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AMENDED AND RESTATED
TECHNICAL REPORT
ALLIED NEVADA GOLD CORP.
HYCROFT MINE, WINNEMUCCA, NEVADA, USA

SEPTEMBER 23, 2010

Prepared by

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1 SUMMARY

1.1 INTRODUCTION

This National Instrument 43-101 ("NI 43-101") compliant Technical Report was prepared by Scott E. Wilson Consulting, Inc. ("SEWC") of Englewood, Colorado for Allied Nevada Gold Corp. ("Allied Nevada"), a Delaware corporation listed on the Toronto Stock Exchange and the NYSE - Amex (Symbol ANV). Allied Nevada owns the Hycroft mine; Maverick Springs (a 45% joint venture with Silver Standard Resources, Inc.), Mountain View, Hasbrouck, Three Hills and Wildcat projects; the Contact, Pony Creek/Elliot Dome property packages; and the exploration rights to more than one hundred other early stage exploration properties. The Hycroft gold mining operation is the subject of this report.

This report describes seven gold and silver deposits collectively referred to as the Hycroft mine, Hycroft project or simply Hycroft. The seven deposits, listed below, contain the mineral resources and mineral reserves addressed in this report.

- Brimstone deposit (Mineral Reserves and Mineral Resources).
- Cut-5 deposit (Mineral Reserves and Mineral Resources).
- Camel Hill deposit (Mineral Reserves and Mineral Resources).
- Bay Area deposit (Mineral Reserves and Mineral Resources).
- Boneyard (Mineral Reserves and Mineral Resources).
- Central Zone (Mineral Reserves and Mineral Resources).
- Vortex Zone (Mineral Resources).

All material at Hycroft has been classified in accordance with the resource classification of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") in compliance with NI 43-101 and Industry Guide 7 of the United States Securities and Exchange Commission. This new Technical Report identifies changes to the NI 43-101 compliant Mineral Reserves and Mineral Resources that were reported in the April 1, 2010 Technical Report, authored by SEWC and published on SEDAR. The changes are current as of the date of June 1, 2010. Numerous sources of information, both digital and hard copy, were used in the preparation of this report. The data comprises over 3,900 exploration holes as well as updated geological interpretations. Ordinary kriging was used to estimate the grade of gold, and inverse distanced cubed to estimate the grade of silver.

Included in this Technical Report is a discussion of the accelerated oxide mining plan. The plan, now being implemented, and a mill scoping study relating to the economic potential of the sulfide mineral resource at Hycroft, is also discussed in the report.

The author of this report, Scott Wilson, a Qualified Person, has visited the Hycroft property on numerous occasions. Most recently, Mr. Wilson visited the Hycroft mine on June 19, 2009, to review pit designs, mine plans, and scheduling options associated with the operation of the Hycroft mine.

The Hycroft mine is an open pit, heap leach gold and silver mine. Hycroft is located 54 miles west of Winnemucca, Nevada, and, historically, has produced in excess of one million ounces of gold and two million ounces of silver. Formerly, the Hycroft mine was known as the Crofoot-Lewis mine. Mining began in the area in 1983 with a small heap leach operation known as the Lewis mine. The Lewis mine production was followed by production from the Crofoot property in the Bay Area, South Central, Boneyard, Gap and Cut-4 pits along the Central fault, and finally the north end of the Brimstone pit and continued until it was placed on a care and maintenance program in December 1998 due to gold prices (below $300/oz).

The Hycroft mine consists of 22 patented claims that comprise approximately 1,794 acres and 2,521 unpatented claims that comprise approximately 51,932 acres. Combining the patented and unpatented claims, Hycroft claims total approximately 53,726 acres. This claim package was acquired by Allied Nevada in a series of transactions:

- The Crofoot property and approximately 3,500 acres of claims were acquired by Vista Gold Corp. ("Vista") in 1988. The Crofoot property, originally held under lease, is owned by Hycroft Resources & Development, Inc. ("HRDI") subject to a 4% NPI retained by the former owners.
- The F.W. Lewis property and approximately 8,700 acres of claims were acquired by Vista in early 1987.
- In 2006, approximately 13,100 acres of additional claims were staked by Vista. These claims were around or contiguous to the original Crofoot and F.W. Lewis claims.
- In 2008, approximately 22,700 acres of additional claims were staked by HRDI contiguous to or around the existing Hycroft claims.
- In 2009, an additional 79 claims were staked by HRDI contiguous to or around the existing Hycroft land holding.
- In 2010, an additional 94 claims were staked by HRDI contiguous to or around the existing Hycroft land holding.

The Crofoot property is held by HRDI, a wholly owned subsidiary of Allied Nevada. A 4% Net Profit Interest ("NPI") is retained by the original Crofoot owners. In 1996, the lease/purchase agreement was amended to provide for minimum advance royalty payments of $120,000 on January 1 of each year in which mining occurs on both patented and unpatented claims. All payments for the Crofoot property are capped at $7.6 million of which $960,000 has been paid to date, after which Allied Nevada will own the property. An additional $120,000 is due if ore production exceeds 4.0 million tons from the Crofoot property, on both patented and unpatented claims, in any calendar year. All advanced royalty payments are available as credit against the 4% NPI.

The leasehold interest in the Lewis property is wholly owned by Allied Nevada.

Of the approximately 53,726 acres of patented and unpatented mineral claims, 8,858 acres are within the current Plan of Operations. Nearly 2,300 acres have been disturbed by mining operations. The Hycroft mine is in full production and has been recovering gold from its processing facilities. Table 1.1 lists the most important aspects of the Hycroft mine that are addressed in this report.

Table 1.1 Hycroft Mine Technical Report Relevant Statistics

Category	Description
Property Name	Hycroft Mine
Company Name	Hycroft Resources and Development Inc.
Owner	Allied Nevada Gold Corp.
Land Position	Public and Private Claims, Nevada and BLM
Nearest Population Center	Winnemucca, Nevada
Mine Location	54 miles west of Winnemucca via Highway 49 (Jungo Road)
Topography	Low Hills
Climate	Arid Desert
Historic Production	Over 1 million oz Au and 2 million oz Ag Since 1983
Reason for NI 43-101 Technical Report	Material Changes to the Mineral Resources and Mineral Reserves at the Hycroft Mine; update oxide mine plan and scoping study of sulfide measured, indicated and inferred resources
Mineralization Type	Fracture Controlled Disseminated Gold
Estimation Type	Ordinary Kriging for Gold
Mine Life	5 Years
Mining Rate	67.5 Million Tons per Year Mined
Mining Method	Open Pit Truck, Wheel Loaders, and Hydraulic Shovel
Processing Method	Heap Leaching
Overall Processing Gold Recovery	60% (combined Crushed & ROM recoveries)
Gold Selling Price - SEWC Financial Model	$800
Mining Cost per Ton Mined	$1.07
Processing and G&A Cost per Ore Ton	$1.51
Crushing cost per Ton	$2.15
Operating Cost per Ore Ton	$3.48
Pre-Tax Cash Flow	$284.2 million
Pre-Tax Net Present Value at 6%	$198.1 million

1.2 GEOLOGY AND MINERALIZATION

The Hycroft mine is located in the Nevada basin and range geologic province on the western flank of the Kamma Mountains, straddling the county line between Humboldt and Pershing Counties, Nevada. Tertiary-to-recent, fault controlled, low sulfidation gold deposits occur over an area measuring three miles in a north-south direction and two miles in an east-west direction.

Based on drilling results, mineralization extends to depths of at least 330 ft from the northwest corner of the Bay Area, to over 1,000 ft in the Brimstone deposit in the eastern portion of the Hycroft property, and over 2,000 ft in the southeastern Vortex Zone.

Five major north-northeast trending, west dipping, normal fault zones broadly bound gold mineralization. The fault zones are referred to as the Central, Boneyard, Albert, Fire and East faults. The Lewis, Bay Area, Central and South Central, Cut-3, and Cut-4 deposits are hosted by sedimentary conglomerate and lacustrine rocks (the "Sulfur Group") in the hanging wall of the Central fault.

The Brimstone deposit is hosted within the hanging wall of the East fault. This portion of the deposit has been highly structurally prepared by at least four phases of alteration. Gold mineralization is thought to have occurred during periods of fracture and breccia controlled chalcedony sulfide mineralization. Oxidation appears to be related to a deep, wide spread acid leaching event, and by descending fluids along the East fault.

The Vortex Zone, bordering the south end of the Brimstone deposit, is hosted in rocks similar to those at Brimstone, but overprinted with extensive hydrothermal brecciation. Alteration in the Vortex Zone is primarily strong silicification capped with hydrothermal argillic and subordinate acid leach alteration. Mineralization in the Vortex Zone is thought to be related to several pulses of fracturing and hydrothermal brecciation, plus quartz and chalcedonic veining. Vortex is bounded on the east by the East fault, and is open to the north, west, south and at depth.

The historically mined Lewis, Bay Area, Central and South Central, Cut-3, and Cut-4 deposits were hosted by the Sulfur Group in the hanging wall of the Central fault. The host rocks in these deposits are silicified conglomerate rocks comprised of sedimentary and volcanic rock fragments. The Central fault provided mineralizing fluids which both altered the host rock and deposited gold and silver. Alteration ranges from strong argillic and acid leaching in the Central, Cut-3 and Cut-4 areas, to passive silicification and hot spring sinters in the Bay Area and Lewis areas.

1.3 DRILLING AND SAMPLING

Exploration and development drilling by Allied Nevada and its predecessors totals 1,507,099 ft of drilling in 3,922 drill holes at the Hycroft mine. In December 2006, Allied Nevada drilled one RC hole at the south end of the Cut-4 pit. Allied Nevada then commenced a drilling program to delineate oxide and sulfide resources throughout the entire Hycroft property. As of June 30, 2010, Allied Nevada had completed 676 holes totaling 534,397 ft. During 2010, 153 holes totaling 138,689 ft were drilled.

Current sample collection, assaying, and certification of assays are consistent with currently accepted mining and operating practices. The sampling methods are standardized and tracked by mine site geologists. Sample preparation, analysis, and security are handled by two reputable laboratories. All data is verified before being entered into the drill hole databases for resource estimation.

1.3.1 RESOURCES

SEWC has developed the breakdown for resources on the Hycroft property. Resources for this report are classified as either oxide or sulfide. The classes are characterized as:

- Oxide Mineralization - The oxide material will be processed by utilizing the existing and expanded heap leach pads and processing facilities. Both gold and silver oxide mineralization are present at Hycroft.

- Sulfide Mineralization – The Measured, Indicated and Inferred resource estimate for sulfide material containing gold and silver was calculated from fire assay data obtained from over 3,900 historic and Allied Nevada drill holes, comprising approximately 1.5 million ft of drilling. The estimate of the grade of silver sulfide mineralization is primarily based on 534,397 ft of drilling completed by Allied Nevada. The sulfide resource has been calculated over a wide area of Hycroft and continues to show the potential for there to be a large mineralized system.

1.3.2 NI 43-101 COMPLIANT MEASURED AND INDICATED MINERAL RESOURCES

The June 1, 2010 Measured and Indicated Resources are reported at a gold equivalent cut off grade of 0.009 opt for oxide mineralization and 0.018 opt for sulfide mineralization. Measured and Indicated Oxide Resources for the Hycroft mine are shown in Table 1.2. Measured and Indicated Sulfide Resources for the Hycroft mine are shown in Table 1.3.

Table 1.2 June 1, 2010 Hycroft Mine Measured and Indicated Oxide Gold and Silver Resources

Cut Off Grade (opt)	Tons	Grade (opt)		Contained Ounces	
AuEq		Au	Ag	Au	Ag
0.009	396,000,000	0.013	0.32	4,904,000	127,786,000

Table 1.3 June 1, 2010 Measured and Indicated Sulfide Gold and Silver Resources

Cut Off Grade (opt)	Tons	Grade (opt)		Contained Ounces	
AuEq		Au	Ag	Au	Ag
0.018	177,000,000	0.018	0.73	3,104,000	131,422,000

1.3.3 NI 43-101 COMPLIANT INFERRED MINERAL RESOURCES

Inferred Oxide Resources are shown in Table 1.4 at a gold equivalent cut off grade of 0.009 opt and Inferred Sulfide Resources are reported in Table 1.5 at a gold equivalent cut off grade of 0.018 opt. Industry accepted standards for resource estimation were used to determine the extent of mineralization at Hycroft. Gold mineralization was estimated using ordinary kriging, and silver mineralization was estimated using inverse distance cubed.

Table 1.4 June 1, 2010 Oxide Inferred Resource @ 0.009 opt Gold Equivalent Cut Off Grade

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Inferred Resource					
Ag Inferred Associated with Au M&I					31,344
Au and Ag Inferred	148,000	0.011	0.55	1,628	81,437
Crofoot Pad	35,000	0.009	-	318	-
Inferred Total	**183,000**	**0.011**	**-**	**1,946**	**112,781**

Table 1.5 June 1, 2010 Sulfide Inferred Resources @ 0.018 opt Gold Equivalent Cut Off Grade

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Inferred Resource					
Ag Inferred Associated with Au M&I					10,267
Au and Ag Inferred	153,000	0.017	1.07	2,601	163,644
Inferred Total	**153,000**	**0.017**	**-**	**2,601**	**173,911**

1.4 OPERATING MINE PLAN

SEWC used current economics to confirm the recently accelerated mine plan for the Hycroft mine. At a mining rate of approximately 67.5 million tons of ore and waste per year, the mine can operate for about 5 years. The ore will be placed on Phases 1 and 2 of the current Brimstone heap. The current pad height will be increased from 200 ft to 400 ft and the pad extended to the north to accommodate approximately 90.7 million tons of additional ore. A new leach pad has been designed just south of the Crofoot pad that will accommodate up to 85 million tons of ore to be used for South Central and South Brimstone pit development. Leaching of the ore will occur over 6 years.

The current plan is for the mine to run 24 hours per day, seven days per week. Mining is expected to average 67.5 million tons of total material per annum. The internal ore cut off grade for ROM ore is 0.005 opt, and the crusher cutoff grade is variable based on location, processing method, and metallurgy. Waste will be used as pit backfill material or placed on waste dumps located adjacent to the production pits.

All ore grade material placed on the leach pad is cross ripped to enhance permeability, and a network of solution drip lines is positioned on top of the ore to apply a cyanide solution to dissolve the gold. Return solution from the pad containing the precious metals is directed to the pregnant solution pond and processed in the CIC or Merrill-Crowe plants.

The Merrill-Crowe process clarifies and de-oxygenates the pregnant solution using two 1,600 ft^2 sparkler filters. Zinc dust is applied to the clarified solution where gold precipitates and is collected on three 48" recessed plate filter presses. Excess solution returns are processed in two CIC trains and gold and silver collected on carbon. The collected precipitate is refined at the new refinery where mercury is removed and the gold and silver are fire refined. Gold and silver on carbon is processed off-site. The solution flow is a closed process so the barren solution from the plant is returned to the leach pad circuit to continue the leaching process. Table 1.6 shows the ROM and crushed recoveries by area.

Table 1.6 Hycroft Ore Body Recovery Estimate Based on Fire Assay

	3/8" Rec, %		3/4" Rec, %		ROM Rec, %	
	Au	Ag	Au	Ag	Au	Ag
Cut-5	59	20	48	15	42	11
Central Fault	61	9	57	8	52	6
Camel Hill	59	11	48	11	42	11
Boneyard	64	8	50	7	45	5
Bay Area	62	16	57	13	50	11
Brimstone	73	28	68	19	57	12
Vortex	73	28	68	19	57	12

Assuming 10% of total ore mined is crushed to 3/8", it is expected that the overall average recovery for gold and silver would be approximately 60% and 14%, respectively.

1.5 NI 43-101 COMPLIANT PROVEN AND PROBABLE MINERAL RESERVES

Economic reserves for the Brimstone, Bay Area, Boneyard, Central Fault and Cut-5 deposits were calculated based on current operational economics and engineered estimates for costs of future years for Hycroft. SEWC verified the economic pit limits of the mineral reserve estimate using Whittle® 4.3.1 software. Table 1.7 summarizes the Hycroft reserves as of June 1, 2010. The stated Mineral Reserve Estimate conforms to the December 23, 2005, CIM definitions of Proven and Probable Mineral Reserves.

Table 1.7 Mineral Reserves at June 1, 2010

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Proven Reserve	135,000	0.014	0.23	1,900	30,600
Probable Reserve	39,000	0.013	0.21	500	8,300
Total Reserve	**174,000**	**0.014**	**0.22**	**2,400**	**38,900**
Waste	179,000				
Total Pit Tons	**353,000**				
Strip Ratio	1.04:1				

The waste material inside the final pit design includes 113 million tons of oxide material grading 0.002 opt AuFA. The waste includes 27 million tons of sulfide material that grades 0.021 opt AuFA above a 0.018 opt AuFA cut off grade. There is an additional 39.5 million tons of sulfide mineralization below a 0.018 cutoff grade. **Though these mineral resources are targeted to be mined within the Hycroft pit, mineral resources that are not mineral reserves do not have demonstrated economic viability.**

1.6 ONGOING RESOURCE DEVELOPMENT

Allied Nevada has undertaken metallurgical studies on material from various zones of the Hycroft property to determine the recoveries associated with the potential to process sulfide and oxide gold and silver material through a mill. The initial results have been encouraging and further studies are planned into 2011. The large sulfide resource has been drill tested and delineated by widely spaced drilling programs. Allied Nevada plans to continue drilling programs to increase the drilling density of the deep sulfide and Vortex Zone inferred gold and silver resource portions of the ore deposit and to expand the extent of the sulfide mineralization. This will allow for the potential to convert additional resources into the measured and indicated mineral resource categories. The resources should continue to be tested for millability.

1.7 CONCLUSIONS

1.7.1 ADEQUACY OF PROCEDURES

SEWC, as well as other reputable firms and consultants, have reviewed the methods and procedures of Allied Nevada and its predecessors. The methods of geological interpretation, geotechnical evaluation, mine planning, metallurgical assessment, and assaying procedures are reasonable and meet generally accepted practices for operating Nevada gold mines.

1.7.2 ADEQUACY OF DATA

SEWC believes that Allied Nevada has conducted exploration and development sampling and analysis programs using industry standard practices. The resulting data can be relied upon to estimate Mineral Resources and Mineral Reserves at the Hycroft project.

1.7.3 ADEQUACY OF FINANCIAL INFORMATION

The economics of this Technical Report are based on actual and predicted information gathered from discussions with Allied Nevada personnel. SEWC believes the cost tracking procedures and assumptions at Hycroft are adequate enough to draw reliable conclusions on the economics of the Hycroft mine.

1.7.4 COMPLIANCE WITH CANADIAN NATIONAL INSTRUMENT NI 43-101 AND U.S. S.E.C. INDUSTRY GUIDE 7

The drill hole database and assaying quality for the Hycroft mine is sufficient for the determination of Measured, Indicated and Inferred Mineral Resources. Additionally, the geological interpretations, metallurgical assumptions, and the spatial drilling densities within the Brimstone, Cut-5, Boneyard, Bay Area, Central Fault and Camel Hill deposits are sufficient to define and state Proven and Probable Mineral Reserves for Hycroft. All of the aforementioned categories are compliant as defined by the December 23, 2005 CIM Standards of Disclosure for Mineral Projects, Form 43-101F1 and Companion Policy 43-101CP. The Proven and Probable Mineral Reserves categories are compliant with the United States Securities and Exchange Commission Industry Guide 7.

1.7.5 NOTE TO U.S. READERS CONCERNING USE OF MEASURED, INDICATED AND INFERRED RESOURCES

The terms “Mineral Resource”, “Measured Mineral Resource”, “Indicated Mineral Resource” and “Inferred Mineral Resource” used in this report are Canadian mining terms as defined in accordance with NI 43-101 under guidelines set out in the CIM Standards on Mineral Resources and Mineral Reserves adopted by the CIM Council on December 11, 2005. While the terms “Mineral Resource”, “Measured Mineral Resource”, “Indicated Mineral Resource” and “Inferred Mineral Resource” are recognized and required by Canadian regulations, they are not defined terms under the United States Securities and Exchange Commission Industry Guide 7. Under United States standards, mineralization may not be classified as a “reserve” unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the reserve calculation is made. As such, certain information contained in this report concerning descriptions of mineralization and resources under Canadian standards is not comparable to similar information made public by United States companies subject to the reporting and disclosure requirements of the United States Securities and Exchange Commission. An "Inferred Mineral Resource" has a great amount of uncertainty as to its existence and as to its economic and legal feasibility. It cannot be assumed that all or any part of an "Inferred Mineral Resource" will ever be upgraded to a higher category or will ever be confirmed or converted into "Reserves," as such a term is defined in Industry Guide 7 of the United States Securities and Exchange Commission. Under Canadian rules, estimates of Inferred Mineral Resources may not form the basis of feasibility or other economic studies. Readers are cautioned not to assume that all or any part of Measured or Indicated Resources will ever be converted into Mineral Reserves. Readers are also cautioned not to assume that all or any part of an "Inferred Mineral Resource" exists, or is economically or legally mineable. In addition, the definitions of "Proven Mineral Reserves" and "Probable Mineral Reserves", under CIM standards differ in certain respects from the standards of the United States Securities and Exchange Commission.

1.8 RECOMMENDATIONS

SEWC recommends that Allied Nevada implement the following resource development plans at Hycroft.

- Evaluate the exploration opportunity to further expand the oxide resource.
- Complete infill drilling and metallurgical evaluation of the sulfide resource and initiate feasibility level review.
- Continue drilling to delimit the Vortex resource and high grade ore exploration program.
- Drilling to improve confidence in the estimate of silver grades in the rest of the Hycroft zones.

2 TERMS OF REFERENCE

2.1 PURPOSE OF TECHNICAL REPORT

SEWC prepared this Technical Report of the Hycroft mine at the request of Allied Nevada. The Hycroft mine is owned by Allied Nevada. Allied Nevada made the decision, in September of 2007, to reactivate its wholly owned Hycroft mine which was placed on care and maintenance in late 1998 due to low metal prices. Open pit mining of the Brimstone pit resumed in the third quarter of 2008. Gold production began in the fourth quarter of 2008.

This report is intended to provide a technical summary of the Hycroft mine gold and silver resources and reserves for Allied Nevada. Also presented is a mill scoping study for the measured, indicated and inferred sulfide resource located directly below the oxide resource. This Technical Report is written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1. Prior to this report, SEWC authored a Technical Report pertaining to the Hycroft mine dated April 1, 2010 (Wilson, 2010). The technical information contained in this Technical Report reflects material changes that have occurred since the April 1, 2010 Technical Report, authored by SEWC. The remaining resources and reserves cited for the Hycroft mine are current as of June 1, 2010.

2.2 SOURCES OF INFORMATION

The scope of this study included a review of pertinent technical reports and data in possession of Allied Nevada relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, and resources and reserves. Material changes discussed in this report are based on the remodeling and re-interpretation of geology at the Brimstone, Cut-5, Bay Area, Boneyard, Camel Hill, Central Zone, and Vortex Zone deposits. Observations and interpretations of geostatistics, geology, grade estimation, and determination of mineralized trends at Hycroft were generated independently by SEWC and discussed internally with Allied Nevada. The Hycroft model was generated and evaluated with Vulcan® 3D scientific software. Economic pit limits were determined with Whittle® Strategic Planning software.

2.3 EXTENT OF INVOLVEMENT OF QUALIFIED PERSON

The author's mandate was to determine the most current property wide oxide and sulfide gold and silver mineral resource estimates for the Hycroft mine. The author was also mandated to determine the Proven and Probable Mineral Reserves for the Hycroft mine and to conduct the mill scoping study included in this report. The author is responsible for the construction of the Hycroft block model and the interpretation of statistics and grade estimation techniques for the Hycroft mine. The author visited the mine site for a personal inspection on June 19, 2009, to validate mine planning options related to production of gold and silver at the Hycroft mine.

2.4 TERMS OF REFERENCE

Unless stated otherwise, all volumes and grades are in US Imperial units and currencies are expressed in constant 2010 US dollars. Distances are expressed in US imperial units. This report is written specifically for the Hycroft mine operation.

2.5 UNITS OF MEASURE

2.5.1 COMMON UNITS

Table 2.1 Common Units

Cubic foot	ft^3
Cubic yard	yd^3
Degree	°
Degrees Fahrenheit	°F
Foot	ft
Gallon	gal
Gallons Per Minute	gpm
Gram	g
Greater than	>
Inch	"
Kilo (thousand)	K
Less than	<
Miles per hour	mph
Million	M
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Pound(s)	Lb(s)
Short ton (2,000 lb)	st
Short ton (US)	t
Short tons per day (US)	tpd
Short tons per hour (US)	tph
Short tons per year (US)	tpy
Square foot	ft^2
Square inch	in^2
Year (US)	yr

2.5.2 COMMON CHEMICAL SYMBOLS

Table 2.2 Symbols

Calcium Carbonate	$CaCO_3$
Copper	Cu
Cyanide	CN
Gold	Au
Hydrogen	H
Iron	Fe
Lead	Pb
Mercury	Hg
Silver	Ag
Sodium	Na
Sulphur	S
Zinc	Zn

2.5.3 COMMON ACRONYMS

Table 2.3 Acronyms

Atomic Absorption	AA
Bureau of Land Management (U.S.)	BLM
Canadian Institute of Mining, Metallurgy and Petroleum Engineers	CIM
Carbon in Column	CIC
Environmental Impact Statement	EIS
Environmental Protection Agency (U.S.)	EPA
Fracture Controlled Chalcedony-Pyrite-Marcasite Mineralization	FCCPM
Gold Equivalent	AuEq
Gold Ounces per Short Ton	Opt Au
International Standards Organization	ISO
Mineral Resources Development, Inc.	MRDI
Net Profit Interest	NPI
Net Smelter Return	NSR
Nevada	NV
Nevada Department of Environmental Protection	NDEP
Ounce per Ton	opt
Quality Assurance/Quality Control	QA/QC
Reverse Circulation	RC or RVC
Right-of-Way	ROW
Rock Quality Designation	RQD
Run-of-Mine	ROM
Scott E. Wilson Consulting, Inc.	SEWC
Silver Ounces per Short Ton	Opt Ag
SRK Consulting (U.S.), Inc.	SRK

3 RELIANCE ON OTHER EXPERTS

The opinions expressed in this report are based on the available information and geologic interpretations as provided by Allied Nevada.

The author is not a metallurgist and has relied upon written reports and statements of other individuals and companies that do business with ANV. Namely the reports of MacDonald, J, et al, on investigations of the Recovery of Gold and Silver from Hycroft Project Sulfide Samples; as well as the work and opinions of recoveries of gold and silver as described by David Hill, Vice President of Metallurgy employed by ANV. These assumptions have been used in the various parts of the accelerated mine plan and the mill scoping study. The author has exercised independence in reviewing the supplied information, believes that the basic assumptions are factual and correct, and the interpretations are reasonable. The author has relied on this data and has no reason to believe that any material facts have been withheld.

The results and opinions expressed in this report are conditional upon the aforementioned data and information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions herein. SEWC reserves the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to SEWC subsequent to the date of this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Hycroft mine is located 54 miles west of Winnemucca in Humboldt County, Nevada (Figure 4.1) with a significant portion of the property in adjacent Pershing County. The mine is easily accessible via Nevada Highway 49 (Jungo Road), an all weather, unpaved road that is maintained by Humboldt County (Wilson, 2008; Prenn, 2006).

The mine property straddles Townships 34, 35, 35.2 and 36 North and Ranges 29 and 30 East with an approximate latitude 40º 52' north and longitude 118º 41'. The mine is situated on the western flank of the Kamma Mountains and on the eastern edge of the Black Rock Desert in un-surveyed Sections 1 and 2, Township 34 North, Range 29 East; Sections 13, 23, 24, 25, 26, 27, 34, 35, 36, Township 35 North, Range 29 East; and Sections 17, 18, 19, 20, 30, 31, Township 35 North, Range 30 East, MDB&M, Humboldt County, and Sections 1, 2, 3, 11, 12, 13, 14, 23, 24, 25, 26, Township 34 North, Range 29 East; and Sections 5, 6, 7, 8, 17, 18, 19, 20, 29, 30, Township 34 North, Range 30 East, MDB&M, Pershing County, Nevada. Allied Nevada staked 25 claims in November, 2007 in Sections 28, 31, 32 and 33, Township 35 North, Range 30 East, MDB&M, Humboldt County, and Sections 1, 11, 12 and 14, Township 34 North, Range 29 East, Pershing County. One claim was staked in January, 2008 in Section 34, Township 35 North, Range 29 East, MDB&M, and Humboldt County. An additional 1,057 unpatented lode mining claims were staked in April and May, 2008 and recorded with the BLM in late June, 2008. These new claims are located in Sections 1, 2, 3, 10, 11, 12, 13, 14, 15, 21, 22, 23, 27, 28, 29, 30, 31, 32 and 33, Township 35 North, Range 29 East; Sections 36, Township 35 North, Range 28 East; Sections 25, 26, 35 and 36, Township 35.2 North, Range 29 East; Sections 4, 5, 6, 7 and 8, Township 35 North, Range 30 East; Sections 28, 32 and 33 Township 36 North, Range 29 East; Sections 19, 28, 29,30,31, 32, 33 and 34, Township 36 North, Range 30 East, MDB&M, Humboldt County, and in Sections 3, 4, 5, 6, 7, 8, 9, 10, 15, 16, 17, 18, 19, 20 and 21, Township 34 North, Range 29 East; Sections 1, 2, 11, 12 and 13, Township 34 North, Range 28 East; MDB&M, Pershing County. Please note that much of the project area is located on un-surveyed public and private land and the sections, ranges, and townships listed above have been interpolated for purposes of this general description. However, all patented claims have been surveyed (Wilson, 2008; Prenn, 2006).

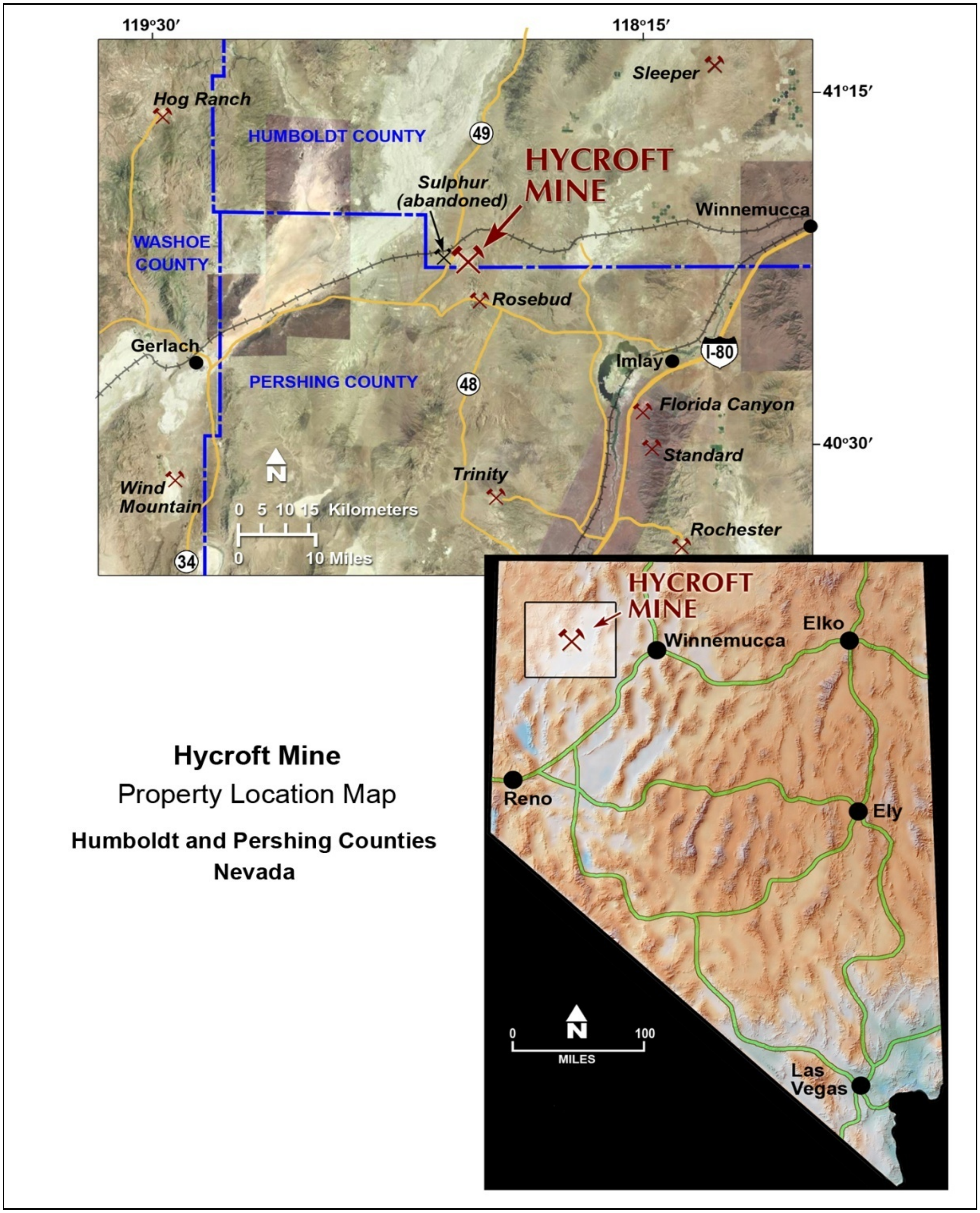
On May 10, 2007, Vista transferred its Nevada assets, including Hycroft, to Allied Nevada.

The Hycroft mine consists of 22 patented claims that comprise approximately 1,794 acres and 2,521 unpatented claims that comprise approximately 51,932 acres. Combining the patented and unpatented claims, Hycroft claims total approximately 53,726 acres. This claim package was acquired by Allied Nevada in a series of transactions:

- The Crofoot property and approximately 3,500 acres of claims were acquired by Vista in 1988. The Crofoot property, originally held under lease, is owned by HRDI subject to a 4% NPI retained by the former owners.
- The F.W. Lewis property and approximately 8,700 acres of claims were acquired by Vista in early 1987.
- In 2006, approximately 13,100 acres of additional claims were staked by Vista. These claims were around or contiguous to the original Crofoot and F.W. Lewis claims.

- In 2008, approximately 22,700 acres of additional claims were staked by HRDI contiguous to or around the existing Hycroft claims.
- In 2009, an additional 79 claims were staked by HRDI contiguous to or around the existing Hycroft land holding.
- In 2010, an additional 94 claims were staked by HRDI contiguous to or around the existing Hycroft land holding.

Figure 4.1 Hycroft Mine Property Location Map



4.2 LAND STATUS

The mine is managed and operated by Allied Nevada under the name of the Hycroft mine.

There are 2,521 unpatented mining claims covering approximately 51,932 acres at the Hycroft site. An additional 1,794 acres are in patented lode and placer claims and are the core property surrounded by the unpatented claims (Figure 4.2). The permitted site disturbance for current and future mining activities total 2,851 acres.

Figure 4.3 shows the property layout including site facilities, mineralized zones, mine workings, and waste deposits.

Figure 4.2 Hycroft Mine Claim Map

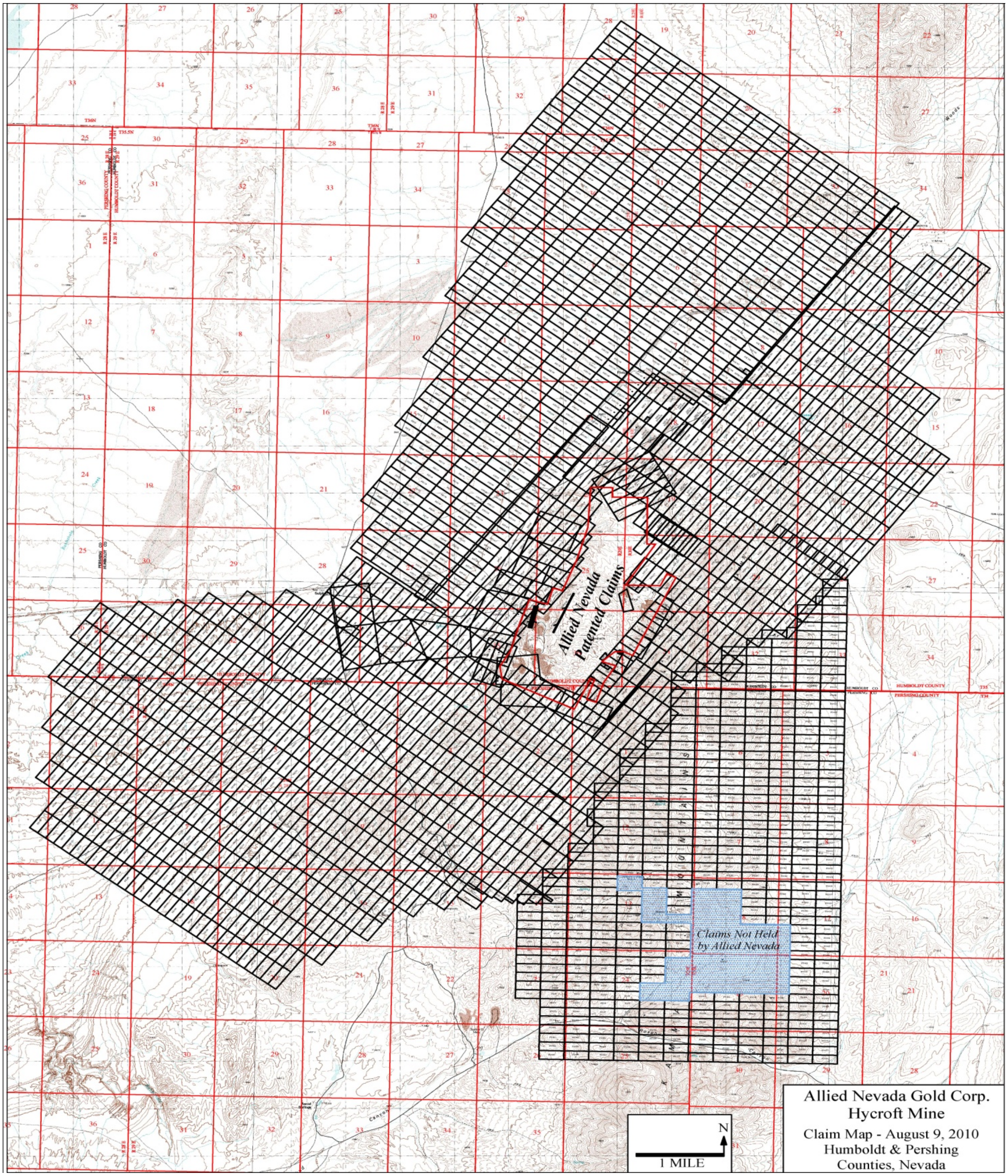
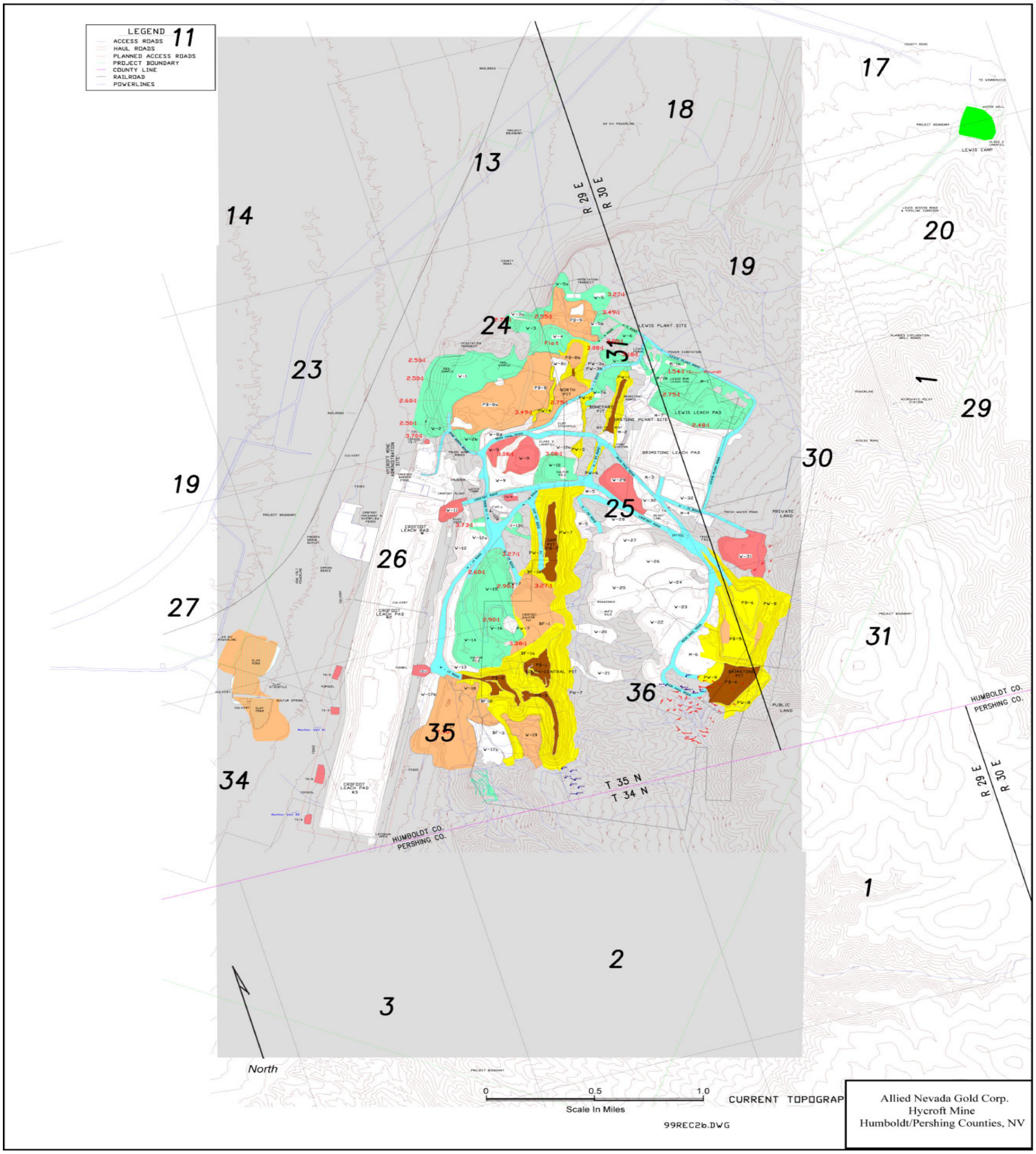


Figure 4.3 Property and Facilities Layout



4.3 AGREEMENTS AND ROYALTIES

The leasehold interests of Hycroft are composed of two primary properties, Crofoot and Lewis. The Crofoot and Lewis properties together comprise approximately 11,829 acres. The Crofoot property covers approximately 3,500 acres and is virtually surrounded by the Lewis property of 8,400 acres.

Vista exercised its option to purchase the Lewis property on December 13, 2005 by purchasing all the outstanding shares of F. W. Lewis, Inc. for $5.1 million. In addition to the Lewis portion of the Hycroft mine, F. W. Lewis, Inc. owned 52 other properties that were retained by Vista and subsequently transferred to Allied Nevada. F. W. Lewis, Inc. also had a 5% NSR royalty on gold and a 7.5% NSR royalty on silver produced from the Lewis portion of the property. With this purchase of F.W. Lewis, Inc., there is no longer any royalty on gold and silver payable on the previous Lewis ownership.

In 2006, approximately 13,100 acres of additional claims were staked by Vista. These claims were around or contiguous to the original Crofoot and F.W. Lewis claims. In 2008, approximately 22,700 acres of additional claims were staked contiguous to or around the existing Hycroft claims. In 2009, an additional 79 claims were staked contiguous to or around the existing Hycroft land holding. In 2010, an additional 94 claims were staked contiguous to or around the existing Hycroft land holding.

The Crofoot property is held by HRDI, a wholly owned subsidiary of Allied Nevada. A 4% NPI is retained by the original Crofoot owners. In 1996, the lease/purchase agreement was amended to provide for minimum advance royalty payments of $120,000 on January 1 of each year in which mining occurs on both patented and unpatented claims. All payments for the Crofoot property are capped at $7.6 million of which $960,000 has been paid to date, after which Allied Nevada will own the property. An additional $120,000 is due if ore production exceeds five million tons from the Crofoot property, on both patented and unpatented claims, in any calendar year. All advanced royalty payments are available as credit against the 4% NPI. Royalty payments to Crofoot have totaled $960,000 since the amended agreement. Table 4.1 shows the royalty amount and other annual land holding costs.

Table 4.1 Hycroft Land Holdings Costs

Month Due	Lessor	Type	Amount
January	Crofoot	Advance Royalty	$120,000
	U.S. BLM, Humboldt & Pershing Counties, and State of Nevada	Claim Fees	$432,000 $466,385
	Communication Site of Floka Peak	Annual Fee	$1,809

4.4 ENVIRONMENTAL LIABILITIES

Gold production began on the property in 1983 and continued through 1985 when Standard Slag opened the Lewis mine. There was a brief gap in mining until HRDI acquired the Lewis mine and the Crofoot claims and started mining in 1988. Mining operations continued until 1998 when pit development was placed on standby due to low metal prices. The process operations continued until 2004 when the property was placed on care and maintenance.

The Mines Group Inc. of Reno, Nevada, revised and updated reclamation plans and cost estimates for the Hycroft mine in 2003. In January 2004, Vista announced that HRDI had reached an agreement with member companies of American International Group, Inc. (“AIG”) to replace the existing bond at its

Hycroft mine with a new package that includes an insurance component and covers all existing reclamation liability at Hycroft. The Reclamation Plan and bonding includes all the historic mining at the Hycroft property. The bond called for an initial payment of $4.0 million and two additional payments of $1.3 million after six months and 11 months from the initial payment. The bonding instrument was accepted by the BLM, and the insurance/assurance bonding instrument replaced the existing bond made up of a $5.1 million non-cash collateralized bond from American Home Assurance Company, letters of credit of $1.7 million posted directly with the BLM, and the existing indemnity agreement.

The 2004 bond cost estimates were revised and increased by HRDI to $7,549,363 at the end of 2006 and the bond amount increased to the BLM.

Allied Nevada contracted SRK to review the Hycroft Plan of Operations to update the site disturbance using the Geographical Information System programs and July 2007 aerial photograph. The updated disturbance and proposed new disturbance was then loaded into the Nevada Standardized Reclamation Cost Estimator ("SRCE") version 1.1.1, with the current Nevada Cost Data File and Hycroft Interim Fluid Management Plan. The revised project reclamation estimate totaling $14,343,100 was approved by both the NDEP and BLM and the bond amount was increased with the BLM.

In July 2009, the NDEP and BLM approved an update to the Plan of Operations and Reclamation Plan for the inclusion of previously permitted areas not included in the 2007 update. The bond instrument was increased with the BLM to a total of $15,714,789.

In December 2009, the NDEP approved an update to the Reclamation Plan for the inclusion of expanded open pit areas and waste rock facilities located entirely on private land. In February 2010, the BLM approved the associated reclamation cost estimate, and a surety bond increase was submitted on February 8, 2010, bringing the total bond amount to $16,621,357. In March, 2010, the BLM approved the surety bond increase.

4.5 PERMITS

Hycroft mine operates under permit authorizations from the BLM, NDEP, and the Nevada Bureau of Mining Regulation & Reclamation. Allied Nevada has posted a bond for its mining operations at Hycroft. All operating and environmental permits, approved by the BLM and NDEP, are in good standing for mining operations at Hycroft.

Table 4.2 summarizes the operating permits, while Table 4.3 shows the miscellaneous permits for the property.

Table 4.2 Hycroft Operating Permits

Operating Permits	Issuing Agency	Number	Status
Plan of Operations	BLM	N64641	Current
Mercury Operating Permit to Construct	NDEP Bureau of Air Quality Planning	AP1041-2255	Current
Permit to Operate a Public Water System	NDEP Bureau of Safe Drinking Water	HU-0864-12NTNC	Current
Class II Air Quality Permit	NDEP Bureau of Air Pollution Control	AP1041-0034.02	Current
Water Pollution Control Permit-Crofoot Project	NDEP Bureau of Mining Regulation & Reclamation	NEV60013	Current
Water Pollution Control Permit-Brimstone Project	NDEP Bureau of Mining Regulation & Reclamation	NEV94114	Current
Bioremediation Facility Permit	NDEP Bureau of Mining Regulation & Reclamation	GNV041995-HGP15	Current
Reclamation Permit	NDEP Bureau of Mining Regulation & Reclamation	134	Current
Mining General Stormwater Pollution Prevention Permit	NDEP Bureau of Water Pollution Control	R300000: MSW-177	Current
Artificial Pond Permit (Brimstone Process Ponds)	NV Dept of Wildlife	28233	Current
Artificial Pond Permit (Crofoot Process Ponds)	NV Dept of Wildlife	S30630	Current
Dam Safety Permit (Crofoot Process Ponds)	NV Division of Water Resources	J-273	Current
Water Rights (Crofoot Process Well #1)	NV Division of Water Resources	60230	Current
Water Rights (Crofoot Process Well #2)	NV Division of Water Resources	60231	Current
Water Rights (Crofoot Potable Well)	NV Division of Water Resources	49533	Current
Hazardous Materials Storage Permit	NV State Fire Marshall	8826	Current

Table 4.3 Hycroft Miscellaneous Permits

Operating Permits	Issuing Agency	Number	Status
ROW Microwave Repeater; Sec. 29, 30	BLM	NVN46292	Current
ROW Wells/Pipeline/Power Line; Sec. 3	BLM	NVN46564	Current
ROW 2 Wells/Pipeline/Power Line	BLM	NVN46959	Current
ROW Road & Waterline (Old Mancamp to Lewis)	BLM	NVN39119	Current
Kamma Peak Station	FCC	WNER344	Current
Sulfur Mine Station	FCC	WNER345	Current
Winnemucca Mtn. Station	FCC	WNER346	Current
Base Station & 45 Mobile Units	FCC	WNKK336	Current
Class 3 Waivered Landfill Permit	NDEP Bureau of Waste Management	SWM1-08-11	Current

Operating and miscellaneous permits that require annual maintenance fees are shown in Table 4.4. Fixed annual fees are required for the stormwater and the public drinking water system permits based upon the current Nevada regulatory structure. The other annual fees are based on annual mining production, quantities, and types of chemicals stored on site, existing and permitted surface disturbance, and the level of actual and permitted air emissions. The variable fees shown are based upon the current operational conditions.

Table 4.4 Hycroft Permits

Permit and Fee Description	Annual Amount
Air Quality Operating Permit	$5,000
Reclamation Permit	$21,000
Stormwater Permit	$750
Artificial Pond Permit	$10,000
Water Pollution Control Permit	$40,000
State Fire Marshall	$3,050
Public Drinking Water System	$225
TOTAL	**$80,025**

Hycroft currently holds four ROW leases with the BLM, as described in Table 4.3. The fee and renewal schedule is shown in Table 4.5.

Table 4.5 ROW Payment and Renewal Schedule

ROW Number	Payment Amount (estimated)	Payment Date	Expiration Date
NVN46292	$2,200	01/10/10	09/09/17
NVN46564	$55	01/01/12	09/22/17
NVN46959	$350	01/01/12	09/29/17
NVN39119	$220	01/10/10	06/25/14

4.6 HYCROFT EXPANSION PERMITTING

Allied Nevada submitted a Plan of Operations amendment for the expansion of the Hycroft mine in April 2010 to the BLM. The proposed action includes the expansion of the following facilities: existing open pits – Brimstone, Bay Area, Boneyard, Central and South Central; existing Brimstone heap leach; and existing waste rock facilities. A new heap leach and waste rock facility is proposed in the southern end of the project area. The draft proposed action currently includes the continued processing of ore via heap leaching and open pits that remain above the groundwater table. Open pits would be backfilled to the extent possible. The submittal of the Plan of Operations to the BLM has initiated a National Environmental Policy Act review of the proposed action. The BLM has determined that an Environmental Impact Statement ("EIS") will be performed.

Continuing operations at Hycroft will include the crushing of ore for placement on the currently permitted Brimstone heap leach facility. The air quality operating permit has been received.

A proposed expansion of the existing operation would require the modification of other permits in addition to the Plan of Operations. The air quality permit is anticipated to be modified for changes to the current portable crushing layout and the expansion of the existing refinery. The Brimstone Water Pollution Control Permit would be modified to include a waste rock characterization and placement plan, new reagent storage areas, the expansion of the Brimstone heap leach facility, and the addition of the proposed South heap leach facility. Allied Nevada is anticipated to refine the design of the proposed process components and submit the Water Pollution Control Permit and Air Quality Operating Permit modifications to the NDEP in 2011. These permit applications would include updated management plans as needed.

4.7 CROFOOT HEAP LEACH FACILITY CLOSURE

Allied Nevada submitted a Final Permanent Closure Plan ("Plan") on July 9, 2009, to the NDEP for the Crofoot processing facilities permitted in Water Pollution Control Permit NEV60013. Facilities proposed for permanent closure include the Crofoot heap leach and associated processing components. The NDEP has approved the activities associated with the closure of the process plant and ponds.

Allied Nevada has initiated the scheduled Plan activities which include closure of the Crofoot barren and diatomaceous earth ponds, characterization of soils in the vicinity of the Merrill-Crowe plant and refinery, construction of a new drain-down collection system along the western heap leach pad boundary, and removal of the Merrill-Crowe plant and refinery equipment in 2010 and 2011. Future activities include re-grading and placement of growth medium on the heap leach pad.

5 ACCESS, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS

Hycroft and its related facilities are located 54 miles west of Winnemucca, Nevada. Access to the Hycroft mine from Winnemucca, Nevada is by means of State Road No. 49 (Jungo Road), a good quality, unpaved road. Access is also possible from Imlay and from Lovelock by dirt roads intersecting Interstate 80. The majority of the mine's employees live in the Winnemucca area. Winnemucca (population 15,000) is a commercial community on Interstate 80, 164 miles northeast of Reno, Nevada. The town is served by a transcontinental railroad and has a small airport. There is access to adequate supplies of water and power. A major east-west railway passes through the Hycroft claim position.

