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TECHNICAL REPORT

ALLIED NEVADA GOLD CORP.

HYCROFT MINE, WINNEMUCCA, NEVADA, USA

MAY 15, 2009

Prepared by

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1 SUMMARY

1.1 INTRODUCTION

This 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, the New York Stock Exchange and the American Stock Exchange (Symbol ANV). Allied Nevada controls more than 100 mineral properties throughout Nevada including the Hycroft Gold Mining Operation which is the subject of this report.

This report variously describes six gold and silver deposits collectively referred to as the Hycroft Mine, Hycroft Project or simply Hycroft. The six 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 Resources)
- Bay Area Deposit (Mineral Resources)
- Boneyard (Mineral Resources)
- Central Fault (Mineral Resources)
- Deep Sulfides Vortex Zone

All material at Hycroft has been classified in accordance with the resource classification of the Canadian Institute on Mining, Metallurgy and Petroleum ("CIM") as in compliance with National Instrument 43-101 ("NI 43-101").

This new Technical Report identifies changes to the NI 43-101 compliant Mineral Reserves and Mineral Resources that were reported in the October 17, 2009 Technical Report, authored by SEWC and published on SEDAR. Numerous sources of information, both digital and hard copy, were used in the preparation of this report. The data comprises over three thousand exploration holes as well as updated geological interpretations. Ordinary Kriging was used as the grade estimation technique.

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 March 3, 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 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, 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 put on a care and maintenance program in December 1998 due to low gold prices (below $300 per ounce).

The Hycroft Mine historically comprised two primary lease holdings named the Crofoot and Lewis properties. These properties comprise approximately 11,829 acres of which the Crofoot property is approximately 3,636 acres and the Lewis property is 8,193 acres. The Lewis property completely surrounds the Crofoot property. In May, September and October of 2006, 717 additional claims were located comprising 14,340 acres in the name of Hycroft Resource and Development Corporation. Together, Allied Nevada holds 2,349 unpatented claims covering approximately 48,000 acres on the Hycroft Property.

The Crofoot property is held by Hycroft Resources and Development Inc., a wholly-owned subsidiary of Allied Nevada. A 4% net profits interest 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, after which Allied Nevada will own the property. An additional $120,000 is due if ore production exceeds 5.0 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% net profits royalty. Royalty payments to Crofoot have totaled $840,000 since the amended agreement.

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

Of the approximately 50,000 acres of patented and unpatented mineral claims 7,700 acres are within the current plan of operations. Nearly 2,600 acres have been disturbed by mining operations. There is one 20 acre claim on the north end of the Central Fault that is not controlled by Hycroft. This claim is not in an area that impacts any current or future 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	Fifty four miles west of Winnemucca via the Jungo Road
Topography	Low Hills
Climate	Arid Desert
Historic Production	Over 1,000,000 Ounces Since 1983
Reason for NI 43-101 Technical Report	Material Changes to the Mineral Resources and Mineral Reserves at the Hycroft Mine
Mineralization Type	Fracture Controlled Disseminated Gold
Estimation Type	Ordinary Kriging
Mine Life	6.5 Years
Production Rate Imperial Tons	25 Million Tonnes per Year Mined - Target of 180,000 Contained Ounces Placed on Pad
Mining Method	Open Pit Truck and Shovel
Processing Method	Run of Mine Heap Leaching
Processing Gold Recovery	56%
Gold Selling Price SEWC Financial Model	$650 USD
Mining Cost per Imperial Ton Mined	$1.14 USD
Processing Cost per Imperial Ore Ton (Includes G&A)	$1.37 USD
Operating Cost per Imperial Ore Ton	$2.59
Pre Tax Cash Flow – SEWC Financial Model	$86.582 Million
Pre Tax Net Present Value at 10% - SEWC Financial Model	$45.416 Million
Pre Tax IRR – SEWC Financial Model	55%
Approximate Time for Payback	42 Months

1.2 GEOLOGY AND MINERALIZATION

The Hycroft Deposit is located in the Nevada Basin and Range geologic province on the western flank of the Kamma Mountains, along the county line between Humboldt and Pershing Counties, Nevada. Tertiary to recent, fault-controlled, low-sulfidation gold deposits occur over an area measuring 3 miles in a north-south direction by 1.5 miles in an east-west direction. Based on drilling results, mineralization extends to depths of at least than 330 ft in the outcropping to near-outcropping portion of the Bay deposit on the northwest side and to over 1000 ft in the Brimstone deposit in the eastern portion of the Hycroft property.

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, Central and South Central, Cut 3, and Cut 4 deposits are hosted by 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 controlled, chalcedony/pyrite/marcasite mineralization. Oxidation appears to be related to a deep acid leaching event.

1.3 DRILLING AND SAMPLING

Exploration and development drilling, by Allied Nevada and its predecessors, totals 1,270,583 feet of drilling, in 3,626, drillholes at the Hycroft Mine. In December 2006, Allied Nevada drilled one reverse circulation hole at the south end of the Cut 4 Pit. The Company then commenced a drilling program of seventy holes in August 2007 to delineate oxide and sulfide resources throughout the entire Hycroft property. As of December 31, 2008, Allied Nevada had completed 393 holes totaling 300,494 feet drilled with encouraging results.

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 drillhole databases for resource estimation.

1.3.1 RESOURCES

SEWC has developed the breakdown for resources on the Hycroft property. Resources for this report are broken down as either 1) Oxide Gold 2) Oxide Silver 3) Sulfide Gold 4) Sulfide Silver. The breakdowns are characterized as:

- *Oxide Mineralization* - The oxide material will be processed by utilizing the existing and expanded heap leach pads and processing facilities. Oxide Silver is included only as Mineral Resources due to limited understanding of oxide silver recoveries. Until additional silver assays are obtained and silver recovery can be determined, no contained silver will be reported for the oxide reserves and resources.
- *Sulfide Gold Mineralization* - The inferred resource estimate for sulfide material containing gold only was calculated from fire assay data obtained from over 3,400 historic exploration drill holes, comprising approximately 1,100,000 feet of drilling. These exploration holes were drilled to a shallow depth and were not fire assayed for silver. These historic holes were designed to identify the extent of oxide (heap leachable) gold only and as such were neither designed to test the depth nor the extent of the sulfide mineralization. Subsequent analysis of the 3,200 historic drill holes determined that approximately 90% of the historic drill holes bottomed in sulfides.
- *Sulfide Silver Mineralization* - The inferred resource estimate for sulfide material containing silver was determined from 393 holes drilled since fall of 2007, comprising approximately 300,000 feet of drilling. These holes were fire assayed for both gold and silver. This inferred resource is calculated over a wide area of Hycroft deposit and shows the potential for there to be a large mineralized system at Hycroft. Drill testing of this resource is sufficient for the calculation of inferred resource. Additional resource development drilling will convert a substantial portion of this deposit into the indicated and measured categories.

1.3.2 NI 43-101 COMPLIANT MEASURED AND INDICATED MINERAL RESOURCES

The March 31, 2009 measured and indicated gold resource is reported at a 0.17 g Au/t cutoff grade. Measured and Indicated Oxide Gold Resources for the Hycroft Mine deposit are shown in Table 1.2. Measured and Indicated Sulfide Gold Resources for the Hycroft Mine deposit are shown in Table 1.3.

Table 1.2 Hycroft Mine Measured and Indicated Oxide Gold Resources

March 31, 2009 Measured and Indicated Gold Resource			
Cutoff g Au/t	Tonnes 1,000's	Grade g Au/t	Contained oz Au
0.17	393,901	0.40	5,095,000

Table 1.3 Hycroft Mine Measured and Indicated Sulfide Gold Resources

March 31, 2009 Measured and Indicated SX Gold Resource			
Cutoff g Au/t	Tonnes 1,000's	Grade g Au/t	Contained oz Au
0.45	46,963	0.53	798,000

The March 31, 2009 measured and indicated silver resource is reported at a 12.41 g Ag/t cutoff grade. Measured and Indicated Oxide Silver Resources for the Hycroft Mine deposit are shown in Table 1.4. Measured and Indicated Oxide Silver Resources for the Hycroft Mine deposit are shown in Table 1.4.

Table 1.4 Hycroft Mine Measured and Indicated Oxide Silver Resources

March 31, 2009 Measured and Indicated Silver Resource			
Cutoff g Ag/t	Tonnes 1,000’s	Grade g Ag/t	Contained oz Ag
12.41	70,385	31.13	70,452,000

Table 1.5Hycroft Mine Measured and Indicated Sulfide Silver Resources

March 31, 2009 Measured and Indicated Silver Resource			
Cutoff g Ag/t	Tonnes 1,000’s	Grade g Ag/t	Contained oz Ag
12.41	34,198	42.79	47,050,000

1.3.3 NI 43-101 COMPLIANT INFERRED MINERAL RESOURCES

The March 31, 2009 Hycroft Inferred Oxide Gold Resources are shown in Table 1.6 at a cutoff grade of 0.17 gpt AuFA. Hycroft Inferred Oxide Silver Resources are shown in Table 1.7 at a cutoff grade of 12.41 gpt AgFA. Inferred Sulfide Gold Resources are reported in Table 1.8 at a gold cutoff grade of 0.45 gpt AuFA. Inferred Sulfide Silver Resources are tabulated in Table 1.9 at a cutoff grade of 12.41 gpt AgFA. Industry accepted standards for resource estimation were used to determine the extent of mineralization at Hycroft. Gold and silver mineralization was estimated using ordinary kriging of 25 foot drillhole composites.

Table 1.6 Hycroft Inferred Oxide Gold Resources (Includes Crofoot Pad)

March 31, 2009 Inferred Gold Resource			
Cutoff g Au/t	Tonnes 1,000's	Grade g Au/t	Contained oz Au
0.17	139,408	0.37	1,643,000

Table 1.7 Hycroft Inferred Oxide Silver Resources

March 31, 2009 Inferred Silver Resource			
Cutoff g Ag/t	Tonnes 1,000's	Grade g Ag/t	Contained oz Ag
12.41	11,503	51.27	18,962,000

Table 1.8 Hycroft Inferred Sulfide Gold Resources

March 31, 2009 Inferred Gold Resource			
Cutoff g Au/t	Tonnes 1,000's	Grade g Au/t	Contained oz Au
0.45	218,524	0.62	4,371,000

Table 1.9Hycroft Inferred Sulfide Silver Resources

March 31, 2009 Inferred Silver Resource			
Cutoff g Ag/t	Tonnes 1,000's	Grade g Ag/t	Contained oz Ag
12.41	103,405	47.82	158,965,000

1.4 OPERATING MINE PLAN

SEWC used current economics to develop a new mine plan for the Hycroft Mine. At a production rate of approximately 23 million tonnes of ore and waste per year the mine can operate for about 6.5 years. The ore will be placed on Phases 1 and 2 of the Brimstone Pad. Leaching of the ore will take approximately 8 years.

The current plan is for the mine to run twenty four hours per day, seven days per week. Production is expected to average 1.9 million tonnes of total mine production per month. The ore cutoff grade is 0.16 g Au/t. Ore will be placed on the Brimstone Pad without crushing (run of mine) and waste will go to one of several dump locations. Much of the waste will be used to backfill the Central fault pit.

All ore-grade material placed on the leach pad will be run of mine and cross-ripped to enhance permeability. A network of solution drip lines will be positioned and the run of mine material will be leached with a cyanide solution for a period of 60 to 90 days before another 30 ft high lift of ore is placed on top of the existing one. Return solution from the pad containing the precious metals is directed to the pregnant solution pond.

The pregnant solution will be processed at a Merrill-Crowe zinc-precipitation plant that has been maintained in pristine condition since it was shut down in 1998. The Merrill-Crowe process clarifies and de-oxygenates the pregnant solution using two 1,600 square foot Sparkler filters. Zinc dust is applied to the clarified solution where gold precipitates and is collected on three 48 inch recessed plate filter presses. The collected precipitate will be refined at a new refinery where mercury will be removed and the gold will be fire refined. This is a closed process so the barren solution is returned to the leach pad circuit to start the process again. Expected recovery of gold is 56.6% of the total gold.

1.5 NI 43-101 COMPLIANT PROVEN AND PROBABLEMINERAL RESERVES

Economic reserves for the Brimstone and Cut 5 deposits were calculated based on current operational economics for Hycroft. The 2008 SEWC reserve block model was used. SEWC verified the economic pit limits of the mineral reserve estimate using Whittle 4.0 software. Table 1.10 summarizes the Hycroft reserves which are unchanged since the October 17, 2008 Technical Report. The stated Mineral Reserve Estimate conforms to the December 23, 2005 CIM definitions of Proven and Probable Mineral Reserves.

Table 1.10 Hycroft Mine Proven and Probable Mineral Reserves at October 17, 2008

Cutoff	Category	Tonnes 1,000's	Grade g Au/t	Ounces
0.17 g Au/t	Proven	42,236	0.55	747,831
0.17 g Au/t	Probable	24,134	0.51	395,347
Total Proven and Probable Mineral Reserves		**66,369**	**0.54**	**1,143,178**

The waste material inside the final pit design includes 2.47 million tonnes of oxide inferred material grading 0.44 gpt AuFA above a 0.17 gpt AuFA cutoff grade. Additionally the waste also includes 13.63 million tonnes of sulfide inferred material that grades 0.45 gpt AuFA above a 0.17 gpt AuFA cutoff grade. Though these mineral resources will 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 several metallurgical studies to determine the recoveries associated with the potential to process sulfide gold and silver material. The results are encouraging. The large sulfide resource has been drill tested and delineated by widely space drilling programs. Allied Nevada should institute drilling programs to increase the drilling density of the deep sulfide and Vortex Zone inferred gold and silver resource portion of the ore deposit. This will allow for the potential to convert resources into the indicated mineral resource category. Indicated resources should be evaluated for the potential to sustain sulfide processing methods.

1.7 CONCLUSIONS

1.7.1 Adequacy of Procedures

SEWC, as well as other reputable firms and consultants, has reviewed the methods and procedures of Allied Nevada and its predecessors. The methods of geological interpretation, geotechnical evaluation, mine planning 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

The drillhole 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 and Cut 5 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.

1.7.5 Cautionary Note to U.S. Readers Concerning Estimates 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 standards of the United States Securities and Exchange Commission. 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. 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.

1. Determine the pre-backfill surface at the Bay Area
2. Investigate gold and silver recovery and processing methods for sulfide material
3. Drill the Vortex Zone on 200 foot centers

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 PURPOSE OF TECHNICAL REPORT

Scott E. Wilson Consulting, Inc. (SEWC) prepared this technical report of the Hycroft Mine at the request of Allied Nevada Gold Corp. (Allied Nevada), a Delaware corporation. The Hycroft Mine is owned by Allied Nevada. Allied Nevada made the decision in September of 2007 to re-activate their wholly owned Hycroft Mine which was placed in care and maintenance program in late 1998 due to low metal prices. Open pit mining of the Brimstone Pit resumed in the third quarter of 2008. Gold production is expected in the fourth quarter of 2008. Approximately 650,000 ounces of gold will be recovered over five years.

This report is intended to provide a technical summary of the Hycroft Mine gold and silver resource and reserves for Allied Nevada. 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 of Englewood Colorado authored a technical report pertaining to the Hycroft Mine dated October 17, 2008 (Wilson, 2008). The technical information contained in this technical report reflects material changes that have occurred since the October 2008 Report. The remaining resources and reserves cited for the Hycroft Mine are current as of March 30, 2009.

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 and Cut 5 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 oxide and sulfide, property-wide gold and silver Mineral Resource estimates for the Hycroft Mine. Also the author was mandated to determine the Proven and Probable Mineral Reserves for the Hycroft Mine. 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 minesite for a personal inspection on March 3, 2009 to validate mine planning options related to production of gold at the Hycroft Mine.

2.4 TERMS OF REFERENCE

Unless stated otherwise, all volumes and grades are in metric units and currencies are expressed in constant 2009 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

Cubic foot	ft^3
Cubic yard	yd^3
Degree	°
Degrees Fahrenheit	°F
Foot	ft
Gallon	gal
Gram	g
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
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
Tonne	t
Year (US)	yr

2.5.2 Common Chemical Symbols

Calcium carbonate	$CaCO_3$
Copper	Cu
Cyanide	CN
Gold	Au
Hydrogen	H
Iron	Fe
Lead	Pb
Silver	Ag
Sodium	Na
Sulfur	S
Zinc	Zn

2.5.3 Common Acronyms

AA	atomic absorption
AuEq	gold equivalent
BLM	U.S. Bureau of Land Management
CIM	Canadian Institute of Mining, Metallurgy and Petroleum Engineers
EIS	Environmental Impact Statement
EPA	U.S. Environmental Protection Agency
FCCPM	Fracture controlled chalcedony-pyrite-marcasite mineralization
ISO	International Standards Organization
NDEP	Nevada Department of Environmental Protection
NPI	Net profit interest
NSR	Net Smelter return
Oz Ag/ton	Silver ounces per short ton
Oz Au/ton	Gold ounces per short ton
ROM	Run of mine

RQD	Rock quality designation
RC or RVC	Reverse circulation

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. SEWC regularly discusses the Hycroft Mine and material information with the following people:

- Mr. Mike Doyle, Vice President of Technical Services, Allied Nevada Gold Corp.
- Mr. Hal Kirby, Vice President and Chief Financial Officer, Allied Nevada Gold Corp.
- Mr. David Flint, Chief Geologist, Allied Nevada Gold Corp.
- Mr. Warren Woods, General Manager, Hycroft Resources and Development, Inc.
- Mr. Todd Sylvester, Mine Manager, Hycroft Resources and Development, Inc.

The author has exercised independence in reviewing the supplied information and 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.

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 the Jungo road, Nevada Highway 49, 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 40° 52’ north and longitude 118° 41’. The mine is situated on the western flank of the Kamma Mountains and on the eastern edge of the Black Rock Desert in unsurveyed 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, Humboldt County. An additional 1,057 unpatented lode mining claims were staked in April and May, 2008 and recorded with the Bureau of Land Management (“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 Gold Corp (“Vista”) transferred its Nevada assets, including Hycroft, to Allied Nevada. The Hycroft Mine historically comprised two primary lease holdings named the Crofoot and Lewis properties. These properties comprise approximately 11,829 acres of which the Crofoot property is approximately 3,636 acres and the Lewis property is 8,193 acres. The Lewis property completely surrounds the Crofoot property. In May, September and October of 2006, Vista Gold located 717 additional claims comprising 14,340 acres. In November of 2007 Allied located 25 claims, and in 2008 Allied located 1,058 claims in January, April and May, comprising 21,660 acres. Allied Nevada currently holds 2,349 unpatented claims covering approximately 48,000 acres.

4.2 MINERAL TENURE

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

The Crofoot property is owned by Hycroft Resources and Development Corporation. A 4% net profits interest is retained by the original Crofoot owners. In 1996 the lease/purchase agreement was amended to provide for minimum advance royalty payments of $60,000 on January 1 of each year in which mining occurs on the patented claims covered under the amended agreement and $60,000 on January 1 of each year mining takes place on the unpatented claims covered under the amended agreement. All payments for the Crofoot property are capped at $7.6 million, after which Allied Nevada will own the property. An additional $60,000 is due if ore production exceeds 5.0 million tons from the patented claims on Crofoot property and an additional $60,000 if production exceeds 5.0 million tons on the unpatented claims on the Crofoot property in any calendar year. All advance royalty payments are available as credit against the 4% net profits royalty. Royalty payments to Crofoot have totaled $840,000 since the amendment agreement.

There are 2,349 unpatented mining claims covering approximately 48,000 acres at the Hycroft site. An additional 1,440 acres are in patented lode and placer claims and are the core property surrounded by the unpatented claims. The permitted site disturbance for current and future mining activities total 2,600 acres. There is one 20 acre claim on the north end of the Central Fault that is not controlled by Hycroft. This claim is not in an area that impacts any current or future operations.

Allied Nevada possesses all of the necessary permits, facilities and infrastructure to allow resumption of mining at Hycroft. Capital investment has been approved for pre-production stripping, leach pad development, the purchase of a used mining fleet and other capital expenditures necessary to restart the Hycroft Mine. Figure 4.2 shows the property layout including site facilities, mineralized zones, mine workings and waste deposits.

Figure 4.1 Hycroft Mine Property Location Map

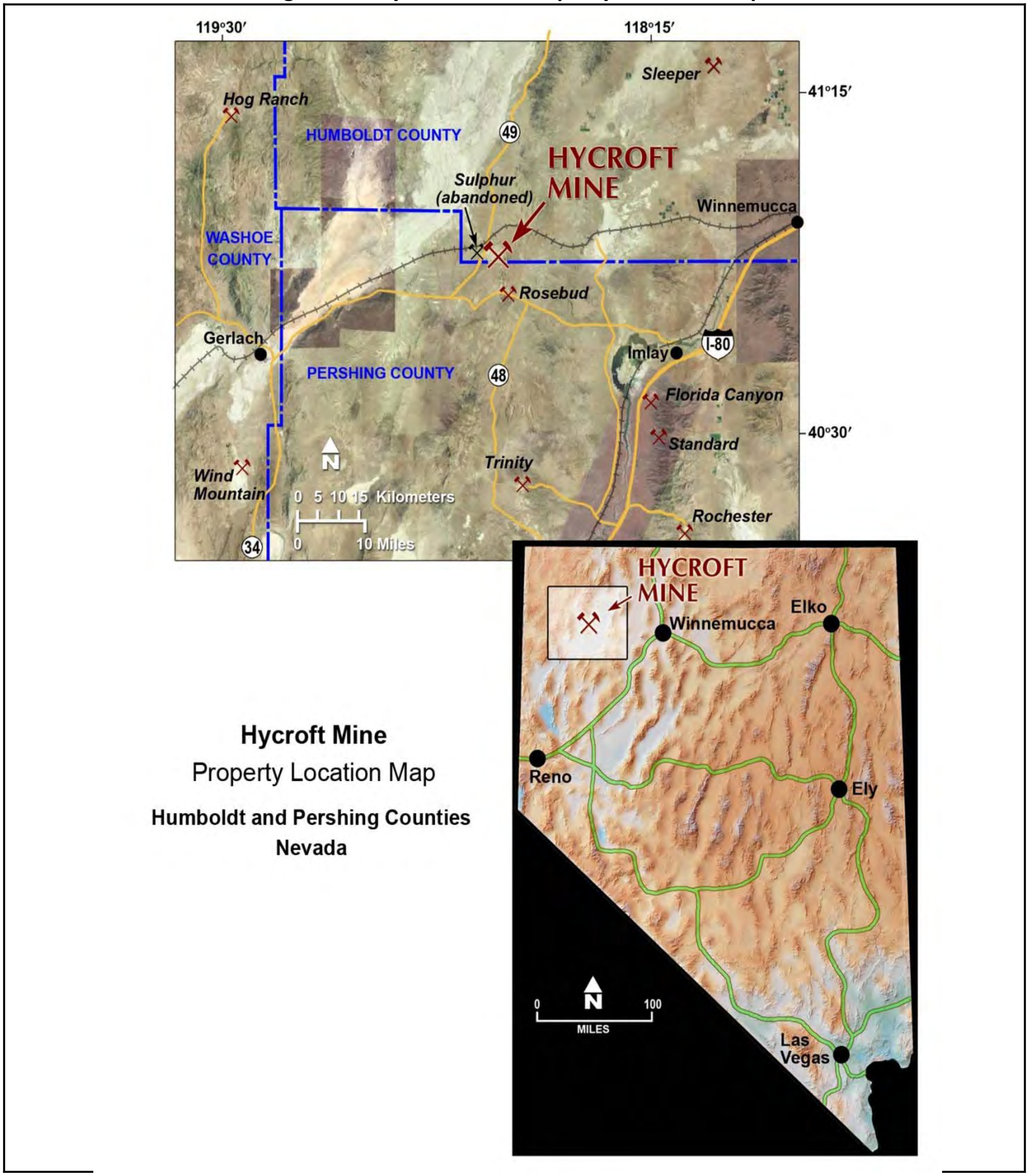
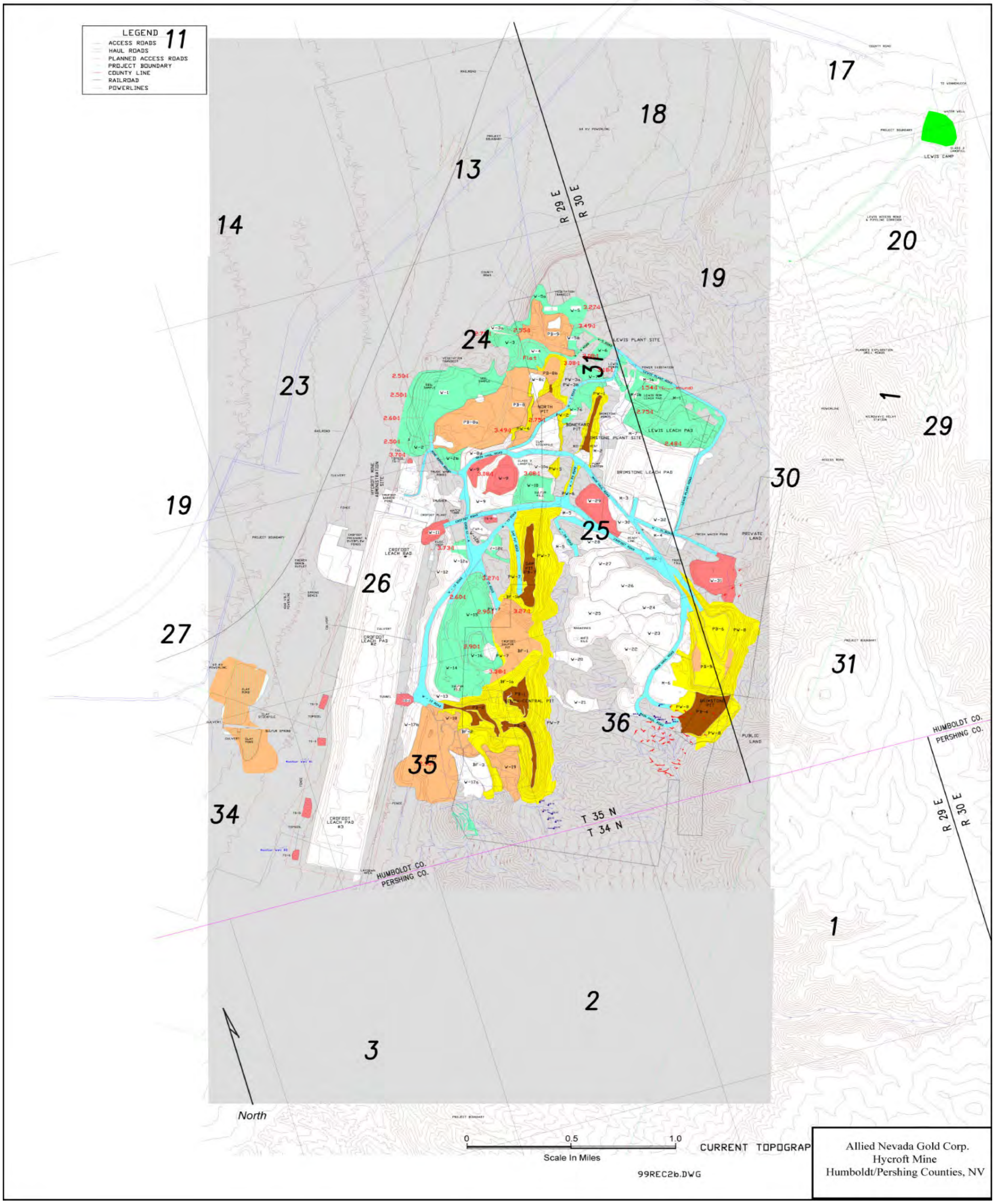


Figure 4.2 Property and Facilities Layout



4.3 AGREEMENTS AND ROYALTIES

The leasehold interests of Hycroft Mine 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 3,636 acres and is virtually surrounded by the Lewis property of 8,193 acres.

Vista exercised their 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. There is no longer any royalty on gold and silver produced from the previous Lewis ownership.

In May, September and October of 2006, Vista Gold located 717 additional claims comprising 14,340 acres in the name of Hycroft Resource and Development Corporation.

The Crofoot property was originally held under two leases and is now optioned by Hycroft Resources and Development Inc. (HRDI), subject to a 4% net profits interest retained by the former owners for production of all minerals and ores except mercury and sulfur. The net profits interest for mercury shall be 7% and Crofoot retains all rights to the sulfur except as that amount needed to maintain production of other ores and minerals. In 1996, the lease/purchase agreement was amended (4th Amendment) to provide for minimum advance royalty payments of $120,000 on January 1 of each year in which mining occurs both on patented and unpatented mining claims. An additional $120,000 is due if ore production exceeds 5.0 million tons from the Crofoot property in any calendar year on both patented and unpatented mining claims. All advance royalty payments are available as credit against the 4% net profits royalty. Under the 1996 amended agreement, the Crofoot royalty is capped at $ 7.6 million of which $3.8 million is for the patented claims and $3.8 million is for the unpatented claims. To date, $0.84 million has been paid to the Crofoot family under the provisions of the 1996 amended agreement.

Figure 4.3 Claim Boundaries

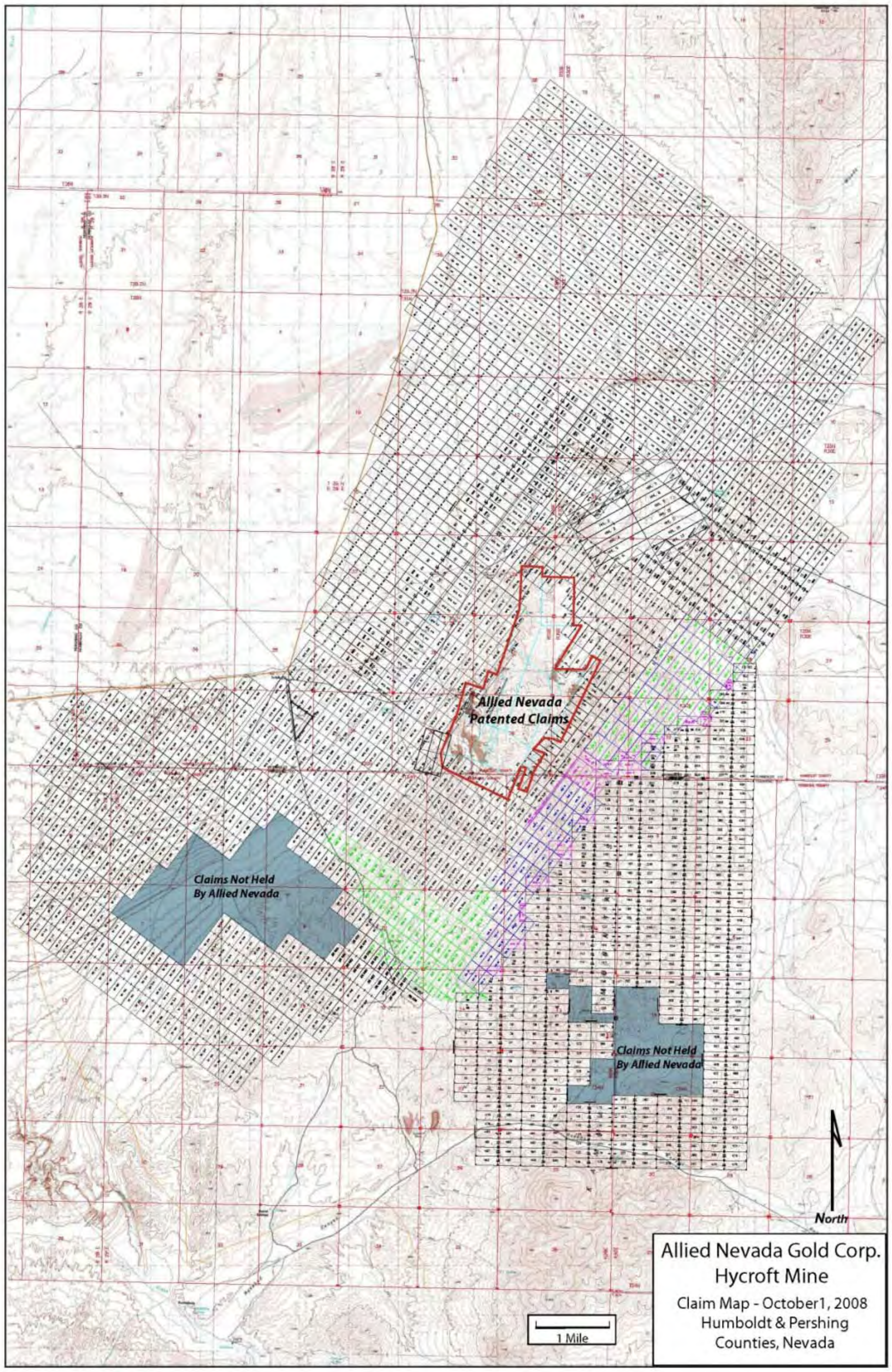


Table 4.1 Hycroft Land Holdings Costs

Month Due	*Lessor*	*Type*	*$ Amount*
January	*Crofoot*	*Advance Royalty*	*$120,000*
	U.S. BLM, Humboldt & Pershing Counties	*Unpatented Claim Fees*	*$63,992*
	Communication Site of Floka Peak	*Annual Fee*	*$1,809*
	Potable Water Permit # Hu-0864-12NCNT State Division of Health	*Annual Fee*	*$225*
	Bio-Remediation Cells Permit #GNV041995 Bureau of Mining Regulation	*Annual Fee*	*$200*
February	*Permit #1182-2354 Nevada State Fire Marshal*	*Annual Fee*	*$150*
October	*Permit #03615- Nevada Board for the Regulation of Liquefied Petroleum Gas*	*Annual Fee*	*$135*

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 Hycroft Resources and Development, Inc. (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 initial payment of $4.0 million and two additional payments of $1.3 million after 6 months and 11 months from the initial payment. The bonding instrument was accepted by the Bureau of Land Management (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 bond amount increased to BLM.

Allied Nevada Gold contracted SRK Consulting of Reno to review the Hycroft Plan of operations, 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 Nevada Cost Data File and Hycroft Interim Fluid Management plan. The new project reclamation estimate totals $14,343,100, which has been approved by both the Nevada Department of Environmental Protection and Bureau of Land Management. The updated financial guarantee has been submitted to the BLM.

4.5 PERMITS

Hycroft Mine operates under permit authorizations from the BLM, Nevada Division of Environmental Protection, 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 & Reclamation Plan	BLM	#N26-87-002P	Current
Reclamation Surety Bond	Am Home Assure Co.	N-64641	Current
Manufacture of High Explosives	Bureau of Alcohol, Tobacco & Firearms	#9-NV-013-20-5C-12087	Current
Class II Air Quality Permit	NV Division of Environmental Protection Bureau of Air Quality	#AP1041-0661.01	Current
Water Pollution Control -Crofoot Operation	NV Bureau of Mining Regulation & Reclamation	NEV60013	Current
Water Pollution Control -Brimstone Operation	NV Bureau of Mining Regulation & Reclamation	NEV94114	Current
Water Pollution Control -Closure of Lewis Facility	NV Bureau of Mining Regulation & Reclamation	NEV89017	Current
Bioremediation Facility Permit	NV Bureau of Mining Regulation & Reclamation	#GNV041995	Current
Reclamation Permit	NV Bureau of Mining Regulation & Reclamation	#0134	Current
Stormwater Pollution	NV Bureau of Water Pollution Control	#NV0050006-10037	Current
Artificial Pond Permit (Brimstone Mine)	NV Dept of Wildlife	S21090	Current
Artificial Pond Permit (Crofoot Mine)	NV Dept of Wildlife	S23123	Current
Crofoot Process Ponds	NV Division of Water Resources	#J-273	Current
Crofoot Process Well #1	NV Division of Water Resources	#60230	Current
Crofoot Process Well #2	NV Division of Water Resources	#60231	Current
Crofoot Potable Well	NV Division of Water Resources	#49533	
Hazardous Materials Storage Permit	NV State Fire Marshall	#1182-2354	Current

Table 4.3 Hycroft Miscellaneous Permits

Operating Permits	Issuing Agency	Number	Status
R/W Communication Site on Floka Peak	BLM	N46292	Current
R/W Potable Water Well/Pipeline/Power Line	BLM	N-46564	Current
R/W Process Wells/Pipeline/Power Line	BLM	N-46959	Current
R/W Road & Waterline (Old Mancamp to Lewis)	BLM	N-39119	Current
R/W Mabel Well Pipe Line to Mancamp	BLM	N-44999	Current
Kamma Peak Station	FCC	WNER344	Current
Sulfur Mine Station	FCC	WNER345	Current
Winnemucca Mtn. Station	FCC	WNER346	Current
Base Station & 45 Mobil Units	FCC	WNKK336	Current
Class 3 Landfill Permit	NV Bureau of Waste Management	#SWM1-08-11	Current
Potable Water Permit	NV Division of Water Resources	#HU-0864-12NCNT	Current
Propane	NV Board for the Regulation of LPG	#03615	Current
Regional General Permit	U.S. Army Corps of Engineers	Section 404 Permit	Current

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 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.

