owner a notice of termination of the Agreement in a form acceptable for recording.

PRODUCTION ROYALTY: 2% NSR for the unpatented claims and 2% of eighty-eight percent (88%) for the patented claims. CSGM may buy down the royalty to 1% by paying Owner US$2,000,000 in increments of US$1,000,000 per one-half of one percent (0.5%) of the NSR. In any event, CSGM shall not be obligated to pay any royalty payments to Owner until the royalty otherwise payable to Owner exceeds US$4,500,000 which is the purchase price for CSGM's purchase of the property. **Note: The US$2,000,000 is to be divided among three parties as set forth in Exhibit C of the Thetford Agreement.**

ANNUAL SUMMARY REPORT: Summary report due on or before March 1st of each lease year.

DATA: If CSGM does not exercise the option to purchase the Property, CSGM must deliver all data, except interpretative data, within 90 days of termination of this Agreement.

AREA OF INTEREST: N/A

PROPERTY TAXES: CSGM to reimburse or pay Owner for any real property taxes assessed against the property.

ASSESSMENT WORK AND/OR MAINTENANCE FEES: Beginning with the assessment work period of September 1, 2011 to August 31, 2012, assessment work to be performed and filed by CSGM unless terminated more than 2 months before the deadline for performance of assessment work for the succeeding annual assessment year.

Beginning with the assessment work period of September 1, 2011 to August 31, 2012, maintenance fees to be paid and filed by CSGM unless terminated more than 2 months before the deadline for payment of the federal annual mining claim maintenance fees the succeeding annual assessment year. **CSGM must provide proof of compliance to Owner by August 15 prior to the succeeding annual assessment year.**

NOTICE OF NON-RESPONSIBILITY: Before commencement of activities on the Property, CSGM shall record, post and maintain "no lien" notices in accordance with Arizona Revised Statutes 33-990. **Done.**

SURRENDER OF PROPERTY: If CSGM does not exercise the option to purchase, upon the expiration or termination of this Agreement, CSGM must fence or secure all shafts, pits and other excavations on the Property, whether or not created by CSGM, and post warning signs at such excavations as required by Arizona Revised Statutes 27-318.

COMMENTS: Dated September 5, 2013

CSGM - Cartmell Agreement Summary

AGREEMENT

NAME: Ralph M. Cartmell et al Mining Lease and Option To Purchase Agreement

PROJECT: Commonwealth

RECORDER: Cochise County, Arizona

FILE NO:

OWNER: **Ralph M. Cartmell, Vivian M. Cartmell, Martha E. Cartmell and David W. Cartmell**
c/o Ralph M. Cartmell
PO Box 146
Pearce, AZ 85625
Telephone number: 520-826-3564

COMPANY: Commonwealth Silver and Gold Corp., an Arizona corporation
5210 E. Williams Circle, Suite 730
Tucson, AZ 85711
Telephone: 520-790-1909

cc: Commonwealth Silver and Gold Mining Inc.
10 King Street East, Suite 801
Toronto, ON, Canada M5C 1C3
Attention: Michael Farrant

DATE: January 25, 2011

TERM: 5 years

PROPERTY:

Thirty Four (34) Unpatented claims

San Ignacio #1-18, AMC 75620-75637
Brindle Steer, Blue Jeep and Blue Jeep #2-8, AMC 67975-67983
Blue Jeep #9, AMC 103278
San Ramon #1-6, AMC 253698-253703

MINIMUM PAYMENTS:

Due Date of Payment	**Amount (US$)**
On signing 12-7-2010 letter of intent:	$10,000
Date of Execution:	$140,000
July 25, 2011	$50,000
January 25, 2012	$50,000
July 25, 2012	$50,000
January 25, 2013	$50,000
July 25, 2013	$50,000
January 25, 2014	$50,000
July 25, 2014	$100,000
January 25, 2015	$100,000
July 25, 2015	$100,000
January 25, 2016	$1,250,000

The minimum payments shall be credited against the Purchase Price on CSGM's exercise of the Option and shall be advance payments of the royalty payable by CSGM on the commencement of commercial production of minerals from the property. **Note: The minimum payments are to be divided among six parties as set forth in Exhibit C of the Cartmell Agreement.**
CSGM may not mine until it exercises the option except for tests and samples, including bulk samples up to 10,000 tonnes.

PURCHASE PRICE: US$2,000,000, less minimum payments previously made to Owner by CSGM

WORK COMMITMENTS: None

ASSIGNMENT BY OPTIONOR: Freely assignable

ASSIGNMENT BY CSGM: Freely assignable, provided that the transferee agrees in writing to assume CSGM's obligations.

TERMINATION BY CSGM: Freely terminable with 30 days notice. Within 10 days after the effective date of termination, CSGM must provide Owner a notice of termination of the Agreement in a form acceptable for recording.

PRODUCTION ROYALTY: 2% NSR, CSGM may buy down the royalty to 1% by paying Owner US$1,000,000. In any event, CSGM shall not be obligated to pay any royalty payments to Owner until the royalty otherwise payable to Owner exceeds US$2,000,000 which is the purchase price for CSGM's purchase of the property. **Note: The US$1,000,000 is to be divided among six parties as set forth in Exhibit C of the Cartmell Agreement.**

ANNUAL SUMMARY REPORT: Summary report due on or before March 1st of each lease year.

