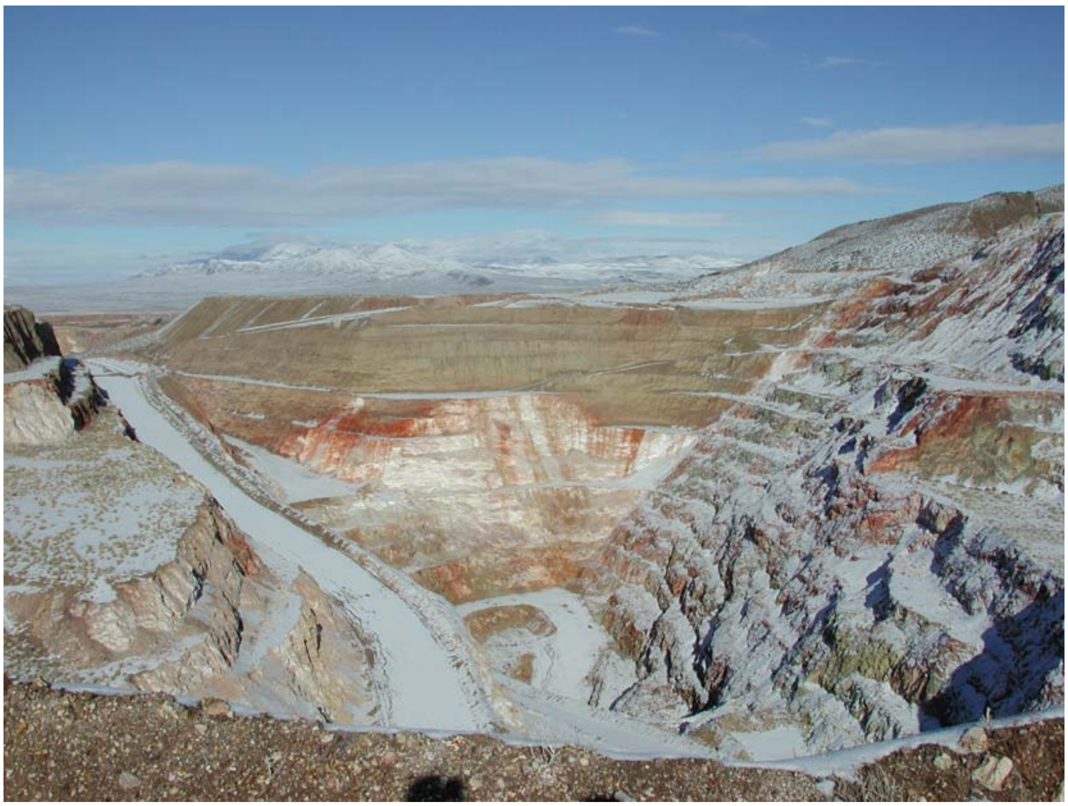
MINE DEVELOPMENT ASSOCIATES

MINE ENGINEERING SERVICES

Technical Report – Vista Gold Corp.

Hycroft Mine, Winnemucca, Nevada, USA



Prepared for:
VISTA GOLD CORP.

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1.0 SUMMARY

1.1 Introduction

Mine Development Associates (MDA) prepared this updated technical report of the Hycroft Mine at Vista Gold Corp's (Vista) request. The initial scope of MDA's work was to review the work completed by Canyon Resources Corporation (Canyon), who optioned the Hycroft property in 2005. Canyon completed 33 drill holes on the property and updated the Hycroft resource estimate during July 2005. Canyon also studied the feasibility of restarting the operation. MDA's scope also included updating the 2004 NI43-101 study of restarting of the Hycroft Mine and updating to current costs. This updated study is based on the July 2005 block model of the deposit completed by Ore Reserve Engineering (ORE) for Canyon. Canyon terminated their option on the Hycroft property during August 2005.

The Hycroft Mine is owned by subsidiaries of Vista: Hycroft Resources and Development, Inc. (HRDI) and Hycroft Lewis Mine, Inc. The Hycroft Mine is an open pit, heap leach gold-silver mine that is currently on a care and maintenance status. This document examines the feasibility of re-opening the mine and developing the reserves in the Brimstone and Albert Deposits, the easternmost of a series of gold-silver deposits on the property.

The Hycroft Mine 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. One small and two large open pit operations comprise the Hycroft Mine. 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. 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 halted in December 1998 due to low gold prices (below $300 per ounce). Leaching and recovery of gold continued for a time, then recovery of gold continued to the present through circulation of fluids to first rinse the heap and then to evaporate the fluids. Table 1.1 is a summary of production from the Hycroft Mine through December 31, 2005.

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Table 1.1 Hycroft Mine Production Summary

Deposit	Years Mined (approximate)	Tons (millions)	Grade Cn Au	Gold ounces produced
Lewis Mine	1983-1985	3.9	N/A	N/A
Bay	1988-1992			
South Central	1992-1995			
Boneyard	1992-1993			
Gap & 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 Brimstone Deposit is fully permitted and Vista possesses the necessary facilities and infrastructure to allow resumption of mining with some capital investment for pre-production stripping, mining equipment and leach pad development.

The total land holdings include 11,829 acres of patented and unpatented mineral claims, of which 7,700 acres are permitted. Of the permitted acres, 2,144 acres have been disturbed.

The Hycroft Mine is composed of two primary properties based on prior leaseholdings called the 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. There is a single 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 leasehold interest in the Lewis property was purchased by Vista on December 13, 2005. The Lewis property was purchased for the remaining purchase option total of $5.1 million and also eliminated the 5% NSR royalty on gold and 7.5% NSR royalty on silver produced from the Lewis property.

The Crofoot property was originally held under two leases and is now owned by Hycroft Resources and Development Corporation subject to a 4% net profits interest retained by the former 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. All payments for the Crofoot property are capped at $7.6 million, after which Vista will own the property. An additional $120,000 is due if ore production exceeds 5.0 million tons from the Crofoot property in any calendar year. All advance royalty payments are available as credit against the 4% net profits royalty. Crowfoot royalty payments since the amended agreement have totaled $600,000.

During 2005, Canyon optioned the Hycroft property and evaluated the purchase of the property by completing an updated property evaluation, including drilling 33 new holes on the property and updating the project resources. This study is based on the updated resources completed by Canyon.

The updated resources were completed for two areas on the property called the Brimstone and Boneyard. The resources and reserves of the Brimstone Deposit at the Hycroft Mine lie partially within the Lewis property and partially within the Crofoot property.

Except where noted, Imperial units are used throughout this report and currency is in U.S. 4th quarter dollars.

1.2 Hycroft Geology and Mineralization

The Hycroft Mine is located in the Basin and Range physiographic province, on the western flank of the Kamma Mountains and consists of Tertiary- to Recent-age, fault-controlled, low-sulfidation gold deposits that occur over an area measuring 3 miles in a north-south direction by 1.5 miles in an east-west direction. Mineralization extends to depths of less than 330 ft in the outcropping to near-outcropping portion of the Bay deposit on the northwest side and to over 990 ft in the Brimstone deposit in the eastern portion of the Hycroft property.

Four major north-northeast-trending, west-dipping, normal fault zones broadly control gold mineralization. From west to east, these fault zones are referred to as the Central, Boneyard, Albert, and East faults. The Lewis, Bay, South Central, Cut 3, and Cut 4 deposits (Central fault deposits) are hosted by the Sulfur Group in the hanging wall of the Central fault.

Gold-bearing rocks at Brimstone are located in the hanging wall of the East fault. These rocks were highly altered by four phases of alteration. Gold 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 ore.

1.3 Hycroft Drilling and Sampling

Since 1981, Vista and its predecessors have completed 958,387 ft of exploration and development drilling in 3,213 drill holes at the Hycroft Mine. This total does not include the holes completed by Canyon during 2005. These drilling efforts have resulted in the discovery and mining of 82 million tons of heap leach material from which over one million ounces of gold have been recovered. Additional reserves remain in the Brimstone deposit and the potential to add resources and reserves is considered to be good.

Prior to the drilling by Canyon, the Brimstone deposit has been tested by a total of 412 drill holes (181,828 ft). Mineralization in the neighboring Albert deposit is defined by a total of 163 drill holes (81,473 ft). Twelve holes were core drilled and the remainder drilled with reverse- circulation rigs. Sample recovery was poor with both drilling techniques because of the soft, friable nature of acid-leach and oxide ores. The assay database for Brimstone and Albert deposits contains 46,552 gold fire assays, 45,660 cyanide-soluble gold assays and 44,981 cyanide-soluble silver assays. Although Canyon completed 33 drill holes during 2005, only 14 drill holes were used to update the model of the Brimstone deposit totaling 1,015 ft of drilling.

In 1999, Vista relogged 410 drill holes in the Brimstone deposit and approximately 160 drill holes in the Albert deposit. A comprehensive system for logging lithology, structure, alteration, oxidation, the presence of sulfur and the percent sulfur was used. These data allowed for major improvements in the interpretation of the location and geometry of acid leach, oxide, sulfide and footwall units. Along with the relogging, eleven twin RC holes were drilled in 1999 to test the hypothesis that previous RC drilling had underestimated gold grades. The new holes returned higher fire and cyanide-soluble gold grades over most intervals. Based on the studies, it was concluded by Vista and Mineral Resources Development, Inc. (MRDI) that the older, wet samples were biased and that the most likely source of the bias was that fine, higher grade material was lost during wet sampled RC drilling. Based on a review of the Vista/MRDI work by Ore Reserves Engineering (ORE), they agreed with the conclusion of bias and the likely source of bias.

1.4 Hycroft Resource

The July 2005 resource estimate was completed for both the Brimstone and Boneyard deposits by ORE for Canyon. This report dated July, 2005 was filed as a technical report by Vista. The Measured and Indicated resources (acid leach and oxide) for the Brimstone deposit are shown in Table 1.2, while Inferred (acid leach and oxide) resources for the Brimstone deposit are shown in Table 1.3. The resources are reported for a cutoff grade of 0.005 cyanide soluble oz Au/ton. Below the oxide material are sulfide materials. Inferred resources are reported for sulfide materials at a 0.20 oz Au/ton cutoff grade in Table 1.4. The resources for the Boneyard deposit are shown in Table 1.5. Gold grade was estimated using inverse-distance estimation but with gold grade selection ranges and capping parameters varying according to the grade zone that was estimated. Measured resources were defined by blocks within a 100-foot drill-hole grid, which is sufficient to define the smaller mineralized and included waste zones. Indicated resources were defined by blocks with two to three drill-hole intersections within a maximum 200-foot drillhole distance from the block.

The resource estimates are based on a two pass approach to modeling the deposit. First, the mineralized zone was defined by using a nearest neighbor approach to assign a code if assays were above 0.005 or below 0.005 oz Au/ton based on cyanide soluble assays. Second, grades were assigned to blocks from bench composites within the zone based on inversed distance raised to the fourth power weighting inside the mineralized zone and inverse distance raised to the 2.5 power to the unmineralized zone. This method was repeated to estimate the fire assay grades, however, a grade of 0.0075 oz Au/ton was used to define the mineralized zone.

Table 1.2 Brimstone Measured and Indicated Resources (Acid Leach and Oxide)

	Measured			Indicated			Measured and Indicated				
Cutoff oz cnAu/t	Tons 000,000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Tons 000,000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Tons 000.000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Total oz Au/t	Contained 000's ounces
0.005	17.2	0.015	254	35.5	0.013	453	52.7	0.013	707	0.019	1,001

Table 1.3 Brimstone Inferred Resources (Acid Leach and Oxide)

	Acid Leach			Oxide			Total Inferred				
Cutoff oz cnAu/t	Tons 000,000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Tons 000,000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Tons 000.000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Total oz Au/t	Contained 000's ounces
0.005	2.5	0.010	25	6.3	0.011	67	8.7	0.011	92	0.015	131

Table 1.4 Brimstone Inferred Sulfide Resources

Cutoff oz Au/t	Tons 000,000's	Total oz Au/t	Contained 000's ounces	Cyanide Soluble oz Au/t
0.020	13.5	0.028	379	0.006

Table 1.5 Boneyard Resources

	Boneyard Indicated Resource					Boneyard Inferred Resource				
Cutoff oz cnAu/t	Tons 000,000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Total oz Au/t	Contained 000's oz Au	Tons 000,000's	Cyanide Soluble oz Au/t	Cyanide Soluble 000's ounces	Total oz Au/t	Contained 000's oz Au
0.004	0.4	0.012	5.0	0.024	10.6	0.3	0.011	3.2	0.020	6.2

1.5 Hycroft Operating Plan

MDA studied the economics of restarting an open pit operation in the Brimstone Deposit at the Hycroft Mine. It is unknown at this time whether the restart would entail using their own personnel and lease or rent mine equipment or hire a contractor for the mining portion of the operation. At a production rate of 24 million tons of ore and waste per year, the mine would be in operation for about 3.25 years after pre-stripping, with the potential to develop additional material. Leaching would continue for an additional 1.75 years. The pre-production stripping is expected to require 6 to 9 months to complete.