5.2 CLIMATE

The climate of the region is arid, with precipitation averaging 7.6 inches 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°'s F at night and near 90° F and above during the days. Winter temperatures are usually in the 20°'s F at night and in the 40°'s 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 ranges in temperatures 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, 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

Permit #	Well Name	Gallons per Well	Total Combined Gallons
60230	Near Process	471,903,000	1,076,502,000
60231	Far Process	471,903,000	
49533	Crofoot Potable	4,831,000	
47423	Lewis Camp	57,970,000	
42675	Mabel Crofoot	59,095,000	
46794	Grable Camp	10,800,000	
60230	Near Process	471,903,000	

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, thoroughly 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 the winter, it becomes a temporary lake.

There are no streams, rivers, or major lakes in the general area. Elevations range from 4,500 to 5,500 feet 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, South Central, Boneyard, Gap and Cut 4 Pits along the Central Fault. The north end of the Brimstone Pit continued until it was placed on a care and maintenance program in December 1998 due to low gold prices below $300 per ounce of gold.

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 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 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. 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 drilling that has identified a large mineralization system below the Brimstone and Cut 5 deposits.

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 1950’s. 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 117.9 Kg/tonne and 12.4 g/tonne gold were reported by Jones (1921). Silver production ceased by 1912 with a total estimated production of 5670 kg of silver. 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 feet in length, 10 feet in width, and 100 feet in dip extent. An estimated 2,500 tons has been mined at an unknown grade between 1932 and 1937.

During the First World War, three 6 – 8 foot 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 were mined as a soil additive (Fulton and Smith, 1932). Vandenburg (1938) estimated that 454 tons of alunite were shipped to the West coast to be used as fertilizer. From 1941 -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, Duval Corporation drilled 20 holes on the property in search of a Frasch-type sulfur deposit (Wallace, 1980). Duval Corporation found no evidence for 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 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 Company 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 tonnes at 1.20 g Au/t. 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

HRDI drilled between 1985 and 1999, a total of 3,123 exploration drill holes, totaling 943,822 ft. The current Hycroft drillhole database consists of the former holes, plus 61 RC holes drilled by Homestake in 1982 and 29 rotary holes completed by Homestake in 1981. The Duval Corporation holes are not included in the database, but did guide some early exploration. 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	# 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
Total			**3640**	**1,274,096**	

Exploration by Hycroft and Homestake resulted in the discovery of seven zones of mineralization. These are described in detail in the exploration section of this document and are shown in Figure 7.1. These zones include:

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 represents the north end of the district, and extends for 2,000 ft in a north-south direction along the Central Fault, between 49,000N and 51,000N. This type of 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-Kspar chalcedony-pyrite. Oxidation forms an 80-100 foot thick blanket over the hypogene mineralization in the form of clay alteration with an abundant zeolite (mordenite). This area was drilled out as the first reserve on the project.

6.2.2 Central Fault, South Central, Gap, Cut 4 and Cut 5 Deposits

These deposits occur in a 10,000 ft segment 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 Synge sediments, and Tertiary Kamma Volcanics. The Camel Conglomerate has been altered to an opal-Kspar 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.

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 deposits.

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

6.2.4 The Fire and Brimstone Deposit

The Fire & Brimstone deposit is hosted in rhyolitic, aphanitic and tuffaceous Kamma volcanics in the southeastern part of the Crofoot Lewis mine area. The deposit consists of 2 major zones of hydrothermal venting, displaying fracture-controlled chalcedony-pyrite-marcasite mineralization as veinlets, hydrofracture fill, and chaotic hydrothermal breccia. The deposit is oxidized by an acid leach/oxidizing event.

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 0.0143 oz Au/ton. The remaining Mineral Reserves at Hycroft are contained in the southern portions of Brimstone.

6.2.5 The Albert Deposit

This area of mineralization is located approximately halfway between the Central Fault and the Fire & Brimstone deposit. The mineralization is hosted in both sedimentary and volcanic rocks. The north-striking west-dipping, Albert Fault separates dominantly sedimentary Camel Conglomerate from Kamma volcanic rock in the footwall of the Albert Fault.

Deep drill holes in the Albert area suggest a deep unconformity between the Kamma Volcanics and the Camel Conglomerate above. The Albert mineralization is included in the Brimstone resources and reserves.

6.3 CANYON RESOURCES 2005 DRILLING PROGRAM

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.

6.4 PRODUCTION HISTORY

Information on the production history of the Hycroft Mine comes from Allied Nevada 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, 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. All data in section 6.4 expressed in US Imperial units.

Table 6.2 Historic Production (US Imperial Units)

Deposit	Years Mined (*approximate*)	Tons (millions)	Grade Cn oz Au/ton	Ounces Au produced
Lewis Mine	*1983-1985*	3.9	N/A	N/A
Bay	*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 ¾ inch or treated as run-of-mine, depending on the blast-hole grade. The Central Fault production was leached 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 carbon 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 an average cyanide soluble gold grade of 0.016 oz Au/ton or 1.31 million ounces of gold placed. A total of 1.053 million ounces of gold has been recovered, as shown in Table 6.3.

Table 6.3 Historic Pad Production (US Imperial Units)

Year	Hycroft Pad Loading Tons (000's)					Ore Tons	Waste Tons	CN Au	Total Oz. Loaded (000's)						000's Oz.
	Pad 1	Pad 2	Pad 3	Pad 4	Pad 5			oz Au/ton	Pad 1	Pad 2	Pad 3	Pad 4	Pad 5	Totals	
1988	3,995.4	0.0	0.0	0.0	0.0	3,995.4	2,450.3	0.021	82.1	0.0	0.0	0.0	0.0	82.1	38.1
1989	5,144.8	104.0	0.0	0.0	0.0	5,248.8	5,682.7	0.019	98.4	2.0	0.0	0.0	0.0	100.4	73.6
1990	3,793.9	1,792.4	0.0	0.0	0.0	5,586.3	8,276.0	0.019	73.3	34.8	0.0	0.0	0.0	108.1	89.3
1991	490.3	5,309.9	0.0	0.0	0.0	5,800.2	8,182.7	0.019	9.2	99.4	0.0	0.0	0.0	108.5	92.6
1992	428.1	5,665.4	0.0	0.0	0.0	6,093.5	9,884.2	0.017	7.2	95.1	0.0	0.0	0.0	102.3	99.1
1993	588.7	4,610.4	521.1	0.0	0.0	5,720.2	16,765.4	0.018	10.7	87.0	7.9	0.0	0.0	105.6	86.5
1994	488.4	3,066.4	5,683.2	0.0	0.0	9,238.0	17,460.5	0.015	7.8	42.2	89.7	0.0	0.0	139.8	94.9
1995	463.8	4,577.7	4,890.0	0.0	0.0	9,931.5	27,263.6	0.014	6.5	53.6	78.8	0.0	0.0	139.0	101.1
1996	2,337.1	3,671.3	5,843.3	1,027.8	0.0	12,879.5	23,822.1	0.013	23.2	35.2	91.5	11.6	0.0	161.5	89.4
1997	664.3	478.8	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	0.0	0.0	0.0	5,469.6	1,647.9	7,117.4	3,009.3	0.015	0.0	0.0	0.0	82.8	24.0	106.8	112.7
1999	0.0	0.0	0.0	0.0	0.0										40.1
2000	0.0	0.0	0.0	0.0	0.0										13.0
2001	0.0	0.0	0.0	0.0	0.0										3.2
2002	0.0	0.0	0.0	0.0	0.0										1.8
2003	0.0	0.0	0.0	0.0	0.0										0.6
Totals	18,394.8	29,276.3	19,078.5	11,130.1	4,334.1	82,213.6	149,568.9	0.016	331.4	458.6	298.9	159.2	62.0	1,310.1	1,053.4

Production from the Brimstone Pit was placed directly on the heaps as run-of-mine. The leach pads used for treating the ore were Pads 4 and 5. Pad 5 consists of extra lifts placed on top of Pads 1 and 2. Pad 4 is a new pad constructed immediately south of the old Lewis Pad and was completed in 1996. Loading of Pad 4 and Pad 5 commenced in October 1996 and July 1997, respectively. A 2,800 gallon per minute 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.

6.5 HISTORICAL RESOURCE AND RESERVE ESTIMATES

Prior historical resource estimates were completed by Mineral Resources Development, Inc. ("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; 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 are considered to be indicated. Blocks with an estimation variance in excess of 0.47 are 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 cyanide-soluble gold cutoff. The grades shown in the 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 summarized the historic ore reserve.

The ore model and resource estimates were re-evaluated by ORE in 2005 and reserves re-estimated by MDA 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** g Au/t	**Cyanide Soluble 000's oz Au**	**Fire Assay g Au/t**	**Contained 000's oz Au**
Measured	21,126,000	0.41	307.6	0.51	385.1
Indicated	21,947,000	0.40	307.3	0.46	369.4
Totals	**43,073,000**	**0.40**	**614.9**	**0.49**	**754.6**

Table 6.5 Historic MAY 2000 Reserves

Category	Tonnes	Cyanide Soluble g Au/t	Cyanide Soluble 000's oz Au	Fire Assay g Au/t	Contained 000's oz Au
Inside Historic Designed Pit	4,726,000	0.39	65.4	0.48	80.4
Outside Designed	6,186,000	0.25	54.7	0.24	53.2
Totals	**10,913,000**	**0.31**	**120.1**	**0.35**	**133.6**

Table 6.6 Production Prior to February 1, 2001

Category	Tonnes	Cyanide Soluble g Au/t	Cyanide Soluble oz Au	Fire Assay oz g Au/t	Contained oz Au	Waste Tons 000's	Total Tons 000's	Strip Ratio
Proven	14,759.0	0.45	234.0	0 .56	293.0			
Probable	14,660.0	0.45	224.2	0 .53				
Totals	**24,919.0**	**0.45**	**458.2**	**0 .54**	**566.5**	**52,431**	**81,850**	**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-Pershing County line. The deposit is hosted in volcanic eruptive breccias and conglomerates associated with the Tertiary Kamma Mountain volcanic event. The volcanics are mainly acidic to intermediate tuffs, flows and coarse volcaniclastic rocks. Fragments of these units dominate the clasts in the eruptive breccias. Volcanic rocks have been block-faulted by dominant north-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. The heap leach method is widely used in the western and southwestern United States and allows the economical treatment of oxidized low-grade ore deposits in large volumes.

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 that comprise the Sulfur District.

See Figure 7.1 Simplified Geological Map of the Sulfur District.

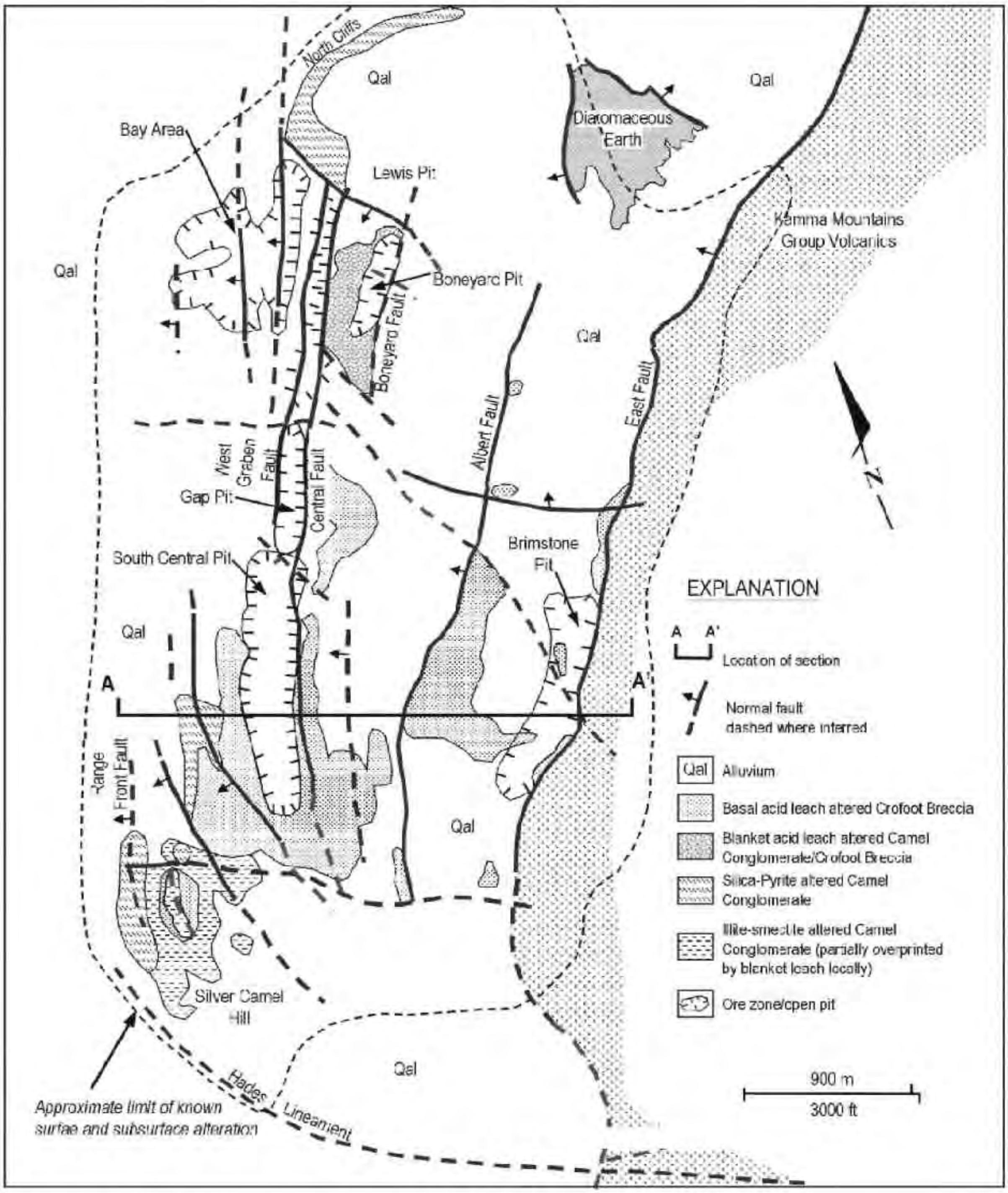
Four major north-northeast-trending, west-dipping, normal to listric fault zones appear to broadly control the location of gold and silver mineralization. From west to east, these fault zones are referred to as the Central, Boneyard, Albert, and East Faults. Figure 7.2a shows a north-looking section through the Hycroft Mine showing structures and volcanic stratigraphy. Figure 7.2b 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 to the west of the Boneyard Fault are Tertiary conglomerates, siltstones and fanglomerates of the Sulfur Group. These rocks are sediments formed from erosion of the underlying Kamma Mountains Group (KMG). Felsic tuffs and massive, flow-banded rhyolites of the KMG are present east of the Boneyard Fault.

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

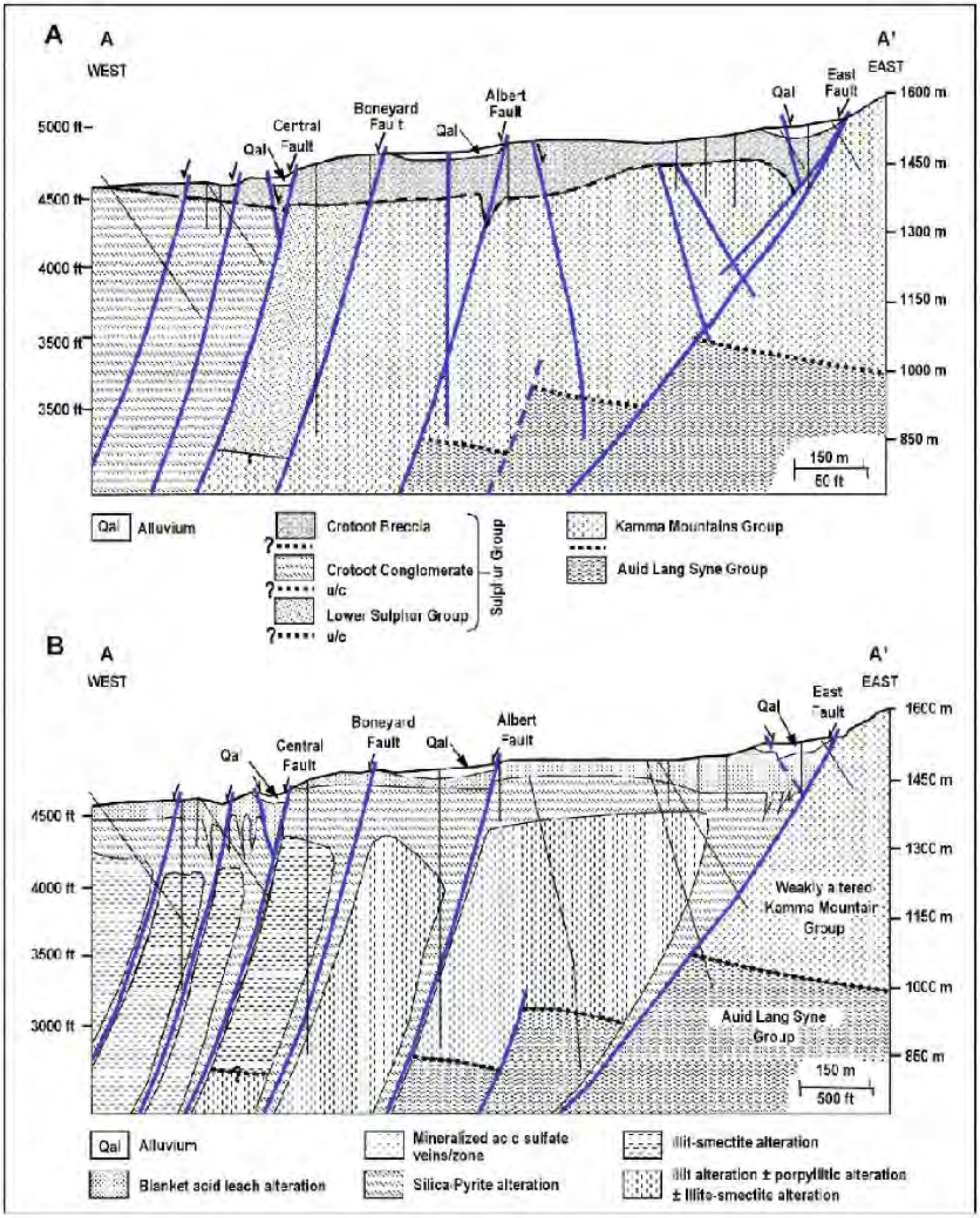
Mineralization in the Albert Zone is present along the Albert Fault, located approximately 2500 feet east of the Central Fault deposits and 2,000 feet west of the Brimstone deposit. The Albert Zone is hosted by KMG eruption breccias and volcanic flows in the hanging wall of the Albert Fault.

Figure 7.1 Simplified Geological Map of the Sulfur District



From Mineral Resources Development, Brimstone Resource Report, June, 2002

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 Fire & Brimstone deposit is hosted by volcanic rocks of the KMG rocks which are present in the hanging wall of the East Fault. The volcanic rocks are principally eruption breccias and volcanic flows proximal to vents. The volcanics overlie deformed and metamorphosed shales, sandstones and siltstones of the Mesozoic Old Lang Syne Group (OLSG). KMG volcanic rocks are strongly altered in the hanging wall of the East Fault, whereas the same units are only weakly altered to the east in the footwall of the Fault.

The East Fault is a north-northeast striking 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° to 85°. As indicated by recent drilling, the East Fault may flatten at depth to a listric fault. 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.

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 HYCROFT PROPERTY GEOLOGY – FIRE & BRIMSTONE DEPOSIT

The Hycroft Mine property consists of Tertiary- to Recent-age, fault-controlled, low-sulfidation gold deposits that occur over an area measuring approximately 3 miles in a north-south direction by 1.5 miles in an east-west direction. Drilling has shown that mineralization extends from depths of less than 330 feet from the outcropping to near-outcropping gold and silver mineralization of the portion of the Bay deposit on the northwest side and to over 1,000 feet in the Fire & Brimstone deposit in the eastern portion of the Hycroft property.

Gold- and silver-bearing rocks exposed in the Brimstone Pit are located in the hanging wall of the East Fault. These rocks were highly altered by at least four phases of alteration. Gold and silver mineralization is thought to occur during a period of fracture-controlled chalcedony-pyrite-marcasite mineralization. A subsequent acid-alteration event produced the current distribution of oxidized and mixed sulfide-oxide ore.

7.2.1 Hanging Wall of the East Fault – Brimstone Pit

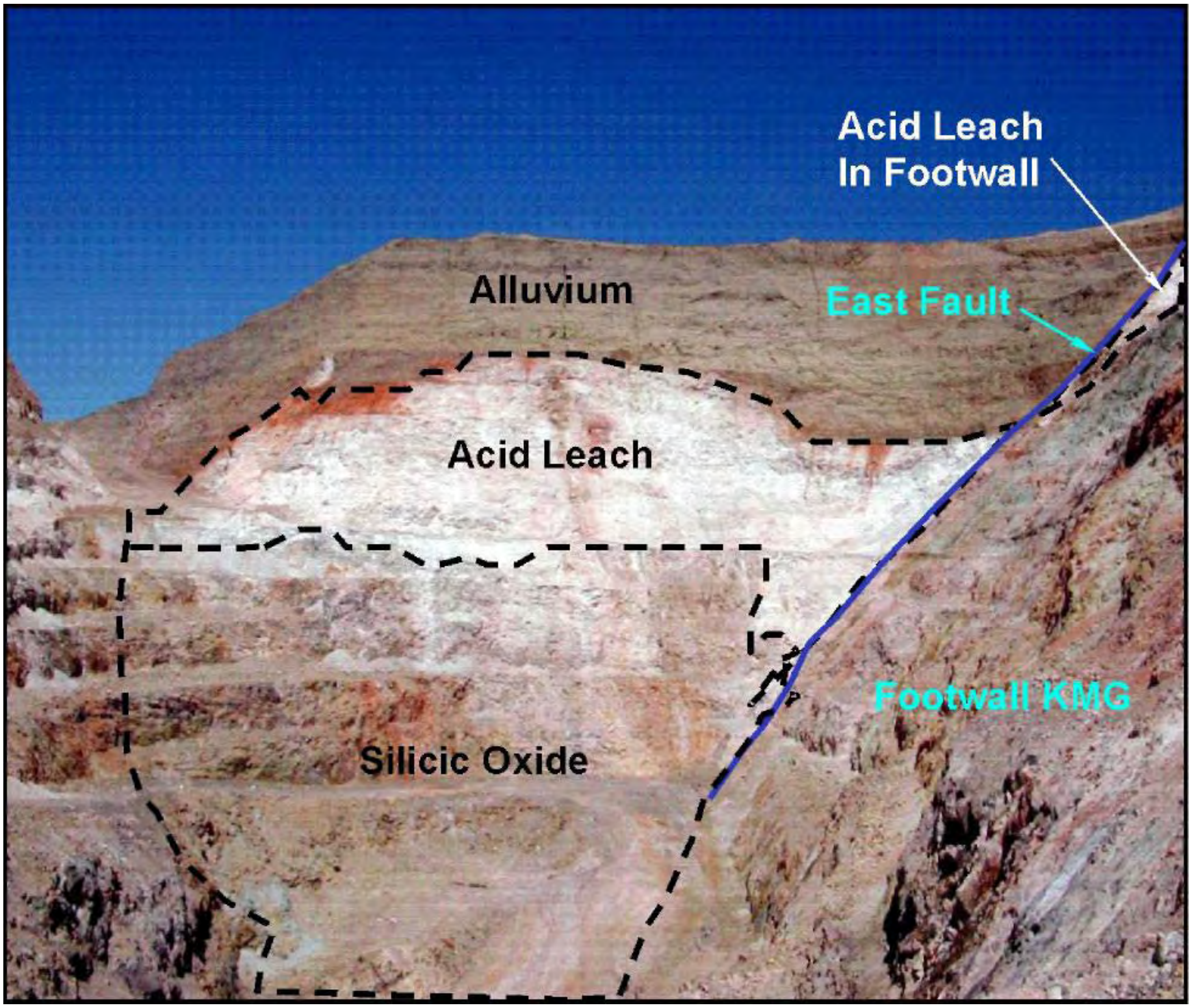
The upper 100 to 200 feet of rock in the hanging wall of the East Fault is a late hydrothermal eruption breccia called the Crofoot Breccia. This breccia is matrix-supported with clasts dominated by Kamma Mountains Volcanics. Locally, clasts of oxidized fracture-controlled chalcedony-pyrite-marcasite gold and silver mineralization are observed in the Crofoot Breccia, indicating a possible syn- or post-mineralization steam-dominated eruption event. The Crofoot Breccia is not found in the footwall of the East Fault. The average fire-assay grade of rocks logged as Crofoot Breccia is less than 0.003 opt gold, pointing to the possibility that this eruption-breccia unit is a post-mineralization barren cap overlying altered and mineralized rocks of clearly magmatic origin.

At the Fire & Brimstone Deposit, the gold- and silver-bearing host rocks are altered felsic-volcanic rocks of the KMG. The KMG rocks in the hanging wall of the East Fault are dominated by epiclastic feldspathic tuffs and aphanitic rhyolite flows. Until recently, correlation of these units has been difficult due to the lack of diamond drilling to provide core in which it is possible to observe macroscopic textures.

7.2.2 Rocks In The Footwall Of The East Fault – Fire & Brimstone Deposit

In the footwall of the East Fault, rocks are exclusively KMG-dominated by flow-banded rhyolite and epiclastic tuffs of felsic composition. Alteration and oxidation of these volcanic rocks are weak, with propylitic alteration, clay alteration, and oxidation occurring within 50 to 150 feet of the East Fault.

Figure 7.3 Brimstone North Pit Wall Geology



8 DEPOSIT TYPE

8.1 GEOLOGICAL MODEL

The Hycroft gold deposits are Tertiary to Recent age low-sulfidation deposits. Radiometric dates of adularia (potassium feldspar) indicate that the main phase of gold and silver mineralization formed four million years ago. Gold mineralization was followed 2 to 0.4 million years ago by an intense event of high-sulfidation, acid leaching of the mineralized volcanics. Acid leaching resulted locally in dissolution of the groundmass of the volcanics and of the matrix of breccias, leaving a silica-alunite-rich rock with abundant pore spaces. Locally, the acid-leached rock contains native sulfur.

8.2 HYCROFT GEOLOGICAL MODEL

The known gold and silver mineralization within the Hycroft Mine property extends for a distance of at least 3 miles in a north-south direction by 1.5 miles in an east-west direction. Gold and silver mineralization extends to depths of less than 330 feet in the outcropping to near-outcropping portion of the Bay deposit on the northwest side of the property and to over 1,000 feet in the Brimstone deposit in the eastern portion of the property.

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 MINERALIZATION

9.1 ALTERATION AND MINERALIZATION IN THE EAST FAULT – HANGING WALL

Highly altered rocks are almost exclusively found in the hanging wall of the East Fault. There are at least four main alteration events that have affected the hanging wall rocks. These alteration events occurred in the following sequence.

- Barren silica-pyrite and gold-bearing chalcedony-pyrite-marcasite replaced volcanic rocks on the west side of the East Fault. This original hypogene alteration and mineralization formed approximately four million years ago. The East Fault most likely served as a conduit for hydrothermal fluids.
- A sulfur-rich hydrothermal system developed along the East Fault approximately 400,000 to 2 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.
- Most recently, supergene oxidation of acid leach, oxide and sulfide mineralization has occurred along the East Fault. This was accompanied by a small amount of normal movement along the Fault, displacing mineralization in the hanging wall downward.

Each alteration and mineralization type is described below in detail.

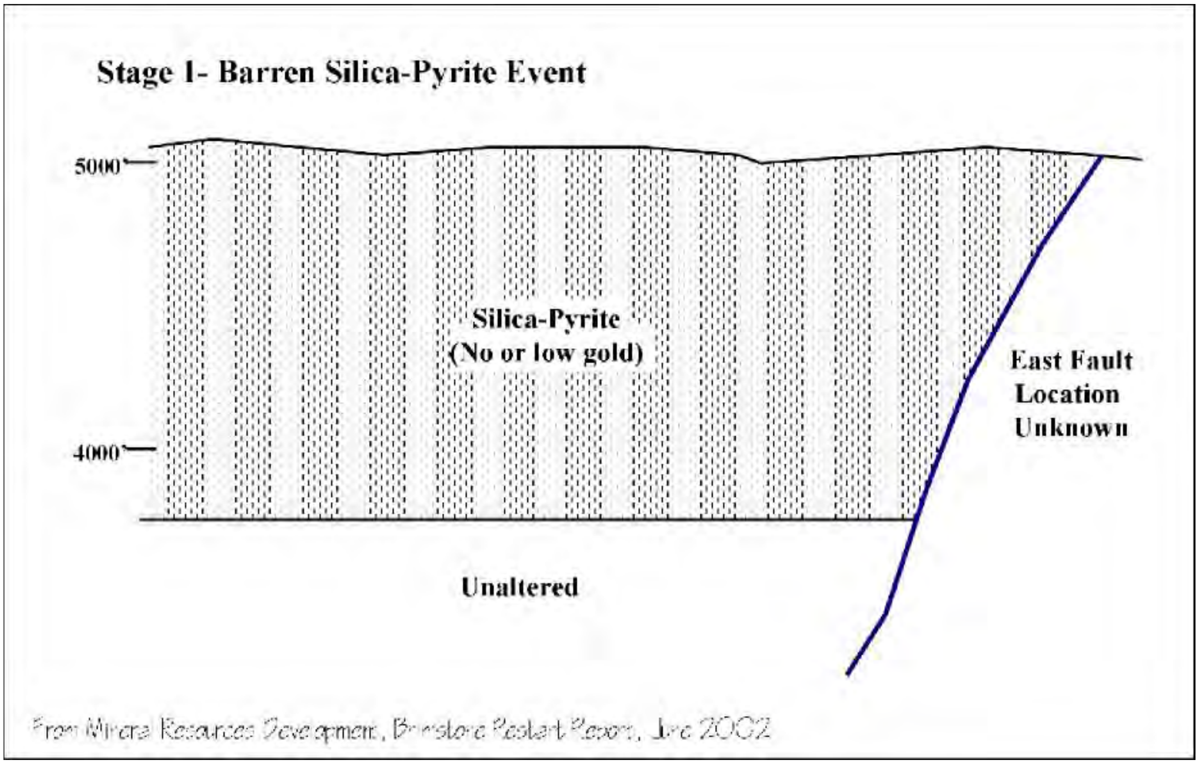
9.1.1 BARREN DISSEMINATED SILICA-PYRITE

The first alteration event was a widespread event of barren silica-pyrite alteration, and was logged as Alteration Code 1. The rocks have a glassy appearance, resulting from strong, fine-grained, disseminated silicification that permeates the rock mass. Fine-grained, euhedral to subhedral pyrite is always associated with this alteration. The pyrite forms 2 to 5% of the rock as fairly uniform grains about 0.2 to 0.5 mm in size. This early phase of pyrite is bright yellow to brassy and is evenly distributed throughout the rock mass. Figure 9.1 shows a schematic section of the distribution of this alteration type.

This alteration type is ubiquitous in the Fire & Brimstone-Albert region, extending for at least 6,000 feet along the strike of the East Fault and at least 2,000 feet west of the East Fault. In cross section, the appearance is funnel shaped, with the first occurrence of unaltered volcanic rock being 2,000 feet west of the East Fault at a depth of approximately 500 feet. As the East Fault is approached from the west, the thickness of this alteration type increases. Until recently very few drill holes passed though the lower contact of this alteration type, although drill hole

96-2888, approximately 600 feet west of the East Fault, crosses into unaltered rock at a depth of 1,100 feet.

Figure 9.1 Barren Silica-Pyrite



9.1.2 FRACTURE CONTROLLED CHALCEDONY-PYRITE-MARCASITE MINERALIZATION

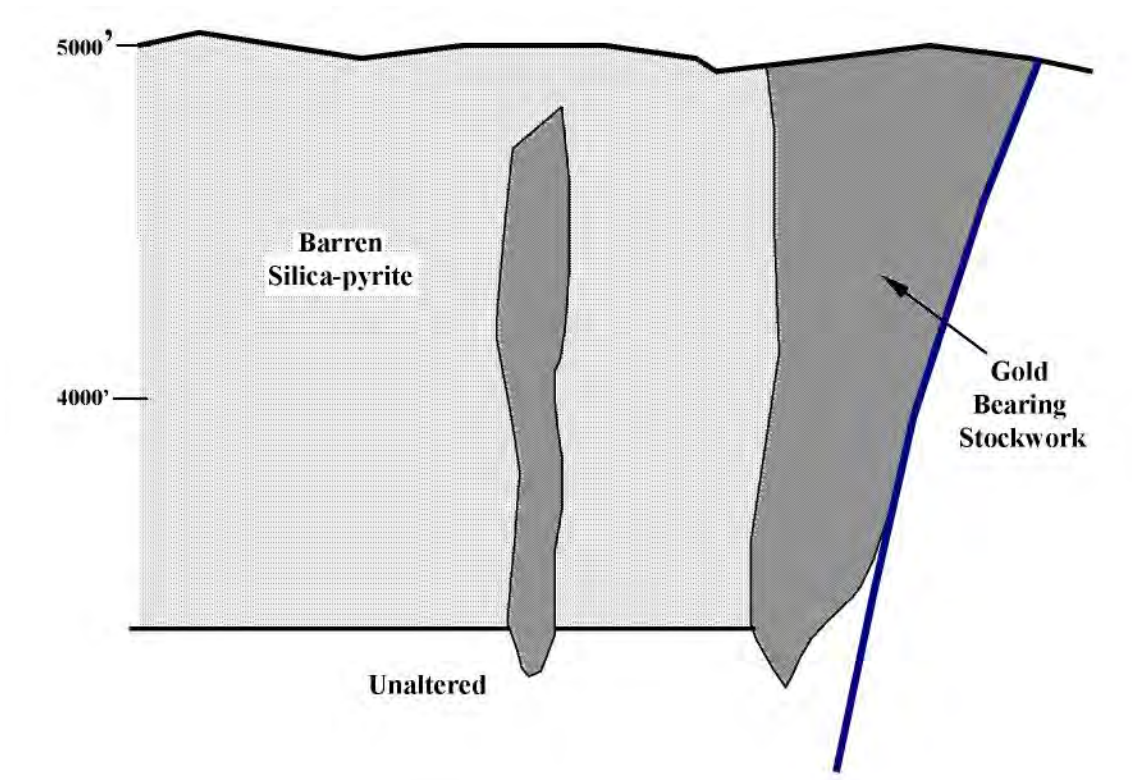
The fracture-controlled chalcedony-pyrite-marcasite mineralizing event was associated with primary gold and possibly silver deposition at Brimstone. Figure 9.2 shows a schematic section of the distribution of this type of mineralization. Mineralization occurs as veinlets, stockworks, in-situ (jig-saw) breccia, and rotational (chaotic) breccia, and was logged as Alteration Code 1, with a Structural Code assigned as described below. This mineralization type clearly crosscuts the earlier barren silica-pyrite alteration, as randomly oriented veinlets, stockwork, in-situ (jig-saw) breccia, or chaotic breccia.

The veinlet mineralization style occurs as 1-mm to 2-cm veinlets forming 2 to 10% of the rock mass. The veinlets are composed of gray to milk-white chalcedony with 5 to 10% sulfides. Chalcedony is rarely banded, but mostly massive. Veinlets were logged as Structural Code 3. Structural Code 6 was used in chips where it was clear that veinlets intersected.