AREA OF INTEREST: N/A

PROPERTY TAXES: CSGM to reimburse or pay Cartmell for any real property taxes assessed against the property.

ASSESSMENT WORK AND/OR MAINTENANCE FEES: Beginning with the assessment work period of September 1, 2011 to August 31, 2012, assessment work to be performed and filed by CSGM unless terminated more than 2 months before the deadline for performance of assessment work for the succeeding annual assessment year.

Beginning with the assessment work period of September 1, 2011 to August 31, 2012, maintenance fees to be paid and filed by CSGM unless terminated more than 2 months before the deadline for payment of the federal annual mining claim maintenance fees the succeeding annual assessment year.

DATA: CSGM to provide **to** provide data to Owner within 30 days of termination.

COMMENTS: Dated September 5, 2013

Appendix C: Drillhole and Underground Channel Sample Information Table

Hole_ID	TD_M	DH_Type	Exploration Company	Surveyor	Survey Type	Assay Certificate	Geology	Downhole Survey
CS-1	82.3	RC	Platoro	Bailey	Total_Station	X	X	
CS-2	48.77	RC	Platoro	Darling	DGPS	X	X	
CS-3	82.3	RC	Platoro	Bailey	Total_Station	X	X	
CS-4	60.96	RC	Platoro	Bailey	Total_Station	X	X	
CS-5	76.2	RC	Platoro	Darling	DGPS	X	X	
DD-76.1	40.84	CORE	Bethex	Bailey	Total_Station		X	Vertical
DD-76.2	45.72	CORE	Bethex	Darling	DGPS		X	Vertical
DD-76.3	71.63	CORE	Bethex	Darling	DGPS		X	Vertical
P-76.6	198.1	RC	Bethex	Darling	DGPS		X	
P-76.7	152.4	RC	Bethex	Darling	DGPS		X	
P-76.8	88.39	RC	Bethex	Darling	DGPS		X	
P-76.9	78.33	RC	Bethex	Darling	DGPS		X	
P-76.10	152.4	RC	Bethex	Darling	DGPS		X	
P-76.11	152.4	RC	Bethex	Darling	DGPS		X	
P-76.13	137.2	RC	Bethex	Darling	DGPS		X	
P-76.14	137.2	RC	Bethex	Darling	DGPS		X	
WC.1A	115.8	RC	Western States	Darling	DGPS	X	X	
WC.2	30.48	RC	Western States	Darling	DGPS	X	X	
WC.3	91.44	RC	Western States	Bailey	Total_Station	X	X	
WC.4B	121.9	RC	Western States	Bailey	Total_Station	X	X	
WC.5	41.15	RC	Western States	Darling	DGPS	X	X	
WC.6	33.53	RC	Western States	Bailey	Total_Station	X	X	
WC.6B	22.86	RC	Western States	Bailey	Total_Station	X	X	
WC.6C	7.62	RC	Western States	Bailey	Total_Station	X	X	
WC.6D	112.8	RC	Western States	Bailey	Total_Station	X	X	
WC.7	121.9	RC	Western States	Bailey	Total_Station	X	X	
WC.9	120.4	RC	Western States	Bailey	Total_Station	X	X	
WC.10	121.9	RC	Bethex	Bailey	Total_Station	X	X	
WC.11B	105.2	RC	Western States	Bailey	Total_Station	X	X	
WC.12B	121.9	RC	Western States	Bailey	Total_Station	X	X	
WC.14	141.7	RC	Western States	Bailey	Total_Station		X	
WC.15	131.1	RC	Western States	Bailey	Total_Station	X	X	
CM-1	121.9	RC	Santa Fe Pacific	Darling	DGPS	X	X	Vertical
CM-2	91.44	RC	Santa Fe Pacific	Darling	DGPS	X	X	Vertical
CM-3	121.9	RC	Santa Fe Pacific	Darling	DGPS	X	X	Vertical
CM-4	121.9	RC	Santa Fe Pacific	Darling	DGPS	X	X	
CM-5	173.7	RC	Santa Fe Pacific	Darling	DGPS	X	X	Vertical
CM-6	121.9	RC	Santa Fe Pacific	Darling	DGPS	X	X	
W-001	62.48	RC	Westland Exploration	Darling	DGPS	X	X	
W-001A	117.4	RC	Westland Exploration	Bailey	Total_Station	X	X	
W-002A	105.2	RC	Westland Exploration	Darling	DGPS	X	X	
W-003	86.87	RC	Westland Exploration	Darling	DGPS	X	X	
W-004	121.9	RC	Westland Exploration	Darling	DGPS	X	X	
W-005	99.06	RC	Westland Exploration	Darling	DGPS	X	X	
W-007	141.7	RC	Westland Exploration	Darling	DGPS	X	X	
W-008	123.4	RC	Westland Exploration	Bailey	Total_Station	X	X	
W-011	91.44	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-012A	64.01	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-013	74.68	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-014	73.15	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-015	91.44	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-016A	27.43	RC	Westland Exploration	Darling	DGPS	X	X	Vertical