This document assumes that Hycroft personnel operate the mine with leased mine equipment as the base case, although some of the mine equipment will be purchased. It is planned that the mine will run two ten hour shifts per day, seven days per week. Production is expected to average 2 million tons of total mine production per month. The ore cutoff grade is 0.0047 cyanide soluble oz Au/ton. Ore will be placed on leach pad 4 without crushing (run of mine) and waste will go to one of several dump locations. The majority 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 for a period of 60 to 90 days before another 30 ft high lift of ore is placed on top of the existing one. The ore is irrigated with a buffered cyanide solution. Return solution from the pad containing the precious metals is directed to the pregnant solution pond.

The pregnant solution is then directed to the Brimstone Merrill-Crowe zinc-precipitation plant where the solution is buffered, cyanide added and then clarified using two 1,600 square foot Sparkler filters. The clarified solution is de-aerated and zinc dust metered into it. Precipitate containing the precious metals is collected using three 48 inch recessed-plate filter presses. Collected precipitate is transported to the Crofoot refinery, retorted to remove mercury, and fire refined. The barren solution is then returned to the leach pad circuit. Expected recovery of gold is 78% of the cyanide soluble gold.

1.6 Hycroft Reserves

New reserves for the Brimstone deposit were calculated based on updated economics and the new block model of the deposit. The designed pit is based on a $450 Lerchs-Grossmann optimization run. Table 1.6 summarizes the Hycroft reserves, which conform to August 20, 2000 CIM definitions.

Table 1.6 Hycroft Mineral Reserve Estimate

Category	Tons 000's	oz Au/t Cyanide Soluble	Contained oz Au Cyanide Soluble	oz Au/t Fire Assay	Contained oz Au	Waste Alluvium 000's Tons	Waste Rock 000's Tons	Total Waste 000's Tons	Total Pit 000's Tons	Strip Ratio t waste/t ore
Proven	11,954	0.016	188,600	0.022	261,000					
Probable	21,366	0.014	292,700	0.019	401,800					
Totals	**33,320**	**0.014**	**481,300**	**0.020**	**662,800**	**4,975**	**45,833**	**50,808**	**84,128**	**1.52**

The waste material inside the final pit design includes 1.525 million tons of inferred material grading 0.009 cyanide soluble oz Au/ton above a 0.0047 cyanide soluble cutoff grade. In addition, the waste totals also include a total of 1.304 million tons of sulfide material that grades 0.009 cyanide soluble oz Au/ton and 0.020 oz Au/ton by fire assay.

1.7 Restart Project Economics

The restart economics were originally studied by Vista in 2000 in an internal feasibility study. This study was updated by MDA during 2004 and again in December 2005 for this report by optimizing new pits for different gold prices, designing new preliminary and final pits, and completing feasibility level capital and operating cost estimates for the project. Table 1.7 illustrates the capital cost of the mine equipment, assuming most of the major mining equipment is leased. Table 1.8 summarizes the equipment leasing costs.

Preproduction stripping totals 8.35 million tons, including 800,000 tons of material placed on the heap.

Table 1.7 Mine Equipment Estimated Capital Cost (Purchased Equipment)

Item	Cost	Frt/Erect	Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Totals
	$000's	$000's	$000's	$000's	$000's	$000's	$000's	$000's	$000's
Loading Equipment									
Spare 27.5 cm Bucket	$425.0	$0.0	$425.0		$425.0				$425.0
Cranes for Assembly	$40.0	$0.0	$40.0	$40.0					$40.0
EX-3600 Parts On-site Inventory	$40.0	$0.0	$40.0	$40.0					$40.0
Haul Trucks									
Spare truck bed	$145.0	$5.0	$150.0		$150.0				$150.0
Support Equipment									
Light Plant	$23.0	$0.6	$23.6	$70.7					$70.7
Mechanics Truck	$100.0	$2.5	$102.5	$205.0					$205.0
Pickup Trucks	$35.0	$0.9	$35.9	$287.0		$287.0			$574.0
Welding Truck/Crane	$60.0	$1.5	$61.5	$61.5					$61.5
45 T Hydraulic Crane	$440.0	$11.0	$451.0	$451.0					$451.0
200 HP Integrated Tool Carrier	$300.0	$7.5	$307.5	$307.5					$307.5
1 CM Loader/Backhoe	$85.0	$2.1	$87.1	$87.1					$87.1
Ambulance and Fire Equipment	$150.0	$3.8	$153.8	$153.8					$153.8
Flatbed Truck	$60.0	$1.5	$61.5	$61.5					$61.5
Crew Vans	$50.0	$1.3	$51.3	$102.5		$102.5			$205.0
Forklift	$40.0	$1.0	$41.0	$82.0					$82.0
Total Equipment Capital				**$1,949.6**	**$575.0**	**$389.5**	**$0.0**	**$0.0**	**$2,914.1**

Table 1.8 Mine Equipment Leasing Cost

Item	Unit Cost $000's	Frt/Erect $000's	Unit Total $000's	Yr -1 $000's	Yr 1 $000's	Yr 2 $000's	Yr 3 $000's	Yr 4 $000's	Totals $000's
Drills									
IR DML Blasthole Drill	$880.0	$22.0	$902.0	$259.8	$844.3	$844.3	$844.3	$140.7	$2,933.3
Loading Equipment									
27.5 cy Front Shovel - Hitachi EX-3600	$4,450.5	$0.0	$4,450.5	$640.9	$1,281.7	$1,281.7	$1,281.7	$213.6	$4,699.7
24 CY Wheel Loader - Cat 994	$2,950.0	$132.0	$3,082.0	$443.8	$887.6	$887.6	$887.6	$147.9	$3,254.6
Haul Trucks									
150 t Truck- Cat 785	$1,665.0	$41.6	$1,706.6	$1,228.8	$4,392.9	$5,038.0	$5,406.6	$778.2	$16,844.4
Dozer - Cat D11R	$1,565.0	$39.1	$1,604.1	$462.0	$1,443.7	$1,443.7	$1,443.7	$240.6	$5,033.7
Rubber Tire Dozer Cat 834	$680.0	$17.0	$697.0	$100.4	$200.7	$200.7	$200.7	$33.5	$736.0
Grader 16H	$650.0	$16.3	$666.3	$95.9	$191.9	$191.9	$191.9	$48.0	$5,769.8
Water Truck - Cat 777	$1,150.0	$28.8	$1,178.8	$169.7	$339.5	$339.5	$339.5	$56.6	$3,602.3
Totals				**$3,401.3**	**$9,582.3**	**$10,227.4**	**$10,596.0**	**$1,659.1**	**$35,466.1**

In addition to the mine equipment, capital costs for the project include 8.53 million tons of pre-production stripping, additional leach pad construction, increased bonding, and initial warehouse stock. The initial capital requirements are summarized in Table 1.9.

Table 1.9 Hycroft Restart Initial Capital Requirements

Capital Expenditures	Year -1
Mine pre-stripping	$11,154.9
Mine Equipment	$1,949.6
Leach Pad	$2,000.0
Inventory	$1,150.0
Total Capital Expenditures	**$16,254.5**

The total cash cost per ounce of gold recovered is $257 per ounce, not including an equipment rental cost of about $85/ounce. Table 1.10 summarizes the economics for the project using a $450/oz gold price and a $7/oz silver price.

Table 1.10 Restart Project Pre-tax Economics – Leased Mine Equipment

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Totals	$/Ounce Au
Production Statistics								
Ore Mined, 000's Tons	800.0	8,312.0	10,639.0	11,514.0	2,055.0		33,320.0	
Waste Mined, 000's Tons	7,554.0	15,688.0	13,361.0	12,486.0	1,719.0		50,808.0	
Total Mined, 000's Tons	8,354.0	24,000.0	24,000.0	24,000.0	3,774.0		84,128.0	
Ore Grade (oz Au/ton)	0.013	0.020	0.021	0.020	0.016		0.020	
Ore Grade (cyanide soluble oz Au/ton	0.011	0.016	0.015	0.014	0.011		0.014	
Contained Ounces Au (000's)	10.5	169.8	221.4	228.7	32.4		662.8	
Contained Soluble Ounces Au (000's)	8.5	129.5	164.0	156.3	23.0		481.3	
Gold Sales (000's oz Au)		66.1	108.4	131.7	60.6	8.6	375.4	
Silver Sales (000's oz Ag)		264.2	433.8	526.9	242.4	34.4	1,501.7	
Strip ratio		1.89	1.26	1.08	0.84		1.52	
Revenue								
Gold Revenue		$29,727.9	$48,798.3	$59,273.5	$27,265.8	$3,873.9	$168,939.3	$450
Silver Revenue		$1,849.7	$3,036.3	$3,688.1	$1,696.5	$241.0	$10,511.8	$28
Gross Revenues	**$0**	**$31,577.6**	**$51,834.6**	**$62,961.7**	**$28,962.3**	**$4,115.0**	**$179,451.1**	$478
Cash Costs								
Mining (excludes cap. pre-strip)	$0.0	$19,057.6	$19,416.2	$19,975.8	$3,207.7	$0.0	$61,657.3	$164
Equipment Leasing	$0.0	$9,582.3	$10,227.4	$10,596.0	$1,659.1	$0.0	$32,064.8	$85
Processing	$0.0	$5,785.5	$6,199.3	$6,435.9	$4,392.5	$3,124.2	$25,937.3	$69
Refining, Freight	$0.0	$231.2	$379.5	$461.0	$212.1	$30.1	$1,314.0	$4
Administration	$0.0	$1,701.5	$1,761.3	$1,784.1	$1,014.2	$716.3	$6,977.4	$19
Jungo road	$0.0	$216.0	$216.0	$216.0	$108.0	$0.0	$756.0	$2
Direct Operating Costs	$0.0	$36,574.2	$38,199.7	$39,468.8	$10,593.5	$3,870.6	**$128,706.8**	$343
Royalty and Nevada Net Proceeds								
Crofoot Royalty - 4% net profit	$0.0	$120.0	$120.0	$120.0	$120.0	$0.0	$480.0	$1
Nevada Net Proceeds			$675.7	$1,168.6	$912.4	$8.6	$2,765.5	$7
Total Cash Costs	**$0.0**	**$36,694.2**	**$38,995.5**	**$40,757.4**	**$11,625.9**	**$3,879.3**	**$131,952.3**	**$351**
Capital Expenditures								
Mine pre-stripping	$11,154.9	$0.0	$0.0	$0.0	$0.0	$0.0	$11,154.9	$30
Mine Equipment	$1,949.6	$575.0	$389.5				$2,914.1	$8
Leach Pad	$2,000.0	$1,400.0	$0.0	$0.0	$0.0	$0.0	$3,400.0	$9
Inventory	$1,150.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,150.0	$3
Total Capital Expenditures	**$16,254.5**	**$1,975.0**	**$389.5**	**$0.0**	**$0.0**	**$0.0**	**$18,619.0**	$50
Other Capital								
Sale of Assets					($437.1)		($437.1)	($1)
Reclamation & Severance (Additional Bondi	$2,233.0	$0.0	$0.0	$0.0	$200.0	$50.0	$2,483.0	$7
Net Cash Flow	**($18,487.5)**	**($7,091.6)**	**$12,449.6**	**$22,204.2**	**$17,573.5**	**$185.7**	**$26,833.9**	$71
Cumulative Cashflow	**($18,487.5)**	**($25,579.1)**	**($13,129.5)**	**$9,074.7**	**$26,648.2**	**$26,833.9**		
Net Present Value and Internal Rate of Ret	Gold Price	$450						
	Silver Price	$7			$ 000's			
			NPV	0%	$26,833.9			
				3%	$21,802.3			
				5%	$18,890.3			
				10%	$12,868.3			
			IRR	29.47%				
			Payback =	31.1	months			

1.8 Economic Sensitivity

The sensitivity of the project to changes in gold price and operating cost is shown in Figure 1.1 for the pre-tax internal rate of return, and on Figure 1.2 in terms of Net Present Value (5%). Table 1.11 summarizes the Net Present Value and internal rate of return for gold prices ranging from $360 to $600 per ounce.