In-situ (jigsaw) breccia shows flooding of the rock fractures with the chalcedony-sulfide assemblage filling a network of fractures. These fractures occupy 5 to 15% of the rock mass; the remaining rock mass can be fit back together, as in a jigsaw puzzle. The in-situ (jigsaw) breccia mineralization was logged as Structural Code 1.

With chaotic breccia, unsorted, angular, wallrock fragments float in a sea of chalcedony-sulfide. Fragments are not aligned and clearly show rotation with respect to adjacent fragments. Breccia mineralization comprises 5 to 20% of the rock mass. Chaotic breccia was logged as Structural Code 2.

Figure 9.2 Schematic Cross Section of Fracture-Controlled Chalcedony-Pyrite Marcasite



The two breccia facies indicate increasing fracture opening and filling by the chalcedony-sulfide mixtures.

FCCPM sulfides are dominated by two species: pyrite and marcasite. Pyrite occurs within the 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 twinned sheaf-like groups of crystals.

As mentioned earlier, gold and possibly silver mineralization were most likely introduced during this event, and evidence for this is two-fold (work remains to be completed for silver mineralization):

- Visible gold (50 to 120 microns in size) has been identified within the chalcedonic veins in thin sections from drillhole 94-2458, and is closely associated with marcasite.
- Assay statistics from RC chip-logging during 1999 show a correlation between FCCPM and gold mineralization. Gold grades by alteration domains are shown in Table 9.1.

Table 9.1 Grade by Domain

Alteration Domain	Avg. FA Au With FCCPM	Avg. FA Au Without FCCPM	% of Domain With FCCPM
Acid leach	0.44	0.25	21
Oxide	0.49	0.16	58
Sulfide	0.47	0.18	57

The data in this table clearly shows that for oxide and sulfide mineralization, the presence of FCCPM correlates with higher gold grades. Samples without the FCCPM-style alteration have average values less than the expected cutoff grade.

The lower percentage of samples observed to contain FCCPM mineralization in the acid-leached rocks is due to:

- The presence of a barren blanket of material above the gold mineralized zone that has been acid-leached, and
- The inability of previous chip loggers to recognize the FCCPM in this highly altered rock-type. The acid-leach alteration obscures the textural evidence of FCCPM.

The presence of gold mineralization in rock units not bearing the FCCPM structural codes can be explained. The FCCPM is only logged when the veinlet concentration is at least 2 to 5% of the rock mass. Lower grade mineralization may simply have an extremely low concentration of veinlets that could not be reliably logged.

The FCCPM mineralization is widespread, but less widespread than barren silica-pyrite alteration. Fracture-controlled mineralization is observed in drill core and chips up to 500 to 1,000 feet west of the East Fault. The north-south extent of this type of mineralization is at least 5,000 feet, from approximately 39,000N to 44,025N. Drillhole 94-2458 intersected this type of mineralization to a depth of 1,000 feet.

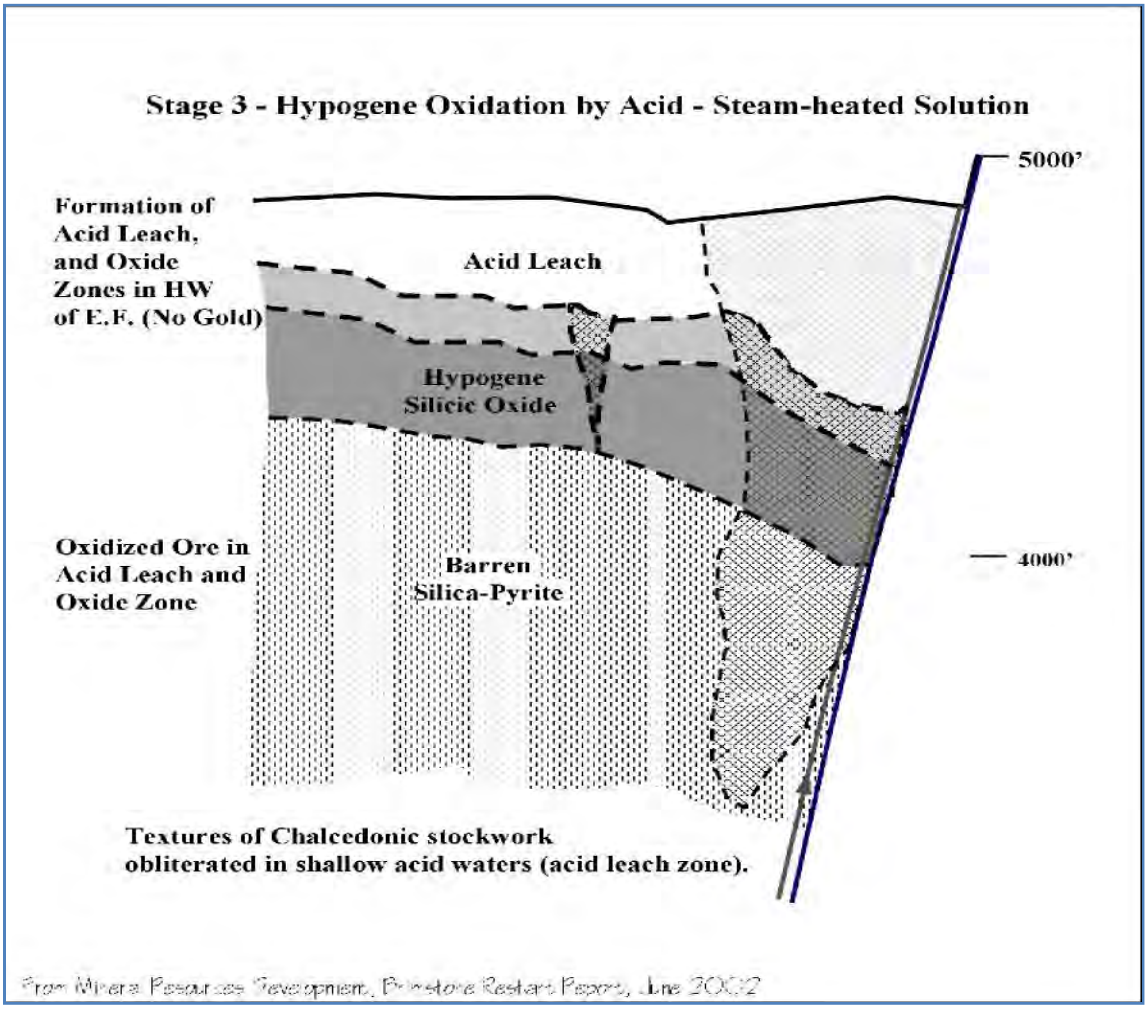
The East Fault clearly cuts FCCPM mineralization, as seen in the bottom of the Brimstone North Pit, evidenced by areas of Fault gouge bearing fragments of this important mineralization type. The best exposures of this type of mineralization are in the bottom three benches of the North Brimstone Pit.

9.1.3 HYPOGENE ACID LEACH OXIDE ALTERATION

The hypogene, 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 9.3 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 as the East Fault is approached. This alteration may be broken into two subtypes, blanket acid-leach and basal acid-leach alteration. Both acid-leach subtypes were logged as Alteration Code 2 under the alteration-coding scheme.

Figure 9.3 Hypogene Oxidation by Acid - Steam Heated Solution



9.1.4 BLANKET ACID LEACH ALTERATION

The dominant blanket acid-leach material covers the entire deposit area and is the uppermost-oxidized alteration-type. On average, blanket acid-leach alteration is 150-200 feet thick over the entire study area, but reaches thicknesses of 450 feet 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. Sizes of the void spaces seen in drill core vary from centimeters to voids of 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 powdery fine-grained alunite or kaolinite, of a few percent at best;

- The absence of iron-bearing minerals, either oxides or sulfides;
- In general, the rock is almost entirely composed of vuggy, fine-grained silica;
- The original textures associated with volcanic deposition are completely obliterated or obscured;
- Accessory minerals are cinnabar, realgar (rare), native sulfur, opal, and gypsum. Native sulfur forms massive veins in acid-leach rock or appears as a disseminated variety when it fills vugs. Native sulfur formation is a late-stage process, with crystals growing into the centers of voids in the already acid-altered wallrock; and
- Blanket acid-leach alteration can be crumbly and incompetent, or hard and competent.

9.1.5 BASAL ACID LEACH ALTERATION

The second form of acid-leach alteration is referred to as basal acid-leach alteration. This form of acid-leach alteration is not as continuous as blanket acid-leach alteration, and is always located at the lower acid-leach/oxide contact.

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 silica;
- Accessory minerals are rare, but native sulfur has been observed;
- Secondary porosity is not as well developed, but occurs as irregular vugs and cavities on the centimeter to decimeter scale; and
- Basal acid-leach-alteration rocks have a conchoidal fracture.

Basal acid-leach alteration is anywhere from 0 to 40 feet thick and horizontal in its lower contact with silicic-oxide alteration. Basal acid-leach alteration was not considered continuous enough to separate as an alteration domain in developing the rock model for Brimstone.

9.1.6 OXIDE ALTERATION

Oxide alteration is composed of two dominant types: silicic-oxide and clay-oxide alteration.

9.1.6.1 SILICIC OXIDE ALTERATION

Silicic-oxide alteration is the dominant type of oxide alteration, forming about 85% of all oxide samples. The silicic-oxide alteration underlies acid-leach alteration and reaches thicknesses of up to 200 feet. The definition used to determine oxide rocks was that at least 25% of the sulfides in a rock had to have been converted to oxides. In the majority of oxide mineralization, all sulfides have been converted to oxides.

Silicic-oxide, as observed in chip trays, is generally fine grained and glassy appearing, with little or no secondary porosity development. Iron oxides, sulfates, and hydroxides are common accessory minerals with the most prevalent oxide being hematite. Other accessory iron-bearing phases include limonite and jarosite. Jarosite most often occurs as amber, euhedral crystals, 1 to 2 mm in size, as fracture coatings and late veinlets. Red, earthy hematite is generally seen replacing pyrite or marcasite. Fine fracture-networks can be observed, often 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. Specular hematite probably results from iron phases being precipitated after being leached from the overlying acid-leach material.

Silicic oxide alteration 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. Silica oxide was coded as XX101X for Alteration Code under the computer logging system. Silicic-oxide is composed of 65 to 85% silica, 5 to 20% clay, and 5 to 15% hematite and jarosite.

9.1.6.2 CLAY OXIDE

Clay-oxide alteration makes up about 15% of material classed as oxide, and represents a more clay-rich zone. Clay zones appear white to yellow to pinkish and are composed of 50% or more clay, with the usual accessory iron oxides. Clays are thought to be mixtures of montmorillonite and kaolinite with accessory alunite.

Clay zones are most common as a layer 30 to 50 feet thick, directly beneath basal acid-leach alteration or as irregular veins or amoeboid-shaped areas scattered throughout the silica-oxide alteration. Clay-oxide alteration is thought to be an intermediate oxidized composition between pure acid-leach and silica-oxide alteration, representing formation under weakly acid-oxidizing conditions.

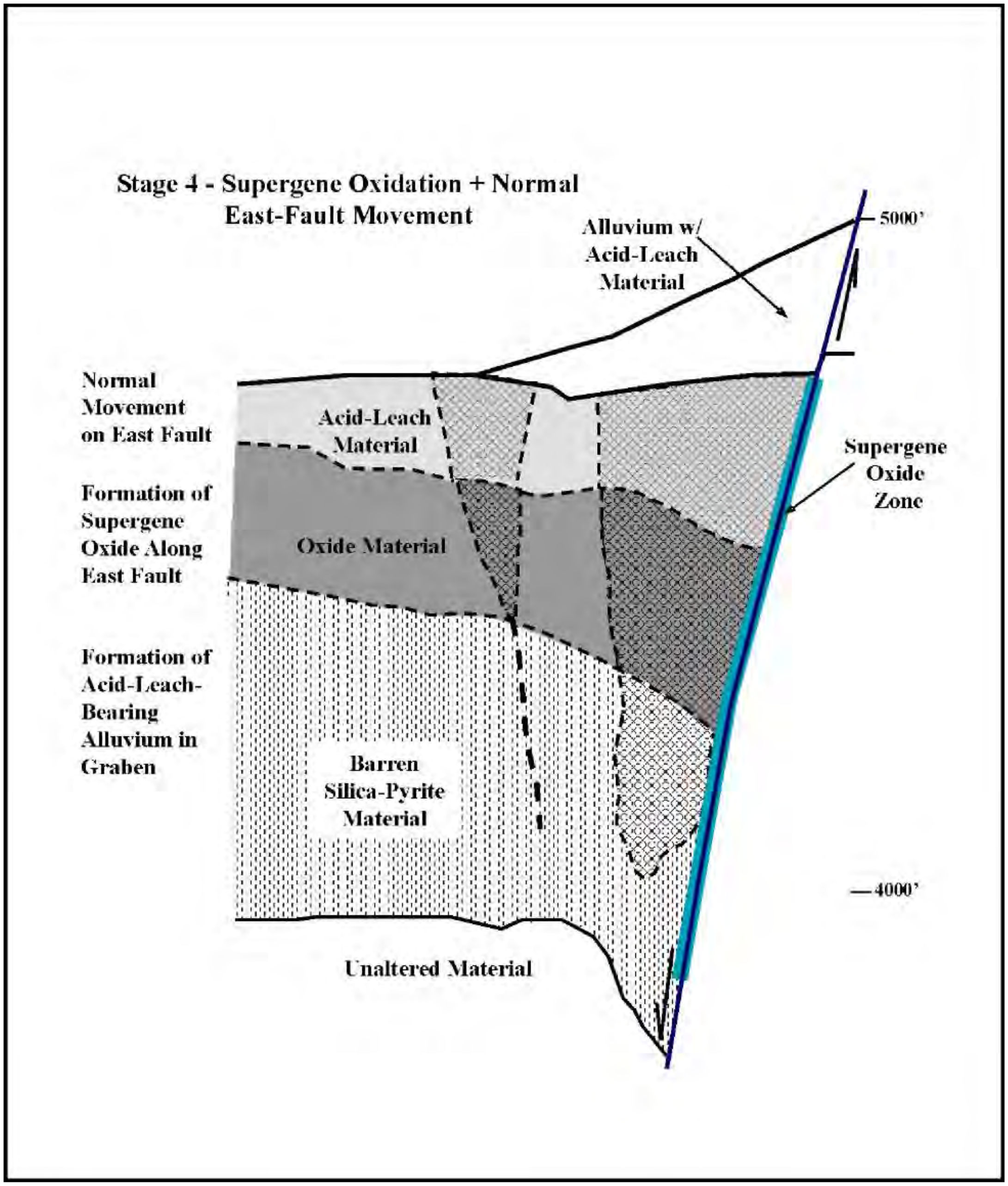
Clay-oxide alteration was coded as XX601X for the Alteration Code in the computer coding system. Clay-oxide alteration was not continuous enough to be separated as a separate alteration domain.

9.1.7 SUPERGENE OXIDATION AND FAULT GOUGE ALTERATION

Supergene oxidation and fault gouge is a zone of oxidation that is literally within the East Fault, and manifests itself as a zone of oxide-stained fault gouge. Figure 9.4 shows a schematic section of the distribution of this alteration. Supergene oxidation was the final alteration event.

The zone appears very similar to silica-oxide alteration, but 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 west-dipping band 20- to 80-feet wide, forming the East Fault-footwall-contact.

Figure 9.4 Supergene Oxidation + Normal Fault Movement



9.2 ZONING OF ACID LEACH AND OXIDE

Oxide mineralization nearly always underlies 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 alteration.

The mineral assemblages in each alteration type and strong geometric zoning suggest that acid-leach alteration and oxide alteration 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 only 2- to 4-weight-percent Al_2O_3, clearly indicating depletion of the aluminum. This depletion requires that the pH of conditions under which acid-leach alteration formed had to be lower than 2.

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-oxide alteration). The upper-level acid fluids were created through oxidation of hydrogen sulfide on reaching the surface, or simply the oxidation of pyrite by surface waters.

10 EXPLORATION

10.1 HISTORIC EXPLORATION AND DEVELOPMENT

Between 1985 and 1999, HRDI drilled a total of 3,123 exploration drillholes, totaling 943,822 ft. Canyon completed 33 drillholes totaling 13,315 ft of reverse circulation drilling during 2005. The current Hycroft drillhole database consists of the former holes, plus 61 RC holes drilled by Homestake in 1982 and 29 rotary holes completed by Homestake in 1981. Drilling completed by the Duval Corporation is not included in the database, but did guide some early exploration. Drilling campaigns are summarized in Table 10.1 by year, operator and drilling type. A breakdown of the drillholes by type and orientation is found in Table 10.2.

Table 10.1 Hycroft Exploration Drill Campaigns

Year	Hole Type	Company	# 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
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	Canyon	33	13,315	Brim, Boneyard
Total			**3246**	**972,702**	

Table 10.2 Exploration Drillholes by Type

Drill Type	Number	Footage
Diamond Drill	27	24,503
RC	3123	938,979
Rotary	29	5,550
Blast	67	3,670
Total	3246	972,702
Angle	1198	
Vertical	2048	

Exploration by Hycroft and Homestake resulted in the discovery of several zones of mineralization. These are briefly described below and are shown in Figure 7.1.

- 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, Cut 5 - a 10,000 foot segment in the immediate hanging wall of the Central Fault. All the deposits are composed of oxidized acid leached Camel Conglomerate.
- 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. The remaining reserves at Hycroft are contained in the southern portions of Brimstone.
- Albert Deposit - located halfway between the Central Fault and the Brimstone deposit.

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 Year	Hole Number	Company	Orientation Of Hole	Present Condition
Cut 4	1977	Duval	Duval	Vertical	Mined
Bay	1981	SR-1	Homestake	Vertical	Mined
South Central	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	To be Mined
Albert	1988	88-1389	Hycroft	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 not substantially outside of previous discovery areas.

10.2 RE-LOGGING AND GEOLOGIC LOGGING CODES

An historic review of the drill logs from holes drilled during the period from 1986 to 1998 on the Brimstone Project led to the following conclusions:

- There were serious problems with the continuity and consistency of logging due to the large number of people involved over the period, their varying levels of experience and expertise, and the lack of a formal written logging scheme;
- The logging was, at times, not based on observation but rather was interpretive. This interpretative method leads to serious problems as the knowledge and understanding of the deposit and the geologic model evolves;
- The generalizations and lack of detail clearly indicate that the loggers did not always use microscopes but rather made broad judgments based on color; and
- When changes in the model occurred and additional features gained importance, samples from the previous drilling were not re-logged. When the drill cuttings were relogged during the 1999 program, it was clear from the condition of the chip trays that they had not been opened since being placed in storage.

Rock types were generally classified as either oxidized or unoxidized felsic-volcanics. This general classification evolved into a logging scheme based on lithology, alteration, and oxidation state that assigned a single numerical value to each five-foot interval as shown in Table 10.4.

Table 10.4 Geological Logging Codes

Code	*Lithology*
1	*Alluvium*
2	*Acid Leach*
5	*Clay*
6	*Quartz Sinter*
7	*Unoxidized Kamma Felsic Volcanics (Footwall)*
8	*Oxidized Kamma Felsic Volcanics (Footwall)*
9	*Unoxidized Felsic Volcanics*
10	*Oxidized Felsic Volcanics*

This scheme effectively combines three separate and distinct geologic parameters (lithology, structure, and ore habit) into a single numerical code. In some cases where this scheme was not in use during the initial logging, codes were later assigned based on the original descriptions or the cyanide-soluble gold-recovery ratio instead of re-logging. The inconsistencies in logging and the grouping of what should be distinct features resulted in inaccurate geologic modeling.

Lacking a formal classification scheme, the classification of material based on the degree of oxidation and alteration (acid-leach, clay-bearing) became completely subjective. This subjectivity leads to inconsistency when numerous people do the logging over the life of a project.

Use of the term quartz sinter is an example of interpretive logging, which was quite misleading. The presence of sinters on the surface, within the district, apparently led to the conclusion that all of the drill intervals, composed mainly of quartz and/or chalcedony were sinters. This assumption is clearly a dangerous and inappropriate conclusion when applied to a deposit with significant occurrences of both quartz and chalcedony veining associated with the mineralization.

In many cases, the presence or absence of pyrite, as support for a conclusion regarding the level of oxidation, could only be determined by use of a microscope. Other rather subjective judgments such as the acid-leach boundary would have been more consistent if a microscope had been used.

The presence of elemental sulfur and its impact on cyanide-soluble assays was recognized rather late in the development of the deposit. Elemental sulfur was observed to depress the cyanide-soluble-gold recovery at the assay level while not significantly impacting the recovery achieved in column testing. An additional code was added to the geologic logs after 1994 to indicate the presence of elemental sulfur. This additional code allowed an upward adjustment of the cyanide-soluble-gold assays to be made which accounted for the artificial depression of the assays. However, despite the importance of this feature, little or no attempt was made to refine logs from earlier drilling. The level of detail in the written logs was insufficient with many having no reference to the presence of sulfur. During the 1999 re-logging program, the number of samples with gold grades greater than or equal to 0.005 opt observed to contain elemental sulfur totaled 1,045 compared to only 85 samples recorded in the old database. This difference contributed significantly to the underestimation of reserves.

As an integral part of the reevaluation of the deposit, two experienced Vista geologists were assigned to re-log all of the available drill chips and core. Prior to the start of logging, a new classification system including five fields was developed. This classification system included fields for lithology, structure (ore habit), alteration, presence or absence of sulfur and/or sulfides, and degree of oxidation.

Approximately half way through the re-logging, an additional field was added to record an estimate of the percent of sulfur. Samples that had already been re-logged were reexamined and the percent sulfur recorded. The logging system was designed to insure consistency and is shown in Table 10.5. The more detailed logging system with each field representing an independent geologic parameter allows for more refined interpretation and better geologic modeling.

Table 10.5 Lithological, Structure, Alteration, Sulfur and Oxidation Codes

Code	Lithology	Code	Structure	code	Alteration
1	Alluvium	1	No Structure	1	Unaltered
2	Gouge Material	2	Jig-Saw Breccia	2	Silicic; Quartz, Chalcedony, K-Feldspars
3	Und. Felsic Volcanics	3	Chaotic Breccia	3	Acid Leach
4	Rhyolite	4	Fractured Zone	4	Propylitic
5	Flow Banded Rhyolite	5	Fault, Gouge, Shear Zone	5	Argillic
6	Rhyolite Tuff	6	Voids	6	Calcite
7	Epiclastic Tuff	7	Stockwork	7	Clay
8	Crofoot Breccia	8			
9	Mafic Volcanics	9	Quartz Vein > 1'		
10	Auld Lang Syne Sedimentary Group Or	10	Calcite Vein > 1'		
		10	Gypsum in Acid Leach		
99	Data Missing	99	Data Missing	99	Data Missing
code	**Presence or Absence of Sulfur and or Sulfide**	Code	**Oxidation State**	Code	**% Native Sulfur (Observed)**
1	No Sulfur and Sulfide	1	< 25 % of the sulfide oxidized	1	Trace
2	Sulfide	2	> 25 % of the sulfide oxidized	2	<5%
3	Sulfur			3	> 5%
4	Sulfur and Sulfide			4	> 10%
9	Data Missing	9	Data Missing	99	Data Missing

All logging was done with the aid of binocular microscopes and the geologists assigned to logging frequently compared notes and chip trays to insure consistency. New codes were recorded on paper log-sheets for each five-foot interval during the re-logging. Holes were grouped by section for logging to insure geological continuity. The geologists responsible for the logging entered the codes into a new database.

The 1999 re-logging program led directly to the recognition of several new geological units, a better understanding of the temporal relations between mineralization and alteration, a better understanding of the structural environment, and a more accurate geological model of the deposit that is still in use today.

10.3 SURVEYING

Drillholes are surveyed in UTM coordinates and converted to Nad27 State Plane coordinates.

10.3.1 Drill Collar Surveys

Standard operating procedure is to lay out planned exploration drill-hole locations by GPS. After drilling is completed on a site, the actual drill-hole location is surveyed with Trimble GPS, and the survey data is entered into the collar file.

10.3.2 Down Hole Surveys

In the past, down-hole surveying of exploration holes was not carried out on a routine basis. During the 1999 drilling-program, down-hole, multi-shot, gyro surveys were done on several of the holes. Results of this work have not shown significant deviations and thus do not indicate that the lack of down-hole surveys in the bulk of the exploration holes poses a problem. All down-hole survey data which is available has been entered into the database. Currently, downhole surveys are carried out on any hole greater than 900 feet in depth.

11 DRILLING

Drilling programs at Hycroft were intended to verify the nature and extent of mineralization. The majority of samples were collected on 5 foot sample intervals. This method of sample collection does not indicate the true thickness of any mineralization at Hycroft.

11.1 HISTORY OF DRILLING AND SAMPLING IN THE FIRE & BRIMSTONE/ALBERT AREA

Reverse circulation drilling of the Fire & Brimstone deposit through 1996 formed the basis for the ore-reserve modeling, and was done with reverse circulation drilling tools utilizing a crossover sub and wet sample-collection. These methods were considered to be standard at the time despite the fact that sample recovery was generally poor due to loss of sample into open spaces in the formation and the potential for down-hole contamination.

In a deposit such as Brimstone, where the fine fraction contains a disproportionately high portion of the gold, poor sample recovery is likely to introduce a low bias into analytical results due to the preferential loss of fines. This bias will be exacerbated if a rigorous sample-collection protocol that insures collection of the entire sample prior to splitting is not followed. Anecdotal evidence suggests that the sample collection protocol employed during the earlier drilling was not sufficiently rigorous in that the sample containers were allowed to overflow during drilling. When sample containers are allowed to overflow, a portion of the very-fine sample-fraction is in suspension and is lost. Also, finely divided sulfides may float off and be lost. Down-hole contamination may result in either a low sample bias when unmineralized material from the upper portion of the hole falls into the mineralized sample intervals or a high bias when higher grade material drifts down-hole below the mineralized zone. The higher than projected production, both tonnage and grade, from North Brimstone suggests that the primary sampling problems during drilling were a combination of contamination of the ore zone with low-grade material and the loss of higher grade fine material.

Modest diamond drilling programs were implemented in 1993 and 1999. The 1993 program was carried out to obtain metallurgical samples through drilling of four PQ-size (3.345") core holes that twinned earlier reverse-circulation holes. The 1999 program was designed to provide both twin-hole information and to fill in some gaps. The twin holes were drilled to test the hypothesis that the earlier reverse-circulation drilling had understated the ore grades. The 1999 program resulted in four twin-holes in the ore zones drilled with HQ-size (2.5") core. These programs both indicated that the previous reverse-circulation programs understated the grade of the deposit.

11.2 1999 TWIN DRILLING PROGRAM

After reviewing the results from the diamond drill twins, it was clear that additional twin drilling was necessary to better quantify possible understatements of resources and reserves in the remaining southern portion of the deposit. After consideration of the problems associated with the diamond drilling and the improvements in reverse-circulation drilling and sampling techniques, the decision was made to implement a new reverse-circulation twin-hole program. Another significant consideration in this decision was the larger sample volume that can be generated during reverse-circulation drilling. A nominal 5.5” reverse-circulation drillhole generates approximately 4.84 times the sample volume of HQ core and 2.70 times that of PQ core.

A 10-hole reverse-circulation twin-hole program was planned with the prospective drill sites selected to provide a representative sampling of two ore types, acid-leach and oxide, with and without elemental sulfur. The sites selected were spread out over the strike and width of the deposit and twinned earlier holes drilled in several different years. 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.

In order to insure the best possible sample recovery, the decision was made to carry out the drilling program with center-return tools, and without water injection. It was recognized at the time that this would result in extremely difficult drilling due to the abrasive, caving ground and the inability to maintain a well-conditioned hole.

Contractor selection was considered to be of critical importance for the planned reverse-circulation twin holes. The most important criteria in the selection process were the availability of an appropriate drill rig, the ability to supply specialized sampling equipment, and the level of cooperation and support which would be necessary to carry out the program under the difficult conditions anticipated. Lang Exploratory Drilling was selected based on these criteria, and proved to be a very good choice in that the program was completed despite conditions that were even more difficult than anticipated.

The drill rig used was a D-40K modified for angle-hole drilling and equipped with 750-cfm/300-psi air supply. The dry-sample collection system provided consisted of three cyclones in series with a filter on the final exhaust. Center-return tools included both tri-cone and hammer systems.

The first two holes were started with a skirted tri-cone bit. Drilling proved to be extremely slow due to the rather low penetration-rate and caving ground that necessitated excessive back reaming. Also, sample recovery while drilling with the skirted tri-cone bit did not appear to be satisfactory. During drilling of the second hole, the tools were changed over to a center-return hammer. An immediate improvement was noticed in sample recovery when this change was made, but the drilling remained extremely difficult. During the balance of the program, one string of pipe with the bit and hammer was lost, one hammer was stuck and actually pulled apart, and two hammers were completely worn out.

The program as completed totaled 5,545 feet of drilling in 11 holes. Seven of the holes were completed to the planned depth, two were abandoned early after reaching a depth sufficient to test the target, and two were abandoned prior to testing the target horizon. Table 11.1 summarizes the drilling completed during the program.

Each hole was started with conventional rotary tools, drilling 10 to 40 feet prior to setting the surface casing. During this phase of the hole, samples were collected on five-foot intervals by setting buckets around the drill string. All of the holes were collared above the mineralized zones, so these samples had no effect on the resource estimates.

Table 11.1 1999 Twin RC Drilling Campaign

Hole	Northing	Easting	Elevation	Attitude	Planned Depth	Actual Depth	Comments
99-2648	40,428.46	22,344.84	5,037.97	-90	700	305	Abandoned, hammers dead
99-1975	41,843.76	23,284.69	4,960.24	-90	600	545	Stuck, Shot Rods
99-1432	42,012.88	23,465.51	4,959.92	-90	500	500	TD
99-1950	41,424.90	22,805.34	4,961.38	-90	500	485	Tight, called TD
99-1419	42,157.92	23,084.14	4,991.56	-90	650	665	TD
99-1504	41,418.09	22,402.90	4,956.37	-90	700	655	Bits worn out, called TD
99-1523	40,788.09	22,529.26	4,965.98	-90	550	435	Twisted hammer off
99-1378	41,091.08	22,441.71	4,937.40	-90	650	650	TD
99-1976	40,524.15	22,376.59	5,027.97	-90	600	600	TD
99-1944	41,615.89	23,148.27	4,961.00	-70 E	550	250	Scrap iron in hole
99-1949	41,744.94	23,290.13	4,961.22	-70 E	450	455	TD

The sample-collection system employed after setting the surface casing consisted of a triple-cyclone setup with an air filter on the final exhaust. This setup insured that virtually the entire sample return was collected with a minimum of fugitive dust. All of the sample return was collected at the rig on five-foot intervals using non-porous plastic bags. Initially, the fine material discharged from the third cyclone was collected separately. However, the amount of fine material actually recovered from the third cyclone was quite small, so it was combined with the coarse material after the first hole.

Standard practice during the drilling was to pull back at the end of the sample interval, allow the sample to clear the inner tube, then open the cyclones and collect the sample. There was no sample volume reduction or splitting carried out at the rig. After the sample was collected, the cyclones were left open and the hole cleaned out prior to the drill string returning to the bottom. When the hole was clean, the cyclones were closed and drilling resumed.

Table 11.2 shows the comparison of fire assays of the twin-hole program, while Table 11.3 shows the comparison of cyanide soluble assays. The 1999 drill results generally indicated higher grades than the older drillhole assays.

Table 11.2 Comparison of the Twin Drillholes (Fire Assays – Imperial Units)

1999 Drillhole	Interval	Feet	Fire oz Au/ton	Cn Soluble oz Au/ton	Old Drillhole	Fire oz Au/ton	Cn Soluble oz Au/ton
99-1378B	180-410	230	0.015	0.012	88-1378	0.013	0.009
99-1419B		565	0.009		89-1419	0.008	
99-1432B	0-565	475	0.017		89-1432	0.009	
99-1504B		570	0.014		90-1504	0.012	
99-1523B	25-500	340	0.016		90-1523	0.017	
99-1944B		250	0.003		92-1944	0.005	
99-1949B	30-600	410	0.013		92-1944	0.011	
99-1950B		405	0.018		92-1950	0.014	
99-1975B	30-370	470	0.027		92-1975	0.023	
99-1976B		405	0.016		92-1976	0.020	
99-2648B	0-250	205	0.004		95-2648	0.004	
Totals		**4,325**	**0.015**			**0.013**	

Table 11.3 Comparison of the Twin Drillholes (Cyanide Soluble Assays)

1999 Drillhole	Interval	Feet	Fire oz Au/ton	Cn Soluble oz Au/ton	Old Drillhole	Fire oz Au/ton	Cn Soluble oz Au/ton
99-1378B	180-410	230	0.015	0.012	88-1378	0.013	0.009
99-1419B		235		0.010	89-1419		0.010
99-1432B		220		0.024	89-1432		0.014
99-1504B	330-565	475		0.009	90-1504		0.006
99-1523B		185		0.021	90-1523		0.027
99-1944B	240-460	250		0.002	92-1944		0.003
99-1949B		410		0.013	92-1944		0.011
99-1950B		405		0.014	92-1950		0.010
99-1975B	125-600	470		0.022	92-1975		0.018
99-1976B		405		0.012	92-1976		0.007
99-2648B		205		0.002	95-2648		0.001
Totals		**3,490**		**0.013**			**0.010**

11.3 CANYON RESOURCES 2005 DRILLING

Canyon completed 33 reverse circulation drillholes using center return bits to improve sample recovery, however the center return hammer broke, and a normal reverse circulation interchange was used for the last four Canyon drillholes.

11.4 DRILL SAMPLE RECOVERY

11.4.1 Pre 1999 Drilling

Prior to the 1999 drilling, no effort was made to estimate sample recovery during reverse circulation drilling. Anecdotal evidence from several employees who worked in the lab during earlier reverse circulation drilling programs indicates that recovery was rather low. This is based on the number of very small samples received for preparation. Vista estimates that pre-1999 RC drilling achieved sample recoveries in the range of 10 to 15 percent.

Core recovery for the 1993 PQ diamond drilling averaged 86 percent. Given that this drilling was done to obtain metallurgical samples, the recovery was generally inadequate.

11.4.2 1999 Drilling

Average core recovery for eight holes drilled in 1999 was 81 percent, although diligent efforts were made to maximize recovery. Twenty percent of drill runs had recoveries of 60 percent or less. Poor recovery was caused by abrasive and loose acid-leach material. This material, combined with strongly oxidized siliceous mineralization, is likely to contain the highest gold values. In MRDI's opinion, the drilling did not meet industry standards for gold deposits of the Brimstone type.

For the 1999 twin-hole program, sample recovery varied with the type of bit used. The upper, barren portions of the first two holes, drilled with a tri-cone bit, had an average recovery of 32 percent. This was inadequate, given the purpose of the drilling, but is still believed to be more than twice the average recovery in previous RC drilling (RC recovery was not carefully measured previously). For the remainder of the program, RC holes were drilled with a center-return hammer. These obtained an average sample recovery of 63 percent. In MRDI's experience, this is above average for dry-drilled RC holes. The calculation of recovery does not make allowances for the significant number of voids encountered in the acid leach zone and thus is somewhat conservative.

MRDI plotted recovery against fire-assay and cyanide-soluble gold values to evaluate the relationship between recovery and gold grades (Figures 11.1 and 11.2). There are no discernible patterns between recovery and either fire-assay or cyanide-soluble gold values. No relationship can be seen between low recovery in the 1999 RC drilling campaign and low gold values. MRDI believes that this is because the drilling was done dry, preventing a separation of particle sizes by the drilling fluid. In other words, all particle sizes may be affected nearly equally by recovery, and gold grains in fine particles are more adequately represented in samples, regardless of drilling recovery, when drilling is done dry rather than wet.