Hole_ID	TD_M	DH_Type	Exploration Company	Surveyor	Survey Type	Assay Certificate	Geology	Downhole Survey
W-016B	91.44	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-017A	28.96	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-017B	45.72	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-018A	48.77	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-019	91.44	RC	Westland Exploration	Darling	DGPS	X	X	Vertical
W-020A	91.44	RC	Westland Exploration	Darling	DGPS	X	X	Vertical
W-020B	60.96	RC	Westland Exploration	Bailey	Total_Station	X	X	
W-021A	18.29	RC	Westland Exploration	Darling	DGPS	X	X	Vertical
W-021B	91.44	RC	Westland Exploration	Darling	DGPS	X	X	
W-022A	30.48	RC	Westland Exploration	Bailey	Total_Station	X	X	Vertical
W-022B	97.54	RC	Westland Exploration	Bailey	Total_Station	X	X	
W-034	96.01	RC	Westland Exploration	Darling	DGPS		X	
W-035	112.8	RC	Westland Exploration	Darling	DGPS		X	
W-036	91.44	RC	Westland Exploration	Darling	DGPS	X	X	
W-037	121.9	RC	Westland Exploration	Darling	DGPS		X	
W-038	91.44	RC	Westland Exploration	Darling	DGPS		X	
W-039	121.9	RC	Westland Exploration	Darling	DGPS		X	
W-040	106.7	RC	Westland Exploration	Darling	DGPS	X	X	
W-041	67.06	RC	Westland Exploration	Bailey	Total_Station		X	
W-042	152.4	RC	Westland Exploration	Darling	DGPS		X	
W-043	150.9	RC	Westland Exploration	Darling	DGPS		X	
W-044	91.44	RC	Westland Exploration	Darling	DGPS		X	
W-045	53.34	RC	Westland Exploration	Bailey	Total_Station		X	
W-046	152.4	RC	Westland Exploration	Darling	DGPS		X	
W-047	45.72	RC	Westland Exploration	Darling	DGPS		X	
W-048	76.2	RC	Westland Exploration	Darling	DGPS		X	
W-049	76.2	RC	Westland Exploration	Darling	DGPS		X	
W-050	91.44	RC	Westland Exploration	Bailey	Total_Station		X	
W-051	91.44	RC	Westland Exploration	Darling	DGPS		X	
W-052	91.44	RC	Westland Exploration	Darling	DGPS		X	
W-054	121.9	RC	Westland Exploration	Darling	DGPS		X	Vertical
W-055	152.4	RC	Westland Exploration	Bailey	Total_Station		X	Vertical
W-056	152.4	RC	Westland Exploration	Darling	DGPS		X	
E-1	50.29	RC	Westland Exploration	Darling	DGPS	X	X	
E-1B	68.58	RC	Westland Exploration	Bailey	Total_Station	X	X	
E-2	99.06	RC	Westland Exploration	Darling	DGPS	X	X	
E-3	123.4	RC	Westland Exploration	Bailey	Total_Station	X	X	
E-9	117.4	RC	Westland Exploration	Darling	DGPS	X	X	
E-14	105.2	RC	Westland Exploration	Bailey	Total_Station	X	X	
E-15	56.39	RC	Westland Exploration	Darling	DGPS	X	X	
GG-1	128	RC	Glamis Gold	Darling	DGPS	X	X	Vertical
GG-2	91.44	RC	Glamis Gold	Darling	DGPS	X	X	
GG-3	85.34	RC	Glamis Gold	Darling	DGPS	X	X	
GG-4	115.8	RC	Glamis Gold	Bailey	Total_Station	X	X	Vertical
GG-5	115.8	RC	Glamis Gold	Darling	DGPS	X	X	Vertical
GG-6	100.6	RC	Glamis Gold	Darling	DGPS	X	X	Vertical
GG-7	103.6	RC	Glamis Gold	Darling	DGPS	X	X	Vertical
GG-8	97.54	RC	Glamis Gold	Bailey	Total_Station	X	X	Vertical
GG-9	103.6	RC	Glamis Gold	Darling	DGPS	X	X	Vertical
GG-10	128	RC	Glamis Gold	Bailey	Total_Station	X	X	
GG-11	121.9	RC	Glamis Gold	Darling	DGPS	X	X	Vertical
GG-12	91.44	RC	Glamis Gold	Bailey	Total_Station	X	X	Vertical