5.2 CLIMATE

The climate of the region is arid, with precipitation averaging 7.6" per year. The majority of the precipitation occurs in the winter and spring months and again in October.

Temperatures during the summer are generally in the 50° F at night and near 90° F and above during the days. Winter temperatures are usually in the 20° F at night and in the 40° F during the day. There is strong surface heating during the day and rapid nighttime cooling because of the dry air resulting in wide daily ranges in temperatures. The average range between the highest and lowest daily temperatures is about 30 to 35° F. Daily temperature ranges are greater in summer than the winter.

Winds are generally light. Dust or sand storms occur occasionally, particularly during the spring. The mine is generally not known to have major delays in production due to inclement weather.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The mine is situated on the eastern edge of the Black Rock Desert and has alkali tolerant, low shrub vegetation.

Water rights, as listed in the Process Management Plan, are shown in Table 5.1. The Near and Far Process wells and the Crofoot potable well are the main sources of water for the mine site.

The mine site has a truck shop, ore processing facilities, and an administration building as well as other service related structures. Electricity is furnished from nearby power lines and there is a telephone system.

Table 5.1 Hycroft Water Wells and Permitted Yearly Consumption

Application No	Certificate No	Consumption (cfs)	Annual Consumption Limit (gallons)	Annual Consumption Limit (ac-ft)	Point of Diversion
18580	5590	0.96	226,400,000	694.80	T35N R29E S34
19477	6040	0.891	Not specified	Not specified	T35N R30E S03
49533	13859	0.4	4,831,000	14.83	T34N R29E S31
51112	13457	3.2	236,171,000	724.78	T35N R29E S31
51113	13458	3.2	98,871,000	303.42	T35N R29E S31
60230		2	471,832,827	1,448.00	T35N R29E S31
60231		2	471,832,827	1,448.00	T35N R29E S31
61255	15722	0.0557	2,179,946	6.69	T36N R36E S31
61256	15723	0.0557	1,697,686	5.21	T35N R32E S16
47423	13448	0.34	57,970,000	177.90	T35N R30E S20
42675	CANCELLED	0.891	96,000,000	294.61	
46794	12237	0.05	10,800,000	33.14	T35N R30E S20
Total			1,678,586,286		

5.4 PHYSIOGRAPHY

The mine is situated on the eastern edge of the Black Rock Desert and on the western flank of the Kamma Mountains between Winnemucca and Gerlach, Nevada.

The Black Rock Desert is a 400 square mile flat, prehistoric lake bed, completely devoid of any vegetation or animal habitat. Its name comes from a large, prominent, dark rock formation located at the north end of the desert. During the summer, the lake bed is primarily a hardpan alkaline playa. During some winters, it may become a temporary lake.

There are no streams, rivers, or major lakes in the general area. Elevations range from 4,500 to 5,500 ft above sea level.

6 HISTORY

6.1 PROPERTY HISTORY

One small and two large open pit operations comprise the Hycroft mine. The mine was formally known as the Crofoot Lewis open pit mine. Mining began in 1983 with a small heap leach operation known as the Lewis mine. Lewis mine production was followed by production from the Crofoot property in the Bay Area, South Central, Boneyard, Gap and Cut-4 pits along the Central fault. Production from the north end of the Brimstone pit continued until it was placed on a care and maintenance program in December 1998 due to gold prices below $300/oz.

The Lewis mine was acquired by Vista in early 1987 from F. W. Lewis, Inc., and the Crofoot mine was acquired by Vista in April 1988. The leasehold interest in the Lewis property was purchased by Vista on December 13, 2005 in consideration of the payment of $5.1 million and the elimination of the 5% NSR royalty on gold and 7.5% NSR royalty on silver produced from the property. The Hycroft mine produced over one million ounces of gold and over two million ounces of silver from the commencement of mining operations in 1987 until the operations were suspended. Gold production from the leaching and rinsing of the heap leach pads continued in 2000 through 2005 and then the mine was placed on care and maintenance.

In May 2007, Allied Nevada acquired the Nevada based holdings of Vista. The Hycroft mine was included as part of the transfer of ownership allowing Allied Nevada to explore, expand, and develop the resources at Hycroft.

In September 2007, the Board of Directors of Allied Nevada approved the reactivation of the Hycroft mine. A used mining fleet was purchased and refurbished at a minimal capital cost, modifications to site facilities were completed, and mining operations resumed in the third quarter of 2008. The first gold pour occurred in December 2008. Mining is currently being accomplished with open pit mining methods, and the ROM heap leach process is being used to recover gold from the ore. Solution from the heap leach process is further processed using CIC and Merrill-Crowe process methods. Silver is produced as a by-product of the gold extraction process.

Since start up, two phases of leach pad expansion have been constructed bringing the total available leach pad space of the Brimstone leach pad to 6.6 million ft^2 by mid 2010. The fully permitted expansion of 9.1 million ft^2 will be complete by the end of 2010. In addition, a new refinery was built to allow for refining of gold and silver from the Merrill-Crowe process on site to reduce the time required to prepare doré for sale. In the third quarter of 2008, mining operations began at Hycroft and initial gold production was achieved in December 2008.

Allied Nevada determined, in 2008, that there was a well defined oxide and sulfide resource outside of Brimstone and Boneyard based on an analysis of all known drilling at the mine site. Allied Nevada has also pursued a successful campaign of deep exploration drilling that has identified a large sulfide mineralization system below the Brimstone and Cut-5 deposits, and at the newly discovered Vortex Zone immediately south of Brimstone.

The earliest recorded mining in the Sulfur District began in the late 1800s following the discovery of significant native sulfur deposits (Couch and Carpenter, 1943; Willden, 1964). Mining of native sulfur was sporadic during the 1900s, with the last significant episode of mining occurring in the 1950s. Based on historical reports, a total of over 181,488 tons of sulfur ore, grading approximately 20-35% sulfur, was mined and milled (McLean, 1991). High grade silver mineralization, consisting of nearly pure seams of cerargyrite (AgCl) plus alunite, was discovered in 1908 at Silver Camel Hill (Vandenburg, 1938). Assays up to 3,439 opt and 0.362 opt gold were reported by Jones (1921). Silver mining ceased by 1912 with a total estimate of 165,374 ounces produced. Minor silver mining has also occurred along the East fault in the Snyder adit region, and silver samples as high as 66 opt were reported by Friberg (1980) and 29 opt by Bates (2000). The stope along the Snyder adit is about 50 ft in length, 10 ft in width, and 100 ft in dip extent. An estimated 2,500 tons was mined at an unknown grade between 1932 and 1937.

During the First World War, three 6-8 ft wide veins of nearly pure alunite were mined in the southern part of the Sulfur District (Clark, 1918). In 1931, several hundred tons of alunite was mined as a soil additive (Fulton and Smith, 1932). Vandenburg (1938) estimated that 454 tons of alunite was shipped to the west coast to be used as fertilizer. From 1941 to 1943, cinnabar was mined from small pits (Bailey and Phoenix, 1944) in the exposed acid sulfate alteration zone. Total mercury production during this period is estimated at 1,900 lbs (McLean, 1991).

In 1966, the Great American Minerals Company began extensive exploration for native sulfur. Approximately 200 shallow holes were drilled and numerous trenches dug (Friberg, 1980). In 1974, the Duval Corporation ("Duval") drilled 20 holes on the property in search of a Frasch type sulfur deposit (Wallace, 1980). Duval found no evidence of a sulfur deposit at depth, but did report elevated gold and silver values. Duval drilled two core holes (DC-1 and DC-2) and 18 rotary holes (DR-3 through 20) (Ware, 1989). In 1977, Cordex Syndicate mapped and rock chip sampled the property, recognizing the potential for a bulk tonnage, low grade precious metal deposit. In 1978, Homestake Mining ("Homestake") became interested in the property recognizing similarities with the McLaughlin hot springs deposit in California. Numerous surface samples were taken and 112 holes drilled (Friberg, 1980), but the option was dropped because of low grades and limited extent. Homestake drilling consisted of eight core holes, (SC-81-1 through 8), nine air track holes (AT-1 through 9) and 95 rotary holes (SR81-1 through 95). In 1983, Standard Slag acquired the Lewis option of the North pit (along the Central fault), which contained a historical, non 43-101 compliant resource of 1.2 million tons at 0.035 opt. Production by Standard Slag commenced at the Lewis mine in 1983 and continued until 1985.

The Crofoot deposit, adjoining the Lewis mine, was discovered in 1985. HRDI acquired the Crofoot claims and the Lewis mine in 1986. Allied Nevada acquired the Crofoot-Lewis mine in 2007.

6.2 EXPLORATION DEVELOPMENT AND HISTORY

Between 1985 and 1999, HRDI drilled 3,123 exploration drill holes totaling 943,822 ft. The current Hycroft drill hole database consists of the former holes, plus 61 RC holes drilled by Homestake in 1982, and 29 rotary holes completed by Homestake in 1981. Canyon Resources drilled a 33 hole program to test extensions of oxide mineralization both laterally and at depth in proximity to the Brimstone deposit. Grade estimates for the current reserves and resources include data from these holes. The historic drilling campaigns are summarized in Table 6.1 by year, operator, and drilling type.

Table 6.1 Historic Drilling

Year	Hole Type	Company	No. of Holes	Footage	Zones Drilled
1981	Rotary	Homestake	29	5,550	North, SC
1982	RC	Homestake	61	10,015	North
1985	RC	Hycroft	195	33,482	North, Cut-4, SC
1986	RC	Hycroft	492	96,877	North, Cut-4 ,SC ,Gap, Brim, Alb
1987	RC	Hycroft	632	138,385	Alb, Cut-4, Gap, North, SC
1988	RC	Hycroft	73	25,855	Alb, Brim, Cut-4, North, SC
1989	RC	Hycroft	43	15,780	Alb, Brim, Cut-4, North, SC
1990	DD	Hycroft	8	11,247	Cut-4, Sulfur
1990	RC	Hycroft	134	52,675	Alb, Brim, Cut-4, North, SC
1991	RC	Hycroft	147	44,360	Cut-4, North, SC
1992	RC	Hycroft	265	83,030	Alb, Brim, Cut-4 ,North, SC
1993	DD	Hycroft	6	2,318	Alb, Brim, SC
1993	RC	Hycroft	297	105,500	Alb, Brim, Cut-4, North, SC
1994	DD	Hycroft	3	4,990	Brim
1994	RC	Hycroft	208	78,650	Alb, Brim, Cut-4, Boneyard, SC
1995	RC	Hycroft	355	157,515	Alb, Brim, Cut-4, Gap, Boneyard, SC
1996	DD	Hycroft	1	1,078	Brim
1996	RC	Hycroft	164	75,000	Alb, Brim, Cut-4, North, SCP
1997	RC	Hycroft	13	3,040	Brim, Boneyard
1998	Blasthole	Hycroft	67	3,670	Brim
1999	DD	Hycroft	9	4,870	Brim
1999	RC	Hycroft	11	5,500	Brim
2005	RC	Vista	33	13,315	Brim
2006	RC	ANV	1	900	Brim
2007	RC	ANV	14	14,944	Alb, Brim
2007	DD	ANV	38	42,930	Alb, Brim, Bay
2008	RC	ANV	281	181,810	Alb, Brim
2008	DD	ANV	60	60,810	Alb, Brim, Bay
2009	DD	ANV	50	39,989	Bay, Cut-4, Vortex, Brim
2009	RC	ANV	79	54,325	Bay, Vortex, Brim
2010	RC	ANV	95	69,217	Bay, Vortex, Brim, Alb
2010	DD	ANV	58	69,472	Bay, Vortex, Brim, Alb
Total			**3,922**	**1,507,099**	

Exploration by Homestake resulted in the discovery of seven zones of mineralization. These are described in detail in the exploration section of this document.

6.2.1 BAY AREA

The Bay Area is a large blanket of oxide mineralization hosted by interbedded sinters and conglomeritic to sandy debris flows (Upper Camel Group). The Bay Area is located at the northwest sector of the district, and extends for 2,000 ft in a north-south direction along the Central fault, between 49,000N and 51,000N. Mineralization extends as far as 2,500 ft to the west of the Central fault. The Bay Area was the focus of exploration drilling during 1985-1987, and can be thought of as the western extension of the Lewis mine, which was the area partially mined by Standard Slag during 1983-1985. Alteration associated with gold values is an assemblage of replacement opal K-spar chalcedony pyrite. Oxidation forms an 80-100 ft thick blanket over the sulfide mineralization. This area was drilled out as the first reserve on the project.

In 2009, 65 rotary and core holes were drilled to: (a) delineate the depth of overlying fill material, (b) delineate favorable host stratigraphy, and (c) provide material for metallurgical test work.

6.2.2 CENTRAL FAULT, SOUTH CENTRAL, GAP, CUT-4 AND CUT-5 DEPOSITS

These deposits occur along a length of 10,000 ft in the immediate hanging wall of the Central fault. All the deposits are composed of oxidized acid leached Camel Conglomerate. This unit is composed of clasts of Triassic Auld Lang Syne ("ALS") sediments, and Tertiary Kamma volcanic rocks. The Camel Conglomerate has been altered to an opal K-spar pyrite assemblage and subsequently was oxidized to a clay hematite or silica alunite assemblage.

The South Central deposit was mined first after the Bay Area, and extends from approximately 42,000N to 46,000N; the Gap was mined second and extends from 46,000N to 49,000N. Cut-4 was mined last along the Central fault, and extends from 39,000N to 42,000N. Cut-5 is a southerly extension of the Cut-4 deposit.

In the Cut-5 and Silver Camel areas, drilling to test shallow oxide and deep sulfide potential was conducted in 2007 and 2008. In 2009, approximately 20 core holes were drilled to provide material for metallurgical testing; in 2010 35 RC and two core holes were drilled for metallurgical testing and to infill resource areas.

6.2.3 THE BONEYARD DEPOSIT

This deposit strikes north-northeast and is located approximately 1,000 ft east of the Bay Area. This deposit is similar in lithology and alteration to the Central fault deposit.

The deposit is about 2,000 ft long and extends in a north-northeast direction from 20,300E to 48,500N. The deposit was mined concurrently with the Gap deposit.

6.2.4 THE BRIMSTONE DEPOSIT

The Brimstone deposit is hosted in rhyolitic, aphanitic and tuffaceous Kamma volcanic rocks in the southeastern part of the Crofoot-Lewis mine area. The deposit consists of two major zones of hydrothermal venting, displaying fracture and breccia controlled chalcedony sulfide mineralization as veinlets, hydrofracture fill, and chaotic hydrothermal breccias. The deposit has been oxidized by an acid leach/oxidizing event**.** Disseminated native sulfur was deposited in the acid leached material.

The system extends from at least 40,000N to 45,000N in the hanging wall of the west dipping, normal East fault. Production records show 15,500,000 tons of ore was mined from the Brimstone deposit with an average cyanide soluble grade of 0.014 opt Au.

Drilling to test shallow oxide and deep sulfide potential was conducted in 2007 and 2008. In 2009, 11 rotary and core holes were drilled for infill and to provide material for metallurgical testing. In 2010, four core and eight RC holes were drilled for geotechnical studies and to infill resource areas.

6.2.5 THE VORTEX ZONE

The Vortex Zone was discovered in 2008 as a result of testing a geophysical anomaly. Ten holes were drilled as part of the discovery phase. In 2009, 30 rotary and core holes were drilled as part of an infill program over an area of 2,000 x 2,000 ft. In 2010, 32 core and 11 RC holes were drilled for exploration step-out and infill. Mineralized core also was submitted for metallurgical testing.

6.2.6 THE ALBERT ZONE

Albert mineralization is located between the Central fault and Brimstone deposits. The mineralization is hosted in both sedimentary and volcanic rocks.

The north striking, west dipping Albert fault down drops, approximately 300 vertical ft, the dominantly sedimentary Camel Conglomerate to the west from Kamma volcanic rock in the footwall of the Albert fault to the east.

Deep drill holes in the Albert Zone suggest a deep unconformity between the Kamma volcanic rocks and the Camel Conglomerate above. There has been no historic mining in the Albert Zone. In 2010, Allied Nevada drilled four RC and one core hole for exploration purposes.

6.3 PRODUCTION HISTORY

Information on the production history of the Hycroft mine comes from Allied Nevada's in-house documents. Production by Standard Slag commenced at the Lewis mine in 1983 and continued until 1985. Ore from the Lewis mine was crushed and stacked on the Lewis pads in the north-central part of the district. Lewis mine production was followed by production from the Bay Area, South Central, Boneyard, Gap and Cut-4 pits along the Central fault, and finally the north end of the Brimstone pit, as outlined below in Table 6.2.

Table 6.2 Pre-Allied Nevada Historic Production

Deposit	Years Mined (approximate)	Tons (millions)	Cyanide Soluble Grade opt Au	Ounces Au Produced
Lewis Mine	1983-1985	3.9	N/A	N/A
Bay Area	1988-1992			
South Central	1992-1995			
Boneyard	1992-1993			
Gap	1994-1995			
Cut-4	1994-1997			
Total Central Fault Production		66.7	0.0163	877,460
North Brimstone	1996-1998	15.4	0.0143	175,954
Hycroft Mine Production		**82.2**	**0.0159**	**1,053,414**

The Central Fault deposits were either crushed to 80% passing 3/8" or treated as ROM, depending on the blast hole grade. The Central Fault production was placed on a series of leach pads referred to as pads 1-3. Pads 1 and 2 were constructed in 1987, and pad 3 was constructed in 1992. Ore placement was made on pad 1 from 1988-1997, on pad 2 from 1989-1997 and on pad 3 from 1993-1997. Solutions from the pads were treated in a Merrill-Crowe plant (Crofoot plant) located on the northeast side of pad 1. Since 2000, solutions have been run through a CIC plant located on the northwest side of pad 1.

Detailed records are not available on historic reserve modeling in the Central Fault and Brimstone deposits, but detailed records are available for the pad loading from these deposits. From 1988-1997, a total of 82.2 million tons of ore were placed on all pads, with average cyanide soluble gold grade of 0.016 opt Au or 1.31 million ounces of gold placed. A total of 1,053,414 ounces of gold has been recovered through 2003, as shown in Table 6.3.

Table 6.3 Historic Pad Production

Year	Hycroft Pad Loading (Tons 000s)					Ore Tons (000s)	Waste Tons (000s)	CnAu (opt)	Total Ounces Loaded (000s)						Au oz Recovered (000s)
	Pad 1	Pad 2	Pad 3	Pad 4	Pad 5				Pad 1	Pad 2	Pad 3	Pad 4	Pad 5	Total	
1988	3,995.4	104.0				3,995.4	2,450.3	0.021	82.1					82.1	38.1
1989	5,144.8	1,792.4				5,248.8	5,682.7	0.019	98.4	2.0				100.4	73.6
1990	3,793.9	5,309.9				5,586.3	8,276.0	0.019	73.3	34.8				108.1	89.3
1991	490.3	5,665.4				5,800.2	8,182.7	0.019	9.2	99.4				108.6	92.6
1992	428.1	4,610.4				6,093.5	9,884.2	0.017	7.2	95.1				102.3	99.1
1993	588.7	3,066.4	521.1			5,720.2	16,765.4	0.018	10.7	87.0	7.9			105.6	86.5
1994	488.4	4,577.7	5,683.2			9,238.0	17,460.5	0.015	7.8	42.2	89.7			139.7	94.9
1995	463.8	3,671.3	4,890.0			9,931.5	27,263.6	0.014	6.5	53.6	78.8			138.9	101.1
1996	2,337.1	478.8	5,843.3	1,027.8		12,879.5	23,822.1	0.013	23.2	35.2	91.5	11.6		161.5	89.4
1997	664.3		2,140.9	4,632.7	2,686.2	10,602.9	26,772.1	0.015	13.1	9.3	30.9	64.8	38.0	156.1	117.4
1998				5,469.6	1,647.9	7,117.5	3,009.3	0.015				82.8	24.0	106.8	112.7
1999															40.1
2000															13.0
2001															3.2
2002															1.8
2003															0.6
2004															
2005															
2006															
2007															
2008				2,405.7		2,405.7	10,097.7	0.007				16.5		16.5	
2009				9,771.7		9,771.7	17,214.9	0.022				214.2		214.2	42.4
2010						5,696.9	6,701.9								50.0
	18,394.8	29,276.3	19,078.5	23,307.5	4,334.1	100,088.1	183,583.4	0.016	331.5	458.6	298.8	389.9	62.0	1,540.8	1,145.8

Production from the Brimstone pit was placed directly on leach pads 4 and 5 as ROM. Pad 5 consists of extra lifts placed on top of pads 1 and 2. Pad 4, constructed immediately south of the old Lewis pad, was completed in 1996. Loading of pads 4 and 5 commenced in October 1996 and July 1997, respectively. A 2,800 gpm Merrill-Crowe leach solution plant was completed and put into operation in February 1997. This is referred to as the Brimstone plant. The plant treats solutions from pad 4 and is located on the northwest side of the pad. Pad 5 solutions were treated in the older Crofoot plant.

In May of 2007, Allied Nevada began the permitting and economic evaluation required to reopen the Hycroft operation. In 2008, Hycroft began staffing the mine, purchased an equipment fleet, initiated pre-development stripping of the Brimstone deposit, and constructed a 1.2 million ft^2 expansion of pad 4. Two small CIC trains with a maximum solution flow capacity of 1,400 gpm were added to the Brimstone plant. Doré was shipped to the refinery in December of 2008. Pad 4 was expanded again in

2009 with 2.4 million ft^2 of liner constructed. Permitting was received and construction of a new refinery was completed at the Brimstone plant site by June of 2009. The mine achieved planned ore production capacity by the end of 2009.

6.4 HISTORICAL RESOURCE AND RESERVE ESTIMATES

Prior historical resource estimates were completed by MRDI as part of their work for Vista in May 2000. MRDI then used the model to re-estimate gold resources using the MRDI adjusted gold and silver database and a new geological interpretation of ore types. **Note that these historical resources which were prepared before February 1, 2001, are not compliant with NI 43-101 since the resource categorization was not addressed and should not be relied upon**; however, MRDI has had a long reputation of producing reliable mineral estimates.

Mineralized blocks with an estimation variance of 0.36 or less were considered to be measured and blocks between 0.36 and 0.47 were considered to be indicated. Blocks with an estimation variance in excess of 0.47 were considered to be inferred. The resource was classified primarily on the basis of estimation variance because it reflects the spatial distribution of the data, not just the distances. The historic Brimstone resource estimate includes material found between the $450 gold floating cones and the $375 gold designed pit and may be considered to be economically mineable at higher gold prices. The historic resource estimate is shown in Table 6.4. The historic resource is summarized using a 0.005 opt cyanide soluble gold cut off grade. The grades shown in Table 6.4 are the estimates generated by multiple indicator kriging of the cyanide soluble gold values. Table 6.5 summarizes the historic inferred resource estimate. Table 6.6 summarizes the historic ore reserve.

The ore model and resource estimates were re-evaluated by Ore Reserve Engineers ("ORE") in 2005 and reserves re-estimated by Mine Development Associates in 2006. Detail of the resource and reserve estimates are included in Section 17 of this report.

Table 6.4 Historic May 2000 Resource

Category	Tons	Cyanide Soluble oz Au/ton	Cyanide Soluble 000s oz Au	Fire Assay oz Au/ton	Contained 000s oz Au
Measured	23,287,000	0.0132	307.6	0.0165	385.1
Indicated	24,192,000	0.0127	307.3	0.0153	369.4
Totals	**47,479,000**	**0.013**	**614.9**	**0.0159**	**754.6**

Table 6.5 Historic May 2000 Inferred Resources

Category	Tons	Cyanide Soluble oz Au/ton	Cyanide Soluble 000s oz Au	Fire Assay oz Au/ton	Contained 000s oz Au
Inside Historic Designed Pit	5,210,000	0.0126	65.4	0.0154	80.4
Outside Designed Pit	6,819,000	0.008	54.7	0.0078	53.2
Totals	**12,029,000**	**0.01**	**120.1**	**0.0111**	**133.6**

Table 6.6 Historic Mineral Reserve Estimate

	Tons (000s)	Cyanide Soluble oz Au/ton	Cyanide Soluble oz Au (000s)	Fire Assay oz Au/ton	Contained oz Au	Waste Tons (000s)	Total Tons (000s)	Strip Ratio
Proven	16,269.0	0.0144	234.0	0.0181	293.0			
Probable	16,160.0	0.0139	224.2	0.0169	273.5			
Totals	**32,429.0**	**0.0141**	**458.2**	**0.0175**	**566.5**	**57,796**	**90,225**	**1.78**

7 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Hycroft mine is located on the western flank of the Kamma Mountains in the basin and range physiographic province of northwestern Nevada astride the Humboldt and Pershing County lines.

The Kamma Mountains were formed during Miocene to Quaternary time from the uplift of Mesozoic basement rocks and Tertiary volcanic rocks along north to northeast trending normal faults. The stratigraphy along the western flank of the range steps down westward along a series of these normal faults. The faults also served as conduits of hydrothermal fluids that formed a series of gold and silver deposits in the Sulfur District.

The Hycroft property consists of Tertiary- to Recent-age, fault controlled, low sulfidation gold deposits that occur over an area measuring approximately three miles in a north-south direction by two miles in an east-west direction. The deposits are hosted in volcanic rock eruptive breccias, flows, and conglomerates associated with the Tertiary Kamma Mountain volcanic event. The volcanic rocks are mainly acidic to intermediate tuffs, flows, and coarse volcaniclastic rocks. Volcanic rock fragments dominate the clasts in the eruptive breccias.

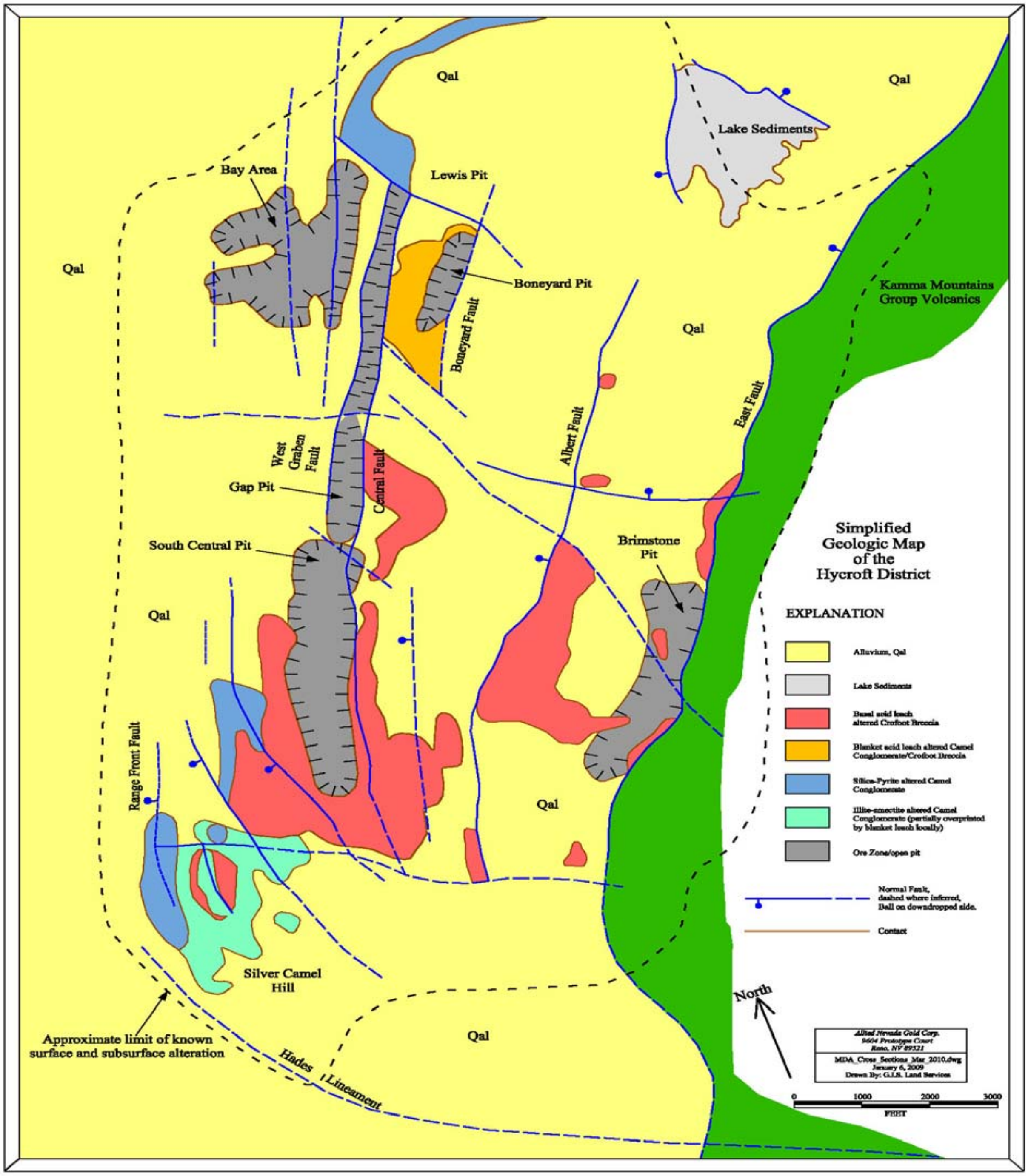
Volcanic rocks have been block faulted by dominant northeast trending structures, which have affected the distribution of alteration and mineralization. The Central fault and East fault control the distribution of mineralization and subsequent oxidation. A post-mineral range front fault separates the gold and silver deposits from the adjacent Pleistocene Lahontan Lake sediments in the Black Rock Desert. The geological events have created a physical setting ideally suited to the open pit, heap leach mining operation at the Hycroft mine.

Five major north-northeast trending, west dipping, normal fault zones appear to broadly control the location of gold and silver mineralization (Figure 7.1). From west to east, these fault zones are referred to as the Central, Boneyard, Albert, Fire, and East faults. Figure 7.2 ‘A’ is a north looking section through the Hycroft mine showing structures and volcanic rock stratigraphy. Figure 7.2 ‘B’ outlines structures and alteration types in the same area. There are also several other parallel fault zones that may have a significant impact on the localization of mineralization. The depth of oxide and mixed sulfide/oxide gold and silver mineralization varies considerably over the area.

Rocks on the western portion of the district are Tertiary conglomerate, siltstone, and fanglomerate of the Sulfur Group (locally termed “Camel Conglomerate”). These rocks are sediment formed from erosion of the underlying Kamma Mountains Group (“KMG”) and Mesozoic basement ALS sediment. The Sulfur Group is divided into three main units: the clast supported coarse conglomerate, the matrix supported conglomerate, and the underlying tuffaceous lake sediment. Felsic tuff, eruption breccia, fanglomerate, aphanitic, and flow banded rhyolite of the KMG are present in the eastern portion of the deposit.

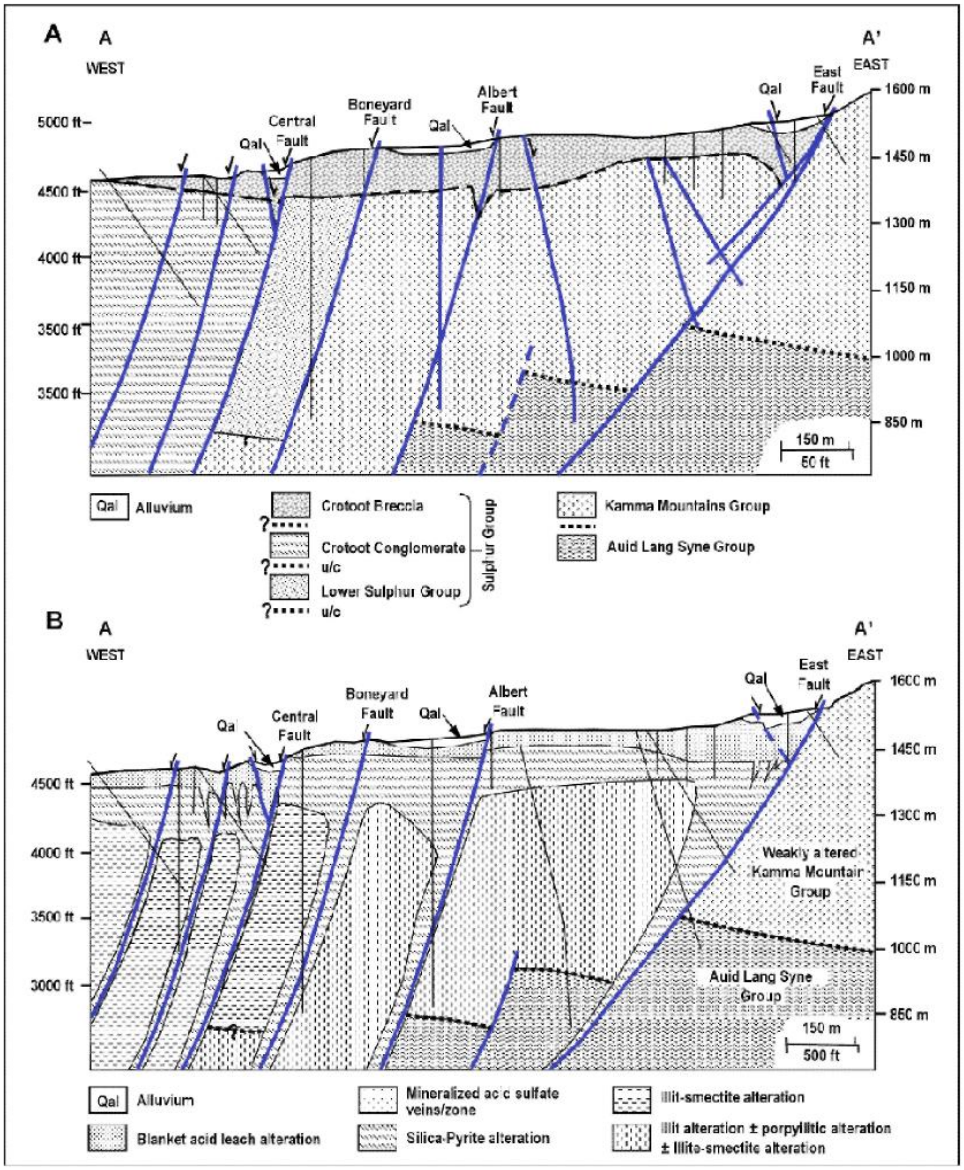
The Lewis, Bay Area, South Central, Cut-3, and Cut-4 deposits (Central Zone deposits) are located in the hanging wall of the Central fault and are hosted by sedimentary rocks of the Sulfur Group.

Figure 7.1 Simplified Geological Map of the Sulfur District



Mineralization in the Albert Zone is present along the Albert fault, located between the Central Zone and Brimstone deposits. The Albert Zone is hosted by fragmental rocks and volcanic rock flows in the hanging wall of the Albert fault.

Figure 7.2 Simplified East-West Cross Sections Through the Sulfur District



A) Simplified East-West Cross Section A-A' Through the Sulphur district.
B) Simplified East-West Cross Section A-A' through the Southern Sulphur district.

From Mineral Resources Development, Brimstone Restart Report, June 2002

The major mineralizing structure bordering the Kamma Mountains is the East fault, which strikes northeast and dips moderately west. The Brimstone deposit occurs in the hanging wall of the East fault.

A post mineral range front fault separates the Hycroft gold and silver deposits from Pleistocene Lahontan Lake sediments in the Black Rock Desert to the west. Recent alluvium overlies bedrock in the district.

7.2 BRIMSTONE DEPOSIT

The Brimstone deposit is hosted by KMG volcanic rocks which are present in the hanging wall of the East fault. The volcanic rocks are principally eruption breccias, tuffs, and volcanic rocks flows proximal to vents and overlie deformed and metamorphosed shale, sandstone, and siltstone of the Mesozoic ALS Group. KMG volcanic rocks are strongly altered in the hanging wall of the East fault, whereas the same units are weakly altered to the east in the footwall of the fault. The stratigraphy at Brimstone includes up to a 100 ft of alluvium (Figure 7.3), underlain by Camel Conglomerate rocks or volcanic rock breccia, all of which overlay Kamma Mountain volcanic rocks (Figure 7.4).

The hanging wall rocks of the East fault include interbedded hydrothermal eruption breccias and conglomerates. This breccia is matrix supported with clasts dominated by Kamma Mountain volcanic rocks, and is not found in the footwall rocks of the East fault. Locally, clasts of oxidized fracture and breccia controlled chalcedony sulfide gold and silver mineralization are observed in the eruptive breccia, indicating a possible syn- or post-mineralization, steam dominated eruption event.

The East fault is a north-northeast striking, west dipping, normal fault with repeated episodes of movement. Where exposed in the Brimstone pit, the fault clearly shows steep normal movement, with slickensides that plunge 80-85°. The fault may have originally served as a conduit to hydrothermal fluids, but most observed movement is post mineral, especially in the North Brimstone pit.

Host rocks were highly altered by at least four phases of alteration. The relatively porous conglomerate and breccias were preferentially acid leached by late stage steaming hydrothermal acid vapors. Acid leach alteration extends to depths of several hundred feet in some areas of the Brimstone deposit (Figure 7.3), indicating that the water table was below the acid leached zone. A siliceous layer (basal acid leach), up to tens of feet thick, occurs at the base of the acid leached material. Underlying the acid leaching is a layer of hydrothermal clay alteration, followed by silica potassium feldspar alteration. Pervasive silicification, veining, and hydrothermal brecciation are primarily restricted to the volcanic rocks and tuffs.

Gold and silver are spatially associated with fracture and breccia controlled chalcedony sulfide mineralization. A subsequent acid alteration event produced the current distribution of oxidized and mixed sulfide oxide ore. The lower acid leach material hosts gold and silver mineralization, as does the underlying silicified and veined KMG rocks. Silicified and brecciated vein material up to 60 ft thick along the East fault hosts mineralization higher in grade than the deposit as a whole. Intersections between steeply dipping, northeast trending faults and the East fault resulted in zones of host rock which were preferentially mineralized.

Drilling has shown that mineralization extends to over 1,200 ft in the Brimstone deposit. Substantial sulfide mineralization underlies the oxide ore currently being mined.

North of the Brimstone deposit, the intensity and nature of the alteration significantly decreases. Condemnation drilling of the leach pad to the north has shown local zones of weak gold and silver mineralization. In the footwall of the East fault, rocks are exclusively KMG dominated by flow banded rhyolite, epiclastic tuffs of felsic composition, and intermediate to mafic andesite and basalt. Alteration and oxidation of these volcanic rocks is weak, with oxidation occurring within 50-150 ft of the East fault, and propylitic or clay alteration extending further.

Linear zones of silicification of limited thickness oriented parallel to the East fault are present in the footwall zone. These may be related to high angle northeast striking faults.

Figure 7.3 Brimstone North Pit Wall Geology

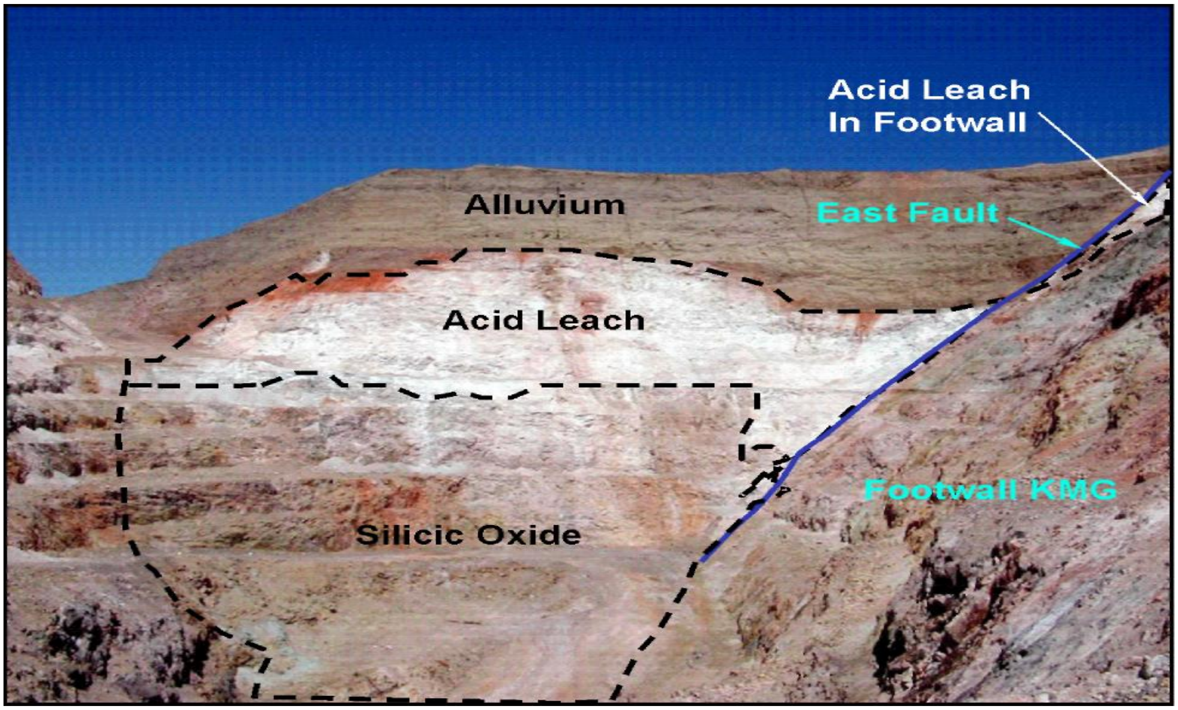
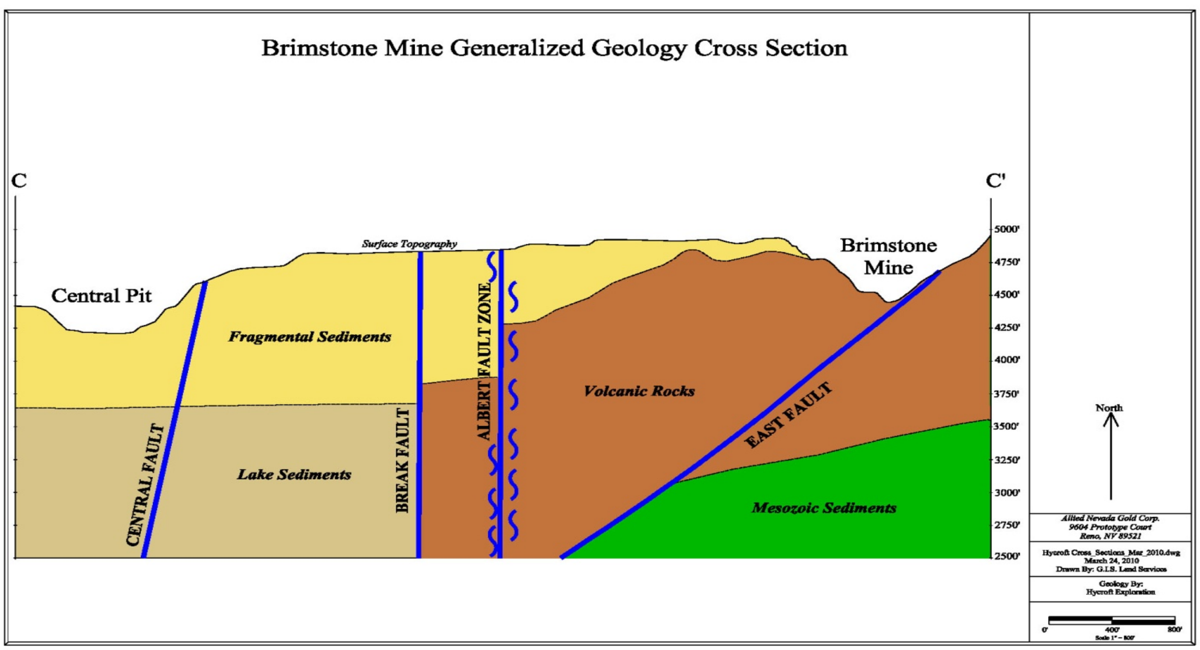


Figure 7.4 Brimstone Generalized Geology Cross Section



7.3 THE VORTEX ZONE

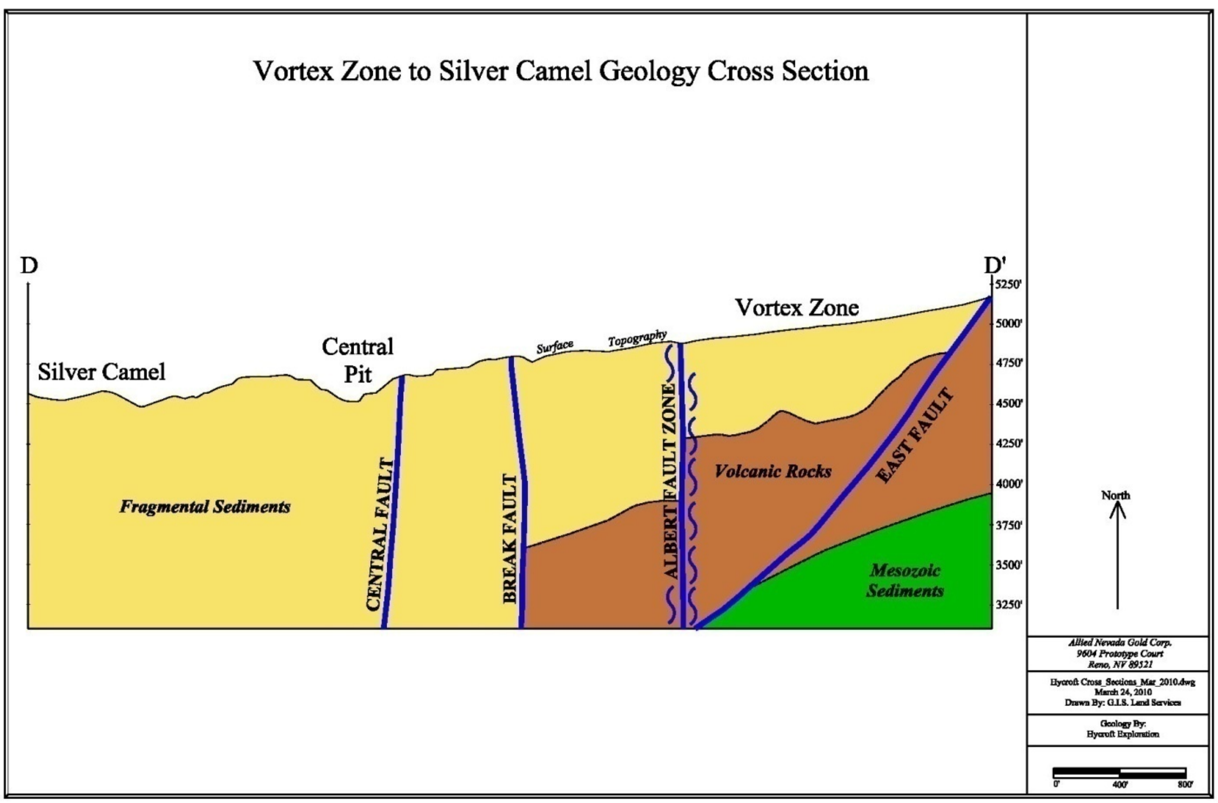
Gold and silver bearing host rocks in the Vortex Zone are correlative with those at the Brimstone deposit immediately to the north. At Vortex, strong silicification to depths of >1,500 ft is due to veining and phreatic hydrothermal brecciation. At least four mineralizing events are present as evidenced by crosscutting vein and breccias zones. The hydrothermal venting may have contributed to the eruption breccias overlying the Brimstone deposit. The same eruptive rock type occurs at Vortex, overlying veined and brecciated volcanic rocks.

The upper levels at Vortex are strongly hydrothermally clay altered, and, close to the East fault, acid leaching is prominent. Acid leaching at Vortex is subordinate to hydrothermal clay alteration, and is notably less extensive than occurs at the Brimstone deposit.

In the footwall of the East fault, rocks are exclusively KMG dominated by aphanitic, porphyritic, and flow banded rhyolite, basalt, and epiclastic tuff of felsic composition (Figure 7.5). Alteration and oxidation of these volcanic rocks is weak, with oxidation occurring within 50-150 ft of the East fault, and propylitic or clay alteration extending further.

The mineralization at Vortex is of both vein and disseminated type, with brecciated and altered rhyolite rocks and tuffs acting as favorable hosts. Very high grade silver has been drilled at Vortex. The predominant silver mineral is pyrargyrite, occurring both in veins and as disseminations. The chemical formula for pyrargyrite is Ag_3SbS_3 (silver, antimony, and sulfur).

Figure 7.5 Vortex Zone to Silver Camel Hill Generalized Section



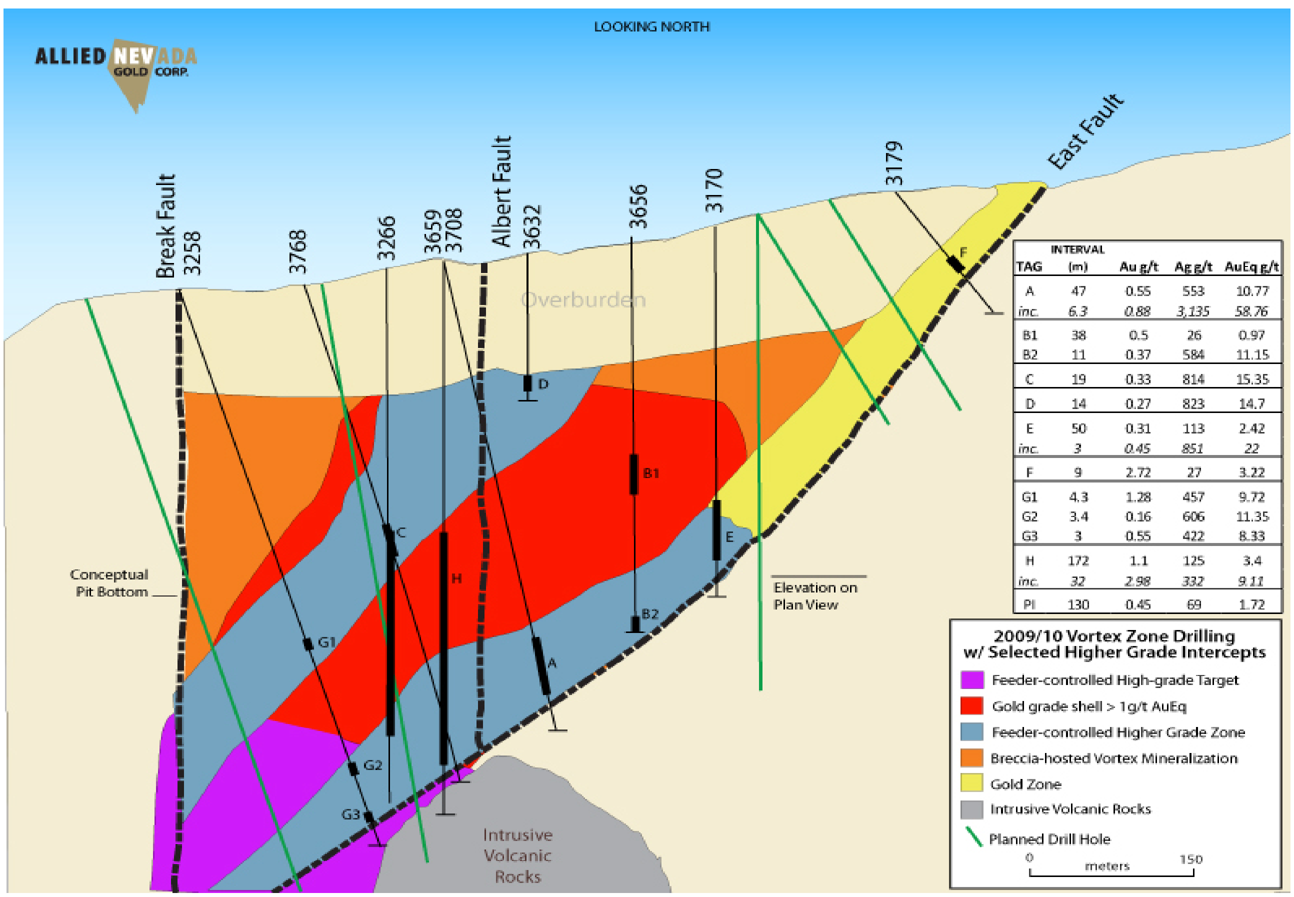
Vortex has characteristics of breccia hosted gold and silver systems that may have high grade feeder veins at depth. Targeting, using this concept in the initial discovery hole, resulted in a high grade pyrargyrite intercept and, in 2009, the same concept was used to intercept even higher grade silver in the form of pyrargyrite veins (Figure 7.6) adjacent to massive quartz veins. The latter hole also extended the Vortex Zone 300 ft to the west, and intercepted a targeted fault system at depth.

Oxide mineralization is present at about a depth of 500 ft below surface. A substantial zone of sulfide hosted mineralization occurs beneath the oxide mineralization. Bonanza grades of silver and gold are present within the zone (Figure 7.7). Limited drilling indicates a flat lying, massive quartz system containing visible pyrargyrite.

As presently known, the Vortex Zone mineralization is open to the north, south, west, and to depth.

Figure 7.6 Pyrargyrite Vein in Vortex Zone

Figure 7.7 Vortex Zone Drilling



7.4 BAY/LEWIS AND BONEYARD

Mineralization in the Bay/Lewis and Boneyard Area is hosted by gently, east dipping conglomerate rocks of the Sulfur Group. Both clast supported and matrix supported conglomerate rocks acted as mineral hosts, fed by high angle faults and fractures, plus lateral fluid flow. Mineralized siliceous hot spring sinters were noted during initial mining activities indicating that this deposit represents the upper levels of a hot spring hydrothermal system.

The predominant alteration type in the Bay/Lewis mineralized zones is silicification. Acid leach alteration in the area is relatively minor, and occurs along high angle structures (Figure 7.8). Strong oxidation is present in the upper portion of the silicified zone (Figure 7.9).

Underlying the silicification are illite-smectite clay altered, and clay dominant rocks. The basal rock type is lake sediments, composed of fine grained clay with minor layers of gravel and conglomerate extending to a depth >1,100 ft (Figure 7.8).