Figure 11.1 Recovery vs. Chemex AuFA

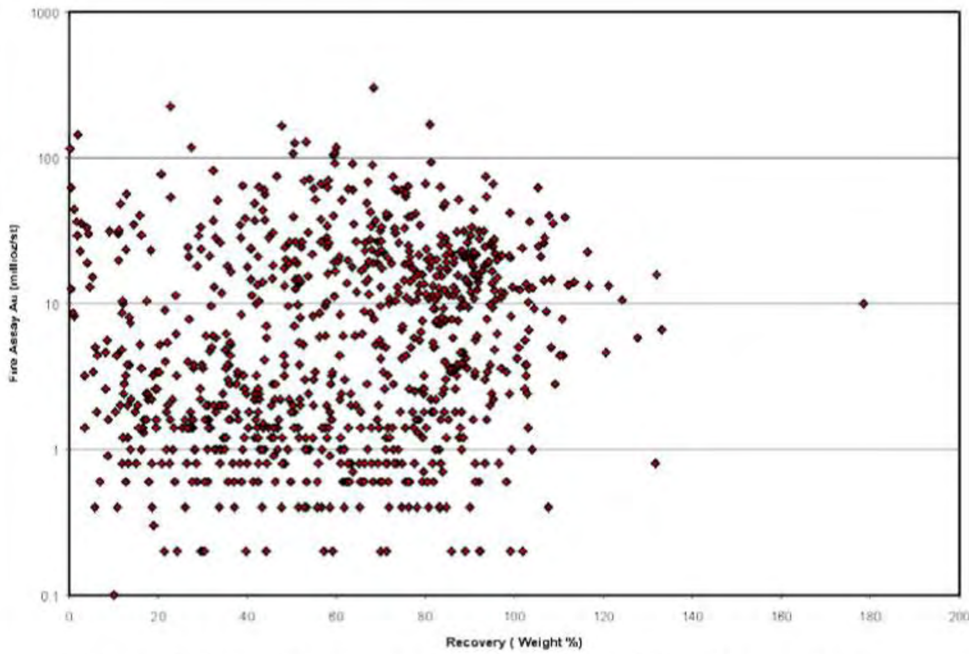


Figure 11.1a Recovery Vs. Chemex Fire Assay Au

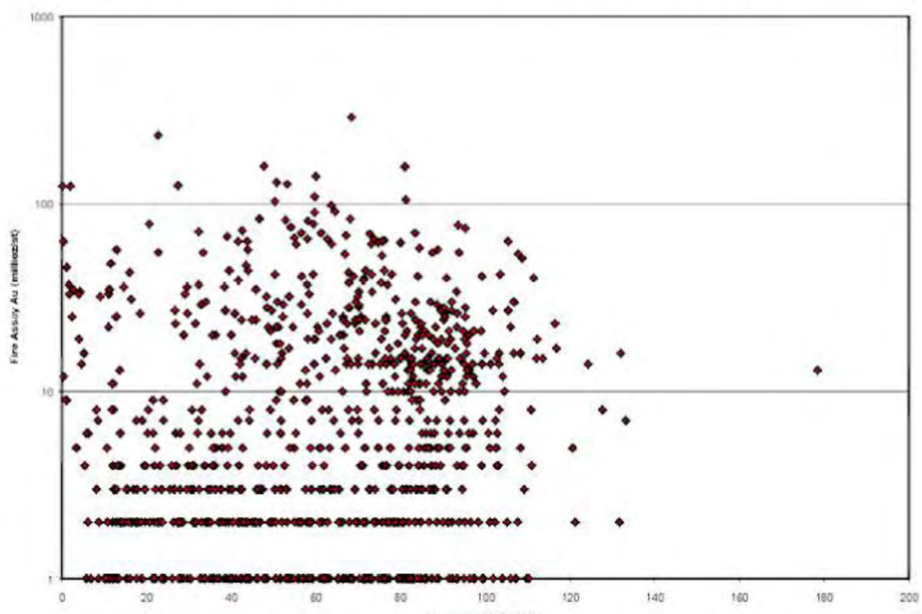


Figure 11.1b Recovery Vs. Hycroft Fire Assay Au

From Mineral Resources Development, Brimstone Restart Report, June 2002

Figure 11.2 Recovery vs. Hycroft AuCN

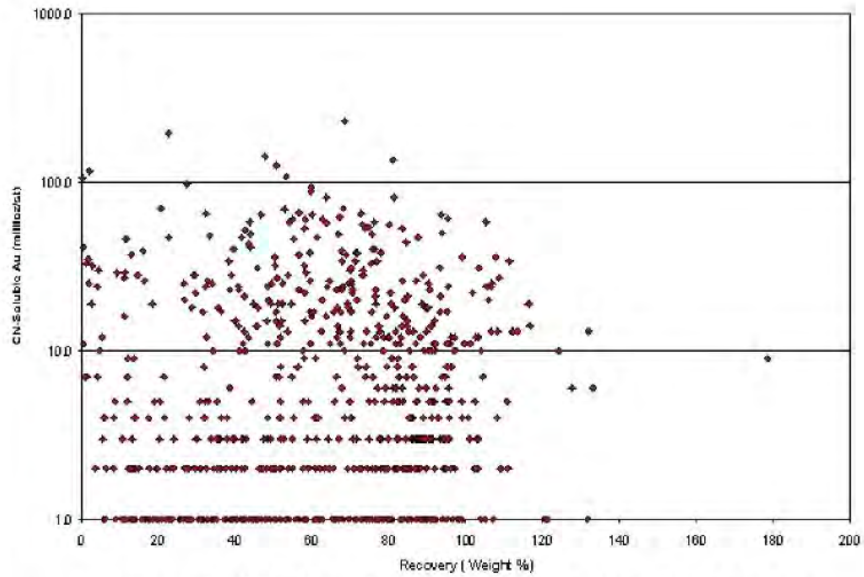


Figure 11.2a Recovery Vs. Hycroft CN-Soluble Au

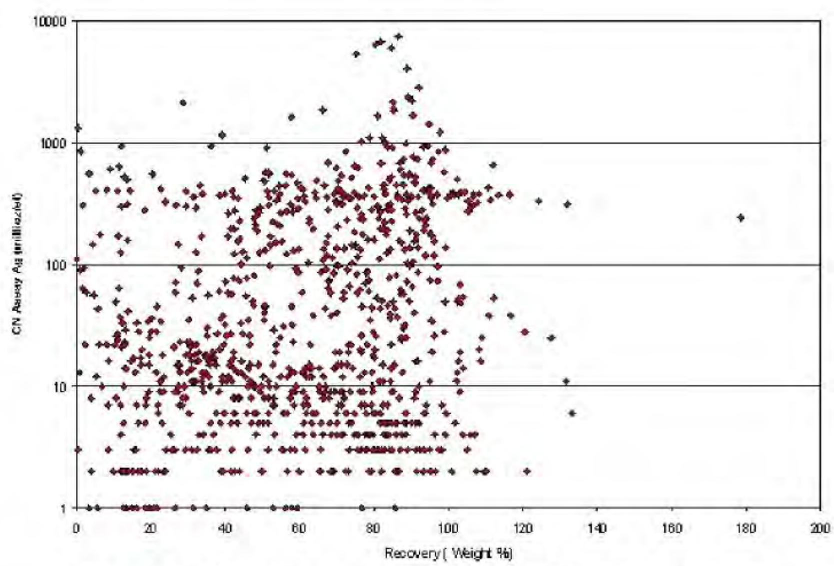


Figure 11.2b Recovery Vs. Hycroft CN-Soluble Ag

From Mineral Resources Development, Brimstone Restart Report, June 2002

11.4.3 Allied Nevada Drilling

Allied Nevada commenced exploration and resource development drilling in August 2007. Since then Allied Nevada has completed 393 drillholes for a total of 300,494 feet drilled on the Hycroft project. Figure 11.3 shows the drilling locations for the drilling program to date. The drilling program focused on the following purposes:

- Oxide delineation
- Wide spaced deep sulfide exploration
- Condemnation drilling for the Hycroft heap leach expansion
- Grade confirmation on the historic Crofoot pad
- Waste dump condemnation

11.4.4 Silver Mineralization in Drillholes

Gold and silver mineralization have been extended at least 1,500 feet south of the southern end of the Central Fault Pit and to at least 2,400 feet south of the Brimstone Pit. Silver contributes a significant value to many mineralized intervals. Allied Nevada conducted a silver fire assay analyses program on 13,168 historical drill pulps, sampled from 185 individual holes (Figure 11.3). Historic drill holes were fire assayed for gold and cyanide soluble analyses were done for gold and silver. Only limited analyses for fire assay silver were completed by previous operators up to 2006.

Historic drill hole sample pulps were identified from some periods of past drilling. 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. Several hundred historical pulps are still available to assay for intervals that have been mined.

SRK Consulting (Elko) managed the collection and submission of the drill hole pulps. SGS Laboratory completed the analyses. Samples were submitted to their Elko prep facility and fire assay analysis was completed in their Toronto laboratory. Standard and blank samples were submitted with the drill hole pulps to evaluate the analytical quality.

The following are summary observations based on the analytical results:

- Based on standards, blanks and comparison to the results of complete silver fire re-assay of historic drillhole 96-3013, analytical results appear to be accurate and reliable.

- Silver is below detection limit (3 ppm) for approximately 76% of the samples.
- Approximately 57% of the holes contain significant silver (≥20 feet of 8 ppm or 0.23 opt) (Figure 11.3).
- Silver values as high as 3240 ppm (94.50 oz/ton) have been received from the laboratory.
- Holes with silver mineralization are located in the Silver Camel, northern Boneyard, and southern Fire and Brimstone zones.

Figure 11.3 Hole Locations and Results of Fire Assay Program – Historic Pulps

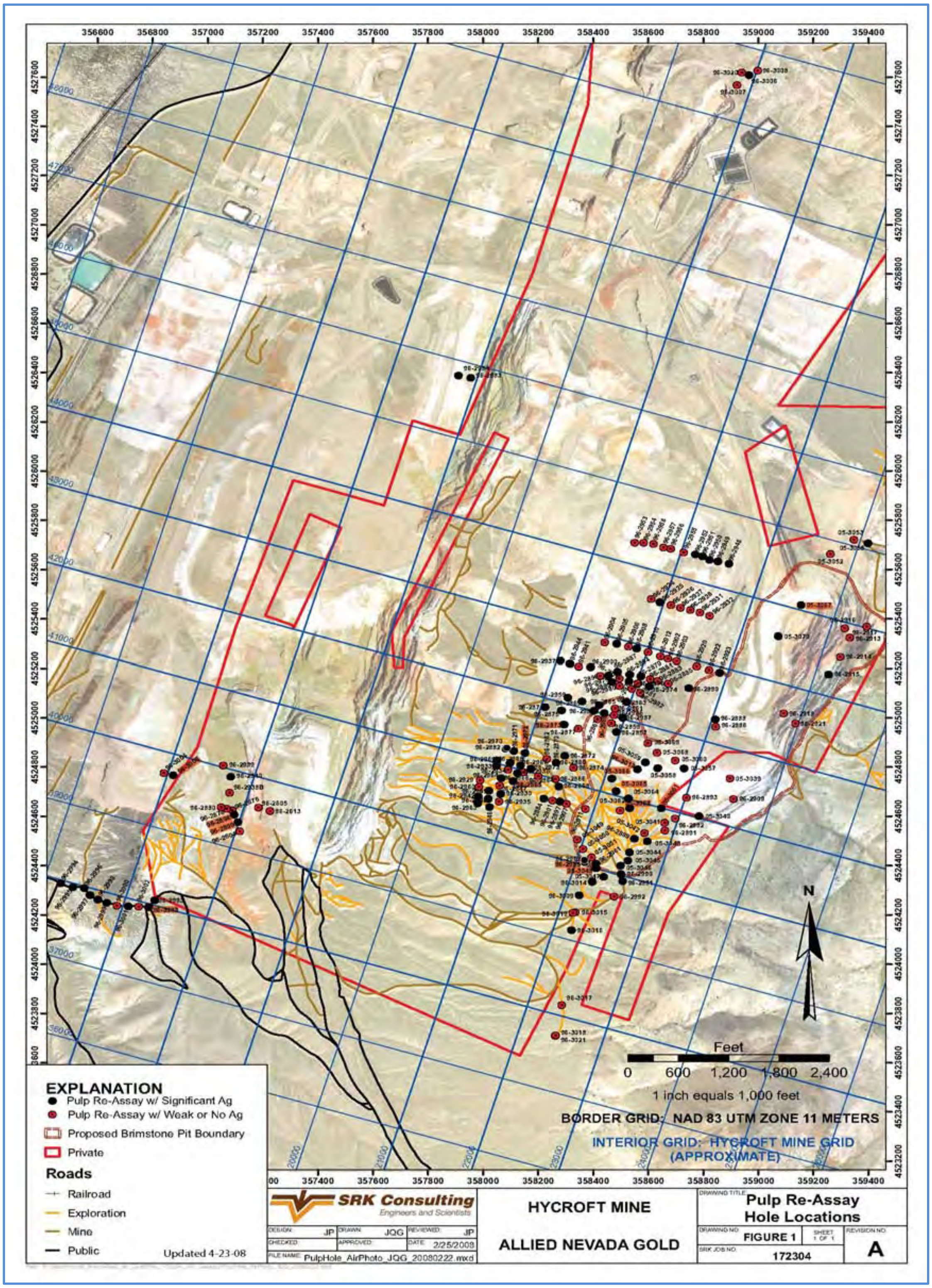


Figure 11.4 Reserves and Resource Development Drill Plan

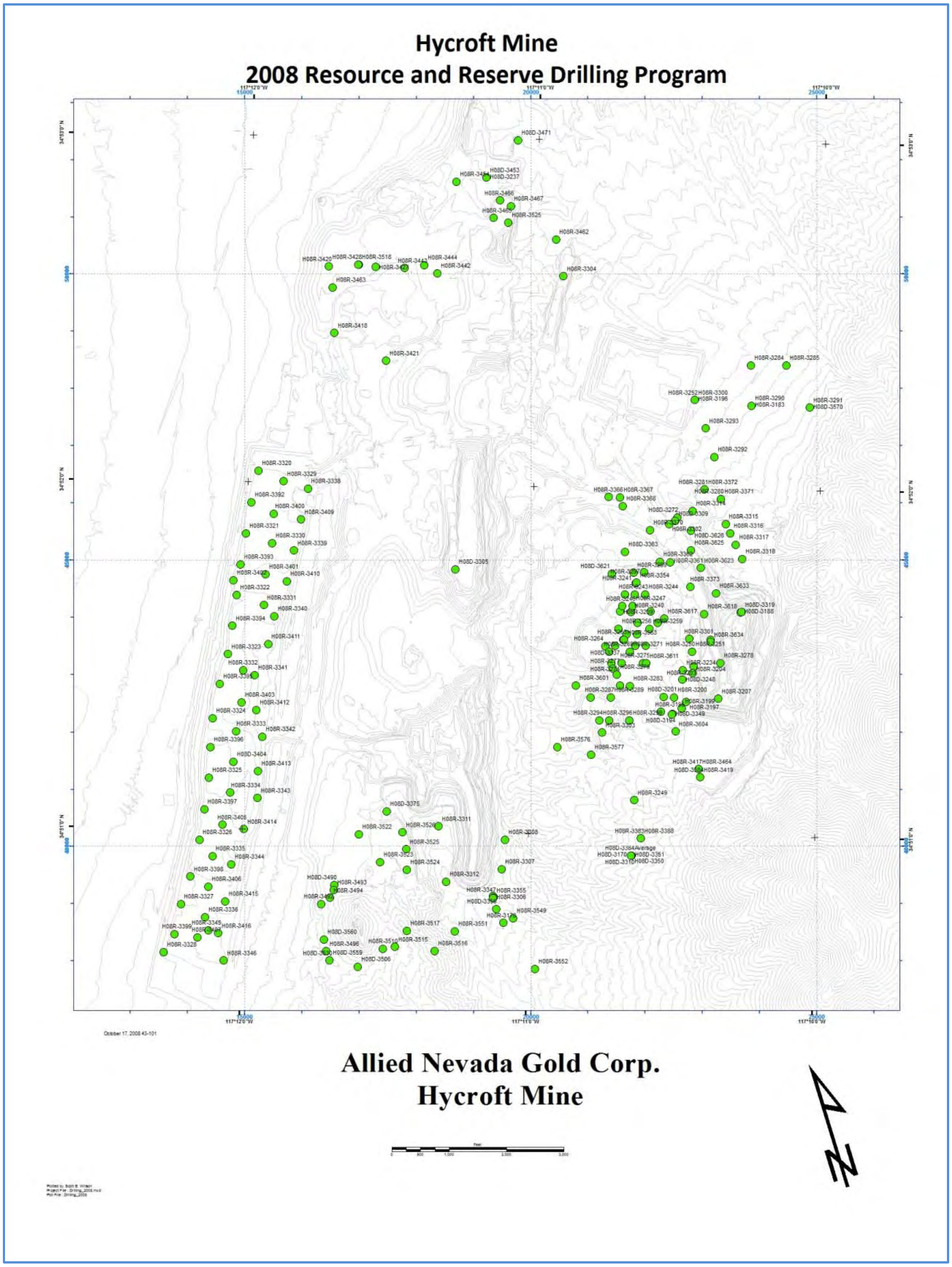


Table 11.4 Allied Nevada Hycroft Drilling Program

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
Hy-06-01 (Vista, 12/06)	C5 – 9	Central-South	Ox & Sulf	0-900	Complete
H07R*[1]-3071	B/S – 7	Brimstone	Ox & Sulf	0-1230	Complete
H07R-3072	B/S – 7	Brimstone	Ox & Sulf	0-1190	Complete
H07D*[2] -3073	B/S – 1	Brimstone	Ox & Sulf	0-1300	Complete
H07R-3074	B/S – 6	Brimstone	Ox & Sulf	0-940	Complete
H07R-3075	C5-8	Central-South	Ox & Sulf	0-1160	Complete
H07R-3076	C5-8	Central-South	Ox & Sulf	0-1040	Complete
H07R-3077	B/S -18	Brimstone	Ox & Sulf	0-1500	Complete
H07D-3078	B/S – 1	Brimstone	Ox & Sulf	0-1429	Complete
H07D-3079	B/S-20	Brimstone	Ox & Sulf	0-1190	Complete
H07R-3080	C5-8	Central-South	Ox & Sulf	0-1160	Complete
H07R-3081	B/S-13	Brimstone	Ox & Sulf	0-1200	Complete
H07R-3082	C5-3	Central-South	Ox & Sulf	0-1120	Complete
H07D-3083	C5-13	Central-South	Ox & Sulf	0-1225	Complete
H07R-3084	B/S-22	Brimstone	Ox & Sulf	0-1500	Complete
H07R-3085	C5-3	Central-South	Ox & Sulf	0-1075	Complete
H07D-3086	C5-13	Central-South	Ox & Sulf	0-600	Complete
H07D-3087	C5-15	Central-South	Ox & Sulf	0-1330	Complete
H07R-3088	C5-3	Central-South	Ox & Sulf	0-1080	Complete
H07D-3089	C5-15	Central-South	Ox & Sulf	0-673	Complete
H07R-3090	B/S-22	Brimstone	Ox & Sulf	0-400	Complete
H07R-3091	B/S-34	Brimstone	Ox & Sulf	0-1155	Complete
H07R-3092	C5-5	Central-South	Ox & Sulf	0-1200	Complete
H07D-3093	C5-17	Central-South	Ox & Sulf	0-1499.5	Complete
H07R-3094	B/S-19	Brimstone	Ox & Sulf	0-1440	Complete
H07R-3095	C5-5	Central-South	Ox & Sulf	0-1160	Complete
H07R-3096	B/S-19	Brimstone	Ox & Sulf	0-1230	Complete
H07D-3097	C5-17	Central-South	Ox & Sulf	0-1200	Complete
H07R-3098	C5-5	Central-South	Ox & Sulf	0-320	Complete
H07R-3099	C5-16	Central-South	Ox & Sulf	0-1300	Complete
H07D-3100	C5-6	Central-South	Ox & Sulf	0-797	Complete
H07D-3101	C5-2	Central-South	Ox & Sulf	0-1140	Complete
H07R-3102	B/S-30	Brimstone	Ox & Sulf	0-1230	Complete
H07R-3103	C5-26	Central-South	Ox & Sulf	0-975	Complete
H07R-3104	B/S-104	Brimstone	Ox & Sulf	0-1200	Complete
H07R-3105	C5-1	Central-South	Ox & Sulf	0-940	Complete
H07R-3106	C5-28	Central-South	Ox & Sulf	0-1065	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H07R-3107	B/S-102	Brimstone	Ox & Sulf	0-1085	Complete
H07R-3108	C5-32	Central-South	Ox & Sulf	0-1150	Complete
H07D-3109	C5-27	Central-South	Ox & Sulf	0-1160	Complete
H07R-3110	C5-30	Central-South	Ox & Sulf	0-1195	Complete
H07R-3111	B/S-105	Brimstone	Ox & Sulf	0-1220	Complete
H07R-3112	C5-25	Central-South	Ox & Sulf	0-1160	Complete
H07R-3113	B/S-101	Brimstone	Ox & Sulf	0-1200	Complete
H07R-3114	C5-29	Central-South	Ox & Sulf	0-1225	Complete
H07R-3115	CC-006-S	Gap	Ox & Sulf	0-1100	Complete
H07D-3116	YN-004-S	Bone Yard	Ox & Sulf	0-145	Complete
H07R-3117	B/S-110	Brimstone	Ox & Sulf	0-1100	Complete
H07R-3118	B/S-36	Brimstone	Ox & Sulf	0-1200	Complete
H07D-3119	YN-004-S	Bone Yard	Ox & Sulf	0-540	Complete
H07R-3120	B/S-111	Brimstone	Ox & Sulf	0-1200	Complete
H07R-3121	CC-006-S	Central Pit	Ox & Sulf	0-1200	Complete
H07R-3122	B/S-32	Brimstone	Ox & Sulf	0-1200	Complete
H07R-3123	B/S-31	Brimstone	Ox & Sulf	0-1120	Complete
H07R-3124	CC-006-S	Central Pit	Ox & Sulf	0-1200	Complete
H07R-3125	B/S-32	Brimstone	Ox & Sulf	0-650	Complete
H07R-3126	BC-053-O	Brimstone	Oxide	0-650	Complete
H07R-3127	BC-055-S	Brimstone	Ox & Sulf	0-1360	Complete
H07R-3128	BC-052-O	Brimstone	Oxide	0-750	Complete
H07R-3129	BC-071-S	Brimstone	Ox & Sulf	0-1200	Complete
H08D-3130	BC-070-S	Brimstone	Ox & Sulf	0-1858	Complete
H07R-3131	BC-070-S	Brimstone	Ox & Sulf	0-700	Complete
H08R-3132	BC-052-O	Brimstone	Oxide	0-750	Complete
H08D-3133	YN-004-S	Bone Yard	Ox & Sulf	0-1000	Complete
H08D-3134	YN-004-S	Bone Yard	Ox & Sulf	0-1230	Pending
H08R-3135	FC-095-S	Fire & Brimstone	Ox & Sulf	0-1200	Complete
H08R-3136	FC-097-S	Fire & Brimstone	Ox & Sulf	0-1200	Complete
H08R-3137	BS-005-O	Brimstone	Oxide	0-750	Complete
H08R-3138	FS-113-S	Fire	Oxide	0-540	Complete
H08R-3139	FC-013-O	Fire	Oxide	0-750	Complete
H08R-3140	FC-096-S	Fire	Ox & Sulf	0-995	Complete
H08R-3141	FC-057-O	Fire	Oxide	0-460	Complete
H08D-3142	FC-043-OS	Fire	Oxide	0-468	Complete
H08D-3143	BC-003-S	Brimstone	Ox & Sulf	0-1028	Complete
H08D-3144	FS-114-S	Fire	Oxide	0-390	Complete
H08R-3145	FS-113-S	Fire	Oxide	0-750	Complete
H08R-3146	FC-056-O	Fire	Oxide	0-750	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08R-3147	FC-056-O	Fire	Oxide	0-580	Complete
H08R-3148	FC-057-O	Fire	Oxide	0-750	Complete
H08R-3149	FC-090-S	Fire & Brimstone	Oxide	0-380	Complete
H08R-3150	FC-094-S	Fire & Brimstone	Oxide	0-940	Complete
H08R-3151	BS-006-OS	Brimstone	Oxide	0-900	Complete
H08R-3152	FC-090-S	Fire	Oxide	0-180	Complete
H08R-3153	FS-012-O	Fire	Oxide	0-500	Complete
H08R-3154	BS-004-OS	Brimstone	Oxide	0-650	Complete
H08D-3155	FS-002-O	Fire	Oxide	0-700	Complete
H08R-3156	FS-009-O	Fire	Oxide	0-500	Complete
H08R-3157	BS-007-O	Brimstone	Oxide	0-700	Complete
H08R-3158	FS-026-OS	Fire	Oxide	0-600	Complete
H08R-3159	FS-033-OS	Fire	Ox & Sulf	0-1000	Complete
H08D-3160	FS-025-O	Fire	Oxide	0-500	Complete
H08D-3161	FC-050-O	Fire	Oxide	0-596	Complete
H08R-3162	FS-030-OS	Fire	Oxide	0-860	Complete
H08R-3163	BC-041-O	Fire & Brimstone	Oxide	0-600	Complete
H08R-3164	BC-042-O	Fire & Brimstone	Oxide	0-650	Complete
H08R-3165	FS-024-OS	Fire	Oxide	0-500	Complete
H08R-3166	FS-035-OS	Fire	Oxide	0-305	Complete
H08R-3167	BC-044-O	Brimstone	Oxide	0-750	Complete
H08R-3168	FS-003-O	Fire	Oxide	0-500	Complete
H08D-3169	BC-009-O	Brimstone	Oxide	0-601	Complete
H08D-3170	BS-24	South Brimstone	Sulfide	0-1358	Complete
H08D-3171	BC-010-O	Brimstone	Oxide	0-550	Complete
H08R-3172	BC-043-O	Fire & Brimstone	Oxide	0-530	Complete
H08R-3173	FS-032-OS	Fire	Oxide	0-500	Complete
H08D-3174	FS-037-OS	Fire	Oxide	0-905	Complete
H08D-3175	FS-008-OS	Fire	Oxide	0-856	Complete
H08R-3176	FS-039-OS	Fire	Oxide	0-400	Complete
H08R-3177	FS-040-OS	Fire	Oxide	0-750	Complete
H08R-3178	C5-14	Central-South	Oxide	0-800	Complete
H08R-3180	FS-043-S	Fire	Oxide	0-350	Complete
H08D-3181	FS-044-OS	Fire	Oxide	0-251	Complete
H08R-3182	FS-045-OS	Fire	Oxide	0-520	Complete
H08D -3183	BC-100-C	Condemnation	Oxide	0-155	Complete
H08R-3184	FS-050-O	Fire	Oxide	0-750	Complete
H08R-3185	FS-046-S	Fire	Oxide	0-380	Complete
H08D-3186	FC-078-OS	Fire	Oxide	0-731	Complete
H08R-3187	BC-033-OS	Fire & Brimstone	Oxide	0-300	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08D-3188	BC-061-S	Brimstone	Ox & Sulf	0- 1764.5	Complete
H08R-3189	BC-011-O	Brimstone	Oxide	0-500	Complete
H08R-3190	BC-012-O	Brimstone	Oxide	0-720	Complete
H08R-3191	BC-014	Brimstone	Oxide	0-500	Complete
H08R-3192	BC-022-OS	Brimstone	Oxide	0-400	Complete
H08D -3193	BC-101-C	Condemnation	Oxide	0-693.5	Complete
H08D -3194	FC-091-S	Fire	Ox & Sulf	0-1321	Complete
H08R-3195	FC-092-S	Fire	Oxide	0-800	Complete
H08R-3196	BC-098-C	Condemnation	Ox & Sulf	0-600	Complete
H08R-3197	BC-048-C	Condemnation	Ox & Sulf	0-750	Complete
H08R-3198	BC-032-O	Brimstone	Oxide	0-350	Complete
H08R-3199	BC-046-O	Brimstone	Oxide	0-740	Complete
H08R-3200	BC-045-S	Brimstone	Ox & Sulf	0-1200	Complete
H08D-3201	FC-093-S	Fire	Ox & Sulf	0-1145	Complete
H08D-3203	BC-038-S	Brimstone	Oxide	0-800	Complete
H08R-3204	FC-098-S	Fire	Oxide	0-800	Complete
H08R-3205	FS-021-OS	Fire	Oxide	0-400	Complete
H08R-3206	BC-019-O	Fire & Brimstone	Oxide	0-400	Complete
H08R-3207	BC-047-O	Brimstone	Oxide	0-500	Complete
H08D-3208	BC-054-S	Brimstone	Oxide	0-400	Complete
H08R-3209	BS-002-O	Brimstone	Oxide	0-400	Complete
H08R-3210	BS-10	Brimstone	Oxide	0-500	Complete
H08R-3211	BS-2	Brimstone	Oxide	0-400	Complete
H08R-3212	FC-003-O	Fire	Oxide	0-500	Complete
H08R-3213	FC-005-OS	Fire	Oxide	0-500	Complete
H08D-3214	FC-009-OS	Fire	Oxide	0-600	Complete
H08R-3215	FC-010-OS	Fire	Oxide	0-500	Complete
H08R-3216	FC-001-OS	Fire	Oxide	0-500	Complete
H08D-3217	FC-015-OS	Fire	Oxide	0-1200	Complete
H08D-3218	FC-046-OS	Fire	Oxide	0-600	Complete
H08R-3219	FC-051-O	Fire	Oxide	0-500	Complete
H08R-3220	FC-052-O	Fire	Oxide	0-500	Complete
H08R-3221	FC-062.O	Fire	Oxide	0-500	Complete
H08R-3222	FC-066-O	Fire	Oxide	0-120	Complete
H08R-3223	FC-067-O	Fire	Oxide	0-600	Complete
H08R-3224	FC-068-O	Fire	Oxide	0-500	Complete
H08D-3225	FC-072-O	Fire	Oxide	0-627	Complete
H08D-3226	FC-079-O	Fire	Oxide	0-600	Complete
H08R-3227	FC-084-O	Fire	Oxide	0-400	Complete
H08R-3228	FC-089-O	Fire	Oxide	0-500	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08R-3229	FS-006-S	Fire	Oxide	0-355	Complete
H08R-3230	FS-008-O	Fire	Oxide	0-500	Complete
H08R-3231	FS-010-OS	Fire	Oxide	0-360	Complete
H08R-3232	BC-004-OS	Brimstone	Oxide	0-400	Complete
H08R-3233	BC-003-OS	Brimstone	Oxide	0-400	Complete
H08R-3234	BC-037-0S	Fire & Brimstone	Ox & Sulf	0- 800	Complete
H08R-3235	FS-108-S	Fire	Oxide	0-360	Complete
H08R-3236	FC-002-OS	Fire	Oxide	0-700	Complete
H08R-3237	CN-004-S	North Central	Ox & Sulf	0-1398	Complete
H08R-3238	FC-019-OS	Fire	Oxide	0-600	Complete
H08R-3239	FC-016-OS	Fire	Ox & Sulf	0-650	Complete
H08R-3240	FC-017-OS	Fire	Oxide	0-490	Complete
H08R-3241	FC-004-OS	Fire	Oxide	0-160	Complete
H08R-3242	FC-006-OS	Fire	Oxide	0-600	Complete
H08R-3243	FC-007-OS	Fire	Oxide	0-500	Complete
H08R-3244	FC-008-OS	Fire	Oxide	0-600	Complete
H08R-3245	BC-024-OS	Brimstone	Ox & Sulf	0-600	Complete
H08R-3246	FC-012-OS	Fire	Ox & Sulf	0-650	Complete
H08R-3247	FC-013-OS	Fire	Ox & Sulf	0-800	Complete
H08D-3248	CN-017-S	North Central	Ox & Sulf	0- 836	Complete
H08R-3249	FS-041-OS	Fire	Oxide	0- 740	Complete
H08R-3250	BC-030-S	Brimstone	Ox & Sulf	0-1200	Complete
H08R-3251	BC-031-S	Brimstone	Ox & Sulf	0-505	Complete
H08R-3252	BC-098-C	Condemnation	Ox & Sulf	0-1200	Complete
H08R-3253	FC-023-OS	Fire	Oxide	0-600	Complete
H08D-3255	FC-025-OS	Fire	Oxide	0-600	Complete
H08R-3256	FC-026-OS	Fire	Ox & Sulf	0-800	Complete
H08R-3259	FC-029-OS	Fire	Ox & Sulf	0-1100	Complete
H08R-3260	FC-030-S	Fire	Ox & Sulf	0-800	Complete
H08R-3261	FC-031-OS	Fire	Ox & Sulf	0-1200	Complete
H08R-3263	FC-033-S	Fire	Ox & Sulf	0-695	Complete
H08R-3264	FC-034-OS	Fire	Oxide	0-650	Complete
H08D-3265	FC-035-OS	Fire	Ox & Sulf	0-1757	Complete
H08R-3267	FC-037-OS	Fire	Ox & Sulf	0-865	Complete
H08R-3268	FC-038-OS	Fire	Oxide	0-1100	Complete
H08R-3269	FC-045-OS	Fire	Oxide	0-750	Complete
H08R-3271	FC-041-S	Fire	Ox & Sulf	0-800	Complete
H08D-3272	BC-09-S	Condemnation	Ox & Sulf	0-200	Complete
H08R-3273	FC-044-OS	Fire	Oxide	0-280	Complete
H08R-3274	FC-045-0S	Fire	Oxide	0-280	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08D-3275	FC-047-OS	Fire	Oxide	0-600	Complete
H08D-3276	FS-044-O	Fire	Oxide	0-319	Complete
H08R-3277	FC-049-O	Fire	Oxide	0-625	Complete
H08R-3278	BC-036-O	Brimstone	Oxide	0-500	Complete
H08R-3279	FC-054-O	Fire	Ox & Sulf	0-650	Complete
H08R-3280	BC-072-S	Condemnation	Ox & Sulf	0-1200	Complete
H08R-3281	BC-073-OS	Condemnation	Ox & Sulf	0-1000	Complete
H08R-3282	FC-060-0	Fire	Oxide	0-600	Complete
H08R-3283	CN-018-S	North Central	Ox & Sulf	0-1000	Complete
H08R-3284	BN-014-C	Condemnation	Ox & Sulf	0-1000	Complete
H08R-3285	BN-015-C	Condemnation	Ox & Sulf	0-1000	Complete
H08R-3287	FC-069-OS	Fire	Ox & Sulf	0-650	Complete
H08R-3289	FC-071-0	Fire	Oxide	0-550	Complete
H08R-3290	BC-100-C	Condemnation	Ox & Sulf	0-1000	Complete
H08D-3291	BC-101-C	Condemnation	Ox & Sulf	0-1049	Complete
H08R-3292	BC-096-C	Condemnation	Ox & Sulf	0-1000	Complete
H08R-3293	BC-097-C	Condemnation	Ox & Sulf	0-1000	Complete
H08R-3294	FC-080-S	Fire	Ox & Sulf	0-670	Complete
H08R-3296	FC-081-S	Fire	Ox & Sulf	0-800	Complete
H08R-3297	FC-101-OS	Fire	Ox & Sulf	0-1200	Complete
H08R-3298	FC-083-O	Fire	Oxide	0-650	Complete
H08R-3299	FC-100-OS	Fire	Ox & Sulf	0-1040	Complete
H08R-3300	BC-098-C	Condemnation	Ox & Sulf	0-900	Complete
H08R-3301	BC-060-S	Brimstone	Ox & Sulf	0-800	Complete
H08R-3302	FC-098-S	Fire	Ox & Sulf	0-800	Complete
H08R-3303	FC-088-OS	Fire	Ox & Sulf	0-1200	Complete
H08R-3304	YN-008-S	Bone Yard	Ox & Sulf	0-1040	Complete
H08D-3305	CC-004-S	Gap	Ox & Sulf	0-400	Complete
H08R-3306	C5-10	Central-South	Oxide	0-400	Complete
H08R-3307	CS-008-O	Central-South	Oxide	0-500	Complete
H08R-3308	CS-006-O	Central-South	Oxide	0-580	Complete
H08D-3309	BC-069-S	Condemnation	Ox & Sulf	0-1000	Complete
H08D-3310	BS-24	S. Brimstone	Oxide	0-649	Complete
H08R-3311	C5-4	Central-South	Oxide	0-500	Complete
H08R-3312	C5-7	Central-South	Oxide	0-460	Complete
H08R-3314	BC-067-S	Brimstone	Ox & Sulf	0-600	Complete
H08R-3315	BC-001-O	Brimstone	Ox & Sulf	0-1110	Complete
H08R-3316	BC-002-O	Brimstone	Oxide	0-1110	Complete
H08R-3317	BC-005-O	Brimstone	Oxide	0-1115	Complete
H08R-3318	BC-006-O	Brimstone	Oxide	0-850	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08R-3319	BC-061-S	Brimstone	Oxide	0-600	Complete
H08D-3337	FC-034-OS	Fire	Ox & Sulf	0-879.5	Complete
H08R-3347	C5-10	Central-South	Oxide	0-380	Complete
H08D-3349	FC-091-S	Fire & Brimstone	Ox & Sulf	0-1337	Complete
H08D-3350	BS-24	S. Brimstone	Ox & Sulf	0-1320	Complete
H08D-3351	BS-24	S. Brimstone	Ox & Sulf	0-1330	Complete
H08R-3354	FC-004-OS	Fire	Oxide	0-850	Complete
H08R-3355	C5-10	Central-South	Oxide	0-840	Complete
H08R-3356	C5-12	Central-South	Oxide	0-800	Complete
H08R-3361	FC-106-OS	Fire	Ox & Sulf	0-650	Complete
H08R-3362	FC-107-OS	Fire	Ox & Sulf	0-260	Complete
H08D-3363	FC-108-OS	Fire	Ox & Sulf	0-1223	Complete
H08R-3366	FC-111-OS	Fire	Ox & Sulf	0-1000	Complete
H08R-3367	FC-112-OS	Fire	Ox & Sulf	0-800	Complete
H08R-3368	FC-113-OS	Fire	Ox & Sulf	0-650	Complete
H08R-3370	FC-115-OS	Fire	Ox & Sulf	0-850	Complete
H08R-3371	BC-072-S	Brimstone	Ox & Sulf	0-700	Complete
H08R-3372	BC-073-S	Brimstone	Ox & Sulf	0-1000	Complete
H08R-3373	BC-065-S	Brimstone	Ox & Sulf	0-800	Complete
H08R-3383	BS-22	Fire	Ox & Sulf	0-1145	Complete
H08D-3384	BS-24	Brimstone	Ox & Sulf	0-1792	Complete
H08R-3388	BS-22	Brimstone	Ox & Sulf	0-1575	Complete
H08R-3392	CF-A2.5-P	Crofoot Pad	Oxide	0-120	Complete
H08R-3393	CF-A3.5-P	Crofoot Pad	Oxide	0-120	Complete
H08R-3394	CF-A4.5-P	Crofoot Pad	Oxide	0-120	Complete
H08R-3395	CF-A3.5-P	Crofoot Pad	Oxide	0-120	Complete
H08R-3396	CF-A6.5-P	Crofoot Pad	Oxide	0-60	Complete
H08R-3397	CF-A7.5-P	Crofoot Pad	Oxide	0-120	Complete
H08R-3398	CF-A8.5-P	Crofoot Pad	Oxide	0-100	Complete
H08R-3399	CF-A9.5-P	Crofoot Pad	Oxide	0-40	Complete
H08R-3400	CF-B2.5-P	Crofoot Pad	Oxide	0-80	Complete
H08R-3401	CF-B3.5-P	Crofoot Pad	Oxide	0-80	Complete
H08R-3402	CF-B4.5-P	Crofoot Pad	Oxide	0-80	Complete
H08R-3403	CF-B5.5-P	Crofoot Pad	Oxide	0-100	Complete
H08R-3404	CF-B6.5-P	Crofoot Pad	Oxide	0-100	Complete
H08R-3405	CF-B7.5-P	Crofoot Pad	Oxide	0-100	Complete
H08R-3406	CF-B8.5-P	Crofoot Pad	Oxide	0-80	Complete
H08R-3407	CF-B9.5-P	Crofoot Pad	Oxide	0-40	Complete
H08R-3409	CF-2.5-P	Crofoot Pad	Oxide	0-40	Complete
H08R-3410	CF-C3.5-P	Crofoot Pad	Oxide	0-40	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08R-3411	CF-C4.5-P	Crofoot Pad	Oxide	0-80	Complete
H08R-3412	CF-A6.5-P	Crofoot Pad	Oxide	0-60	Complete
H08R-3414	CF-C7.5-P	Crofoot Pad	Oxide	0-80	Complete
H08R-3415	CF-C8.5-P	Crofoot Pad	Oxide	0-40	Complete
H08R-3416	CF-C9.5-P	Crofoot Pad	Oxide	0-40	Complete
H08R-3417	BS-008-O	Brimstone	Oxide	0-480	Complete
H08R-3418	RN-001-S	Bay Area	Ox & Sulf	0-960	Complete
H08R-3419	BS-008-O	Brimstone	Oxide	0-600	Complete
H08R-3421	RN-003-S	Bay Area	Ox & Sulf	0-100	Complete
H08R-3427	SN-007-OS	Bay Area	Ox & Sulf	0-1060	Complete
H08R-3428	SN-008-OS	Bay Area	Ox & Sulf	0-280	Complete
H08R-3442	WN-012-S	Bay Area	Ox & Sulf	0-1005	Complete
H08R-3443	WN-011-S	Bay Area	Ox & Sulf	0-1040	Complete
H08R-3444	WN-012-S	Bay Area	Ox & Sulf	0-700	Complete
H08D-3453	CN-004-S	North Central	Ox & Sulf	0-1645	Complete
H08R-3454	CN-005-S	North Central	Ox & Sulf	0-660	Complete
H08R-3462	YN-010-S	North Bone Yard	Ox & Sulf	0-1200	Complete
H08R-3463	RN-002-S	Bay Area	Ox & Sulf	0-1140	Complete
H08R-3464	CN-011-S	North Central	Ox & Sulf	0-1200	Complete
H08R-3465	CN-011-S	North Central	Ox & Sulf	0-1200	Complete
H08R-3466	CN-012-S	North Central	Ox & Sulf	0-1160	Complete
H08R-3467	CN-013-S	North Central	Ox & Sulf	0-960	Complete
H08R-3468	CN-014-S	North Central	Ox & Sulf	0-860	Complete
H08D-3471	CN-017-S	North Central	Ox & Sulf	0-1348	Complete
H08D-3476	FC-004-OS	Fire	Ox & Sulf	0-924.5	Complete
H08D-3490	RS-001-S	Silver Camel	Ox & Sulf	0-997	Complete
H08R-3491	FC-034-OS	Fire	Ox & Sulf	0-1020	Complete
H08R-3492	RS-003-S	Silver Camel	Ox & Sulf	0-990	Complete
H08R-3493	RS-004-S	Silver Camel	Ox & Sulf	0-900	Complete
H08R-3494	RS-005-S	Silver Camel	Ox & Sulf	0-800	Complete
H08R-3495	RS-007-S	Silver Camel	Ox & Sulf	0-1020	Complete
H08R-3505	SS-002-S	Silver Camel	Ox & Sulf	0-1000	Complete
H08D-3506	SS-005-S	Silver Camel	Ox & Sulf	0-897	Complete
H08R-3507	SS-008-S	Silver Camel	Ox & Sulf	0-1000	Complete
H08R-3515	WS-001-S	Silver Camel	Ox & Sulf	0-1200	Complete
H08R-3516	WS-002-S	Silver Camel	Ox & Sulf	0-1000	Complete
H08R-3517	WS-003-S	Bay Area	Ox & Sulf	0-1000	Complete
H08R-3518	SN-008-S	Bay Area	Ox & Sulf	0-1200	Complete
H08R-3520	FS-104-S	South Brimstone	Ox & Sulf	0-1030	Complete
H08R-3522	WS-016-S	Devil's Corral	Ox & Sulf	0-790	Complete