Hole_ID	TD_M	DH_Type	Exploration Company	Surveyor	Survey Type	Assay Certificate	Geology	Downhole Survey
GG-13	67.06	RC	Glamis Gold	Bailey	Total_Station	X	X	Vertical
PHR-001	65.53	RC	Western States	Bailey	Total_Station	X	X	
PHR-002	60.96	RC	Western States	Bailey	Total_Station	X	X	Vertical
PHR-003	59.44	RC	Western States	Darling	DGPS	X	X	Vertical
PHR-004	109.7	RC	Western States	Darling	DGPS	X	X	Vertical
PHR-005	91.44	RC	Western States	Darling	DGPS	X	X	Vertical
PHR-006	169.2	RC	Western States	Bailey	Total_Station	X	X	Vertical
PHR-007	158.5	RC	Western States	Darling	DGPS	X	X	Vertical
PHR-008	115.8	RC	Western States	Darling	DGPS	X	X	Vertical
PHR-009	115.8	RC	Western States	Bailey	Total_Station	X	X	Vertical
PHR-010	60.96	RC	Western States	Darling	DGPS	X	X	Vertical
PHR-011	73.15	RC	Western States	Bailey	Total_Station	X	X	Vertical
PCW-1	185.9	RC	Pegasus Gold	Bailey	Total_Station	X	X	X
PCW-2	182.9	RC	Pegasus Gold	Bailey	Total_Station	X	X	X
PCW-3	231.7	RC	Pegasus Gold	Darling	DGPS	X	X	X
PCW-4	213.4	RC	Pegasus Gold	Bailey	Total_Station	X	X	X
PCW-5	167.6	RC	Pegasus Gold	Bailey	Total_Station	X	X	X
C-94-1	123.4	RC	Harvest Gold	Darling	DGPS	X	X	Vertical
C-94-2	106.7	RC	Harvest Gold	Darling	DGPS	X	X	Vertical
C-94-3	150.9	RC	Harvest Gold	Bailey	Total_Station	X	X	Vertical
C-94-4	141.7	RC	Harvest Gold	Bailey	Total_Station	X	X	Vertical
122-1	121.9	RC	Atlas Corp	Darling	DGPS	X	X	
122-2	76.2	RC	Atlas Corp	Darling	DGPS	X	X	
122-3	106.7	RC	Atlas Corp	Darling	DGPS	X	X	
122-4	128	RC	Atlas Corp	Darling	DGPS	X	X	X
122-5	80.77	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-6	79.25	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-7	103.6	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-8	76.2	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-9	102.1	RC	Atlas Corp	Bailey	Total_Station	X	X	Vertical
122-10	140.2	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-11	15.24	RC	Atlas Corp	Bailey	Total_Station	X	X	Vertical
122-12	115.8	RC	Atlas Corp	Bailey	Total_Station	X	X	
122-13	83.82	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-14	89.92	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-15	67.06	RC	Atlas Corp	Bailey	Total_Station	X	X	Vertical
122-16	99.06	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-17	76.2	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-18	60.96	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-19	88.39	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-20	112.8	RC	Atlas Corp	Darling	DGPS	X	X	Vertical
122-21	94.49	RC	Atlas Corp	Darling	DGPS	X	X	X
122-22	158.5	RC	Atlas Corp	Darling	DGPS	X	X	X
122-23	91.44	RC	Atlas Corp	Darling	DGPS	X	X	X
122-24	158.5	RC	Atlas Corp	Darling	DGPS	X	X	X
122-25	134.1	RC	Atlas Corp	Darling	DGPS	X	X	X
122-26	109.7	RC	Atlas Corp	Darling	DGPS	X	X	X
124-1	67.97	CORE	Atlas Corp	Darling	DGPS	X	X	
124-2	54.56	CORE	Atlas Corp	Darling	DGPS	X	X	
124-3	139.6	CORE	Atlas Corp	Darling	DGPS	X	X	
124-4	109.1	CORE	Atlas Corp	Darling	DGPS	X	X	
CSG-001	60.96	CORE	CSGC	Darling	DGPS	X	X	X