Figure 7.8 Bay Area Geology Cross Section

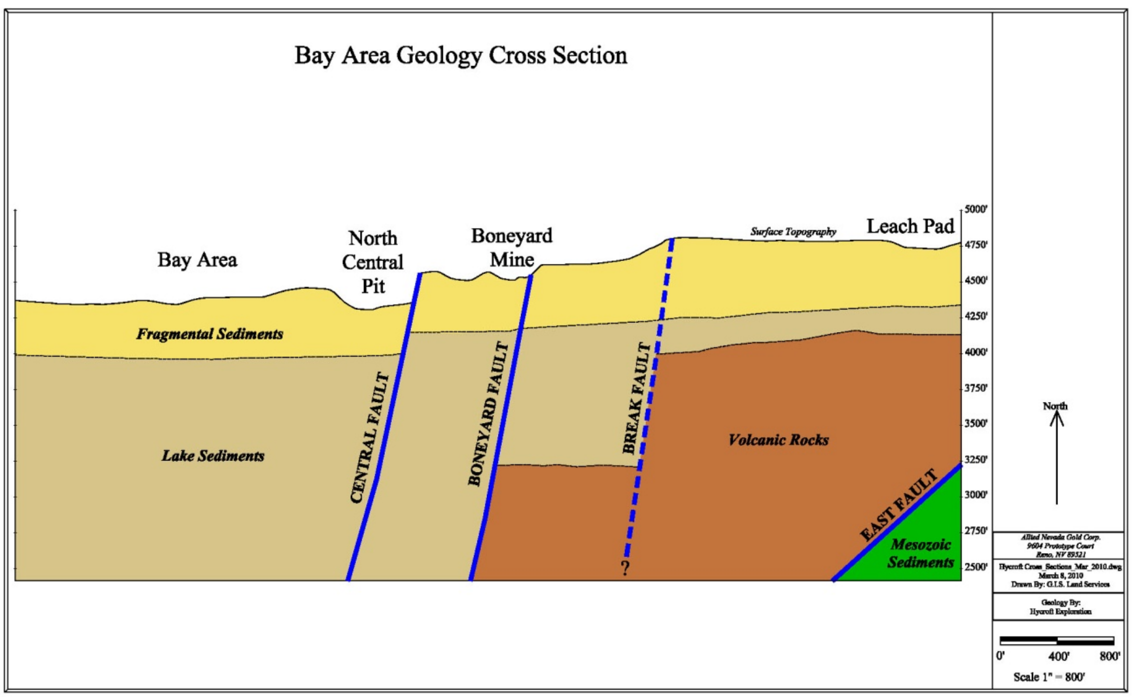


Figure 7.9 Bay Area Looking North

7.5 CENTRAL ZONE

The Central Zone geology is similar in nature to that of the Bay/Lewis Area, with mineralization and alteration fed by high angle faults and fractures, plus lateral fluid flow through the porous conglomerate rocks (Figure 7.11). High angle acid leach and chalcedonic veins, targeted for their higher gold grade, were mined along with the bulk tonnage material, especially in and near the Cut-4 pit.

The main Central Fault is a zone of sub-parallel faults that strike north-east and extend from the Bay Area deposit through the Cut-5 area, approximately three miles to the south. Offsets due to individual faults are minimal, with most measured in tens of feet.

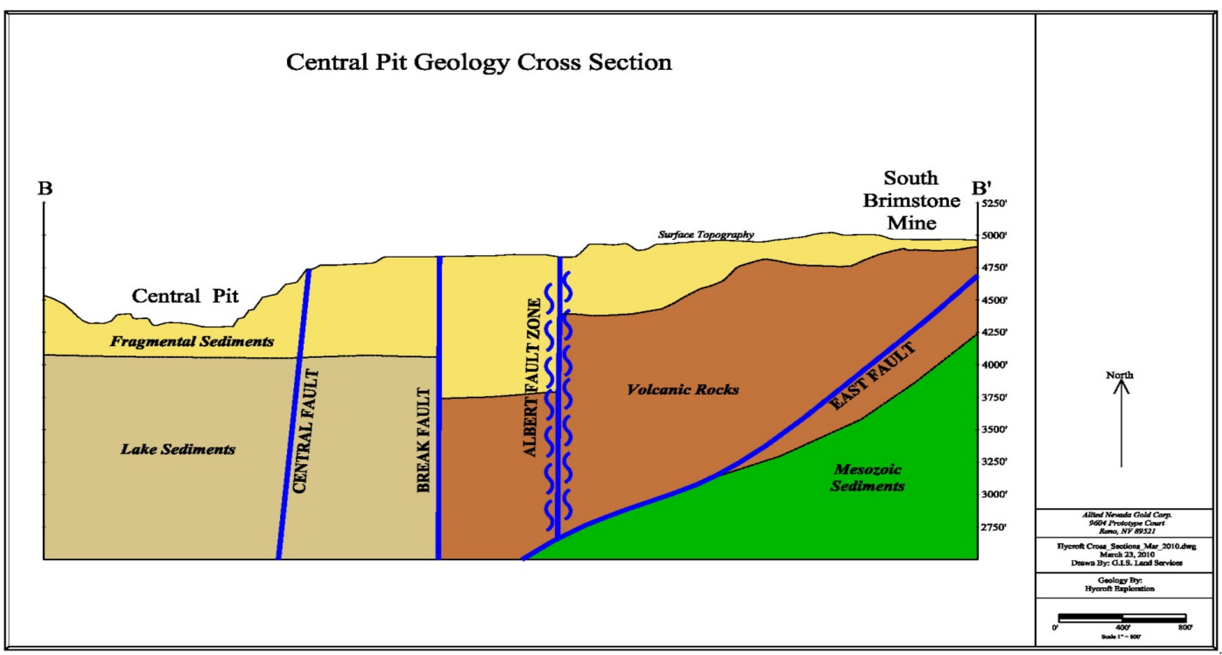
Alteration along the Central Zone is similar to that of the Bay/Lewis Area, minus the hot spring sinters. Acid Leach alteration is stronger and more widespread than at the Bay/Lewis Area (Figure 7.10), and is extensive in the southern portion of the pit. The acid leaching overlies silicified conglomerate rocks, except along the immediate trace of the Central fault where silicification dominated as the alteration type. Oxidation extends down roughly to the elevation of the historical pit bottoms of approximately 400 ft.

Underlying the silicification and acid leaching are illite-smectite clay altered and clay dominant rocks. The basal rock type is clay with minor layers of gravel and conglomerate extending to a depth >1,100 ft.

Gold and silver mineralization is spatially associated with a well developed set of through going planar chalcedonic and quartz alunite veins, oriented sub-parallel to the Central Fault Zone.

Figure 7.10 Central Zone Acid Leach

Figure 7.11 Central Pit Geology Cross Section



7.6 SILVER CAMEL/CUT-5 AREA

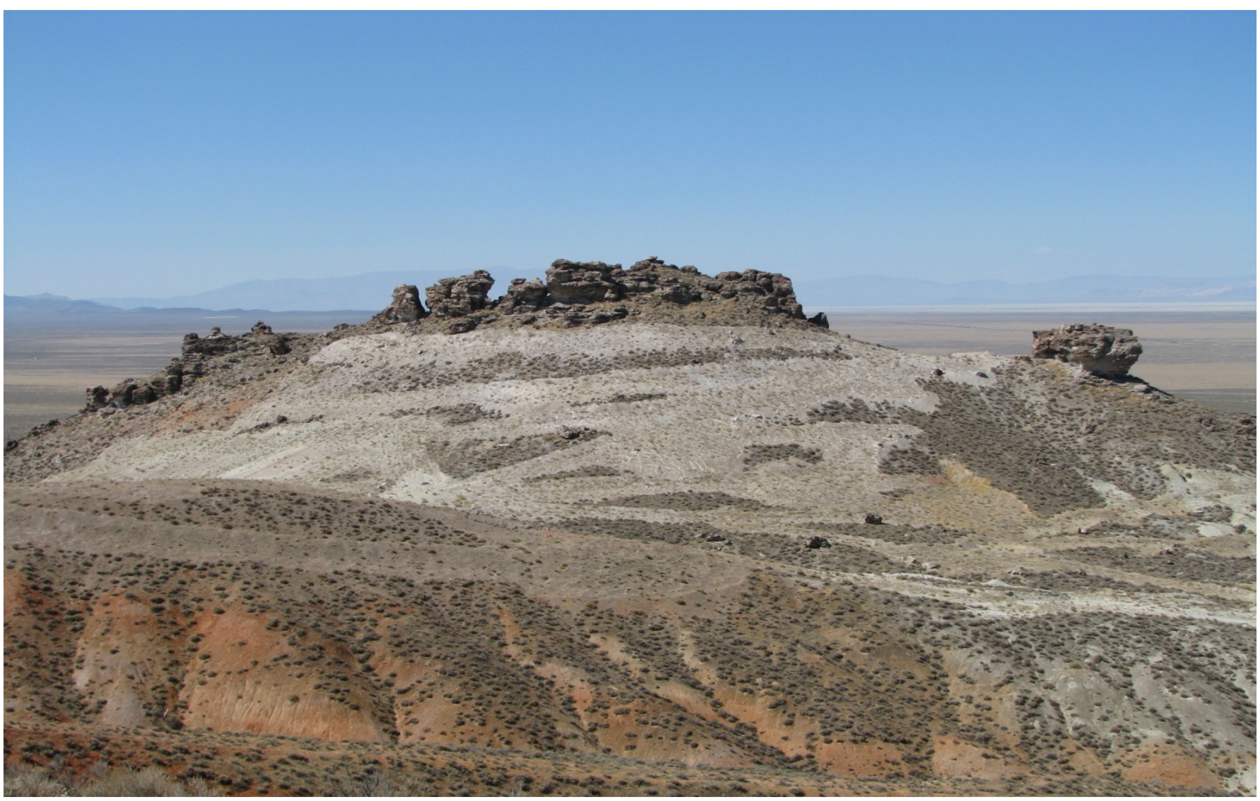
Mineralization in the Silver Camel/Cut-5 area is hosted by conglomerate rocks, and occurs as both disseminated gold and silver associated with pyrite and marcasite, and higher grade veins, including silver bearing pyrargyrite veins.

The nature of the basal rocks is markedly different than that immediately to the north along the Central pit, as conglomerate rocks persists to depth. The porous conglomerate rocks are mineralized and altered at greater depths than along the Central pit. The clay content of the rocks varies, and it appears that the clay rich basal rocks immediately to the north are present, but in much smaller quantities, and do not form extensive layers.

Alteration south of the Central pit, and in the Silver Camel Hill area is predominantly silicification and clay alteration (Figure 7.12). Hydrothermal clays, overlying silicified conglomerate rocks, and basal illite-smectite clay altered rocks are present. Notably, acid leaching in the area is relatively minor, especially with respect to the intensity and amount of acid leaching in the Central Zone/Cut-4 area immediately to the northeast.

Mineralization in the Camel/Cut-5 area extends to depths >1,000 ft in places, and is of both disseminated and vein type. Some pyrargyrite veins have been noted at depth, and mineralized siliceous veins on strike with veins in the Cut-4 pit to the north, are common. Oxidation extends to depths >200 ft, and an area of intense oxidized mordenite alteration is present between the Cut-5 and Silver Camel deposits.

Figure 7.12 Silver Camel Looking Southwest



8 HYCROFT GEOLOGICAL MODEL

Radiometric dates of adularia (potassium feldspar) indicate that the main phase of gold and silver mineralization formed four million years ago when gold and silver were deposited by a low sulfidation hot spring system. Fluids were fed by high angle, normal faults. Low grade gold and silver mineralization was co-deposited with silica and potassium feldspar throughout porous rock types.

A subsequent drop in permeability, due to sealing of the system, led to over pressuring and subsequent repeated hydrothermal brecciation. Additional precious metal mineralization was deposited during this event as irregular breccia zones, veins, and sulfide flooding.

Gold mineralization was followed 400,000 to two million years ago by an intense event of high sulfidation acid leaching of the mineralized volcanic rocks coincident with a regional water table drop, allowing steam heated sulfur gasses to condense into sulfuric acid and leach the upper portion of the mineralized rocks.

Supergene oxidation and post mineral normal faulting continue to present.

Not all the mineralized zone is completely oxidized and the depth of oxide and mixed sulfide and oxide ore varies considerably over the area of the deposits. The determination of whether or not mineralized material can be mined and processed economically by heap leach technology is dependent on the grade of gold and silver mineralization, the depth of overburden, and the degree of oxidation.

9 ALTERATION AND MINERALIZATION

9.1 ALTERATION

The main alteration events occurred in the following sequence:

- Barren propylitic alteration of the Kamma volcanic rocks.
- Barren illite-smectite clay alteration of the Camel Conglomerate and sedimentary rocks on the western portion of the Hycroft deposit, and related alteration of large areas of the Kamma volcanic rocks (illite + quartz + pyrite).
- Hydrothermal activity produced a layer of kaolinite-montmorillonite clay at the top of the opal chalcedony flooding, and above hydrothermal breccias.
- Widespread opal chalcedony, K-spar, pyrite and marcasite (termed “silica sulfide”) flooding of the sedimentary Camel Conglomerate, hydrothermal breccias ejecta, and related fragmental rocks was synchronous with the illite-smectite event. The resultant altered rocks are dense and colored dark gray.
- Blanket acid sulfate (acid leach) alteration formed a vertically zoned layer of upper residual quartz, and a lower layer of intense opalization, termed basal acid leach.
- Hypogene alteration oxidized silica and clay rich rocks at the base of acid leach alteration.
- Mordenite alteration (zeolite and clays) overprints the opal K-spar alteration, especially in the Gap and Bay Areas, and reaches depths of 160 ft in places. The alteration is commonly strongly oxidized and characteristically colored a reddish brown.
- Most recently, supergene oxidation of acid leach, oxide, and sulfide mineralization occurred along major faults, accompanied by small amounts of normal movement along the fault, displacing mineralization in the hanging wall downward.
- Each alteration and type is described below in detail.

9.1.1 PROPYLITIC

Propylitic alteration has only been noted in volcanic rocks of the Kamma Mountains, both in drill samples and from surface mapping in the mountains. Propylitic alteration is pervasive in the Kamma Mountains and affects the rocks in the Hycroft deposit both as pervasive and as propylitic veining, especially in rhyolite flows and intrusive rocks at Vortex and southern Brimstone. Typically, the propylitic alteration gives the rocks a bright green color, and minerals consist of chlorite, quartz K-spar, calcite and pyrite.

9.1.2 ILLITE-SMECTITE CLAY ALTERATION

This alteration type underlies the near surface silicification in the western portion of the district in the sedimentary rocks of the Camel Conglomerate and basal clay rocks. Rocks have been altered to a mixed layer Illite-smectite plus/minus quartz, calcite, pyrite, kaolinite, and pyrhotite assemblage. The alteration gives the rocks a gray to greenish-gray color, and extends to depths >1,000 ft. The composition of the Illite-smectite varies with distance from faults and with depth, with increasing Illite content indicating higher temperature with depth and proximity to silicified conduits.

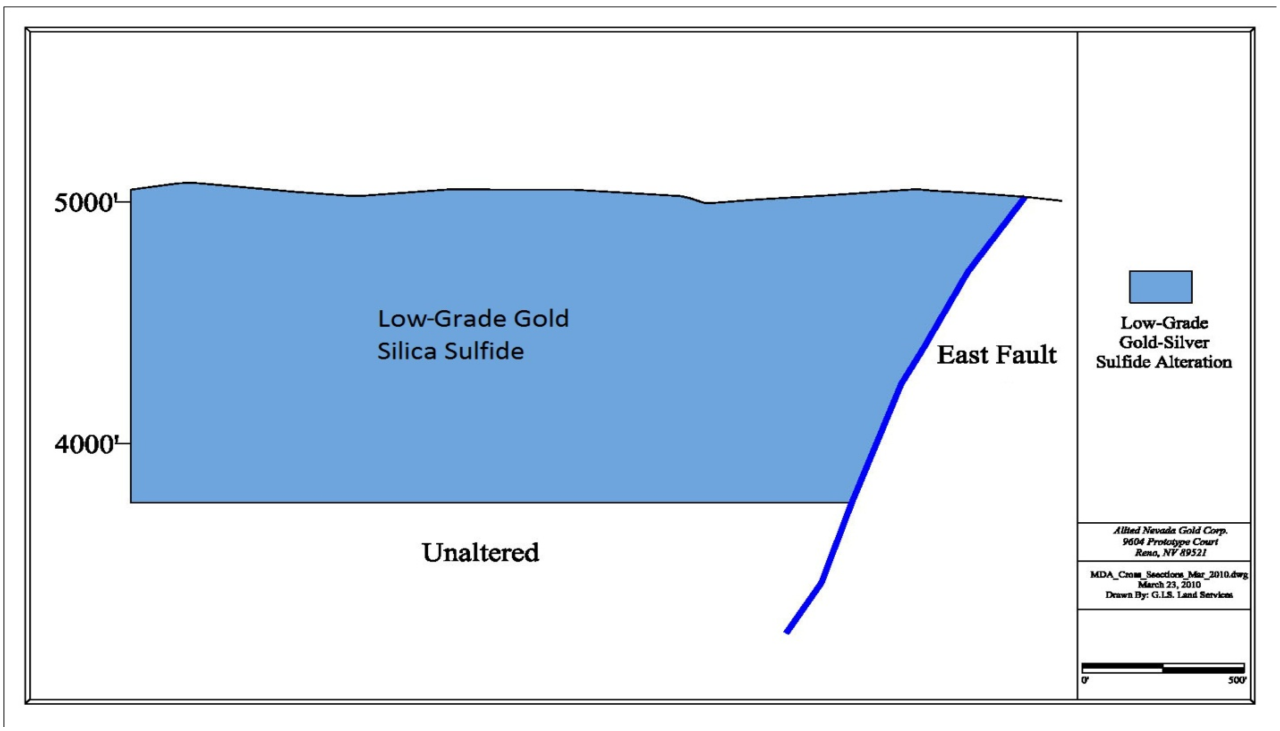
The contact between this alteration type and the opal K-spar alteration is transitional and suggests that the timing of the two events is roughly synchronous.

9.1.3 OPAL K-SPAR ALTERATION

A widespread event of barren silica-pyrite-potassium feldspar alteration created a blanket of silica sulfide alteration, having a glassy appearance, resulting from silicification that permeates the rock mass. Fine grained, euhedral to subhedral pyrite is always associated with this alteration. Pyrite forms 2-5% of the rock as fairly uniform, bright yellow to brassy grains, about 0.2-0.5 mm in size, and is evenly distributed throughout the rock mass. Up to 50% of the rock mass is composed of microscopic potassium feldspar. Figure 9.1 shows a schematic section of the distribution of this alteration type.

This alteration type created a blanket of silicification in the Bay Area, and is ubiquitous in the Brimstone/Albert region, extending for at least 6,000 ft along the strike of the East fault and at least 2,000 ft west of the East fault. The alteration affected some brecciated material that is interpreted to be hydrothermal (phreatic) explosion ejecta. This relationship, if it holds, suggests that the phreatic brecciation occurred at least as early as the start of the opal K-spar event. In cross section, the appearance of the opal K-spar alteration is funnel shaped and is deeper along faults.

Figure 9.1 Low Grade Gold Silica Pyrite



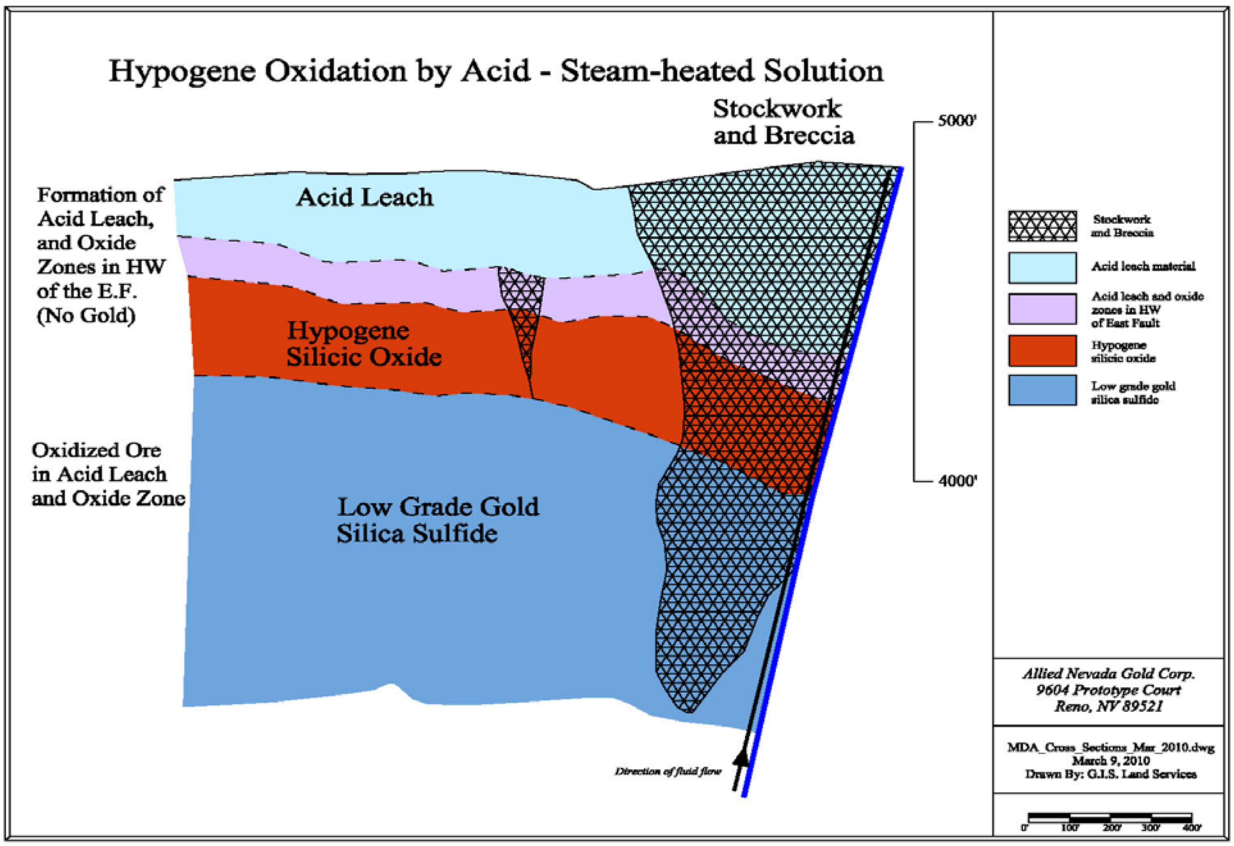
9.1.4 ACID LEACH ALTERATION

A sulfur rich hydrothermal system developed along the East fault and the southern portion of the Central fault system approximately 400,000 to two million years ago. Older silica sulfide mineralization was strongly leached by acids generated above the paleo water table. Downward percolation of acids formed a zoned pattern, from top to bottom, of blanket acid leach material, basal acid leach, and oxide. Oxide is older silica sulfide material in which sulfides have been altered to iron oxides.

The acid leach, oxidation alteration event determined the distribution of the two dominant types of oxidized material, "acid leach" and "oxide" rocks. The alteration is geometrically zoned suggesting that a single event produced the zoning. Acid leach and oxide alteration clearly overprint both earlier sulfide phases of alteration. Figure 9.2 shows a schematic section of the distribution of this alteration.

In general, acid leach alteration forms a horizontally oriented blanket, but has a "V" shaped aspect near major feeder faults. This alteration may be broken into two sub-types, blanket acid leach and basal acid leach alteration.

Figure 9.2 Hypogene Oxidation by Acid - Steam Heated Solution



Blanket acid leach material covers an area from the Brimstone pit to Cut-4, and is the uppermost oxidized alteration type. On average, blanket acid leach alteration is 150-200 ft thick over the area, reaching thicknesses of 450 ft in the immediate hanging wall of the East fault. Blanket acid leach alteration is characterized by the following properties:

- The ubiquitous presence of secondary porosity development at all scales of observation. Depending on the original composition of the rock, open spaces are developed after feldspars, fine grained rock fragments, or as simple vugs ranging in size from centimeters to less than 0.1 mm. Void spaces are due to the loss of most of the aluminous mineralogy in the original rock (feldspar, mica, or clay). Remaining aluminous mineralogy is almost always a white colored, powdery, fine grained alunite or kaolinite, of a few percent at best;

- The absence of iron bearing minerals, either oxides or sulfides;
- The rock is almost entirely composed of vuggy, fine grained silica;
- The original textures associated with volcanic rock deposition are completely obliterated or obscured;
- Accessory minerals are cinnabar, realgar (rare), native sulfur, opal, and gypsum. Native sulfur forms late stage massive veins or disseminations in acid leach rock; and
- Blanket acid leach alteration can be crumbly and incompetent, or hard and competent.

Basal acid leach alteration, where it occurs, is horizontal to the lower acid leach/oxide contact and ranges up to 40 ft thick.

Basal acid leach alteration is characterized by the following properties:

- Basal acid leach alteration rocks are extremely hard, being composed almost entirely of very fine grained, conchoidal fracturing silica;
- Accessory minerals are rare, but native sulfur has been observed; and
- Secondary porosity is not well developed occurring as irregular vugs and cavities on the centimeter to decimeter scale.

9.1.5 QUARTZ-CHALCEDONY VEINING AND SILICA FLOODING

Quartz-chalcedony veins cut acid leach alteration in the Brimstone deposit and Vortex Zone. These veins and associated silica flooding of wallrock may be related to hydrothermal brecciation and phreatic eruption events. If so, the eruptive breccias, which is bedded with the Camel Conglomerate and is itself acid leached in places, may have been episodic and lasted from well before the acid leaching event to later than the main acid leach event.

At depth in the Vortex Zone, and to a lesser extent at the Brimstone deposit, are lythophysae (small voids lined with fine grained euhedral quartz crystals), ‘finger breccias’, jigsaw breccias, and monolithic breccias veins and bodies from centimeters to meters thick, with material transported from depth, all consistent with phreatic brecciation. At depth in Brimstone and Vortex, the rock is brecciated and cut by high angle to moderately dipping veins and silicified breccias bodies. These veins and breccias may be from millimeters to meters thick, show several stages of formation by their crosscutting nature, and often are associated with sulfide selvages and fine grained sulfide flooding (locally termed ‘sooty sulfides’), giving the rock a dark gray to black color.

Veins are commonly banded and often contain brecciated fragments of other veins. The presence of calcite as euhedral rhombs, and replacement of calcite by quartz, is common. Along the East fault, massive calcite veins, some horizontally banded, are present.

Breccia bodies, comprised of monolithic fragments, some of which show an alteration ‘rind’ of silica overprinting less silicified fragments, are present from depths of >1,500 ft to just below what is interpreted as the eruptive fragmental rocks. Some breccias bodies are multi-lithic and contain clasts of basement ALS rocks, and fragments of propylitic volcanic rocks. At Vortex, brecciated and silicified basement ALS rocks have been intercepted in drill core.

9.1.6 HYPOGENE OXIDATION

This original hypogene alteration is composed of two dominant types: silicic oxide and clay oxide. Iron oxide minerals occur on fractures and in original sulfide sites. Silicic oxidation comprises about 85% of all oxide samples. Silicic oxidation underlies acid leach alteration and reaches thicknesses of up to 200 ft. In the majority of oxide mineralization, all sulfides have been converted to iron oxides. Silicic oxide is fine grained and glassy appearing, with little or no secondary porosity development. Iron oxides, sulfates, and hydroxides are common accessory minerals, with hematite the most prevalent oxide. Other accessory iron bearing phases include limonite and jarosite. Jarosite often occurs as amber, euhedral crystals, one to two mm in size, or as fracture coatings and late veinlets. Pyrite and marcasite are replaced by red, earthy hematite. Fine fracture networks can be observed, filled with hematite, limonite, and minor clay.

Black to metallic gray specular hematite is observed as fracture coatings and pisolitic masses filling minor openings in the rock have been found in both the Brimstone pit and in the northern Bay/Lewis Area along the Central fault. Specular hematite results from iron phases being precipitated after being leached from the overlying acid leach material.

Silicic oxidation can have a variety of dominant colors; from white to yellow to red and even purple, depending on the relative amounts of iron oxides, hydroxides, and sulfates. Silicic oxide is composed of 65-85% silica, 5-20% clay, and 5-15% hematite and jarosite.

Clay oxidation makes up about 15% of material classed as oxide, and is thought to be the result of hydrothermal alteration of in situ rock, representing formation under weakly acid oxidizing conditions. Clay zones appear white to yellow to pinkish and are composed of 50% or more clay, with the usual accessory iron oxides. Clays are mixtures of montmorillonite and kaolinite with accessory alunite, and occur as discontinuous layers 30-50 ft thick directly beneath basal acid leach alteration, as irregular veins, or amoeboid shaped areas scattered throughout the silica oxide alteration. At the Vortex Zone, clay oxidation may extend to depths >200 ft.

Iron oxides nearly always underlie acid leach alteration. Within acid leach alteration, there are remnant pods of unoxidized rock containing sulfide mineralization. These remnant pods of sulfides are always surrounded by a clay oxide rim, suspended in acid leach altered material.

The mineral assemblages in each alteration and oxidation type, plus the strong geometric zoning, suggest that acid leach alteration and oxidation formed from the interaction of the oxidized fluids at the water table with descending acid fluids.

Whole rock geochemical analysis shows that the acid leach material contains two to four weight percent Al_2O_3, indicating depletion of the aluminum. This depletion requires that the pH of conditions under which acid leach alteration formed was lower than two.

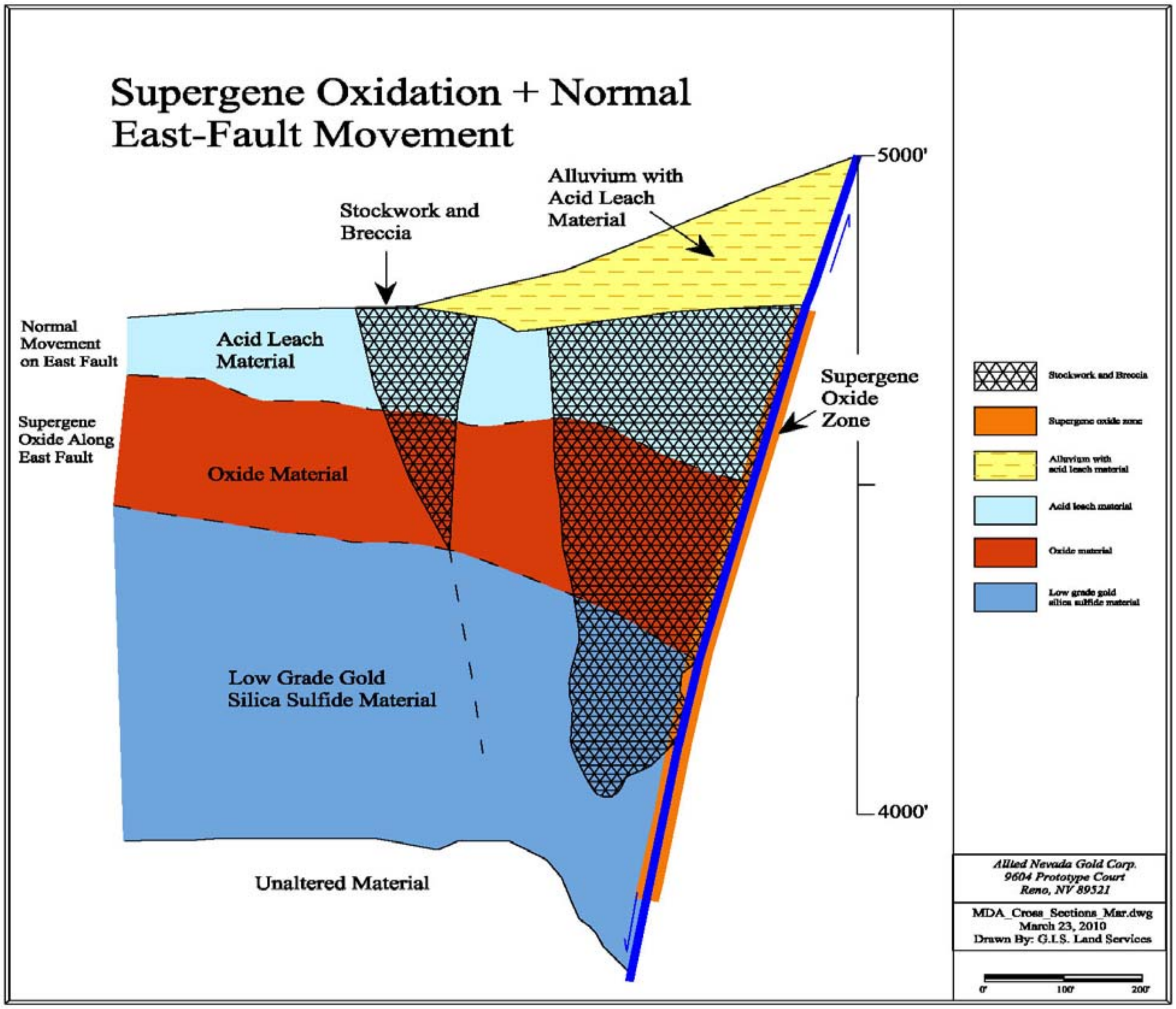
The absence of iron phases in acid leach alteration supports a low pH, since iron is soluble in acid but insoluble under neutral oxidizing conditions. Iron was transported to the neutral waters from overlying acid waters and precipitated as specular hematite or jarosite in oxidizing neutral water (silicic oxide alteration), or weakly acid oxidized water (clay oxidation). The upper level acid fluids were created through oxidation of hydrogen sulfide on reaching the surface, or by oxidation of pyrite by surface waters.

9.1.7 SUPERGENE OXIDATION

Supergene oxidation extends to depth along faults, manifested as a zone of oxide stained fault gouge. Figure 9.3 shows a schematic section of the distribution of this alteration. Supergene oxidation was the final alteration event.

The zone appears very similar to silica oxidation, and small fragments of acid leach alteration are caught up in this material. Bright red hematite most often coats all fragments in this zone. In deeper levels of the North Brimstone pit, black manganiferrous oxides also occur. Supergene oxidation forms a band 20-80 ft wide in faulted contacts.

Figure 9.3 Supergene Oxidation + Normal Fault Movement



9.2 MINERALIZATION

Several styles of mineralization exist at the Hycroft deposit. The early silica sulfide flooding event deposited relatively low grade gold and silver mineralization. Steeply dipping quartz alunite veins created a mineralized zone below the basal acid leach, in open space voids and fractures in the acid leach blanket. Hypogene enrichment of gold and silver occurred at the base of the acid leach blanket.

Quartz chalcedony veins cut the acid leached rock and carry gold and silver. These veins occur in the Brimstone, Boneyard, Cut-4 and Vortex Zone areas, and may be related to hydrothermal brecciation. Late stage silver bearing pyrargyrite veins are found in the Vortex Zone and at depth in the Cut-5 area. Late to present supergene oxidation along faults has liberated precious metals from sulfides and enriched gold and silver.

9.2.1 OPAL K-SPAR

Large, near surface silica sulfide alteration deposited low grade gold and silver minerals. Where these were oxidized, the gold and silver became amenable to heap leaching. The Bay Area has a blanket of this mineralization that was overprinted with mordenite alteration, and the resulting oxidation left a near surface exposure of gold and silver that was mined. The mineralization in the Bay Area is both fracture and structurally influenced, and controlled by oxidation along bedding. Some Bay Area mineralization occurred in hot spring siliceous sinters interbedded with the opal K-spar mineralization. Gold and silver occur as micron sized grains associated with sulfides, and in the matrix of the rocks.

9.2.2 QUARTZ ALUNITE VEINS

Steeply dipping, acid leach altered veins host gold and silver. The veins have been deposited in fractures below the basal acid leach, and in fractures and voids in the main acid leach blanket. In the Central pit, the veins were noted to carry higher values than the average grade of the deposit as a whole. This mineralization is important at the Brimstone deposit, and oxidization of the veins is noted to extend deeper along faults and suspected breccia bodies there**.**

Pervasive zones of mineralized acid leach hosted alteration consist mainly of white chalcedony quartz fragments surrounded by fine grained silica, alunite, kaolinite, and montmorillonite. The fragments are opal coated or opalized remnants of the earlier opal K-spar altered rocks. Pervasive acid leach mineralization gives way to narrow, steeply dipping, mineralized, acid sulfate veins. The tenor of gold and silver mineralization decreases with depth in these veins.

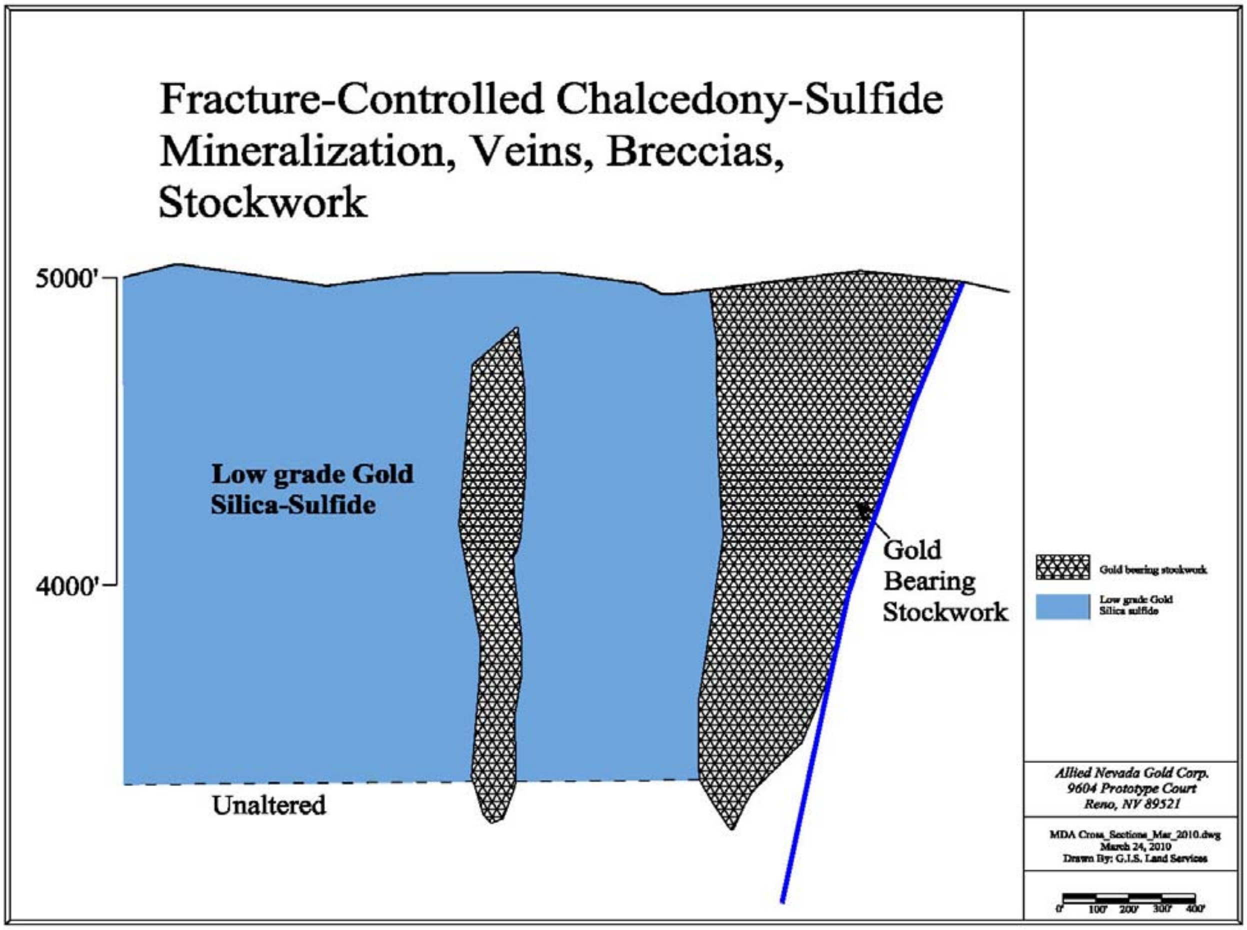
Gold occurs as micron sized electrum grains (averaging 30% silver) within and adjacent to opal, alunite, and clay minerals. Silver also occurs as cerargyrite (AgCl) and iodargyrite (Agl) associated with alunite, clays, or jarosite.

9.2.3 QUARTZ CHALCEDONY VEINS

Fracture and breccias controlled chalcedony-pyrite-marcasite mineralization was associated with primarily gold and possibly silver deposition at Brimstone and the Vortex Zone, occurring as veinlets, stock works, in situ breccias, and rotational (chaotic) breccias. This mineralization type, as shown in Figure 9.4, clearly crosscuts the earlier low grade silica pyrite alteration and acid leached material. The veinlet mineralization occurs as one mm to two cm veinlets forming 2-10% of the rock mass. The veinlets are composed of gray to milk white chalcedony with 5-10% sulfides. Chalcedony veins are commonly massive. In situ breccias shows flooding of the rock fractures with the chalcedony sulfide assemblage filling a network of fractures occupying 5-15% of the rock mass.

Chaotic breccias occurs as unsorted, angular wall rock fragments floating in a matrix of chalcedony sulfide. Fragments are not aligned and clearly show rotation with respect to adjacent fragments. Breccia mineralization comprises 5-20% of the rock mass. This brecciation type is suspected of being the result of phreatic (hydrothermal) brecciation.

Figure 9.4 Schematic Cross Section of Quartz Chalcedony Sulfide Mineralization



Fracturing, veining, and brecciation related to phreatic explosions led to both mineral deposition and discontinuous blankets of fragmental ejecta. The ejected fragmental rocks are often acid leached and some are cut by quartz chalcedony veining. This relationship suggests that the phreatic events were long lived and lasted throughout the acid leach event.

Sulfides are dominated by pyrite and marcasite (both minerals share the chemical formula FeS_2, however, marcasite is more brittle and crystallizes in a different, more unstable crystal system, which is the main defining difference between the two minerals). Pyrite occurs within veinlets as irregular anhedral masses which are sub-parallel to the veinlet edges and from 0.5 mm to 0.5 cm long. Marcasite occurs as similar sized masses and as single crystals. Marcasite is euhedral to subhedral, with masses forming twin sheaf like groups of crystals. Gold, and possibly silver, mineralization was most likely introduced during this event. Visible gold (50-120 microns in size) has been identified within chalcedonic veins in thin sections, and is closely associated with marcasite, and assay statistics show a correlation between fracture controlled veining and gold mineralization. At Vortex, sulfide veins associated with this style of mineralization are higher in grade for gold and silver than surrounding wall rock.

This mineralization is less widespread than barren silica pyrite alteration. Fracture and brecciation controlled mineralization is observed in drill core and chips up to 2,000 ft west of the East fault and intercepted in drill holes at depths >1,500 ft at Vortex. The north-south extent of this type of mineralization is at least 5,000 ft, from approximately 39,000N to 44,025N. The East fault clearly post dates this mineralization, as evidenced by areas of fault gouge in the Brimstone pit bearing fragments of this mineralization type.

In the Boneyard and Cut-4 areas, massive quartz chalcedony veins are through going, parallel the Central Fault Zone, and are up to meters wide. The veins are not associated with phreatic brecciation, and appear to be the result of open space fracture filling. Some of the veins are horizontally banded, the result of the proto-silica gel slumping from fracture walls. Early calcite in the veins suggests a vigorously boiling system at depths from the base of the paleo water table up to several hundred meters. The calcite was later replaced by quartz and chalcedony. Brecciated vein material is common in the massive veins. Pyrargyrite quartz chalcedony veining has been found at depth in the Cut-5 area immediately to the south of Cut-4. The current working theory is that the fracture and brecciation related veining at Brimstone and Vortex, and the more massive chalcedonic veining in Boneyard and Cut-4/5, occurred as a general result of the same regional hydrothermal event, but manifested in different forms due to local host rock and structural characteristics.

9.2.4 LATE STAGE QUARTZ SILVER SULFIDE VEINS

At Vortex, late stage veins of pyrargyrite are associated with zones of quartz veining. The pyrargyrite occurs as discrete veins, selvages along quartz veins, and as crystals deposited on small, euhedral, drusy quartz lining small voids. The quartz veins are both banded and massive, with the mineralized veins ranging from sub-millimeter to centimeters in size. The massive quartz veins are white quartz, often have a ‘moth eaten’ appearance due to numerous vugs, and are from meters to tens of meters thick. These veins are interpreted as feeder veins, possibly along structures and cut earlier brecciation and their geometry has yet to be detailed.

This mineralization style appears to favor a horizon just below gold mineralization, and has been intercepted at roughly the same elevation at depths ranging from 900-1,375 ft within the Vortex Zone from several holes spaced approximately 500 ft apart.

9.2.5 HYPOGENE MINERALIZATION

Mineralization, as a result of hypogene alteration below the acid leach zones, consists of oxidized sulfides in the silica and clay. This oxidation made the material amenable to heap leach recovery. The oxidation formed from the interaction of the oxidized fluids at the water table with descending acidic fluids.

9.2.6 SUPERGENE OXIDATION OF SULFIDES

Descending oxidized fluids along fault planes and in fractures, oxidized sulfides and liberated precious metals, making the material amenable to heap leach recovery.

10 EXPLORATION

Homestake conducted the initial modern exploration of the district beginning in 1981. The predecessors to HRDI gained control of the district in 1985 and drilled a total of 3,123 exploration holes, totaling 943,581 ft. Canyon Resources ('Vista') then completed 33 drill holes totaling 13,310 ft of RC drilling during 2005. The current Hycroft drill hole database consists of historic holes and drilling by Allied Nevada from 2006 to 2009.

Between August 2007 and July 2010, Allied Nevada drilled 676 holes totaling 534,397 ft. In 2010, Allied Nevada drilled 153 holes: 95 RC and 58 diamond core, totaling 138,689 ft.

Objectives of the Allied Nevada drilling program incorporated step out exploration, resource infill delineation, and reserve definition. Drilling campaigns are summarized in Table 10.1 by year, operator and drilling type. A breakdown of the drill holes by type and orientation is found in Table 10.2.

Table 10.1 Hycroft Exploration Drill Campaigns

Year	Hole Type	Company	No. of Holes	Footage	Zones Drilled
1981	Rotary	Homestake	29	5,550	North, SC
1982	RC	Homestake	61	10,015	North
1985	RC	Hycroft	195	33,482	North, Cut-4, SC
1986	RC	Hycroft	492	96,877	North, Cut-4 ,SC ,Gap, Brim, Alb
1987	RC	Hycroft	632	138,385	Alb, Cut-4, Gap, North, SC
1988	RC	Hycroft	73	25,855	Alb, Brim, Cut-4, North, SC
1989	RC	Hycroft	43	15,780	Alb, Brim, Cut-4, North, SC
1990	DD	Hycroft	8	11,247	Cut-4, Sulfur
1990	RC	Hycroft	134	52,675	Alb, Brim, Cut-4, North, SC
1991	RC	Hycroft	147	44,360	Cut-4, North, SC
1992	RC	Hycroft	265	83,030	Alb, Brim, Cut-4 ,North, SC
1993	DD	Hycroft	6	2,318	Alb, Brim, SC
1993	RC	Hycroft	297	105,500	Alb, Brim, Cut-4, North, SC
1994	DD	Hycroft	3	4,990	Brim
1994	RC	Hycroft	208	78,650	Alb, Brim, Cut-4, Boneyard, SC
1995	RC	Hycroft	355	157,515	Alb, Brim, Cut-4, Gap, Boneyard, SC
1996	DD	Hycroft	1	1,078	Brim
1996	RC	Hycroft	164	75,000	Alb, Brim, Cut-4, North, SCP
1997	RC	Hycroft	13	3,040	Brim, Boneyard
1998	Blasthole	Hycroft	67	3,670	Brim
1999	DD	Hycroft	9	4,870	Brim
1999	RC	Hycroft	11	5,500	Brim
2005	RC	Vista	33	13,315	Brim
2006	RC	ANV	1	900	Brim
2007	RC	ANV	14	14,944	Alb, Brim
2007	DD	ANV	38	42,930	Alb, Brim, Bay
2008	RC	ANV	281	181,810	Alb, Brim
2008	DD	ANV	60	60,810	Alb, Brim, Bay
2009	DD	ANV	50	39,989	Bay, Cut-4, Vortex, Brim
2009	RC	ANV	79	54,325	Bay, Vortex, Brim
2010	RC	ANV	95	69,217	Bay, Vortex, Brim, Alb
2010	DD	ANV	58	69,472	Bay, Vortex, Brim, Alb
Total			**3,922**	**1,507,099**	

Table 10.2 Exploration Drill Holes by Type

Drill Type	Number	Footage
Diamond Drill	233	237,704
RC	3,593	1,260,175
Rotary	29	5,550
Blast	67	3,670
Total	**3,922**	**1,507,099**
Angle	1,488	
Vertical	2,434	

Exploration by Hycroft, Homestake and Allied Nevada resulted in the discovery of the following interpretations of zones of mineralization:

- Bay Area – a large blanket of oxide mineralization hosted by interbedded sinters and conglomeritic to sandy debris flows (Upper Camel Group). The Bay Area represents the north end of the district.
- Central Fault deposits – South Central, Gap, Cut-4, and Cut-5 – a 10,000 ft segment in the immediate hanging wall of the Central fault.
- Boneyard deposit – strikes north-northeast and is located approximately 1,000 ft east of the Bay Area. This deposit is similar in lithology and alteration to the Central Fault deposits.
- Brimstone deposit – located in the hanging wall of the west dipping, normal East fault.
- Albert deposit – located between the Central Fault and Brimstone deposits.
- Vortex deposit – discovered by Allied Nevada in early 2008, followed by delineation drilling in late 2008 through the first half of 2010. The 2010 drilling program also extended to the western most known extent of KMG volcanic rocks by approximately 800 ft.

The discovery year of each oxide deposit is shown below in Table 10.3

Table 10.3 Discovery Years of Hycroft Oxide Zones

Deposit	Discovery Yr	Hole No.	Company	Hole Orientation	Present Condition
Cut-4	1977	Duval	Duval	Vertical	Mined
Bay Area	1981	SR-1	Homestake	Vertical	Mined
South	1981	SR-27	Homestake	Angle	Mined
Boneyard	1986	86-230	Hycroft	Vertical	Mined
Gap	1986	86-290	Hycroft	Angle	Mined
Brimstone	1986	86-256	Hycroft	Angle	Mining On-Going
Albert	1988	88-1389	Hycroft	Vertical	Mineralization
Vortex	2008	H08D-3170	ANV	Vertical	Mineralization

Early work by Homestake and Duval led to the discovery of ore zones on the south and north ends of the Central fault. Additional oxide discoveries were made by Hycroft in a short period of drilling during

1986. No new oxide zones have been discovered since 1988, although the current drill pattern is now substantially outside of previous discovery areas.

10.1 GEOLOGIC LOGGING

A variety of geologic logging systems have been utilized during the 29 year exploration history of the Hycroft deposits. Vista reviewed drill logs from holes drilled during the period from 1986 to 1998 on the Brimstone project, which led to the conclusion that issues existed with continuity and consistency of logging observations. Vista geologists re-logged available drill chips and core.

In 2007, Allied Nevada further refined the logging system to provide more detail on the intensity, style, and distribution of the geologic attributes. All of the previous logging, where possible, was converted to the current Allied Nevada system. The current logging system is shown in Table 10.4.

Table 10.4 Allied Nevada Logging Code Fields

Log Form Identifier	Description
Hole	Drill hole name
From	Interval start footage
To	Interval ending footage
Formation	Formation code
Lithology	Lithology code
Malt_T	Main alteration type
Malt_S	Main alteration intensity
Malt_M	Main alteration mode
2Alt_T	Secondary alteration type
2Alt_S	Secondary alteration intensity
2Alt_M	Secondary alteration mode
Vein_T	Vein type
Vein_S	Vein intensity
Vein_M	Vein mode
Min_T	Mineral type
Min_S	Mineral intensity
Min_M	Mineral mode
Sulf_T	Sulfur type
Sulf_S	Sulfur intensity
Sulf_M	Sulfur mode
Ox_T	Oxidation type
Ox-S	Oxidation intensity
Ox-M	Oxidation mode
Struct	Structure
Texture	Rock texture or structure modifier

10.2 ALLIED NEVADA LOGGING CODES DESCRIPTIONS

Starting in June 2008, the following text-based logging codes were instituted and have been in use since. The logs are entered by the geologist in paper format, and the paper logs digitally entered into spreadsheets, which are then compiled into a master logging data list.

Hycroft Logging Form Codes Updated: 3/1/2010 (Usage started June 2008)

Formation Code	**Formation Description**
NS	No Sample for the interval
Dmp	Waste dumps, road fill
Qal	Quaternary alluvium, includes 'Rosebud gravels'
Tal	Tertiary older alluvium
Tcp	Camel pyroclastic and related rocks, (depreciated July 29, 2009. Use Tc instead as it eliminates misleading 'pyroclastic' term). Existing Tcp codes can be translated directly into Tc.
Tc	Tertiary Camel formation includes sedimentary and volcanic rock clasts, lithologies: Tpw, Tcc, Tcm, and Tsg.
Tk	Kamma volcanic rocks
Ja	Jurassic (ALS basinal sediments)
Rock Code	**Lithology Description**
NS	No Sample for the interval. Use NS for both Formation and Lithology.
Dmp	Dump material, describe in comments. Use Dmp for both Formation and Lithology.
Qal	Quaternary alluvium. Use Qal for both Formation and Lithology. Qal generally reacts moderately to strongly with Hcl acid.
Tal	Tertiary older alluvium, unconsolidated, may be hydrothermally altered. Use Tal for both Formation and Lithology. Suspicion that Tal is in part pyroclastic/phreatic in origin due to abundant maroon crystal bearing rhyolite frags, unsorted layering, and argillic alteration in many places. Type local is Brimstone north ramp, also extensive under new leach pad. Tal reacts mildly to none with Hcl acid. Often mistaken for Tcm or Qal, especially in rotary holes.
Tpw	Sedimentary/conglomerate rocks, water lain or reworked, undifferentiated, (formerly Camel Conglomerate, use regardless of Fm). Includes Tcc and Tcm. Used mostly in logging RC holes when difference between Tcc/Tcm is not apparent.