Drill Hole	Station	Resource Area	Target	Depth (ft) From - To	Assay Status
H08R-3523	SS-007-S	Silver Camel	Ox & Sulf	0-1000	Complete
H08R-3524	WS-017-S	Devil's Corral	Ox & Sulf	0-640	Complete
H08R-3525	WS-005-S	Devil's Corral	Ox & Sulf	0-1000	Complete
H08R-3526	WS-018-S	Devil's Corral	Ox & Sulf	0-1000	Complete
H08D-3529	SS-005-S	Silver Camel	Ox & Sulf	0-1105	Complete
H08D-3530	RS-004-S	Silver Camel	Ox & Sulf	0-1786	Complete
H08R-3545	BC-024-OS	Brimstone	Ox & Sulf	0-880	Complete
H08R-3546	YS-002-S	Albert	Ox & Sulf	Moving	Complete
H08R-3548	YS-004-S	Albert	Ox & Sulf	0-590	Complete
H08R-3549	YS-005-S	Albert	Ox & Sulf	0-720	Complete
H08R-3551	CS-002-S	Cut 5	Ox & Sulf	0-960	Complete
H08R-3552	CS-001-S	Cut 5	Ox & Sulf	0-1000	Complete
H08D-3559	RS-001-S	Silver Camel	Ox & Sulf	0-1957	Complete
H08D-3560	RS-001-S	Silver Camel	Ox & Sulf	0-1387	Complete
H08R-3561	CN-018-S	North Central	Ox & Sulf	0-1000	Complete
H08R-3563	FC-033-S	Fire	Ox & Sulf	0-1075	Complete
H08R-3576	FS-011-S	South Fire	Ox & Sulf	0-1200	Complete
H08R-3577	FS-016-S	South Fire	Ox & Sulf	0-1120	Complete
H08R-3579	FS-102-S	South Brimstone	Ox & Sulf	0-910	Complete
H08R-3580	FS-109-S	South Brimstone	Ox & Sulf	0-590	Complete
H08R-3581	FS-109-OS	South Brimstone	Ox & Sulf	0-900	Complete
H08R-3583	FS-104-S	South Brimstone	Ox & Sulf	0-600	Complete
H08D-3584	BS-009-O	Brimstone	Ox & Sulf	0-692	Complete
H08D-3617	BC-3617-S	Brimstone	Ox & Sulf	0-925	Complete
H08D-3618	BC-3618-S	Brimstone	Ox & Sulf	0-1050	Complete
H08D-3621	BC-3621-S	Brimstone	Ox & Sulf	0-1527	Complete
H08R-3623	BC-3623-S	Brimstone	Ox & Sulf	0-1000	Complete
H08R-3625	BC-025-S	Brimstone	Ox & Sulf	0-1000	Complete
H08D-3626	BC-026-S	Brimstone	Ox & Sulf	0-1171	Complete
H08D-3633	BC-3633-S	Brimstone	Ox & Sulf	0-805	Complete
H08D-3634	BC-3634-S	Brimstone	Ox & Sulf	0-160	Complete
49 Shallow Holes	CF-A2-P To CF-C10-P & CF-A205-P To CF-C9.5-P	Crofoot Pad	Oxide	4020 Total feet	
Totals: 371 Holes Drilled				**300,494 Feet Drilled**	**393Holes Assays Complete**

12 SAMPLING METHOD AND APPROACH

12.1 SAMPLING METHODS

12.1.1 Reverse Circulation Rotary

Reverse circulation of the Brimstone Deposit prior to 1999 was done with reverse circulation tools utilizing a crossover sub and wet sample collection. These methods were considered to be standard at the time, despite the fact that sample recovery was generally poor due to loss of sample into open space in the formation and loss of fines due to sample overflow. The exact amount of sample recovery is unknown, because sampling weights were not recorded. HRDI staff that were involved in some of these earlier drilling campaigns estimate that average sample recovery ranged from 10 to 15 percent.

Eleven twin RC holes were drilled in 1999 to test the hypothesis that previous RC drilling had underestimated gold grades. The twin drilling was done dry, using a triple-cyclone sampling system and tricone and center-return hammer bits. Tricone was used for the uppermost portions of some drillholes (405 ft total), where previous drilling indicated barren rock; the average drilling recovery of the tricone drilling is 31 percent. An average recovery of 61 percent was obtained from intervals drilled with a center-return hammer (4,800 ft). Recovery could not be accurately measured from the very tops of holes where casing was being set (250 ft).

New holes returned higher fire and cyanide-soluble gold grades than the original holes over most intervals. Based upon analysis of assay and drilling recovery data, MRDI found that low recovery is not associated with high grades, as would occur if mineralized material was preferentially recovered or un-mineralized material was preferentially lost.

MRDI's analysis of decay and cyclicity in RC assay profiles indicate that neither down-hole contamination nor down-hole dilution was a problem in any of the RC holes drilled to date.

Allied Nevada drilling continues to perform as expected although there are areas near Brimstone, Cut 5, South Central Fault and Boneyard where recoveries are difficult. The sample weights are recorded and the assay results are reviewed by site geologists for reliability. Sample numbering has been standardized and a system is in place for error checking.

12.1.2 Core Drilling

At the rig, core drillers are responsible for obtaining a complete and representative sample of the cored interval, generally in runs not to exceed 5 feet and in shorter increments in difficult conditions. Coring is 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; if possible, the core is pumped out of the tube hydraulically. Recoveries for both core diameters are excellent.

12.2 SAMPLE QUALITY

12.2.1 Reverse Circulation Recovery

Reverse circulation sample recovery is excellent, with full 10-lb. bags collected from nearly every interval of every hole, with the exception of about 100 samples in the entire Allied Nevada set of samples collected to date. The missing samples occur in isolated zones of voids or badly broken ground.

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. Core sample recovery is also excellent, in excess of 99% of the bedrock cored, once the broken (alluvial) ground of the surface 50' is breached. There are few instances of core loss below the bedrock contact. The vast majority of such losses were due to voids within the stratigraphy.

12.3 SAMPLE LOCATION

12.3.1 Downhole Surveys

Downhole surveying is conducted by International Directional Services (IDS) of Elko Nevada. Gyroscopic techniques are used to locate drillhole deviations, an industry norm. Most historic drilling was not downhole surveyed.

12.3.2 Final Collar Surveys

Upon drillhole completion, Allied Nevada uses a licensed surveying company to locate the collar coordinates of exploration drillholes. Once the mine is brought back into production and staffed, drillholes will be located internally.

12.4 SAMPLING INTERVALS

12.4.1 Reverse Circulation

Rock chips are collected continuously down the hole. Samples are collected every five feet. Such was the case with the Allied Nevada drill program. Samples are submitted for assay, as collected on the rig; in addition standards, blanks and duplicates are inserted into the sample sequence as described in the section on QA/QC.

12.4.2 Core

The Allied Nevada core program uses a 10' barrel to collect samples. After the core is logged it is the geologist's responsibility to determine appropriate sample intervals and boundaries. 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 are inserted by the geologist to compensate:

- A sample must NEVER cross a lithology boundary.
- A sample must not cross an obvious alteration boundary, including oxidation.
- A sample must not exceed 7 feet in length, and only be that long if sure to be barren; 5 feet maximum is better.
- 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.

13 SAMPLE COLLECTION, PREPARATION, ANALYSIS AND SECURITY

13.1 SUMMARY

The sample preparation procedure prior to 1999 is not documented. MRDI, in their review of the procedures, believed that a small pulp (perhaps 150 to 300 grams) was prepared from a split of nominal 10 mesh material (crusher output from reverse-circulation drilling is generally about 50 percent passing a 10 mesh screen and 95 percent passing either a ½ or 3/8 inch screen). The combination of large particle size and small sample mass taken in the first split is substandard relative to current industry practice for gold deposits containing visible gold. Sample preparation in 1999 consisted of drying an 11-22 pound split at 175 degrees F, crushing the entire sample to 95 percent passing 10 mesh, splitting 400-800 grams and pulverizing the split to 95 percent passing 150 mesh.

Allied Nevada uses two laboratories for assay analysis. The companies are ALS Chemex and SGS Laboratories. ALS is ISO9001:2000 compliant. SGS conforms to CAN-P-1579, CAN-P-4E (ISO/IEC 17025:2005) standards.

It is the SEWC's opinion that Allied Nevada is using best industry practices with regard to sample preparation, security and analysis.

13.2 SAMPLE PREPARATION AND ANALYSIS

Sample preparation by Allied Nevada personnel is limited to site technicians who saw core samples. No officers, directors or associates of the issuer are involved with the sample preparation process.

13.2.1 Pre 1999 Sample Preparation

The sample collection method is not documented; however, it is likely that dry samples were collected by splitting the reverse circulation cuttings at the drill with a riffle splitter and wet samples were collected by using a wet rotary splitter. Prior to the end of 1991, all of the samples were prepared for shipment to Barringer Labs in Reno, Nevada, which were submitted for fire assay. Follow-up cyanide soluble assays were requested for selected intervals after the fire assays were received.

The samples collected after the end of 1991 were prepared for assay at the laboratory facilities at the mine.

Industry standard methods used during this time frame were to dry the drill sample (typically 5-15 lbs), crush to 10 mesh (sometimes this step was omitted), take a split to pulverize (usually 300-600 grams), pulverize, and prepare a one assay ton pulp for assay. It is not known if these methods were employed at Hycroft during the exploration programs prior to 1999.

13.2.2 Post 1999 Sample Preparation

The samples were transported to the sample-preparation facility at the Hycroft laboratory for processing prior to shipment to the outside analytical laboratory. All samples were logged in and weighed as received with the data recorded on the Sample Collection Data Sheet designed for this program. The sample preparation protocol established for the mine bucking-room required that the entire sample be retained. The sample was to be split into duplicate laboratory samples, "A" and "B", each weighing between 5 and 10 kilograms (11 and 22 pounds) with the balance of the material bagged as a coarse reject. The weight of the sample received was recorded on a Sample Splitting Data Sheet and the number of splits required to provide laboratory samples of the appropriate weight was determined as shown in Table 13.1.

Table 13.1 Laboratory Sample Sizes

Sample Weight Received	"A" Split	"B" Split	Coarse Reject
< 22 Pounds	100%	0	0
22 – 44 Pounds	50%	50%	0
44 – 88 Pounds	25%	25%	50%
88 – 176 Pounds	12.5%	12.5%	75%
> 176 Pounds	6.25%	6.25%	87.5%

The samples were then passed through a single-stage Gilson Adjustable Splitter the appropriate number of times and the samples bagged. After splitting, the resulting samples were weighed and the weights were recorded on the Splitting Data Sheet. This allowed for a check on the splitting and insured that the sample was split properly. The "A" splits were then lined up for shipment to the analytical laboratory (ALS Chemex) and the others were placed in storage at the core shed at the mine.

All sample preparation was performed at the ALS Chemex facility located in Sparks, Nevada. The sample preparation protocol established for this program included:

- Weigh each sample as received. This weight was reported and recorded as the wet weight.
- Oven-dry the samples at a temperature not to exceed 175° F. This temperature was selected to minimize the volatilization of trace elements and sulfur.
- Weigh each sample after drying. This weight was reported and recorded as the dry weight.
- Crush the entire sample to 95% passing 10-mesh prior to any splitting.
- Pass the crushed sample through a Jones splitter to obtain 400 to 800 grams for pulverization. Retain the entire coarse reject for return to Hycroft.
- Pulverize the 400- to 800-gram split to 95% passing 150-mesh.

- Riffle-split the pulp with one split retained by ALS Chemex for analysis and the other returned to Hycroft.
- The wet and dry weights were used to adjust the total-sample weights that were then used to calculate the sample recovery.

All drill samples were analyzed for gold by one-assay-ton fire assay performed both by the ALS Chemex laboratory in Vancouver, BC and the Hycroft Mine laboratory. ALS Chemex used an AA finish with the detection limit reported at 0.0002 opt gold while Hycroft used a gravimetric finish with the detection limit reported at 0.001 opt gold. The standard operating procedure has not included the calculation of a fire assay silver value. Thus, there are virtually no fire assay values for silver in the database.

Elemental sulfur analyses were performed on all samples that were reported to contain total-gold concentrations greater than or equal to 0.005 opt. This threshold was selected in order to insure that any interval that could be “ore grade” would be run. The analyses were performed by ALS Chemex using a carbon tetrachloride leach and gravimetric finish. The results of these analyses were then used to validate the geologic logging of sulfur in the samples and to assess the impact of sulfur on the cyanide-leach analyses.

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. These analyses are used in the resource modeling and for grade control during the mining phase. The following analytical procedures are followed:

- 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 atomic absorption 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.

Attempts to validate the laboratory methodology during the recent program demonstrated that it is quite sensitive to several parameters in addition to the reagent concentrations as follows:

- Temperature is critical to any cyanide-soluble gold analysis. Any relative change in temperature will affect the reaction rate and, thus, how far the reaction proceeds during the essentially fixed leach time. Work with ALS Chemex highlighted the requirement to maintain the appropriate temperature when cold cyanide-leach results failed to compare favorably with the Hycroft results.
- Leach time and agitation are critical. Assuming a consistent leach time, agitation will also affect the reaction rate. In order to duplicate a method, agitation must be consistent both in the attitude of the sample and in the agitation rate. This effect of agitation was also confirmed during the work with ALS Chemex.
- The presence of elemental sulfur in samples has a significant effect on the cyanide-soluble-gold recovery. Historically, samples containing significant amounts of elemental sulfur have yielded much lower than anticipated cyanide-soluble-gold recovery. During the test work with ALS Chemex, it was found that elemental sulfur not only suppressed the gold solubility but that it could also be "preg-robbing". This cyanide-soluble-gold-in-the-presence-of-sulfur assay problem was demonstrated when several sample solutions were read after sequential leach times with depressed results after the longer time-intervals. This phenomenon would make the time intervals between leaching, centrifuging, and reading critical for duplication of results from samples containing elemental sulfur.

After the initial test work with ALS Chemex, American Assay Laboratories located in Sparks, Nevada, was selected to run a series of samples using the Hycroft methodology.

American Assay Laboratories was provided with the written methodology and one heated agitating water-bath from the Hycroft Laboratory. A meeting was held where the methodology and potential problems were discussed and 106 pulps from drill samples were submitted for analysis. The American Assay Lab checks indicated good correlation with the Hycroft Laboratory cyanide soluble gold assays, but on the average were about 7% lower than the Hycroft results.

Presently, samples are prepared as follows:

- Samples are weighed, dried and re weighed.
- A 1 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)
- 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)

13.2.3 Post 1999 Sample Collection

Historically, all drill sampling was completed with wet samples collected through the cyclone and a 36" rotary wet splitter. Samples were collected on five ft sample intervals directly into 20" x 24" sample bags placed in 5-gallon buckets. A thin polymer (EZ Mud) mix was prepared for use as a flocculent with some added to each bag prior to sample collection.

Initially, the rotary splitter was set to deliver 25% of the cuttings returned to the sample port using the "pie" covers. Assuming sample return would be averaging about 60% (actual recovery during the 1999 RC Twin Program) this arrangement would yield samples between 13 and 20 pounds depending on the bit size and material being drilled. The sample return was monitored during drilling and the "pie" covers removed to deliver 50% of the return when circulation appeared to be falling off. The splitter setting was noted and recorded to allow for calculation of actual recoveries.

Drill water injection was regulated to minimize the fluid return while maintaining sufficient flow for drilling and sample return. One 5 gallon bucket was sufficient for most of the intervals when collecting 25% of the return. When it appeared that one bucket would be insufficient, a second bucket was used to collect the balance of the sample. If two buckets were used for a sample, they were set aside, flocked, allowed to settle, decanted, and combined. Sample bags were tied closed, set aside, and allowed to weep prior to transport.

Allied Nevada geologists now provide drill crews with a set of bags pre-numbered in order representing the footage interval completed. The sample numbering sequence includes blanks and standards inserted every 20th sample, the driller's sampler only has to keep track of the actual footage drilled, with respect to the footage marked on the bags. The actual insertion of blanks and standards is handled independently by geologists, who create duplicate numbers at appropriate intervals, post scripted with “S” for standard and “B” for blank. The driller’s sampler is provided with chip trays accurately numbered. Cuttings are collected as a continuous fraction of the return stream from the drill rig. The cuttings are diverted to a 10"x17" mesh bag, and tray chips are diverted to a kitchen strainer. Filled chip trays are collected by an Allied Nevada geologist for logging under a binocular microscope. Sample bags are shipped to the analytical laboratory for preparation and assaying.

Sample bags are allowed to dry and drain at the drill site or in a holding area near the core facility. Samples are then brought down to the shipment staging area, where Allied Nevada personnel finish preparing extra bags for standards and blanks. The samples are then loaded in 4' x 4' x 3' wooden crates for delivery to the laboratory.

13.2.4 Post 1999 Sample Collection - Core

The drill crews place the core in waxed cardboard core boxes, with tops and bottoms accurately labeled as to Company – Property– Hole ID – Box # -– From – To. 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. There are 5 rows or trays. 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 two numbers on the larger surface:

C [cut] – m.n feet R [recovered] – m.n feet

The Cut number results from measured rod footage and the Recovered number stems from a taped measurement of core in situ.

The geologists provide the sample prep technicians with working copies of one document: the basic sample sequence list, which contains the drill-hole number and the appropriate sample intervals indicated. The sample technician works from the sample list in conjunction with the core to see why and how sample boundaries were picked. This provides a redundancy check on the accuracy of the sample list.

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. The portion to be saved remains in the core box, in its proper position, with core blocks in place, and the boxes are stacked on pallets for storage in a core shed. The split portion of core is bagged and boxed in 4' x 4' x 3' wooden crates.

13.3 SAMPLE SECURITY

13.3.1 Reverse Circulation and Core Samples

Crated 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 range of sample numbers, and the total sample count. The lab has no coordinate 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 the blanks (samples of quartz sand post scripted by “B” in Kraft envelopes), standards (pulp powder post scripted by “S” in Kraft envelopes), but will have no knowledge of the accepted value of each, as a variety of standards are submitted, ranging from 0.2 ppm to 9.0 ppm. In addition, a sample of unmineralized decorative rock is submitted as the lead sample in each drillhole group of samples.

At present, site sawn samples are delivered to ALS Chemex in Winnemucca by Allied Nevada staff, or picked up by ALS Chemex drivers for delivery to their facilities in Winnemucca and Elko.

Chain of custody is established by transmittal sheets. Some uncut core is hand-delivered to a contract core cutter in Elko who provides a handwritten receipt for delivery. After cutting, a transmittal sheet is prepared for submission to SGS Laboratories in Elko.

13.3.2 Analytical Results

Following analysis, results are posted to a digital laboratory database on which Allied Nevada has secure permission privileges. Managers download the data to Excel files, 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, and a follow-up hard-copy certificate is mailed to company offices.

13.4 QA/QC, CHECK SAMPLES AND CHECK ASSAYS

Until 1992, selected mineralized intervals were analyzed for cyanide-soluble gold and cyanide-soluble silver by Barringer Laboratories, Reno. When contacted by MRDI during the 1999 drill program, Barringer Laboratories' 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 Mine laboratory. Fire assays were also performed. In most cases, if the fire assay was below detection, the cyanide-soluble assays were not performed. No decipherable QA/QC data exist for these assays. There are QA/QC data for the Hycroft blast hole assays.

All samples in the 1999 Reverse-Circulation Twin Drillhole program were fire-assayed for gold by ALS Chemex, Vancouver, and Hycroft. Comparison between Hycroft and ALS Chemex revealed a number of outliers, prompting the use of Cone Geochemical as an Umpire for the disagreements. Cone check assays on 40 pairs with disagreement were in better agreement with ALS Chemex than with Hycroft. Consequently, the Chemex data were used for calculating correction factors to fire assay results for the block model.

Sample bags that are intended as Standards and Blanks are labeled at the office. For standards, the identifying code number of the blank or standard and grade value is written on the sampler's sheet at the correct sample ID value, and inserted into the appropriately marked sample bag.

Standards and Blanks, numbered as described above, are inserted in the crates at a rate of one standard and one blank for every 40 drill samples.

Each crate contains the raw samples, duplicates, standards and blanks intended for each hole. The crates also contain bags for non-recovered samples. To the extent possible, all samples for one hole are aggregated together, and sample transmittal sheets are filled out at least in duplicate (one to the lab, one for file retention), with one job number assigned to each hole shipment.

An Allied Nevada copy of the transmittal sheet is stored in a file cabinet in the office. Once assays have been received, a copy of the assay sheets will be 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 in on a dedeicated Hycroft computer, indexed by hole number. Originals of all logs and assays are stored in file cabinets on a per-hole basis, also indexed by hole number. Allied Nevada personnel contact the lab to obtain a job number assignment for hole or partial hole shipment, and arranges for sample pickup by the lab's driver. In some cases, an Allied geologist returning to Winnemucca for the evening may deliver a crate directly to the lab.

To increase the integrity of the sample handling process, from collection to shipment to assay, standards are inserted in the sample stream at a rate of one standard and one blank for every 40 drill samples. The reference standards were prepared by Mineral Exploration Geochemistry (MEG) of Carson City Nevada.

Standards are stored in plastic bins at the office. MEG has noted the gold content and identifying code on a gummed label on each Kraft envelope. When a sample shipment is being prepared, standards with different gold concentrations are selected randomly from the available group and inserted into the sample stream. When the standard or blank is used for the sample stream, the stick-on identifying label is removed from the envelope and the information is transferred to the digital sample record. The actual label is preserved on a separate sheet of paper filed with the drill hole. The actual Kraft envelope containing the powdered standard, and turned in to the lab, is NOT labeled.

14 DATA VERIFICATION

14.1 INTEGRITY OF DATABASE

A review and validation study was performed on the database in 2000 (MRDI, 2000) This work and the results are described in the following sections. As part of the project restart pre-feasibility, SRK Consulting of Elko, NV has performed a 100% data validation on the exploration databases and that work is described at the end of this section. SEWC verified that the databases were correct before any information was used in this report.

14.1.1 Data Selection

Five different criteria were used to select assay data for checking. Two of these criteria consisted of random selections from assayed intervals, from two mutually exclusive lists of assayed intervals; intervals with gold fire assays greater than 0.01 opt, or closer than 15 feet from an interval with gold greater than 0.01 opt made up the potential ore zone group and all other intervals were placed in the probable waste group. Eight percent of the intervals in the potential ore zone group were randomly selected for checking, and one percent of the probable waste group was selected. Random selections such as these allow error frequency rates for data entry to be estimated.

Three other groups of samples were selected for sampling to check for certain types of errors. Because these directed checks are selected on the basis of certain characteristics that may correlate with an increased likelihood that data entry errors have been made, the error frequency rate may be higher, and is not representative of the database as a whole. One directed check was based upon selecting intervals where the cyanide-soluble gold result markedly exceeds the gold fire assay result; a sample was selected if the [cyanide-soluble gold / fire-assay gold] ratio exceeded 1.2 and the cyanide-soluble gold result was at least 0.03 opt gold higher than the gold fire result. Another directed check was made by selecting intervals that have two nearest neighbors (one above and one below) with the same geologic characteristics (oxide, sulfur/sulfide, and alteration type) but with nearly an order of magnitude difference in grade; an interval was selected if it was either more than eight times higher than both its neighbors, or its neighbor.

14.1.2 Assay Selection

Assay data were checked against source documents. Source documents consist of photocopies of Barringer assay certificates, or handwritten entries from the Hycroft Mine laboratory. A tabulation of errors found for groups chosen by the various selection criteria is shown in Tables 14.1 through 14.3.

Table 14.1 Error Frequencies by Selection Criteria

	No. Samples	No. Checked	No. Errors	% Errors
Cyanide grade significantly greater than fire	116	88	36	40.9%
Value (>8x factor) between interval and its 2 nearest neighbors	43	41	11	26.8%
The 75 (unique) highest concentration gold	22	22	3	13.6%
Random selection in >.01 grade envelope (Rand-ore)	920	876	34	3.9%
Random selection outside grade envelope (Rand-waste)	232	214	26	12.2%
Totals	1,333	1,241	110	8.9%

Table 14.2 Tabulation of Errors in Rand-ore Category

Number of Errors (total=34)	Percent Errors (total = 3.9%)	Description of Error
5	0.6%	Missing samples entered as 0.001 oz/ton
15	1.7%	CN Ag mistype error
14	1.6%	FA Au and/or CN Au mistype

Table 14.3 Tabulation of Errors in Random Waste Category

Number of Errors (total=26)	Percent Errors (total = 12.2%)	Description of Error
4	1.9%	Missing samples entered as 0.001 oz/ton
13	6.1%	CN Ag mistype error
9	4.2%	FA Au and/or CN Au mistype

Of the randomly selected samples, 0.8 percent of them were in error regarding missing samples entered as 0.001 opt, 2.6 percent were in error regarding cyanide-soluble silver mistype errors, and 2.1 percent were in error regarding fire-assay-gold and/or cyanide-soluble-gold mistype errors. The mistype errors exceeded the industry standard of 1 percent; therefore, Vista reviewed cyanide-soluble and fire-assay gold entries for ore holes and corrected any errors found. MRDI did not re-audit the corrected database.

14.1.3 Geological Data Checks

The new geologic logging was checked for data entry errors. In addition to the drillhole name and depths (from and to), there are six fields containing single digit integers corresponding to geologic observations. Approximately five percent of the relogged drillhole intervals were selected at random. Nearly every relogged drillhole had at least one interval selected for checking. Of 1,740 selected intervals, logs were available for 1,696 (a few of the new logs had been misplaced at the time of the audit). Seventy-seven (4.5 percent) of the selected intervals were found to have an error in one of the fields. Because there are six different fields for each interval, the error rate was found to be 0.8 percent. Drillhole logs having errors were rechecked by Vista in their entirety. This led to detection and correction of some additional entry errors.

Subsequent investigation (MRDI, 2000) revealed entries where an interval had native sulfur observed, but no estimate of percentage (a separate field). This led to some additional re-logging to correct these discrepancies.

14.1.4 Collar Survey Checks

MRDI checked every drillhole collar location against entries in the original drill logs. Two large errors were found in collar locations and these were corrected in the collar database. In one case, two drillholes differing by one number were given the same collar coordinates. In the other case, the drillhole number had two of its digits transposed. These errors were corrected or resolved (in at least one case, a source document was reported to be in error) by Vista geologists.

14.1.5 Downhole Survey Checks

Very few drillholes had down-hole surveys. All drillholes with down-hole surveys were spot checked. No errors were found.

14.2 ANALYSIS OF SAMPLING BIAS AND CORRECTION OF EXPLORATION DRILLING ASSAYS

The reconciliation of Brimstone production indicated that the Brimstone Model slightly over-predicted ore grade 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 may be 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.

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.

Most of the exploration samples, and all of the blast-hole samples, were assayed by the mine laboratory using the same cyanide-soluble gold protocol. Vista hypothesized that the samples collected during exploration reverse-circulation drilling were biased low, as a consequence of preferential loss of fines. Exploration drilling was performed wet, and sample-collection buckets were allowed to overflow, without any effort to capture the fines. In such circumstances, if the fine fraction has a higher grade than the rest of the sample, the sample will have a low bias, relative to what would be obtained from a properly collected, representative sample.

MRDI and Vista's work in 1999 and early 2000 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 exceeds 5.0 percent), 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. A preg-robbing effect has not been noticed on the heap robbing recoveries, most likely because native sulfur typically occurs as much larger fragments when found in run-of-mine ore.

Correction factors for fire-assay and cyanide-soluble gold due to sampling biases and the presence of native sulfur were derived by MRDI from three sources of comparative data:

- Comparison of blast-hole cyanide-soluble gold assays to cyanide-soluble gold assays of nearby exploration holes.
- Comparison of fire assay and cyanide-soluble gold in new twin RC holes and fire assay and cyanide-soluble gold in old exploration holes.
- Correction of cyanide-soluble assays for the presence of sulfur, using paired sulfur-bearing intervals in twin holes and old holes.

These studies produced the following method of correction:

- For intervals with native sulfur logged at high (>5 percent) levels, the cyanide-soluble gold assays were discarded and replaced with an estimate derived from CN-sol Au to fire Au 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 14.4
- CN-sol gold: After corrections for sulfur were made, the following adjustments were applied to the assays with gold <0.045 oz/ton:
    - o Acid Leach Ore Original assay x 1.40
    - o Oxide Ore Original assay x 1.19
- Fire Assay gold: Adjustments were made to assays with gold, 0.08 oz/ton:
    - o Acid Leach Ore Original assay x 1.39
    - o 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. This was not undertaken because silver is a byproduct; it was estimated that even a large adjustment of silver assays would produce only a very small, perhaps negligible, change in the resource model.