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CSG-002	157.6	CORE	CSGC	Darling	DGPS	X	X	X
CSG-003	176.2	CORE	CSGC	Darling	DGPS	X	X	X
CSG-004	133.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-005	124.6	CORE	CSGC	Darling	DGPS	X	X	X
CSG-006	151.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-007	169.6	CORE	CSGC	Darling	DGPS	X	X	X
CSG-008	74.79	CORE	CSGC	Darling	DGPS	X	X	
CSG-009	74.99	CORE	CSGC	Darling	DGPS	X	X	X
CSG-010	158.7	CORE	CSGC	Darling	DGPS	X	X	X
CSG-011	135.2	CORE	CSGC	Darling	DGPS	X	X	X
CSG-012	112.5	CORE	CSGC	Darling	DGPS	X	X	Vertical
CSG-013	128.5	CORE	CSGC	Darling	DGPS	X	X	X
CSG-014	99.67	CORE	CSGC	Darling	DGPS	X	X	X
CSG-015	140.2	CORE	CSGC	Darling	DGPS	X	X	Vertical
CSG-016	105.2	CORE	CSGC	Darling	DGPS	X	X	Vertical
CSG-017	81.08	CORE	CSGC	Darling	DGPS	X	X	Vertical
CSG-018	100.6	CORE	CSGC	Darling	DGPS	X	X	X
CSG-019	60.05	CORE	CSGC	Darling	DGPS	X	X	X
CSG-020	144.8	CORE	CSGC	Darling	DGPS	X	X	X
CSG-021	200.3	CORE	CSGC	Darling	DGPS	X	X	X
CSG-022	161.5	CORE	CSGC	Darling	DGPS	X	X	X
CSG-023	99.97	CORE	CSGC	Darling	DGPS	X	X	X
CSG-024	53.95	CORE	CSGC	Darling	DGPS	X	X	Vertical
CSG-025	98.76	CORE	CSGC	Darling	DGPS	X	X	Vertical
CSG-026	163.7	CORE	CSGC	Darling	DGPS	X	X	X
CSG-027	199	CORE	CSGC	Darling	DGPS	X	X	X
CSG-028	151.5	CORE	CSGC	Darling	DGPS	X	X	X
CSG-029	157.6	CORE	CSGC	Darling	DGPS	X	X	X
CSG-030	169.5	CORE	CSGC	Darling	DGPS	X	X	X
CSG-031	239.9	CORE	CSGC	Darling	DGPS	X	X	X
CSG-032	201.2	CORE	CSGC	Darling	DGPS	X	X	X
CSG-033	168.6	CORE	CSGC	Darling	DGPS	X	X	X
CSG-034	139.3	CORE	CSGC	Darling	DGPS	X	X	X
CSG-035	191.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-036	188.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-037	169	CORE	CSGC	Darling	DGPS	X	X	X
CSG-038	141.7	CORE	CSGC	Darling	DGPS	X	X	X
CSG-039	149.4	CORE	CSGC	Darling	DGPS	X	X	X
CSG-040	157.9	CORE	CSGC	Darling	DGPS	X	X	X
CSG-041	80.62	CORE	CSGC	Darling	DGPS	X	X	X
CSG-042	151.2	CORE	CSGC	Darling	DGPS	X	X	X
CSG-043	56.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-044	195.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-045	178.9	CORE	CSGC	Darling	DGPS	X	X	X
CSG-046	66.9	CORE	CSGC	Darling	DGPS	X	X	X
CSG-047	151.5	CORE	CSGC	Darling	DGPS	X	X	X
CSG-048	145.1	CORE	CSGC	Darling	DGPS	X	X	X
CSG-049	171.8	CORE	CSGC	Darling	DGPS	X	X	X
CSG-050	47.55	CORE	CSGC	Darling	DGPS	X	X	X
CSG-051	200	CORE	CSGC	Darling	DGPS	X	X	X
CSG-052	225	CORE	CSGC	Darling	DGPS	X	X	X
CSG-053	242	CORE	CSGC	Darling	DGPS	X	X	X

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CSG-054	52.12	Met_Core	CSGC	Darling	DGPS	X	X	X
CSG-055	25	Met_Core	CSGC	Darling	DGPS	X	X	X
CSG-056	60.05	Met_Core	CSGC	Darling	DGPS	X	X	X
CSG-057	32.31	Met_Core	CSGC	Darling	DGPS	X	X	X
CSG-058	20.42	Met_Core	CSGC	Darling	DGPS	X	X	
1CHN-19030	7.62	UG_CHN		Bailey	Brunton		X	X
1CHN-19035	4.57	UG_CHN		Bailey	Brunton		X	X
1CHN-19038	1.52	UG_CHN		Bailey	Brunton		X	X
1CHN-19039	1.52	UG_CHN		Bailey	Brunton		X	X
1CHN-19040	1.52	UG_CHN		Bailey	Brunton		X	X
1CHN-19041	1.52	UG_CHN		Bailey	Brunton		X	X
1CHN-7153	13.72	UG_CHN		Bailey	Brunton			X
1CHN-7162	6.1	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7166	1.52	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7167	45.72	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7197	18.29	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7209	3.05	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7211	4.57	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7214	1.52	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7215	13.72	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7224	4.57	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7227	3.05	UG_CHN		Bailey	Brunton	X	X	X
1CHN-7229	9.14	UG_CHN		Bailey	Brunton	X	X	X
1CHN-9033	21.34	UG_CHN		Bailey	Brunton	X	X	X
2CHN-19000	4.57	UG_CHN		Bailey	Brunton		X	X
2CHN-19003	4.57	UG_CHN		Bailey	Brunton		X	X
2CHN-19006	4.57	UG_CHN		Bailey	Brunton		X	X
2CHN-19009	4.57	UG_CHN		Bailey	Brunton		X	X
2CHN-7083	16.76	UG_CHN		Bailey	Brunton	X	X	X
2CHN-7094	9.14	UG_CHN		Bailey	Brunton	X	X	X
2CHN-8000	7.62	UG_CHN		Bailey	Brunton	X	X	X
2CHN-8005	7.62	UG_CHN		Bailey	Brunton	X	X	X
2CHN-8010	9.14	UG_CHN		Bailey	Brunton	X	X	X
3CHN-10	13.72	UG_CHN		Bailey	Brunton	X	X	X
3CHN-11	9.14	UG_CHN		Bailey	Brunton	X	X	X
3CHN-2	18.29	UG_CHN		Bailey	Brunton	X	X	X
3CHN-3	3.05	UG_CHN		Bailey	Brunton	X	X	X
3CHN-5	15.24	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6	22.86	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6669	47.24	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6900	13.72	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6909	7.62	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6914	4.57	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6917	16.76	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6931	24.38	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6957	3.05	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6977	32	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6994	4.57	UG_CHN		Bailey	Brunton	X	X	X
3CHN-6997	4.57	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7	21.34	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7000	115.8	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7076	7.62	UG_CHN		Bailey	Brunton	X	X	X