Figure 1.1 Project Pre-tax Internal Rate of Return Sensitivity

Figure 1.2 Project Pre-tax Net Present Value (5%) Sensitivity

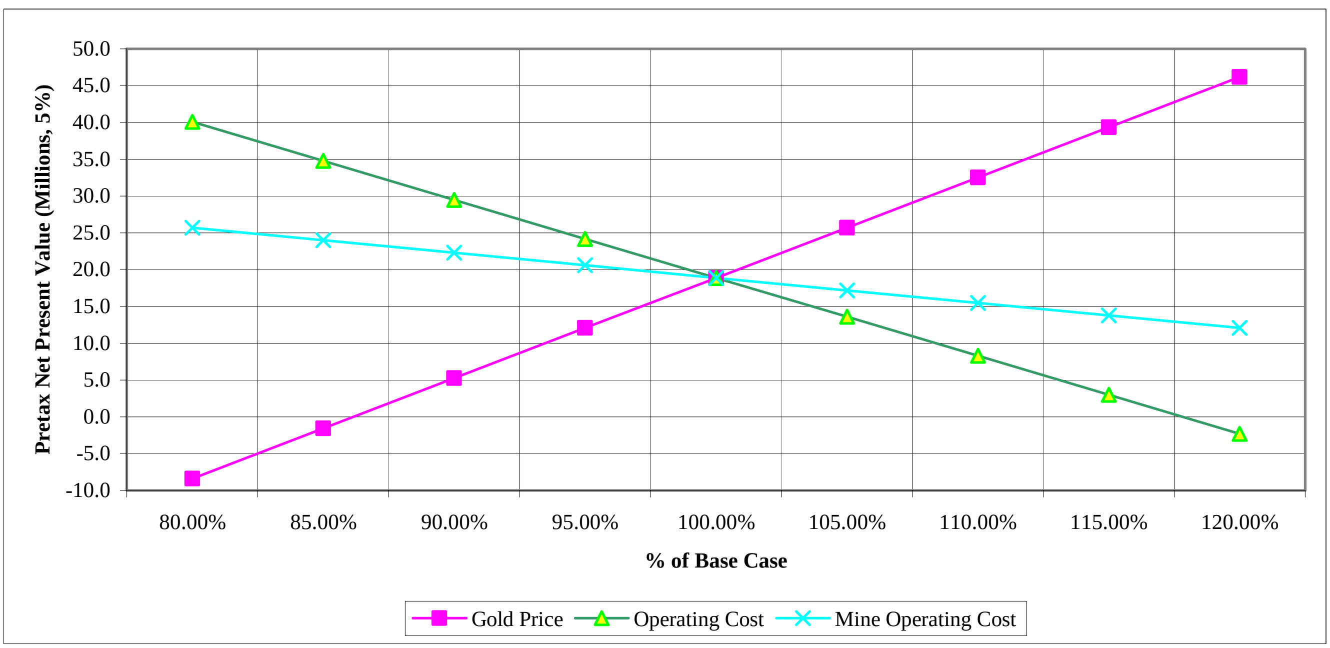


Table 1.11 Project Sensitivity to Gold Price

Gold Price $/oz Au	Silver Price $/oz Ag	NPV ($000's) 0%	NPV ($000's) 3%	NPV ($000's) 5%	IRR
$360.0	$5.0	($8,454)	($9,982)	($10,819)	-9.9%
$360.0	$7.0	($5,573)	($7,388)	($8,394)	-6.5%
$382.5	$7.0	$2,529	($90)	($1,573)	2.9%
$405.0	$7.0	$10,630	$7,207	$5,248	12.0%
$427.5	$7.0	$18,732	$14,505	$12,069	20.8%
$450.0	$7.0	$26,834	$21,802	$18,890	29.5%
$472.5	$7.0	$34,936	$29,100	$25,711	38.0%
$495.0	$7.0	$43,037	$36,397	$32,532	46.4%
$517.5	$7.0	$51,139	$43,695	$39,353	54.8%
$540.0	$7.0	$59,241	$50,992	$46,174	63.1%
$550.0	$7.0	$62,842	$54,236	$49,206	66.7%
$550.0	$9.0	$65,722	$56,830	$51,631	69.6%
$600.0	$7.0	$77,960	$67,853	$61,934	82.0%

1.9 Recommendations

Currently Hycroft's Brimstone deposit contains only enough reserves to permit mining for 3.25 years, however, the project appears to be economic based on a $450/oz gold price and current cost estimates. Restart of the project is justified at this time.

Drilling is recommended to potentially increase the proven or probable reserves and may improve the project economics. Canyon completed a portion of a Phase 1 drilling program developed by Vista during 2005. Drilling and geophysics for the Phase 1 program should be continued to improve the economics of the known mineralization and test the economics of the sulfide material. The estimated cost of the Phase 1 program is shown in Table 1.12

Table 1.12 Phase 1 Program

Program	# Holes	Footage	Drill Type	Cost
Brimstone Oxide Extension	32	18,335	RC	330,000
Brimstone Infill Drilling	29	18,065	RC	325,000
Geophysics				65,000
Total Phase 1	**61**	**36,400**		**720,000**

Vista also developed a Phase 2 program that should be updated and completed based on the results of the Phase 1 program. The estimated cost of the Phase 2 program is shown in Table 1.13.

Table 1.13 Phase 2 Program Drilling Oxide Reserve Extensions Bulk Tonnage Sulfide/High Grade

Program	# Holes	Footage	Drill Type	Cost
Brimstone Oxide Extensions (6)	75	45,000	RC	810,000
Sulfide Bulk Tonnage, High-Grade (4)	14	10,720	RC-DD	276,960
Oxide Outside Brimstone, Cut5 (3)	5	2,450	RC	44,100
Total Phase 2	**94**	**58,170**	**RC-DD**	**1,131,060**

2.0 INTRODUCTION AND TERMS OF REFERENCE

Mine Development Associates prepared this technical report of the Hycroft Mine at Vista Gold Corp's request. Vista Gold Corp. is based in Littleton, Colorado and evaluates and acquires gold projects with defined gold resources, maximizing the value of the projects through exploration and engineering studies to prepare projects for eventual development. The corporation's projects include:

- Hasbrouck Project in Nevada
- Hycroft Mine in Nevada
- Maverick Springs Project in Nevada
- Mountain View Project in Nevada
- Three Hills Project in Nevada
- Wildcat Project in Nevada
- Long Valley Project in California
- Yellow Pine Project in Idaho
- Paradones Amarillos Project in Mexico
- Guadalupe de los Reyes Project in Mexico
- Amayapampa Project in Bolivia
- Awak Mas Project in Indonesia
- Lewis Properties in Nevada

Hycroft Mine is owned by subsidiaries of Vista: Hycroft Resources & Development, Inc. and Hycroft Lewis Mine, Inc. Both subsidiaries are incorporated under the laws of the state of Nevada. The Hycroft Mine is an open pit, heap leach gold-silver mine that elected to curtail mining operations beginning in 1997. All mining of new ore was suspended by the end of 1998. Since that time, leaching has continued to recover gold and silver contained in the heaps. This document examines the feasibility of re-opening the mine and developing the reserves in the Brimstone and Albert Deposits, the easternmost of a series of gold-silver deposits on the property.

This report's purpose is to comply with disclosure and reporting requirements set forth in the Canadian Venture Exchange ("CDNX") Corporate Finance Manual, National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1. The remaining reserves and resources for Hycroft Mine cited within this document are as of December 31, 2003.

The scope of this study included a review of pertinent technical reports and data in possession of Vista relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, resources and reserves. Each scope element was addressed in the context of the company's target concepts, recent results, and proposed activities.

The author's mandate was to prepare resource and reserve estimates and a mine plan using Vista's block model and to comment on substantive public or private documents and technical information listed in the Section 23 (References). The Vista block model was prepared by ORE during July 2005. The mandate also required an on-site inspection and preparation of an independent qualifying report containing the author's observations, conclusions and recommendations. A total of 25 man days were required to complete the mandate, including a site inspection that was conducted January 7, 2004 at Hycroft Mine.

Currency, units of measure, and conversion factors used in this report include:

Linear Measure

1 inch	= 2.54 centimeters
1 foot	= 0.3048 meter
1 yard	= 0.9144 meter
1 mile	= 1.6 kilometers

Area Measure

1 acre		= 0.4047 hectare
1 square mile	= 640 acres	= 259 hectares

Capacity Measure (liquid)

1 US gallon	= 4 quarts	= 3.785 liter

Weight

1 short ton	= 2000 pounds	= 0.907 tonne
1 pound = 16 oz	= 0.454 kg	= 14.5833 troy ounces

Analytical Values	Percent	grams per metric ton	troy ounces per
1%	1%	10,000	291.667
1 gm/tonne	0.0001%	1	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
10 ppb			0.00029
100 ppm			2.917

Currency Unless otherwise indicated, all currency ($) in this report is expressed in United States dollars.

$1 US = $1.333 CAD (rate March 15, 2004)

$1 CAD= $0.750 US (rate March 15, 2004)

Frequently used acronyms and abbreviations

AA	atomic absorption spectrometry
Ag	silver
Au	gold
AuEq	gold equivalent
BLM	U.S. Bureau of Land Management
C	Celsius
CIM	Canadian Institute of Mining, Metallurgical, and Petroleum Engineers
cy	cubic yard
EIS	Environmental Impact Statement
EPA	U.S. Environmental Protection Agency
F	Fahrenheit
FCCPM	Fracture-controlled chalcedony-pyrite-marcasite mineralization
Fe	iron
ft	foot or feet
gpm	gallons per minute
hp	horsepower
HRDI	Hycroft Resources and Development, Inc.
ISO	International Standards Organization
Ma	million years before present
MDA	Mine Development Associates
MRDI	Mineral Resources Development, Inc.
NDEP	Nevada Department of Environmental Protection
NPI	net profit interest
NSR	net smelter return
ORE	Ore Reserves Engineering
oz Ag/ton	ounces silver per short ton
oz Au/ton	ounces gold per short ton
ROM	run of mine (leaching of uncrushed materials)
RQD	rock quality designation
RC or RVC	reverse circulation drilling method
ton	short ton
tpd	short tons per day
Vista	Vista Gold Corp.

3.0 DISCLAIMER

MDA has relied almost entirely on data and information derived from work done by Vista or Canyon for the Hycroft Property. In particular, the most recent resource model was developed by ORE who completed the work for Canyon during 2005.

The author, Neil Prenn has visited the property, collected enough samples to verify that mineralization of the character described exists, and verified that the geology as seen in the field is consistent with the geology described herein. Nevertheless, the authors have made extensive use of information contained in geological reports prepared by other geoscientists, as listed in Section 21. Sources of information are acknowledged throughout the text, where the information is used. None of the reports cited contain authors' certificates. MDA has not determined, nor is it practical for MDA to determine, who if anyone amongst the authors of the reports cited may have been a Qualified Person as defined in NI 43-101.

Vista provided MDA with copies of documentation regarding the status of the mineral rights that comprise the Hycroft Property. While the present authors are generally knowledgeable concerning mineral rights in Nevada, they are not "Qualified Persons" for assessing the validity of the mining claims, the contractual rights of Vista, and other legal matters relating to the mineral rights. MDA believes that the mineral rights held by Vista at Hycroft are as stated in this report, but this is not a professional opinion. Readers requiring assurance on such legal matters should consult qualified experts.

The present authors are not Qualified Persons with respect to environmental science. Discussions of environmental matters in this report are not professional opinions. Readers requiring assurance on environmental matters should consult qualified experts.

4.0 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 small portion of the property in adjacent Pershing County. The mine straddles Townships 34 and 35 North and Ranges 29 and 30 East with an approximate latitude 40^{o} 52’ north and longitude 118^{o} 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, Nevada. 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.

4.2 Land Area

The acreage owned or leased by Hycroft is approximately 11,829 acres. A total of 588 unpatented claims are maintained at the property. There are twenty owned or leased patented claims on the property. Hycroft is the owner of approximately 3,636 acres acquired when Vista exercised its options to convert its leasehold interest in the Crofoot property into 100% ownership interest in the patented mining claims, a 100% possessory interest in the unpatented claims, and a 100% interest in the incidental rights thereto, subject to a 4% net profits interest retained by the former owners. Of the Hycroft controlled acreage, approximately 10,017 acres are on public lands and 1,812 acres are on private lands.

4.3 Mining Claim Description

Figure 4.2 illustrates the location of the mining claims, fee lands, open pit, plant, and related facilities. There is a single claim of approximately 20 acres contained within the greater area that is not controlled by Hycroft. However, this claim should not affect the planned mining operation. A list of the patented mining claims is shown in Table 4.1, while a list of unpatented claims can be found in Appendix A.