Table 14.4 Adjustments to Cyanide-Soluble Gold for Presence of Sulfur

Acid Leach	
Native Sulfur Logged Observation, Drill Year	***Adjustment (CN is CNsol 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 14.5 shows the correction factors applied to the cyanide soluble assays from the twin drillholes.

Table 14.5 Correction Factors Applied to the 1999 Twin Drilling (US Imperial Units)

						Original	**Corrected**
1999 Drillhole	***Interval***	***Feet***	***AuFA/ton***	***AuCN/ton***	***Old Drillhole***	***AuFA/ton***	***AuCN/ton***
99-1378B	*180-*	*230*	*0.015*	*0.012*	*88-1378*	*0.009*	*0.012*
99-1419B	*410*	*235*		*0.010*	*89-1419*	*0.010*	*0.011*
99-1432B		*220*	*0.003*	*0.024*	*89-1432*	*0.014*	*0.018*
99-1504B	*330-*	*475*	*0.013*	*0.009*	*90-1504*	*0.006*	*0.007*
99-1523B	*565*	*185*	*0.018*	*0.021*	*90-1523*	*0.027*	*0.029*
99-1944B		*250*	*0.027*	*0.002*	*92-1944*	*0.003*	*0.004*
99-1949B	*240-*	*410*	*0.016*	*0.013*	*92-1944*	*0.011*	*0.013*
99-1950B	*460*	*405*	*0.004*	*0.014*	*92-1950*	*0.010*	*0.011*
99-1975B		*470*		*0.022*	*92-1975*	*0.018*	*0.020*
99-1976B	*125-*	*405*		*0.012*	*92-1976*	*0.007*	*0.010*
99-2648B	*600*	*205*		*0.002*	*95-2648*	*0.001*	*0.001*
Totals		***3,490***		***0.013***		***0.010***	***0.012***

14.3 ANALYSIS OF SAMPLING BIAS AND CORRECTION OF EXPLORATION DRILLING ASSAYS - 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 (1.0), a decision was made to assume that the power is one (1.0), 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 (1.0).

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, so the overall difference between the MRDI and ORE adjusted grades is less than 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 above 0.75, so should samples that contain sulfur. The correction equations were derived as follows:

1) The drillhole data contains codes identifying the quantity of sulfur in the sample based on visual examination of drill cuttings by the geologist. Sulfur categories are:
    a) No Sulfur,
    b) Trace Sulfur,
    c) <5% Sulfur,
    d) 5% to 10% Sulfur, and
    e) >10% Sulfur

2) 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) greater than 0.004 opt Au to minimize problems calculating ratios when the assay values approach the precision of the assay.
3) Logarithmic correction curves were fit to the QQ points in the form:

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

Two curves were used for the 5% to 10% Sulfur, and >10% Sulfur categories, because the low ratio end of the corrections were not linear.

4) 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 of Elko Nevada completed a one hundred percent 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 ANV 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

ANV 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 main mine reception and sign-in area. Additional survey data were located in the former survey office adjacent to the bull room. The data were organized as individual files 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 were boxed and transported to SRK's Elko office for verification and further processing.

14.4.2 Data Collection – Electronic Data

The Hycroft electronic database was provided by ANV in MS 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 were 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 were 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 ounces per ton (opt) to parts per million (ppm) and parts per billion (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 analyses:

- FAU Fire assay gold;
- FAG Fire assay silver; (rarely assayed/reported)
- CNAU Cyanide soluble gold; and
- 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 Laboratories. 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 Mine 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 2007 only minor analytical work was done, all by off-site laboratories such as American Assay Laboratories, Chemex and ALS Chemex.

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

14.4.4 Verification of Geological Data

Geologic data were checked and validated by previous workers (MRDI, 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 concludes 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 were 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.

Down-hole surveys are uncommon in the database. There were no historic records to which the electronic data could be compared. An examination of the drillhole 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. These 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 Mine 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 NEW RESOURCE MODEL

A great level of detail and correction went into the 561 drillholes at Brimstone that were used for the ORE model. For this report Allied Nevada logged and additional 1,205 drillholes for geology. This provided enough information to apply the same MRDI and ORE derived factors the recently logged drilling data. The 561 originally factored drillholes 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

The Brimstone Deposit is being mined and processed using a ROM heap leach pad and a Merrill-Crowe plant. Production data from the pads give the best possible indication of future processing recoveries. The feasibility of mining is based on the past performance of the Brimstone leach pads. The ore types that will be placed are the same ore types that were placed on the pads until 1998. The Pad 4 was continuously leached after mining ceased and the recovery for that pad stands at 56.6% of the total gold by fire assays. The estimated recovery for the remainder of Brimstone is projected to be 56.6% of 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 square feet. A 3 Million square foot addition was constructed during the 2nd quarter of 2008. ROM ore will be placed on the leach pad with trucks. Successive leach pad lifts will be 30 ft in height.

The ore lifts on the pad will be cross ripped for enhanced permeability. The ore will be irrigated at a rate of 0.0025-0.0030 gpm/ft2 with a buffered cyanide solution. The anticipated leach period will be 60 to 90 days.

16.1.2 Brimstone Plant

The Brimstone plant comprises four solution ponds, a Merrill-Crowe zinc precipitation plant and a refinery. There are two primary ponds; the pregnant pond and the barren pond. Each primary pond has a capacity of 2.6 million gallons each. The third pond is an emergency pond and has a capacity of 2.8 million gallons. The last pond is the old Lewis pregnant pond and it has a capacity of 4.0 million gallons.

Solution processing and precious metal recovery will be accommodated with a 2,800 gpm Merrill-Crowe plant. Pregnant solution is buffered, fortified with cyanide and then clarified with Sparkler filters. The clarified solution is de-aerated with vacuum pumps and a packed vacuum tower. Zinc dust is added to the clarified/de-aerated solution. Gold and silver precipitates are captured with three 48 inch recessed plate filters. The collected precipitates are transported to the refinery, retorted to remove mercury and then fire refined. Barren solution is discharged to the barren pond and the re-circulated back to the Brimstone pad.

16.1.3 Recovery

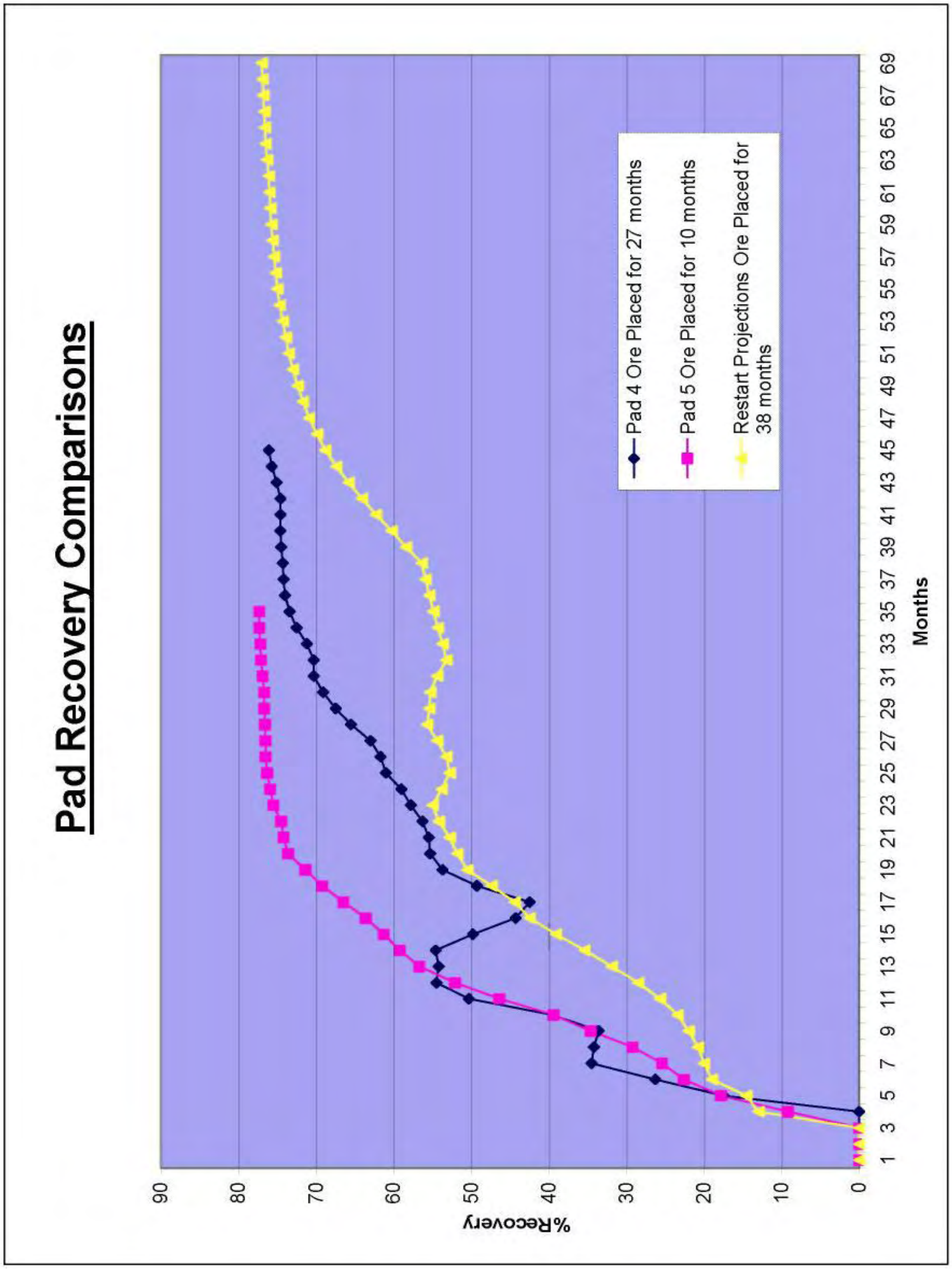
Actual final gold recovery from Pad 4 for all previous operations was 79.5% (pad #4, historic results). Considering all the information available, the projected recovery of 56.6% of AuFA represents a realistic estimate of recovery for the remaining ores in the Brimstone pit. Historic production figures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

Table 16.1 Production Pad Loading and Recoveries (US Imperial units)

Pad	Tons of Ore	Gold Loaded Oz	CN Sol Grade Opt	Recovery Gold Oz	Actual % Recovery
4	11,130,054	159,206	0.0143	126,622	79.5
5	4,334,061	61,991	0.0143	49,348	79.6

Figure 16.1 shows the actual recoveries of gold from the Brimstone pad. The figure shows gold recovery versus time for the ROM ore that was placed on pads 4 and 5. Since the same material types will be placed by trucks, just as in the past, these are the best indicators that recoveries will remain the same for future ore placement and leaching. Estimated recoveries are also shown on the graph. It shows a lower recovery based on minimal ore placement in the beginning of the mine plan. It is anticipated that once production hits its peak and additional testing of solution to the pad are evaluated that metallurgical recovery will improve.

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 were reviewed by the author and are briefly revisited in this section. In 1994, a metallurgical program was initiated at the Hycroft mine to evaluate the gold recovery that could be expected from run-of-mine 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 breccia" and "acid-leach." The acid-leach material, which generally forms the upper part of the Brimstone deposit, is fine and friable, whereas the silicified breccia is 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 breccia core samples were available for testing. As a result, good confidence in the recoveries from acid-leach material were obtained through test work while additional testing needed to be undertaken to improve the confidence in the expected recovery from the silicified-breccia 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 percent of siliceous oxide (previously called silicified-breccia) 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 drillhole data was re-logged, and together with pit mapping and blast hole data, the geology of the Brimstone deposit was reinterpreted, resulting in 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 breccia," 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.

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 percent 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 were as shown in Table 16.2.

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 percent, but if the lowest recovery test is rejected, the average gold recovery is 75 percent.

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.

16.2.2 Test Work – 2000

In 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 16.3.

Table 16.3 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 &3, which employed samples taken from intact core not representative of Run of Mine 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 indicate that future ores will yield similar metallurgical performance to previously mined ore.

16.3 PREVIOUSLY MINED ORE COMPARED TO REMAINING MINERAL RESERVES

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 16.4 and 16.5.

Table 16.4 South Brimstone Drill Intercepts (US Imperial Units)

Ore Type	Footage Included	% Of Total Footage	CN Sol Au(opt)	%CN Sol Recovery
Siliceous	13,873.0	45.7%	0.0155	73.5%
Acid Leach	12,677.1	41.8%	0.0191	76.7%
Clay-bearing	3,165.7	10.4%	0.0239	80.2%
Other	615.0	2.0%	0.0142	85.9%
Total Average	30,330.8		0.0178	76.0%

Table 16.5 North Brimstone Drill Intercepts US Imperial Units

Ore Type	Footage Included	% Of Total Footage	CN Sol Au(opt)	%CN Sol Recovery
Siliceous	11,355.0	35.7%	0.0137	75.3%
Acid Leach	18,727.0	58.8%	0.0160	75.8%
Clay-bearing	1,485.0	4.7%	0.0136	79.5%
Other	260.0	2.0%	0.0088	59.5%
Total Average	31,827.0		.0150	75.7%

These results indicate that there is virtually no 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% versus 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.

16.4 METALLURGICAL TESTING OF UNOXIDZED MATERIAL

In September 2008 Allied Nevada sent a bulk sample of representative unoxidized material to SGS Mineral Services to investigate recovery methods for this material type. SEWC believes this step 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.

16.4.1 Ore Characterization

Test work indicated the following:

- Gold and silver grades were slightly lower than anticipated at 0.52 g/t Au and 16.0 g/t Ag. The average gold and silver values anticipated, based on information provided by Allied Nevada were 0.61 g/t Au and 17.3 g/t Ag.
- At 17.2 (metric), 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 sulphide mineral observed at an estimated 1-5% in all eight composites. Marcasite which is essentially pyrite with a chemical composition slightly weighted toward S (S content in marcasite is 53.5% and only 53.4% in pyrite, both are FeS_2) was present in the range of 1-5% 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.

16.4.2 Metallurgical Testing

- 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 sulphide 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 ~80-85% gold, silver and sulphide 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. We feel 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 (P80), 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 ~20 µm (P80), 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 P80 = 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 ~99% sulphide 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 sulphide ore.

The economic viability of these pre-treatment oxidation processes needs to be evaluated. Considerable test work is clearly required in order to optimize unit processes and to define the metallurgical flow sheet required for treatment of the Hycroft sulphide ore. Final product recoveries can only be confirmed accurately after the flow sheet has been tested and optimized.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The resources stated for Hycroft Mine in this report conform to the definitions adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), December 23, 2005, and meet the criteria of Measured Mineral Resources, Indicated Mineral Resources and Inferred Mineral Resources

The Hycroft Mineral Resource and Mineral Reserves are not materially affected by any known environmental, permitting, and 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

Drillhole data for the Hycroft property is maintained in an Microsoft database by Allied Nevada. Allied Nevada validates the database constantly and has certified the data to be clean and error free. The drillhole database has been converted to a Vulcan Isis database named Feb09_2.dhd.isis.

17.1.1.2 Assay Corrections

No assay corrections were applied to the new resource block model. There is no geologic logging associated with the majority of the historic drilling data. Hycroft geologists are presently re-logging the majority of the drill cuttings for future geological modeling exercises. The Hycroft assay data were corrected for resource estimation as follows:

- If the ratio of cyanide-soluble gold (AuCN) to fire-assay gold (AuFA) was greater than 1.00, and AuCN was less than 0.002 opt Au greater than AuFA, AuCN was set equal to AuFA.

Brimstone and Cut 5 assay data were 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, October, 2007) was used as the base topography at Hycroft. Aerographics provided the topo in Autocad format and SEWC converted the topo to a Vulcan triangulation surface. The surface was used to separate air versus ground in the block model.

SRK 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 back-filled pits. The difference between the aerial survey surface and the SRK surface equals back fill material and this fill material was coded to the Vulcan block model.

- Aerial Survey Surface triangulation – *Ocober_topo.00t*
- As-Mined-Pre-Backfill triangulation – *4sw_prebackfill_cont5.00t*

17.1.1.4 Geological Model – Excluding Brimstone and Cut5

The quaternary alluvium/bedrock surface was interpreted by SRK and Hycroft site geologists. A Vulcan surface was provided and was used to code the block model with this break between rock types. A block that is greater than 50% in the ground is coded as bedrock otherwise it is coded as alluvium.

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 1) Oxide, 2) Mixed and 3) Sulfide by the following methods and formulas.

1. *Two columns (redox, ratio) were added to the drillhole database.*
2. *The ratio of AuCN/AuFA was stored in the ratio column.*
3. *If (ratio >= 70%) then redox = 1, where 1 indicates Oxide*
4. *If(ratio <= 30%) the redox = 3, where 3 indicates Sulfide*
5. *All other assay intervals set to 2, where 2 indicates Mixed*
6. *This results in the redox field containing ones, twos or threes.*

The redox values were combined into 25 foot composites based on the majority value of the five 5 foot redox values.

7. *Determined the anisotropic preferred direction of the redox indicators.*
8. *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, 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 lumped together as Oxide for the purposed of reporting Mineral Resources as either Oxide or Sulfide.

17.1.1.5 Geological Model – Brimstone and Cut 5

The scope of this report was to update the reserve level detail of Brimstone and Cut 5. Therefore, interpreted geologic shapes were required for increased confidence in the mineral estimate for these two deposits.

Allied Nevada staff, along with input from SRK Elko, logged the geology of all the holes in the proximity of Brimstone and Cut 5. The resulting interpretations of rock units and alteration zones, in the form of Vulcan® triangulations, were provided to SEWC. The definition of rock units allow for the proper distribution of bulk material densities. This in turn helps provide a more reliable estimate of the contained metal of the deposit. The alteration shapes also help determine the rock type densities but also help subdivide the deposit for further metallurgical characterizations. Table 17.1 lists the rock units and alteration shapes used for the Brimstone and Cut 5 geological model.

Table 17.1 Vulcan® Rock Units and Alteration Shapes

Vulcan® Triangulation	Geologic Zone
B_AcidminusEBlk.00t – Brimstone C_AcidLeach.00t – Cut 5	Acid Leach (Alteration)
Alluvium SEWC	Alluvium (Rock Type)
B_ArgillicminusEBlk.00t – Brimstone C_Argillic.00t – Cut 5	Argillized Rocks (Alteration)
B_OX-CN_080623MS.00t – Brimstone C_OX-CN_080623MS.00t – Cut 5	Oxide Zone (Redox)
Vulcan® Triangulation	**Geologic Zone**
B_PropminusEBlk.00t – Brimstone	Propylitically Altered Rocks (Alteration
B_SilicicMinusEBlk.00t – Brimstone C_Silicic.00t – Cut 5	Silicified Rocks (Alteration)
B_UnaltMinusEBlk.00t – Brimstone C_Unalt.00t	No Alteration

17.1.1.6 Tonnage Factors

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

Table 17.2 Brimstone Tonnage Factors

Geologic Zone	Tonnage Factor (ft^3/ton)
Alluvium	18.00
Backfilled Pits	20.00
Acid Leach	17.50
Oxide Silicic Alteration	13.70
Oxide Propylitic Alteration	14.00
Oxide Argillic Alteration	16.00
Sulfide Rocks	13.00
Unaltered	13.00
Undefined	14.25

17.1.1.7 Drillhole Compositing

Drill-hole assays were composited using 25-foot down-the-hole composites for the entire Hycroft Project. The start of the composite is the collar of the drillhole. If the down-hole length of the composite was less than 12.5 feet then no composite was generated. Assay values of -9 were ignored and then a new composite would be generated at that point. Assay values of -8 were set to 0.0001 opt for AuFA, AuCN and AgFA.

Geologic zone codes were added to the Brimstone 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 polygons on the same bench as the centroid of the composite.

17.1.1.8 Composite Statistics – Brimstone

Basic statistics were compiled for exploration drill-hole data using geologic codes transferred from the Brimstone geologic model polygons, as summarized in Table 17.3. Observations from these statistics include:

- The highest average gold grade is in the silicified unit with an average fire-assay grade of 0.4098 opt AuFA. This composite was mined out in the Vista mining days.
- The highest average grade in unaltered in situ rock is 0.288 opt AuFA.
- The highest grade in situ oxide is 0.228 in acid leach.

Table 17.3 Brimstone Composite Statistics (Imperial Units)

Brimstone	All Composites				Composites at 0.005 Cutoff AuFA			
Geologic Zone	Number	Average oz Au/t	Std Deviation	C.V	Number	Average oz Au/t	Std. Deviation	C.V.
Alluvium	838	0.0034	0.0059	1.7422	126	0.0138	0.0099	0.7140
Undefined	1471	0.0043	0.0067	1.5468	377	0.0110	0.0106	0.9624
Acid Leach	5982	0.0074	0.0120	1.6297	2071	0.0175	0.0159	0.9075
Silicic	5332	0.0114	0.0121	1.0605	3604	0.0157	0.0126	0.8063
Argillic	782	0.0112	0.0155	1.3844	414	0.0193	0.0177	0.9190
Propylitic	157	0.0103	0.0104	1.0077	107	0.0138	0.0108	0.7822
Unaltered	1310	0.0059	0.0077	1.2995	469	0.0134	0.0088	0.6564
Brimstone	**Oxide Composites**				**Oxide Composites at 0.005 Cutoff AuFA**			
Geologic Zone	**Number**	**Average oz Au/t**	**Std Deviation**	**C.V**	**Number**	**Average oz Au/t**	**Std. Deviation**	**C.V.**
Alluvium	786	0.0030	0.0060	1.7940	112	0.0143	0.0102	0.7163
Undefined	469	0.0050	0.0067	1.3504	142	0.0115	0.0092	0.8024
Acid Leach	5879	0.0074	0.0121	1.6295	2041	0.0176	0.0160	0.9074
Silicic	3245	0.0123	0.0137	1.1179	2196	0.0170	0.0144	0.8471
Argillic	463	0.0146	0.0189	1.2906	268	0.0238	0.0203	0.8517
Propylitic	25	0.0159	0.0191	1.2015	18	0.0212	0.0202	0.9545
Unaltered	824	0.0042	0.0070	1.6724	179	0.0135	0.0106	0.7905
Brimstone	**Sulfide Composites**				**Sulfide Composites at 0.005 Cutoff AuFA**			
Geologic Zone	**Number**	**Average oz Au/t**	**Std Deviation**	**C.V**	**Number**	**Average oz Au/t**	**Std. Deviation**	**C.V.**
Alluvium	52	0.0043	0.0046	1.0760	14	0.0102	0.0050	0.4948
Undefined	1002	0.0040	0.0067	1.6555	235	0.0107	0.0113	1.0587
Acid Leach	103	0.0047	0.0057	1.2146	30	0.0114	0.0067	0.5882
Silicic	2087	0.0100	0.0088	0.8773	1408	0.0135	0.0087	0.6464
Argillic	319	0.0063	0.0057	0.9116	146	0.0110	0.0053	0.4860
Propylitic	132	0.0092	0.0071	0.7754	89	0.0123	0.0067	0.5417
Unaltered	486	0.0088	0.0079	0.8938	290	0.0133	0.0074	0.5550

Lognormal cumulative frequency plots were compiled by geologic unit to further evaluate the gold grade distributions in drill-hole composites. The plots indicate a lognormal distribution at all grade ranges.

It is observed that the distributions of gold in the acid leach oxide units are similar but the acid-leach unit contains a larger fraction of mineralized material (Figure 17.1). Figure 17.2 shows that at the 0.005 Cutoff grade there a nearly equal amounts of Silicic and acid leach alteration.

Figure 17.1All Mineralized Brimstone Composites

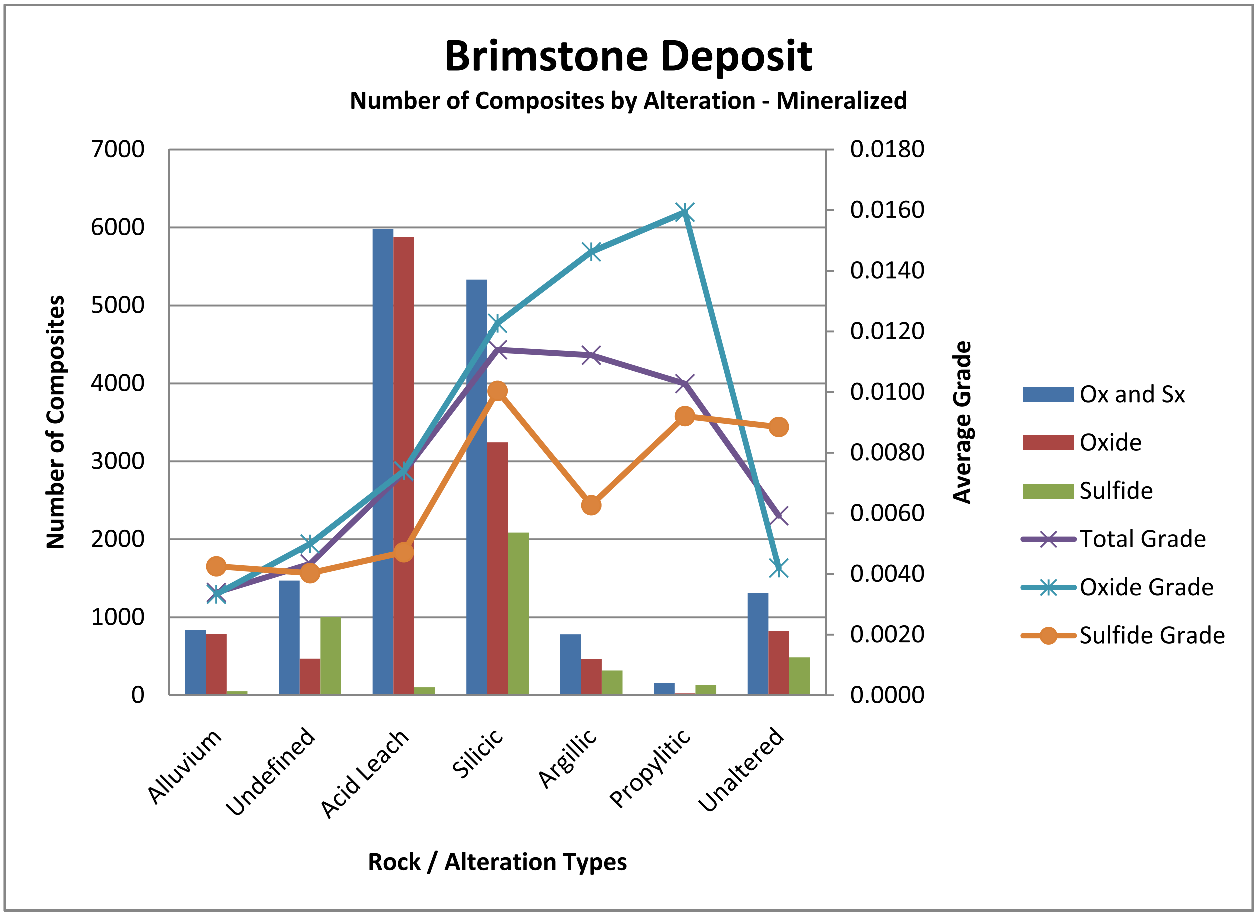
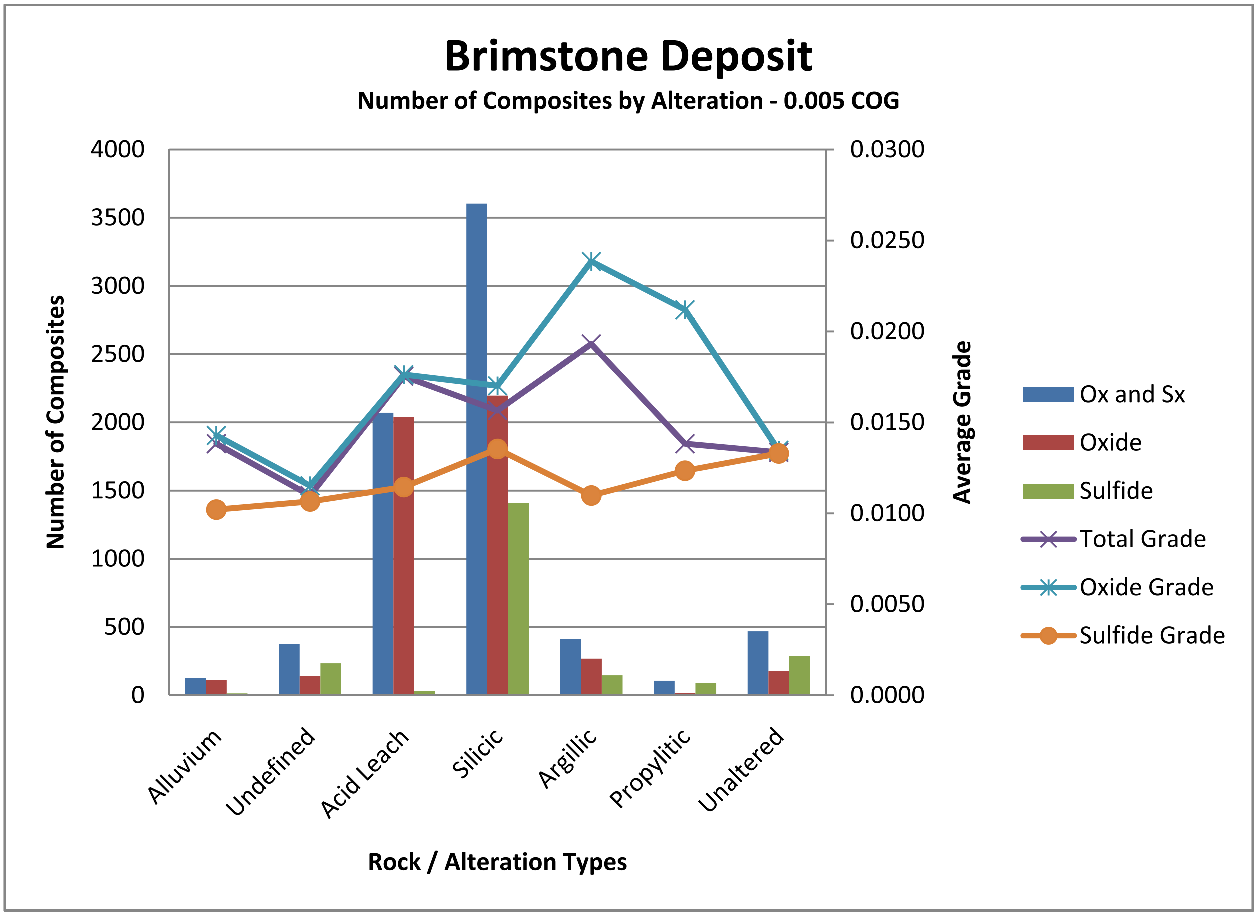


Figure 17.2Brimstone Composites at the 0.005 AuFA Cutoff Grade



17.1.1.9 Composite Statistics – Cut 5

Basic statistics were compiled for exploration drill-hole data using geologic codes transferred from the Brimstone geologic model polygons, as summarized in Table 17.4. Distributions of grade by rock type are shown in Figure 17.3 at the 0.000 cutoff grade and 17.4 at the 0.005 cutoff grade..

Table 17.4 Cut 5 Composite Statistics

Cut 5	All Composites				Composites at 0.005 Cutoff AuFA			
Geologic Zone	Number	Average oz Au/t	Std Deviation	C.V	Number	Average oz Au/t	Std. Deviation	C.V.
Alluvium	951	0.0083	0.0149	1.8027	414	0.0168	0.0194	1.1508
Undefined	312	0.0050	0.0060	1.1861	103	0.0117	0.0063	0.5395
Acid Leach	2923	0.0036	0.0064	1.7712	584	0.0122	0.0105	0.8602
Silicic	1612	0.0166	0.0202	1.2194	1451	0.0181	0.0208	1.1498
Argillic	282	0.0098	0.0154	1.5680	139	0.0181	0.0186	1.0260
Unaltered	7971	0.0108	0.0167	1.5533	5245	0.0153	0.0191	1.2495
Cut 5	Oxide Composites				Oxide Composites at 0.005 Cutoff AuFA			
Geologic Zone	Number	Average oz Au/t	Std Deviation	C.V	Number	Average oz Au/t	Std. Deviation	C.V.
Alluvium	803	0.0083	0.0160	1.9377	306	0.0192	0.0219	1.1371
Undefined	81	0.0062	0.0072	1.1566	34	0.0124	0.0073	0.5862
Acid Leach	2854	0.0035	0.0064	1.8232	532	0.0123	0.0109	0.8795
Silicic	788	0.0215	0.0259	1.2007	734	0.0229	0.0263	1.1465
Argillic	185	0.0127	0.0180	1.4229	107	0.0207	0.0202	0.9760
Unaltered	3939	0.0136	0.0218	1.5947	2657	0.0193	0.0246	1.2735
Cut 5	Sulfide Composites				Sulfide Composites at 0.005 Cutoff AuFA			
Geologic Zone	Number	Average oz Au/t	Std Deviation	C.V	Number	Average oz Au/t	Std. Deviation	C.V.
Alluvium	148	0.0081	0.0051	0.6640	108	0.0101	0.0046	0.4616
Undefined	231	0.0043	0.0054	1.2557	69	0.0113	0.0057	0.5054
Acid Leach	69	0.0083	0.0052	0.6352	52	0.0103	0.0044	0.4254
Silicic	824	0.0119	0.0108	0.9075	717	0.0131	0.0110	0.8350
Argillic	97	0.0044	0.0050	1.1452	32	0.0094	0.0060	0.6385
Unaltered	4032	0.0080	0.0086	1.0854	2588	0.0112	0.0093	0.8315

Figure 17.3 Grade and Rocktype Distributions at Cut 5 (All Mineralization)

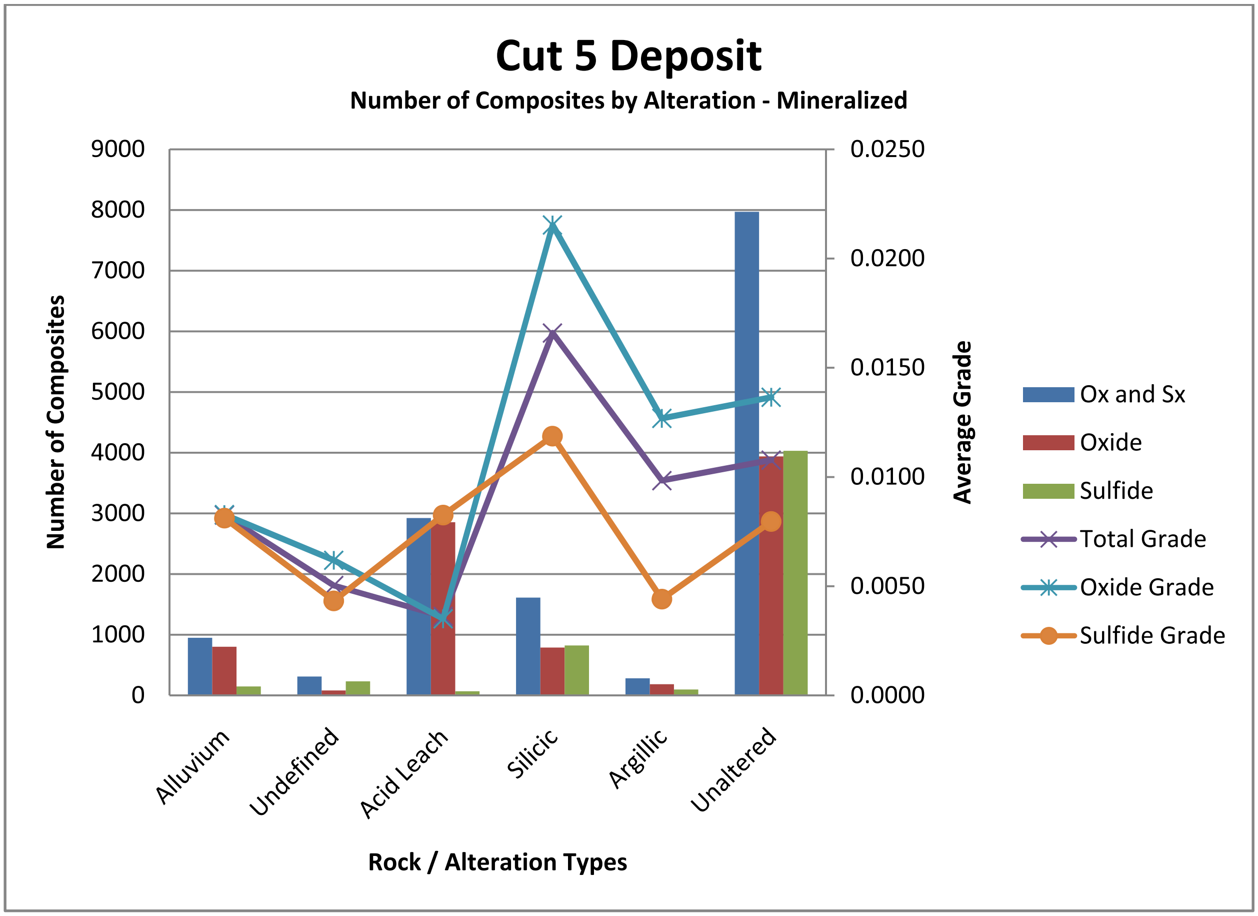
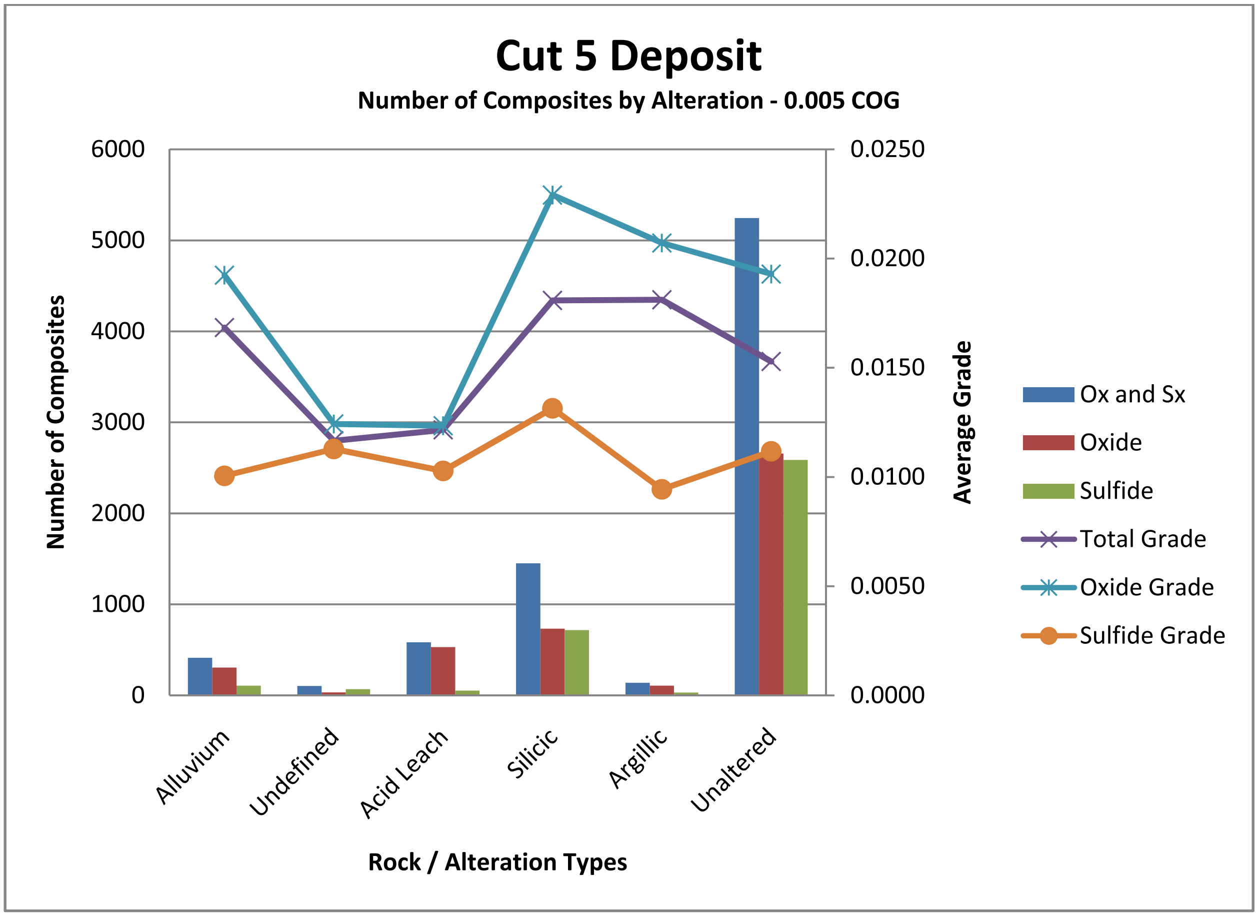


Figure 17.4Grade and Rocktype distribution at the 0.005 cutoff grade



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 25ft by 25ft by 25 ft cubed 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 drillhole, etc., are also stored in each individual block for descriptive evaluations. Physical information stored in the blocks can include 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 17.5. The model is saved as Feb09_5.*bmf.* All the blocks are orthogonal at the Selective Mining Unit of 25 feet.