Tcc	Tertiary Camel Conglomerate, classic clast supported water lain/worked. Typically comprised of imbricate flattened, or rounded, clasts of sedimentary rock, especially ALS, blocky white quartz, sand/silt layers, and cross bedding. Volcanic rocks if present constitute the minority of clasts. Tcc may be volumetrically much less than Tcm. Type local: North pit cliffs.
Tcm	Matrix supported water lain/worked volcanic rock and sedimentary clasts. Tcc grades downward into Tcm, and differs in that Tcm has more volcanic rock clasts, is generally more angular than Tcc, has fewer ALS fragments, and contains interbeds from centimeters to meters thick of ash and tuff. May alter to illite-smectite, such as in Central pit east wall (type locality). Beds are generally centimeters to meters thick. Some soft sediment deformation and cross beds noted.
Tsg	Historically part of the 'Sulfur Group', lower member. Lacustrine to gravel sediments. Matrix may be in part tuffaceous. Typically gray, muddy, sandy, gravel, +/- sparse coarser clasts. Contains thin interbeds of Tcm, and contact gradational from Tcm downward into Tsg. Contains brown layers which may be stratigraphic markers. These layers are suspected of being intermediate to mafic derivatives (tuff, erosional debris), and will typically react to HCl.
Tbg	Red matrix epiclastic/fanglomerate unit capping Kamma volcanic rocks. Same as Rosebud 'Badger' unit. Unconformably overlies Kamma, thickness tens to hundred feet. Clasts mainly volcanic rock. Mapped on surface (east side of Kammas) dipping 10-30°s, both east and west dips. Currently intercepted in only two core holes including Duval-2. Type local first ridge east of Floka peak radio tower.
Tlaf	Lithic ash flow – angular to sub-rounded lithic fragments up to X cm in size, often tan lithics and matrix. May contain packed or matrix supported lithics, and pale green lithic fragments are common. Found at depth in Vortex, and at ~500-700' at Brimstone.
Tms	Mudstone: Typically dark brown, almost black. May show soft sediment deformation and mild clay alteration. Can be mistaken for ALS, but differentiated by stratigraphic position well above ALS contact. Finely laminated, dark reddish-brown, tuffaceous (?) unit in footwall of East Fault at the south end of Brimstone may be the same or similar unit, but less deformed and altered.
Tri	Intrusive rhyolite: Aphanitic to banded. Aphanitic type shows chilled margins, can cut core at a high angle, and appears to cross cut as well as be concordant to other units. Banded rhyolite has bands near-parallel to contact margin, and normal to contact in interior of dike or sill.
Tpa	Depreciated: Discontinue use (7/2009) . Pyroclastic rocks/ash, matrix supported (formerly Camel unit # C2, use regardless of formation Fm).
Tpu	Pyroclastic/Epiclastic rocks, undifferentiated. Tpu and Tcm are most often the upper unit between Albert and East faults, where Tcm is often found overlying Ttu or Tr. May be interbedded with Tpw/Tcm west of the Albert fault. Angular volcanic rock clasts, may in part be derived from phreatic events, fanglomerates, slope wash. Tpu is difficult to distinguish from Tcm, especially in RC chips, and the keys are angularity, more mono-lithic nature, and clasts predominantly volcanic rock in origin.

Ttb	Banded tuff, fine grained and thinly layered (~1-3 mm).
Ttl	Lithic Tuff (Includes Ttf fragmental tuff; probably subjacent Kamma sequence).
Ttu	Tuff, undifferentiated (Includes identified varieties: Tra, air-fall tuff; Trw, water lain tuff; probably subjacent Kamma sequence). Typically fine grained and massive, includes bedded and banded (but not welded) units.
Ttw	Welded tuff, ignimbrite. Often evidenced by flattened fiamme structure (eutaxitic). May contain crystals, other lithics. Chloritic alteration of fiamme may be present giving rock a dark green streaked appearance. Sulfides may preferentially replace fiamme in more silicified rock giving the rock a gray wispy appearance. Type local on dirt road to Floka Peak (radio tower hill).
Ttx	Crystal tuff, typically feldspar rich, may contain unaltered sanadine phenocrysts or minor lithics.
Tfu	Depreciated: Discontinue use (7/2009) Felsic volcanic rocks, undifferentiated (presumed Kamma)
Tr	Rhyolite (presumed Kamma). May be purple to red in hand specimen on surface, typically red-brown (green if propylitic) in drill holes, often aphanitic and can easily be mistaken for altered tuff when completely silicified. RC drills bring up peculiar translucent brown water when drilling Tr, yet the chips may be white-grey quartz with only minor brown silica patches. Tr in Vortex noted to be in part intrusive, with flow banding parallel to contacts, chill margins. Tr in footwall of East fault in south Brimstone and Vortex is porphyritic, has biotite, sanadine, 1-2 mm quartz phenocrysts, and 1-4 mm feldspars.
Trb	Rhyolite, flow banded (presumed Kamma). Often medium to dark brown in drill holes. Autobrecciation noted in many sore intervals. Odd medium brown translucent drill fluid noted in RC holes when drilling Tr or Trb. RC chips typically hard to classify: most Tr has silica veining – RC chips will typically have medium brown to tan pieces of Tr on clear to gray silica.
Td	Dacite (presumed Kamma but age uncertain)
Ta	Andesite (presumed Kamma but age uncertain). Variant: trachytic andesite has large lath shaped feldspars. May have significant magnetite as accessory/contact mineral.
Tvc	Volcanic rocklastic rocks, may be in part same as Tpu. Often interbedded with rhyolite flows, may be sandy, gravels, conglomerates, but most clasts volcanic rock in origin. Different from ACG in that few ALS fragments are seen.
Tmu	Intermediate to mafic volcanic rocks, undifferentiated (presumed Kamma but age uncertain). May have significant magnetite as accessory/contact mineral.
Acg	ALS derived sedimentary unit. Gravel to matrix supported clasts. Primarily ALS derived, may include Kamma and other sedimentary clasts. Typically immediately overlies ALS rocks.

ALS ALS rocks, undifferentiated as to shale, sandstone, siltstone, carbonate. Typically black with white quartz veins. Fractures in core to flat, thin 'hockey puck' looking pieces. May have white quartz veining and be silicified.

Formation members note:

Generally formation and lithology pairs follow the correlation:

Formation	**Age**	**Lithology**
Qal	quaternary	Qal
Dmp	Recent	Dmp
TC	Tertiary	Includes Tcm, Tcc, Tsg
Tk	Tertiary	Units starting with "T", other than noted above
Ja	Jurassic	ALS

Alteration Codes	**Alteration Description (refers to primary and secondary types)**
Arg	Clay, hydrothermal kaolinite
AL	Acid Leach
Illite	Illite clay alteration, plus/minus smectite
Prop	Propylitic
Si	Silica

Silica Code	**Species/Description (if in veins, code in Vein columns as well)**
Ch	Chalcedonic quartz
Op	Opaline quartz
Col	Colloform quartz
Qtz	White or clear quartz
Si	Silica undifferentiated

Vein notes: Any current mineral code can be entered in the vein columns.

Modes of Occurrence (for silica, sulfur species, minerals)

P	Pervasive/flooded/massive
D	Disseminated
B	Blebs
V	Vein (Generic vein, not banded)
BV	Banded Vein
CV	Colloform Vein
Fx	Fracture coatings

Mineral Code	Description
Adl	Adularia
Aln	Alunite
Cal	Calcite
Chl	Chlorite
Cinn	Cinnabar
Gyp	Gypsum
Ill	Illite (waxy, light green, gray green)
Kao	Kaolinite (chalky white, typically in fault zones)
Ksp	Potassium feldspar
Mag	Magnetite
Mar	Marcasite
Orp	Orpiment
Py	Pyrite
Pyrg	Pyrargyrite (ruby red silver mineral)
Real	Realgar
Ser	Sericite
S	Native Sulfur
Sulfur/Sulfide Code	**Description**
Ac	Acanthite
Ar	Argentite
Ag	Silver sulfide undifferentiated (black, often anhedral, sometimes sectile)
Su	Sulfide, undifferentiated
VG	Visible gold
Iron Code	**Description**
Hem	Hematite
Lim	Limonite (generic); Goethite
Jar	Jarosite
Structure Code	**Description**
F	Fault

Texture Code	Description
Bx	Breccia/brecciated (structural, not hydrothermal), undifferentiated
Fg	Fault gouge. Clay filled interval due to faulting. Alteration should be Arg.
HBx	Hydrothermal breccia, typically result of phreatic explosion.
JBx	'Jigsaw'/Matrix breccias, may be HBx.
Lph	Lithophysa: Small voids typically lined with fine grained quartz crystals, may be evidence of boiling activity +/- phreatic event in phreatic zones.
Qpc	Quart psuedomorphs of calcite blades. Replacement of any calcite by silica.
Rbl	Rubble zone suspected to be fault caused
Sh	Shear zone/ shearing
Void	Void

Intensities (for Alteration types (Si, Arg, AL, Prop), Sulfur, Iron)

- None.
- Low, trace, mild (this will be the former 'Mixed' oxidation style).
- Moderate, abundant.
- Intense, very abundant, pervasive.
- For main and secondary alteration columns the total of intensities should not exceed 3. For example, main alteration of Si, intensity 3, leaves no 'room' for secondary alteration of Arg, intensity 2. Exceptions are when the main alteration is intensity 3, and the secondary alteration is intensity 1, with mode expressed as "B", blebs (widely spaced small particles), or "Fx" fracture coatings.
- For modes other than "B" or "Fx", the main alteration column should have the highest intensity, and the secondary alteration intensity equal or lesser than the main alteration. For example, Malt_S = 2, Malt_M = "D", 2Alt_S = 2, 2Alt_S = "D" is not allowed, but 2Alt_S = 1 is allowed.

10.3 SURVEYING

Prior to Allied Nevada drilling, drill holes were surveyed in UTM coordinates and converted to Nad 27 State Plane coordinates. Starting in 2007, drill holes were surveyed in UTM NAD 83 and converted to mine grid coordinates. From late 2008 to 2010, mine surveyors located drill holes using accurate GPS equipment, reporting directly in mine grid coordinates.

10.3.1 DRILL COLLAR SURVEYS

Standard operating procedure is for the mine surveyors to lay out planned exploration drill hole locations by accurate GPS. After drilling is completed on a site, the actual drill hole location is surveyed with the GPS and the survey data is entered into the collar file in mine grid coordinates.

10.3.2 DOWNHOLE SURVEYS

Downhole surveying of early exploration holes was not carried out on a routine basis. During the 1999 drilling program, downhole, multi-shot, gyro surveys were completed on several of the holes. Results of this work have not shown significant deviations and thus do not indicate that the lack of downhole surveys in the bulk of the exploration holes poses a problem. All downhole survey data, which is available, has been entered into the database. Current Allied Nevada practice is to contract survey the holes using gyroscopic instruments, the accepted industry practice. These instruments record orientation, deflection, and temperature. During 2009 and 2010, most holes deeper than 500 ft were surveyed.

11 DRILLING

A combination of RC rotary and core drilling techniques have been utilized to verify the nature and extent of mineralization. The majority of samples have been collected using air RC rotary drilling methods on five foot sample intervals. This method of sample collection does not indicate the true thickness of any mineralization at Hycroft. RC drilling of the Brimstone deposit through 1996 formed the basis for the ore reserve modeling, and was done with RC drilling tools utilizing a crossover sub and wet sample collection. Sample recovery was generally poor due to loss of sample into open spaces in the formation and created the potential for downhole contamination.

Modest diamond drilling programs were implemented in 1993 and 1999. The 1993 program was carried out to obtain metallurgical samples in holes that twinned earlier RC holes. These programs both indicated that the previous RC programs understated the grade of the deposit.

A 10-hole RC twin hole program was implemented in 1999 to provide a representative sampling of two ore types, acid leach and oxide, with and without elemental sulfur.

Allied Nevada commenced systematic exploration and resource development drilling in August 2007. Since then, Allied Nevada has completed 676 drill holes for a total of 534,397 ft drilled on the Hycroft project.

The drilling program accomplished the following:

- Oxide delineation.
- Wide spaced, deep sulfide exploration throughout the district.
- Close spaced sulfide delineation over Vortex and parts of Brimstone.
- Condemnation drilling for the Hycroft heap leach expansion.
- Grade confirmation on the historic Crofoot pad.
- Waste dump condemnation.
- Assays for silver.
- Metallurgical sampling in the Bay/Lewis, Cut-4, Brimstone and Vortex areas.

11.1 DRILL SAMPLE RECOVERY

Prior to the 1999 drilling, no effort was made to estimate sample recovery during RC drilling. Anecdotal evidence from several employees who worked in the lab during earlier RC drilling programs indicates that recovery was rather low, ranging from 10-15%, due to loss of sample into open space in the formation, and loss of fines due to sample overflow. Core recovery for the 1993 PQ diamond drilling averaged 86%.

In 1999, as part of a twin hole program, a total of 12 sites were selected to allow for the loss or abandonment of holes if conditions would not allow for drilling to sufficient depth. The program, as completed, totaled 5,543 ft of drilling in 11 holes. Seven of the holes were completed to the planned depth. Drill results generally indicated higher grades than the original drill hole assays.

Allied Nevada's RC drilling recoveries are good, although there are areas near Brimstone, Cut-5, South Central Fault and Boneyard where recoveries are difficult.

Most recovery problems with both core and RC were in the upper tens of feet of loose material. Where previous drilling had indicated that the upper material was unmineralized alluvium, the core drillers tricone down to a depth specified by the geologist, then start coring. This procedure increased core recovery and reduced caving hole and contamination problems. Recoveries for both core diameters are excellent, with the average core recovery for 2010 at >91%.

The ground conditions that cause difficult drilling are primarily the loose dump material and the upper acid leached material that crumbles easily. Some issues have been experienced in structurally broken material as well.

12 SAMPLING METHOD AND APPROACH

12.1 METHODS

12.1.1 REVERSE CIRCULATION SAMPLING METHODS

RC drilling of the Brimstone deposit prior to 1999 was done with RC tools utilizing a crossover sub and wet sample collection. Allied Nevada era drilling using RC methods are similar, with the addition of using a center return tricone drill bit for drilling below the water table. RC drillers clean the hole between rod changes and wait for a sample return before collecting assay samples. Drills utilized in 2010 included a Schramm 685, an Explorer Buggy Rig and a TH-75DH.

Rock chips are collected continuously down the hole, with individual samples taken over five foot intervals. Samples are submitted for assay, as collected on the rig, with standards, blanks and duplicates inserted into the sample sequence as described in the section on QA/QC. The drill crews pre number the bags, in order, representing the footage interval completed. The driller's sampler only has to keep track of the ending footage drilled with respect to the footage marked on the bags. The drill crews are provided with 20 slot chip trays, representing 100 ft total per tray, and number them with hole number, start and stop footage, and ending depth for each five foot interval. The Hycroft ore deposit is considered a disseminated ore deposit, therefore five foot samples are representative of the ore deposit at Hycroft.

RC air rotary drill sampling was completed with wet samples collected through the cyclone and a 36" rotary wet splitter discharging into five gallon buckets. A flocculent was added to each collection bucket prior to sample collection.

Drill water injection was regulated to minimize the fluid return while maintaining sufficient flow for drilling and sample return. Currently, Allied Nevada geologists provide drill crews with 20 x 24" bags. Cuttings are collected as a continuous fraction of the return stream from the drill rig by way of a rotary vane splitter. The splitter has vane covers that can be added or removed to provide the desired weight of sample for each interval. The cuttings are diverted to a clean, five gallon, plastic bucket that contains a small amount of a polymer flocculent. When a bucket is full of water and sample, it is removed and allowed to settle while another bucket is placed under the sample spout. If the drilled material contains clay, more flocculent may be added to the settling bucket and the contents stirred. When the five foot sample run is complete, the last sample bucket is removed, another clean bucket placed under the spout, and the previous interval buckets are carefully decanted and their contents poured into another bucket that holds the 20 x 24" mesh bag.

During drilling, a kitchen strainer is placed under the waste discharge spout to collect chips for the character chip tray. At the end of each run, the drill sampler fills the character box slot for the sample interval and discards the rest. The contents of the strainer are not introduced into the sample bags. The bags are placed on a plastic sheet, when freezing temperatures are expected, to prevent them from freezing to the ground and ripping when picked up. Sample bags are allowed to dry and drain at the drill site or in a holding area near the sample processing facility.

Filled chip trays are field checked for numbering accuracy during visits to the drill rig and collected by an Allied Nevada geologist for logging under a binocular microscope.

Samples are then brought down to the shipment staging area where Allied Nevada personnel finish preparing extra bags for certified standards and blanks. Insertion of blanks and standards is handled independently by geologists who create duplicate numbers at appropriate intervals, post scripted with "S" for standard, "Q" for certified quartz blanks, and "B" for blank bulk material.

The samples are then loaded in 4 x 4 x 3 ft plastic bins for delivery to the laboratory. Normally, the bins of bagged samples are picked up by the laboratory staff.

12.1.2 CORE DRILLING SAMPLING METHODS

Core drills used in 2009 included LF-70, LF-90, and LF-100 models, utilizing a wire line retrieval system and five foot stroke rod advancing systems. All drills were capable of drilling PQ sized core, and reducing as far as NQ if required.

Core drillers are responsible for obtaining a complete and representative sample of the cored interval, generally in runs not to exceed 10 ft, and in shorter increments in difficult conditions. Coring is generally begun with large diameter (PQ) rods until ground conditions dictate reducing to permit conventional diameter (HQ, 2.5") drilling. Core is recovered from the barrel by using a wire line core tube.

At the drill site, the crews place the core in waxed, cardboard core boxes, with tops and bottoms accurately labeled as to Company, Property, Hole ID, Box #, and starting and ending depths. The bottom of the core box is laid out long ways from left to right, with the marked or labeled end to the left and the unlabeled end to the right. The first portion of core is laid in the upper left hand tray, and continuously laid in the tray from left to right, advancing "down" one row as each tray is completed. The bottom of the core is terminated in the lower right corner. A wooden block is inserted at the end of each run, and in locations deemed important by the drillers, to note adverse conditions, such as caving, voids, or mislatches (situations where the core tube failed to seat properly in the core barrel). The ending block for the run is marked with an ending footage on the thin edge, and both the cut footage and recovered footage on the larger surface.

The Allied Nevada core program uses five foot or ten foot core barrels to collect samples, depending upon ground conditions. After the core is logged, it is the geologist's responsibility to determine appropriate sample intervals and boundaries. Sample intervals are representative of the mineralization at Hycroft. Original core blocks, used by core drillers to mark the end of a cored run, ordinarily serve as the primary sample boundary, subject to the rules below. Where a conflict exists between the blocks and those rules, the rules prevail and extra blocks or metal tags labeled with the depth are inserted by the geologist to indicate sample intervals.

A sample must never cross a lithologic boundary.

A sample must not cross an obvious alteration boundary, including oxidation.

A sample must not generally exceed seven feet in length.

Distinct vein zones are sampled separately.

Any core blocks that do not mark a sample boundary, for whatever reason (such as 'cave', 'loss', 'void', etc.), must be labeled in black magic marker for photographic visibility.

12.2 SAMPLE QUALITY

12.2.1 REVERSE CIRCULATION RECOVERY

RC sample recovery is excellent as judged by both field observations and recovered material weights. The average sample intervals recovered was >96% and the average weight of the material collected for a five foot sample in 2010 was 17.3lbs. Field estimates and measurements indicate that the average overall recovery volume was approximately 70%. The majority of missing samples occur in isolated zones of voids, such as in loose dumps, or in badly broken ground. Deep drilling in areas of fault zones using rotary methods was revised in late 2009 to drill such targets using diamond core methods. These factors do not materially impact the accuracy and reliability of the results.

12.2.2 CORE RECOVERY

Core recovery is measured by the ratio of length of material returned in the tube versus the total length drilled for the run and expressed as a percent. In 2010, core sample recovery was also excellent; in excess of 97% of the bedrock cored, once intersected. There have been a few instances of core loss below the bedrock contact, the majority of which are due to voids within the stratigraphy. In some areas, where historical drilling indicated that no shallow mineralization was present, the drillers were allowed to tricone drill the upper unmineralized alluvium to a depth set by the geologist (2,073 total feet were triconed in 2010, or 4% of total footage drilled). These intervals were not sampled. There is no impact on the reliability of the results based on these excellent recoveries.

12.3 SAMPLE LOCATION

12.3.1 DOWNHOLE SURVEYS

Down hole surveying is conducted by International Directional Services (“IDS”) of Elko, Nevada. Gyroscopic techniques are used to locate drill hole deviations, an industry norm. Most historic drilling was not down hole surveyed.

12.3.2 FINAL COLLAR SURVEYS

Upon drill hole completion, the Hycroft mine surveyor locates the collar coordinates of drill holes using an accurate GPS device, and reports data in the mine grid coordinate system

12.3.3 SIGNIFICANT DRILLING SINCE APRIL 1, 2010 TECHNICAL REPORT

Exploration drilling subsequent to the previous Technical Report was primarily directed towards infill and expansion of the Vortex zone. A number of drill holes intersected important intervals of mineralization. These intercepts are listed in Table 12.1 and discussed below. The locations of the holes added to the resource are shown in Figure 12.1.

Hole 10-3266 intersected 425 feet of mineralization grading 0.013 opt Au and 2.0 opt Ag (0.050 opt AuEq) starting at a depth of approximately 1,380 feet. Included in the 425 feet intercept is a 34 foot interval grading 0.031 opt Au and 12.6 opt Ag (0.264 opt AuEq). Nearer surface, a high-grade intercept of 75.5 feet at 0.013 opt Au and 2.5 opt Ag (0.058 opt AuEq) was also intersected.

Hole 10-3286, which was drilled on the southern boundary of the Vortex drill pattern, intersected 583 feet of mineralization grading 0.045 opt AuEq (0.019 opt Au and 1.5 opt Ag). This hole extends known Vortex mineralization to the south.

Hole 10-3288, drilled on the far west boundary of the Vortex Zone, intersected 835 feet grading 0.068 opt AuEq (0.021 opt Au and 2.7 opt Ag). This hole was collared 1,175 feet west of hole 10-3659. The intercept includes an interval of higher grade mineralization (470 feet grading 0.102 opt AuEq) that occurs in the hanging wall of the East Fault. The mineralization intersected in hole 10-3288 indicates significant potential to further expand the Vortex Zone westward.

Hole 10-3382 intersected 1,498 feet grading 0.046 opt AuEq (0.022 opt Au and 1.4 opt Ag) including a 510 foot interval grading 0.089 opt AuEq (0.041 opt Au and 2.7 opt Ag) . This intercept includes a zone of higher-grade mineralization (230 feet grading 0.136 opt AuEq) that is present in the hanging wall of the East Fault. Hole 10-3382 has been drilled as a 180-feet step-out vertical hole from hole 10-3659. Both of these holes have intersected long intervals of mineralization that enclose higher grade mineralization in the hanging wall of the East Fault.

Hole 10-3659, a vertical core hole drilled approximately 160 feet west of the Albert Fault in the central region of the Vortex Zone, intersected 565 feet of mineralization grading 0.032 opt Au and 3.6 opt Ag (0.096 opt AuEq). This intersection includes 105 feet grading 0.087 opt Au and 9.7 opt Ag (0.256 opt AuEq). The high-grade mineralization in hole 10-3659 occurs in the hanging wall of the East Fault.

Hole 10-3833/10-3353 intersected 461 feet grading 0.044 opt AuEq (0.012 opt Au and 1.8 opt Ag), including a 105 foot higher grade intercept of 0.135 opt AuEq (0.007 opt Au and 7.3 opt Ag). This hole was rotary drilled (3833) to a depth of 1,260 feet and then cored (3353) another 715 feet. 10-3833/10-3353 was collared in the southwestern region of the Vortex drill pattern, and confirms high-grade silver mineralization in the hanging wall of the East Fault in that region.

Hole 10-3840 was collared from the same station as hole 10-3382, and drilled at -75 degrees to the east. The hole intersected 1,164 feet grading 0.033 opt AuEq (0.014 opt Au and 1.1 opt Ag).

Hole 10-3843, a 325 feet step-out hole from 10-3659 intersected 541 feet grading 0.055 opt AuEq (0.022 opt Au and 1.9 opt Ag) including a 122 foot interval grading 0.107 opt AuEq (0.024 opt Au and 4.7 opt Ag).

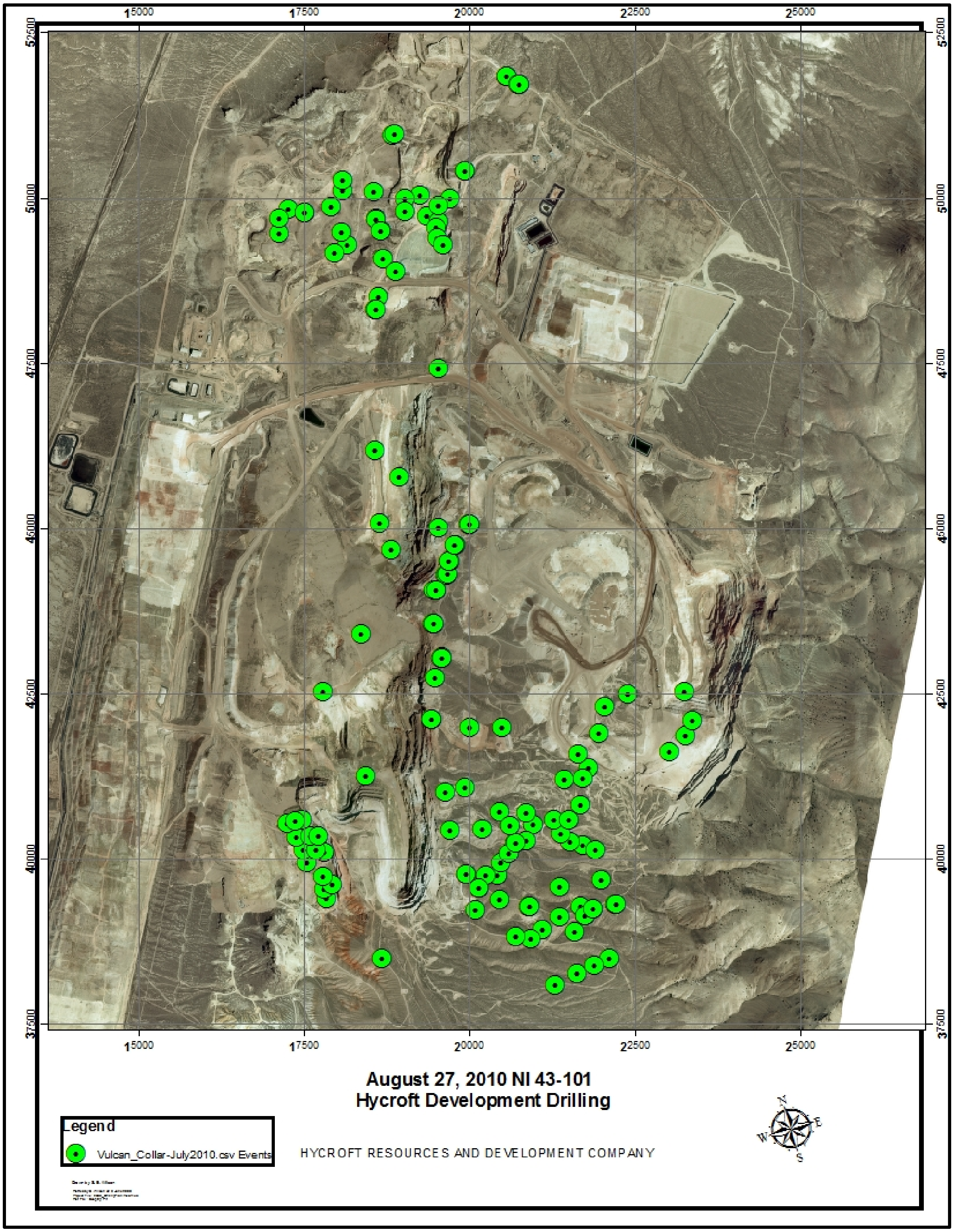
Hole 10-3848, collared approximately 1,000 feet northeast of hole 10-3659, intersected 135 feet grading 0.020 opt Au and 2.8 opt Ag (0.071 opt AuEq). The intersection in this hole includes 55 feet grading 0.036 opt Au and 5.7 opt Ag (0.140 opt AuEq).

Table 12.1 Significant Drilling Intercepts

-	From (ft)	To (ft)	Interval (ft)	Au opt	Ag opt
H10D-3266					
	231	306.5	75.5	0.013	2.5
&	1379	1804	425	0.013	2.0
including	1507	1541	34	0.031	12.6
H10D-3286					
	514.5	1097	583	0.019	1.5
including	779	897	118	0.046	1.0
H10R-3288					
	1235	2070	835	0.021	2.7
including	1275	1745	470	0.031	4.1
including	1630	1705	75	0.022	15.8
H10D-3382					
	554	2051.5	1497.5	0.022	1.4

-	From (ft)	To (ft)	Interval (ft)	Au opt	Ag opt
including	1469	1979	510	0.041	2.7
including	1488.5	1718	229.5	0.043	5.3
H10R-3659					
	469	1945	1476	0.018	1.6
including	1380	1945	565	0.032	3.6
including	1525	1630	105	0.087	9.7
including	1525	1545	20	0.191	8.4
including	1570	1600	30	0.120	13.7
H10R-3833/H10D-3353					
	690	760	70	0.013	0.2
&	810	875	65	0.012	0.2
&	915	1376	461	0.012	1.8
including	1271.5	1376	104.5	0.007	7.3
H10D-3840					
	239	315	76	0.024	2.6
&	496	1660	1164	0.014	1.1
including	1387	1650	263	0.019	2.6
including	1552	1614	62	0.016	5.1
H10D-3843					
	1039	1580	541	0.022	1.9
including	1423	1545	122	0.024	4.7
including	1462	1489	27	0.037	9.4
H10R-3848					
	345	480	135	0.010	0.6
&	750	770	20	0.015	0.2
&	1270	1405	135	0.020	2.8
including	1295	1350	55	0.036	5.7

Figure 12.1 New Drilling since April 1, 2010 Technical Report



13 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

13.1 SAMPLE PREPARATION

The sample preparation procedure prior to 1999 was not documented. Sample preparation in 1999 consisted of drying, crushing, splitting, and pulverizing the split.

Currently, samples are prepared from a split of 70% passing -6mm if pieces are too large to fit in the pulverizer, and further crushing of 70% passing -2mm. A one kilogram split is taken and pulverized to 85% passing -75 micron.

Sample preparation by Allied Nevada personnel is limited to site technicians who saw core samples. Some uncut core is hand delivered to contract core cutters in Elko and Winnemucca who provide a handwritten receipt for delivery. Some core is shipped whole to ALS Chemex ("Chemex") in Reno and cut by their staff. After cutting, a transmittal sheet is prepared for submission to the laboratory. The geologists provide the sample prep technicians with a list containing the drill hole number and the appropriate sample intervals. The intervals of "no sample recovery" are identified, tagged, and accounted for separately in the sample lists so that the lab reports them as "no sample" rather than "0" or some other arbitrary value.

The sampling operation avoids bias, wherever possible, by cutting the core in half perpendicular to the trace of the visible bedding. When prominent veins are noted during logging, the geologist will mark the trace of the cut to ensure a representative sample. The portion to be saved remains in the core box, in its proper position, with core blocks in place. Core boxes are stacked on pallets for storage. The split portion of core is bagged and shipped in bins to the lab.

No officers, directors, or associates of the issuer are involved with the sample preparation process.

13.2 ASSAY METHODS

Prior to 1992, all samples were sent to Barringer Laboratories, Golden, Colorado, ("Barringer") for fire assay. Selected intervals were cyanide soluble analyzed. From 1992 to 1999, samples were processed at the Hycroft laboratory. After 1999, samples were sent to outside laboratories for processing.

From 1992 to 1999, all of the samples were analyzed for cyanide soluble gold and silver at the Hycroft laboratory. The method employed at Hycroft is a non-standard procedure that has been developed to provide a semi-quantitative measurement of recoverable gold.

Allied Nevada currently uses two laboratories for assay analysis. The companies are Chemex and Inspectorate Laboratories ("Inspectorate"), both located in Reno, Nevada. Chemex is ISO9001:2000 compliant and has ISO 17025 accreditation. Inspectorate has 9001:2008 certification.

13.2.1 PRECIOUS METAL FIRE ASSAY ANALYSIS

All Allied Nevada drill samples have been routinely assayed for gravimetric gold and silver; however earlier operators primarily fire assayed only for gold.

As silver contributes a significant value to many mineralized intervals, Allied Nevada conducted a silver fire assay analysis program on 13,168 historical drill pulps, sampled from 185 individual holes. These pulps had been stored in a drill material shed at the Hycroft mine and were in good condition when collected in November 2007. Each sample had been labeled with the drill hole number and from/to depth. Collar coordinate and downhole survey data, where available, are recorded in the historical drill hole database. Only those intervals representing presently un-mined material were selected for silver fire assay analysis. Standard and blank samples were submitted with the drill hole pulps to evaluate the analytical quality.

13.2.2 CYANIDE SOLUBLE PRECIOUS METAL ANALYSIS

The following hot cyanide analytical procedure, originally developed by the Hycroft lab, was conducted on most pre-Allied Nevada drill intervals:

- The sample pulps are blended on a roll cloth and 20 grams are stippled out and placed in 50 ml plastic centrifuge tubes.
- 20 grams of 20 lb per ton NaCN solution containing 20 lb per ton of NaOH are dispensed into each tube.
- The tubes are capped and shaken until homogenized. The tubes are then inserted in racks that are placed in an agitating water bath at a temperature of 160°F. The racks are placed so the centrifuge tubes are in a horizontal position.
- The tubes are shaken at a moderately slow speed, approximately 60 rpm on the eccentric, for one hour.
- The sample tubes are removed from the water bath, allowed to cool for several minutes, and then centrifuged.
- The liquid phase is then analyzed for gold and silver using AA spectrophotometry.

This methodology has been consistent through the life of the project and has proved to be reliable based on metallurgical testing and production results.

In 2009, Allied Nevada assayed approximately 18,000 pulps from previous drilling, for gold and silver, using a hot cyanide method modified from the Hycroft method by Chemex (see below). All 2009 and 2010 drill hole samples were routinely analyzed using this method.

- Samples are weighed, dried and re-weighed.
- A one kg split of sample, pulverized to better than 85% passing 75 microns (PREP-31B).
- Au-Ag fire assay followed by gravimetric AA finish (Au-GRA21; detection range 0.05 to 1,000 ppm).
- Au-Ag hot cyanide assay (Au-AAHmc13, Ag-AAHmc13, detection: 0.03 – 1,000 ppm).
- Multi-element ICP by 4-acid 'near-total' digestions (ME-ICP61; detection range varies by element).
- Hg by aqua regia digestion, AAS (Hg-CV42; detection range 0.01 to 100 ppm).

The hot cyanide method modified by Chemex generates results that match the previous Hycroft method. The method is detailed in Table 13.1.

Table 13.1 Hot Cyanide Method

Item	Amount	Notes
Quantity Pulverized	1,000	grams
Sample Size	30	grams
Solution Quantity	30	ml
Solution/Solid Ratio	1	
NaCN Solution Strength	1%	
NaOH Solution Strength	1%	
Temperature	158	F
Particle Size/Pulverize	85%	-200 M
Pulverizer	ring and puck	
Time	1	hour

The method process is as follows:

1. Samples received from the sample preparation area are first logged on to an assay ticket with any discrepancies noted and resolved with the supervisor prior to assaying.
2. Every 10th sample is duplicated and a control sample is completed with each load of 36 analysis.
3. The samples are blended on a rolling cloth and 20 grams is stippled out and placed in a 50 ml plastic centrifuge tube.
4. 20 grams of 20 lb per ton NaCN solution containing 20 lb per ton of NaOH is dispensed into the tubes. The automatic dispenser is routinely calibrated using a top loading balance.
5. The tubes are capped and shaken until homogenized. The racks containing the tubes are placed into an agitating water bath at a temperature of 160°F and are placed horizontally.
6. The samples are then removed from the water bath shaker, cooled for several minutes and centrifuged.
7. The liquid phase of the sample is assayed using AA spectrophotometry using similar matrix standards.

13.2.3 ICP MULTI-ELEMENT ANALYSIS

All drilling by Allied Nevada in 2007 and 2008 was assayed using a 35 element, total digestion, multi-element method, generating 56,327 individual ICP assays. As this program provided a broad distribution of trace element data, only a limited number of samples were assayed by ICP in 2010. In addition, a limited number of sample intervals were assayed for trace elements during previous drill campaigns.

13.2.4 ICP PLOTS

Table 13.2 shows the global mean averaged analyses for several ICP elements.

Table 13.2 Mean Values of ICP Elements

Element	As	Cr	Cu	Fe	Hg	Mo	Pb	Sb	Zn
Units	Ppm	ppm	Ppm	%	ppm	ppm	ppm	ppm	ppm
No. Samples	54,002	54,002	54,002	54,002	53,759	54,002	53,094	54,002	54,002
Mean	119.3	23.8	20.0	2.5	6.1	2.8	16.7	53.6	61.6
STD	171.5	33.7	64.3	1.5	58.5	13.1	28.6	103.6	91.1

The spatial distribution of ICP elemental values is shown on the following plots of drill hole traces. Hotter (red) colors indicate higher values, while cold (blue) colors indicate relatively low values. The scale of values is individual to each plot, and cannot be used to compare absolute values between plots.

Figure 13.1 Arsenic Distribution

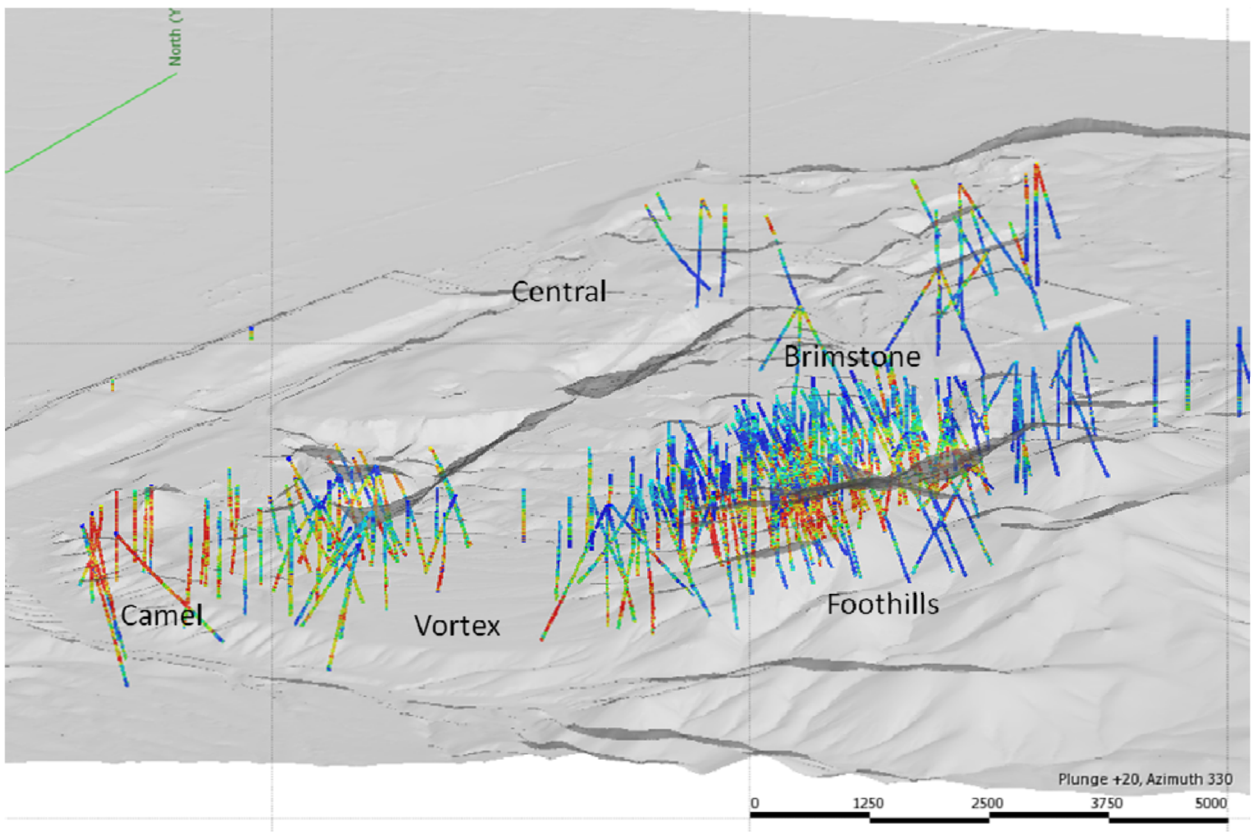


Figure 13.2 Copper Distribution

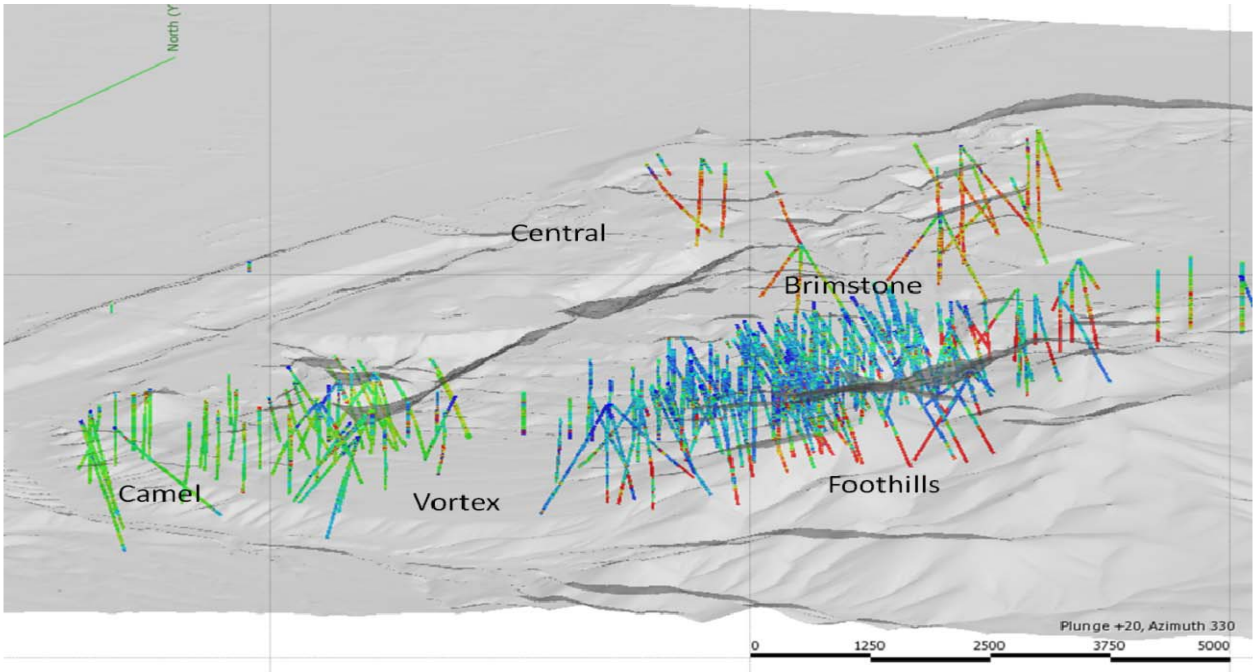


Figure 13.3 Mercury Distribution

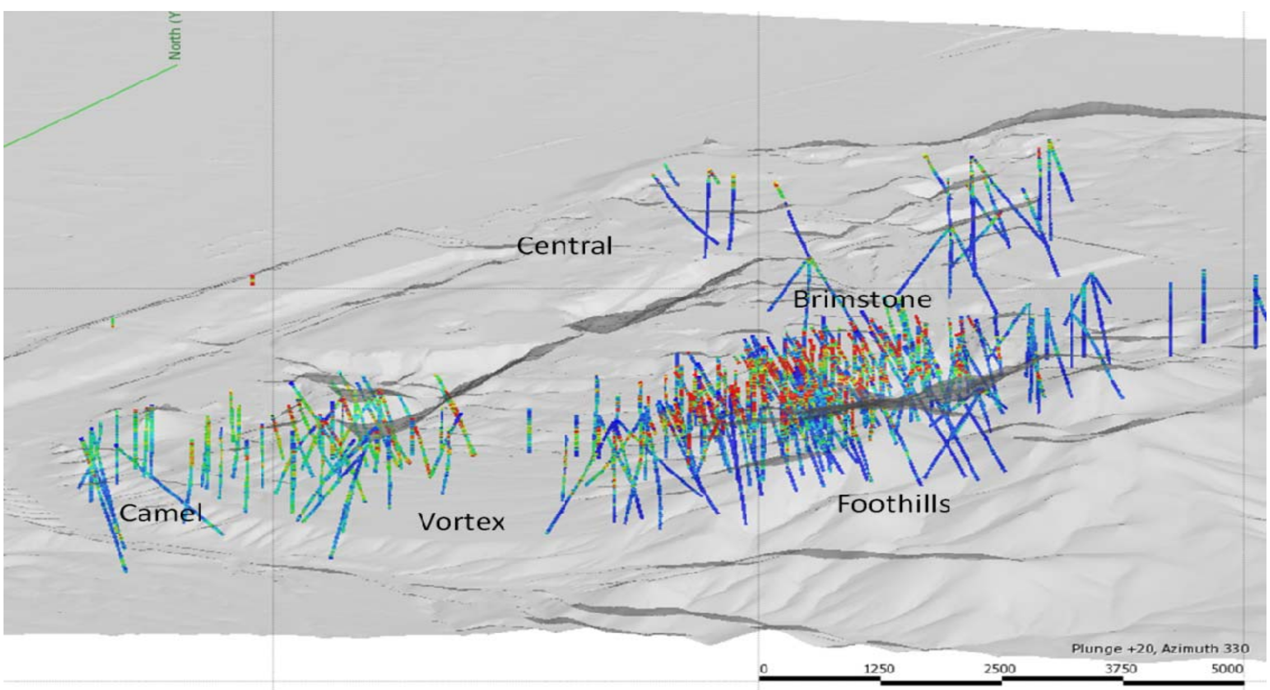


Figure 13.4 Lead Distribution

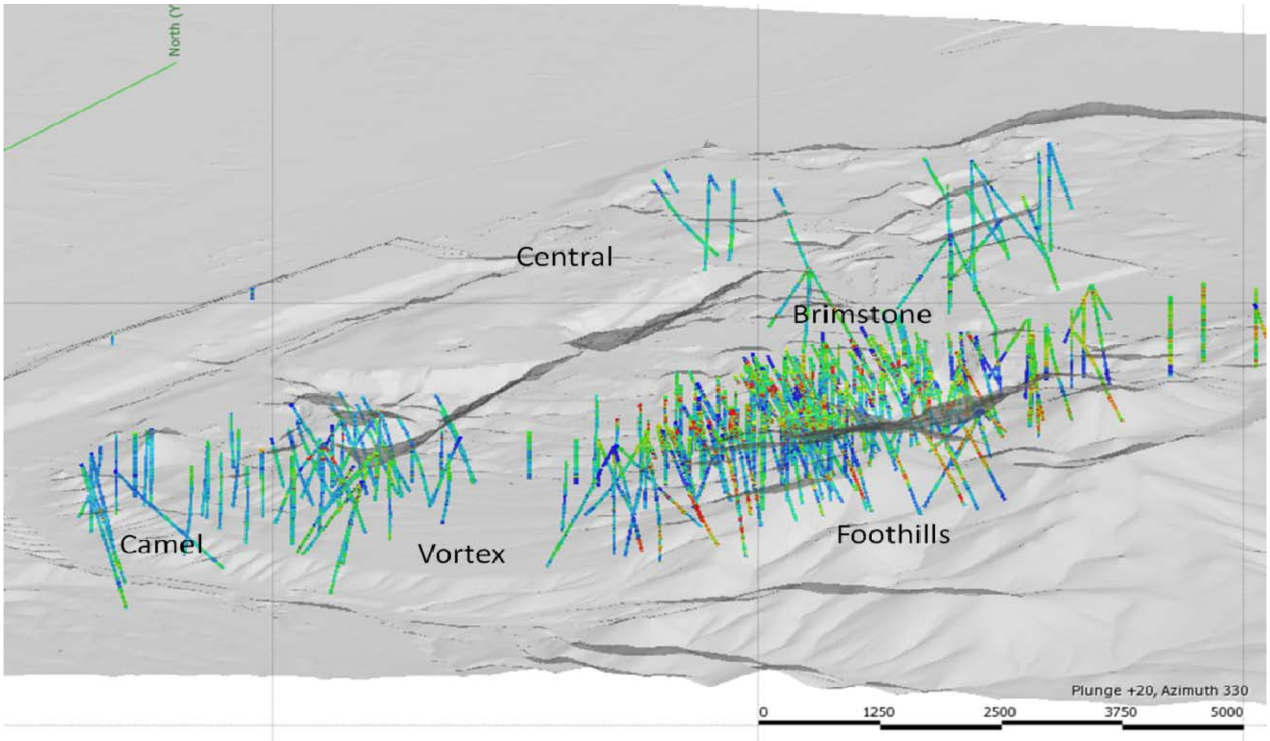


Figure 13.5 Antimony Distribution

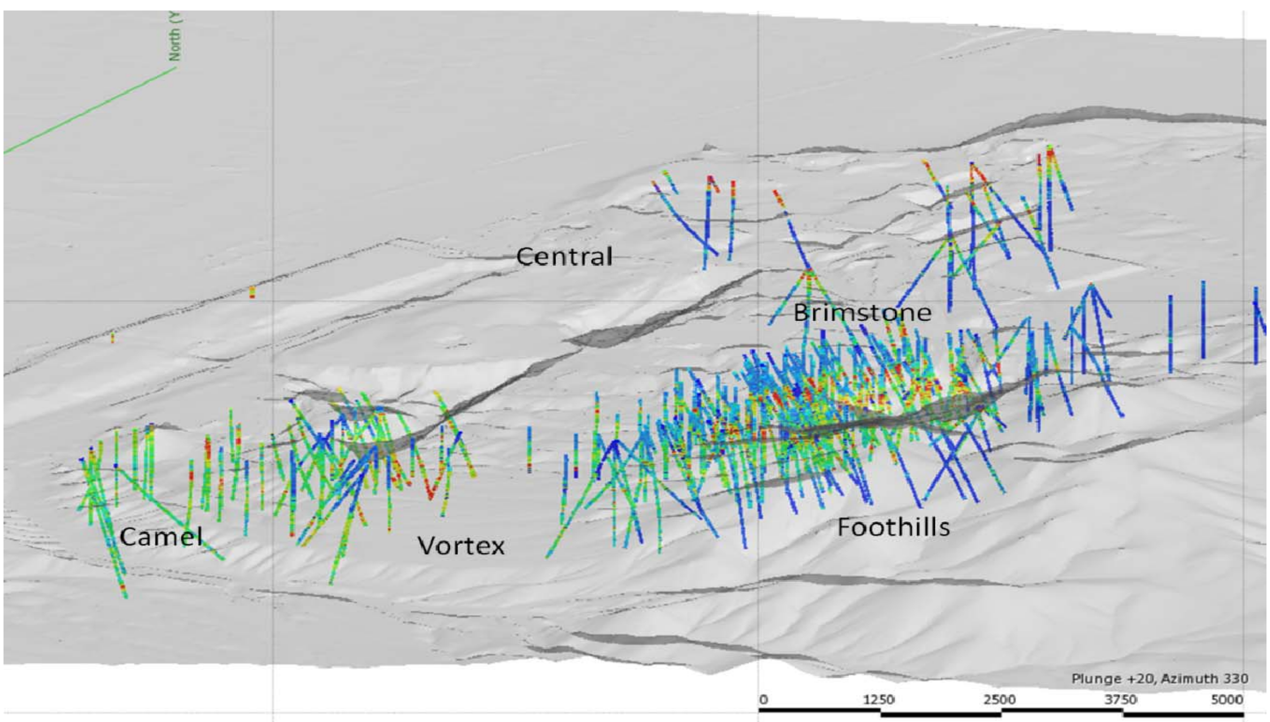
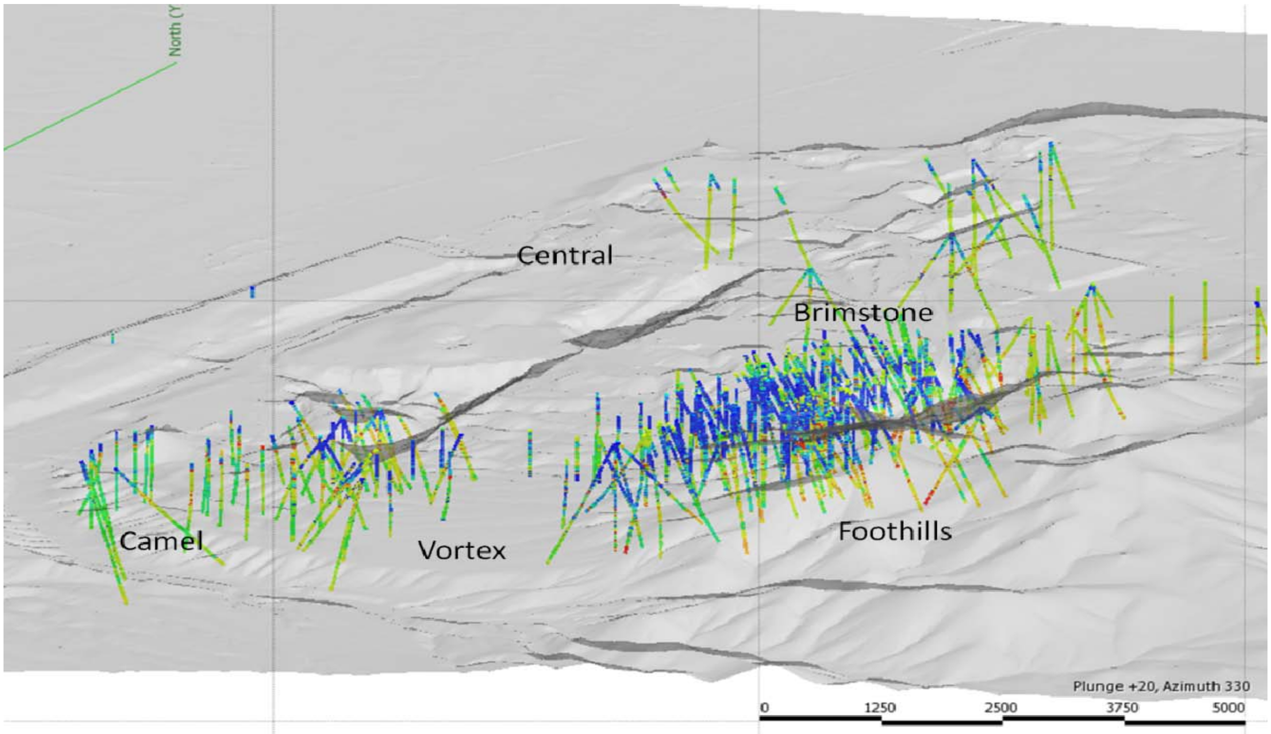


Figure 13.6 Zinc Distribution



13.3 SAMPLE SECURITY

Samples are delivered to the analytical laboratory in the numbered bags, along with a transmittal sheet stating whether the samples are “cuttings” or “core”, the list of sample numbers, and the total sample count. The lab has no knowledge of the spatial reference of the individual samples, beyond knowing the footage of a particular hole.