Figure 4.1 Location Map of the Hycroft Mine, Humboldt County, Nevada

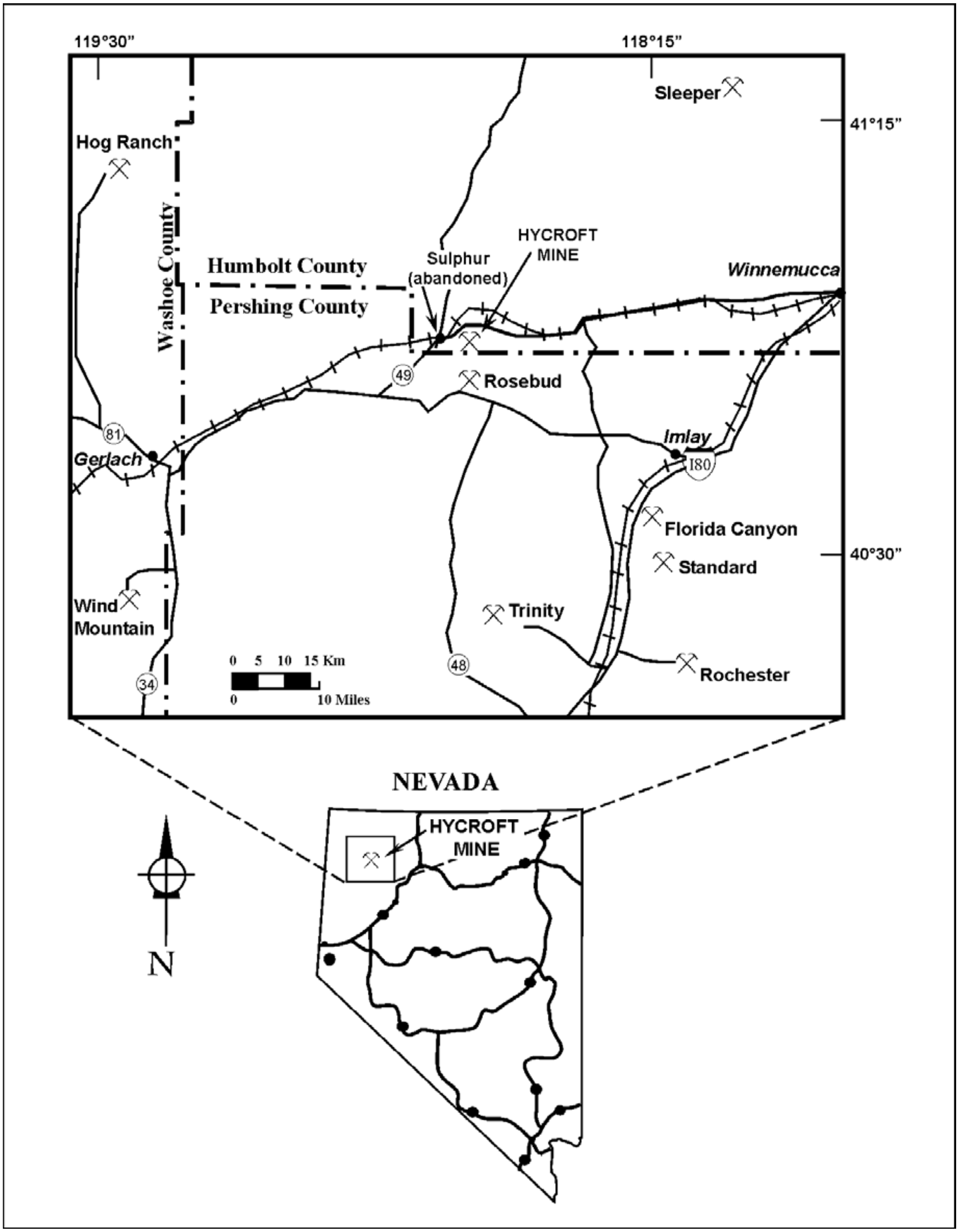
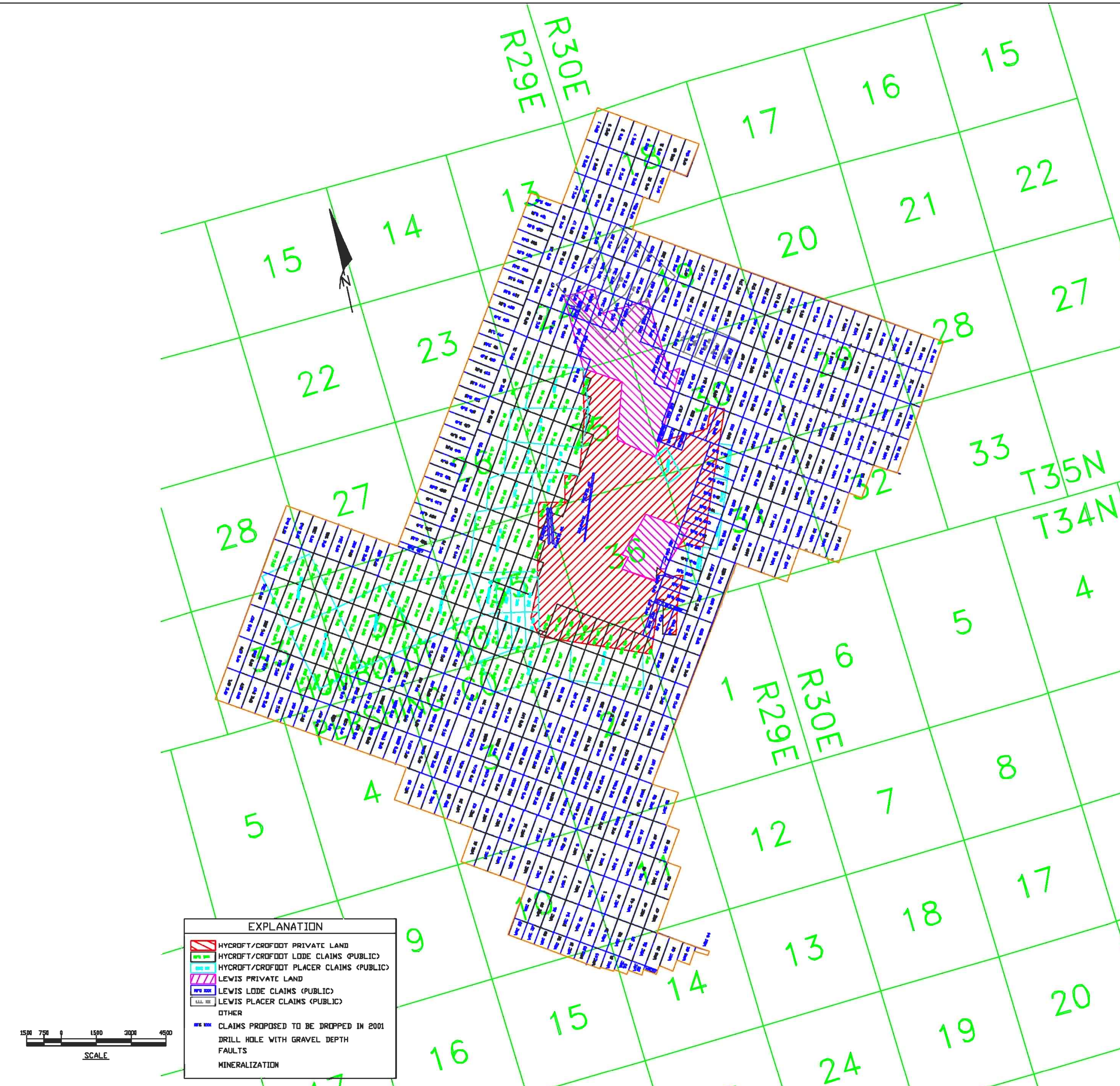


Table 4.1 Hycroft Patented Claims

Hycroft / Crofoot Patented Claims Humboldt and Pershing Counties	
	Sheol Sulfur Mine No. 1
	Sheol Sulfur Mine No. 2
	Sheol Sulfur Mine No. 3
	Sheol Sulfur Mine No. 4
	Sheol Sulfur Mine No. 5
	Sheol Sulfur Mine No. 6
	Sheol Sulfur Mine No. 7
	Sheol Sulfur Mine No. 8
	Swager Place
	Green Rock # 1
	Green Rock # 2
	Green Rock # 3
	Green Rock # 4
	Admission Placer
Hycroft / Synder Patented Claims Humboldt County	
	West Virginia
	West Virginia No. 1
	Blackrock
Hycroft / Lewis Patented Claims Humboldt County	
	Hilltop Placer
	Occult Placer
	Sheol Sulfur Mine No. 9



Vista Gold Corp
Hyrcroft Property
Claim Map

NV

AS A MUTUAL PROTECTION TO OUR CLIENTS, THE PUBLIC, AND MDA ALL REPORTS AND DRAWINGS ARE SUBMITTED FOR THE CONFIDENTIAL INFORMATION OF OUR CLIENT FOR A SPECIFIC PROJECT AND AUTHORIZATION FOR USE OR PUBLICATION OF DATA, STATEMENTS, CONCLUSIONS OR ABSTRACTS FROM OR REGARDING OUR REPORTS AND DRAWINGS IS RESERVED PENDING OUR WRITTEN APPROVAL.

MINE DEVELOPMENT ASSOCIATES
Reno Nevada

DATE	08 Mar 04
DRAWN BY	Vista Gold
CHECKED BY	MDA
SCALE	not to scale

FIGURE NO.
4.2

CAD CONTROL BLOCK REF: neil\proj\vista-hycroft\43-101 2000\Claim Map
DATE: 4/8/04

4.4 Agreements and Encumbrances

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. Besides the Lewis portion of the Hycroft mine, F. W. Lewis, Inc. owned 52 other properties that are retained by Vista. F. W. Lewis, Inc. also had a 5% NSR royalty on gold and a 7.5% NSR royalty on silver produced from the Lewis property. There is no longer any royalty on gold and silver produced from the Lewis property.

The Crofoot property was originally held under two leases and is now owned by Hycroft Resources and Development Corporation subject to a 4% net profits interest retained by the former 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. An additional $120,000 is due if ore production exceeds 5.0 million tons from the Crofoot property in any calendar year. All advance royalty payments are available as credit against the 4% net profits royalty. The Crofoot royalty is capped at $2.8 million of which $0.6 million has been paid to date.

Table 4.2 Hycroft Land Holding 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 Liquified Petroleum Gas	Annual Fee	$135

4.5 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 and HRDI acquired the Lewis Mine and the Crofoot claims and started mining in 1988. All the earlier mining areas were incorporated into the Hycroft Mine and as such, are included with the current reclamation plans and bonding.

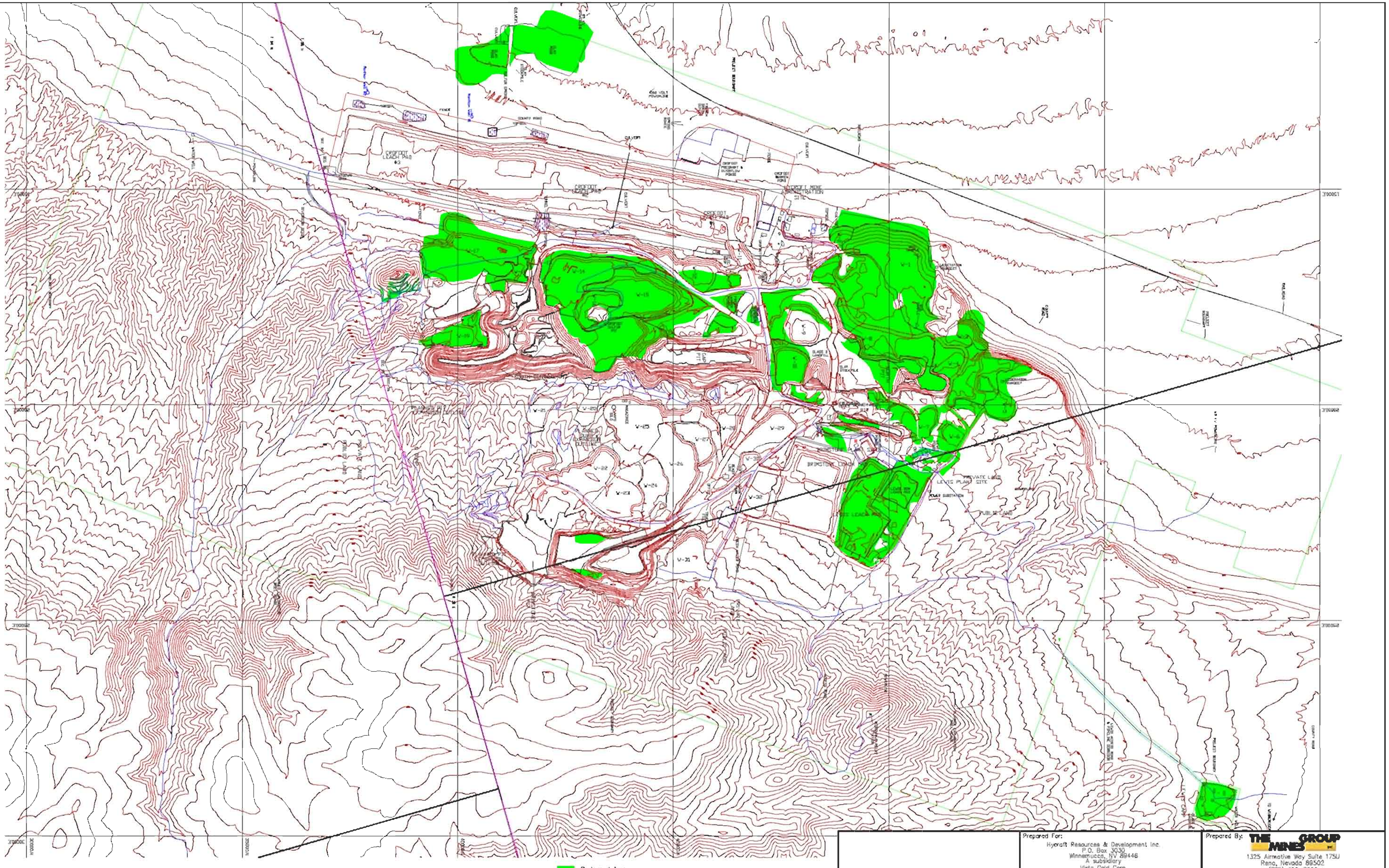
In January 2004 Vista announced that HRDI had reached agreement with member companies of American International Group, Inc. (AIG) to replace the existing bond at its Hycroft Mine with a new package that includes an insurance component and covers all existing reclamation liability at Hycroft. The bond package also allows for future increases when Hycroft moves back into production. Figure 4.3 shows the currently reclaimed areas, while Figure 4.4 shows the disturbed areas.