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3CHN-7081	3.05	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7100	39.62	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7133	13.72	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7135	3.05	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7136	3.05	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7138	4.57	UG_CHN		Bailey	Brunton	X	X	X
3CHN-7268	51.82	UG_CHN		Bailey	Brunton	X	X	X
3CHN-8	35.05	UG_CHN		Bailey	Brunton	X	X	X
3CHN-8101	4.57	UG_CHN		Bailey	Brunton	X	X	X
3CHN-9	13.72	UG_CHN		Bailey	Brunton	X	X	X
5CHN-1	45.72	UG_CHN		Bailey	Brunton		X	X
5CHN-7281	19.81	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7305	9.14	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7310	30.48	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7313	12.19	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7321	51.82	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7355	62.48	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7396	18.29	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7408	12.19	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7435	6.1	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7439	9.14	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7445	1.52	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7446	6.1	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7694	10.67	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7701	6.1	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7705	6.1	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7711	4.57	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7712	7.62	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7717	4.57	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7720	6.1	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7724	16.76	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7735	10.67	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7748	10.67	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7759	16.76	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7760	25.91	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7777	10.67	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7784	13.72	UG_CHN		Bailey	Brunton	X	X	X
5CHN-7793	1.52	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7450	47.24	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7481	3.05	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7483	3.05	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7485	18.29	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7497	3.05	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7502	6.1	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7503	1.52	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7504	4.57	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7510	6.1	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7511	39.62	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7537	12.19	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7545	4.57	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7548	3.05	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7550	7.62	UG_CHN		Bailey	Brunton	X	X	X

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6CHN-7557	4.57	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7558	39.62	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7584	1.52	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7585	1.52	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7586	25.91	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7603	25.91	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7620	1.52	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7621	7.62	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7626	1.52	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7908	15.24	UG_CHN		Bailey	Brunton	X	X	X
6CHN-7981	143.3	UG_CHN		Bailey	Brunton	X	X	X
7CHN-1	111.3	UG_CHN		Bailey	Brunton	X	X	X
7CHN-112	16.76	UG_CHN		Bailey	Brunton	X	X	X
7CHN-123	21.34	UG_CHN		Bailey	Brunton	X	X	X
7CHN-137	16.76	UG_CHN		Bailey	Brunton	X	X	X
7CHN-148	13.72	UG_CHN		Bailey	Brunton	X	X	X
7CHN-157	12.19	UG_CHN		Bailey	Brunton	X	X	X
7CHN-165	15.24	UG_CHN		Bailey	Brunton	X	X	X
7CHN-175	12.19	UG_CHN		Bailey	Brunton	X	X	X
7CHN-183	18.29	UG_CHN		Bailey	Brunton	X	X	X
7CHN-195	41.15	UG_CHN		Bailey	Brunton	X	X	X
7CHN-222	16.76	UG_CHN		Bailey	Brunton		X	X
7CHN-233	27.43	UG_CHN		Bailey	Brunton		X	X
7CHN-251	27.43	UG_CHN		Bailey	Brunton		X	X
7CHN-74	30.48	UG_CHN		Bailey	Brunton	X	X	X
7CHN-94	27.43	UG_CHN		Bailey	Brunton	X	X	X
B3CHN-7416	19.81	UG_CHN		Bailey	Brunton	X	X	X
B3CHN-7429	4.57	UG_CHN		Bailey	Brunton	X	X	X
B3CHN-7432	3.05	UG_CHN		Bailey	Brunton	X	X	X
B3CHN-7434	1.52	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7640	15.24	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7659	36.58	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7677	4.57	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7679	3.05	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7680	1.52	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7682	3.05	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7684	3.05	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7685	1.52	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7686	4.57	UG_CHN		Bailey	Brunton	X	X	X
B5CHN-7692	19.81	UG_CHN		Bailey	Brunton	X	X	X
B6CHN-7687	4.57	UG_CHN		Bailey	Brunton	X	X	X
B6CHN-7691	3.05	UG_CHN		Bailey	Brunton	X	X	X
SCHN-10012	13.72	S_CHN		Bailey	Brunton		X	X
SCHN-10013	13.72	S_CHN		Bailey	Brunton		X	X
SCHN-10022	3.05	S_CHN		Bailey	Brunton		X	X
SCHN-10024	4.57	S_CHN		Bailey	Brunton		X	X
SCHN-10027	3.05	S_CHN		Bailey	Brunton		X	X
SCHN-10029	6.1	S_CHN		Bailey	Brunton		X	X
SCHN-11000	25.91	S_CHN		Bailey	Brunton	X	X	X
SCHN-11017	18.29	S_CHN		Bailey	Brunton	X	X	X
SCHN-11029	13.72	S_CHN		Bailey	Brunton	X	X	X
SCHN-11038	1.52	S_CHN		Bailey	Brunton	X	X	X