By inspection of the submitted sample bags, the lab can identify which are blanks, samples of certified quartz sand (post scripted by “Q” in Kraft envelopes), and standards (pulp powder post scripted by “S” in Kraft envelopes), but has no knowledge of the accepted value of each. A variety of certified standards are submitted, ranging from 0.2-9.0 ppm gold. In addition, a sample of unmineralized rock (marble, granite, or scoriaceous lava) is submitted as the blank and is the first sample in each drill hole group of samples.

At present, site sawn samples are delivered to Chemex or Inspectorate in Reno, Nevada by Allied Nevada staff, or picked up by Chemex or Inspectorate drivers for delivery to their facilities in Reno (both labs) and Elko (Chemex only).

Chain of custody is established by transmittal sheets, sample receipt documents from the lab, and by work orders and certificates.

An Allied Nevada copy of the transmittal sheet is stored in a file cabinet in the corporate office in Reno, Nevada. Once assays have been received, a copy of the assay sheet is stored with the drill logs and the original with the transmittal sheets. The transmittal sheets are indexed by job number.

Copies of the sample sequence list, the lithology log, and assays are stored on a dedicated Hycroft computer and indexed by hole number. Originals of all logs and assays are stored in file cabinets on a per hole basis, indexed by hole number. Allied Nevada personnel contact the lab to obtain a job number assignment for whole or partial hole shipment and arrange for sample pickup by the lab's driver.

Coarse reject material from RC drilling and sample pulps are returned to the Hycroft mine by laboratory staff and stored on site. Access to the sample preparation and storage area is limited to geologic staff.

13.4 ANALYTICAL RESULTS

Following analysis, results are posted to a digital laboratory database on which Allied Nevada has secure permission privileges. Managers download the data where the sample results are cross referenced to sample numbers. Each drill hole carries a unique self-identifying sample number, simplifying the cross referencing. The completed digital file for each drill hole is emailed to Allied Nevada by the lab, and a follow up, hard copy certificate is mailed to company offices.

Data is proofed checked by geologists and loaded into SQL Server data tables. The data is further checked using electronic methods, and then translated into ounce per ton values and loaded to the modeling database for display and visual QA/QC checking.

13.5 QA/QC CHECK SAMPLES AND CHECK ASSAYS

13.5.1 HISTORICAL QA/QC PROGRAM

Until 1992, selected mineralized intervals were analyzed for cyanide soluble gold and cyanide soluble silver by Barringer. Barringer's successor company was unable to provide details of the methodology used during this period.

All exploration samples subsequent to 1991 that were assayed for cyanide soluble gold and cyanide soluble silver were assayed at the Hycroft laboratory. Fire assays were also performed. No decipherable QA/QC data exists for these assays. All samples in the 1999 RC twin drill hole program were fire assayed for gold by Chemex and the Hycroft lab.

13.5.2 ALLIED NEVADA QA/QC PROGRAM

The Allied Nevada QA/QC program includes analysis of standard reference materials ("standards"), inert blanks, and duplicate pulps, as well as check assays by umpire laboratories. The program was designed to ensure that at least one standard and one blank is inserted into the drill sample stream for every 40 drill samples, which are the number of Allied Nevada samples in each AAL analytical batch. In practice, the insertion rates for the QA/QC samples are somewhat higher.

13.5.2.1 Certified Standards

Standards are used to evaluate the analytical accuracy and precision of the assay laboratory during the time the drill samples were analyzed.

Allied Nevada acquired four certified reference standards from Minerals Exploration and Environmental Geochemistry of Reno, Nevada ("MEG") for use in their 2009 and 2010 Hycroft drilling programs (Table 13.2). These standards have a range of certified gold values that is representative of the deposit. Four standards were prepared by MEG from both high and low grade mineralized material from the Hycroft deposits for use in the 2009 and 2010 drilling programs (results not discussed herein).

The standards were assigned sample numbers in sequence with their accompanying drill samples, and were inserted into the drill sample stream of all holes from the 2010 drilling program for which assay data was included in the resource update. The Hycroft Chief Geologist and CPG compiled 331 analysis of these standards and 302 analysis of blanks, which were inserted into the sample sequence of the holes drilled by Allied Nevada. This equates to an insertion rate of one standard or blank for every 26 drill samples (there are a total of 16,406 drill sample assays for these holes in the resource database). The compilation included all control samples for which data was received from the lab. All of the analysis were completed by Chemex.

Table 13.3 Certified Standards

Standard	Standard Source	Certified Value (ppm)	Standard Deviation
Cove - 1	MEG	0.475	0.088
Cove - 2	MEG	0.663	0.152
Cove - 3	MEG	0.848	0.071
Cove - 4	MEG	2.058	0.294

The following discussion of the standard results includes graphical representations of the data. These graphs show the distribution of the assay certificates ordered along the x-axis, the gold grade of the standard assays on the y-axis, the certified or accepted values of the standards as green line and vertical bars, and plus three standard deviation limits of the standards as red lines, respectively. The Chemex analysis are purple stars. Moving range charts demonstrate the analytical lab's ability to reproduce the standard over time by graphing the absolute value of successive differences in data points.

In the case of normally distributed data (note that most assay data sets from metal deposits are positively skewed), 95% of the standard analysis should lie within the two standard deviation limits of the certified/accepted value, while only 0.3% of the analysis should lie outside of the three standard deviation limits. As it is statistically unlikely that two consecutive samples would lie outside of the two standard deviation limits, such samples are considered failures unless further investigation proves otherwise. All samples outside of the three standard deviation limits are considered to be failures. Failures should trigger laboratory notification of potential problems and a re-run of all samples included with the failed standard result.

The 331 assays from the MEG standards are presented in Figures 13.7 through 13.10. These standards were submitted with samples from the holes in the 2009 sequence contained in the resource update.

Figure 13.7 Cove -1 Standard Results

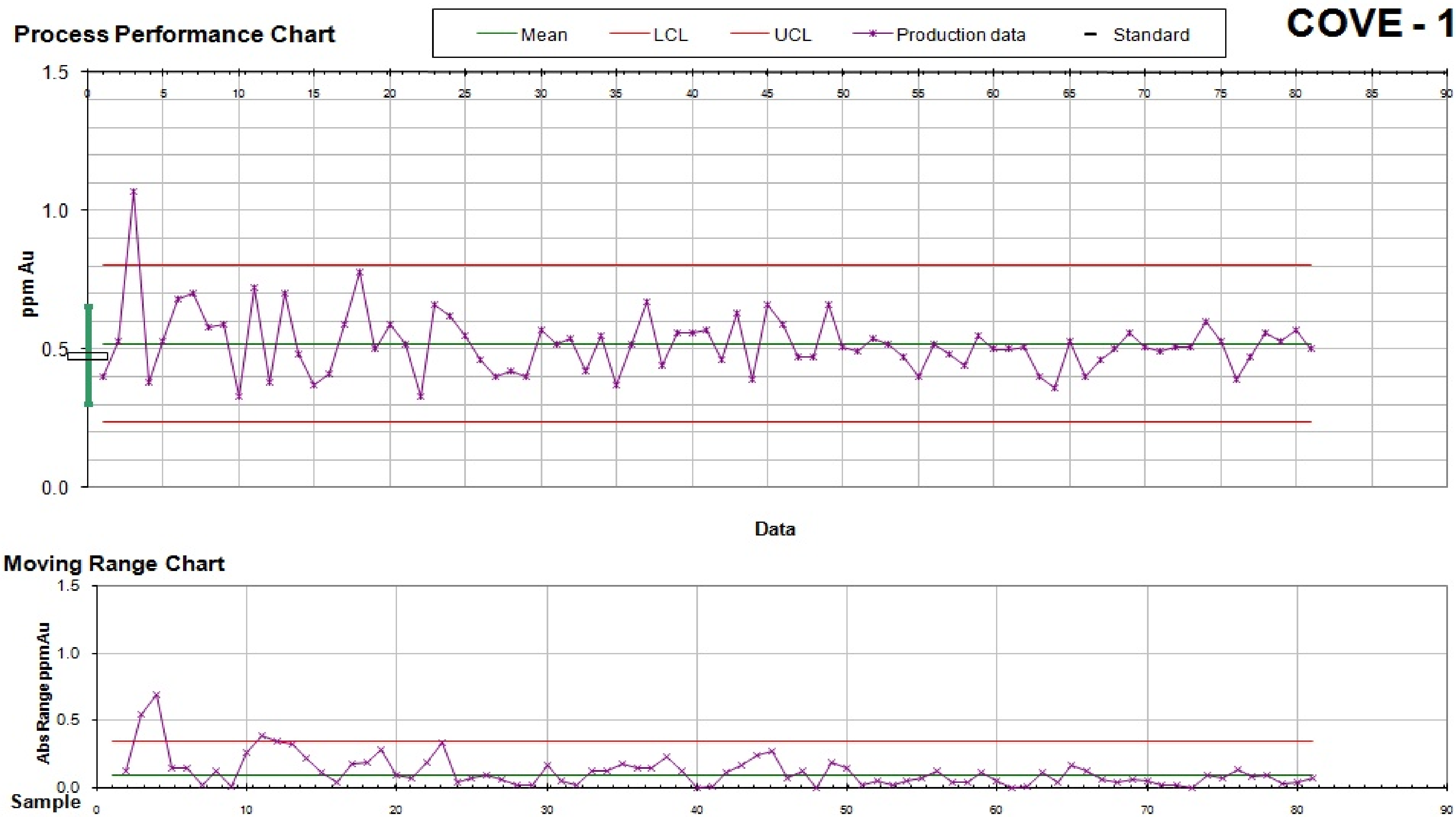


Figure 13.8 Cove-2 Standard Results

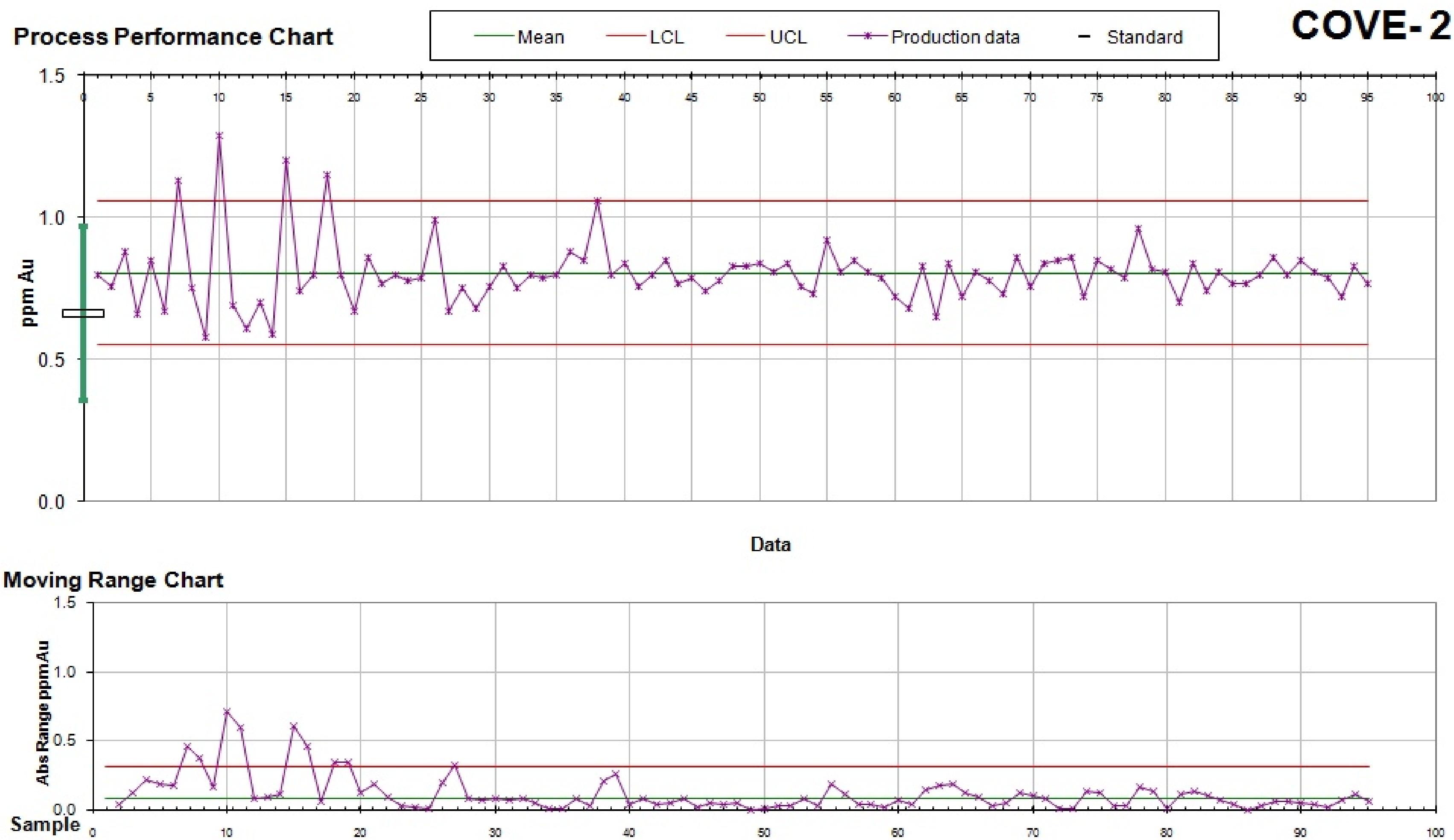


Figure 13.9 Cove-3 Standard Results

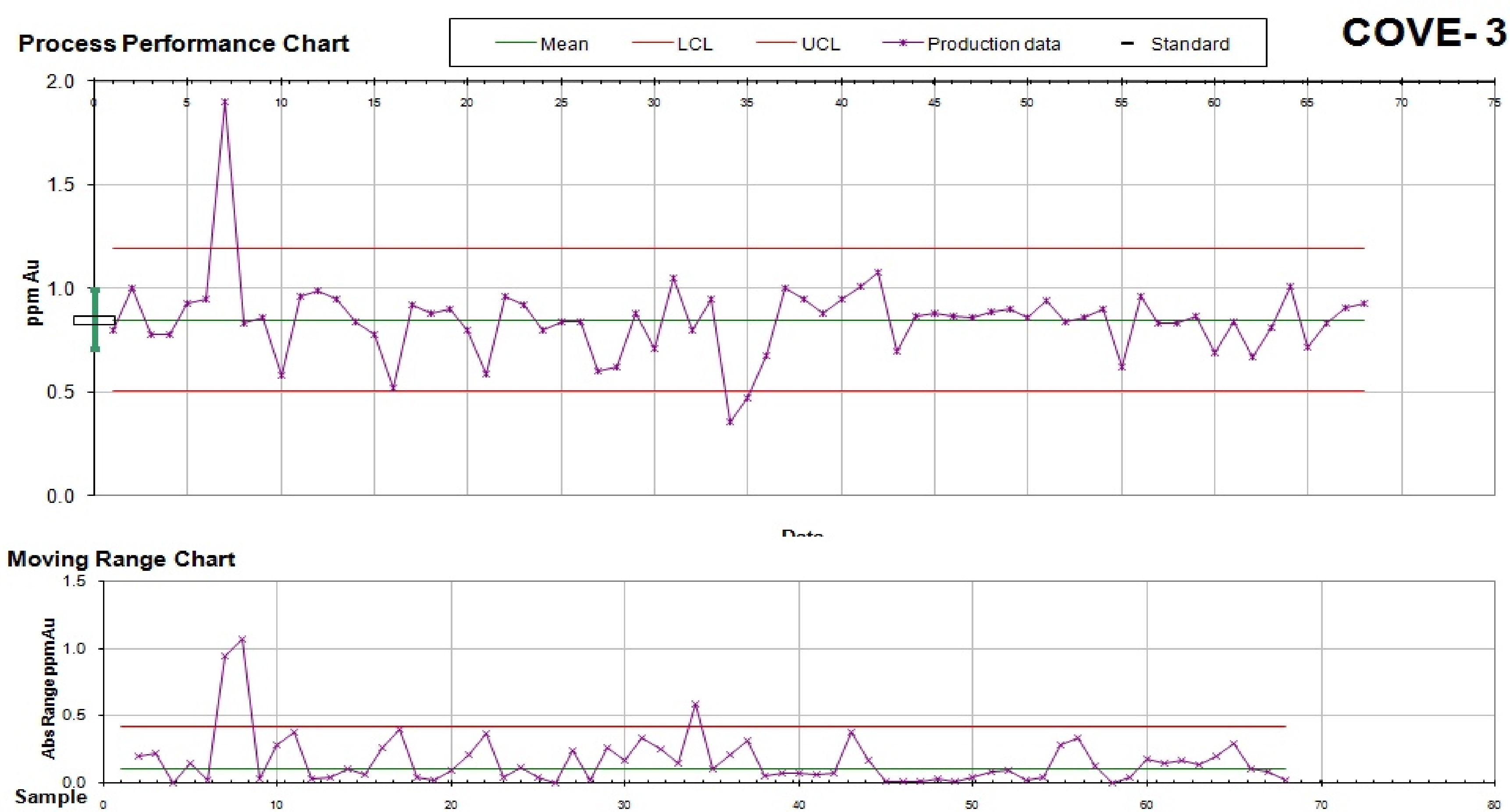
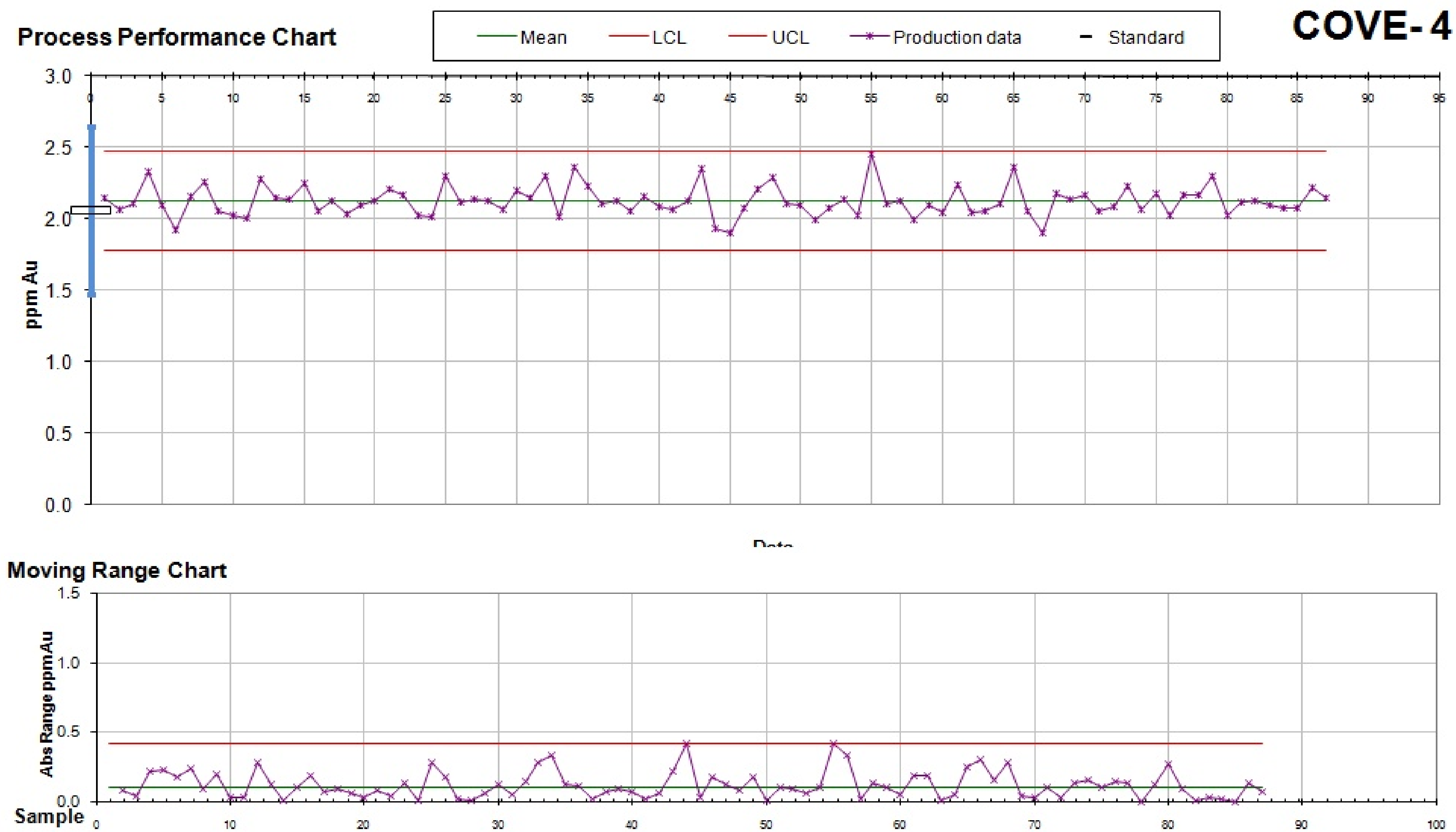


Figure 13.10 Cove-4 Standard Results



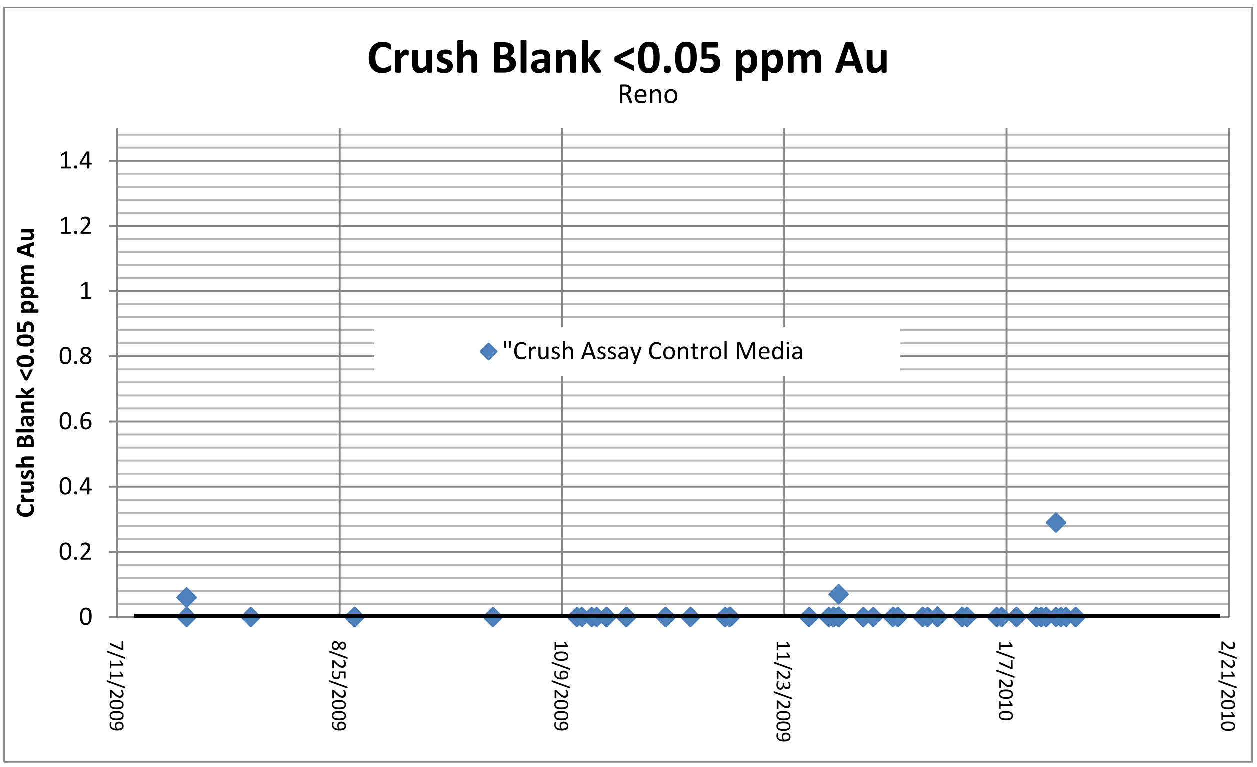
13.5.2.2 Certified and Bulk Blanks

Two types of blanks were submitted for QA/QC with assay samples during the 2009 drilling program at Hycroft; crusher blanks and certified washed quartz sand. Crusher blanks are coarse, inert, non-mineralized material, such as a barren marble or granite control sample.

A total of 121 crusher blanks were submitted during the drilling program at Hycroft, which equates to approximately one blank submitted with each work order processed by Chemex. Of this total, 11 samples were processed by the Elko, NV prep facility and 110 samples were processed in Reno, NV. All of the crusher blanks were assayed at the Chemex lab in Reno, Nevada.

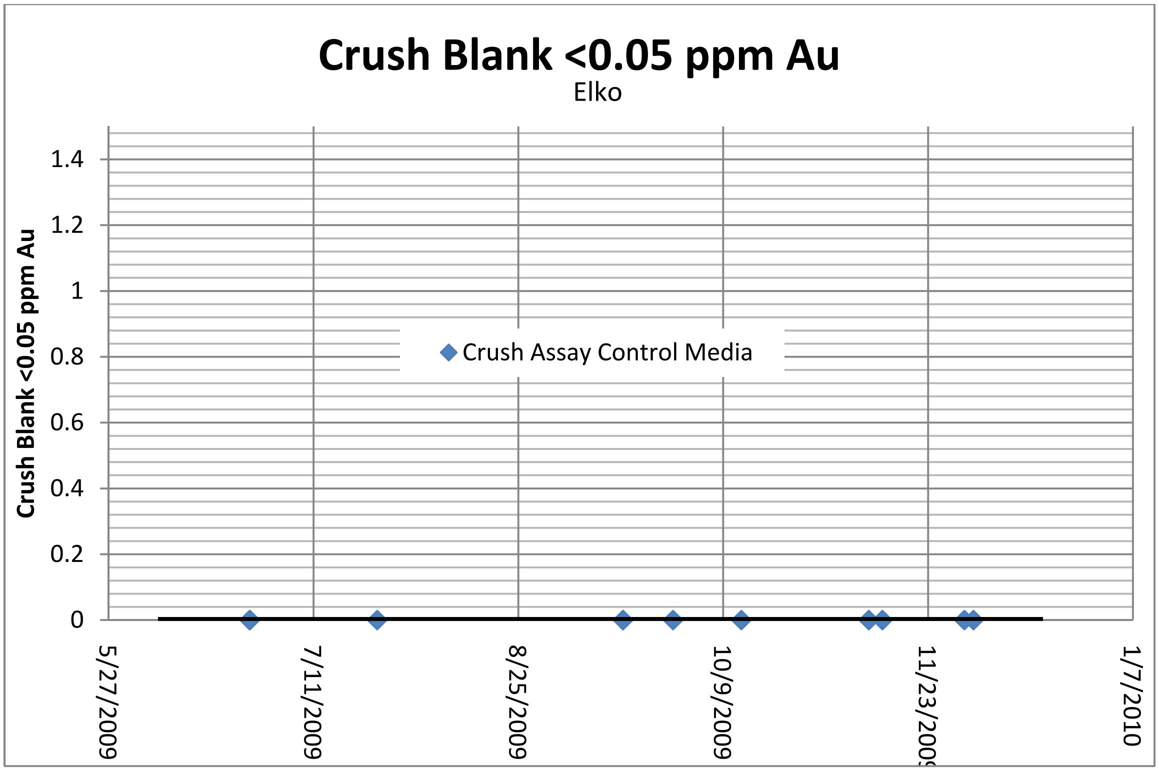
Of the 11 samples processed in Elko, NV, there were no failures for a reproducibility rate of 100%. Of the 110 samples processed at the Reno, NV facility, there were three failures for a reproducibility rate of 97%. Two of the failures were very close to the lab detection limit and fall within the range of analytical error. Figures 13.11 and 13.12 illustrate the results of the analysis.

Figure 13.11 Crushed Blank Analysis Results – Reno Laboratory



N=11

Figure 13.12 Crushed Blank Analysis Results – Elko Laboratory

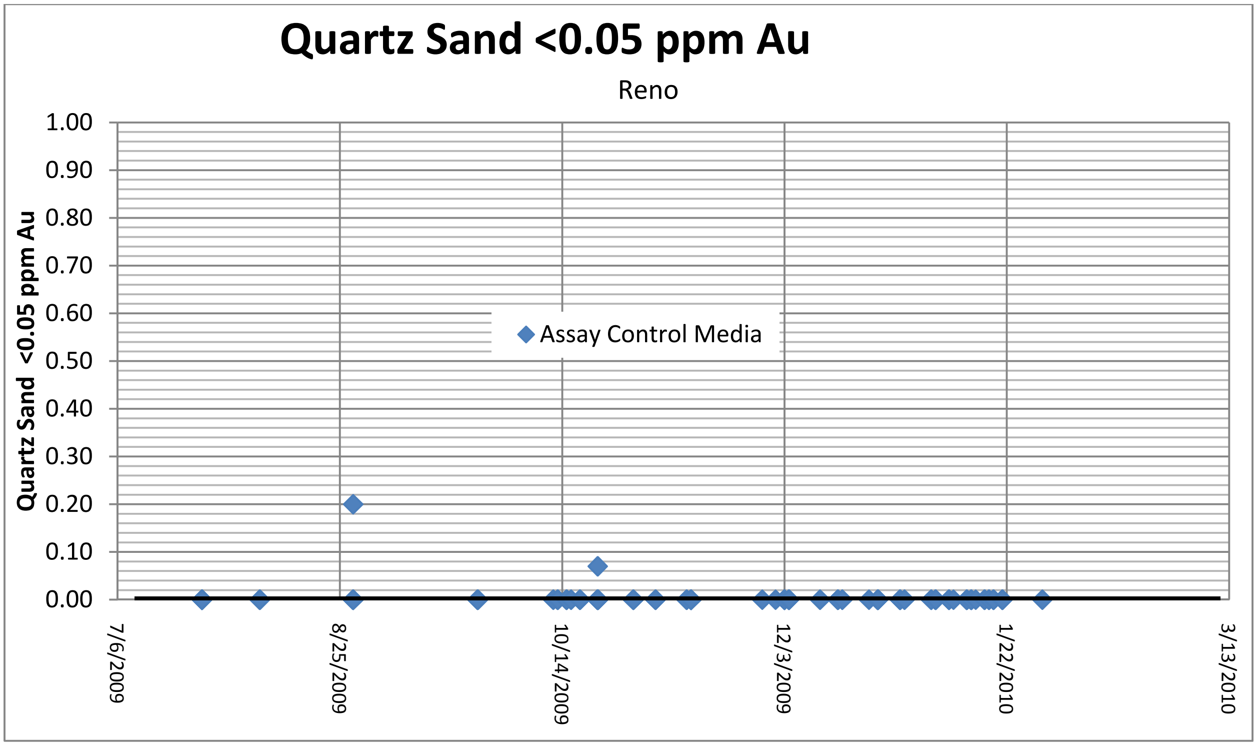


N=11

A total of 301 quartz sand blanks were submitted with assay lots of samples from the 2009 drilling program at Hycroft. No data was received for 11 of these samples and they have been excluded from the population. Of the 290 remaining samples, 37 were processed at the Chemex prep facility in Elko, NV, and assayed in Reno, NV. 253 blanks were shipped directly to the Reno lab for assay.

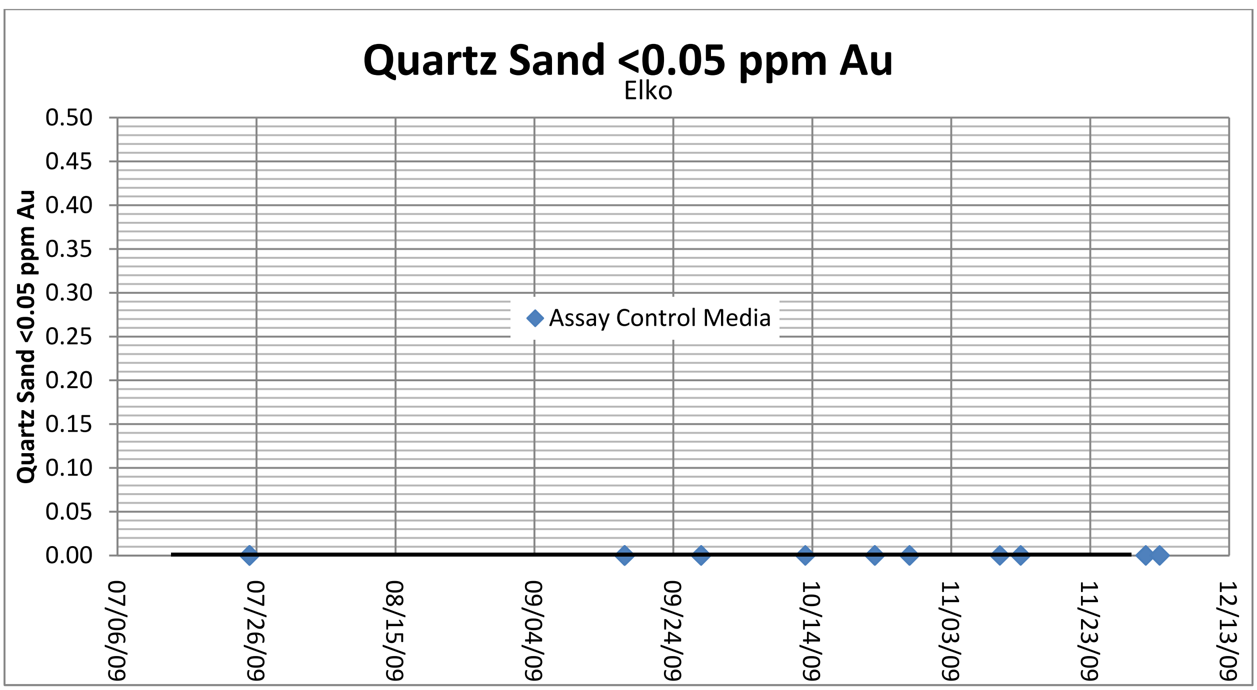
For the 37 samples shipped to the Elko facility, there were no failures reported for a reproducibility rate of 100%. Of the 253 samples shipped and processed in Reno, there were three reported failures for a reproducibility rate of 99%. All 290 quartz sand blank assays for which data was reported were conducted in Reno. Figures 13.13 and 13.14 illustrate the data analysis.

Figure 13.13 Quartz Sand Analysis Results – Reno Laboratory



N = 253

Figure 13.14 Quartz Sand Analysis Results – Elko Laboratory



N = 37

13.5.2.3 Duplicate Assay Program

Pulped samples totaling 200 RC and 200 core were selected at random from 2010 drilling. The criterion for selection was an initial gold assay value >0.005 opt, as determined by Chemex. The re-assaying process is in progress.

13.5.2.4 Assay Methods

The majority of samples were analyzed by Chemex using the method summarized in Table 13.4.

Table 13.4 Chemex Analysis Methods

Element	Method Name	Description	Level of Detection
Gold	MEGRA21	Fire with gravimetric finish	0.05 ppm
Silver	MEGRA21	Fire with gravimetric finish	5.00 ppm
Gold	13Hmc	Hot cyanide, AA finish	0.03 ppm
Silver	13Hmc	Hot cyanide, AA finish	0.03 ppm

Inspectorate methods for gold and silver are summarized in Table 13.5.

Table 13.5 Inspectorate Analysis Methods

Element	Method Name	Description	Level of Detection
Gold	2-FA-11	Fire with gravimetric finish	0.003 opt
Silver	2-FA-11	Fire with gravimetric finish	0.01 opt

The sample preparation, security and analytical procedures are adequate and within industry accepted norms for a Nevada mine such as Hycroft.

14 DATA VERIFICATION

14.1 INTEGRITY OF DATABASE

Pre-Allied Nevada assay data was verified by SRK in 2008. Assay certificates were visually checked against the electronic database, and errors resolved. Source documents consist of photocopies of Barringer assay certificates or handwritten entries from the Hycroft laboratory. The pre-Allied Nevada databases comprise a large part of the current resource model data set.

A review and validation of the 2010 assay, collar coordinate, downhole survey, and assay data were performed by a third modeler. SEWC verified that the databases were correct before any information was used in this report. The 2010 data was verified then checked electronically by a third party.

14.1.1 DATA SELECTION

A complete check against laboratory certificates of all electronic assay data used in the resource model was accomplished. All assay, survey, and logging data were checked electronically as described in the following section.

14.1.2 ASSAY SELECTION

The April 2010 resource assay database file for all assayed drill holes was visually checked against assay certificates by SRK. No assay errors were detected. Laboratory source data files for all holes were electronically checked against the 2010 resource assay database file and no errors were found.

The 2010 resource assay database was checked to detect:

- Overlaps and gaps in data.
- Minimum and maximum data values for outliers.
- From footage values were less than to footage values.
- Blank data fields.
- Non-numeric characters in numeric fields.
- Hole naming errors (two typos were found and corrected).
- Maximum assay depths against downhole survey, planned, and logged depths.

14.1.3 GEOLOGICAL DATA CHECKS

Geologic logs were checked for conformance to approved logging codes and format. The data was checked electronically using scripts to compare data against source files, and to detect anomalies. Logs were visually checked against electronic data and by examining core photos where necessary. During 2010, approximately 40 pre-2010 RC holes were logged or re-logged using the new Allied Nevada format. These holes were checked against nearby holes visually using the Vulcan® display software for detection of inconsistent or odd alteration and rock types. Where such inconsistencies were noted, the material was re-examined and corrected, if necessary.

Start and ending log footages were checked for gaps and overlapping values. Holes were checked to ensure that the encoding of RC and Core ("R" or "D") in the hole name was correct.

Total depths were checked against total depths as drilled, and against maximum assay depths. Several hole maximum depths as logged were found to be short by a few feet. These were the result of a missing sample for those intervals not recorded in the geologic logs. The differences were deemed to be minor in nature and would not affect modeling efforts. The list of these was sent to the geologist for correction.

14.1.4 COLLAR SURVEY CHECKS

Collar surveys were range checked for minimum and maximum northing, easting, and elevation. The coordinates were checked against the planned coordinates to detect errors in either set of numbers and for reversal (swapped coordinates). All holes were visually checked in Vulcan® and on topographic photo based maps to confirm that holes were on the correct drill pads and map coordinates. Two holes were found with errors; one was a mismatch between the planned and surveyed coordinates and the other had coordinates from a nearby hole. Both errors were corrected.

14.1.5 DOWNHOLE SURVEY CHECKS

All downhole surveys, Allied Nevada era and historic, were checked electronically for minimum and maximum azimuth, inclination, and depths. Surveys for the 2010 model were checked against the planned azimuth and inclination to detect errors. Surveys were allowed to be taken within 200 ft of expected hole total depth to ensure that the survey was completed before the drill finished. Surveys were projected by the downhole surveyor to the expected total depth.

In some cases, the water temperature was too hot to continue surveying, and the survey projected to the expected total depth. Total depths for downhole surveys, where different than actual depths, were extended or truncated to the actual depths using the projected data.

14.2 ANALYSIS OF SAMPLING BIAS AND CORRECTION OF VISTA DRILLING ASSAYS

While mining the Brimstone deposit, Vista found that it was recovering more gold than was predicted from the resource model. The blast hole samples were also returning higher cyanide soluble gold assays (blast hole samples were not fire assayed) than predicted by the resource model for cyanide soluble gold.

The reconciliation of Brimstone production indicated that the Brimstone model slightly over predicted ore tons (2.2%), but substantially under predicted the grade of the material sent to the leach pad (21%). This reconciliation and the results of the 1999 twin hole comparison indicated that a sampling bias was responsible for the under prediction of the grade of the material mined. MRDI studied this in detail and concluded that the older samples in the database should be corrected to better predict the grade of the material mined from the Brimstone deposit.

MRDI and Vista's work determined that drilling prior to 1999 was clearly biased low in cyanide soluble gold relative to blast holes, and that the source of this bias most likely was loss in fines with the wet drilling method. In addition, MRDI found that cyanide soluble gold values are depressed in samples containing native sulfur (as seen where drill log visually estimated sulfur >5.0%), compared to assays of samples where native sulfur is not observed. This is most likely a consequence of a preg-robbing effect by fine particles of sulfur created in sample preparation.

The resultant correction factors for fire assay and cyanide soluble gold due to sampling biases and the presence of native sulfur are:

- For intervals with native sulfur logged at high (>5%) levels, the cyanide soluble gold assays were discarded and replaced with an estimate derived from cyanide soluble gold to fire gold ratios from nearby intervals (of the same alteration type) without observed native sulfur.

- Intervals with native sulfur logged at low or moderate levels were tagged, and cyanide soluble gold was adjusted with the factors determined by the year of the sampling campaign. Five different adjustments were possible, depending on the ore type and year of assay. These are listed in Table 14.1.
- Cyanide soluble gold: After corrections for sulfur were made, the following adjustments were applied to the assays with gold <0.045 opt:
  - Acid Leach Ore Original assay x 1.40
  - Oxide Ore Original assay x 1.19
- Fire Assay gold: Adjustments were made to assays with gold < 0.08 opt:
  - Acid Leach Ore Original assay x 1.39
  - Oxide Ore Original assay x 1.19

Corrections to cyanide soluble gold assays were validated using blast hole cyanide soluble gold assays for the north half of the Brimstone deposit. No adjustments were made to cyanide soluble silver grades.

Table 14.1 Adjustments to Cyanide Soluble Gold for Presence of Sulfur

Acid Leach	
Native Sulfur Logged Observation, Drill Year	Adjustment (CN is CN-sol Au)
Trace S (S=0)	
Barringer (pre 1991)	no adjustment
1999	no adjustment
Mine Lab, 1992-1998	y = 0.6386*(AuCN) + 0.2944*(Fire)
Minor S (S=1)	
1988 - 1997, AuCN/Fire < 0.4	y = 1.450*(AuCN) + 0.160*(Fire)
1988 - 1997, AuCN/Fire 0.4 to 0.9	y = 0.3143*(AuCN) + 0.6143*(Fire)
1999	no adjustment
Other Oxide (not acid leach)	
Trace S (S=0) or Minor (S=1)	
AuCN/Fire < 0.33	y = 1.387*(AuCN) + 0.2157*(Fire)
AuCN/Fire 0.33 to 0.9	y = 0.2923 *(AuCN) + 0.6788*(Fire)

Table 14.2 Correction Factors Applied to the 1999 Twin Drilling

1999 Drill Hole	Interval	Feet	AuFA/ton	AuCN/ton	Old Drill Hole	Original AuFA/ton	Corrected AuCN/ton
99-1378B	180-410	230	0.015	0.012	88-1378	0.009	0.012
99-1419B	330-565	235	0.003	0.01	89-1419	0.01	0.011
99-1432B	240-460	220	0.013	0.024	89-1432	0.014	0.018
99-1504B	125-600	475	0.018	0.009	90-1504	0.006	0.007
99-1523B	195-380	185	0.027	0.021	90-1523	0.027	0.029
99-1944B	0-250	250	0.016	0.002	92-1944	0.003	0.004
99-1949B	0-410	410	0.004	0.013	92-1944	0.011	0.013
99-1950B	0-405	405	-	0.014	92-1950	0.01	0.011
99-1975B	75-545	470	-	0.022	92-1975	0.018	0.02
99-1976B	175-580	405	-	0.012	92-1976	0.007	0.01
99-2648B	100-305	205	-	0.002	95-2648	0.001	0.001
Totals		**3,490**		**0.013**		**0.01**	**0.012**

14.3 ANALYSIS OF SAMPLING BIAS AND EXPLORATION DRILL ASSAY CORRECTION – ORE

ORE also evaluated the original assays and the corrections applied by MRDI. ORE used slightly different correction factors compared to MRDI, as described by Noble (2005).

Since powers from the regression analysis were generally close to one, a decision was made to assume that the power is one, which causes the power curve to transform to a simple constant that is multiplied times the uncorrected grade. Using a simple constant rather than the power curve, introduces a slight conservative bias for resource estimation, since higher grade assays are corrected less than would be indicated for the power curve, when the power is greater than one.

A correction factor of 1.19 was used for oxide zone assays and 1.32 for acid leach zone assays. The 1.19 factor for oxide zone assays is the same as that developed previously by MRDI. The 1.32 factor for acid leach zone assays is 6% lower than the 1.40 correction used by MRDI. MRDI did not correct cyanide soluble gold assays above 0.045 opt AuCN, however, while all assays were corrected for this study, the overall difference between the MRDI and ORE adjusted grades is <1%.

MRDI used different correction factors for fire assay gold and cyanide soluble gold based on regression analysis of the RC twin data. ORE recommends use of the same factors for fire assay gold and cyanide soluble gold because the amount of twin hole data is too small to establish different bias corrections between the two assays, particularly in the sulfide zone where any difference would be most significant.

It has been shown that high sulfur content is associated with lower than expected cyanide soluble gold assays and that some correction of those assays is justified. Since some of the high sulfur samples have high AuCN:AuFA ratios and some low sulfur samples have low AuCN:AuFA ratios, it is clear that not all high sulfur samples should be corrected and that the amount of correction is not entirely related to sulfur content.

A method of correction for the high sulfur cyanide soluble gold assays was developed based on the assumption that the distribution of the AuCN:AuFA ratio should depend only on the degree of oxidation. Thus, if 50% of the well oxidized samples with no sulfur have AuCN:AuFA ratios >0.75, so should samples that contain sulfur. The correction equations were derived as follows:

- The drill hole data contains codes identifying the quantity of sulfur in the sample based on visual examination of drill cuttings by the geologist. Sulfur categories are:
  - No Sulfur.
  - Trace Sulfur.
  - <5% Sulfur.
  - 5-10% Sulfur.
  - >10% Sulfur.
- Cumulative frequency distributions were prepared for each sulfur category. QQ plots were prepared where the sulfur bearing ratios were plotted on the log scaled X axis and the sulfur free ratios were plotted on the normal scaled Y axis. As expected, these curves imply greater corrections for higher sulfur samples. The cumulative plots were prepared using only those data points with fire assay gold grades (after adjustment for RC bias) >0.14 opt Au to minimize problems calculating ratios when the assay values approach the precision of the assay.
- Logarithmic correction curves were fit to the QQ points in the form:

  $Y = A \ln(X) + B$, where A and B are constants.

- Two curves were used for the 5-10% sulfur, and >10% sulfur categories, because the low ratio end of the corrections were not linear.
- Sulfur corrections on the AuCN assay were then made by looking up the appropriate correction equation for the sulfur content category, calculating the uncorrected AuCN:AuFA ratio, calculating the corrected ratio from the correction curve, then multiplying the corrected ratio times the original AuFA assay.
- A second set of correction curves was developed for partially oxidized materials using the above method.
- The equations developed using the QQ correlation studies were used to correct cyanide soluble gold assays in well oxidized and poorly oxidized samples. Cyanide soluble gold assays were not corrected in sulfides.

14.4 ELECTRONIC DATABASE VALIDATION – SRK

SRK completed a 100% data check of the database in February 2008. The database is certified clean for use with all future grade estimation models at Hycroft. This Technical Report is based on the clean database. The primary purpose of the verification program was to identify and correct data entry errors to the Hycroft electronic analytical database using all available historic assay certificates, drill logs, and surveys. The electronic database provided to SRK by Allied Nevada contained approximately 3,183 drill holes including 186,123 records. SRK was able to locate and check original hard copy assay certificates for 175,002 records (94%). In the process, the drill collar file was supplemented with additional details regarding laboratories and analytical detection limits. The data verification program was carried out from October 2007 through January 2008 and included the following activities:

- Data collection.
- Verification of assays.
- Verification of geologic data.
- Verification of surveys.
- Results and database compilation.

14.4.1 DATA COLLECTION – ASSAY CERTIFICATES AND GEOLOGICAL LOGS

Allied Nevada required that all of the original assay certificate data remain on the Hycroft site. This practice minimized the risk of data loss or damage during transport as there were initially no duplicate hard copy data available.

To satisfy this requirement, SRK sent field technicians to Hycroft where they worked during normal day shift hours to create a duplicate of all of the analytical, geological, and survey data.

The analytical and geological (drill log) data were found in a group of eight filing cabinets located in the bull room of the former engineering office near the mine's main reception and sign-in area. Additional survey data was located in the former survey office adjacent to the bull room. The data was organized as individual files and arranged sequentially by hole number. Approximately 60% of the files contained analytical data sheets only, while 40% contained both assays and drill logs. Once reproduced, the data was boxed and transported to SRK's office for verification and further processing.

14.4.2 DATA COLLECTION – ELECTRONIC DATA

The Hycroft electronic database was provided by Allied Nevada in Microsoft Access format. SRK examined the contents of four historic data sets before determining the most complete set. The database used for verification and development was called "hyc2000.db1.mdb." To this database, the data from the 2005 Canyon Resources drilling program was added.

14.4.3 VERIFICATION OF ASSAYS

Prior to data verification, SRK developed a pre-backfill topographic surface using historic electronic topographic files and photographs. Analytical data was coded using this surface resulting in an "in situ" or "mined out" designation for each interval. All available intervals coded in situ were checked. A total of 10-15% of the mined out intervals were also checked. The Access database was converted to Excel format. Formulae were written to convert analytical results from opt to ppm and ppb for rapid assessment. The database was then subdivided into four equal parts for verification. Each of the parts was addressed by a two person team in which one team member was working with the electronic worksheet and the other was working with the assay certificate. Analytical data sheets contained results for four analysis:

- FAu Fire assay gold.
- FAg Fire assay silver (rarely assayed/reported).
- CNAu Cyanide soluble gold.
- CNAg Cyanide soluble silver.

From 1983 to 1992, some full hole sample sets and other partial hole sets (selected mineralized ranges) were analyzed by Barringer. SRK was unable to locate the assay methodology or QA/QC procedures from Barringer.

From 1991 to 1999, all exploration samples were analyzed on site by the Hycroft laboratory. No QA/QC records are available for this period of testing. On the occasion when Barringer check assays were available, in addition to Hycroft results, the Barringer check assay results were considered most reliable (as Barringer was an accredited facility). From 1999 to 2006, only minor analytical work was done; all by off-site laboratories such as American Assay Laboratories and Chemex.

Rarely, multi-element data was available from Barringer and recent laboratories. While this data was not entered as individual fields and records, the presence of this data was recorded in the updated drill collar file for the project.

14.4.4 VERIFICATION OF GEOLOGICAL DATA

Geologic data was checked and validated by MRDI in 2000. As part of the MRDI program, 1,740 drill logs were selected for checking against the electronic files. A total of 0.8% error was identified, suggesting the data was accurate. SRK followed up on the previous work, selecting 150 drill logs, at random, for confirmatory checking. Localized errors were observed in some of the six fields of geological data, but no systematic errors were identified, i.e. large ranges of intervals with mismatched data. SRK concluded that there is sufficiently low incidence of entry error for use in resource calculations.

14.4.5 VERIFICATION OF SURVEYS

Several survey record books were located in the files of the Hycroft engineering office. The books contained collar coordinates of drill holes. Approximately 100 holes listed in the survey record books were checked in the electronic database. No errors were found. All of the drill data was imported into a 3-D modeling program. The collar elevations were checked against the topographic surface appropriate for the time in which they were drilled. All of the holes examined correlated in elevation with topography.

Downhole surveys are uncommon in the database. There were no historic records to which the electronic data could be compared. An examination of the drill hole traces in 3-D, using the modeling program, indicated reasonable projections for the surveyed holes.

14.4.6 RESULTS AND DATABASE COMPILATION

Following rigorous, record-by-record checks of the analytical database, the temporary electronic worksheets were re-assembled into a single database which serves as the “assay” file for the project. The revised database contains original and updated fields for the four main analyses as follows:

ORIGINAL	NEW	DESCRIPTION
FAu	NFAu	Fire assay gold
FAg	NFAg	Fire assay silver (rarely assayed/reported)
CNAu	NCNAu	Cyanide soluble gold
CNAg	NCNAg	Cyanide soluble silver

The population of assay intervals was 186,123. SRK checked 175,002 intervals (94%). The total errors were 13%, of which 7% were related to missing data or data below detection limit. A total of 6% of the database contained substantive numerical errors, which were replaced by new values from the assay certificates. Compared to the original values, the new values resulted in positive adjustments at a ratio of 2:1. The most common errors were single shifts, where all records of an assay certificate were shifted by one interval (up or down). Next, there were many examples of missing grades in the original electronic database for which certificate values existed. The certificate values were entered into the appropriate fields. Finally, there were occasional decimal errors made during input. Those were corrected.