The new bond calls 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 new bonding instrument was accepted by the BLM, and the new insurance/assurance bonding instrument replaced the existing bond made up of a $5.1 million non-cash collaterized bond from American Home Assurance Company, letters of credit of $1.7 million posted directly with the BLM and the existing indemnity agreement between HRDI and Vista.

The Mines Group, Inc. of Reno, Nevada revised and updated reclamation plans for the Hycroft Mine in 2003 and estimates the cost of reclamation to total $6,767,000. Table 4.3 shows a breakdown of the estimated costs.

Table 4.3 Hycroft Estimated Reclamation Costs

Item	**Estimate $000's**
Earthwork/Recontouring	$ 3,146
Revegetation/Stabilization	$ 580
Detoxification/Disposal of Wastes	$ 1,277
Administration	$ 1,664
Final Closure Design & Plan	$ 100
Total	**$ 6,767**



References	DRAWING NUMBER	DESCRIPTION	Revisions	REV	DATE	DESCRIPTION	BY	CHKD	APPVD

Designed		
Drawn	Robert Breen	08/27/02
Checked		
Approved		
Approved		
Approved		

Engineer

Prepared For:
Hycroft Resources & Development Inc.
P.O. Box 3030
Winnemucca, NV 89446
A subsidiary
Vista Gold Corp.

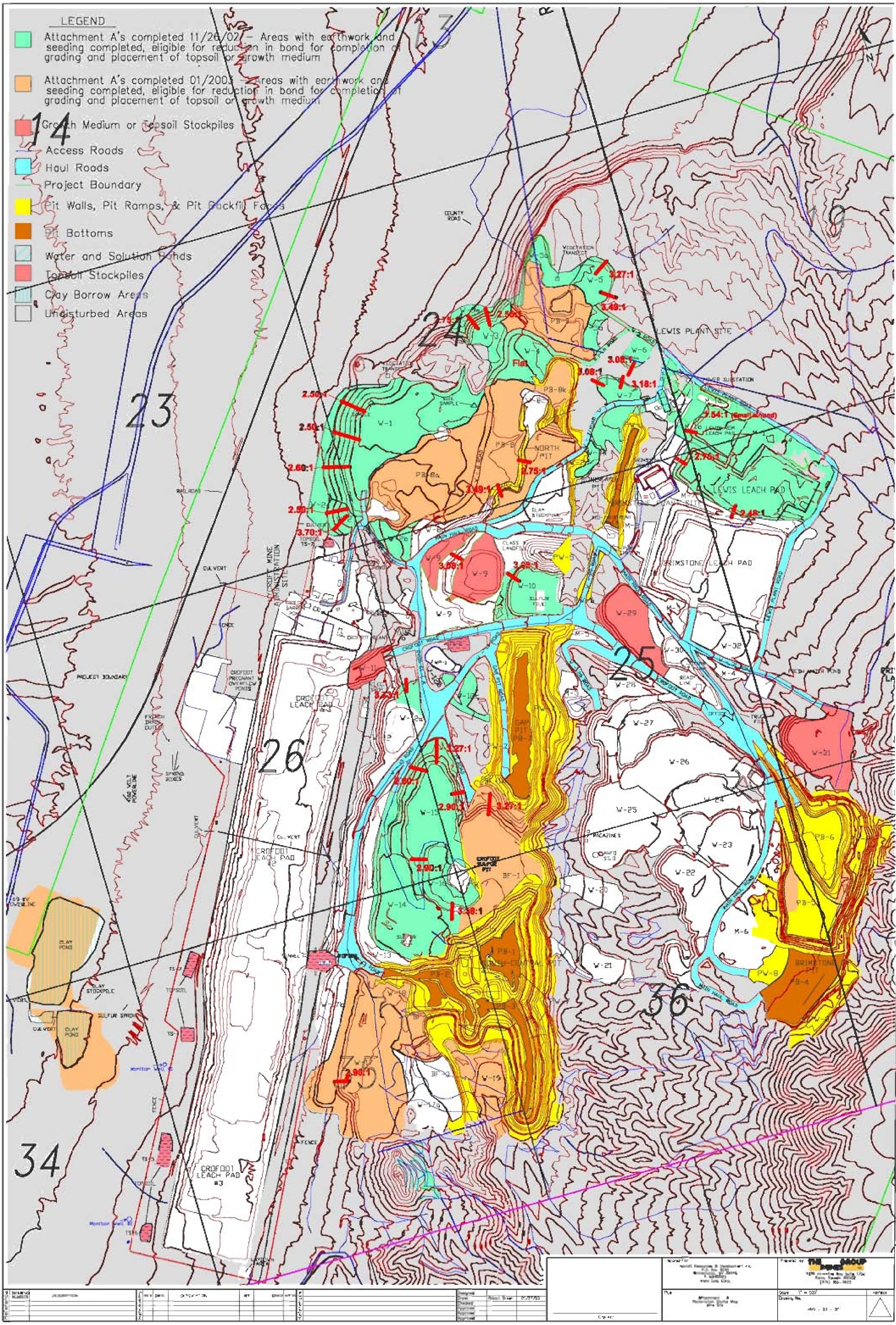
Prepared By: THE MINES GROUP INC
1325 Airmotive Way Suite 175U
Reno, Nevada 89502
(775) 322-7622

Title
Reclamation Cross Sections
Reclamation Plan Review and Update
August 2002
Reclaimed Area Map

Scale 1" = 1000'

Drawing No
02-1002-05

Revision


LEGEND
Attachment A's completed 11/26/02 - Areas with earthwork and seeding completed, eligible for reduction in bond for completion of grading and placement of topsoil or growth medium
Attachment A's completed 01/2003 - Areas with earthwork and seeding completed, eligible for reduction in bond for completion of grading and placement of topsoil or growth medium
Growth Medium or Topsoil Stockpiles
Access Roads
Haul Roads
Project Boundary
Pit Walls, Pit Ramps, & Pit Backfill Faces
Pit Bottoms
Water and Solution Ponds
Topsoil Stockpiles
Clay Borrow Areas
Undisturbed Areas
LEWIS PLANT SITE
LEWIS LEACH PAD
BRIMSTONE LEACH PAD
CROFOOT LEACH PAD
CROFOOT LEACH PAD #2
CROFOOT LEACH PAD #3
NORTH PIT
GAP PIT
CROFOOT MINE ADMINISTRATION SITE
MAGAZINES
CLAY POND
14
13
23
24
19
25
26
36
35
34

4.6 Permits

Hycroft Mine operates under permit authorizations from the BLM, Nevada Division of Environmental Protection, and the Nevada Bureau of Mining Regulation & Reclamation. Additional state and federal permits are required for air and water quality, exploration and other specific items.

Table 4.4 summarizes the operating permits, while Table 4.5 shows the miscellaneous permits for the property. Figure 4.4 illustrates the reclaimed site.

Table 4.4 Hycroft Operating Permits

Operating Permits	Issuing Agency	Number	Issued	Expires
Plan of Operations & Reclamation Plan	BLM	#N26-87-002P	6/10/2003	6/10/2006
Reclamation Surety Bond	Am Home Assure Co.	N-64641	5/22/1998	Life of Project
Manufacture of High Explosives	Bureau of Alcohol, Tobacco & Firearms	#9-NV-013-20-5C-12087	11/1/2002	3/1/2005
Class II Air Quality Permit	NV Division of Environmental Protection Bureau of Air Quality	#AP1041-0661.01	5/3/2001	10/31/2005
Water Pollution Control - Crofoot Operation	NV Bureau of Mining Regulation & Reclamation	NEV60013	Pending	Renewal pending since 4/30/2001
Water Pollution Control - Brimstone Operation	NV Bureau of Mining Regulation & Reclamation	NEV94114	6/6/2001	5/1/2006
Water Pollution Control - Closure of Lewis Facility	NV Bureau of Mining Regulation & Reclamation	NEV89017	1/24/2000	1/24/2005
Bioremediation Facility Permit	NV Bureau of Mining Regulation & Reclamation	#GNV041995	2/19/1995	Life of Project
Reclamation Permit	NV Bureau of Mining Regulation & Reclamation	#0134	5/22/98	Life of Project
Stormwater Pollution Prevention Permit	NV Bureau of Water Pollution Control	#NV0050006-10037	9/13/2000	9/13/2005
Artificial Pond Permit (Brimstone Mine)	NV Dept of Wildlife	S21090	2/1/2002	1/31/2007
Artificial Pond Permit (Crofoot Mine)	NV Dept of Wildlife	S23123	6/1/2003	5/31/2008
Crofoot Process Ponds	NV Division of Water Resources	#J-273	12/15/1987	Life of Project
Crofoot Process Well #1	NV Division of Water Resources	#60230	11/4/2003	8/6/2004
Crofoot Process Well #2	NV Division of Water Resources	#60231	11/4/2003	8/6/2004
Crofoot Potable Well	NV Division of Water Resources	#49533		Must be renewed
Hazardous Materials Storage Permit	NV State Fire Marshall	#1182-2354	3/1/2004	2/28/2005

Table 4.5 Hycroft Miscellaneous Permits

Operating Permits	Issuing Agency	Number	Issued	Expires
R/W Communication Site on Floka Peak	BLM	N46292	1/1/2004	Annual
R/W Potable Water Well/Pipeline/Power Line	BLM	N-46564	1/12002	1/1/2007
R/W Process Wells/Pipeline/Power Line	BLM	N-46959	1/1/2002	1/1/2007
R/W Road & Waterline (Old Mancamp to Lewis)	BLM	N-39119	1/1/2000	1/1/2005
R/W Mabel Well Pipe Line to Mancamp	BLM	N-44999	Dropped	1/1/2004
Kamma Peak Station	FCC	WNER344	4/28/2002	5/14/2012
Sulfur Mine Station	FCC	WNER345	4/28/2002	5/15/2012
Winnemucca Mtn. Station	FCC	WNER346	4/28/2002	5/16/2012
Base Station & 45 Mobil Units	FCC	WNKK336	11/5/2002	12/1/2012
Class 3 Landfill Permit	NV Bureau of Waste Management	#SWM1-08-11	7/16/1993	Life of Project
Potable Water Permit	NV Division of Water Resources	#HU-0864-12NCNT	7/17/2003	7/31/2004
Propane	NV Board for the Regulation of LPG	#03615	10/1/2003	10/1/2004
Regional General Permit	U.S. Army Corps of Engineers	Section 404 Permit	N/A	N/A

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

5.1 Access

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. In the past, the majority of employees lived in the Winnemucca area but some also lived in Gerlach. Winnemucca (population 15,000) is an important commercial community on Interstate 80, 164 miles northeast of Reno, Nevada. The town is served by a transcontinental railroad and has a small airport.

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^{o}'s F at night and near 90^{o} F and above during the days. Winter temperatures are usually in the 20^{o}'s F at night and in the 40^{o}'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. Even after the hottest days, the nights are usually cool. The average range between the highest and lowest daily temperatures is about 30 ^{o} to 35 ^{o} F. Daily ranges are larger in summer than the winter.

Winds are generally light. Dust or sand storms occur occasionally, particularly during the spring.

The mine did not generally have any downtime that was weather related.

5.3 Local Resources and Infrastructure

Mining began in the area in 1983 in the Lewis pit and continued until 1998 with a two year gap in 1976 and 1977. In 1998, mining was curtailed due to low gold prices, however, leaching and gold recovery continued for a time on the existing heaps. October 2000 was the last introduction of cyanide to the leach circuits. At that time, Hycroft modified its recovery process to employ carbon absorption in place of the Merrill-Crowe process. The loaded carbon is transported to an external commercial facility for stripping, thus eliminating the need to add even a small volume of cyanide required for carbon stripping.

The leach pads have been progressively flushed with barren solution, thus reducing the volume of solution circulating in the heaps through evaporation by about 90%. Over the past several years, Hycroft has introduced fresh water into the larger Crofoot leach pad system and, at current levels, the Crofoot pad does not require further rinsing to meet Nevada closure standards for cyanide and pH. It is now ready for re-grading and reclamation.