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SCHN-11039	15.24	S_CHN		Bailey	Brunton	X	X	X
SCHN-11049	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-11053	3.05	S_CHN		Bailey	Brunton	X	X	X
SCHN-11055	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-11059	18.29	S_CHN		Bailey	Brunton	X	X	X
SCHN-11071	36.58	S_CHN		Bailey	Brunton	X	X	X
SCHN-11083	21.34	S_CHN		Bailey	Brunton	X	X	X
SCHN-11090	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-11094	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-11097	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-11101	3.05	S_CHN		Bailey	Brunton	X	X	X
SCHN-11103	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-11106	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-11110	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-19012	9.14	S_CHN		Bailey	Brunton		X	X
SCHN-19018	1.52	S_CHN		Bailey	Brunton		X	X
SCHN-19019	1.52	S_CHN		Bailey	Brunton		X	X
SCHN-19020	7.62	S_CHN		Bailey	Brunton		X	X
SCHN-6367	7.62	S_CHN		Bailey	Brunton	X	X	X
SCHN-6372	9.14	S_CHN		Bailey	Brunton	X	X	X
SCHN-6378	28.96	S_CHN		Bailey	Brunton	X	X	X
SCHN-6397	9.14	S_CHN		Bailey	Brunton	X	X	X
SCHN-6403	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-6407	21.34	S_CHN		Bailey	Brunton	X	X	X
SCHN-6421	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-6424	12.19	S_CHN		Bailey	Brunton	X	X	X
SCHN-6435	45.72	S_CHN		Bailey	Brunton	X	X	X
SCHN-6465	19.81	S_CHN		Bailey	Brunton	X	X	X
SCHN-6471	22.86	S_CHN		Bailey	Brunton	X	X	X
SCHN-6477	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-6481	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-6484	18.29	S_CHN		Bailey	Brunton	X	X	X
SCHN-6497	12.19	S_CHN		Bailey	Brunton	X	X	X
SCHN-6552	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-6556	22.86	S_CHN		Bailey	Brunton	X	X	X
SCHN-6571	6.1	S_CHN		Bailey	Brunton	X	X	X
SCHN-6575	16.76	S_CHN		Bailey	Brunton	X	X	X
SCHN-7150	18.29	S_CHN		Bailey	Brunton	X	X	X
SCHN-8016	13.72	S_CHN		Bailey	Brunton	X	X	X
SCHN-8025	12.19	S_CHN		Bailey	Brunton	X	X	X
SCHN-8034	42.67	S_CHN		Bailey	Brunton	X	X	X
SCHN-8062	18.29	S_CHN		Bailey	Brunton	X	X	X
SCHN-8077	7.62	S_CHN		Bailey	Brunton	X	X	X
SCHN-8082	12.19	S_CHN		Bailey	Brunton	X	X	X
SCHN-8090	22.86	S_CHN		Bailey	Brunton	X	X	X
SCHN-9006	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-9009	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-9012	12.19	S_CHN		Bailey	Brunton	X	X	X
SCHN-9020	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-9023	4.57	S_CHN		Bailey	Brunton	X	X	X
SCHN-9026	7.62	S_CHN		Bailey	Brunton	X	X	X
SCHN-9031	3.05	S_CHN		Bailey	Brunton	X	X	X

Hole_ID	TD_M	DH_Type	Exploration Company	Surveyor	Survey Type	Assay Certificate	Geology	Downhole Survey
SCHN-9048	13.72	S_CHN		Bailey	Brunton	X	X	X
SCHN-9057	19.81	S_CHN		Bailey	Brunton	X	X	X
SCHN-9070	13.72	S_CHN		Bailey	Brunton	X	X	X
SCHN-9079	3.05	S_CHN		Bailey	Brunton	X	X	X
SCHN-9081	3.05	S_CHN		Bailey	Brunton	X	X	X
SCHN-UG1	15.24	S_CHN		Bailey	Brunton		X	X

Appendix D: Certificate of Author

CERTIFICATE of QUALIFIED PERSON

I, Zachary J. Black, SME-RM, do hereby certify that:

1. I am currently employed as Resource Geologist by:

   Hard Rock Consulting, LLC
   1746 Cole Blvd, Ste. 140
   Lakewood, Colorado 80401
   U.S.A.

2. I am a graduate of the University of Nevada, Reno with a Bachelor of Science in Geological Engineering, and have practiced my profession continuously since 2005. Engineering in 1995.

3. I am a registered member of the Society of Mining and Metallurgy and Exploration (No. 4156858RM):

4. I have worked as a Geological Engineer/Resource Geologist for a total of eight years since my graduation from university; as an employee of a major mining company, a major engineering company, and as a consulting geologist. I have 8+ years of experience working on structurally controlled gold and silver resources in the Sierra Madre Occidental of Mexico and the southern United States.

5. 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.

6. I personally inspected the Commonwealth Silver and Gold project field office on May 2nd and May 3rd, 2013.

7. I am responsible for the preparation of the report titled "National Instrument 43-101 Technical Report – Preliminary Economic Assessment, Commonwealth Silver and Gold Project, Cochise County, Arizona," dated April 30, 2014, with an effective date of November 30, 2013 (the "Technical Report"), with specific responsibility for sections 1, 11, 12, 14, 25, and 26.