Drill hole coordinates were compiled into a new "collar file" for the database. In addition to collar coordinate information, the collar file was also used to track the laboratory used for each drill hole, as well as the detection limits for the major elements tested.

In the assay database, records with no sample, no data, or missing data were coded as -9. For all intervals whose value was below the detection limit for that element, the intervals were coded as -8. Since the detection limits for each element are recorded in the collar file, the intervals below detection can be readily re-coded with real numbers if desired.

It is important to note that there are many grade values in the database listed as 0.00. The Hycroft laboratory, which generated most of the historic results, did not report values below detection. Instead, they reported 0.00 on the certificates. For these records, SRK did not substitute -8.

14.5 ADJUSTMENT OF ASSAY VALUES FOR THE NEW RESOURCE MODEL

A great level of detail and correction went into the 561 drill holes at Brimstone that were used for the ORE model. For this report, Allied Nevada logged an additional 1,205 drill holes for geology. This provided enough information to apply the same MRDI and ORE derived factors to the recently logged drilling data. The 561 originally factored drill holes were left unchanged. SEWC applied factors only to fire assays that were identified as either acid leach or oxide.

- Acid leach fire assays were factored by the equation: AuFA * 1.32.
- Oxide fire assays were factored by the equation: AuFA * 1.19.

No other factors were applied to the raw data.

15 ADJACENT PROPERTIES

There are no properties adjacent to the Hycroft project with recent Mineral Resource or Mineral Reserve estimates.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

Hycroft mine acid leach ores are presently being mined and placed on the pad and being processed using ROM and crushing for the heap leach. This material will continue through the end of 2010. More competent siliceous material is being mined mid-year 2010 and will also be processed through the crusher and placed on the leach pad. The deposits in the mineral inventory for crushing include Cut-5, Bay Area, Boneyard, Central Fault, and Vortex.

Ore grade materials are transported to the leach pad and cross ripped to enhance permeability. Solution drip line is placed and the material leached with a buffered cyanide solution. The gold solution from the pad is processed through either the CIC circuit or the Merrill-Crowe plant. Past production data from the leach pads give the best possible indication of future processing recoveries. The feasibility of mining is based on the past performance of the various leach pads. The ore types that will be placed are the same ore types that were placed on the pads until 1998. Pad 4 was continuously leached after mining ceased and the recovery for that pad stands at about 79.5% of the total cyanide soluble gold. The estimated recovery for the various ROM ore materials is projected to average about 56.6% of the fire assay gold based on reviews of historic production and test work.

16.1 PROCESSING FACILITIES

16.1.1 BRIMSTONE LEACH PAD

The existing heap leach pad is permitted for an expansion of 9.1 million ft^2. In addition to the existing 2 million ft^2 of pad constructed in Phase 1 prior to 2008, Allied has completed 4.6 million ft^2 in Phase 2 through the first half of 2010. An additional 2.5 million ft^2 of phase 2 pad construction is planned for the second half of 2010 and this will complete the permitted expansion.

16.1.2 BRIMSTONE PLANT

The Brimstone plant comprises five solution ponds, a CIC circuit, and a Merrill-Crowe zinc precipitation plant and refinery. There are three primary ponds; the pregnant, lean, and barren ponds. Each primary pond has a volume capacity of 2.65 million gallons. The fourth pond is a DE settling pond, also with a volume capacity of 2.65 million gallons. The last pond is the old Lewis pregnant pond which has a capacity of 4.0 million gallons, which is used as the emergency storage pond.

Processing of the buffered pregnant solution and precious metal recovery is through a 1,400 gpm CIC circuit (two CIC trains) and a 3,500 gpm Merrill-Crowe plant.

The CIC circuit (two separate trains) consists of five tanks in each series containing activated carbon. The pregnant solution flows upward through each column in series, maintaining the carbon particles in suspension. The solution flows from column 1 to 2, 3, 4, and 5 and then to the barren pond. The activated carbon is moved counter current to the solution from column 5 to 4, 3, 2, and then 1. The carbon in column 1 contains the highest gold and silver loading, which is removed from the column to be stripped of the metals and eventually placed back into the circuit. The carbon from columns 2 through 5 are individually moved forward with new or stripped carbon placed in column 5. The gold loaded carbon from column 1 is processed into doré by an off-site facility and sent to the refinery.

The Merrill-Crowe plant pregnant solution is fortified with cyanide and clarified using sparkler filters. The clarified solution is de-aerated using vacuum pumps in a packed vacuum tower. Zinc dust is added to the clarified/de-aerated solution. Gold and silver precipitates are captured with three 48” recessed plate filters. The collected precipitate is transported to the refinery, retorted to remove mercury, and fire refined. Barren solution is discharged to the barren pond and then re-circulated back to the Brimstone pad.

16.1.3 RECOVERY

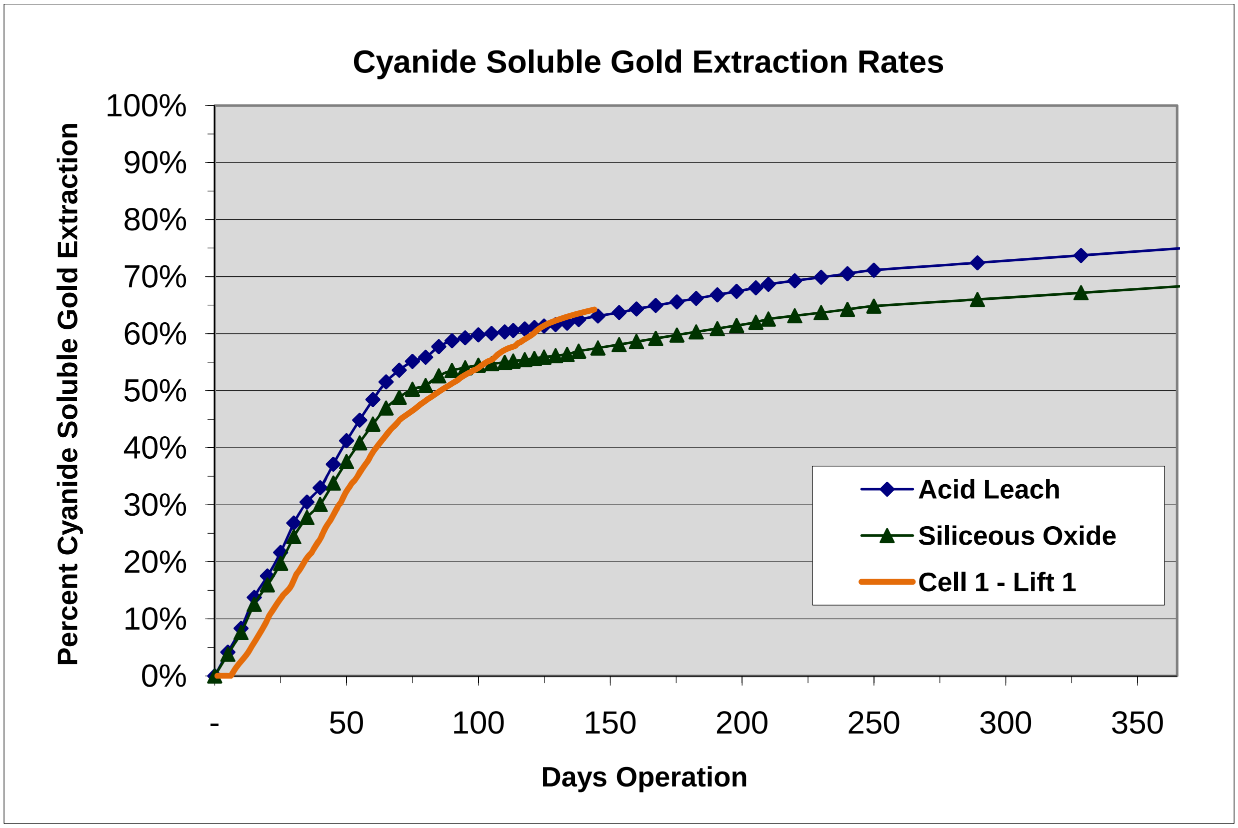
Actual final gold recovery from Pad 4 for all previous operations was 79.5% of the cyanide soluble content (pad 4, historic results). Considering all the information available, the projected gold recovery of 56.6% of total gold content represents a conservative estimate for the remaining ores in the Brimstone pit. Historic production figures for Brimstone pads 4 and 5 are shown in Table 16.1 below.

Table 16.1 Production Pad Loading and Recoveries

Pad	Tons of Ore	Gold Loaded oz	CN Sol Grade opt Au	Recovery Gold oz	Actual % Recovery
4	11,129,940	159,206	0.014	126,622	79.5
5	4,334,017	61,991	0.014	49,348	79.6

Figure 16.1 shows the actual historic recoveries of gold from the Brimstone pad compared with the gold recoveries from material placed on Phase II, Cell 1, first lift in 2009. Gold recovery curves are similar. Cell 1, first lift was shut down after about 140 days to allow the second lift to be placed. The gold extraction curve of Cell 1, first lift is expected to follow the past recovery curves. Silver assays from the Phase II, Cell 1, first lift material are 0.264 opt total by FA and 0.200 opt by CNAA. Silver recoveries are calculated to be 14.8% based on the CNAA assay and 11.8% based on the FA.

Figure 16.1 Hycroft Leach Pad Recovery (Cyanide Soluble) Comparisons



16.2 METALLURGICAL TEST WORK

Several metallurgical studies have been undertaken on the Hycroft ore. These studies are briefly discussed in this section. In 1994, a metallurgical program was initiated at the Hycroft mine to evaluate the gold recovery that could be expected from ROM leaching of the Brimstone ore body. It was apparent, at the start of the Brimstone evaluation, that two basic ore types existed which were classified at the time as “silicified breccias” and “acid leach.” The acid leach material, which generally forms the upper part of the Brimstone deposit, is fine and friable, whereas the silicified breccias are significantly more competent. During the initial testing of the Brimstone ores, relatively good bulk samples of acid leach material were available for column and heap leaching tests, while a limited quantity of silicified breccias core samples were available for testing. As a result, good confidence in the recoveries from acid leach material was obtained through test work, while additional testing needed to be undertaken to improve the confidence in the expected recovery from the silicified breccias material.

When mining started at Brimstone at the north end of the deposit, ore was trucked to pad 4 which was constructed solely for Brimstone ore, and to pad 5, which was Brimstone ore placed on top of the old Crofoot pad 1. As a result of this placement of ore, recovery from pad 5 could be biased by some residual leaching from pad 1 below it. Pad 4, however, was exclusively used for Brimstone ore and, therefore, gold production from this pad accurately reflects actual gold recovery achieved from

Brimstone ore. Ore placed on pad 4 was predominately acid leach material but did include approximately 27% of siliceous oxide (previously called silicified breccias) ore.

Due to sustained low gold prices, mining in the Brimstone pit was halted in December 1998 and no further metallurgical test work was completed. It was apparent, however, that significantly more gold had been placed on the Brimstone heap than was reported in the mine model, so a detailed study of the Brimstone ore body mined to date and future reserves was undertaken. During the course of this study, all the existing drill hole data was re-logged, and together with pit mapping and blast hole data, the geology of the Brimstone deposit was reinterpreted resulting in a much better understanding of the relationship between the ore material types and metallurgical response. While there remains two predominant ore types, they are now referred to as "acid leach" and "siliceous oxide," instead of "acid leach" and "silicified breccias," and there is only one potential subset that has any significance – clay bearing oxide. In light of this additional information, the samples used for all previous metallurgical work were re-reviewed to see which ore type was represented. In addition, areas in the pit where specific ore types are now exposed were identified and new samples were collected for additional test work.

16.2.1 HISTORIC TEST WORK

A significant amount of test work was completed in 1994 prior to making the decision to proceed with the development of the Brimstone deposit. This work included bottle roll tests, barrel tests, column tests, and two test heaps. The majority of the work focused on acid leach material, which was more readily available, and led to the conclusion that at least 75% recovery of cyanide soluble gold was achievable.

Four column/barrel tests were run for a 3" rock size on material designated "transition oxide" material and "silicified oxide" material. The composition of ore samples, which were used for these tests, was reviewed to determine whether or not the columns can be considered representative under the new definition of oxide ore. The conclusion is that the samples were representative. Gold recoveries achieved from these column/barrel tests are as shown in Table 16.2 below.

Table 16.2 Column/Barrel Test Results on "Transition Oxide" and "Silicified Oxide" Ore

Test Number	CN-Soluble Gold Recovery (%)	Fire Assay Gold Recovery (%)
94-13A	72.7	61.9
94-13B	77.6	69.7
94-13C	65.3	52.2
94-13D	74.7	65.9

The first recovery figure is based on cyanide soluble gold assays while the second figure is based on fire assays. The average cyanide soluble gold recovery for these tests was 72.6%, but if the lowest recovery test is rejected, the average gold recovery is 75%.

The results of the tests on acid leach and oxide ores provided the basis for proceeding to production. The actual results of production for the ROM pads demonstrated significantly higher recoveries over time.

Various ore bodies at Hycroft were mined, crushed to a nominal 3/8", and placed on the Crofoot leach pad. These ore bodies included the Cut-5, Bay Area, Boneyard, Central Fault, and Brimstone materials. A monthly composite of the crushed materials was column leached to predict gold recoveries. An evaluation of column test work from the monthly crushed composites, shown in Table 16.3, indicate average CNAA gold recoveries ranging from 71.7% to 81.5%.

Table 16.3 Yearly Average Crusher Column Results for Oxide Ore

Year Average	Au CNAA opt	Ag CNAA opt	Au CNAA Rec. %	Ag CNAA Rec. %
1989	0.018	ND	71.7	ND
1990	0.019	0.069	73.6	35.1
1991	0.020	0.141	78.7	34.9
1992	ND	ND	ND	ND
1993	0.018	0.133	81.5	33.8
1994	0.017	0.044	75.4	26.3
1995	0.014	0.053	80.7	22.8
1996	0.015	0.096	78.5	28.4

ND – No Data Available

16.2.2 TEST WORK YEAR 2000

A test program was initiated to better understand the metallurgical response of ore types that would be encountered in future mining. The tests included column testing of core samples and drum testing of bulk samples collected from the pit. The results are tabulated in Table 16.4.

Table 16.4 Column Leach Results for Oxide Ore

Sample	Material	Current Gold Extraction		90 Day Projected Gold Extraction (%)	R^2
		% CN-Sol Au	% FA Au		
4636	Clay Bearing Oxide	83.2	76.9	90.5	0.99
4434	Clay Bearing Oxide	77.5	69.9	86.7	0.99
4400	Clay Bearing Oxide	79.6	72.4	84.0	0.99
Core 1	Silicified Oxide	61.7	50.5	70.3	0.99
Core 2	Silicified Oxide	64.3	55.7	70.4	0.99
Core 3	Silicified Oxide	70.4	60.4	77.0	0.99

The results of column tests core 1, 2 and 3, which employed samples taken from intact core not representative of ROM material, showed similar results to previous tests. The drum tests were more representative, based on test work carried out on bulk samples taken from the blasted ore in the pit, with a more appropriate size distribution. The 90 day projected recoveries for three drum tests varied from 84.0 to 90.5%. The drum samples were, however, a little higher grade than the grade of the future reserves. The lower or average grade ores will probably not achieve quite as high a recovery. However, in a production situation, the placed ore is leached for much longer than 90 days, which would tend to recover more gold. An important point to note is that the drum test results and subsequent tailings analysis indicates that future ores will yield similar metallurgical performance to previously mined ore.

Management is currently reviewing the potential benefit of milling all or a portion of the oxide material at Hycroft to improve recoveries of both gold and silver.

16.3 PREVIOUSLY MINED ORE COMPARED TO REMAINING MINERAL RESERVES

16.3.1 BRIMSTONE

An indication of future metallurgical performance is to compare the cyanide soluble data of samples representative of the ore obtained during previous mining of the Brimstone ore with samples representative of the remaining Brimstone reserves. A detailed comparison of the cyanide soluble data as compared to the fire assay results for samples of the south Brimstone drill intercepts and north Brimstone drill intercepts was completed. South Brimstone is typical of previously mined Brimstone ores and north Brimstone is representative of future Brimstone reserves. The results of these comparisons are in Tables 16.5 and 16.6.

Table 16.5 South Brimstone Drill Intercepts

Ore Type	Feet Included	% of Total Feet	CN Sol Au (opt)	% CN Sol Recovery
Siliceous	13,868	45.70%	0.015	73.50%
Acid Leach	12,674	41.80%	0.019	76.70%
Clay bearing	3,165	10.40%	0.024	80.20%
Other	613	2%	0.014	85.90%
Total Average			**0.018**	**76.00%**

Table 16.6 North Brimstone Drill Intercepts

Ore Type	Feet Included	% of Total Feet	CN Sol Au (opt)	% CN Sol Recovery
Siliceous	11,352	35.70%	0.014	75.30%
Acid Leach	18,722	58.80%	0.016	75.80%
Clay bearing	1,486	4.70%	0.014	79.50%
Other	259	2.00%	0.009	59.50%
Total Average			**0.015**	**75.70%**

These results indicate that there is no significant difference between the overall percentage of cyanide gold recovery for the north and south portions of the Brimstone pit. The average percentage of cyanide soluble gold in both sample sets, within experimental limits of sampling, is identical; 76.0 vs. 75.7%. The conclusion to be drawn from the cyanide soluble comparison, the production data, and completed test work, is that all described ore types, within the error of quantifiable results, are metallurgically identical in a ROM situation.

Bottle and column test work on Brimstone deposit core samples, shown in Tables 16.7 and 16.8, were conducted to indicate the effect of particle size on leach recovery. Results indicate a particle size effect similar to historical information. Column cyanidation gold leach recovery results were 57% from -3/4” and 71% from -3/8” materials.

Table 16.7 Brimstone Bottle Cyanidation Leach Results

Deposit	Interval	SB	10M	3/4"
		Au %	Au %	Au %
Brimstone	3694 137.5-155.5	81.5	73.1	41.9
Brimstone	3695 105.5-113.2	100	100	33.3

SB = Shatterbox Material
10M = Mesh Material

Table 16.8 Brimstone Column Cyanidation Leach Results

Deposit	Fire Assay Recovery 80% Passing 0.75”		Fire Assay Recovery 80% Passing 0.375”	
	Au %	Ag %	Au %	Ag %
Brimstone	57.1	23.3	71.4	30.7

Column test works conducted on siliceous Brimstone material, shown in Table 16.9, indicate total gold recoveries of 71 to 75% from ROM, -2.0”, and -0.5” materials. Silver recoveries, based on fire assay, range from 17.7% with ROM material to 27.7% from material crushed to passing 0.5”.

Table 16.9 Siliceous Brimstone Column Test Results

Ore type	Crush Size	% Fire Assay Ag Rec.	% Fire Assay Au Rec.
Siliceous	ROM	17.7	70.6
Siliceous	-2.0”	21.2	75.0
Siliceous	- 0.5”	27.7	73.3

As shown in Table 16.10, column cyanidation test work on Brimstone acid leached materials containing low and high sulfur contents (3.9% vs. 7.4%), indicate gold recovery decreased from 89.1% with the lower sulfur to 80.0% with the higher sulfur contents.

Table 16.10 Acid Leach Brimstone Column Test Results

Ore type	Crush Size	% Fire Assay Ag Rec.	% Fire Assay Au Rec.
Low S Content	As Rec	8.3	89.1
High S Content Average	As Rec	6.7	80.0

16.3.2 CUT-5

Bottle and column test work on Cut-5 deposit core samples, shown in Tables 16.11 and 16.12, show the effect of particle size on leach recovery. This data indicates gold recovery is particle size dependent. Bottle test recoveries ranged from 27 to 71% from -10 mesh material and from 9 to 50% from -3/4” material. Column test gold recoveries were 37% and 44% from -3/4” and -3/8” materials, respectively.

Table 16.11 Cut-5 Bottle Cyanidation Leach Results

Deposit	Interval	SB	10M	¾”
		Au %	Au %	Au %
Cut-5	3686 281-311.5	85.7	71.4	50.0
Cut-5	3686 311.5-340	80.0	60.0	37.5
Cut-5	3691 141.5-158	50.0	50.0	28.6
Cut-5	3691 158-174.5	60.0	27.3	9.1
Cut-5	3693 115-130,1725-180	87.5	62.5	42.9
Cut-5	3693 52-60,70-80,90-95	66.7	41.7	23.1
Cut-5	3693 60-70,85-90,95-105	66.7	45.5	21.4

SB = Shatterbox Material
10M = 10 Mesh Material

Table 16.12 Cut-5 Column Cyanidation Leach Results

Deposit	Fire Assay Recovery 80% Passing 0.75”		Fire Assay Recovery 80% Passing 0.375”	
	Au %	Ag %	Au %	Ag %
Cut-5	37.5	9.1	44.4	22.7

16.3.3 BONEYARD

Bottle and column test work on Boneyard deposit core samples are shown in Tables 16.13 and 16.14. This data indicates gold recoveries were particle size dependent with bottle test gold recoveries ranging from 25 to 83% from -3/4” material, column gold recoveries of 50 to 78% from -3/8” material, and column gold recovering of 43 to 80% from ¾” material. Historic data also indicates the Boneyard material was generally crushed to a nominal 3/8” particle size prior to placement on the leach pad.

Table 16.13 Boneyard Bottle Cyanidation Leach Results

Deposit	Interval	SB	10M	¾”
		Au %	Au %	Au %
Boneyard	3710 0-20	87.5	75.0	55.6
Boneyard	3710 20-40	92.3	76.9	80.0
Boneyard	3710 40-60	100	85.7	83.3
Boneyard	3710 60-79.5	88.9	72.2	64.7
Boneyard	3710 79.5-112	86.4	56.5	41.2
Boneyard	3710 112-134.5	87.5	50.0	25.0

SB = Shatterbox Material
10M = 10 Mesh Material

Table 16.14 Boneyard Column Cyanidation Leach Results

Deposit	Fire Assay Recovery 80% Passing 0.75"		Fire Assay Recovery 80% Passing 0.375"	
	Au %	Ag %	Au %	Ag %
Boneyard #1	80.0	0	77.8	0
Boneyard #2	42.9	6.3	50.0	11.8

16.3.4 BAY AREA

Bottle and column test work on Bay Area deposit core samples are shown in Tables 16.15 and 16.16. This data confirms historic information indicating gold recoveries were very particle size dependent. Bottle cyanidation results (Table 16.15) indicate gold recoveries ranged from 17 to 88% from -3/4" materials and from 12 to 67% from -2" materials. Column cyanidation test work resulted in gold recoveries of 58 to 77% from -3/8" materials. Historic data indicates the Bay Area materials were generally crushed to a nominal 3/8" particle size prior to placing on the leach pad.

Table 16.15 Bay Area Bottle Cyanidation Results

Deposit	Interval	SB	10M	3/4"	2"
		Au %	Au %	Au %	Au %
Bay Area	3698 65-71-81-85-98-101	88.5	59.3	36.0	31.0
Bay Area	3698 71-81.5	100	70.0	40.7	30.4
Bay Area	3698 87-98	100	100	50.0	43.8
Bay Area	3699 109.5-127	87.5	87.5	87.5	70.0
Bay Area	3699 127-144.5	100	83.3	87.5	66.7
Bay Area	3699 144.5-163	84.6	90.0	60.0	62.5
Bay Area	3699 163-181	96.7	74.2	37.9	22.2
Bay Area	3699 194-215	87.5	66.0	43.1	38.6
Bay Area	3699 215-234	56.5	39.1	19.0	12.0
Bay Area	3699 234-247	76.9	63.6	17.4	27.3
Bay Area	3700 150-170	88.9	58.3	71.4	66.7
Bay Area	3700 170-190	81.8	66.7	50.0	42.9
Bay Area	3703 63-79.5	100.9	77.5	47.1	30.8
Bay Area	3703 86-98	86.7	86.7	69.0	52.6
Bay Area	3705 61-100	82.0	68.3	40.7	27.3
Bay Area	3706 109.5-125.5	62.5	37.5	28.6	21.4
Bay Area	3706 125.5-140.5	60.7	44.4	24.1	17.1
Bay Area	3706 70.5-109.5	87.3	63.2	43.4	29.1

SB = Shatterbox Material
10M = 10 Mesh Material

Table 16.16 Bay Area Column Test Results

Deposit	Fire Assay Recovery 80% Passing 0.75"		Fire Assay Recovery 80% Passing 0.375"	
	Au %	Ag %	Au %	Ag %
Bay Area SA	51.1	12.5	57.5	33.3
Bay Area AS	72.2	0	76.5	0

16.3.5 SUMMARY

The gold and silver recoveries based on fire assays, shown in Table 16.17, are estimated using historical and present metallurgical test work. Recovery estimates for Brimstone, Cut-5, Boneyard, and Bay Area were estimated based on historical and present metallurgical data, representative of each ore deposit, using a recovery versus log particle crush size plot. Limited data was available from the Camel and Vortex deposits; therefore, gold recoveries from Cut-5 and Brimstone deposits were used as the nearest ore deposit data, respectively.

Table 16.17 Hycroft Ore Body Recovery Estimate Based on Fire Assay

	3/8" Rec, %		3/4" Rec, %		ROM Rec, %	
	Au	Ag	Au	Ag	Au	Ag
Cut-5	59	20	48	15	42	11
Central Fault	61	9	57	8	52	6
Camel Hill	59	11	48	11	42	11
Boneyard	64	8	50	7	45	5
Bay Area	62	16	57	13	50	11
Brimstone	73	28	68	19	57	12
Vortex	73	28	68	19	57	12

Assuming 10% of total ore mined is crushed to 3/8", it is expected that the overall average recovery for gold and silver would be approximately 60% and 14%, respectively.

16.4 METALLURGICAL TESTING OF UNOXIDIZED MATERIAL

In September 2008, Allied Nevada sent eight individual composites that were combined to form a master composite, to represent unoxidized material to SGS to investigate recovery methods. SEWC believes this is the first step required to move the deep sulfides into the indicated category. The purpose of the program was to develop a preliminary process flow sheet. The program incorporated ore characterization tests as well as the evaluation of a number of metallurgical processing options including gravity separation, flotation, and cyanidation. Analysis of a cleaner flotation concentration from the master composite, shown in Table 16.18, indicates the material contained mainly pyrite, with moderate amounts of quartz.

Table 16.18 Composite Cleaner Flotation Concentrate Mineralogy

Mineral Constituent	Mineral Mass %
Pyrite/Marcasite	72
Chalcopyrite	0.13
Arsenopyrite	0.26
Other Sulfides	0.07
Quartz	16.6
Feldspar	3.15
Clays	2.01
Micas	4.15
Other Silicates	0.04
Ti Oxides	0.19
Other Oxides	0.04
Native Sulfur	0.25
Sulfates	1.04
Carbonates	0.01
Ag Mineral	0.01
Other	0.05
Total	**100%**

The master composite cleaner flotation concentrate, shown above, contains a high pyrite/marcasite mass which would produce an exothermic reaction in both a roaster and autoclave. This reaction would reduce fuel requirements and overall process costs.

Additional test work conducted in 2009 included flotation tests on non-representative deposit samples from Cut-5, Bay Area, Boneyard, and Vortex for variability analysis to confirm the master composite results.

The recovery of the mill sulfide feed is a combination of both flotation and cyanidation of the tails as follows:

	Au	Ag
Flotation recovery	79.7%	71.9%
Cyanidation of the tails	8.3%	14.1%
Total Recovery	88%	86%

Based on recent metallurgical test work and weighted via domain, the recovery for the oxide and oxide/sulfide material that would go to the mill is:

Au	Ag
84%	65.6%

And the overall recovery weighted by tonnage for gold and silver for material delivered to the mill is:

87.2%	81.7%

16.4.1 MASTER COMPOSITE

The master composite, representative of Hycroft ore, was compiled from eight individual composites (Zone A through H) estimated to represent the known deposit. Test work included head analysis and metallurgical test work on the individual composites and the master composite.

Individual and master composite head analysis indicates:

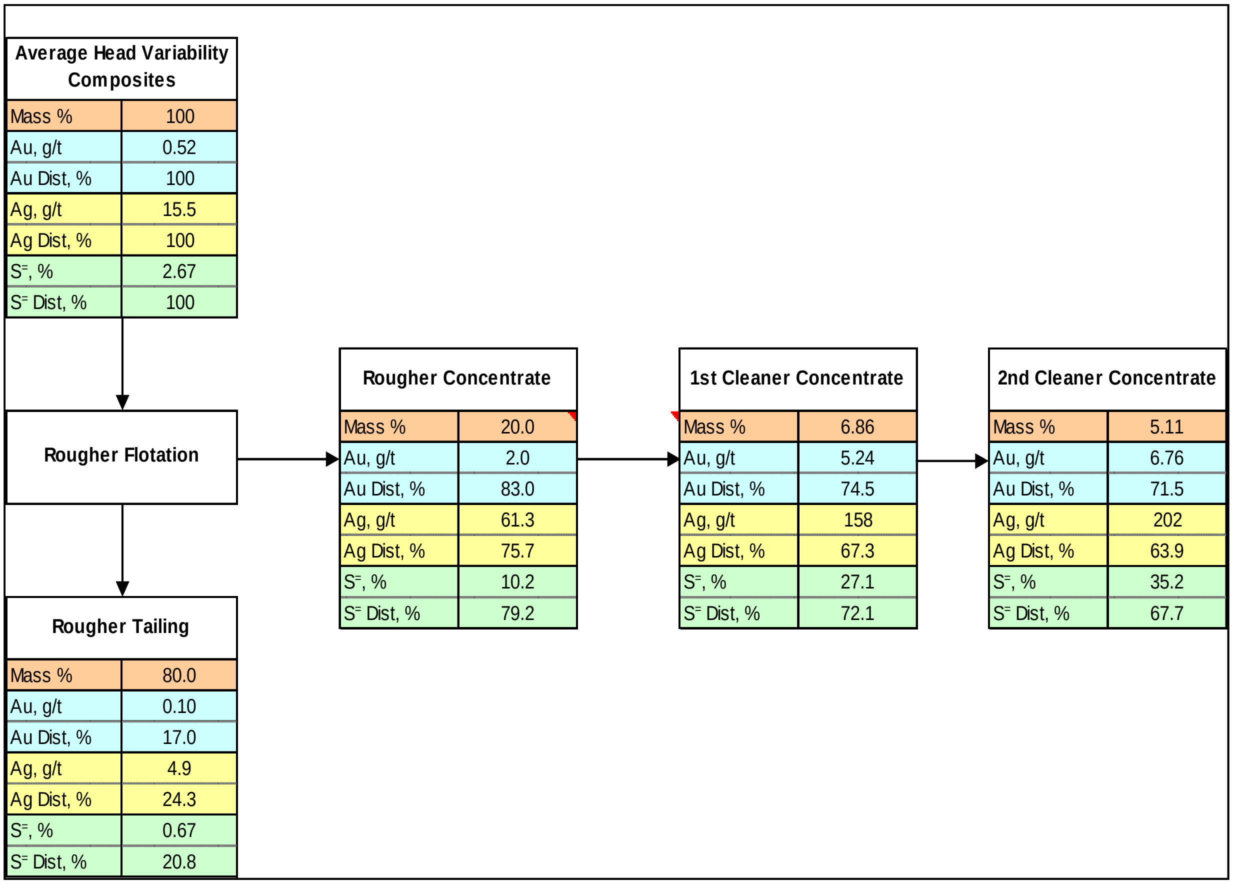
- Gold and silver grades were slightly lower than anticipated at 0.015 opt Au and 0.467 opt Ag. The average gold and silver values anticipated, based on information provided by Allied Nevada, were 0.018 opt Au and 0.505 opt Ag.
- At 15.6 Kwh/ton, the bond ball mill work index is considered to be moderately hard in terms of grindability in comparison to the SGS grindability database.
- A scoping level mineralogical evaluation revealed that pyrite was the principle sulfide mineral observed at an estimated 1 to 5% in all eight composites. Marcasite, which is essentially pyrite with a chemical composition slightly weighted toward Sulfur (Sulfur content in marcasite is 53.5% and only 53.4% in pyrite, both are FeS_2), was present in the range of 1 to 5% in zone composites A, B and C and <1% in the others. Pyrrhotite occurred at <1% in zone composites E, F and G only. Stibnite, at <1% was noted only in zone composite B. Graphite was identified in zone A. Note that the mineralogy study was intended for very basic scoping level reconnaissance information only. A more in depth study may reveal other significant parameters not identified here.
- Gravity separation test work indicated that there is very limited potential for significant gold/silver recovery through gravity methods. Gold and silver content appears to be mainly in intimate association with sulfide species, which, in turn, appear to be fairly fine and not amenable to efficient recovery by gravity methods.
- Flotation through this program has developed to the point where approximately 80 to 85% gold, silver, and sulfide rougher recoveries are attainable in a relatively simple flow sheet. Further flow sheet development is clearly required to attain cleaner (higher grade) rougher concentrates and to improve recoveries. Allied Nevada believes that there are still a significant number of options available for testing in the rougher circuit, including:
  - Optimizing pulp viscosity and density conditions.
  - Evaluating the potential of desliming prior to rougher flotation.
  - Investigating coarser primary grinding (the test work in this program was somewhat inconclusive with regard to establishing the upper grind limit).

    - Conducting a more in depth evaluation of activators and activator combinations in the rougher circuit.
- Flotation cleaning efficiencies were reasonably high and appear to present no insurmountable challenges.
- Whole ore cyanidation yielded very low extractions for both gold and silver. At a primary grind size of 65 µm (P_{80}), gold extraction was 15% while silver was 45%. Based on the available metallurgical evidence, it appears that the associations of gold and silver differ somewhat.

Cyanidation tests completed on cleaner flotation concentrate also yielded poor unit (and overall) extraction values. At a feed size of approximately 20 µm (P_{80}), gold and silver extractions were 13.8% and 55.9% respectively. Considering flotation recovery, those values correspond to 8.5% (gold) and 36% (silver) overall recovery. Ultra fine grinding to P_{80} = 4 µm improved extraction somewhat but also resulted in rather disappointing overall recovery values of 14% and 54% for gold and silver respectively.

A single aqua regia acid leach test completed on the cleaner flotation concentrate resulted in approximately 99% sulfide oxidation and yielded excellent similar degrees of gold and silver extraction. This result indicated that the refractory pre-treatment oxidation process will be necessary to achieve adequate gold and silver recovery from the Hycroft sulfide ore.

Figure 16.2 Master Composite Flotation Results



16.5 VARIABILITY TESTING

A total of 27 composites from the Cut-5, Boneyard, Bay Area, and Vortex deposits were tested at SGS between autumn 2009 and spring 2010. The first study included seven composites from the Vortex deposit. The second study included 20 additional composites from the four deposits.

Testing of the initial seven samples confirmed the results from the previous test work. Head assays ranged from 0.014 to 0.037 opt Au, 0.004 to 1.668 opt Ag and from 2.06 to 4.26% Sulfide. Flotation test results indicate recoveries of gold and silver averaging approximately 88% and 86%, respectively. Sulfide content of the cleaner concentrate was about 25%. Results from this data verify the master composite results using the identical flotation test procedures.

The additional 20 variability samples do not fully represent the various deposits, but were used for comparison to confirm the original master composite results. Head assays for sulfide mineralization ranged from 0.008 to 0.048 opt Au, from 0.050 to 3.333 opt Ag and from 1.0-8.0% sulfide. These samples were geologically identified and ranged from highly siliceous to highly argillaceous.

Rougher flotation gold recovery averaged 79.7%, ranging from 52% to approximately 96%. Silver recovery averaged 71.7% and ranged from 22% to approximately 97%. Average rougher sulfide recovery was 82.1%, ranging from 52 to 97%.

Cleaning in two stages resulted in an average gold recovery of 52% and ranged from 6 to 84%. Second cleaner concentrate silver recovery averaged 47.9% while ranging from approximately 4 to 92%.

The variability in metallurgical response that was evident in the original master composite components test work was also observed in the work completed on the 27 variability samples. It should be noted that, as these were essentially intended as variability tests, no metallurgical optimization test work was completed prior to variability testing. There is little doubt that the apparently negative metallurgical outcomes (recovery) observed in the test work completed on several of these composites, would be easily improved with minimal metallurgical effort.

It is important to recognize that the flotation flow sheet applied in this test work had been developed for one specific composite sample (master composite) and, due to mineralogical differences among the many individual samples evaluated, metallurgical response naturally varied. Further testing, for the purpose of developing a robust and industrially sound flotation flow sheet, should focus on first identifying the ore components that are specifically responsible for the variability observed (the argillaceous nature of many of the ores tested is likely to be significant). Having identified those, the ores should be grouped into similar sets, in terms of mineralogy and general metallurgical response or expectation. The basic flow sheet, so far developed, is very likely to be the core flow sheet that will apply across the board for the Hycroft sulfide ores. Identifying the required modifications in specific parameters (pH, activator, depressant/dispersant, collector, etc.) for each individual ore type is very likely the key to maximizing flotation response for all ore types. The argillaceous fraction of the deposits is estimated to be low.

The tailing components that represent "end of pipe" material in an industrial plant, specifically the rougher and first cleaner scavenger tailings, were proportionally recombined and subjected to basic cyanide leach testing. As the flotation results, in some cases, have not been optimized at this point, the proportion of values (Au/Ag) reporting to the tailing stream (cyanide leach feed) was higher than ought to be anticipated in practice in those cases.

On average, cyanidation of the combined tailings contributed an additional 8.3% gold and 14.1% silver to the overall circuit gold and silver recoveries. The range in overall tailings gold recovery contribution by cyanidation was from 1% to approximately 32%. The range in silver recovery contribution was from 7.5% to approximately 28%.

It is anticipated that these overall recovery contributions would be limited somewhat as the flotation conditions were optimized (i.e., much of the values leached in the cyanidation circuit will report to the cleaner flotation concentrate in an optimized flotation flow sheet).

16.6 ONGOING METALLURGICAL TEST WORK

Additional testing in underway on sulfide, oxide and mixed oxide/sulfide materials to evaluate a grinding circuit and response to flotation for production of a concentrate. Additional test work will evaluate recoveries obtainable by cyanidation of the flotation tailings. Current metallurgical test work includes:

- 12 column tests in progress.
- Six composites for column tests sent to laboratory.
- 27 composites being evaluated for direct cyanidation.

- Two bulk samples, each weighing approximately 3 tons sent to laboratories for flotation pilot plant studies.
- 19 composites being compiled for flotation and cyanidation of tailings
- Crofoot heap leach pad drilled and sampled for evaluation

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The resources stated for the Hycroft mine in this report conform to the definitions adopted by the CIM, December 23, 2005, and meet the criteria of Measured Mineral Resources, Indicated Mineral Resources and Inferred Mineral Resources.

The Hycroft Mineral Resources and Mineral Reserves are not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, political or other relevant issues. The estimates of Mineral Resources and Mineral Reserves may be materially affected if mining, metallurgical, or infrastructure factors change from those currently anticipated at the Hycroft mine.

Inferred Mineral Resources have not been used in the economic analysis of the Hycroft mine.

17.1 HYCROFT MINERAL RESOURCE

17.1.1 RESOURCE DATA FOR GRADE ESTIMATION AND BLOCK MODELING

The resource estimate in this report was completed by SEWC under the direction of Scott E. Wilson, an independent Qualified Person as defined in NI 43-101.

17.1.1.1 Drilling Data

Drill hole data for the Hycroft property is maintained in a Microsoft database by Allied Nevada. Allied Nevada validates the database constantly and has certified the data to be clean and error free. The drill hole database has been converted to a Vulcan® Isis database named July2010_v1.dhd.isis.

17.1.1.2 Assay Corrections

No assay corrections were applied to the new resource block model. The Hycroft assay data was corrected for resource estimation as follows:

- If the assay interval was a pre-1999 hole and was not factored by ORE, and the alteration was acid leach, then the AuFA grade was multiplied by 1.32 to correct for drilling bias.
- If the assay interval was a pre-1999 hole and was not factored by ORE, and the alteration was oxide, then the AuFA grade was multiplied by 1.19 to correct for drilling bias.

17.1.1.3 Topographic Data

The most recent aerial survey (Aerographics - Salt Lake City, Utah, September 2009) was used as the base topography at Hycroft. Aerographics provided the topography in Autocad format and SEWC converted the topography to a Vulcan® triangulation surface. The triangulation surface was then updated with survey points from Hycroft mine operations to reflect mining up to the end of May 2010. The surface was used to separate air versus ground in the block model.

Allied Nevada generated an “as-mined”, “pre-backfill” surface that was used to code the block model with blocks that are still remaining in the ground, underneath dumps, and backfilled pits. The difference between the aerial survey surface and the as-mined and pre-backfill surface equals backfill material and this fill material was coded to the Vulcan® block model.

- Aerial Survey Surface triangulation – May2010SEWC.00t
- As-Mined Pre-Backfill triangulation – HardBottom_July10.00t

17.1.1.4 Geological Model

Geologic shapes representing lithology, alteration zones, and fault surfaces have been utilized to code the block model. The shapes were developed by the Hycroft geologic team based on surface mapping and drill hole logging data. Individual alteration shapes were constructed within the following Hycroft zones:

- Brimstone
- Vortex
- Leach pad
- Central Zone
- Camel
- Foothills

Major structural features were utilized to define zone boundaries. In addition, the boundary between backfill and the pre-backfill surface was reinterpreted for the Cut 5 area based on information from a significant number of holes drilled in 2010. Material above this interface was coded to the block model as fill. Table 17.1 lists the rock units and alteration shapes defined in the Hycroft block model.

17.1.1.4.1 Determination of Oxide vs. Mixed vs. Sulfide Zones

Zones of the various oxidation states of the Hycroft mineralization were determined with indicator kriging of the AuCN:AuFA ratios of the 5 foot raw assay information. Oxidation was classified as either oxide, mixed or sulfide by the following methods and formulas:

- Two columns (redox, ratio) were added to the drill hole database.
- The ratio of AuCN:AuFA was stored in the ratio column.
- If (AuFA – AuCN <= 0.005 and AuFA >0.005) then redox = 1, where 1 indicates oxide.
- If (ratio <= 30% and AuFA >0.005) then redox = 3, where 3 indicates sulfide.
- All other assay intervals set to 2, where 2 indicates mixed.
- This results in the redox field containing 1s, 2s or 3s.
- The redox values were combined into 25 foot composites based on the majority value of the five foot redox values.
- Determined the anisotropic preferred direction of the redox indicators.
- Estimated the redox indicators into the block model variable "redox."

The end result is the indicated oxidation state, as interpreted from statistical modeling of the ratios of AuFA:AuCN, and is stored in the block model. Based on observed mineralized rock coming from the Brimstone pit, it was determined that oxide and mixed material could be combined as oxide for the purpose of reporting Mineral Resources as either oxide or sulfide.

Table 17.1 Vulcan® Rock Units and Alteration Shapes

Vulcan® Triangulation	Geologic Zone
000_1007_Brim_acid.00t 000_1007_Camel.acid.00t 000_1007_Central_acid.00t 000_1007_Leachpad_Acid.00t 000_1007_Vortex_Acid.00t	Acid Leach (Alteration)
000_1007_Alluv_surf.00t	Alluvium (Rock Type)
000_1007_Brim_arg.00t 000_1007_Camel.arg.00t 000_1007_Central_arg.00t 000_1007_Leachpad_Arg.00t 000_1007_Vortex_Arg.00t	Argillized Rocks (Alteration)
000_1007_Brim_si.00t 001_1007_Camel.si.00t 000_1007_Central_si.00t 000_1007_Foothills_si.00t 000_1007_Leachpad_si.00t 000_1007_Vortex_si.00t	Silicified Rocks (Alteration)

17.1.1.5 Tonnage Factors

The densities were adjusted based on the previous technical report to the new rock type characterizations. Table 17.2 lists the densities used in this report.

Table 17.2 Hycroft Tonnage Factors

Geologic Zone	Tonnage Factor (ft^3/ton)
Alluvium	18
Backfilled Pits	20
Acid Leach	17.5
Oxide Silicic Alteration	13.7
Oxide Propylitic Alteration	14
Oxide Argillic Alteration	16
All Other Geologic Zone	14.25
Unaltered	14.25
Undefined	14.25

17.1.1.6 Drill Hole Compositing

Drill hole assays were composited using 25 ft down the hole composites for the entire Hycroft project. The start of the composite is the collar of the drill hole. If the downhole length of the composite was less than 12.5 ft, then no composite was generated. Intervals with no assays were ignored and then a new composite was generated at that point. Values below assay detection limits were set to 0.0001 opt for AuFA, AuCN and AgFA.

Geologic zone codes were added to the composites using the same geologic model solids that were used to define the geologic block model. Codes were assigned based on the location of the composite centroid relative to the geologic model solid.

17.1.1.7 Composite Statistics

Basic statistics were compiled for exploration drill hole data using geologic codes transferred from the geologic model polygons. Observations from these statistics include:

- The highest average gold grade is in the silicified unit, with an average fire assay grade of 0.410 opt AuFA. This composite was mined out in the Vista mining days.
- The highest grade of in situ oxide is 0.375 opt Au in the silicified unit.

17.1.2 RESOURCE ESTIMATION

17.1.2.1 Resource Model Definition

The Vulcan® resource block model for the Hycroft project subdivides the ore deposit into 50 x 50 x 25' blocks. All of the required information about the deposit is stored in each individual block. This includes estimated characteristics such as gold and silver grades. Statistical characteristics such as kriging variances, number of samples used in an estimate, distances to the nearest drill hole, etc., are also stored in each individual block for descriptive evaluations. Physical information stored in the blocks, which includes rock types, bulk densities, contained metal, and alteration is stored in order to evaluate engineering, production, and geotechnical parameters that might be utilized to determine the viability of mining the ore deposit.

17.1.2.2 Resource Model Dimensions

The Vulcan® model dimensions are listed in Table 17.3. The model is saved as July2010.bmf. All the blocks are orthogonal at the Selective Mining Unit of 25 ft. The model contains 7,650,720 blocks.

Table 17.3 Block Model Dimensions

	East	North	Elevation
Minimum Mine Coordinates	15,600	37,000	2,250
Maximum Mine Coordinates	24,800	53,500	5,400
Number of Blocks	184	330	126

17.1.3 HYCROFT GRADE MODEL

17.1.3.1 Variography Parameters

Fire assay and cold cyanide assays were estimated into the block model using ordinary kriging. Variograms were calculated for each ore deposit or domain on the property, and the results tabulated in Table 17.4.

Table 17.4 Hycroft Variograms

Variography Domin	Rotation About Z Axis	Rotation About X Axis	Rotation About Y Axis	Major Axis Length	Semi-Major Axis Length	Minor Axis Length	Nugget	Sill Differential	Range at 95% of Sill
Acid Leach	180	20	30	292	130	85	0.19	1.09	225
Silicified	160	10	0	221	170	80	0.17	1.11	175
Brimstone and Vortex - No Acid Leaching or Silicification	145	-13	61	350	345	30	0.17	0.8	180
Bay Area - No Acid Leaching or Silicification	264	-20	0	482	322	79	4	0.98	200
Central Fault - No Acid Leaching or Silicification	160	10	0	218	75	44	0.09	1.18	175
Global Variogaram - Structure 1	20	20	0	147	130	104	0.29	0.76	250
Global Variogaram - Structure 2	20	20	0	406	300	172	0.29	0.129	250

17.1.3.1.1 Gold Grade Estimation

Gold grade was estimated using ordinary kriging estimation with gold grade selection ranges and parameters varying according to the variography of each domain. The general procedure for creation of the gold grade model was as follows:

- The major axis of the search ellipse was oriented at the same angles as the variograms.
- A composite length had to be 12.5 ft, or half the SMU, in order to be used in the grade estimation run.
- AuCN grades were estimated using the gold variography parameters.

17.1.3.1.2 Resource Classification

Resource classes were based on the distance range of the variogram at 95% of the sill and the full length of the major axis of the variogram. Inferred Mineral Resources are defined at any block receiving an estimated grade, where there was at least one hole within the search ellipse. Indicated Resources are defined as being within the range at 95% of the sill (d95) and having at least two drill holes in the estimate. Measured Mineral Resources require a minimum of two drill holes for the estimate where at least one hole is within d95 and one hole must be within half the d95 distance, expressed as d95/2. Table 17.6 identifies the classification criteria for the Hycroft ore deposits.

Table 17.5 Hycroft Resource Classification Criteria

Domain	Measured (ft)	Indicated Distance (ft)	Inferred Distance (ft)
Acid	<112.5	112.5 – 225	>225
Silica	<87.5	87.5 – 175	>175
Brimstone/Vortex	<90	90 – 180	>180
Boneyard/Bay Area	<100.5	100 – 200	>200
Central	<87.5	87.5 – 175	>175
Foothills	<125	125 – 250	>250
Global	<125	125 – 250	>250

17.1.3.1.3 Resource Summary

The June 1, 2010 reserves are reported at a gold equivalent cut off grade of 0.009 for oxide mineralization and 0.018 for sulfide mineralization.

The remaining Hycroft Measured and Indicated Oxide, Sulfide and Combined Oxide/Sulfide Resource as of June 1, 2010 are shown in Tables 17.6 through 17.8. Resources have been tabulated separately for blocks where (a) the estimate of both gold and silver grades satisfy the Measured and Indicated criteria , and (b) only the estimate of gold grade satisfies the criteria. Silver, for the latter category, has been assigned a grade of zero for the overall Measured and Indicated total.

Table 17.6 Oxide Resources @ 0.009 opt Gold Equivalent Cut Off Grade

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Measured Resources					
Au & Ag Measured	105,000	0.013	0.49	1,365	50,925
Au Measured, Ag Inferred	65,000	0.014	-	910	-
Total	**170,000**	**0.013**	**0.30**	**2,275**	**50,925**
Indicated Resources					
Au & Ag Indicated	173,000	0.012	0.44	2,046	76,861
Au Indicated, Ag Inferred	53,000	0.011	-	583	-
Total	**226,000**	**0.012**	**0.34**	**2,629**	**76,861**
Measured & Indicated Resources					
Au & Ag Measured and Indicated	278,000	0.012	0.46	3,411	127,786
Au M & I, Ag Inferred	118,000	0.013	-	1,493	-
M & I Total	**396,000**	**0.013**	**0.32**	**4,904**	**127,786**

Table 17.7 Sulfide Resources @ 0.018 opt Gold Equivalent Cut Off Grade

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Measured Resources					
Au & Ag Measured	66,000	0.017	0.88	1,122	57,882
Au Measured, Ag Inferred	14,000	0.020	-	280	-
Total	**80,000**	**0.018**	**0.72**	**1,402**	**57,882**
Indicated Resources					
Au & Ag Indicated	87,000	0.017	0.85	1,515	73,540
Au Indicated, Ag Inferred	10,000	0.019	-	187	-
Total	**97,000**	**0.018**	**0.73**	**1,702**	**73,540**
Measured & Indicated Resources					
Au & Ag Measured and Indicated	153,000	0.017	0.86	2,637	131,422
Au M & I, Ag Inferred	24,000	0.019	-	467	-
M & I Total	**177,000**	**0.018**	**0.73**	**3,104**	**131,422**

Table 17.8 Combined Resources

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Measured Resources					
Au & Ag Measured	171,000	0.015	0.64	2,487	108,807
Au Measured, Ag Inferred	79,000	0.015	-	1,190	-
Total	**250,000**	**0.015**	**0.44**	**3,677**	**108,807**
Indicated Resources					
Au & Ag Indicated	260,000	0.014	0.58	3,561	150,401
Au Indicated, Ag Inferred	63,000	0.012	-	770	-
Total	**323,000**	**0.013**	**0.46**	**4,331**	**150,401**
Measured & Indicated Resources					
Au & Ag Measured and Indicated	431,000	0.014	0.60	6,048	259,208
Au M & I, Ag Inferred	142,000	0.014	-	1,960	-
M & I Total	**573,000**	**0.014**	**0.45**	**8,008**	**259,208**

The remaining Hycroft Inferred Oxide and Sulfide Resource as of June 1, 2010 is shown in Tables 17.9 through 17.11. The tabulation also includes the contained silver resource attributable to the Measured and Indicated Resources where only the gold grade satisfies the corresponding estimation criteria. The Inferred Oxide Resource includes the Crofoot pad gold resource as described in the March 2009 NI 43-101 Technical Report.

Table 17.9 Oxide Inferred Resources @ 0.009 opt Gold Equivalent Cut Off Grade

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Inferred Resource					
Ag Inferred Associated with Au M&I					31,344
Au and Ag Inferred	148,000	0.011	0.55	1,628	81,437
Crofoot Pad	35,000	0.009	-	318	-
Inferred Total	**183,000**	**0.011**	**-**	**1,946**	**112,781**

Table 17.10 Sulfide Inferred Resources @ 0.018 opt Gold Equivalent Cut Off Grade

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Inferred Resource					
Ag Inferred Associated with Au M&I					10,267
Au and Ag Inferred	153,000	0.017	1.07	2,601	163,644
Inferred Total	**153,000**	**0.017**	**-**	**2,601**	**173,911**

Table 17.11 Combined Inferred Resources

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Inferred Resource					
Ag Inferred Associated with Au M&I					41,611
Au and Ag Inferred	301,000	0.014	0.081	4,229	245,081
Crofoot Pad	35,000	0.009	-	318	-
Inferred Total	**336,000**	**0.014**	**-**	**4,547**	**286,692**

Tables 17.12 and 17.13 show the oxide and sulfide resource broken down by geologic domain.

Note that the following four tables are for informational use to show the distribution of grade and tonnage by domain and may not add up exactly to the reported resource totals due to rounding.