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 any future operations. The pumps and/or power drives have been removed from the other wells.

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	

Power to the site is supplied by Sierra Pacific Power Company.

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. There are no streams, rivers, or major lakes in the general area.

Elevations in the mine area range between 4,500 and 5,500 ft above sea level.

6.0 HISTORY

Much of the following information in the history sections is taken from Vista in-house documents with other historical sources referenced. Details for the references are found in the reference section of this report.

6.1 Property History

The current proper term for the Hycroft Mine is the Crofoot-Lewis Mine. The name derives from two underlying claim owners. The Crofoot mine is held in HRDI's name and refers to grounds subject to the Crofoot Royalty Agreement. The Lewis property is held by Vista, who purchased the Lewis Royalty in December 2005. HRDI operates mining and exploration on those grounds.

The Crofoot- Lewis Mine has been referred to in the past as the Sulfur district, Hycroft mine, and Silver Camel. Limited mining for sulfur, alunite and silver has been carried out as early as the late 1800s.

The following five paragraphs are taken from Ebert, 1996.

The earliest recorded mining in the Sulfur district began in the late 1800's following the discovery of significant native sulfur deposits (Couch and Carpenter 1943, Willden 1964). Mining of native sulfur was sporadic during the 1900's, 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 mining 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 2500 tons has been mined at an unknown grade between 1932 and 1937.

During World War 1, three 1.8-2.4 meter wide veins of nearly pure alunite were mined in the southern part of the Sulfur district (Clark 1918). In 1931 several hundred tons of alunite was mined as a soil additive (Fulton and Smith, 1932). Vandenburg (1938) estimated that 454 tonnes 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 862 kg (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 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.

6.2 Exploration and Development History

Between 1985 and 1999, HRDI drilled a total of 3,123 exploration drill holes, totaling 943,822 ft. The current Hycroft drill hole database consists of the former holes, plus 61 RC holes drilled by Homestake in 1982 and 29 rotary holes completed by Homestake in 1981. 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.

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.

Table 6.1 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,Cut4,Gap,North,SC
1988	RC	Hycroft	73	25,855	Alb,Brim,Cut4,North,SC
1989	RC	Hycroft	43	15,780	Alb,Brim,Cut4,North,SC
1990	DD	Hycroft	8	11,247	Cut 4,Sulfur
1990	RC	Hycroft	134	52,675	Alb,Brim,Cut4,North,SC
1991	RC	Hycroft	147	44,360	Cut 4, North,SC
1992	RC	Hycroft	265	83,030	Alb,Brim,Cut4,North,SC
1993	DD	Hycroft	6	2,318	Alb,Brim,SC
1993	RC	Hycroft	297	105,500	Alb,Brim,Cut4,North,SC
1994	DD	Hycroft	***3***	***4,990***	Brim
1994	RC	Hycroft	208	78,650	Alb,Brim,Cut4,Boneyard,SC
1995	RC	Hycroft	355	157,515	Alb,Brim,Cut4,Gap,Boneyard,SC
1996	DD	Hycroft	1	1,078	Brim
1996	RC	Hycroft	164	75,000	Alb,Brim,Cut4,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
Total			**3213**	**959,387**	

Note: Drill programs in bold italics are twin studies for metallurgic purposes or sampling and assaying verification.

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 deposits; South Central, Gap, Cut 4

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 Syne 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.

6.2.3 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 Brimstone Deposit

The Brimstone deposit is hosted in rhyolitic, aphanitic and tuffaceous Kamma volcanics in the southeastern part of the Crofoot Lewis mine area. The deposit is a zone 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 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 were mined from the Brimstone deposit with an average cyanide soluble grade of 0.0143 oz Au/ton. The remaining reserves at Hycroft are contained in the southern portions of Brimstone.

6.2.5 Albert Deposit

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

Deeper 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.

6.2.6 Canyon Resources Drilling 2005

Canyon completed 33 drill holes totaling 13,315 ft of drilling. All of the holes were completed by reverse circulation methods. A center return bit was used for most of the drilling. Two holes were drilled from the bottom of the Brimstione pit, five holes were drilled north of the Brimstone pit to test for an extension, and 26 holes were drilled within and adjacent to the remaining Brimstone deposit.

6.3 Production History

Information on the production history of the Hycroft Mine comes from Vista 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.

Table 6.2 Hycroft Mine Production Summary

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 Hycroft Pad Loading and Production by Year

	Hycroft Pad Loading Tons (000's)					Ore	Waste	CN Au	Total Oz. Loaded (000's)						000's Oz. Au
Year	Pad 1	Pad 2	Pad 3	Pad 4	Pad 5	Tons (000's)	Tons (000's)	oz Au/ton	Pad 1	Pad 2	Pad 3	Pad 4	Pad 5	Totals	Recovered
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 all 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.

6.4 Historical Resource and Reserve Estimates

The prior resource estimate was completed by MRDI as part of their work for Vista in 2000. MRDI then used the model to re-estimate gold resources using the MRDI adjusted gold and silver database and the new geological interpretations of ore types.

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 “resources” are tabulated from the 2004 restart feasibility study completed by MDA based on the MRDI model. 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 6.4 Brimstone Historic "Measured and Indicated Resources" (0.005 opt MIK CNsol Au Cutoff Grade)

Category	Tons	Cyanide Soluble oz Au/ton	Cyanide Soluble 000's oz Au	Fire Assay oz Au/ton	Contained 000's oz Au
Measured	23,287,000	0.0132	307.6	0.0165	385.1
Indicated	24,192,000	0.0127	307.3	0.0153	369.4
Totals	**47,479,000**	**0.0130**	**614.9**	**0.0159**	**754.6**

Table 6.5 Brimstone Historic "Inferred Resources" (0.005 opt MIK CNsol Au Cutoff Grade)

Category	Tons	Cyanide Soluble oz Au/ton	Cyanide Soluble 000's oz Au	Fire Assay oz Au/ton	Contained 000's oz Au
Inside Historic Designed Pit	5,210,000	0.0126	65.4	0.0154	80.4
Outside Designed Pit	6,819,000	0.0080	54.7	0.0078	53.2
Totals	**12,029,000**	**0.0100**	**120.1**	**0.0111**	**133.6**

MDA summarized the historic mineral "reserve" estimate from the 2004 restart feasibility, which is given in Table 6.6.

Table 6.6 Hycroft Mineral Reserve Estimate

Category	Tons	Cyanide Soluble oz Au/ton	Cyanide Soluble oz Au	Fire Assay oz Au/ton	Contained oz Au	Waste Tons 000's	Total Tons 000's	Strip Ratio
Proven	16,269.0	0.0144	234.0	0.0180	293.0			
Probable	16,160.0	0.0139	224.2	0.0169	273.5			
Totals	**32,429.0**	**0.0141**	**458.2**	**0.0175**	**566.5**	**57,796**	**90,225**	**1.78**

Both the historic resource and reserve estimates are 43-101 compliant.

7.0 GEOLOGICAL SETTING

Much of the information contained in the geology sections of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000 and MRDI's Brimstone Restart Report, June 2002.

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. 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 downward to the west 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.

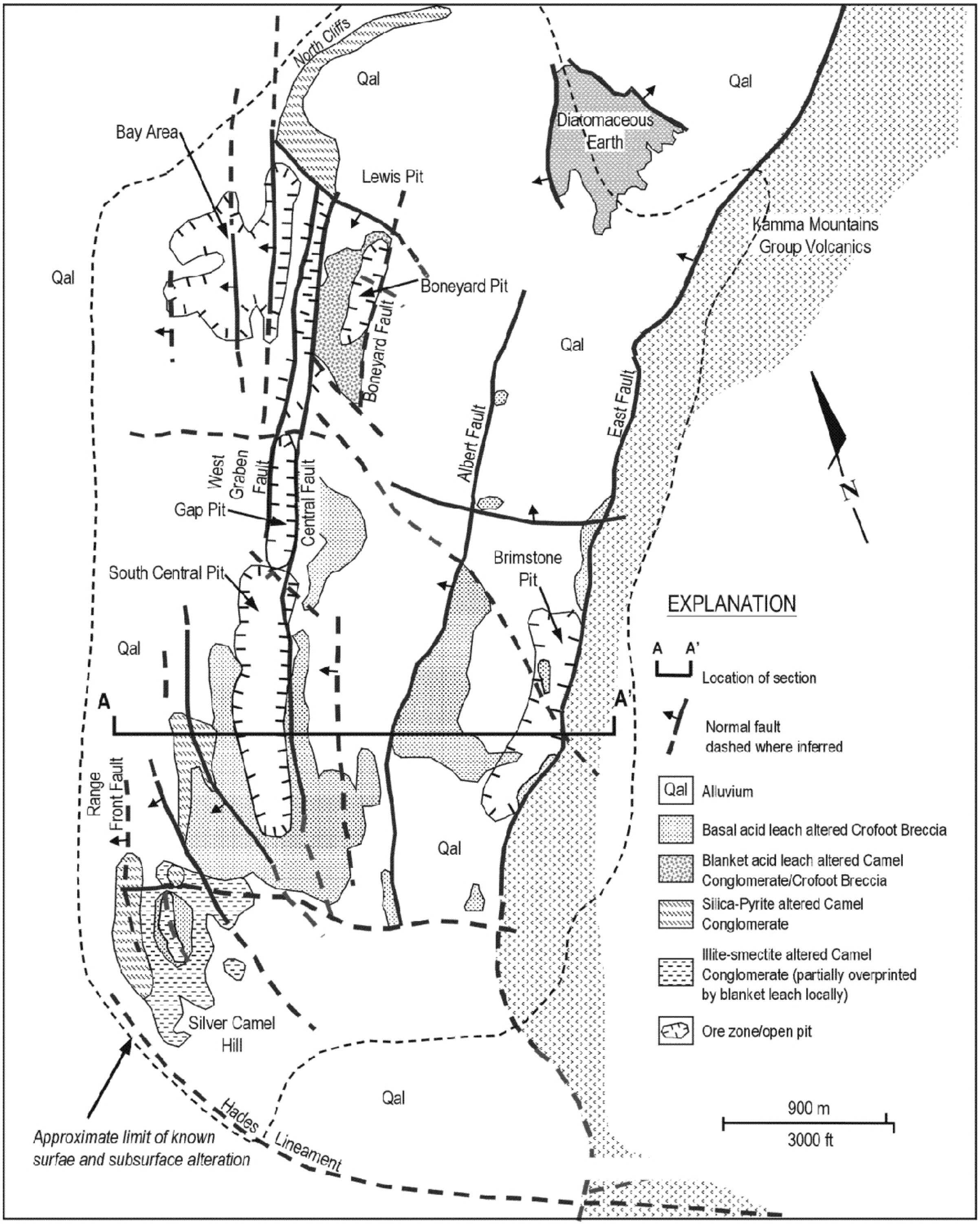
Four major north-northeast-trending, west-dipping, normal fault zones broadly control the location of gold mineralization. From west to east, these fault zones are referred to as the Central, Boneyard, Albert, and East faults. Figure 7.2a shows a north-looking section through the Hycroft Mine outlying structures and volcanic stratigraphy. Figure 7.2b outlines structures and alteration types in the same 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.