Table 17.5 CombinedResource_aug08.bmf Dimensions

	East	North	Elevation
Minimum Mine Coordinates	16000	37000	2350
Maximum Mine Coordinates	24700	53000	5200
Number of Blocks	348	640	114

17.1.2.3 Hycroft Grade Model

17.1.2.3.1 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 17.6.

Table 17.6 Hycroft Variograms

Domain	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
Bay Area	320	-40	16	250	220	160	0.577	0.428	73
Boneyard	241	-58	-51	125	100	30	0.399	0.581	90
Brimstone	0	-10	0	152	145	106	0.158	0.906	120
Cut 5	40	-8	65	150	90	70	0.156	1.36	75
Camel Hill	159	-36	-72	265	140	80	0.553	0.703	165
Central Fault	90	0	0	265	125	80	0.126	0.878	115
Deep Sulfides	0	0	0	500	500	125	0.795	0.233	195

17.1.2.3.2 Gold Grade Estimation Parameters

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 had to be 12.5 feet, or half the SMU, in order to be used in the grade estimation run
- AuCN and AgFA grades were estimated using the gold variography parameters

The grade estimation search parameters are listed in Table 17.7.

Table 17.7 Estimation Sampling Parameters

Domain	Minimum Samples	Maximum Samples per Estimate	Maximum Samples per Hole	Discretization
Bay Area	1	9	3	4X4X1
Boneyard	1	9	3	4X4X1
Brimstone	1	10	4	4X4X1
Cut 5	1	10	4	4X4X1
Camel Hill	1	9	3	4X4X1
Central Fault	1	9	3	4X4X1
Deep Sulfides	1	9	3	4X4X1

17.1.2.3.3 Reconciliation of Resource Model to Validate Performance

There are no historic block models, except at Brimstone, to compare the accuracy of the resource model. However, grades were estimated using Nearest Neighbor and Inverse Distance Squared grade estimation techniques. All three techniques yield nearly the same grade and tonnage. SEWC believes kriging is the best approach to handle local variances of grades so this method was chosen to report grade and tonnages for the Hycroft Project.

The most recent grade estimate at Brimstone was complete by ORE in 2005. It is difficult to reconcile the total resource at Brimstone with the new resource model due to the fact that over 270 additional holes were used in the new Brimstone estimate. Since there is a pit design in existence for Brimstone that area of the resource can be checked for model performance. Table 17.8 lists the reconciliation of the ORE model to the SEW model.

Table 17.8 Reconciliation of ORE to SEWC Resource Model Estimation (US Imperial Units)

	ORE Model			SEWC Model			Difference of SEW to ORE		
Cut Off	Ktons	AuFA	K Oz	Ktons	AuFA	K Oz	Tons	Grade	Ounces
0.500	1,669	0.071	118	702	0.061	43	-138%	-16%	-177%
0.040	2,746	0.060	165	1,733	0.051	88	-58%	-18%	-86%
0.030	5,087	0.049	249	3,963	0.042	166	-28%	-17%	-50%
0.020	12,232	0.035	428	11,415	0.030	342	-7%	-17%	-25%
0.018	14,924	0.032	478	14,227	0.028	398	-5%	-14%	-20%
0.016	19,364	0.028	542	17,697	0.026	460	-9%	-8%	-18%
0.014	22,794	0.026	593	21,722	0.024	521	-5%	-8%	-14%
0.012	27,994	0.024	672	26,298	0.022	579	-6%	-9%	-16%
0.010	30,432	0.023	700	31,898	0.020	638	5%	-15%	-10%
0.009	33,342	0.022	734	34,973	0.019	664	5%	-16%	-10%
0.008	38,864	0.020	777	37,921	0.018	683	-2%	-11%	-14%
0.007	41,540	0.019	789	41,004	0.017	697	-1%	-12%	-13%
0.006	44,856	0.018	807	43,983	0.017	748	-2%	-6%	-8%
0.005	44,856	0.018	807	46,959	0.016	751	4%	-13%	-7%
0.004	53,546	0.016	857	50,637	0.015	760	-6%	-7%	-13%
0.003	60,628	0.014	849	55,626	0.014	779	-9%	0%	-9%
0.002	75,417	0.012	905	62,765	0.013	816	-20%	8%	-11%
0.001	82,366	0.011	906	74,138	0.011	816	-11%	0%	-11%
Total Pit	82,366	0.011	906	88,466	0.009	796	7%	-22%	-14%

The reconciliation shows that, at the cutoff grade of 0.005 the tons are fairly close to the ORE model estimate. There is substantial difference at other cutoff grades but these are attributed to the fact that the ORE model handles AuFA grades as an afterthought. The afterthought was because Vista was concerned with the modeling of recoverable gold rather than the entire metal content of the ore deposit. The SEWC model handles the total gold and silver content of the ore deposit.

17.1.2.3.4 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 2 drillholes in the estimate. Measured Mineral Resources require a minimum of 2 drillholes for the estimate where at least 1 hole is within d95 and one hole must be within half the d95 distance, expressed as d95/2. Table 17.9 identifies the Classification Criteria for the Hycroft ore deposits.

Table 17.9 Hycroft Resource Classification Criteria

Domain	Measured	Indicated Distance	Inferred Distance
Bay Area	< 36.5	36.5 – 73	> 73
Boneyard	< 62.5	62.5 – 115	> 115
Brimstone	< 60	60-120	> 120
Central Fault	< 82.5	82.5 – 165	> 165
Cut 5	< 37.5	37.5 – 75	> 75
Camel Hill	< 57.5	57.5 – 115	> 115
Deep Sulfides	< 97.5	97.5 – 195	> 195

17.1.2.3.5 Resource Summary - Gold

The remaining measured and indicated in situ gold resource as at March 31, 2009 is summarized in Table 17.10. The total inferred in situ resource is summarized in Table 17.11.

Table 17.10 Hycroft Measured and Indicated In Situ Gold Mineral Resources

March 31, 2009 Measured Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
0.17	Oxide Gold	218,894	0.42	2,986
0.45	Sulfide Gold	26,963	0.46	399
March 31, 2009 Indicated Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
0.17	Oxide Gold	175,008	0.37	2,109
0.45	Sulfide Gold	20,000	0.62	399
March 31, 2009 Measured and Indicated Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
0.17	Oxide Gold	393,901	0.40	5,095
0.45	Sulfide Gold	46,963	0.53	798

Table 17.11 Hycroft Inferred In Situ Gold Mineral Resources

March 31, 2009 Inferred Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
0.17	Oxide Gold	107,337	0.38	1,325
0.45	Sulfide Gold	218,524	0.62	4,371

17.1.2.3.6 Resource Summary - Silver

The remaining measured and indicated in situ silver resource is summarized in Table 17.12. The total inferred resource is summarized in Table 17.13.

Table 17.12 Measured and Indicated In Situ Silver Resources

March 31, 2009 Measured Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
12.41	Oxide Silver	49,296	26.50	41,997
12.41	Sulfide Silver	19,665	35.83	22,654
March 31, 2009 Indicated Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
12.41	Oxide Silver	21,088	41.97	28,455
12.41	Sulfide Silver	14,533	52.21	24,396
March 31, 2009 Measured and Indicated Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
12.41	Oxide Silver	70,385	31.13	70,452
12.41	Sulfide Silver	34,198	42.79	47,050

Table 17.13 Inferred In Situ Oxide Silver Resources

March 31, 2009 Inferred Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
12.41	Oxide Silver	11,503	51.27	18,962
12.41	Sulfide Silver	103,405	47.82	158,965

17.1.2.3.7 Resource Summary – Crofoot Pad Inferred Gold Resource

Allied Nevada drilled a widely spaced, 500 by 500 foot, grid of exploration holes to determine whether any mineral resources remain on the previously leached pad. The drilling, sampling assaying and QA/QC procedures, used for exploration, were applied to the Crofoot Pad drilling program. Several large, contiguous zones of identifiable mineralization were delineated on the closed heap leach pad. Ten representative samples were submitted to McClelland Laboratories Inc. for bottle roll testing to determine the gold recoveries for this material. Results are not available as of the publication of this technical report. SEWC believes that the widely spaced drilling, on the Crofoot Pad, is sufficient to establish continuity of mineralization; however, until metallurgical recoveries are established, this mineralization only meets the standards of Inferred Mineral Resources as defined by CIM 2005. Table 17.14 summarizes the tonnage and grade of the mineral resources on the Crofoot Pad.

Table 17.14 Crofoot Pad Oxide Inferred Mineral Resources

March 31, 2009 Crofoot Inferred Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
0.34	Crofoot Pad 1	7,118	0.44	110
0.17	Crofoot Pad 2	24,954	0.24	208

17.1.2.4 Resource Summary – Grade/Tonne Charts

Table 17.15 March 31, 2009 Oxide - Measured and Indicated - Gold

Cutoff	Measured Oxide Au			Indicated Oxide Au			M & I Oxide Au		
g Au/t	K Tonnes	g Au/t	K Oz	K Tonnes	g Au/t	K Oz	K Tonnes	g Au/t	K Oz
0.09	266,090	0.37	3,166	240,958	0.31	2,379	507,048	0.34	5,545
0.12	241,020	0.39	3,031	204,212	0.34	2,254	445,231	0.37	5,285
0.16	218,893	0.43	2,995	175,007	0.36	2,031	393,899	0.40	5,026
0.19	197,691	0.46	2,924	148,206	0.40	1,886	345,897	0.43	4,810
0.22	177,132	0.48	2,734	124,287	0.45	1,781	301,420	0.47	4,515
0.25	158,778	0.49	2,525	106,620	0.48	1,645	265,398	0.49	4,171
0.28	141,104	0.53	2,400	91,226	0.51	1,508	232,330	0.52	3,909
0.31	123,743	0.56	2,242	77,583	0.55	1,368	201,326	0.56	3,610
0.34	107,669	0.58	2,018	65,812	0.57	1,202	173,481	0.58	3,220
0.37	92,814	0.62	1,842	55,328	0.60	1,072	148,142	0.61	2,913
0.40	79,268	0.65	1,660	45,506	0.64	931	124,774	0.65	2,591
0.44	67,775	0.69	1,494	37,320	0.67	805	105,095	0.68	2,299
0.47	57,402	0.70	1,295	30,932	0.71	701	88,334	0.70	1,996
0.50	48,720	0.74	1,153	25,726	0.76	628	74,446	0.74	1,781
0.53	41,151	0.77	1,020	21,783	0.79	556	62,934	0.78	1,576
0.56	34,775	0.81	900	18,322	0.84	497	53,097	0.82	1,397
0.59	29,299	0.84	791	15,614	0.88	441	44,914	0.85	1,231
0.62	24,477	0.87	688	13,344	0.90	385	37,820	0.88	1,073

Table 17.16 March 31, 2009 Sulfide - Measured and Indicated - Gold

Cutoff	Measured Sulfide Au			Indicated Sulfide Au			M & I Sulfide Au		
g Au/t	K Tonnes	g Au/t	K Oz	K Tonnes	g Au/t	K Oz	K Tonnes	g Au/t	K Oz
0.09	184,784	0.34	2,037	449,558	0.31	4,460	634,342	0.32	6,497
0.12	169,426	0.38	2,054	399,450	0.34	4,403	568,876	0.35	6,458
0.16	156,299	0.41	2,067	355,852	0.34	3,923	512,151	0.36	5,990
0.19	142,085	0.41	1,879	310,990	0.38	3,771	453,076	0.39	5,650
0.22	127,189	0.45	1,823	265,768	0.41	3,516	392,958	0.42	5,338
0.25	112,777	0.48	1,740	224,026	0.45	3,210	336,803	0.46	4,951
0.28	97,143	0.48	1,499	184,019	0.45	2,637	281,161	0.46	4,136
0.31	82,690	0.51	1,367	149,364	0.48	2,305	232,055	0.49	3,672
0.34	69,860	0.55	1,232	118,791	0.51	1,964	188,651	0.53	3,196
0.37	58,271	0.58	1,092	93,735	0.55	1,653	152,006	0.56	2,745
0.40	48,208	0.62	957	73,892	0.58	1,385	122,100	0.60	2,341
0.44	39,940	0.65	836	57,809	0.65	1,211	97,749	0.65	2,047
0.47	33,235	0.69	733	45,263	0.69	998	78,498	0.69	1,731
0.50	27,432	0.72	635	35,541	0.72	823	62,974	0.72	1,458
0.53	22,378	0.75	543	27,972	0.75	678	50,350	0.75	1,221
0.56	18,065	0.79	458	22,045	0.79	559	40,110	0.79	1,017
0.59	14,528	0.82	384	17,551	0.82	464	32,080	0.82	849
0.62	11,683	0.86	322	13,960	0.89	400	25,642	0.88	722

Table 17.17 March 31, 2009 Oxide - Measured and Indicated - Silver

Cutoff	Measured Oxide Ag			Indicated Oxide Ag			M & I Oxide Ag		
g Au/t	K Tonnes	g Ag/t	K Oz	K Tonnes	g Ag/t	K Oz	K Tonnes	g Ag/t	K Oz
15.55	32,285	32.65	33,890	13,865	56.09	25,003	46,150	39.69	58,893
31.10	6,756	68.19	14,810	4,930	116.20	18,416	11,685	88.44	33,226
46.66	2,264	122.97	8,952	1,445	291.66	13,551	3,709	188.69	22,503
62.21	1,169	183.76	6,908	1,024	386.89	12,740	2,194	278.61	19,648
77.76	758	241.81	5,896	928	419.04	12,503	1,686	339.34	18,399
93.31	533	304.55	5,223	857	445.83	12,288	1,391	391.64	17,511
108.86	445	342.86	4,900	792	473.60	12,059	1,236	426.60	16,958
124.41	404	365.74	4,747	730	503.12	11,813	1,134	454.21	16,559
139.97	368	386.44	4,576	715	510.95	11,743	1,083	468.61	16,319
155.52	315	424.41	4,295	704	516.32	11,686	1,019	487.92	15,981
171.07	283	452.10	4,114	667	534.91	11,467	950	510.23	15,581
186.62	269	466.31	4,026	662	537.59	11,446	931	517.02	15,472
202.17	254	480.19	3,921	643	547.44	11,320	897	528.40	15,242
217.72	234	500.98	3,770	578	582.87	10,829	812	559.27	14,599
233.28	221	516.09	3,673	557	595.56	10,665	778	572.96	14,338
248.83	209	531.55	3,566	542	604.13	10,537	751	583.97	14,102
264.38	193	550.55	3,420	534	609.27	10,467	728	593.68	13,887
279.93	179	569.82	3,274	524	615.10	10,369	703	603.59	13,643
295.48	165	591.05	3,137	513	621.96	10,249	678	614.43	13,387
311.04	155	607.55	3,030	499	629.50	10,098	654	624.30	13,128

Table 17.18 March 31, 2009 Sulfide - Measured and Indicated - Silver

Cutoff	Measured Sulfide Ag			Indicated Sulfide Ag			M & I Sulfide Ag		
g Au/t	K Tonnes	g Ag/t	K Oz	K Tonnes	g Ag/t	K Oz	K Tonnes	g Ag/t	K Oz
15.55	24,938	42.17	33,813	27,402	72.69	64,037	52,341	58.15	97,849
31.10	8,121	80.57	21,037	10,873	147.43	51,540	18,995	118.85	72,577
46.66	3,835	124.46	15,344	6,177	229.04	45,484	10,012	188.98	60,828
62.21	2,139	176.92	12,167	3,484	360.01	40,331	5,624	290.37	52,498
77.76	1,399	230.06	10,347	2,751	435.78	38,549	4,150	366.45	48,896
93.31	984	286.98	9,081	2,310	501.27	37,223	3,294	437.24	46,304
108.86	721	351.78	8,157	2,022	557.16	36,221	2,743	503.17	44,378
124.41	582	405.95	7,589	1,953	572.24	35,934	2,535	534.09	43,523
139.97	514	440.24	7,280	1,911	581.85	35,756	2,426	551.82	43,036
155.52	459	473.50	6,988	1,867	591.45	35,501	2,326	568.17	42,488
171.07	445	483.10	6,904	1,818	602.76	35,230	2,263	579.25	42,134
186.62	435	489.96	6,845	1,777	612.02	34,968	2,212	588.04	41,813
202.17	406	509.16	6,638	1,715	626.07	34,530	2,121	603.72	41,168
217.72	388	521.84	6,514	1,675	636.02	34,243	2,063	614.53	40,757
233.28	374	532.13	6,394	1,648	641.85	34,014	2,022	621.57	40,408
248.83	357	545.50	6,253	1,620	648.70	33,791	1,977	630.09	40,044
264.38	327	570.19	5,987	1,550	665.16	33,155	1,877	648.64	39,141
279.93	317	578.76	5,891	1,516	673.39	32,818	1,833	657.04	38,710
295.48	309	584.59	5,814	1,486	680.59	32,514	1,795	664.05	38,328
311.04	298	594.53	5,688	1,452	688.82	32,164	1,750	672.79	37,852

Table 17.19 March 31, 2009 Oxide - Inferred - Gold

	Inferred Oxide Au		
Cutoff	K Tonnes	g Au/t	K Oz
0.09	142,950	0.33	1,501
0.12	123,778	0.36	1,440
0.16	107,339	0.40	1,369
0.19	92,075	0.41	1,218
0.22	79,393	0.45	1,138
0.25	68,156	0.48	1,052
0.28	57,366	0.51	949
0.31	46,930	0.55	828
0.34	39,915	0.58	748
0.37	32,669	0.64	669
0.40	27,375	0.67	591
0.44	22,441	0.72	523
0.47	18,660	0.76	455
0.50	15,443	0.81	403
0.53	13,185	0.86	366
0.56	11,433	0.90	331
0.59	10,160	0.93	305
0.62	9,143	0.95	280

Table 17.20 March 31, 2009 Sulfide - Inferred - Gold

	Inferred Sulfide Au		
Cutoff	K Tonnes	g Au/t	K Oz
0.09	952,937	0.31	9,454
0.12	853,844	0.34	9,412
0.16	765,293	0.34	8,436
0.19	665,915	0.38	8,075
0.22	563,919	0.41	7,459
0.25	474,084	0.45	6,794
0.28	378,942	0.48	5,848
0.31	299,267	0.51	4,948
0.34	236,263	0.55	4,167
0.37	185,367	0.58	3,474
0.40	144,807	0.62	2,873
0.44	114,454	0.69	2,523
0.47	91,576	0.72	2,120
0.50	72,847	0.79	1,847
0.53	58,698	0.82	1,553
0.56	47,347	0.89	1,357
0.59	40,624	0.93	1,209
0.62	35,005	0.96	1,080

Table 17.21March 31, 2009 Oxide - Inferred - Silver

	Inferred Oxide Ag		
Cutoff	K Tonnes	g Ag/t	K Oz
15.55	10,178	55.96	18,311
31.10	8,129	64.03	16,735
46.66	1,537	147.84	7,304
62.21	1,012	194.43	6,329
77.76	721	240.95	5,587
93.31	529	292.75	4,978
108.86	344	391.03	4,322
124.41	233	519.13	3,891
139.97	230	523.95	3,881
155.52	226	530.31	3,851
171.07	199	579.68	3,703
186.62	191	595.65	3,648
202.17	177	624.18	3,550
217.72	153	683.88	3,371
233.28	150	695.68	3,348
248.83	149	698.42	3,341
264.38	148	701.16	3,333
279.93	145	709.39	3,310
295.48	142	714.88	3,273
311.04	140	723.11	3,248

Table 17.22March 31, 2009 Sulfide - Inferred - Silver

	Inferred Sulfide Ag		
Cutoff	K Tonnes	g Ag/t	K Oz
15.55	33,518	78.17	84,241
31.10	17,373	128.92	72,008
46.66	8,421	221.15	59,875
62.21	6,147	281.49	55,631
77.76	3,951	394.98	50,170
93.31	2,976	493.73	47,232
108.86	2,343	597.62	45,022
124.41	2,024	672.36	43,750
139.97	1,998	679.22	43,622
155.52	1,860	717.62	42,907
171.07	1,579	813.97	41,331
186.62	1,568	818.77	41,265
202.17	1,548	826.65	41,132
217.72	1,468	859.22	40,547
233.28	1,452	865.74	40,425
248.83	1,442	870.20	40,329
264.38	1,430	875.00	40,220
279.93	1,415	880.83	40,076
295.48	1,400	887.34	39,933
311.04	1,382	894.20	39,720

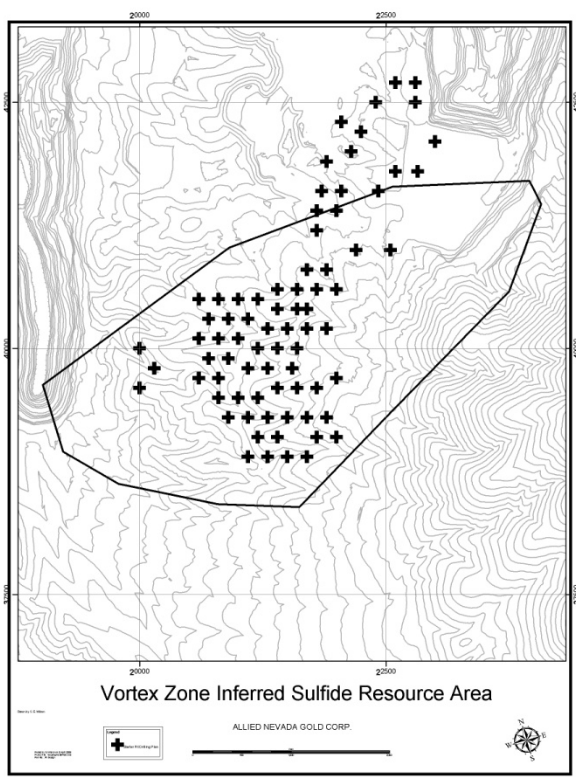
17.1.3 VORTEX ZONE – NEWLY DEFINED SUBSET OF HYCROFT RESOURCE

The results of the deep sulfide drilling program identified an area south of Brimstone that has been named the Vortex. There were several successful drilling intercepts in this area that suggest a large blanket of sulfide mineralization exists directly beneath the oxide mineralization in this area. There is evidence that a significant portion of the 4 million inferred sulfide ounces can be converted to a higher confidence category with infill drilling. SEWC recommends that Allied Nevada drill this zone at a high enough density of drillholes to significantly expand the known indicated mineral resources at Hycroft. Table 17.23 lists the inferred mineral resources within the deep Vortex Zone. Figure 17.5 shows the location of the Vortex Zone at Hycroft.

Table 17.23 Vortex Zone Inferred Resources

March 31, 2009 Vortex Inferred Resource				
Cutoff	Type	Tonnes	Grade g/t	Ounces
0.45	Sulfide Gold	61,973	0.78	1,548
12.41	Sulfide Silver	48,080	68.14	105,328

17.5Vortex Zone with Suggested Drilling Targets



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 and Cut 5 deposits were considered for reserve level determinations. The reported reserves meet 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 have 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 do not have 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 Whittles' Lerchs-Grossman© algorithm.
- Export pit shells from Whittle® and import into Vulcan®.
- Use pit shells as guides to design 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 17.23. Physical design parameters for the Reserve Pits are shown in Table 17.25.

Table 17.24 Economic Design Parameters

Value	Description	Units (US$)
$750	Gold price	$/oz
$1.52	Cost of mining	$/ton
$1.40	Cost of processing	$/ton ore
$0.22	Cost of administration, Jungo road, environmental, reclamation	$/ton ore
56.6%	AuFA gold recovery	%

Table 17.25 Pit Design Parameters

Description	Value
Slope Angle	46 degrees
Bench Height	25 feet
Road Width	90 -110 feet
Maximum Loaded Ramp Grade	13%
Minimum Mining Width	200 feet
Tonnage Factor ft3/ton	13-20

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 Cutoff Grades

SEWC evaluated the cutoff grade based on the current costs associated with the mining of the Brimstone Deposit. SEWC used a cutoff grade of 0.17 g Au/t and a Gold selling price of $650 USD (the approximate 3 year average London Fixed Gold Price) to verify reserves for the Brimstone Pit.

17.2.6 Hycroft Mine Mineral Reserves Statement

Table 17.26 October 17, 2008 Hycroft Mineral Reserve Estimate

Material	Tonnes X 1000	g Au/t	Au Ounces
Proven Reserve	42,236	0.55	747,831
Probable Reserve	24,133	0.51	395,347
Total Reserve	66,369	0.54	1,143,178
Waste	86,366		
Total Pit Tonnes	152,735		
Strip Ratio	1.29		

18 OTHER RELEVANT DATA AND INFORMATION

There is no other data or information that could be relevant to the Hycroft Mine.

19 INTERPRETATIONS AND CONCLUSIONS

SEWC reviewed pertinent data from the Hycroft Gold 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:

- Exploration drilling, sampling, sample preparation, assaying, density measurements and drillhole 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 Gold's Mineral Resource and Mineral Reserve Statements.

20 RECOMMENDATIONS

The Vortex Zone of the Hycroft Deposit is a large new addition to the Hycroft Inferred resources. The shape of mineralization is unknown but the lateral extents are continuous enough to warrant a drilling program to convert this sulfide silver and gold deposit into a higher confidence category; indicated mineral resources at minimum. SEWC recommends a resource development drilling program to confirm the robust nature of the Vortex Zone.

20.1 Vortex Sulfide Resource Development Plan

- Sixty RC holes
- 20 Diamond Core Holes
    - Cost $7 Million USD
    - Plan to commence as soon as practical to be finish by the fourth quarter 2009

Success of this phase would be measured by a conversion of Inferred Mineral Resources to the Indicated Mineral Resource Category (Figure 20.1).

20.2 Sulfide Resource Development Plan

Combine samples from the 20 Core drilling program into a representative sample for future metallurgical testing.

- Cost $400,00 USD

20.3 Hard Bottom Drilling Plan

The large increase in Oxide Resources, as compared to the October 2007 Technical report, is due in large part to the addition of the entire drilling data set for resource analysis. The north end of the Central Fault, the Bay Area and the Boneyard deposits all contribute to the Measured and Indicated Resources at Hycroft. These areas were among the first parts of the Hycroft deposit to be mined. Topography and mining progress was kept in hard copy form and never converted to digital formats. It is known that some parts of the three areas were backfilled in areas. Since those areas are uncertain, SEWC recommends that a drilling program (Figure 20.2) be implemented to confirm the hard bottom and unmined surface. This plan would entail the drilling of several strategically located drillholes meant to give SEWC the confidence to pull remaining Measured and Indicated Mineral resources into the Proven and Probable Reserve category.

- Cost $500,000 USD

Figure 20.1 Phase 1 Sulfide and Vortex Development Plan

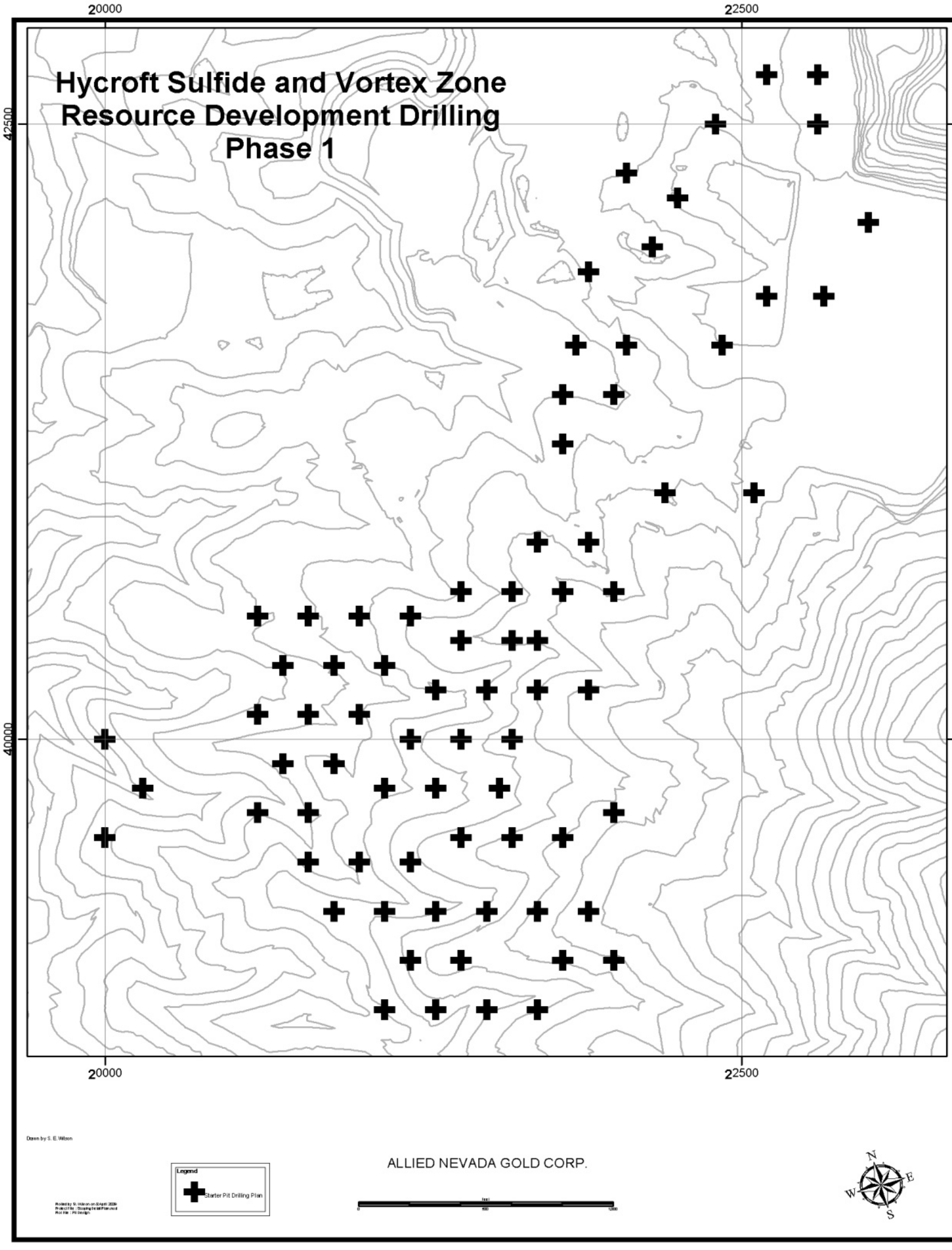
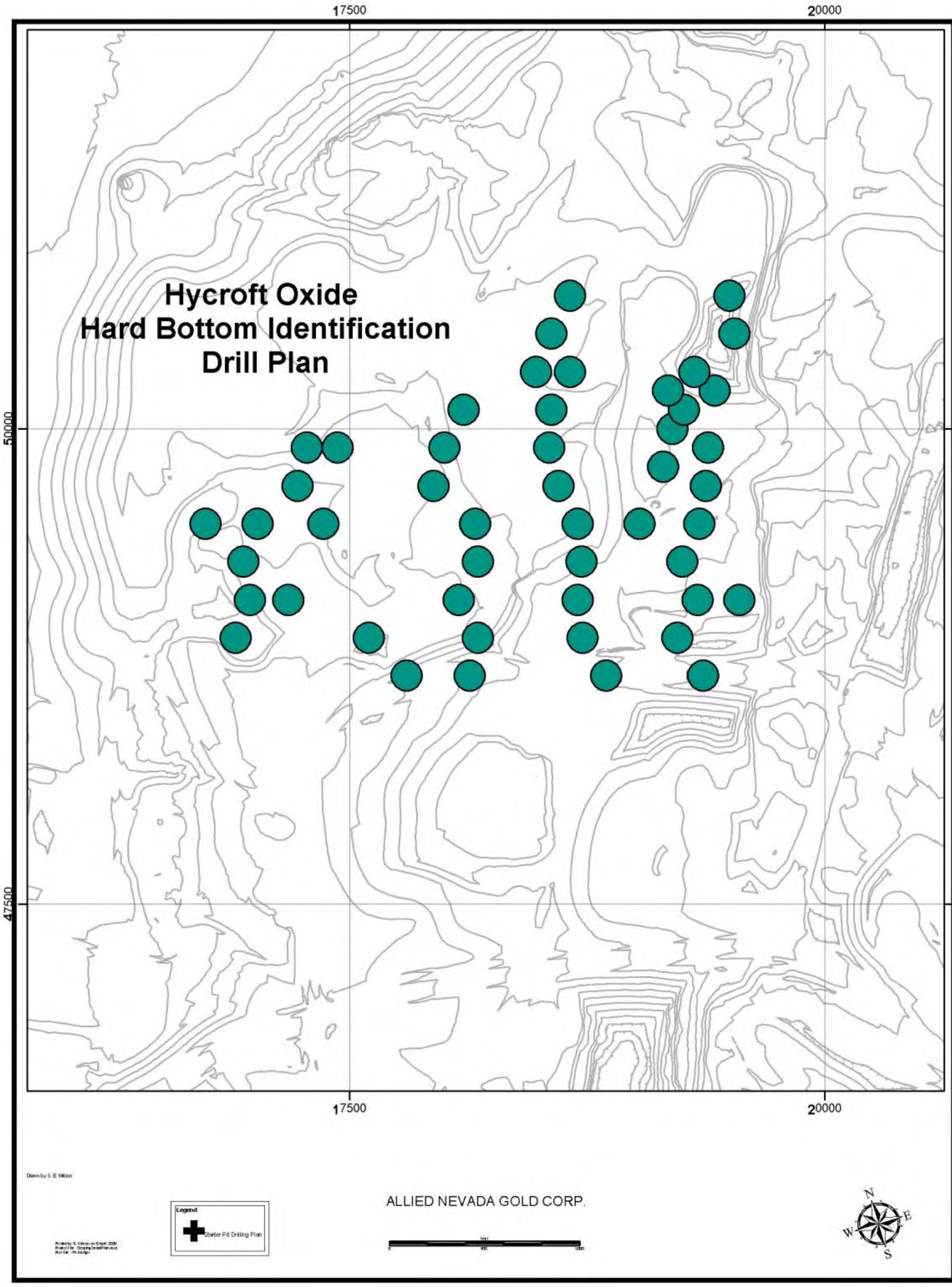


Figure 20.2Drill Plan to convert Oxide to Proven and Probable



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22 DATE

The effective date of this report is May 15, 2009.