Table 17.12 Oxide Resources by Domain @ 0.009 opt Gold Equivalent Cut Off Grade

Area	Classification	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
			Au	Ag	Au	Ag
Bay	Au & Ag Measured and Indicated	23,487	0.017	0.12	399	2,844
	Au M & I, Ag Inferred	15,038	0.017	-	251	-
	M & I Total	**38,525**	**0.017**	**0.07**	**650**	**2,844**
Brimstone	Au & Ag Measured and Indicated	81,937	0.013	0.45	1,046	36,933
	Au M & I, Ag Inferred	6,652	0.012	-	77	-
	M & I Total	**88,589**	**0.013**	**0.42**	**1,123**	**36,933**
Camel Hill	Au & Ag Measured and Indicated	11,963	0.011	0.60	128	7,153
	Au M & I, Ag Inferred	6,275	0.012	-	73	-
	M & I Total	**18,238**	**0.011**	**0.39**	**200**	**7,153**
Central	Au & Ag Measured and Indicated	72,792	0.012	0.33	859	24,253
	Au M & I, Ag Inferred	73,861	0.013	-	970	-
	M & I Total	**146,653**	**0.012**	**0.17**	**1,829**	**24,253**
Foothills	Au & Ag Measured and Indicated	25,623	0.008	0.54	203	13,740
	Au M & I, Ag Inferred	8,660	0.008	-	67	-
	M & I Total	**34,283**	**0.008**	**0.40**	**270**	**13,740**
Leachpad	Au & Ag Measured and Indicated	5,265	0.010	0.48	52	2,509
	Au M & I, Ag Inferred	1,525	0.010	-	15	-
	M & I Total	**6,790**	**0.010**	**0.37**	**67**	**2,509**
Vortex	Au & Ag Measured and Indicated	57,482	0.012	0.71	680	40,677
	Au M & I, Ag Inferred	5,533	0.009	-	49	-
	M & I Total	**63,015**	**0.012**	**0.65**	**729**	**40,677**
Undefined	Au & Ag Measured and Indicated	198	0.005	0.31	1	62
	Au M & I, Ag Inferred	-	-	-	-	-
	M & I Total	**198**	**0.005**	**0.31**	**1**	**62**

Table 17.13 Sulfide Resources By Domain @ 0.018 opt Gold Equivalent Cut Off Grade

Area	Classification	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
			Au	Ag	Au	Ag
Bay	Au & Ag Measured and Indicated	6,577	0.026	0.15	170	984
	Au M & I, Ag Inferred	2,699	0.023	-	62	-
	M & I Total	**9,276**	**0.025**	**0.11**	**232**	**984**
Brimstone	Au & Ag Measured and Indicated	49,801	0.017	0.76	833	37,635
	Au M & I, Ag Inferred	2,846	0.016	-	46	-
	M & I Total	**52,647**	**0.017**	**0.71**	**879**	**37,635**
Camel Hill	Au & Ag Measured and Indicated	14,086	0.018	0.75	253	10,564
	Au M & I, Ag Inferred	3,451	0.017	-	59	-
	M & I Total	**17,537**	**0.018**	**0.60**	**312**	**10,564**
Central	Au & Ag Measured and Indicated	24,059	0.017	0.58	417	14,065
	Au M & I, Ag Inferred	13,603	0.019	-	252	-
	M & I Total	**37,662**	**0.018**	**0.37**	**670**	**14,065**
Foothills	Au & Ag Measured and Indicated	5,849	0.014	0.81	79	4,714
	Au M & I, Ag Inferred	375	0.012	-	4	-
	M & I Total	**6,225**	**0.013**	**0.76**	**83**	**4,714**
Leach pad	Au & Ag Measured and Indicated	1,710	0.014	0.83	23	1,413
	Au M & I, Ag Inferred	506	0.014	-	7	-
	M & I Total	**2,216**	**0.014**	**0.64**	**31**	**1,413**
Vortex	Au & Ag Measured and Indicated	54,619	0.017	1.15	908	62,583
	Au M & I, Ag Inferred	719	0.016	-	11	-
	M & I Total	**55,339**	**0.017**	**1.13**	**919**	**62,583**

Tables 17.14 and 17.15 show the inferred oxide and sulfide resource broken down by geologic domain.

Table 17.14 Oxide Inferred Resource by Domain @ 0.009 opt Gold Equivalent Cut Off Grade

Area	Classification	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
			Au	Ag	Au	Ag
Bay	Ag Inferred Associated with Au M+I					1,246
	Au and Ag Inferred	2,870	0.013	0.10	38	286
	Inferred Total	**2,870**	**0.013**		**38**	**1,533**
Brimstone	Ag Inferred Associated with Au M+I					3,186
	Au and Ag Inferred	33,881	0.011	0.36	358	12,220
	Inferred Total	**33,881**	**0.011**		**358**	**15,405**
Camel Hill	Ag Inferred Associated with Au M+I					3,037
	Au and Ag Inferred	14,162	0.011	0.26	150	3,718
	Inferred Total	**14,162**	**0.011**		**150**	**6,755**
Central	Ag Inferred Associated with Au M+I					15,475
	Au and Ag Inferred	36,609	0.011	0.31	408	11,373
	Inferred Total	**36,609**	**0.011**		**408**	**26,847**
Foothills	Ag Inferred Associated with Au M+I					5,972
	Au and Ag Inferred	10,105	0.007	0.52	75	5,250
	Inferred Total	**10,105**	**0.007**		**75**	**11,222**
Leach pad	Ag Inferred Associated with Au M+I					302
	Au and Ag Inferred	4,652	0.009	0.42	44	1,970
	Inferred Total	**4,652**	**0.009**		**44**	**2,272**
Vortex	Ag Inferred Associated with Au M+I					1,970
	Au and Ag Inferred	46,022	0.012	1.01	575	46,591
	Inferred Total	**46,022**	**0.012**		**575**	**48,561**
Undefined	Ag Inferred Associated with Au M+I					
	Au and Ag Inferred	154	0.006	1.70	1	262
	Inferred Total	**154**	**0.006**		**1**	**262**

Table 17.15 Sulfide Inferred Resources by Domain @ 0.018 opt Gold Equivalent Cut Off Grade

Area	Classification	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
			Au	Ag	Au	Ag
Bay	Ag Inferred Associated with Au M+I					249
	Au and Ag Inferred	293	0.018	0.20	5	58
	Inferred Total	**293**	**0.018**		**5**	**307**
Brimstone	Ag Inferred Associated with Au M+I					1,566
	Au and Ag Inferred	27,963	0.017	0.69	469	19,169
	Inferred Total	**27,963**	**0.017**		**469**	**20,735**
Camel Hill	Ag Inferred Associated with Au M+I					1,603
	Au and Ag Inferred	17,431	0.018	0.36	309	6,263
	Inferred Total	**17,431**	**0.018**		**309**	**7,866**
Central	Ag Inferred Associated with Au M+I					5,278
	Au and Ag Inferred	38,553	0.017	0.43	636	16,515
	Inferred Total	**38,553**	**0.017**		**636**	**21,794**
Foothills	Ag Inferred Associated with Au M+I					348
	Au and Ag Inferred	6,850	0.017	0.54	118	3,722
	Inferred Total	**6,850**	**0.017**		**118**	**4,070**
Leach pad	Ag Inferred Associated with Au M+I					303
	Au and Ag Inferred	3,688	0.011	0.98	40	3,626
	Inferred Total	**3,688**	**0.011**		**40**	**3,929**
Vortex	Ag Inferred Associated with Au M+I					581
	Au and Ag Inferred	58,160	0.017	1.96	984	114,084
	Inferred Total	**58,160**	**0.017**		**984**	**114,665**
Undefined	Ag Inferred Associated with Au M+I					
	Au and Ag Inferred	162	0.013	0.61	2	99
	Inferred Total	**162**	**0.013**		**2**	**229,429**

17.1.3.2 Resource Summary – Grade/Tonnage Charts

The tonnage versus grade chart for all categories of mineralization combined is shown below in Table 17.16 (oxide) and Table 17.17 (sulfide). The AuEq, AuFA, and AgFA grades are shown in opt.

Table 17.16 Grade Ton Chart Oxide Measured, Indicated & Inferred

Cut Off	Tons	Au Eq	Eq Oz	Au FA	Au Oz	Ag FA	Ag Oz
0.005	980,351,101	0.014	13,724,915	0.009	8,725,125	0.29	283,713,609
0.006	839,202,373	0.015	12,923,717	0.010	8,140,263	0.33	272,992,532
0.007	726,680,797	0.017	12,208,237	0.010	7,557,480	0.36	262,985,780
0.008	630,408,621	0.018	11,473,437	0.011	7,060,577	0.40	252,541,694
0.009	544,744,420	0.020	10,731,465	0.012	6,536,933	0.44	241,049,406
0.010	474,587,404	0.021	10,061,253	0.013	6,027,260	0.49	230,744,396
0.011	414,087,071	0.023	9,441,185	0.013	5,548,767	0.53	221,122,496
0.012	363,744,741	0.024	8,875,372	0.014	5,128,801	0.58	212,463,303
0.013	321,285,761	0.026	8,321,301	0.015	4,755,029	0.64	204,723,287
0.014	286,468,911	0.027	7,849,248	0.015	4,411,621	0.69	197,634,902
0.015	255,480,639	0.029	7,408,939	0.016	4,062,142	0.75	190,971,778
0.016	228,550,618	0.031	6,993,649	0.017	3,771,085	0.81	184,737,465
0.017	204,777,028	0.032	6,593,820	0.017	3,481,209	0.87	178,545,091
0.018	182,568,157	0.034	6,207,317	0.018	3,194,943	0.94	172,143,515
0.019	163,947,056	0.036	5,869,305	0.018	2,967,442	1.01	166,242,315
0.020	147,646,512	0.038	5,551,509	0.019	2,746,225	1.09	160,506,523
0.021	133,130,589	0.040	5,258,658	0.019	2,542,794	1.16	154,990,632
0.022	121,143,498	0.041	4,991,112	0.020	2,362,298	1.24	150,145,251
0.023	110,898,623	0.043	4,768,641	0.020	2,217,972	1.31	145,676,431
0.024	101,963,865	0.045	4,557,785	0.020	2,080,063	1.39	141,362,702
0.025	94,141,152	0.046	4,368,149	0.021	1,958,136	1.46	137,370,769

Table 17.17 Grade Ton Chart Sulfide Measured, Indicated & Inferred

Cut Off	Tons	Au Eq	Eq Oz	Au FA	Au Oz	Ag FA	Ag Oz
0.005	1,581,568,160	0.015	23,723,522	0.010	16,131,995	0.29	457,863,982
0.006	1,467,971,422	0.016	23,487,543	0.011	15,560,497	0.31	450,667,227
0.007	1,353,950,741	0.017	23,017,163	0.011	15,028,853	0.33	443,283,473
0.008	1,233,887,337	0.018	22,209,972	0.012	14,313,093	0.35	434,204,954
0.009	1,106,740,084	0.019	21,028,062	0.012	13,391,555	0.38	422,664,038
0.010	978,321,914	0.020	19,566,438	0.013	12,424,688	0.42	409,525,553
0.011	856,275,783	0.021	17,981,791	0.013	11,388,468	0.46	395,428,157
0.012	747,125,204	0.023	17,183,880	0.014	10,385,040	0.51	381,631,554
0.013	646,290,778	0.024	15,510,979	0.015	9,371,216	0.57	367,287,049
0.014	562,195,581	0.026	14,617,085	0.015	8,432,934	0.63	353,621,020
0.015	490,065,854	0.028	13,721,844	0.016	7,645,027	0.70	340,742,788
0.016	428,889,293	0.029	12,437,789	0.016	6,905,118	0.77	328,314,754
0.017	378,460,433	0.031	11,732,273	0.017	6,282,443	0.84	316,657,844
0.018	333,998,554	0.033	11,021,952	0.017	5,677,975	0.91	305,408,278
0.019	296,128,443	0.035	10,364,496	0.018	5,182,248	0.99	294,558,962
0.020	263,453,288	0.037	9,747,772	0.018	4,715,814	1.08	284,371,479
0.021	235,620,591	0.039	9,189,203	0.018	4,311,857	1.17	275,558,281
0.022	210,317,748	0.041	8,623,028	0.019	3,911,910	1.27	266,788,063
0.023	189,081,870	0.043	8,130,520	0.019	3,592,556	1.37	258,456,008
0.024	169,849,688	0.045	7,643,236	0.019	3,261,114	1.48	250,749,094
0.025	151,909,945	0.047	7,139,767	0.020	2,977,435	1.60	242,478,654

17.2 HYCROFT MINE MINERAL RESERVES

Mineral reserves at Hycroft were determined by applying current economic criteria that are valid for the Hycroft mine. These limitations were applied to the SEWC resource model in order to determine which part of the Measured and Indicated Resource is economically extractable. Of the total Hycroft Mineral Resources, only the Brimstone, Cut-5, Camel, Bay Area, Central, and Boneyard deposits were considered for reserve level determinations. The reported reserves meeting the standards as set forth in NI-43-101, December 23, 2005, where:

- *A Probable Mineral Reserve is defined* as 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 defined as* 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.

SEWC evaluated Allied Nevada's current economic forecasts and used these criteria to report the Proven and Probable Mineral Reserves for the Hycroft project. Mining sequences were determined and mining plans were developed. Allied Nevada will employ these mine plans and recommendations into its life-of-mine scheduling and budgeting processes. All the necessary federal, state, and local operating permits are in place and a bond payment has been posted. SEWC believes that there have been sufficient evaluations of mining, processing, economic, and environmental factors to support the determination of Proven and Probable Reserves for the Hycroft mine.

The results of SEWC's economic calculations are that there are sufficient quantities of gold and silver within the Brimstone deposit to sustain all of the mining, processing, general, and administrative costs associated with the operation of the Hycroft mine. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.

17.2.1 RESERVE DETERMINATION

The resource model was built by SEWC with Vulcan® Scientific Modeling software. SEWC uses Vulcan® mine planning software for reserve calculations and pit designs. SEWC uses Whittle® 4X software to determine the economic limits of pits. Whittle® 4X uses the Lerchs-Grossman© economic algorithm, which is an industry standard method for optimizing open pit resources. SEWC evaluates numerical data and generates mining schedules and sequences with Microsoft® Excel spreadsheet software.

17.2.2 RESERVE DETERMINATION PROCEDURES

- Export selected data from Vulcan® in a format suitable to be evaluated with Whittle® 4X software.
- Calculate economic and physical constraints to be used as inputs to Whittle® 4X.
- Optimize pit shells with Whittle's® Lerchs-Grossman© algorithm.
- Export pit shells from Whittle® and import into Vulcan®.
- Use pit shells as guides to complete the ultimate economic pit design.
- Design suitable mining phases that can allow for logical extraction of ore and waste from the ore deposit.
- Tabulate Measured and Indicated Resources within the ultimate pit that meet the criteria to be reported as Proven and Probable Reserves.

17.2.3 ECONOMIC PARAMETERS TO DETERMINE RESERVE LEVEL PIT DESIGN

Economic inputs for calculating cut off grades and for input to Whittle® 4X are listed in Table 17.18. Physical design parameters for the Reserve pits are shown in Table 17.19.

Table 17.18 Economic Design Parameters

Value	Description	Units
$800.00	Gold price	$/oz
$14.00	Silver price	$/oz
$1.28	Cost of mining	$/ton
$1.41	Cost of processing	$/ton ore
$2.51	Cost of crushing (for increased recovery of high grade ore)	$/ton ore
$0.47	Cost of administration, Jungo Road, environmental, reclamation	$/ton ore
Variable	AuFA gold recovery based on location, processing method and metallurgy	%

Table 17.19 Pit Design Parameters

Description	Value
Slope Angle	40 to 46°
Bench Height	25 ft
Road Width	60 to 120 ft
Maximum Loaded Ramp Grade	10%
Minimum Mining Width	200 ft
Tonnage Factor ft^3/ton	13 to 20

17.2.4 DILUTION

SEWC believes that the model blocks are of sufficient size to account for dilution related to open pit mining practices. Allied Nevada is using industry accepted ore control software and methods to delineate ore and waste boundaries. Periodic reconciliations will be done to account for and quantify dilution problems. Adjustments will be made, as necessary, to ensure that waste and ore are categorized properly during the mining process.

17.2.5 CUT OFF GRADES

SEWC evaluated the cut off grade based on the current costs associated with the mining of the Brimstone deposit. SEWC used a gold equivalent cut off grade of 0.005 opt Au, a gold selling price of $800, and a silver selling price of $14.00 (the approximate three year average London Fixed Gold Price) to verify reserves for the Brimstone pit.

17.2.6 HYCROFT MINE MINERAL RESERVES STATEMENT

Table 17.20 June 1, 2010 Mineral Reserves

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Proven Reserve	135,000	0.014	0.23	1,900	30,600
Probable Reserve	39,000	0.013	0.21	500	8,300
Total Reserve	**174,000**	**0.014**	**0.22**	**2,400**	**38,900**
Waste	179,000				
Total Pit Tons	**353,000**				
Strip Ratio	1.04:1				

18 OTHER RELEVANT DATA AND INFORMATION

18.1 OXIDE EXPANSION

Allied Nevada is implementing an accelerated oxide mining plan. This mine plan is designed to accelerate extraction of the oxide reserves based on a number of unique mining phases allowing for several to be mined simultaneously.

The accelerated mining rate will be phased in over the next two years (2010 and 2011), with expected production increasing from approximately 100,000 ounces in 2010 to over 264,000 ounces of gold in 2012 and averaging over 300,000 ounces in each of 2013 and 2014. This accelerated plan assumes that an average mining rate of 85 million tons of total material per year will be achieved by 2013.

Note, that within the 353 million tons being mined during the heap leach operation, 27 million tons of sulfide material grading 0.021 opt Au at a cutoff grade of 0.018 opt has been identified. This sulfide material will be stockpiled in anticipation of future processing.

The total capital required to implement this mining plan is expected to be approximately $212 million over the life of the mine.

Allied Nevada expects to receive permits in 2012 that will allow mining in Boneyard, Bay Area, Cut-5, Central Pit, and Camel Hill, in addition to extensions in Brimstone. New areas for overburden storage will be developed north of the Bay/Boneyard areas, and on the east side of the Crofoot pad. These newly permitted areas will allow development of three to five mining faces at one time resulting in both accelerated waste removal and ore exploitation. Please see Figure 23.1 showing the mining areas and 23.2 showing the dumps and leach pads.

The Brimstone heap leach pad will be expanded to the north 5,000 ft, from 49,000N to 54,000N. The new pad will tie into the existing pad starting at 4,500 ft in width and tapers down to 1,000 ft at the north extent. The majority of the construction for the north pad occurs in 2012.

The south heap leach pad is located south-southwest of the Crofoot pad, starting at 37,000N and extending the south to 34,000N. An area is left between the Crofoot pad and the new South heap leach pad to allow for drainage from the mountains located east of the mine. The pad will be approximately 3,000 ft north-south and 4,000 ft east-west, and will be built in stages as required to minimize on upfront capital.

Two new Merrill-Crowe plants will be constructed for the mine during the expansion – one at the north end of the Brimstone heap leach pad to maximize gravity drainage of the Brimstone pad. The second Merrill-Crowe plant will be built to the west of the South pad also taking advantage of gravity flow. The concentrates from these Merrill-Crowe plants will be delivered to the existing refinery located at the Merrill-Crowe plant west of the Brimstone heap leach pad.

The Center Dump will be full in 2012. New dumps to the north of the Bay Area/Boneyard pits will be opened in 2012. A dump on the east side of the Crofoot pad will be re-opened for waste being mined from the Bay Area and Central fault pits in 2013. In-pit dumping of waste will occur as the pits are fully exploited. As the Bay Area is mined from north to south, the mined out areas will be backfilled.

Scheduling of the pits is based on maximizing NPV along with balancing waste hauls.

The pits are designed with 120 ft wide haul roads grading 10% allowing for 240 to 320 ton haul trucks to move either ore or waste.

Based on metallurgy, the ore will be delivered to the ROM heap leach pads or the crusher, crushed and then delivered to the heap leach pads. Optimal crushing size is to -3/8" allowing for the maximum recovery of the gold and silver. Approximately 10% of the oxide ore will be crushed, using the new MACS (Mobile Aggregate Crushing System, Terex/Cedar Rapids) jaw crusher expected to be operational at the start of 4th qtr 2010. Previously, a contractor crushed a small amount of more competent oxide ore for use as over liner.

The plan, based on Proven and Probable Mineral Reserves, has a mine life of five years, through 2015. Infill drilling and additional metallurgy test work is underway to assess the potential to expand the mine life.

Table 18.1 shows the mine plan, operating costs and cash flow for the LOM, including all of 2010. Based on a discount rate of 6% the discounted cash flow is $198 million.

Please refer to Figure 23.1 showing the mining areas and Figure 23.2 showing the mining areas along with the infrastructure (dumps and pads).

Table 18.1 Operating Costs

Tons of Ore Processed	175,258,972
Grade opt Au	0.014
Ounces of gold sold	1,404,133
Total Revenue	$1,123,306,329
Revenue per Ton Processed @ $800 Au	$6.41
Cost per Ton Mined	$1.07
Processing Cost per Ton	$1.37
Crushing Cost per Ton	$2.15
General and Administration Cost per Ton Processed	$0.144
Mining Cost per Ton of Ore Processed	$2.24
Net Present Value at 6%	$198,106,602

18.2 MILL SCOPING PROJECT

Large sulfide resources at the Hycroft property lay immediately below the current Hycroft mine oxide reserve and resources. An initial sulfide scoping study (the "Hycroft Mill Scoping Study"), prepared by SEWC, indicates that the sulfide resource could be extracted and processed concurrently with the oxide ore reserve. This section discusses the Hycroft Mill Scoping Study. The study relates to the economic potential of the Hycroft mine sulfide deposit based on the measured, indicated and **inferred** resource and is preliminary in nature, and accordingly subject to a high degree of uncertainty.

Since the Hycroft Mill Scoping Study includes inferred mineral resources we caution the reader that inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves and that there is no certainty that the Hycroft Mill Scoping Study will be realized. A preliminary feasibility or definitive feasibility study will be required to evaluate future project economics related to the Hycroft Mill Scoping Study.

With the latest drilling information, both oxide and sulfide resources have increased. Looking beyond the current oxide heap leaching plan and using economic factors shown in Table 18.2, the pits were expanded, as part of the study, to understand to what extent the remaining resources could be extracted. The resulting pit contains 688 million tons of combined heap leach and mill material in all resource categories and 1.1 billion tons of waste and sub-grade material, in addition to the current proven and probable reserves.

Allied Nevada currently envisions that mining will take place via open pit mining methods. The sulfide ore will be crushed, ground, and a flotation concentrate produced utilizing a flotation mill at an assumed rate of 100 k tpd. Cyanidation will be applied to the flotation tails for added incremental recovery. The sulfide concentrate would be shipped off-site to be oxidized via pressure oxidation, roaster, or bio-leach techniques, and the oxidized concentrate would then be leached and processed, resulting in gold and silver doré. Alternatively, the concentrate could be oxidized and leached and a gold/silver doré produced on site with the construction of an oxidation circuit. The doré from the sulfide process circuit would be shipped to an independent metal refinery to be refined into gold and silver for sale. For the purposes of this study, costs associated with shipping a sulfide concentrate off-site for the oxidation and leaching steps are assumed.

During the years that the oxide heap leach operation are producing concurrently with the milling operation, the study indicates that the combination could produce an annual average of 610,000 ounces of gold and 27,000,000 ounces of silver over a ten year life.

Note, that the sulfide pit generated for this study contains approximately equal amounts of oxide and sulfide mineral material above an economic cutoff grade. Ongoing and future metallurgical testing will ascertain the optimum method of treatment.

Receipt of required federal and state permits is expected to take approximately three years. Table 18.2 lists the estimates used in this mill scoping project.

Utilizing capital and operating costs developed for this study, current metallurgical test work related to the sulfides, a gold price of $800 per ounce, and a silver price of $14.00 per ounce, the study indicates that the sulfide project has the potential to generate $1.78 billion of free cash flow, and pays back the initial project capital of $1.1 billion in 2.9 years.

Table 18.2 Mill Scoping Study – Key Operating and Financial Parameters

LOM production	
Oxide Ore Tons Mined – Heap Leach	303,246,082
Oxide Ore Tons Mined – Mill	64,983,870
Sulfide Ore Tons Mined – Mill	319,666,100
Pre-strip Overburden	85,000,000
Total Waste Tons Mined	1,070,000,000
Total Tons	**1,757,896,052**
Strip Ratio	1.56
Cont Ounces Au, Oxide	3,635,000
Cont Ounces Au, Sulfide	4,911,000
Cont Ounces Ag, Oxide	163,266,000
Cont Ounces Ag, Sulfide	266,504,000
Ounces AuEq	16,066,975
Stockpile Sulfide Feed from Oxide Plan	27,000,000
Average Annual Production (including Stockpile)	
Mine Life	13
Plant Life	12
Ore Tons Mined, Heap Leach	25,786,231
Ore Tons Mined, Mill	32,708,331
Waste Tons Mined	77,194,357
Total Tons Mined (not including pre-strip)	**135,688,920**
Mining Rate, tpd	371,750
Ore Tons Processed, Heap Leach	25,786,231
Ore Tons Processed, Mill (including stockpile)	35,004,249
Recovered Heap Leach Ounces Au	123,211
Recovered Mill Ounces Au	486,925
Recovered Heap Leach Ounces Ag	496,388
Recovered Mill Ounces Ag	26,530,282
Total Recovered Ounces Au	610,136
Total Recovered Ounces Ag	27,026,670
Recovered Ounces AuEq	1,083,103
Capital Cost (millions)	
Mill, Tailings Facility and Leach Pads	$883.6
Additional Mine Equipment	$214.0
Other	$20.0
Total Capital	**$1,117.6**
Operating Costs	
Mining Cost per Ton	$1.29
Milling Cost per Ton	$7.65
Transportation of Concentrate	$20.00
Oxidation of the Concentrate	$21.93
Concentrate Ratio	20:1
Heap Leach Cost per Ton of Ore	$1.40
G&A per Ton of Ore	$0.20
Mill Au Grade (opt)	0.016
Mill Ag Grade (opt)	0.952
Mill Au Recovery	87.2%
Mill Ag Recovery	81.7%
Heap leach Au Grade (opt)	0.008
Heap leach Ag Grade (opt)	0.160
Heap leach Au Recovery	56.6%
Heap leach Ag Recovery	12.0%
Cost of sales per ounce of gold sold (after silver byproduct credit) ($/Oz)	$348.79
* The scoping study assumes 350 working days in a year.	

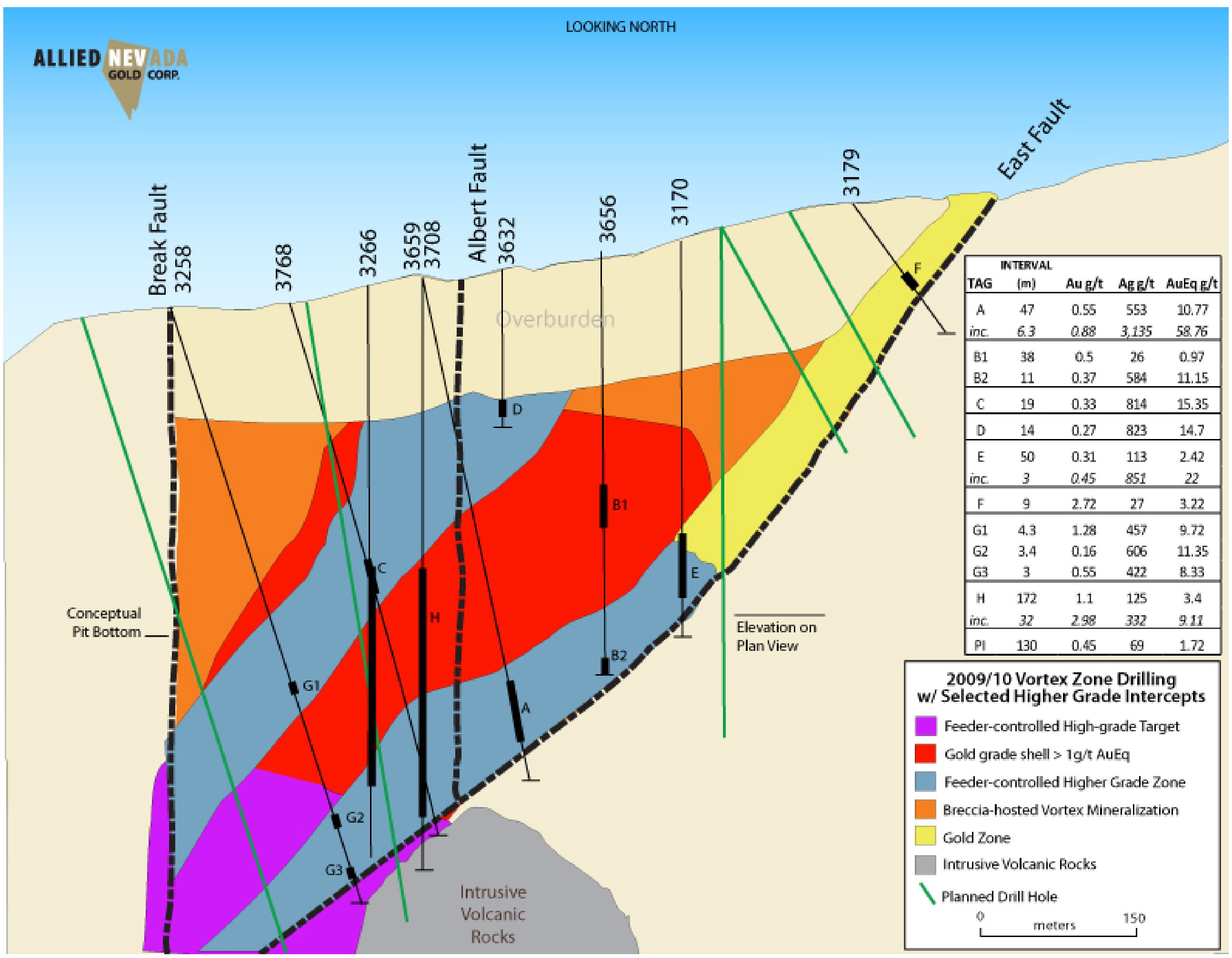
18.2.1 SULFIDE GEOLOGY

The large Hycroft sulfide resource underlies the oxide reserve and resource throughout much of the mine area. The sulfide system has been explored along a strike length of three miles and a width of two miles.

At Brimstone, sulfide mineralization occurs below the oxide reserve presently being mined. The sulfide hosted gold and silver resource occurs in pervasively silicified, veined, and brecciated volcanic rocks of the Kamma Mountains Group. Most sulfide mineralization is in the hanging wall of the East fault.

Gold and silver bearing host rocks in the Vortex Zone are correlative with those at the Brimstone deposit. Vortex sulfide mineralization is associated with silicified, highly brecciated, and abundantly veined volcanic rocks. Bonanza grades of silver (pyrargyrite) and gold mineralization are present within the zone (Figure 18.1). The Vortex Zone mineralization, which is the focus of current exploration drilling, is open to the south, west, and to depth.

Figure 18.1 Vortex Zone Drilling



TAG	INTERVAL (m)	Au g/t	Ag g/t	AuEq g/t
A	47	0.55	553	10.77
inc.	*6.3*	*0.88*	*3,135*	*58.76*
B1	38	0.5	26	0.97
B2	11	0.37	584	11.15
C	19	0.33	814	15.35
D	14	0.27	823	14.7
E	50	0.31	113	2.42
inc.	*3*	*0.45*	*851*	*22*
F	9	2.72	27	3.22
G1	4.3	1.28	457	9.72
G2	3.4	0.16	606	11.35
G3	3	0.55	422	8.33
H	172	1.1	125	3.4
inc.	*32*	*2.98*	*332*	*9.11*
PI	130	0.45	69	1.72

Bay Area sulfide mineralization is hosted in a flat laying, pervasively silicified, conglomerate of the Sulfur Group. Exploration of Bay Area sulfide mineralization is in progress north and south of the defined resource.

Central Zone sulfide mineralization is also hosted in pervasively silicified conglomerate of the Sulfur Group. Higher grade gold and silver mineralization is spatially associated with a well developed set of through going planar chalcedonic and quartz alunite veins, oriented sub-parallel to the Central Fault zone. Additional exploration of the Central Zone sulfide system will be conducted in 2010.

Mineralization in the Cut-5 area extends to depths greater than 1,000 ft in places, and is of both disseminated and vein type. Some pyrargyrite veins have been noted at depth, and mineralized siliceous veins, on strike with veins in the Central Zone Cut-4 pit to the north, are common. Additional exploration of the Cut-5 sulfide system will be conducted in 2010.

Silver Camel sulfide mineralization is hosted by pervasively silicified conglomerate rocks. Gold and silver occur disseminated and in veins, associated with pyrite and marcasite. Higher grade veins, including silver bearing pyrargyrite veins, have been intersected.

18.2.2 EXPLORATION RELATED TO SULFIDES

The focus of Allied Nevada's recent exploration efforts has been to increase drill density and therefore confidence in the sulfide gold and silver resource located at Hycroft.

Through mid 2010, 676 holes or 534,397 ft of drilling by Allied Nevada have been completed at Hycroft.

Approximately 50% of this drilling was directed towards definition of the sulfide resource at Hycroft. Additional drilling is currently underway. The current drill program has been designed to: (a) improve confidence in the current estimate of sulfide mineralization, (b) expand the sulfide gold and silver resource, and (c) obtain additional samples for metallurgical test work. Management intends to continue exploration over the next 12 to 18 months.

18.2.3 SULFIDE RESOURCE

A large Measured and Indicated Gold and Silver Resource contained in the sulfide system has been delineated and addressed in this report. The Measured, Indicated and Inferred Mineral Resources are listed in Table 18.4. The Measured, Indicated and Inferred resources form the basis of the Hycroft Mill Scoping Study.

Table 18.3 Measured, Indicated and Inferred Sulfide Gold and Silver Resources at June 1, 2010

	Tons (000s)	Grade (opt)		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Measured & Indicated Resources					
Au & Ag Measured & Indicated	153,000	0.017	0.86	2,637	131,422
Au M&I, Ag Inferred	24,000	0.019	-	467	
M&I Total	**177,000**	**0.018**	**0.73**	**3,104**	**131,422**
Inferred Resources					
Ag Inferred Resource Associated with M&I Au					10,267
Au and Ag Inferred	153,000	0.017	1.07	2601	163,644
Total	**153,000**	**0.017**		**2,601**	**173,911**

18.2.4 SULFIDE METALLURGY

Metallurgical test work conducted on the sulfide material to date indicates that an overall gold recovery of 88% and an overall silver recovery of 86% may be achieved. Test work indicates that the preferred method for processing the sulfide material is to crush it, followed by grinding and flotation to produce a concentrate containing gold, silver, silica, and sulfides. Testing also shows that additional incremental recovery can be achieved by cyanidation of the flotation tails. The concentrate would likely be transported off site for further processing, though on-site processing will be investigated.

18.2.5 METALLURGICAL TESTING OF UNOXIDIZED MATERIAL

In September 2008, Allied Nevada sent eight individual composites, combined to form a master composite, to represent unoxidized material to SGS to investigate recovery methods. This was the first step required to move the deep sulfides into the indicated category. The purpose of the program was to develop a preliminary process flow sheet. The program incorporated ore characterization tests as well as the evaluation of a number of metallurgical processing options including gravity separation, flotation, and cyanidation. Analysis of a cleaner flotation concentration from the master composite, shown in Table 18.4, indicates the material contained mainly pyrite, with moderate amounts of quartz.

Table 18.4 Composite Cleaner Flotation Concentrate Mineralogy

Mineral Constituent	Mineral Mass %
Pyrite/Marcasite	72
Chalcopyrite	0.13
Arsenopyrite	0.26
Other Sulfides	0.07
Quartz	16.6
Feldspar	3.15
Clays	2.01
Micas	4.15
Other Silicates	0.04
Ti Oxides	0.19
Other Oxides	0.04
Native Sulfur	0.25
Sulfates	1.04
Carbonates	0.01
Ag Mineral	0.01
Other	0.05
Total	**100%**

The master composite cleaner flotation concentrate, shown above, contains a high pyrite/marcasite mass which would produce an exothermic reaction in both a roaster and autoclave. This reaction would reduce fuel requirements and overall process costs.

Additional test work conducted in 2009 and in 2010 included flotation tests on individual deposit samples from Cut-5, Bay Area, Boneyard, and Vortex for variability analysis to confirm the master composite results.

18.2.5.1 Sulfide Master Composite Description

The master composite was compiled from eight individual composites (Zone A through H) to represent the known deposit. Test work included head analysis and metallurgical test work on the individual composites and the master composite.

Individual and master composite head analysis indicate:

- Gold and silver grades were slightly lower than anticipated at 0.015 opt Au and 0.467 opt Ag. The average gold and silver values anticipated, based on information provided by Allied Nevada, were 0.018 opt Au and 0.505 opt Ag.
- At 15.6 Kwh/ton, the bond ball mill work index is considered to be moderately hard in terms of grindability in comparison to the SGS grindability database.
- A scoping level mineralogical evaluation revealed that pyrite was the principle sulfide mineral observed at an estimated 1 to 5% in all eight composites. Marcasite, which is essentially pyrite with a chemical composition slightly weighted toward Sulfur (Sulfur content in marcasite is

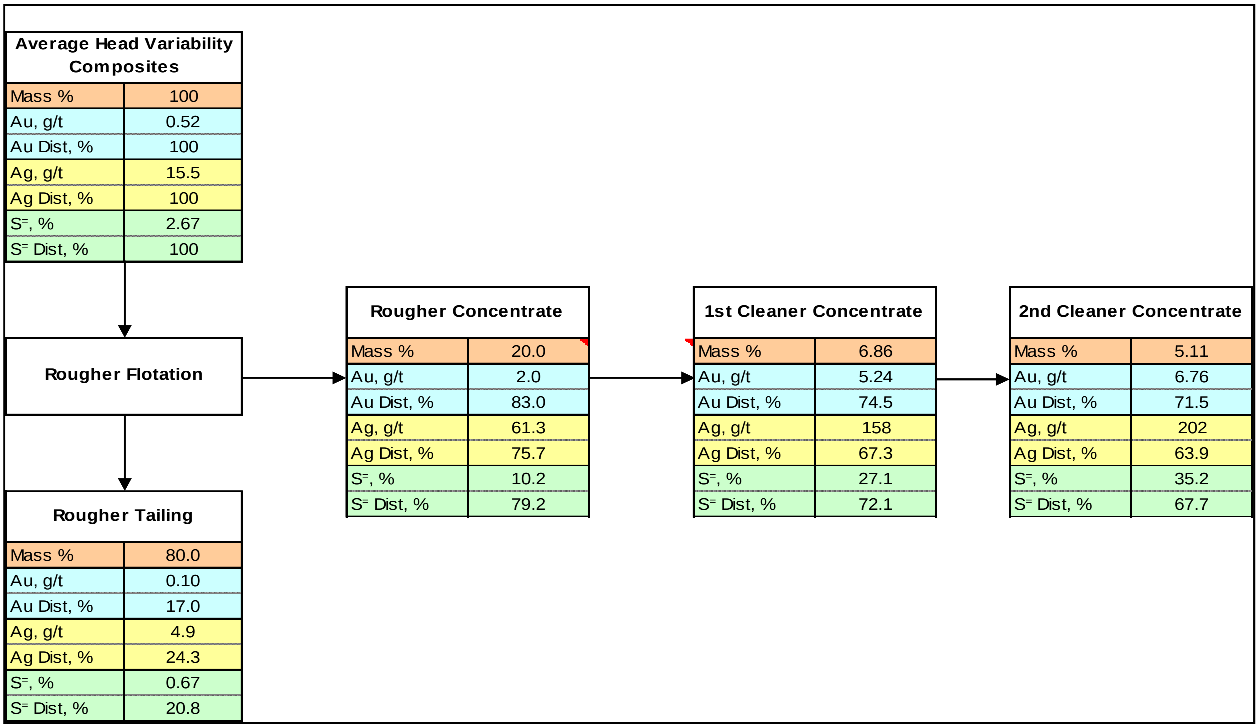
53.5% and only 53.4% in pyrite, both are FeS_2), was present in the range of 1 to 5% in zone composites A, B and C and <1% in the others. Pyrrhotite occurred at <1% in zone composites E, F and G only. Stibnite, at <1% was noted only in zone composite B. Graphite was identified in zone A. Note that the mineralogy study was intended for very basic scoping level reconnaissance information only. A more in depth study may reveal other significant parameters not identified here.

- Gravity separation test work indicated that there is very limited potential for significant gold/silver recovery through gravity methods. Gold and silver content appears to be mainly in intimate association with sulfide species, which, in turn, appear to be fairly fine and not amenable to efficient recovery by gravity methods.
- Flotation through this program has developed to the point where approximately 80 to 85% gold, silver, and sulfide rougher recoveries are attainable in a relatively simple flow sheet. Further flow sheet development is clearly required to attain cleaner (higher grade) rougher concentrates and to improve recoveries. Allied Nevada believes that there are still a significant number of options available for testing in the rougher circuit, including:
    - Optimizing pulp viscosity and density conditions.
    - Evaluating the potential of desliming prior to rougher flotation.
    - Investigating coarser primary grinding (the test work in this program was somewhat inconclusive with regard to establishing the upper grind limit).
    - Conducting a more in depth evaluation of activators and activator combinations in the rougher circuit.
- Flotation cleaning efficiencies were reasonably high and appear to present no insurmountable challenges.
- Whole ore cyanidation yielded very low extractions for both gold and silver. At a primary grind size of 65 µm (P_{80}), gold extraction was 15% while silver was 45%. Based on the available metallurgical evidence, it appears that the associations of gold and silver differ somewhat.

Cyanidation tests completed on cleaner flotation concentrate also yielded poor unit (and overall) extraction values. At a feed size of approximately 20 µm (P_{80}), gold and silver extractions were 13.8% and 55.9% respectively. Considering flotation recovery, those values correspond to 8.5% (gold) and 36% (silver) overall recovery. Ultra fine grinding to P_{80} = 4 µm improved extraction somewhat but also resulted in rather disappointing overall recovery values of 14% and 54% for gold and silver respectively.

A single aqua regia acid leach test completed on the cleaner flotation concentrate resulted in approximately 99% sulfide oxidation and yielded excellent similar degrees of gold and silver extraction. This result indicated that the refractory pre-treatment oxidation process will be necessary to achieve adequate gold and silver recovery from the Hycroft sulfide ore.

Figure 18.2 Master Composite Flotation Results



18.2.5.2 Variability Testing of sulfides

A total of 27 composites from Cut-5, Boneyard, Bay Area, and Vortex deposits were tested at SGS between autumn 2009 and spring 2010. The first study included seven composites from the Vortex deposit. The second study included 20 additional composites from the other deposits.

Testing of the initial seven samples confirmed the results from the previous test work. Head assays for sulfide mineralization ranged from 0.014 to 0.037 opt Au, 0.004 to 1.668 opt Ag and from 2.06 to 9.26% Sulfide. Flotation test results indicate recoveries of gold and silver averaging approximately 88% and 86%, respectively. Sulfide content of the cleaner concentrate was about 25%. Results from this data verify the master composite results using the identical flotation test procedures.

The additional 20 variability samples do not represent the various deposits, but were used for comparison to confirm the original master composite results. Head assays ranged from 0.008 to 0.048 opt Au, 0.050 to 3.333 opt Ag and from 1.0 to 8.0% sulfide sulfur. These samples were geologically identified and ranged from highly siliceous to highly argillaceous.

Rougher flotation gold recovery averaged 79.7% while ranging from 52% to approximately 96%. Silver recovery averaged 71.7% and ranged from 22% to approximately 97%. Average rougher sulfide recovery was 82.1%, ranging from 52% to 97%.

Cleaning in two stages resulted in an average gold recovery of 52% and ranged from 6 to 84%. Second cleaner concentrate silver recovery averaged 47.9% while ranging from approximately 4 to 92%.

The variability in metallurgical response that was evident in the original master composite components test work was also observed in the work completed on the 27 variability samples. It should be noted that, as these were essentially intended as variability tests, no metallurgical optimization test work was completed prior to variability testing. There is little doubt that the apparently negative metallurgical outcomes (recovery) observed in the test work completed on several of these composites would be easily improved with minimal metallurgical effort.

It is important to recognize that the flotation flow sheet applied in this test work had been developed for one specific composite sample (master composite) and, due to significant mineralogical differences among the many individual samples evaluated, metallurgical response naturally varied. Further testing, for the purpose of developing a robust and industrially sound flotation flow sheet, should focus on first identifying the ore components that are specifically responsible for the variability observed (the argillaceous nature of many of the ores tested is likely to be significant). Having identified those, the ores should be grouped into similar sets, in terms of mineralogy and general metallurgical response or expectation. The basic flow sheet, so far developed, is very likely to be the core flow sheet that will apply across the board for the Hycroft sulfide ores. Identifying the required modifications in specific parameters (pH, ORP requirements, activator, depressant/dispersant, collector, etc.) for each individual ore type, is very likely the key to maximizing flotation response for all ore types. The argillaceous fraction of the deposits is estimated to be low.

The tailing components that represent "end of pipe" material in an industrial plant, specifically the rougher and first cleaner scavenger tailings, were proportionally recombined and subjected to basic cyanide leach testing. As the flotation results in some cases have not been optimized at this point, the proportion of values (Au/Ag) reporting to the tailing stream (cyanide leach feed) was higher than ought to be anticipated in practice in those cases.

On average, cyanidation of the combined tailings contributed an additional 8.3% gold and 14.1% silver to the overall circuit gold and silver recoveries. The range in overall tailings gold recovery contribution by cyanidation was from 1% to approximately 32%. The range in silver recovery contribution was from 7.5% to approximately 28%.

It is anticipated that these overall recovery contributions would be limited somewhat as the flotation conditions were optimized (i.e., much of the values leached in the cyanidation circuit will report to the cleaner flotation concentrate in an optimized flotation flow sheet).

18.2.6 SULFIDE MINING

Due to the configuration of the sulfide resource, and the consistent characteristics of the contained gold and silver, the most economic means to extract the sulfide resource appears to be by open pit methods. The study envisions mining and processing the oxide ore reserve and the sulfide Measured and Indicated Resource simultaneously.

The scoping study indicates that the Hycroft sulfide resource is large with a low strip ratio. By incorporating these factors, the study shows that a large scale, bulk tonnage, open pit mining operation is the most economic method to extract the sulfide resource.

The study indicates that a mining rate of approximately 136 million tons per year of combined ore and waste would be associated with mining the sulfide resource. Annually, the mill will process 35 million tons of ore while the ROM heap leach will process 26 million tons.

The study envisions that mining would be conducted via a typical truck and shovel mining operation. Ore and waste would be drilled on 50 ft high mining benches. The material would be blasted and loaded by 45 yd^3 electric mining shovels into 320 and 200 ton haul trucks. The ore would be transported by truck to either the oxide (heap leach) processing facility or to the sulfide/oxide milling facility. Waste would be transported by truck to the waste storage facilities. Future engineering work will evaluate the economic potential of the use of conveyor transport of ore and waste.

18.2.7 PROCESSING SULFIDES

The scoping study indicates that a 100 K tpd milling and flotation process facility is preferred. Ore would be crushed by a large gyratory crusher and transported via conveyor to a large diameter semi-autogenic grinding ("SAG") mill. The SAG mill discharge would be fed to two ball mills. The ore will be ground and pumped to the flotation circuit. On average, approximately 5,000 tons of concentrate per day would be produced. Each ton of concentrate would contain approximately 0.279 ounces of gold and 15.561 ounces of silver. This material would be oxidized using pressure oxidation or roasting to remove the sulfides in order to be able to leach the gold and silver from the concentrate. A doré containing gold and silver would be produced at this facility. This doré would be transported to a refinery for production of gold and silver bullion.

Oxide ores would be processed using the heap leach and Merrill-Crowe processing methods. See Section 16 of this NI 43-101 report for more details. The scoping study indicates that processing higher grade oxide ores concurrently with sulfide is preferred.

Please see Figure 18.3 for the mill option layout.

18.2.7.1 Environmental Permitting

A project of the size and magnitude described in the scoping study will require extensive federal and state environmental permits in order to be constructed and operated. The project will require a site wide EIS.

Preliminary work required for the EIS, such as base line water sampling, archaeology surveys, etc. began late in 2009. The scoping study assumes receipt of all required state and federal permits to construct and operate the sulfide mine and plant in three years from the submittal of the applications.

18.2.7.2 Production Plan

Sulfide mining (not including the heap leach material) is expected to produce an average of 487,000 ounces of gold and 26,500,000 ounces of silver per year over the life of the project. As the accelerated mining alternative continues, heap leach production is expected to average over 123,000 ounces of gold and nearly 500,000 ounces of silver annually. Allied Nevada expects that these operations will run concurrently as drilling and metallurgical work is being completed to potentially expand the overall oxide reserve. While running in parallel, Allied Nevada envisions that the combined operations could produce approximately 610,000 ounces of gold and 27,000,000 ounces of silver per year. Allied Nevada is addressing the potential that sulfide inferred resources can be upgraded through further drilling and metallurgical work.

Allied Nevada is also reviewing the possibility of improving gold and silver recoveries from oxide mineralization utilizing a grinding and milling circuit. The positive results of metallurgical testing may indicate that transitioning to a full milling scenario sufficient to process oxide and sulfide ore may be a more optimal plan than milling sulfides only.

18.2.7.3 Operating Cost Estimate

Industry operating cost data for similar sized mining, milling, and flotation processes located in North America was used to develop the operating costs used in the Mill Scoping Study. See Table 18.2. Operating costs shown for the oxide ore reserve were obtained from the Hycroft Oxide Feasibility Study. See Table 18.1.

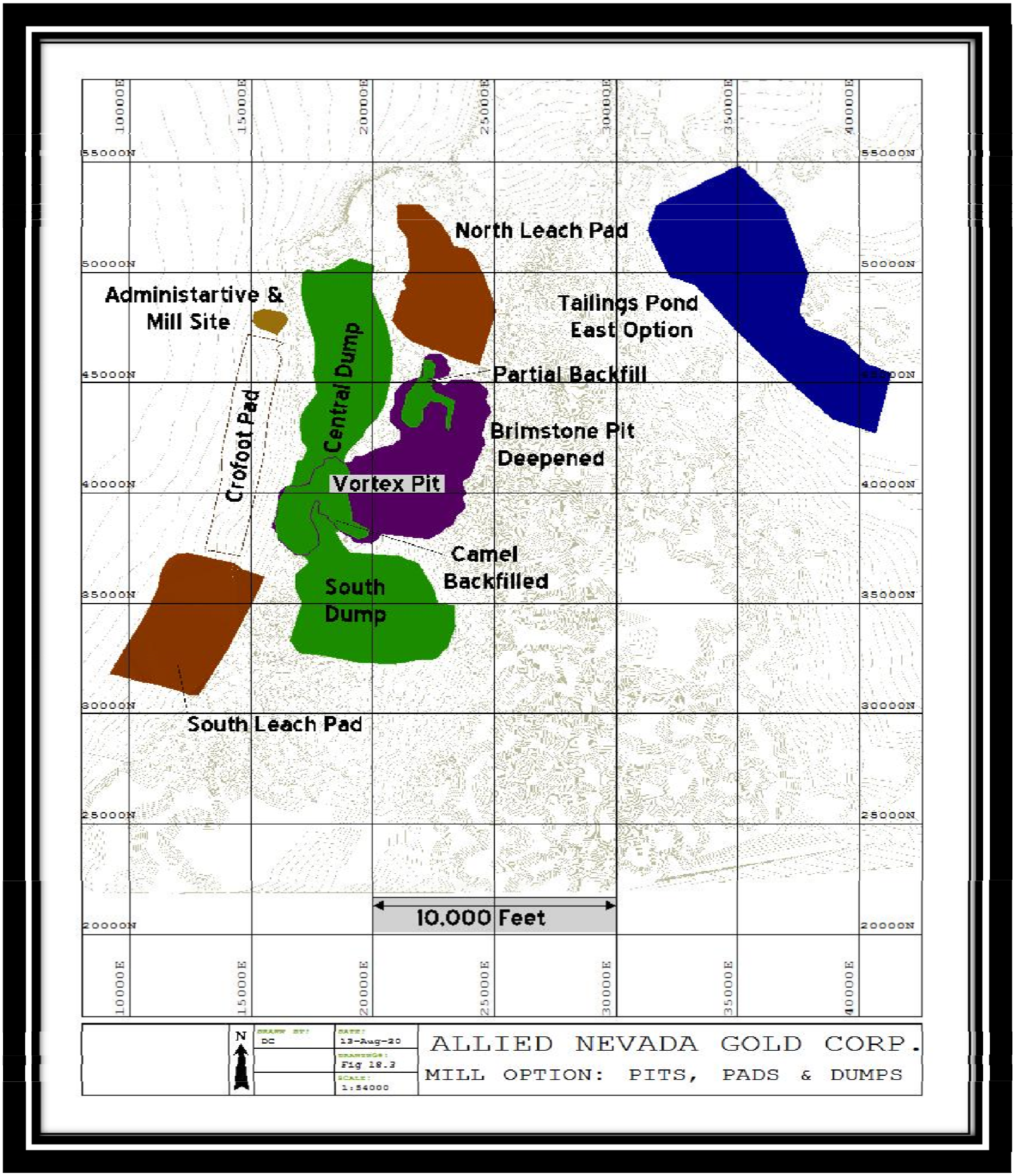
Table 18.5 shows metal price sensitivity for the milling operation envisioned and detailed in Table 18.2 and described in the preceding pages.

Table 18.5 Metal Price Sensitivity

Metal Prices	NPV @ 0%	NPV @ 6%	IRR	CASH COST
$800 Au/$14 Ag	$1,782 million	$961 million	24.21%	$348.79
$1000 Au/$16 Ag	$3,756 million	$2,238 million	42.02%	$273.67
$1200 Au/$18 Ag	$5,723 million	$3,513 million	57.19%	$199.51
$1400 Au/$20 Ag	$7,690 million	$4,788 million	70.94%	$125.35

Note: Cash Cost include silver as a credit to costs.

Figure 18.3 Mill Option Layout



18.2.7.4 Capital Cost Estimate

Recent North American industry average capital costs were utilized to develop the capital cost used in the scoping study. Capital costs for the oxide mine were obtained from the oxide feasibility study.