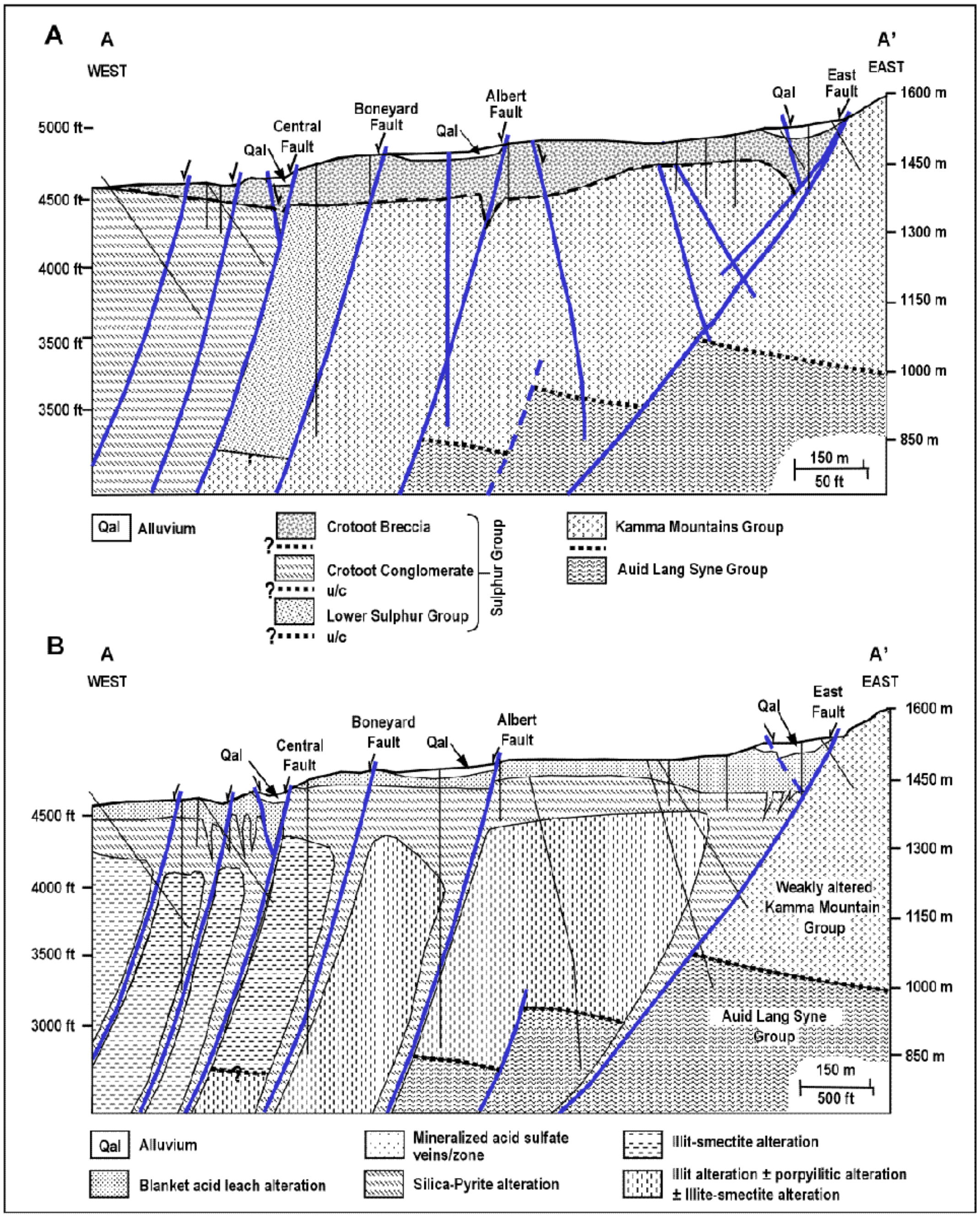
Mineralization in the Albert Zone is present along the Albert fault, located approximately 2500 feet east of the Central fault deposits and 2,000 feet west of the Brimstone deposit. The Albert Zone is hosted by KMG eruption breccias and volcanic flows in the hanging wall of the west-dipping fault.

Figure 7.1 Simplified Geological Map of the Sulfur District



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

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



A) Simplified East-West Cross Section A-A' Throught 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 Brimstone deposit is hosted by volcanic rocks of the KMG present in the hanging wall of the East fault. The volcanic rocks are principally eruption breccias and volcanic flows proximal to vents. The volcanics overly 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 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. The fault clearly shows steep normal movement, with slickensides that plunge 80-85-degrees south. 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 ore bodies 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 - Brimstone Deposit

The Hycroft Mine consists of Tertiary- to Recent-age, fault-controlled, low-sulfidation gold deposits that occur over an area measuring 3 miles in a north-south direction by 1.5 miles in an east-west direction. Mineralization extends to depths of less than 330 feet in the outcropping to near-outcropping portion of the Bay deposit on the northwest side and to over 990 feet in the Brimstone deposit in the eastern portion of the Hycroft property.

Gold-bearing rocks at Brimstone are located in the hanging wall of the East fault. These rocks were highly altered by four phases of alteration. Gold 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 ore.

7.2.1 Hanging Wall of the East Fault – Brimstone Deposit

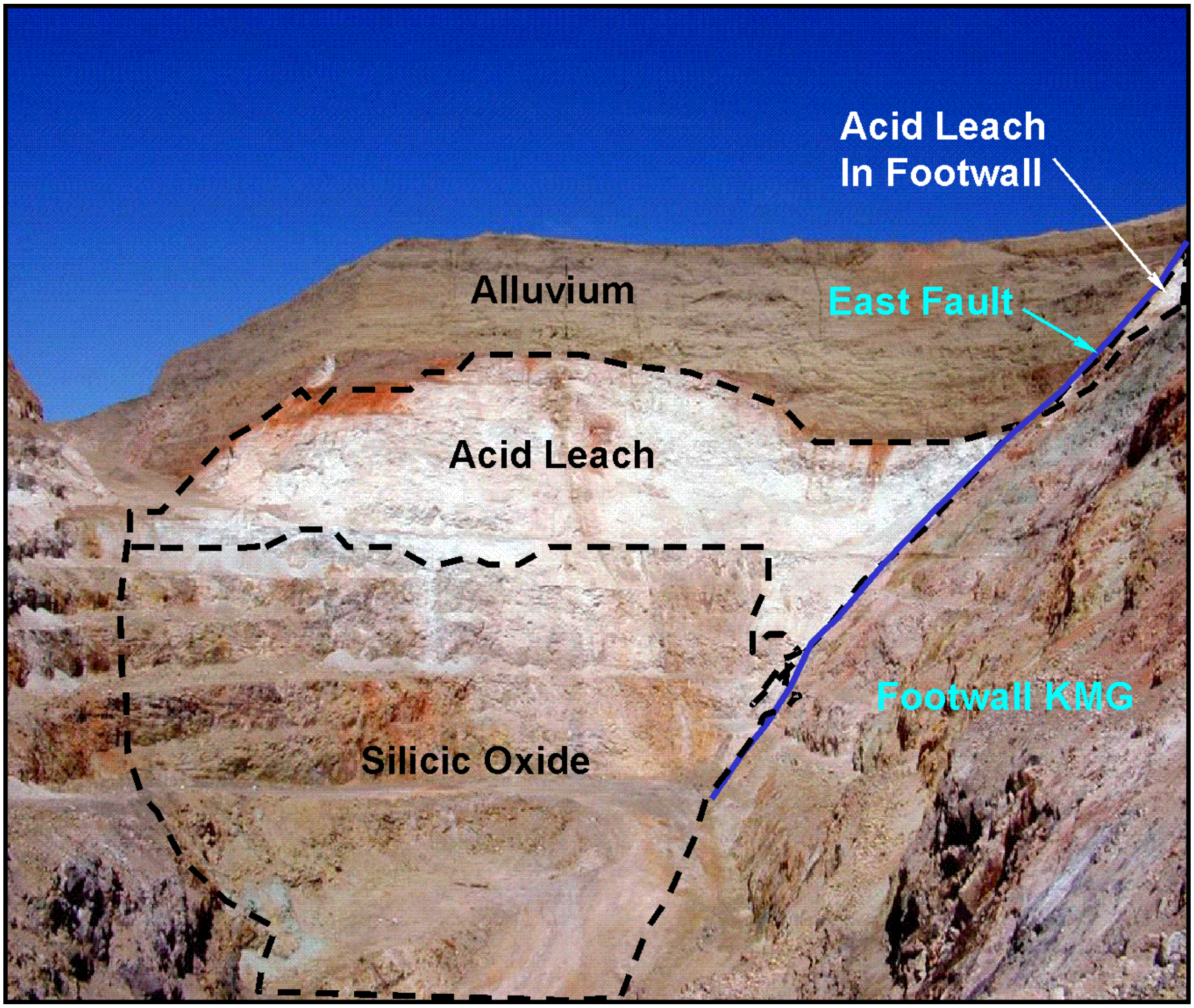
The upper one hundred to two hundred feet of rock in the hanging wall of the East fault is a late hydrothermal eruption breccia called the Crofoot Breccia. This breccia is matrix-supported with clasts dominated by Kamma Volcanics. Rarely, clasts of oxidized fracture-controlled chalcedony-pyrite-marcasite mineralization are observed in the Crofoot Breccia, indicating a possible syn- or post-mineralization steam-dominated eruption event. No Crofoot Breccia is observed 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 Brimstone Deposit, the gold-bearing host rocks are the altered felsic-volcanic rocks of the KMG. The rocks of this group, in the hanging wall of the East fault, are dominated by epiclastic feldspathic tuffs and aphanitic rhyolite flows. Correlation of these units is difficult due to the lack of diamond drilling to provide core in which it is possible to observe macroscopic textures, and the obliteration of original textures by later acid-leach alteration.

7.2.2 Rocks in the Footwall of the East Fault – 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 is weak, with propylitic alteration, clay alteration, and oxidation occurring within 50 to 150 feet of the East fault

Figure 7.3 Brimstone North Pitwall Geology



8.0 DEPOSIT TYPE

The information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000.

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 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

The known gold mineralization within the Hycroft Mine property extends for a distance of 3 miles in a north-south direction by 1.5 miles in an east-west direction. 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 990 feet in the Brimstone deposit in the eastern portion of the property.

Not all the mineralized zone is oxidized, and the depth of oxide ore varies considerably over the area of the deposits. The determination of whether or not mineralized material can be mined economically is dependent on the grade of mineralization, the depth of overburden, and the degree of oxidation.

9.0 MINERALIZATION

The information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000 and MRDI's Brimstone Restart Report, June 2002.

9.1 Alteration and Mineralization in the East Fault Hanging Wall – Brimstone Deposit

9.1.1 Introduction

Highly altered rocks are almost exclusively found in the hanging wall of the East fault. There are 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.*

The following sections describe each alteration and mineralization type in detail.

9.1.2 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 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. Very few drill holes pass 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.*

9.1.3 Fracture-Controlled Chalcedony-Pyrite-Marcasite Mineralization (FCCPM)

The fracture-controlled chalcedony-pyrite-marcasite mineralizing event was associated with primary gold 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 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 rockmass. Chaotic breccia was logged as Structural Code 2.