23 ADDITIONAL REQUIREMENTS FOR DEVELOPING OR PRODUCING PROPERTIES

The Hycroft cash flow models are base on forecast and actual costs. All costs, volumes and grades in section 23 are reported in US Imperial units.

23.1 OPEN PIT MINING OPERATIONS

All mining from currently scheduled ore reserves will be conducted by open pit methods. The reserves will be mined entirely from two pits; the Brimstone Pit and the Cut 5 pit. Brimstone will be mined in five mining phases. Cut 5 will be mined from the top down in one single sequence. The following pit design criteria are used:

- 25 ft bench height on primary stripping benches and final walls
- Inter-ramp wall slope angles of approximately 50° depending on wall orientation or geology
- Bench face angles ranging from 55° to 60° depending on material type
- 13% maximum haul road grade
- 110 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 was successfully mined and has remained intact with these parameters.

The mining is accomplished with a typical drill, blast, load, and haul cycle. All mine material is blasted and the ore is hauled and direct dumped on the Brimstone Leach 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 blasthole assays. A production block model is built using inverse distance numerical modeling of the blasthole 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 blasthole 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 2009 and quarterly, thereafter, through the life of the mine. The plans include scheduling of major equipment to make sure that forecast ounce flows are viable.

Mining operations will be conducted 24 hours per day, seven days per week. Leach grade ore will be placed directly on heap leach pads. Waste will be dumped on permitted dump locations and in some cases historically mined pits will be filled with waste.

The Brimstone pit and site facilities are shown in Figure 23.1. Mining phases for Brimstone are shown in Figure 23.2.

Figure 23.1 Pit Site and Facilities

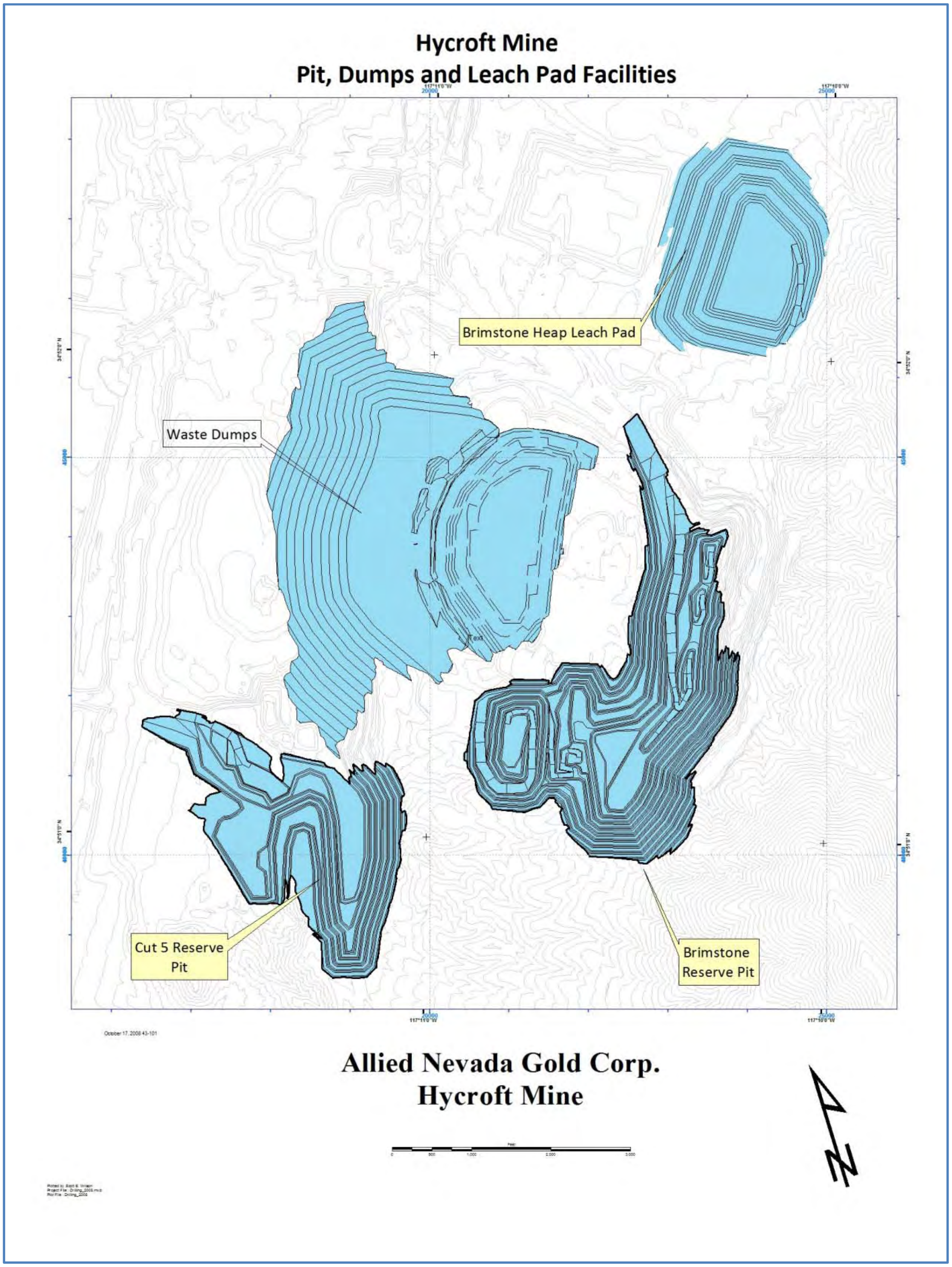
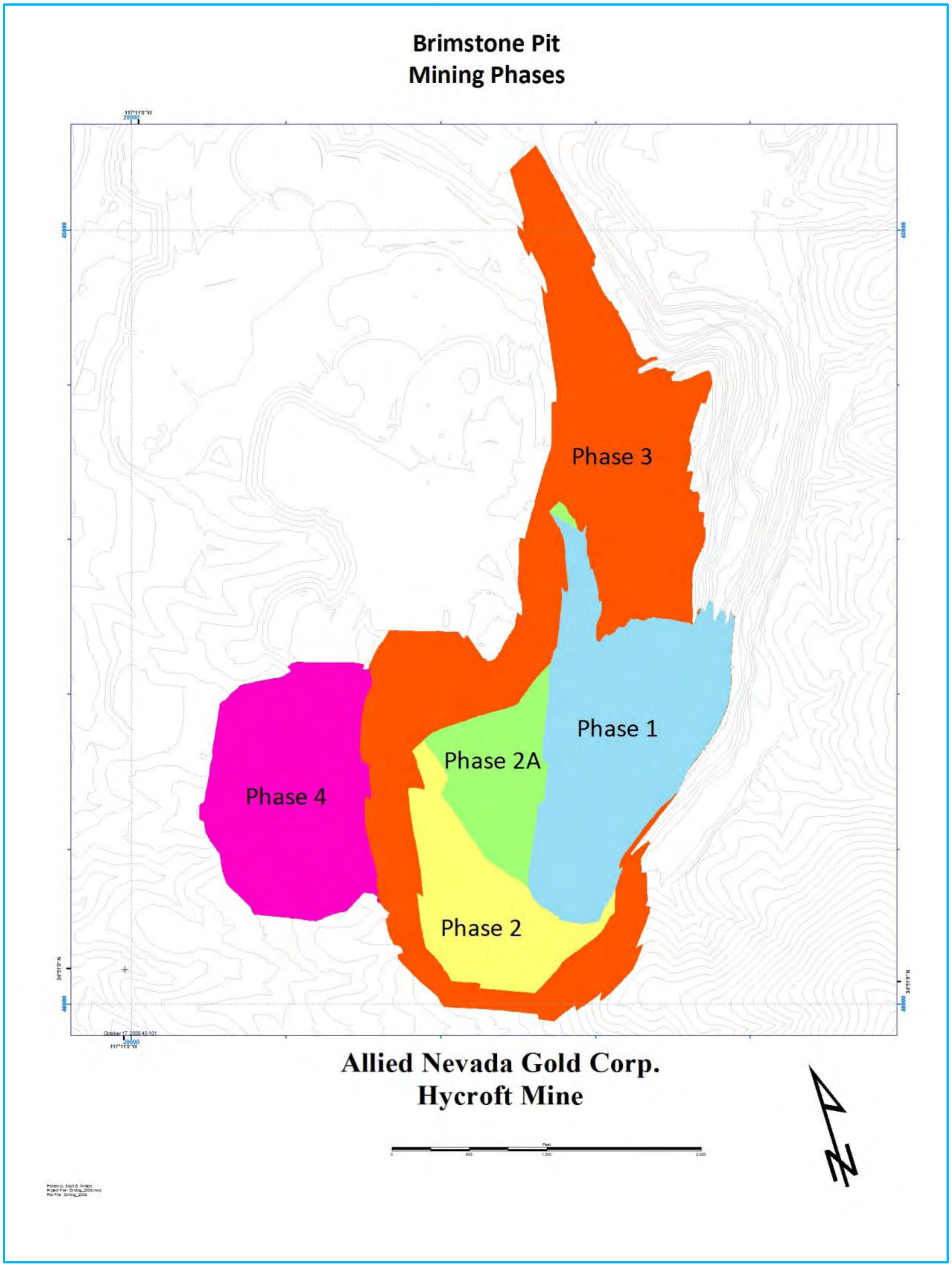


Figure 23.2 Hycroft Mining Phases



23.2 MINING FLEET

Allied Nevada is mining with a Komatsu Mining equipment fleet. Included in the fleet are five 730E haul trucks (two hundred ton class trucks), two WA1200 wheel loaders, one WA900 wheel loader, one WA380 wheel loader, one D475 bulldozer, two D375-A bulldozers, one D65-W bulldozer, one Drill Tech D45KS blasthole drill, two ATLAS-COPCO DML blasthole drills and one PC200LC excavator. There is also an older backup fleet that comprises one Caterpillar 16G motor grader, one Caterpillar 994 wheel loader, two Dresser 385-M water trucks and one Drill Tech D65W blasthole drill.

23.3 PROCESSING AND RECOVERIES

Mined ore will be placed on the Phase II expansion of the Brimstone pad. There will be an additional 6 million square feet of pad constructed to accommodate the Brimstone ore. Pregnant solutions were applied in the 4th quarter 2008.

Merrill Crowe solution capacity is approximately 3,000 gpm. Ponds and pumping stations were modified to increase efficiency by being able to direct solutions to the pads or to the zinc plant. This gives Hycroft flexibility to maximize the pregnant solution grades and to minimize downtime.

A new refinery has been constructed in close proximity to the Merrill Crew facility. Dore will be produced onsite. The refinery will meet Nevada emission standards for mercury.

Expected recoveries (based on past performance) for the Brimstone ore is 56.6% in the first 455 days; 42.1% will be recovered in the first 90 days and 52.0% recovered in the first 180 days.

23.4 PERSONNEL

Table 23.1 lists the personnel for the mine. These employees are responsible for the daily operation of the mine, leach pads, processing facilities and technical support. Allied Nevada is providing corporate support for accounting, purchasing and other non site essential activities. 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 transportation to the mine site.

Table 23.1 Hycroft Mine Personnel

Item	Number
Mining	
Supervision	10
Operators	77
Maintenance	19
Subtotal Mine Operations	**106**
Mine Engineering and Geology	
Engineers	2
Geologists	2
Surveyors	2
Subtotal Engineering and Geology	**6**
Processing	
Supervision	6
Operators	10
Maintenance	7
Assayers and Refiners	12
Subtotal Processing	**35**
Administration	
General Manager	1
Mine Manager	1
Office Manager	1
Senior Accountant	1
Accounting Clerks	1
Purchasing	3
Human Resources	2
Warehouse	3
Safety	6
Environmental	2
Information Technology	1
Utility Maintenance	2
Subtotal Administration	**24**

23.5 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 $14,343,100, which has been approved by both the Nevada Department of Environmental Protection and Bureau of Land Management. The financial guarantee bond has been submitted to the BLM.

23.6 TAXES AND MARKETS

Cash flows and analysis for the Hycroft Mine are developed for this report on a pretax basis at a gold selling price of $650. However, there are two taxes that are applicable to the Hycroft Mine:

- Income Tax – The rate applicable to taxable income from mining operations is 35%
- Nevada Net Proceeds Tax – The rate applicable to gold sales is 2.2%

23.7 CAPITAL AND OPERATING COST ESTIMATES

In On April 1, 2008, Allied Nevada entered into an underwriting agreement where underwriters purchased 14,375,000 common shares of ANV stock which generated net proceeds of approximately $69.3 Million. The proceeds are currently being consumed to fund mining operations at Hycroft until the mine produces positive cash flow from gold production. Cash flow is projected to be $72.3 million over 8 years. Allied Nevada's contracts for mining, and leach pad construction are within industry norms. Table 23.2 lists the operating costs for the Hycroft Project. Table 23.3 shows the capital costs anticipated for the project.

Table 23.2 Operating Costs (Imperial Units)

Tons of Ore Processed (000's)	73,495
Grade AuFA/ton	0.016
Gold Recovered (Ounces 000's)	650
Total Revenue (US $ 000's)	$422,525
Revenue Per Ton Processed @ $650 Au	$5.75
Cost per Ton Mined	$1.14
Processing Cost per Ton Processed	$1.05
General and Administration Cost per Ton Mined	$0.32
Mining Cost per Ton of Ore Processed	$2.59
Pre Tax Internal Rate of Return	55%

Note: No silver was used in the economic evaluation of the Hycroft Project Mine Plan.

Table 23.3 Capital Expenditures (thousands of dollars)

Capital	$34,892
Truck Components	$ 7,049
Loader Components	$ 4,432
Total Capital Cost	$46,373

23.8 ECONOMIC ANALYSIS

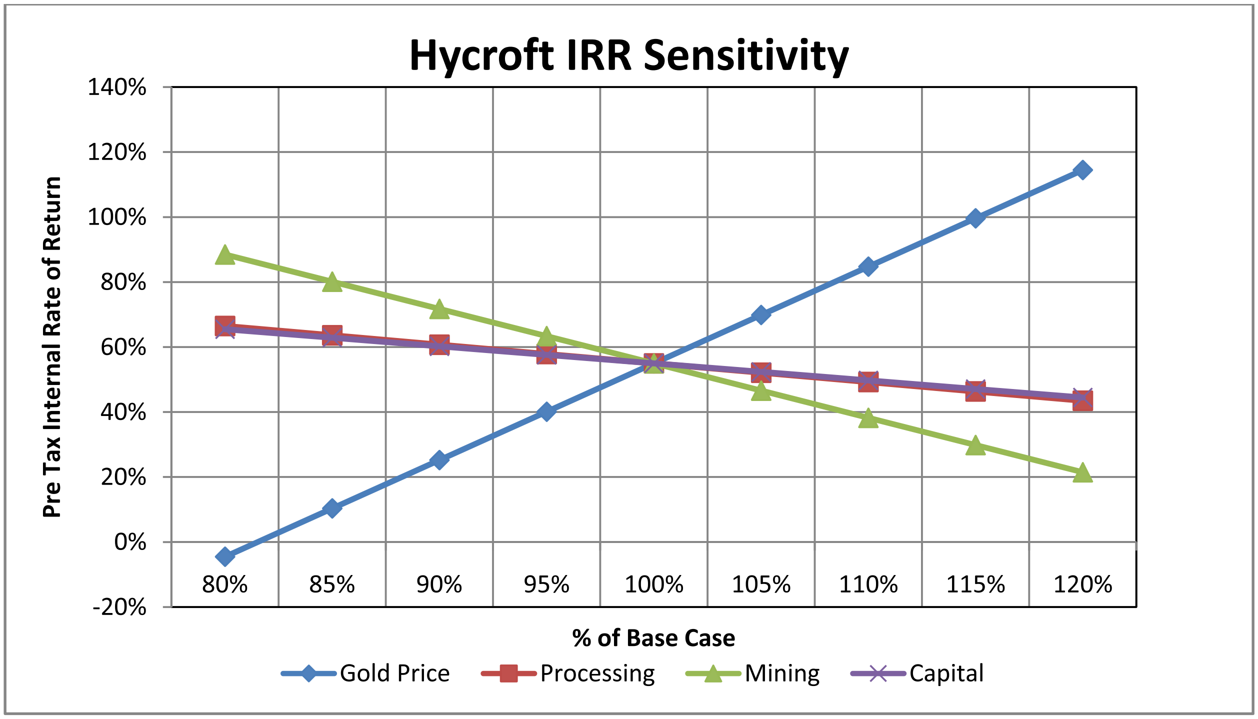
The Life-of-Mine cash flow schedule for the Hycroft property is listed in Table 23.4. The pre tax IRR sensitivity analysis for the Hycroft Mine is shown in Figure 23.3. The graph shows that the project is most sensitive to gold price and mining cost. The estimated pre tax IRR of the project is 55% at a gold selling price of $650.

The payback period for the Hycroft re-commissioning is approximately 42 months. Life of mine the project will generate and NPV of $45.4 Million at a 10% discount rate.

Table 23.4Hycroft Project Anual Cashflow Summary (US Imperial Units)

MATERIAL SUMMARY	2008	2009	2010	2011	2012	2013	2014	2015	TOTAL
Brimstone Pad from Pit									
Tons	1,567,740	11,138,056	10,286,638	14,317,049	13,246,725	13,514,060	7,316,466	2,108,842	**73,495,575**
Grade	0.005	0.010	0.011	0.008	0.008	0.009	0.008	0.010	**0.009**
Ounces Gold	8,199	106,819	112,678	110,238	107,939	121,071	58,483	21,612	**647,039**
Flow-Through Ounces	3,000								**3,000**
Total Process Ounce Placed	11,199	106,819	112,678	110,238	107,939	121,071	58,483	21,612	**650,039**
Total Process Ounce Poured	2,815	87,682	110,621	100,926	115,699	118,802	71,016	42,478	**650,039**
Pit Tons Mined	9,753,047	28,558,220	24,905,216	25,267,792	25,740,548	25,130,579	24,978,930	2,849,159	**167,183,490**
Pit Tons Mined (TPD)	27,017	79,109	68,990	69,994	71,303	69,614	69,194	7,892	
Pit Stripping Ratio	5.22	1.56	1.42	0.76	0.94	0.86	2.41	0.35	**1.27**
FINANCIAL SUMMARY									
Cost of Production:									
Mining	$9,807	$34,290	$30,117	$28,066	$28,497	$28,951	$26,739	$3,586	**$190,052**
Ore Processing	$2,193	$11,398	$10,948	$13,079	$12,513	$12,654	$9,377	$4,655	**$76,817**
General Plant	$40	$163	$163	$163	$163	$163	$163	$9	**$1,030**
Administration	$820	$3,611	$3,509	$3,449	$3,449	$3,449	$3,389	$1,046	**$22,723**
Freight, refining and insurance	$14	$427	$539	$492	$564	$579	$346	$207	**$3,169**
Direct Mining Expenses	$12,874	$49,890	$45,276	$45,249	$45,187	$45,797	$40,015	$9,504	**$293,791**
CAPEX Total	$2,209	$14,220	$13,784	$4,550	$7,258	$2,952	$1,399	$0	**$46,373**
Misc. Costs									
Royalties	$30	$120	$120	$120	$120	$120	$120	$30	**$780**
Salvage								($5,000)	**($5,000)**
Total Costs	$15,113	$64,230	$59,180	$49,919	$52,565	$48,869	$41,534	$9,534	**$340,944**
Revenue @ Au Price of $650	$1,830	$56,993	$71,904	$65,602	$75,205	$77,221	$46,160	$27,611	**$422,525**
Cash Flow	($13,283)	($7,237)	$12,723	$15,683	$22,640	$28,352	$4,627	$23,077	**$86,582**
Descriptive Statistics:									
Pit Mining Cost / Ton	$1.01	$1.20	$1.21	$1.11	$1.11	$1.15	$1.07	$1.26	**$1.14**
Ore Processing Cost / Ton	$1.40	$1.02	$1.06	$0.91	$0.94	$0.94	$1.28	$2.21	**$1.05**
G&A Cost / Ton	$0.55	$0.34	$0.36	$0.25	$0.27	$0.27	$0.49	$0.50	**$0.32**
Mining Cost / Ore Ton Processed	$6.26	$3.08	$2.93	$1.96	$2.15	$2.14	$3.65	$1.70	**$2.59**
Net Present Value Cash Flow 10%	**$45,416**								
Total Cash Flow	**$86,582**								
Total Ounces	**650,039**								
Pre Tax IRR	**55%**								

Figure 23.3 IRR Sensitivity



23.9 MINE LIFE DISCUSSION AND EXPLORATION POTENTIAL

The Proven and Probable Mineral Reserves at the Hycroft mine as of October 17, 2008 totals 73.2 Mt containing 1,143,178 ounces of gold at an average grade of 0.016 opt. This is sufficient for an approximate mine life of 6.5 years and an approximate gold production of 8 years.

There is excellent potential to add mine life to 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. SEWC believes Allied Nevada will add oxide reserves and resources by increasing the density of drilling near the Brimstone and Albert zones.

The Cut4 and Cut5 areas have drilling intercepts of higher grade mineralization. Allied Nevada has planned holes to quantify the potential to add higher grade sulfide resources.

Figures 23.4, 23.5 and 23.6 are cross sections through historic and current drillholes and relevant geology.

Figure 23.4 Cut 4 Cross Section

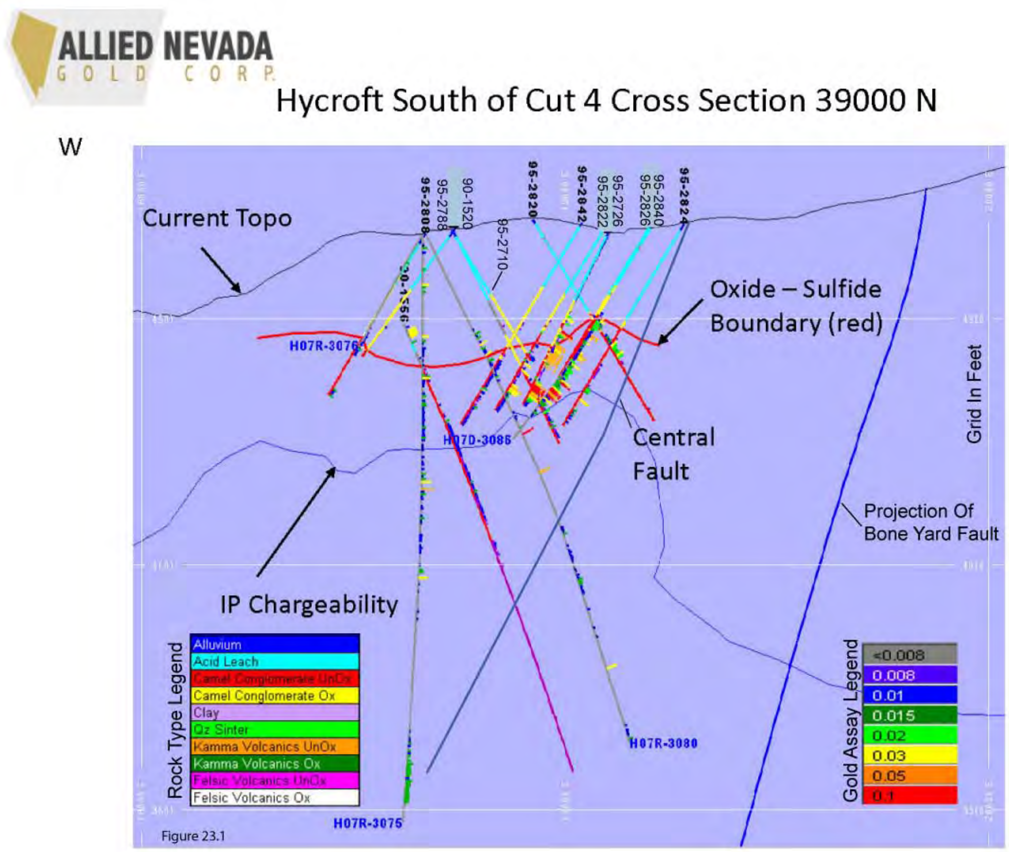


Figure 23.5 Bay Area Cross Section

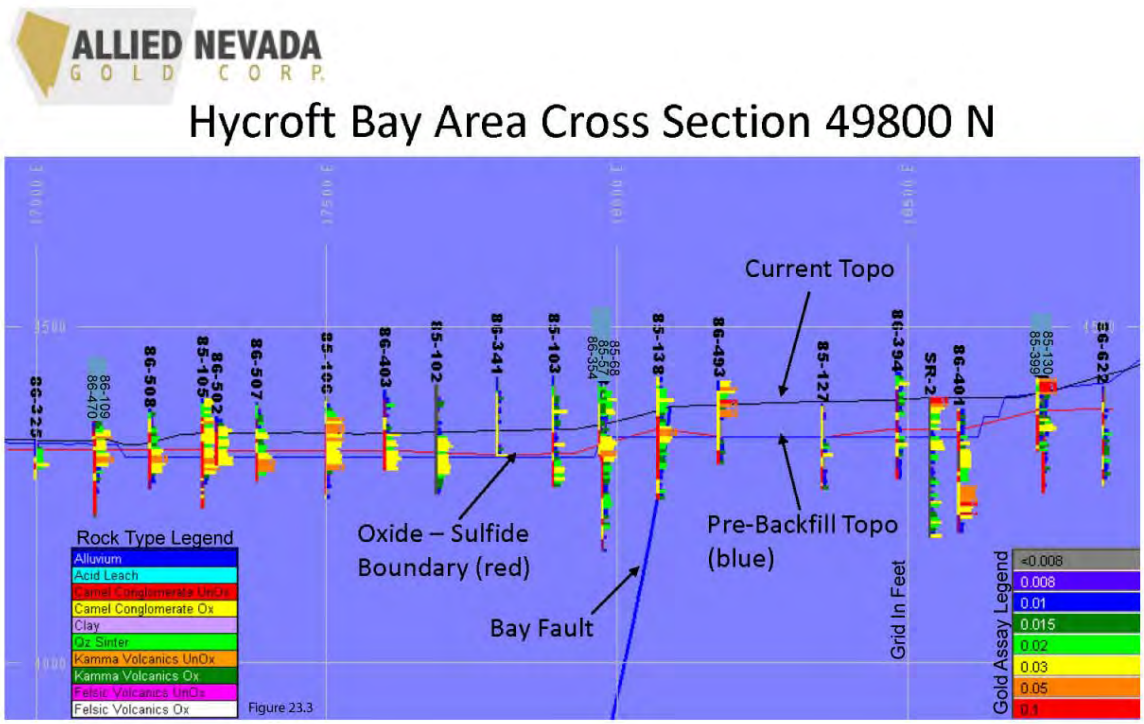
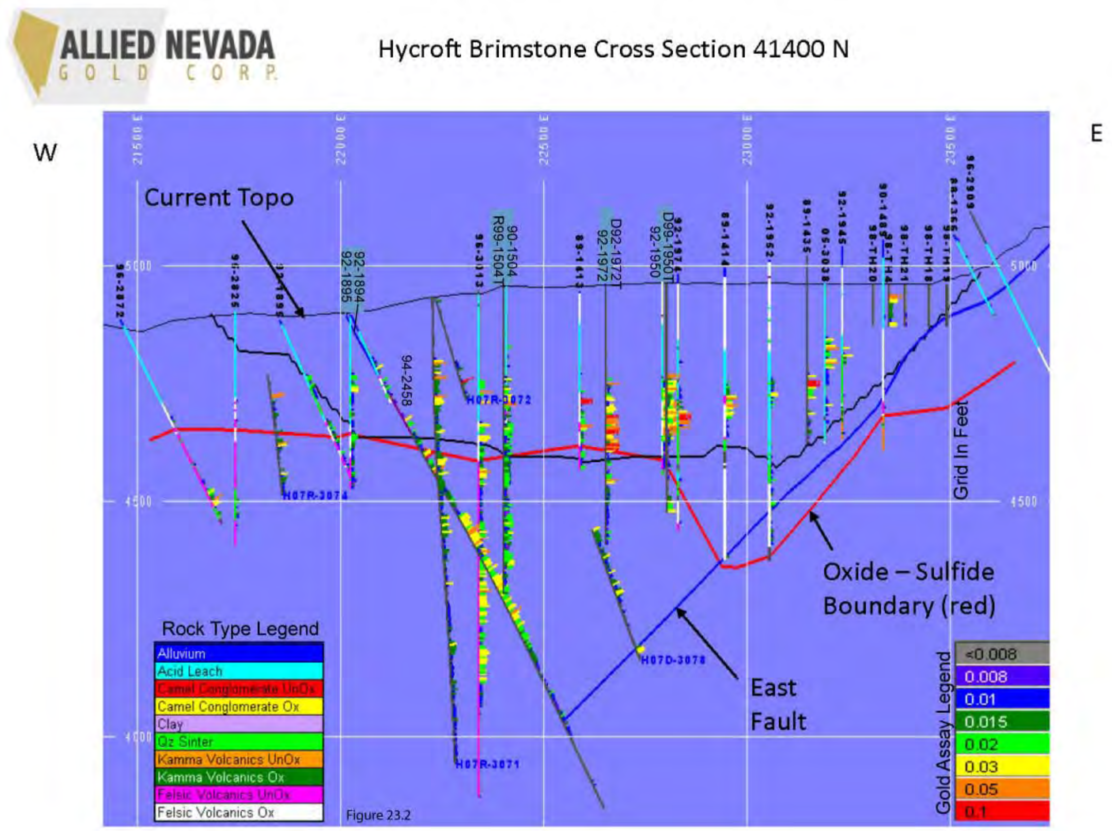
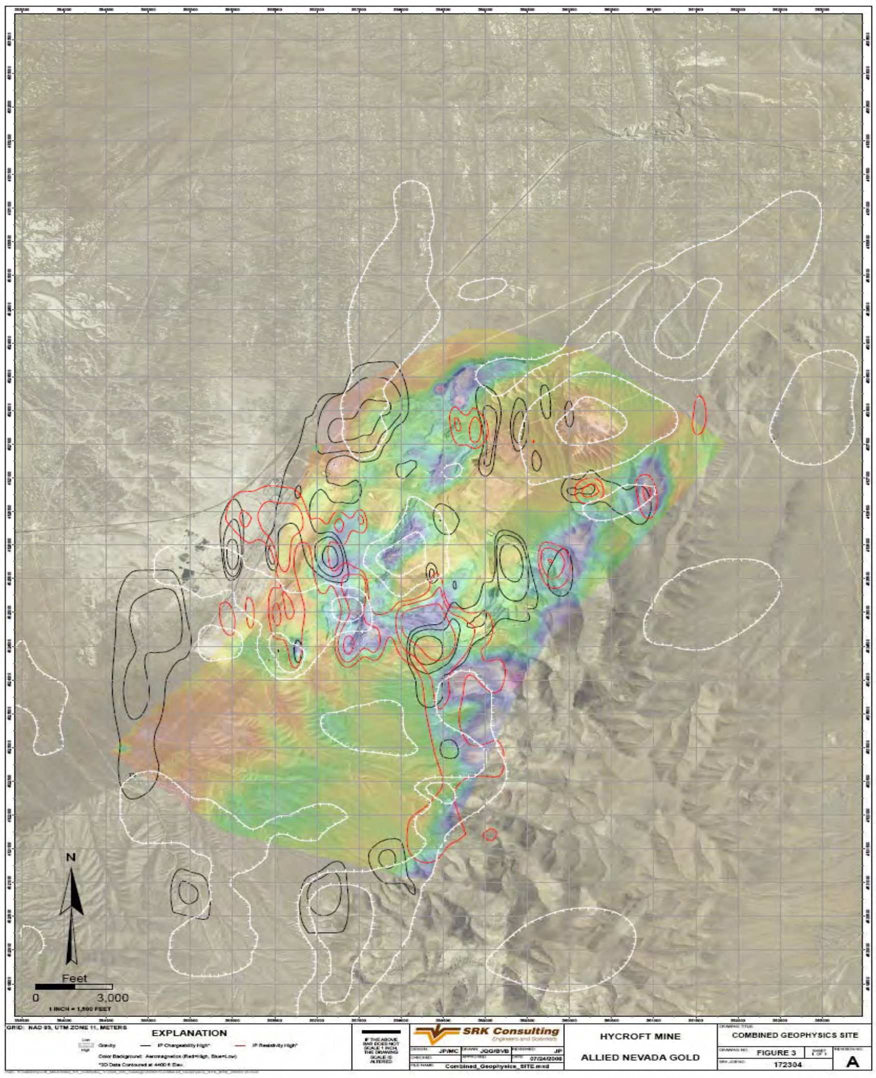


Figure 23.6 Brimstone Cross Section



Allied Nevada has retained Zonge Geoscience to complete a ground dipole-dipole induced polarization/resistivity program. Results were shown to Allied Nevada on November 10 with target recommendations. Figure 23.7 shows the geophysics lines that were interpreted at the Hycroft property.

Figure 23.7 Geophysics

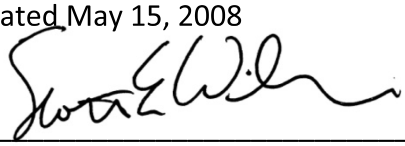


24 AUTHOR'S CERTIFICATE

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 20 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 March 3, 2009.
7. I have had prior involvement with Allied Nevada as the author of 3 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 May 15, 2009, 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 May 15, 2008



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Signature of Qualified Person

Scott E. Wilson

Printed Name of Qualified Person