18.2.7.5 Financials

The scoping study indicates that the sulfide project is economically viable. Utilizing the operating and capital costs developed by the study, as well as the metallurgical performance provided by current metallurgical test work, a gold price of $800/oz and a silver price of $14.00/oz, the project generates a discounted NPV of $961 million at 6% and pays back initial capital in 2.9 years.

The scoping study shows that the economics of the sulfide project are most sensitive to metal price and metal recovery.

18.3 CONCLUSIONS AND RECOMMENDATIONS

The favorable result of the Hycroft mill scoping study warrants the continuation of work and expenditures on the project. Allied Nevada estimates that a feasibility study related to the sulfide Measured and Indicated Resource can be completed in the first half of 2011. The estimated total cost to complete the feasibility study is $50 million. Allied Nevada has spent approximately $25 million to date working on the sulfide feasibility study and it is expected the remaining $25 million will be spent over the next 12 to 18 months. The remaining funds will be spent on further exploration and delineation drilling, metallurgical test work, hydrology, pit and dump slope stability, environmental permitting activities, facility design, definitive operating and capital cost estimates, and other ancillary costs associated with the completion of a feasibility study.

Allied Nevada envisions that the oxide and sulfide mineralization would be mined concurrently. As such, metallurgical work is being conducted to assess the benefit of milling oxide material with the intention of improving gold and silver recoveries. Should the metallurgical testing result in a positive outcome, management believes reviewing the addition of a milling circuit to accommodate oxide and sulfide material may be warranted. It is expected that, based on North American industry average costs, the capital costs to build an 100 Ktpd mill is expected to be approximately $775 million. Operating costs are expected to be similar to those presented in the mill scoping study case. Further, for sulfide mineralization, eliminating the costs to produce, transport, and oxidize the concentrate with an onsite facility would reduce operating costs partially offset by the costs associated with a leaching circuit.

19 INTERPRETATIONS AND CONCLUSIONS

SEWC reviewed pertinent data from the Hycroft mine regarding exploration data and methods, resource estimates, metallurgy, process performance, and reserve estimates. SEWC determined that Allied Nevada's statement of mineral resources and mineral reserves at the Hycroft mine are in accordance with Canadian National Instrument 43-101, as set forth in the CIM Standards on Resources and Reserves, Definitions and Guidelines (2005). SEWC completed its review of the project in preparation for this Technical Report. SEWC met its objective and concludes:

- Assaying, density measurements, and drill hole surveys have been carried out in accordance with best industry standard practices and are suitable to support resource estimates.
- Sampling and assaying includes quality assurance procedures, including submission of blanks, reference materials, pulp duplicates and coarse reject duplicates, and execution of check assays by a second laboratory.
- The Hycroft gold and silver deposit resource models were developed using industry accepted methods.
- Mine designs have been developed using industry standard practices and appropriate design criteria.
- Proven and Probable Mineral Reserves are developed from Measured and Indicated Resources with appropriate application of cost and design criteria.
- Mineral resources are classified as Measured and Indicated Mineral Resources and as Inferred Mineral Resources. Resource classification criteria are appropriate in terms of the confidence in grade estimates and geological continuity and meet the requirements of National Instrument 43-101 and CIM Standards on Resources and Reserves, Definitions and Guidelines (2005).
- Metallurgical studies have been carried out on a sufficient number and sufficiently representative suite of samples to estimate gold recovery for oxide leach material. Estimated recoveries are based on historic production. Similar materials will be processed as in the past and SEWC believes this is the best estimation of future recoveries.
- Historically, the heap leach and Merrill-Crowe facilities at Hycroft have performed as designed.
- SEWC has validated Allied Nevada's Mineral Resource and Mineral Reserve Statements.

20 RECOMMENDATIONS

A significant resource, of both oxide and sulfide mineralization, has been identified at Hycroft. In addition, the continuing Vortex drill program has confirmed the higher grade nature of the zone. Drill plans have been designed to: (a) improve confidence in the estimate of gold and silver grade of the inferred resource, and (b) expand and conjoin the extent of the overall known resource. Material from core holes will be utilized for metallurgical test work. Eight drills (four core and four rotary) are currently operating and devoted to exploration. Under an approved 2010 exploration budget, these programs are ongoing. Due to the increase in oxide ore reserves, it would be prudent for Allied Nevada to review a mining plan which may extract those ounces at an accelerated rate. The author has made no recommendations for successive phases.

20.1 ANV OXIDE RESOURCE DEVELOPMENT PLAN

Planned oxide resource development drilling in 2010 will be directed towards the Albert and Cut-4 zones. The 2010 program includes:

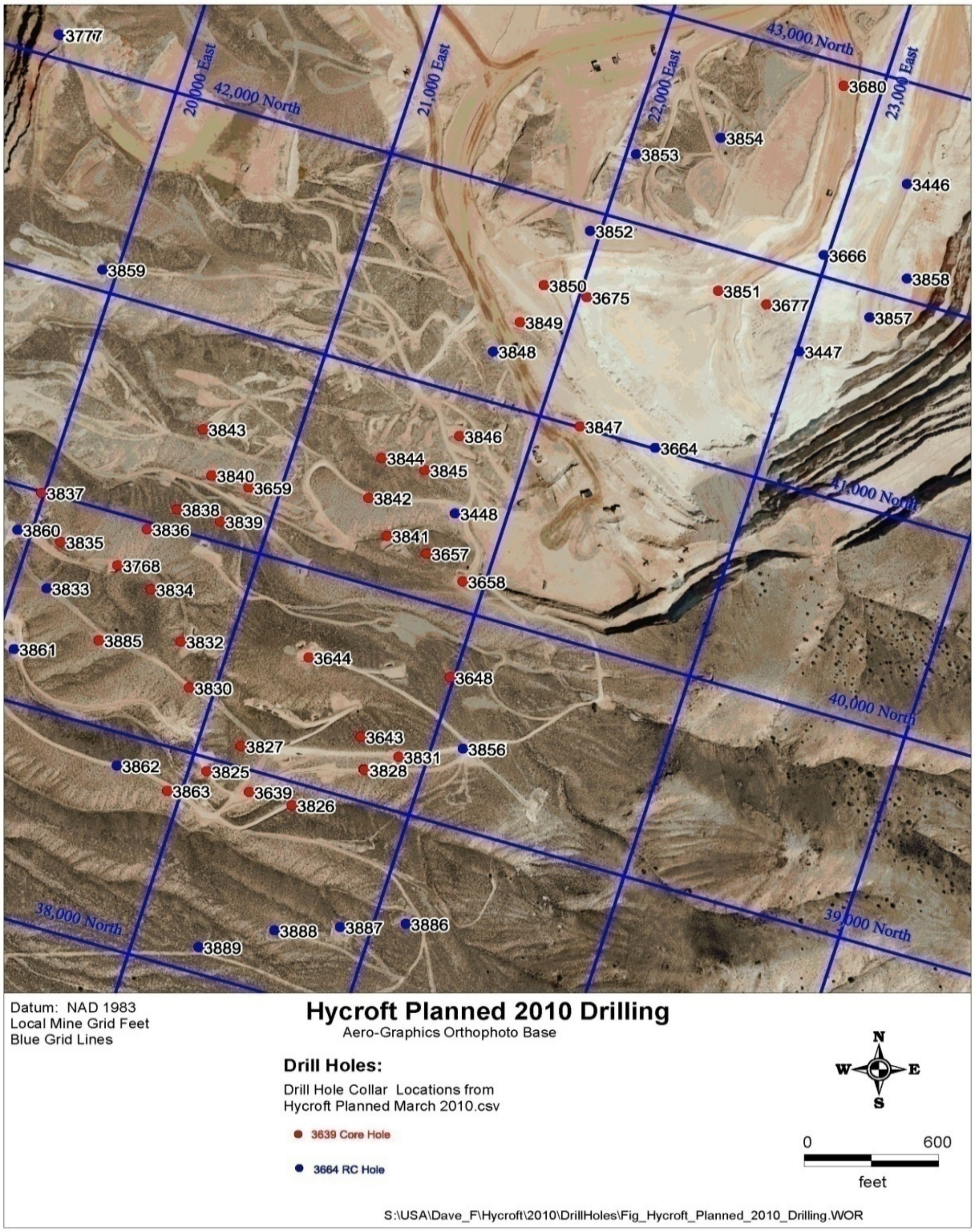
- 100 rotary holes.
- 17 core.
- $10 million in planned spending.
- Drilling to commence immediately and be complete by 2010 year end.

20.2 ANV SULFIDE RESOURCE DEVELOPMENT PLAN

Further, planned sulfide resource development drilling in 2010 will be directed toward exploration of the Albert Zone and expansion of the Vortex Zone (Figure 20.1). Significant additional core drilling to provide material for ongoing metallurgical assessment will also be conducted. The 2010 program includes:

- 32 rotary holes.
- 50 core.
- $10 million in planned spending.
- Drilling to commence immediately and to be complete within 9 months.

Figure 20.1 ANV Phase 2 Sulfide and Vortex Development Plan



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22 DATE

The effective date of this report is September 23, 2010.

23 ADDITIONAL REQUIREMENTS FOR DEVELOPING OR PRODUCING PROPERTIES

The Hycroft cash flow models are based on forecast and actual costs. All costs, volumes, and grades are reported in US Imperial units. Section 23 is based on Proven and Probable Mineral Resources only.

23.1 OPEN PIT MINING OPERATIONS

All mining from currently scheduled ore reserves is conducted by open pit methods. The reserves will be mined from: the Brimstone, Boneyard, Bay Area, Central Fault, Silver Camel and Cut-5 pits. The following pit design criteria are used:

- 25 to 30 ft bench height on primary stripping benches and final walls.
- Overall wall slope angles of approximately 40 to 46°, depending on wall orientation or geology.
- Bench face angles are 60°.
- 10% haul road grade, with bottom of ramps at 15%.
- 60 to 120 ft wide haul roads.
- Minimum mining width of 200 ft, but narrower widths are mined over short distances when unavoidable.

The currently excavated Brimstone pit is being successfully mined and has remained intact with these parameters; however, there is an ongoing geotechnical study to optimize the mining criteria.

Mining is accomplished with a typical drill, blast, load, and haul cycle. All mine material is blasted with the low grade ore being hauled as ROM to the heap leach pads and high grade ore crushed prior to being placed on the pad. The waste is hauled and placed directly in permanent waste dumps, some of which will backfill previously mined pits.

Ore and waste is segregated based on modeling of blast hole assays. A production block model is built using inverse distance numerical modeling of the blast hole data. The ore and waste zones are flagged in the field to provide visual guidance to the production crews and equipment operators. SEWC recommends the use of blast hole models for use in production control. The ore control at Hycroft is performed using industry accepted standards of production geology.

Short term mine plans consist of rolling three month plans that are updated on a monthly basis. Life-of-mine plans were scheduled by month through 2010 and 2011 and quarterly, thereafter. The plans include scheduling of major equipment to make sure that forecast ounce flows are viable.

Mining operations are conducted 24 hours per day, seven days per week. ROM leach grade ore is placed directly on heap leach pads and approximately 10% of the ore mined will be crushed to -3/8” in order to enhance recovery. Waste is dumped on permitted dump locations and, in some cases mined pits will be filled with waste.

Mining phases for Hycroft are shown in Figure 23.1. The Hycroft pits, dumps, pads, pits, and site facilities are shown in Figure 23.2.

Figure 23.1 Hycroft Mining Phases

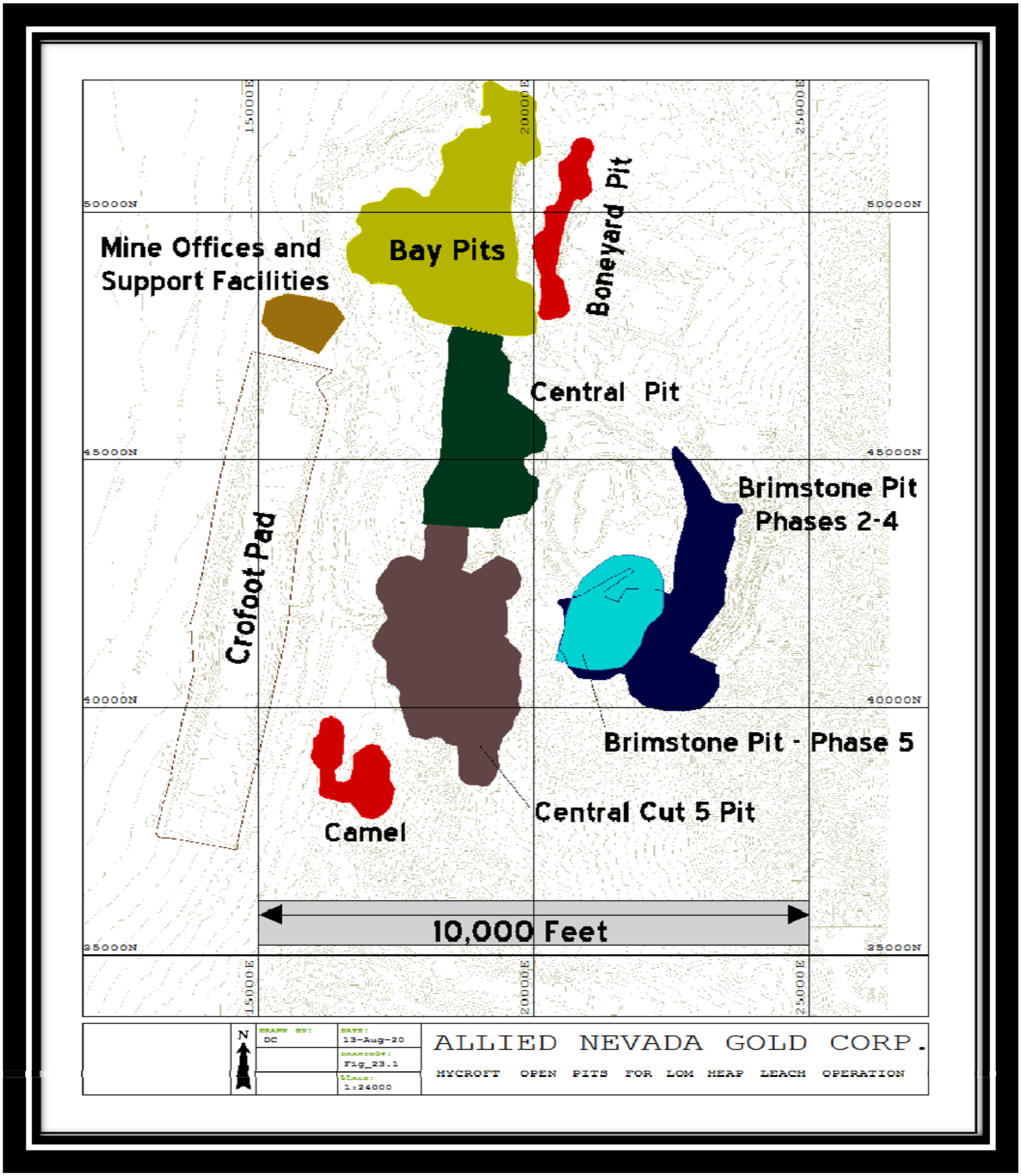
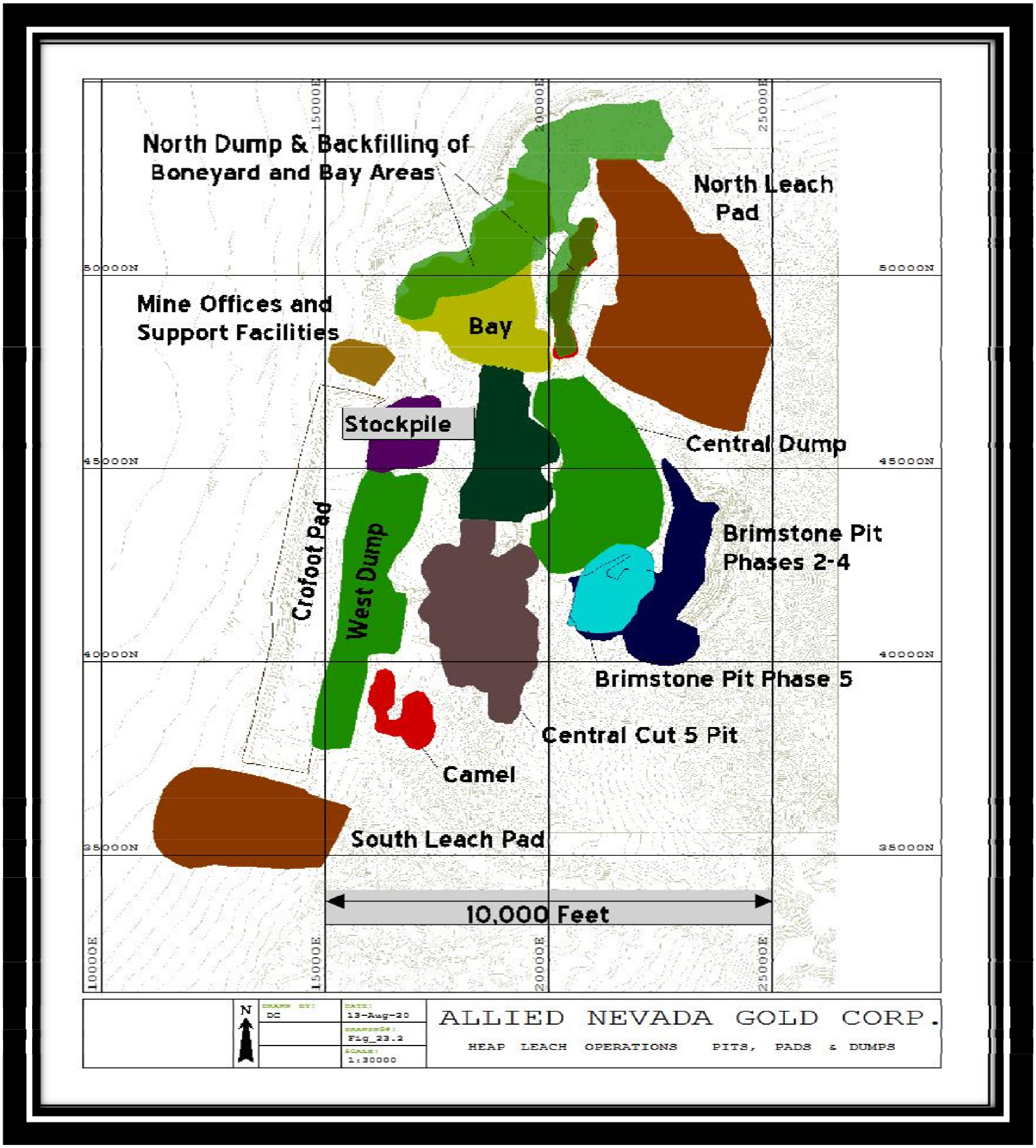


Figure 23.2 Hycroft Pit Sites and Facilities



23.2 MINING FLEET

Allied Nevada is mining with primarily a Komatsu mining fleet, see Table 23.22.

For the approved mining case, the mining fleet will be expanded using 240 to 320 ton class haul trucks, 28-35 yard class loaders, and additional DML drills and support equipment to optimize development of the oxide ore reserve. Total material moved is increasing from approximately 28 million tons per year to 79 million tons per year over an 18 month period and then peaking at 85 million tons per year in 2013 and 2014.

23.3 PROCESSING AND RECOVERIES

Mined ore will be placed on the Phase I and Phase II expansion of the Brimstone leach pad. In addition to the existing 2 million ft^2, 4.6 million ft^2 of the Phase II pad has been constructed. An additional 2.5 million ft^2 is scheduled for construction in the second half of 2010 to accommodate the Brimstone and Cut-5 ore. Brimstone acid leach ores are presently being mined and placed on the pad using a ROM and crushed ore heap leach procedures. This material will continue through the end of 2010. More competent siliceous material will be mined mid-year 2011 which will be processed through a crusher and placed on the leach pad using trucks and/or conveyors and stackers.

The site currently has the capacity to pump 6,000 gpm to the pad, which is expected to be expanded to 8,000 gpm with the 2010 pad construction. Ponds and pumping stations were modified to increase efficiency by being able to direct solutions to the various pad cells and then to either the pregnant pond for processing or to the lean pond to be used for solution grade enhancement. Solution returns from the pad are processed in the Merrill-Crowe plant at a maximum rate of 3,500 gpm, and through two CIC trains with a capacity of approximately 1,400 gpm.

In 2009, a new refinery was constructed in close proximity to the Merrill-Crowe facility. Doré from the Merrill-Crowe plant is produced onsite. The refinery meets Nevada emission standards for mercury. Loaded carbon from the CIC process is currently being processed into doré by an off-site plant and then directly shipped to an independent refinery for final refining.

The Brimstone process facility is expected to be expanded to accommodate expansion of the Brimstone pad and the new south pad system. Solutions from both of the new pads will report to their respective reclaim ponds and be pumped to the Brimstone facility.

Expected recoveries (based on past performance) for the Brimstone ROM ore is 42.1% over the first 90 days, 52.0% recovered in the first 180 days, and overall 56.6%.

The expanded oxide operations are expected to achieve annual production of more than 260,000 ounces of gold per year by 2012, with peak production averaging 300,000 ounces in 2013 and 2014. Average annual silver production is expected to be in excess of 1.1 million ounces in 2014 and 2015.

23.3.1 RECOVERY

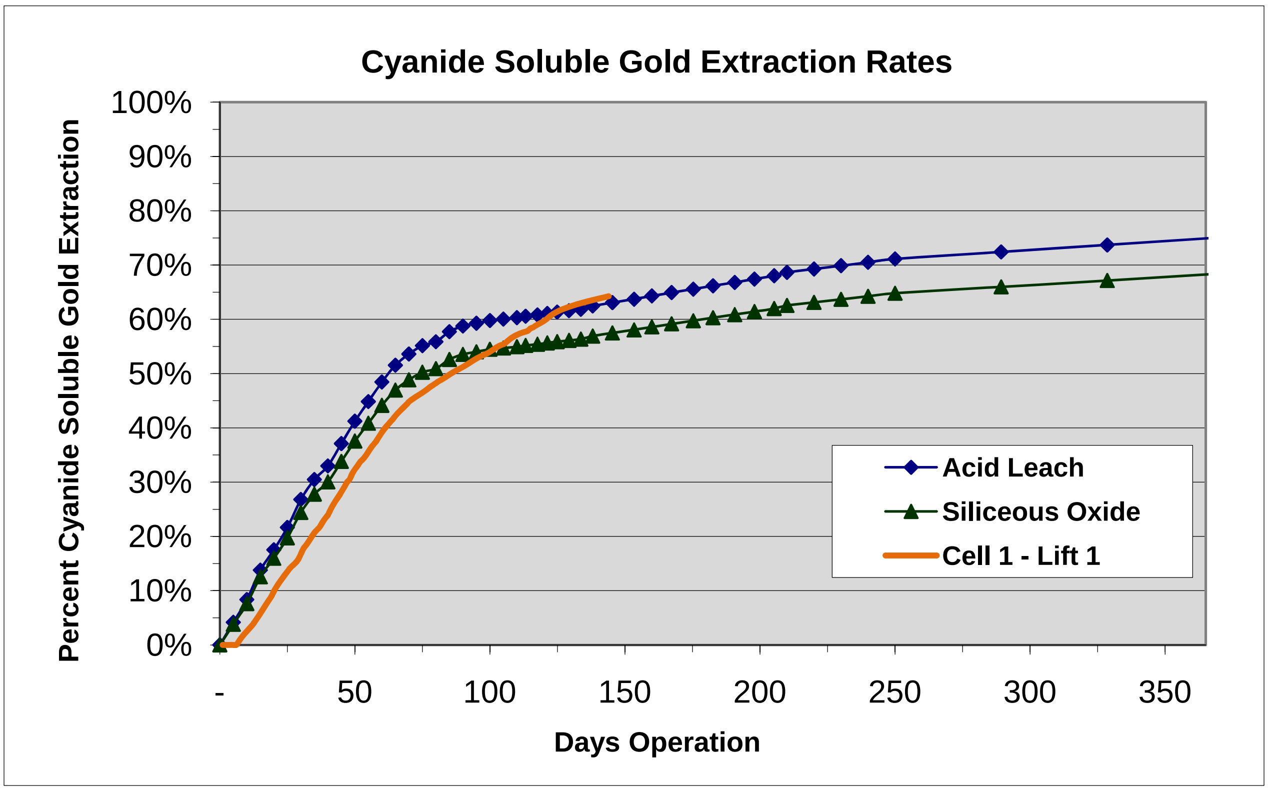
Actual final gold recovery from pad 4 for all previous operations was 79.5% of the cyanide soluble content (pad 4, historic results). Considering all the information available, the projected gold recovery of 60% of total gold content represents a realistic estimate for the ores included in the ore reserve. Historic production figures for Brimstone pads 4 and 5 are shown in Table 23.1.

Table 23.1 Production Pad Loading and Recoveries

Pad	Tons of Ore	Gold Loaded oz	CN Sol Grade opt Au	Recovery Gold oz	Actual % Recovery
4	11,129,940	159,206	0.014	126,622	79.5
5	4,334,017	61,991	0.014	49,348	79.6

Figure 23.3 shows the actual historic recoveries of gold from the Brimstone pad compared with the gold recoveries from material placed on Phase II, Cell 1, first lift in 2009. Gold recovery curves are similar. Cell 1, first lift was shut down after about 140 days to allow the second lift to be placed. The gold extraction curve of Cell 1, first lift is expected to follow the past recovery curves. Silver assays from the Phase II, Cell 1, first lift material are 0.264 opt Au total by FA and 0.200 opt by CNAA. Silver recoveries are calculated to be 14.8% based on the CNAA and 11.8% based on the FA.

Figure 23.3 Hycroft Leach Pad Recovery (Cyanide Soluble) Comparisons



23.4 METALLURGICAL TEST WORK

Several metallurgical studies have been undertaken on the Hycroft ore. These studies are briefly discussed in this section. In 1994, a metallurgical program was initiated at the Hycroft mine to evaluate the gold recovery that could be expected from ROM leaching of the Brimstone ore body. It was apparent, at the start of the Brimstone evaluation, that two basic ore types existed which were classified at the time as “silicified breccias” and “acid leach.” The acid leach material, which generally forms the upper part of the Brimstone deposit, is fine and friable, whereas the silicified breccias are significantly more competent. During the initial testing of the Brimstone ores, relatively good bulk samples of acid leach material were available for column and heap leaching tests, while a limited quantity of silicified breccias core samples were available for testing. As a result, good confidence in the recoveries from acid leach material was obtained through test work, while additional testing needed to be undertaken to improve the confidence in the expected recovery from the silicified breccias material.

When mining started at Brimstone at the north end of the deposit, ore was trucked to pad 4, which was constructed solely for Brimstone ore, and to pad 5, which was Brimstone ore placed on top of the old Crofoot pad 1. As a result of this placement of ore, recovery from pad 5 could be biased by some residual leaching from pad 1 below it. Pad 4, on the other hand, was exclusively used for Brimstone ore and, therefore, gold production from this pad accurately reflects actual gold recovery achieved from Brimstone ore placed on pad 4. Ore placed on pad 4 was predominately acid leach material but did include approximately 27% of siliceous oxide (previously called silicified breccias) ore.

Due to sustained low gold prices, mining in the Brimstone pit was halted in December 1998 and no further metallurgical test work was done at that time. It was apparent, however, that significantly more gold had been placed on the Brimstone heap than was reported in the mine model, so a detailed study of the Brimstone ore body mined to date and future reserves was undertaken. During the course of this study, all the existing drill hole data was re-logged, and, together with pit mapping and blast hole data, the geology of the Brimstone deposit was reinterpreted resulting in a much better understanding of the relationship between the ore material types and metallurgical response. While there remains two predominant ore types, they are now referred to as “acid leach” and “siliceous oxide,” instead of “acid leach” and “silicified breccias,” and there is only one potential subset that has any significance – clay bearing oxide. In light of this additional information, the samples used for all previous metallurgical work were re-reviewed to see which ore type they represented. In addition, areas in the pit where specific ore types are now exposed were identified and new samples were collected for additional test work.

23.4.1 HISTORIC TEST WORK

A significant amount of test work was completed in 1994, prior to making the decision to proceed with the development of the Brimstone deposit. This work included bottle roll tests, barrel tests, column tests, and two test heaps. The majority of the work focused on acid leach material, which was more readily available and led to the conclusion that at least 75% recovery of cyanide soluble gold was achievable from acid leach ore.

Four column/barrel tests were run at a 3” rock size on material designated “transition oxide” material and “silicified oxide” material. The composition of ore samples which were used for these tests was reviewed to determine whether or not the columns can be considered representative under the new definition of oxide ore. The conclusion is that the samples were representative. Gold recoveries achieved from these column/barrel tests are as shown in Table 23.2.

Table 23.2 Column/Barrel Test Results on "Transition Oxide" and "Silicified Oxide" Ore

Test Number	CN-Soluble Gold Recovery (%)	Fire Assay Gold Recovery (%)
94-13A	72.7	61.9
94-13B	77.6	69.7
94-13C	65.3	52.2
94-13D	74.7	65.9

The first recovery figure is based on cyanide soluble gold assays while the second figure is based on fire assays. The average cyanide soluble gold recovery for these tests was 72.6%, but if the lowest recovery test is rejected, the average gold recovery is 75%.

The results of the tests on acid leach and oxide ores were the basis for proceeding in production. The actual results of production for the ROM pads demonstrated significantly higher recoveries over time.

Various ore bodies at Hycroft were mined, crushed to a nominal 3/8", and placed on the Crofoot leach pad. These ore bodies included the Cut-5, Bay Area, Boneyard, Central Fault, and Brimstone materials. A monthly composite of the crushed materials was column leached to predict gold recoveries. An evaluation of column test work from the monthly crushed composites, shown in Table 23.3, indicate average CNAA gold recoveries ranging from 71.7% to 81.5%.

Table 23.3 Yearly Average Crusher Column Results for Oxide Ore

Year Average	Au CNAA opt	Ag CNAA opt	Au CNAA Rec. %	Ag CNAA Rec. %
1989	0.018	ND	71.7	ND
1990	0.019	0.069	73.6	35.1
1991	0.020	0.141	78.7	34.9
1992	ND	ND	ND	ND
1993	0.018	0.133	81.5	33.8
1994	0.017	0.044	75.4	26.3
1995	0.014	0.053	80.7	22.8
1996	0.015	0.096	78.5	28.4

ND – No Data Available

23.4.2 TEST WORK YEAR 2000

A test program was initiated to better understand the metallurgical response of ore types that would be encountered in future mining. The tests included column testing of core samples and drum testing of bulk samples collected from the pit. The results are tabulated in Table 23.4.

Table 23.4 Column Leach Results for Oxide Ore

Sample	Material	Current Gold Extraction		90 Day Projected Gold Extraction (%)	R^2
		% CN-Sol Au	% FA Au		
4636	Clay Bearing Oxide	83.2	76.9	90.5	0.99
4434	Clay Bearing Oxide	77.5	69.9	86.7	0.99
4400	Clay Bearing Oxide	79.6	72.4	84.0	0.99
Core 1	Silicified Oxide	61.7	50.5	70.3	0.99
Core 2	Silicified Oxide	64.3	55.7	70.4	0.99
Core 3	Silicified Oxide	70.4	60.4	77.0	0.99

The results of column tests core 1, 2 and 3, which employed samples taken from intact core are not representative of ROM material, showed similar results to previous tests. The drum tests were more representative, based on test work carried out on bulk samples taken from the blasted ore in the pit, with a more appropriate size distribution. The 90 day projected recoveries for three drum tests varied from 84.0% to 90.5%. The drum samples were, however, a little higher grade than the grade of the future reserves. The lower or average grade ores will probably not achieve quite as high a recovery. However, in a production situation, the placed ore is leached for much longer than 90 days, which would tend to recover more gold. An important point to note is that the drum test results and subsequent tailings analysis indicated that future ores will yield similar metallurgical performance to previously mined ore.

Management is currently reviewing the potential benefit of milling all or a portion of the oxide material at Hycroft to improve recoveries of both gold and silver.

23.5 PREVIOUSLY MINED ORE COMPARED TO REMAINING MINERAL RESERVES

23.5.1 BRIMSTONE

An indication of future metallurgical performance is to compare the cyanide soluble data of samples representative of the ore obtained during previous mining of the Brimstone ore with samples representative of the remaining Brimstone reserves. A detailed comparison of the cyanide soluble data for samples of the south Brimstone drill intercepts and north Brimstone drill intercepts was completed. South Brimstone is typical of previously mined Brimstone ores and north Brimstone is representative of future Brimstone reserves. The results of these comparisons are in Tables 23.5 and 23.6.

Table 23.5 South Brimstone Drill Intercepts

Ore Type	Feet Included	% of Total Feet	CN Sol Au (opt)	% CN Sol Recovery
Siliceous	13,868	45.70%	0.015	73.50%
Acid Leach	12,674	41.80%	0.019	76.70%
Clay bearing	3,165	10.40%	0.024	80.20%
Other	613	2%	0.014	85.90%
Total Average			**0.018**	**76.00%**

Table 23.6 North Brimstone Drill Intercepts

Ore Type	Feet Included	% of Total Feet	CN Sol Au (g/t)	% CN Sol Recovery
Siliceous	11,352	35.70%	0.014	75.30%
Acid Leach	18,722	58.80%	0.016	75.80%
Clay bearing	1,486	4.70%	0.014	79.50%
Other	259	2.00%	0.009	59.50%
Total Average			**0.015**	**75.70%**

These results indicate that there is virtually no significant difference between the overall percentage of cyanide gold recovery for the north and south portions of the Brimstone pit. The average percentage of cyanide soluble gold in both sample sets, within experimental limits of sampling, is identical; 76% vs. 75.7%. The conclusion to be drawn from the cyanide soluble comparison, the production data, and completed test work, is that all described ore types, within the error of quantifiable results, are metallurgically identical in a ROM situation.

Bottle and column test work on Brimstone deposit core samples, shown in Tables 23.7 and 23.8, were conducted to indicate the effect of particle size on leach recovery. Results indicate a particle size effect similar to historical information. Column cyanidation gold leach results were 57% from -3/4” and 71% from -3/8” materials.

Table 23.7 Brimstone Bottle Cyanidation Leach Results

Deposit	Interval	SB	10M	3/4"
		Au %	Au %	Au %
Brimstone	3694 137.5-155.5	81.5	73.1	41.9
Brimstone	3695 105.5-113.2	100	100	33.3

SB = Shatterbox Material
10M = 10 mesh Material

Table 23.8 Brimstone Column Cyanidation Leach Results

Deposit	Fire Assay Recovery 80% Passing 0.75”		Fire Assay Recovery 80% Passing	
	Au %	Ag %	Au %	Ag %
Brimstone	57.1	23.3	71.4	30.7

Column test works conducted on siliceous Brimstone material, shown in Table 23.9, indicate total gold recoveries of 71 to 75% from ROM, -2.0”, and -0.5” materials. Silver recoveries, based on fire assay, ranging from 17.7% with ROM material to 27.7% from material crushed to passing 0.5”.

Table 23.9 Siliceous Brimstone Column Test Results

Ore type	Crush Size	% Fire Assay Au Rec.	% Fire Assay Ag Rec.
Siliceous	ROM	70.6	17.7
Siliceous	-2.0”	75.0	21.2
Siliceous	- 0.5”	73.3	27.7

As shown in Table 23.10, column cyanidation test work on Brimstone acid leached materials containing low and high sulfur contents (3.9% vs. 7.4%) indicate gold recovery decreased from 89.1% with the lower sulfur to 80.0% with the higher sulfur contents.

Table 23.10 Acid Leach Brimstone Column Test Results

Ore type	Crush Size	% Fire Assay Au Rec.	% Fire Assay Ag Rec.
Low S Content	As Rec	89.1	8.3
High S Content Average	As Rec	80.0	6.7

23.5.2 CUT-5

Bottle and column test work on Cut-5 deposit core samples, shown in Tables 23.11 and 23.12, show the effect of particle size on leach recovery. This data indicates gold recovery is particle size dependent. Bottle test recoveries ranged from 27 to 71% from -10 mesh material and from 9 to 50% from -3/4” material. Column test gold recoveries were 37 and 44% from -3/4” and -3/8” materials, respectively.

Table 23.11 Cut-5 Bottle Cyanidation Leach Results

Deposit	Interval	SB	10M	¾”
		Au %	Au %	Au %
Cut-5	3686 281-311.5	85.7	71.4	50.0
Cut-5	3686 311.5-340	80.0	60.0	37.5
Cut-5	3691 141.5-158	50.0	50.0	28.6
Cut-5	3691 158-174.5	60.0	27.3	9.1
Cut-5	3693 115-130,1725-180	87.5	62.5	42.9
Cut-5	3693 52-60,70-80,90-95	66.7	41.7	23.1
Cut-5	3693 60-70,85-90,95-105	66.7	45.5	21.4

SB = Shatterbox Material
10M = Mesh Material

Table 23.12 Cut-5 Column Cyanidation Leach Results

Deposit	Fire Assay Recovery 80% Passing 0.75”		Fire Assay Recovery 80% Passing 0.375”	
	Au %	Ag %	Au %	Ag %
Cut-5	37.5	9.1	44.4	22.7

23.5.3 BONEYARD

Bottle and column test work on Boneyard deposit core samples are shown in Tables 23.13 and 23.14. This data indicates gold recoveries were particle size dependent with bottle test gold recoveries ranging from 25 to 83% from -3/4" material, and column gold recoveries of 50 to 78% from -3/8" material and 43 to 80% from -34" material. Historic data also indicates the Boneyard material was generally crushed to a nominal 3/8" particle size prior to placement on the leach pad.

Table 23.13 Boneyard Bottle Cyanidation Leach Results

Deposit	Interval	SB	10M	¾"
		Au %	Au %	Au %
Boneyard	3710 0-20	87.5	75.0	55.6
Boneyard	3710 20-40	92.3	76.9	80.0
Boneyard	3710 40-60	100	85.7	83.3
Boneyard	3710 60-79.5	88.9	72.2	64.7
Boneyard	3710 79.5-112	86.4	56.5	41.2
Boneyard	3710 112-134.5	87.5	50.0	25.0

SB = Shatterbox Material
10M = 10 mesh Material

Table 23.14 Boneyard Column Cyanidation Leach Results

Deposit	Fire Assay Recovery 80% Passing 0.75"		Fire Assay Recovery 80% Passing	
	Au %	Ag %	Au %	Ag %
Boneyard #1	80.0	0	77.8	0
Boneyard #2	42.9	6.3	50.0	11.8

23.5.4 BAY AREA

Bottle and column test work on Bay Area deposit core samples are shown in Tables 23.15 and 23.16. This data confirms historic information indicating gold recoveries were very particle size dependent. Bottle cyanidation results (Table 23.15) indicate gold recoveries ranged from 17 to 88% from -3/4" materials and from 12 to 67% from -2" materials. Column cyanidation test work resulted in gold recoveries of 58 to 77% from -3/8" materials. Historic data indicates the Bay Area materials were generally crushed to a nominal 3/8" particle size prior to placing on the leach pad.

Table 23.15 Bay Area Bottle Cyanidation Results

Deposit	Interval	SB	10M	¾"	2"
		Au %	Au %	Au %	Au %
Bay Area	3698 65-71-81-85-98-101	88.5	59.3	36.0	31.0
Bay Area	3698 71-81.5	100	70.0	40.7	30.4
Bay Area	3698 87-98	100	100	50.0	43.8
Bay Area	3699 109.5-127	87.5	87.5	87.5	70.0
Bay Area	3699 127-144.5	100	83.3	87.5	66.7
Bay Area	3699 144.5-163	84.6	90.0	60.0	62.5
Bay Area	3699 163-181	96.7	74.2	37.9	22.2
Bay Area	3699 194-215	87.5	66.0	43.1	38.6
Bay Area	3699 215-234	56.5	39.1	19.0	12.0
Bay Area	3699 234-247	76.9	63.6	17.4	27.3
Bay Area	3700 150-170	88.9	58.3	71.4	66.7
Bay Area	3700 170-190	81.8	66.7	50.0	42.9
Bay Area	3703 63-79.5	100.9	77.5	47.1	30.8
Bay Area	3703 86-98	86.7	86.7	69.0	52.6
Bay Area	3705 61-100	82.0	68.3	40.7	27.3
Bay Area	3706 109.5-125.5	62.5	37.5	28.6	21.4
Bay Area	3706 125.5-140.5	60.7	44.4	24.1	17.1
Bay Area	3706 70.5-109.5	87.3	63.2	43.4	29.1

SB = Shatterbox Material
10M = 10 mesh Material

Table 23.16 Bay Area Column Test Results

Deposit	Fire Assay Recovery 80% Passing 0.75"		Fire Assay Recovery 80% Passing 0.375"	
	Au %	Ag %	Au %	Ag %
Bay Area SA	51.1	12.5	57.5	33.3
Bay Area AS	72.2	0	76.5	0

23.5.5 SUMMARY

The gold and silver recoveries based on fire assays, shown in Table 23.17, are estimated using historical and present metallurgical test work. Recovery estimates for Brimstone, Cut-5, Boneyard, and Bay Area were estimated based on historical and present metallurgical data using a recovery versus log particle crush size plot. Limited data was available from the Camel and Vortex deposits; therefore, gold recoveries from the Cut-5 and Brimstone deposits were used as the nearest ore deposit data, respectively.

Table 23.17 Hycroft Ore Body Recovery Estimate Based on Fire Assay

	3/8" Rec, %		3/4" Rec, %		ROM Rec, %	
	Au	Ag	Au	Ag	Au	Ag
Cut-5	59	20	48	15	42	11
Central Fault	61	9	57	8	52	6
Camel Hill	59	11	48	11	42	11
Boneyard	64	8	50	7	45	5
Bay Area	62	16	57	13	50	11
Brimstone	73	28	68	19	57	12
Vortex	73	28	68	19	57	12

23.6 PERSONNEL AND EQUIPMENT

Table 23.18 lists the currently budgeted personnel for the mine. These employees are responsible for the daily operation of the mine, leach pads, processing facilities, and technical support. Allied Nevada understands that employee retention is vital for the success of the Hycroft mine. Allied Nevada has developed a strategy for attracting and retaining employees. Wages and salaries are competitive with other Northern Nevada mining operations. The Hycroft mine provides employee transportation to and from the mine site.

Table 23.22 shows existing and planned mining equipment for the Hycroft operation.

Table 23.18 Hycroft Mine Personnel

	Number
Mining	
Supervision	8
Operators	111
Maintenance	31
Subtotal Mine Operations	150
Mine Engineering and Geology	
Engineers	3
Geologists	3
Surveyors	4
Subtotal Engineering and Geology	10
Processing	
Supervision	6
Operators	22
Maintenance	9
Assayers and Refiners	22
Subtotal Processing	59
Administration	
General Manager	1
Senior Accountant	1
Accounting Clerks	3
Purchasing	2
Human Resources	3
Warehouse	4
Safety	7
Environmental	3
Information Technology	1
Utility Maintenance	2
Subtotal Administration	27
TOTAL EMPLOYEES	**246**

23.7 ENVIRONMENTAL

The updated disturbance and proposed new disturbance was calculated with the Nevada Standardized Reclamation Cost Estimator (“SRCE”) version 1.1.1, with the Nevada Cost Data File and Hycroft Interim Fluid Management Plan. The project reclamation estimate totals $16,621,357, which has been approved by both the NDEP and BLM. The financial guarantee bond is being held by the BLM and supported by insurance backed and surety bonding.

Hycroft has initiated the permitting to develop the 1.3 million ounces in additional oxide ore reserves, and has included the sulfide resource as reasonably foreseeable future mining activity in the permitting process.

23.8 TAXES AND MARKETS

Cash flows and analysis for the Hycroft mine are developed for this report on a pre-tax basis at a gold selling price of $800. However, there are two taxes that are applicable to the Hycroft mine:

- Income Tax – Allied Nevada has net operating loss carryovers of $68.2 million as of December 31, 2009 for federal income tax purposes. Taxable income generated from mining operations will initially be used to offset the above net operating loss carryovers. Taxable income from mining operations in excess of $68.2 million would be taxed at current statutory rates of 35%.
- Nevada Net Proceeds Tax – The State of Nevada taxes gross proceeds less allowable deductions in the amount of 5% of net proceeds.

23.9 CAPITAL REQUIREMENTS

Allied Nevada issued 11,500,000 shares of common stock through an underwriting agreement in August 2009 which resulted in net proceeds of $91.5 million, and 14,375,000 shares of common stock through an underwriting agreement in April 2008, which resulted in cash proceeds of $74.4 million. In June 2010, 13,500,000 shares of common stock were sold through an underwriting agreement raising $254.3 million. Use of proceeds was the funding of mine startup capital and exploration drilling until the mining operations became cash flow positive and to strengthen the Company’s balance sheet. Undiscounted cash flow is projected to be $284 million over 6 years. Allied Nevada’s contracts for mining and leach pad construction are within industry norms. See Table 23.19 showing a summary of the operating and capital costs, based on proven and probable reserves, as of January 1, 2010.

Table 23.19 Summary of Operating and Capital Costs

Tons of Ore Processed	175,258,972
Grade opt Au	0.014
Gold sold (Ounces)	1,404,133
Total Revenue	$ 1,123,306,239
Revenue Per Ton Processed @ $800 Au	$ 6.41
Cost per Ton Mined	$ 1.07
Processing Cost per Ton	$ 1.37
Crushing Cost per Ton	$ 2.15
General and Administration Cost per Ton Mined	$ 0.144
Mining Cost per Ton of Ore Processed	$ 2.24
Remaining Capital Requirements	$183,084,000
Net Present Value at a 6% discount rate	$198,106,602

23.10 ECONOMIC ANALYSIS AND PRODUCTION FORECAST

The life-of-mine cash flow schedule for the Hycroft property is listed in Table 23.20. The pre-tax NPV at 6% sensitivity analysis for the Hycroft mine is shown in Table 23.21. The table shows that the project is most sensitive to gold price and mining cost. The estimated pre-tax NPV of the project at a 6% discount rate is $198.1 million at a gold selling price of $800.

23.20 Accelerated Case Mine Plan Financials

	Allied Nevada Gold Corp.							
	43-101 Financial Analysis							
	Financial Summary (Accelerated Mining Case)							
	2010	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**Project Summary**
Financial Summary:								
Revenues - gold	80,000,000	104,596,628	214,583,956	225,286,571	246,716,249	189,559,935	62,562,898	1,123,306,239
Operating Costs:								
Mining costs	40,069,960	47,456,293	77,424,738	87,311,417	88,205,429	51,738,518	-	392,206,355
Crushing costs	2,552,753	4,913,526	7,386,470	4,946,471	8,706,709	7,084,946	-	35,590,875
Processing costs	16,707,710	23,815,252	38,505,084	44,176,713	49,109,323	28,897,853	-	201,211,935
Processing costs -leachpad tail	-	-	-	-	-	-	3,128,145	3,128,145
Site administration and general costs	5,174,421	4,000,000	4,000,000	4,000,000	4,000,000	4,000,000	-	25,174,421
Silver as a byproduct credit	(3,978,729)	(3,742,149)	(5,516,660)	(8,716,599)	(12,712,967)	(16,469,025)	(6,559,935)	(57,696,064)
Production royalties	120,000	120,000	2,000,000	2,000,000	2,000,000	520,000	-	6,760,000
Total Cash-Basis Operating Costs	60,646,115	76,562,922	123,799,632	133,718,002	139,308,494	75,772,292	(3,431,790)	609,807,457
Abnormal Stripping adjustment	(11,000,000)	(9,000,000)	(5,000,000)			-	-	(25,000,000)
WIP Adjustment	(5,837,029)	(7,434,506)	1,669,994	(4,004,593)	(3,988,389)	19,682,000	34,923,725	35,011,202
Total Production Costs	43,809,086	60,128,416	120,469,626	129,713,409	135,320,105	95,454,292	31,491,935	619,818,659
Operating Margin	36,190,914	44,468,213	94,114,330	95,573,162	111,396,145	94,105,644	31,070,963	506,919,370
Add back non-cash items:								
Abnormal Stripping adjustment	(11,000,000)	(9,000,000)	(5,000,000)	-	-	-	-	(25,000,000)
WIP Adjustment	(5,837,029)	(7,434,506)	1,669,994	(4,004,593)	(3,988,389)	19,682,000	34,923,725	35,011,202
	(16,837,029)	(16,434,506)	(3,330,006)	(4,004,593)	(3,988,389)	19,682,000	34,923,725	10,011,202
Capital Expenditures:								
Process plant	-	-	(8,600,000)	(8,600,000)	-	-	-	(17,200,000)
Leach pad expansions	(2,190,000)	-	(18,000,000)	(16,000,000)	(4,000,000)	-	-	(40,190,000)
Mobile Mine Equipment	(21,338,000)	(44,421,000)	(62,263,000)	(1,500,000)	-	-	-	(129,522,000)
Other Capital	(5,300,000)	(7,000,000)	(10,900,000)	(600,000)	(600,000)	(600,000)	-	(25,000,000)
	(28,828,000)	(51,421,000)	(99,763,000)	(26,700,000)	(4,600,000)	(600,000)	-	(211,912,000)
Other Cash Inflow (outflows):								
Salvage value on fixed assets							5,000,000	5,000,000
Net Proceeds Tax	(967,694)	(1,401,685)	(4,539,216)	(4,578,428)	(5,370,388)	(5,689,382)	(3,299,734)	(25,846,529)
Reclamation spending								-
	(967,694)	(1,401,685)	(4,539,216)	(4,578,428)	(5,370,388)	(5,689,382)	1,700,266	(20,846,529)
CASH FLOW	(10,441,809)	(24,788,979)	(13,517,892)	60,290,140	97,437,368	107,498,261	67,694,954	284,172,044
Cost per ounce sold	438	460	449	461	439	403	403	441
Net Present value @ 6% discount rate	$198,106,602							

The following table illustrates the resulting NPV at 6% from changes in the price of gold, the mining costs, the processing costs, and the capital expenditures.

Table 23.21 NPV Sensitivity of LOM, Including all of 2010

	20% Decrease (millions)	10% Decrease (millions)	Base Case (millions)	10% Increase (millions)	20% Increase (millions)
Gold Price	30.0	$114.1	$198.1	$282.2	$366.2
Mining costs	258.9	228.2	198.1	168.0	138.0
Processing Costs	228.8	213.4	198.1	182.8	167.5
Capital Expenditures	234.5	216.3	198.1	179.9	161.8

Please note that this table uses a NPV value as an internal rate of return cannot be determined when all of the cash flow figures are positive. The result is infinity which does not provide a meaningful measure of profitability. For the same reasons, a payback period cannot be calculated.

23.11 MINE LIFE DISCUSSION AND EXPLORATION POTENTIAL

The Proven and Probable Mineral Reserves at the Hycroft mine as of June 1, 2010, totals 2.4 million contained ounces of gold at an average grade of 0.014 opt and 38.9 million contained ounces of silver at an average grade of 0.22 opt. This is sufficient for a remaining mine life of 5 years and approximate gold production of 6 years at the current operating rates. The site should continue with the expansion plans and scale up production through the permitting process to achieve optimum production levels indicated in Table 23.20 as soon as possible.

There is excellent potential to extend the mine life of the Hycroft project. SEWC evaluated many parts of the Hycroft project in cross section. There are areas where drilling has a high potential to convert Inferred Mineralization to Measured or Indicated status. These areas include wider spaced drilling in the periphery of the pit, as well as mineralization being “open”, in many cases, due to a lack of exploration drilling.

The Cut-4 and Cut-5 areas have drilling intercepts of higher grade mineralization. Allied Nevada is currently in the process of drilling holes to quantify the potential to add higher grade sulfide resources.

Table 23.22 Existing and Planned Mining Fleet at Hycroft

Quantity	
Primary Equipment	
2	WA1200 Wheel loaders
1	WA900 Wheel loader
1	WA380 Wheel loader
1	EX3500 Hydraulic Shovel
2	D475 Track dozer
2	D375-A Track dozers
1	D65-W Track dozer
1	Driltech D45KS blast hole drills
2	Atlas Copco DML blast hole drills
1	DML Blast Hole Drill
6	730E haul trucks
1	Komatsu 785 Water Truck – 20,000 gal
1	Volvo A40D Articulated Water Truck – 9,000 gal
1	PC2LC excavator
Back Up Equipment	
2	Caterpillar 16G motor graders
1	Caterpillar 994 wheel loader
Purchased Equipment	
2	930E Komatsu haul trucks (Sept 2010)
1	930E Komatsu haul trucks (Nov 2010)
2	930E Komatsu haul trucks (Mar 2011)
2	930E Komatsu haul trucks (Apr 2011)
9	930E Komatsu haul trucks (2012)
1	EX5500 Komatsu Hydraulic Shovel (April 2011)
1	EX5500 Komatsu Hydraulic Shovel (August 2011)

24 AUTHOR'S CERTIFICATE

I, Scott E. Wilson, of Highlands Ranch, Colorado, do hereby certify:

1. I am currently employed as President by Scott E. Wilson Consulting, Inc., 6 Inverness Court East, Suite 110, Englewood, CO 80112.
2. I graduated with a Bachelor of Arts degree in Geology from the California State University, Sacramento in 1989.
3. I am a Certified Professional Geologist and member of the American Institute of Professional Geologists (CPG #10965) and a Registered Member (#4025107) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as either a geologist or an engineer continuously for a total of 21 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 made a personal inspection of the Hycroft mine on June 19, 2009 for one day.
7. I have had prior involvement with Allied Nevada as the author of six prior technical reports regarding the Hycroft mine.
8. I am responsible for the preparation of the technical report titled Technical Report – Allied Nevada Gold Corp., Hycroft Mine, Winnemucca, Nevada, USA dated September 23, 2010, relating to the Hycroft property
9. As of the date of the report, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
10. That I have read NI 43-101 and Form 43-101F1, and that this technical report was prepared in compliance with NI 43-101.
11. I am independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated September 23, 2010

Signature of Qualified Person

Scott E. Wilson

Printed Name of Qualified Person