8. I have had prior involvement with the property that is the subject of this Technical Report. I previously acted as a Qualified Person responsible for preparation of the report titled "National Instrument 43-101 Technical Report on Resources for the Commonwealth Silver and Gold Project, Cochise County, Arizona," dated September 5, 2013.

9. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated this 30th day of April, 2014.

"Signed" Zachary J. Black

Signature of Qualified Person

Zachary J. Black
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Jeffery W. Choquette, P.E., do hereby certify that:

1. I am currently employed as Principal Engineer by:

   Hard Rock Consulting, LLC
   1746 Cole Blvd, Ste. 140
   Lakewood, Colorado 80401
   U.S.A.

2. I am a graduate of Montana College of Mineral Science and Technology and received a Bachelor of Science degree in Mining Engineering in 1995.

3. I am a:

   - Registered Professional Engineer in the State of Montana (No. 12265)
   - QP Member in Mining and Mineral Reserves in good standing of the Mining and Metallurgical Society of America (No. 01425QP)

4. I have practiced mining engineering and project management for seventeen years. I have worked for mining and exploration companies for sixteen years and as a consulting engineer for two and a half years.

5. 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.

6. I have not personally inspected the Commonwealth Silver and Gold Project.

7. I am responsible for the preparation of the report titled "National Instrument 43-101 Technical Report – Preliminary Economic Assessment, Commonwealth Silver and Gold Project, Cochise County, Arizona," dated April 30, 2014, with an effective date of November 30, 2013 (the "Technical Report"), with specific responsibility for sections 15 through 24 of the report.

8. I have had prior involvement with the property that is the subject of this Technical Report. I previously acted as a Qualified Person responsible for preparation of the report titled "National Instrument 43-101 Technical Report on Resources for the Commonwealth Silver and Gold Project, Cochise County, Arizona," dated September 5, 2013.

9. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated this 30th day of April, 2014.

"Signed and sealed" Jeffery W. Choquette

Signature of Qualified Person

Jeffery W. Choquette
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Jennifer J. Brown, P.G., do hereby certify that:

1. I am currently employed as a contract Geologist by:

   Hard Rock Consulting, LLC
   1746 Cole Blvd, Ste. 140
   Lakewood, Colorado 80401
   U.S.A.

2. I am a graduate of the University of Montana with a Bachelor of Arts in Geology (1996), and I have practiced my profession continuously since 1997.

3. I am a licensed Professional Geologist in the States of Wyoming (PG-3719) and Idaho (PGL-1414), and am a Registered Member in good standing of the Society of Mining, Metallurgy, and Exploration (#4168244RM) with recognized special expertise in geology and mineral resources. I am also a member of the American Institute of Professional Geologists (MEM-0174).

4. I have worked as a geologist for a total of 17 years since graduation from the University of Montana, as an employee of four separate engineering and geologic consulting firms and the U.S.D.A Forest Service. I have ten collective years of experience directly related to mining and or economic and saleable minerals exploration and resource development, including geotechnical exploration, geologic analysis and interpretation, resource evaluation, and technical reporting.

5. 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.

6. I personally inspected the Commonwealth Silver and Gold Project on May 2nd and May 3rd, 2013.

7. I am responsible for the preparation of the report titled "National Instrument 43-101 Technical Report – Preliminary Economic Assessment, Commonwealth Silver and Gold Project, Cochise County, Arizona," dated April 30, 2014, with an effective date of November 30, 2013 (the "Technical Report"), with specific responsibility for Items 2 through 10, 23 and 27 of the report.

8. I have had prior involvement with the property that is the subject of this Technical Report. I previously acted as a Qualified Person responsible for preparation of the report titled "National Instrument 43-101 Technical Report on Resources for the Commonwealth Silver and Gold Project, Cochise County, Arizona," dated September 5, 2013.

9. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated this 30th day of April, 2014.

"Signed" Jennifer J. Brown, P.G.

Signature of Qualified Person

Jennifer J. Brown, P.G.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Deepak Malhotra, PhD do hereby certify that:

1. I am President of:

   Resource Development, Inc. (RDi)
   11475 W. I-70 Frontage Road North
   Wheat Ridge, CO, USA, 80033 USA

2. I graduated with a degree in Master of Science from Colorado School of Mines in 1973. In addition, I have obtained a PhD in Mineral Economics from Colorado School of Mines in 1977.

3. I am a registered member of the Society of Mining, Metallurgy and Exploration, Inc. (SME), member No. 2006420RM.

4. I have worked as a mineral processing engineer and mineral economist for a total of 40 years since my graduation from university.

5. 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.

6. I am responsible for the preparation of the report titled "National Instrument 43-101 Technical Report – Preliminary Economic Assessment, Commonwealth Silver and Gold Project, Cochise County, Arizona," dated April 30, 2014, with an effective date of November 30, 2013 (the "Technical Report"), with specific responsibility for Item 13 of the report.

7. I have had no prior involvement with the property that is the subject of this Technical Report..

8. As of the date of this certificate and as of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated this 30th Day of April, 2014.

"signed" Deepak Malhotra

Deepak Malhotra, PhD
Print name of Qualified Person