Figure 9.1 Schematic Cross Section of Barren Silica-Pyrite Alteration

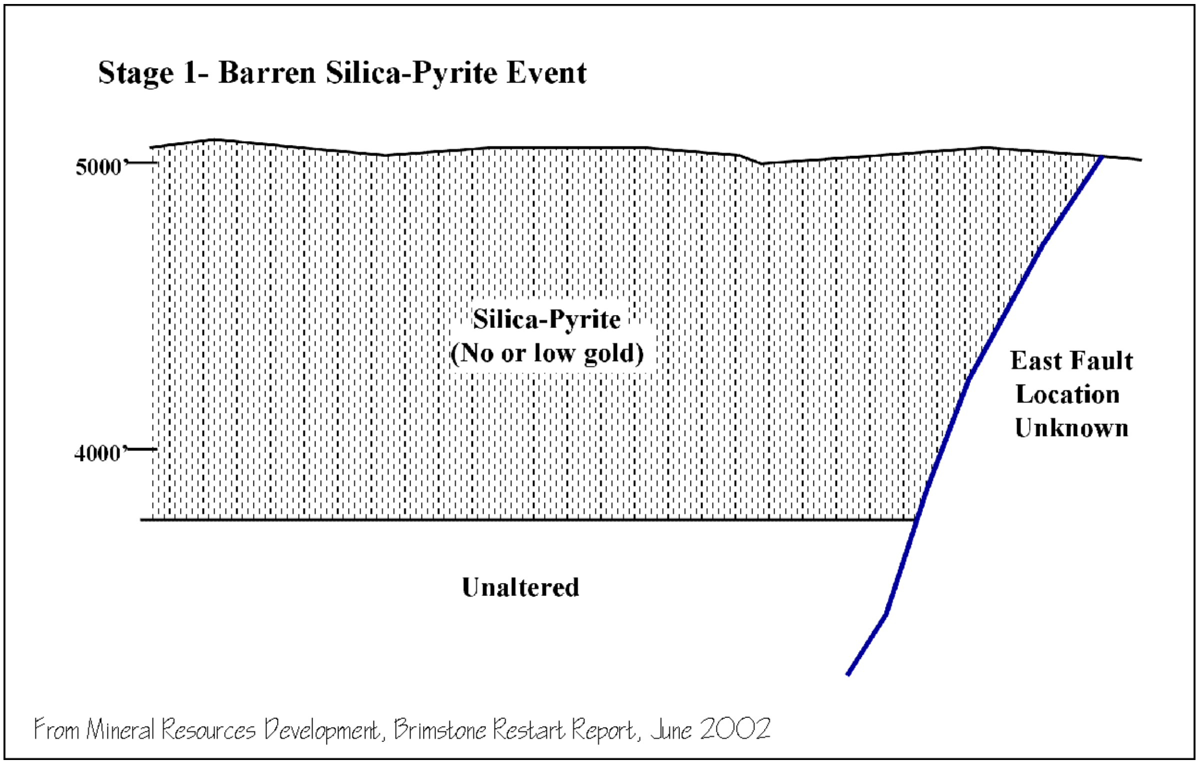
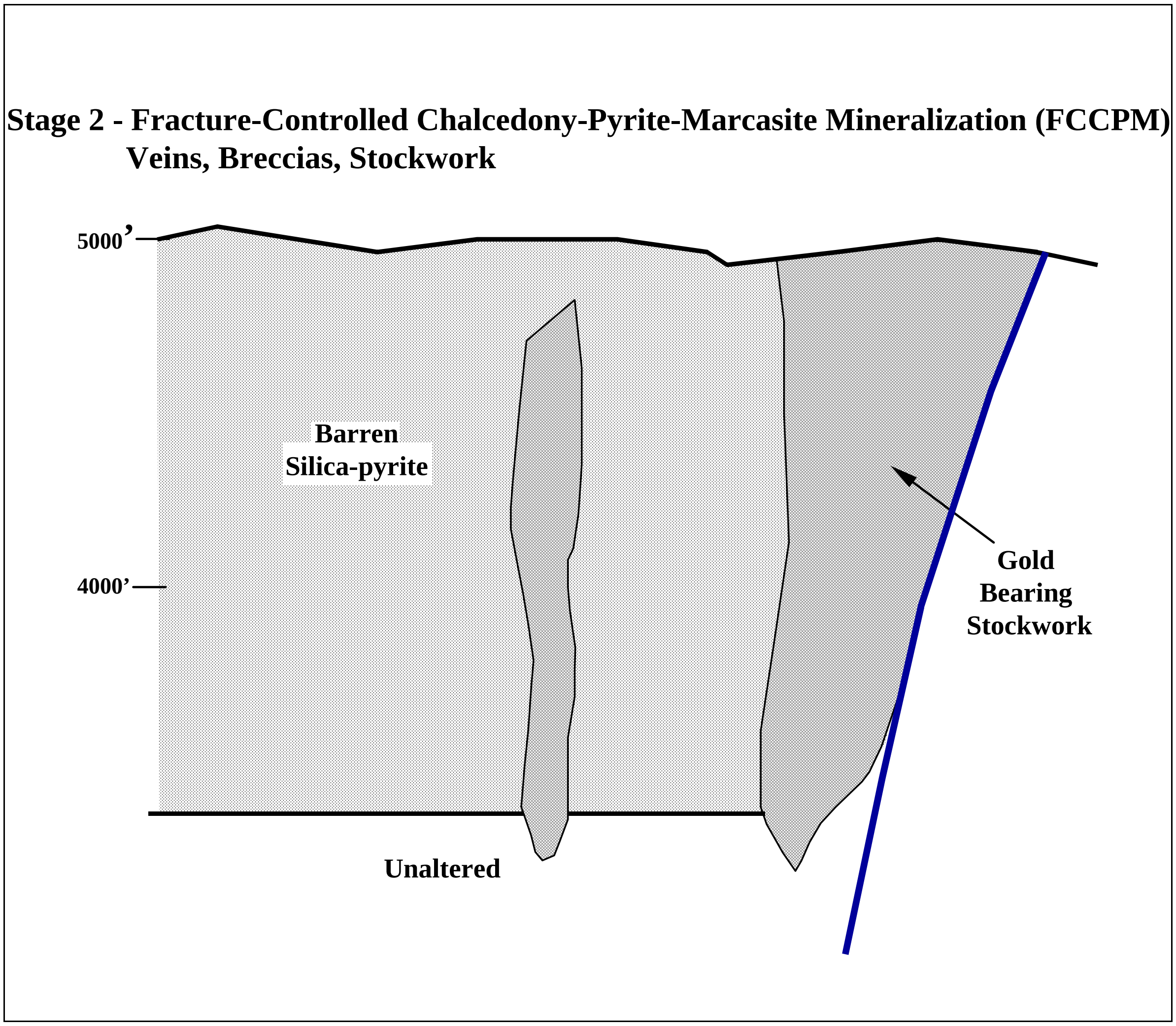


Figure 9.2 Schematic Cross Section of Fracture-Controlled Chalcedony-Pyrite Marcasite Alteration



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 subparallel 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 mineralization was most likely introduced during this event, and evidence for this is two-fold:

- *Visible gold (50 to 120 microns in size) has been identified within the chalcedonic veins in thin sections from drill hole 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 Average Total Gold Grades of Rocks With and Without FCCPM
(after MRDI)

Alteration Domain	*Avg. FA Au With FCCPM*	*Avg. FA Au Without FCCPM*	*% of Domain With FCCPM*
Acid leach	*0.014*	*0.008*	*21*
Oxide	*0.016*	*0.005*	*58*
Sulfide	*0.015*	*0.006*	*57*

The data in this table clearly shows that for both 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 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. Drill hole 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.

9.1.4 Hypogene Acid Leach-Oxide Alteration

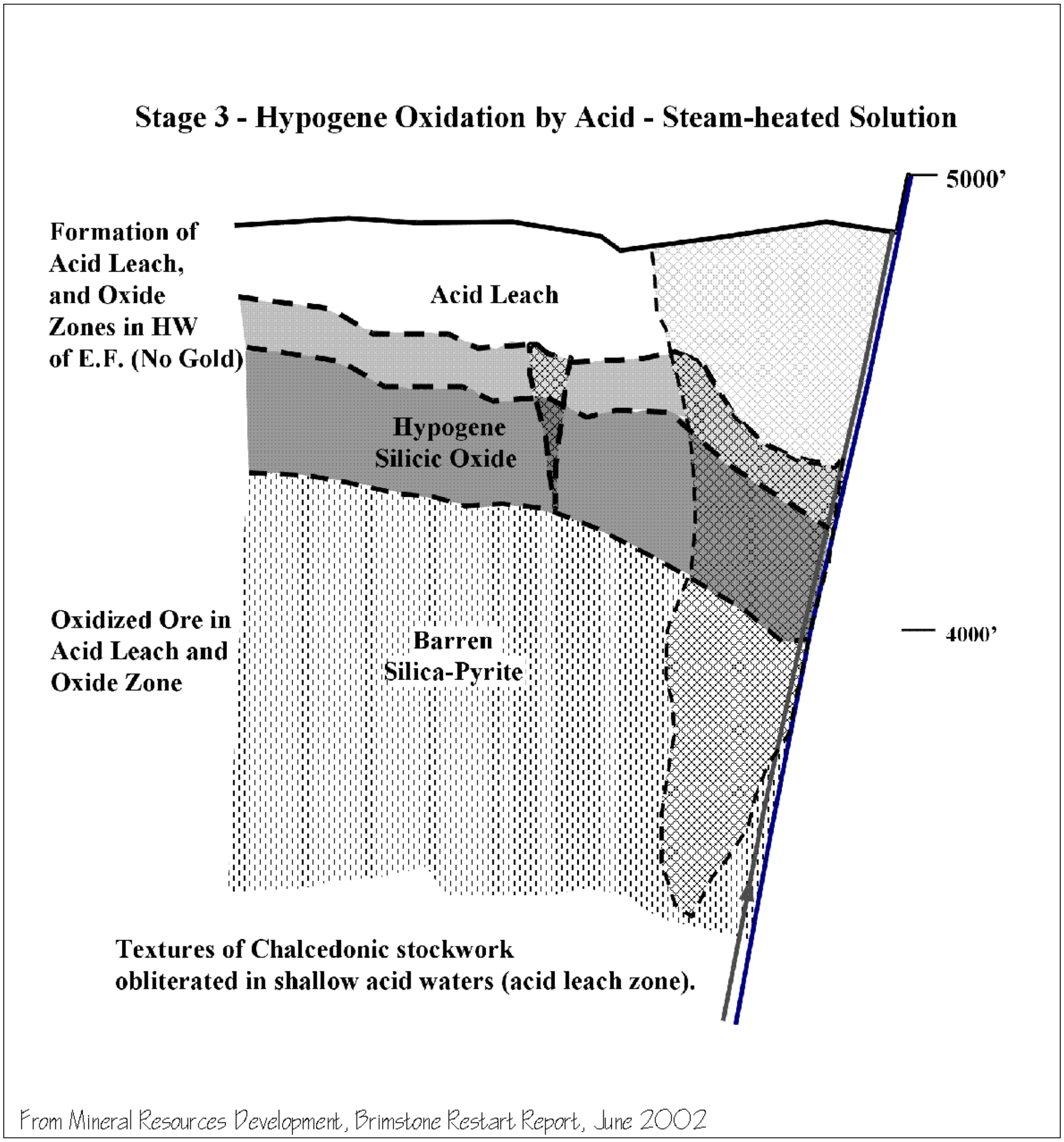
The hypogene, acid-leach, oxidation-alteration event determined the distribution of the two dominant types of oxidized material, "acid-leach" and "oxide" rocks. The alteration is geometrically zoned, suggesting that a single event produced the zoning. Acid-leach and oxide alteration clearly overprint 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.

9.1.5 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- to 200-feet-thick over the entire study area, but reaches thicknesses of 450 feet in the immediate hanging wall of the East fault.

Figure 9.3 Schematic Cross Section of Acid Leach and Oxide Alteration



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.*

9.1.6 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.

9.1.7 Oxide Alteration

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

9.1.7.1 Silicic Oxide

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.

9.1.7.2 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 either 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.

9.1.8 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 9.5 is a photograph of the north pit wall at Brimstone with the geologic contacts outlined. It is clear from the photograph that movement along the East fault, in a normal sense, of at least 200 feet has occurred. A sliver of acid-leach material can be seen in the footwall of the East fault above the main body of the acid-leach material in the hanging wall.

Figure 9.4 Supergene Oxidation Plus Normal East Fault Movement

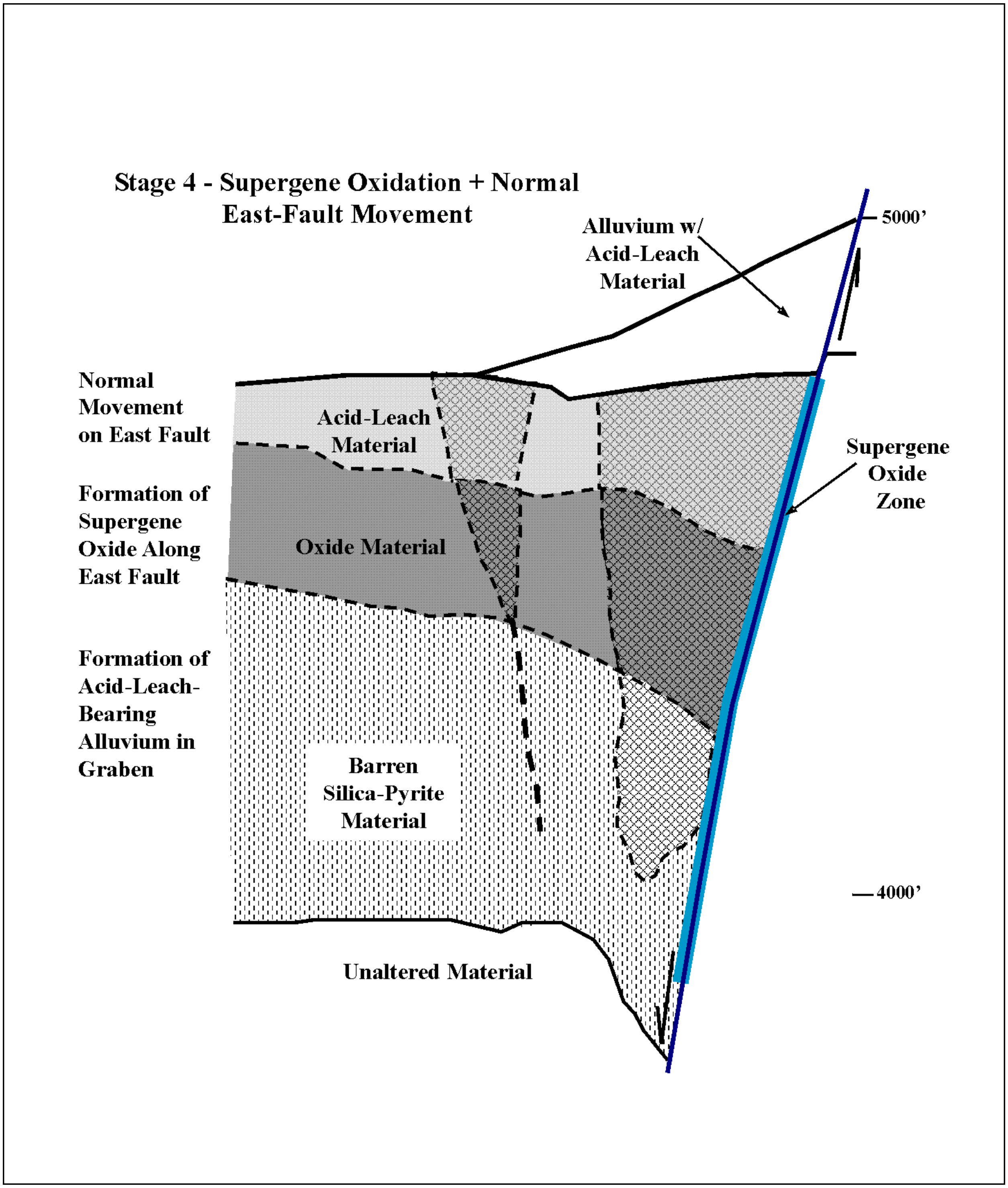
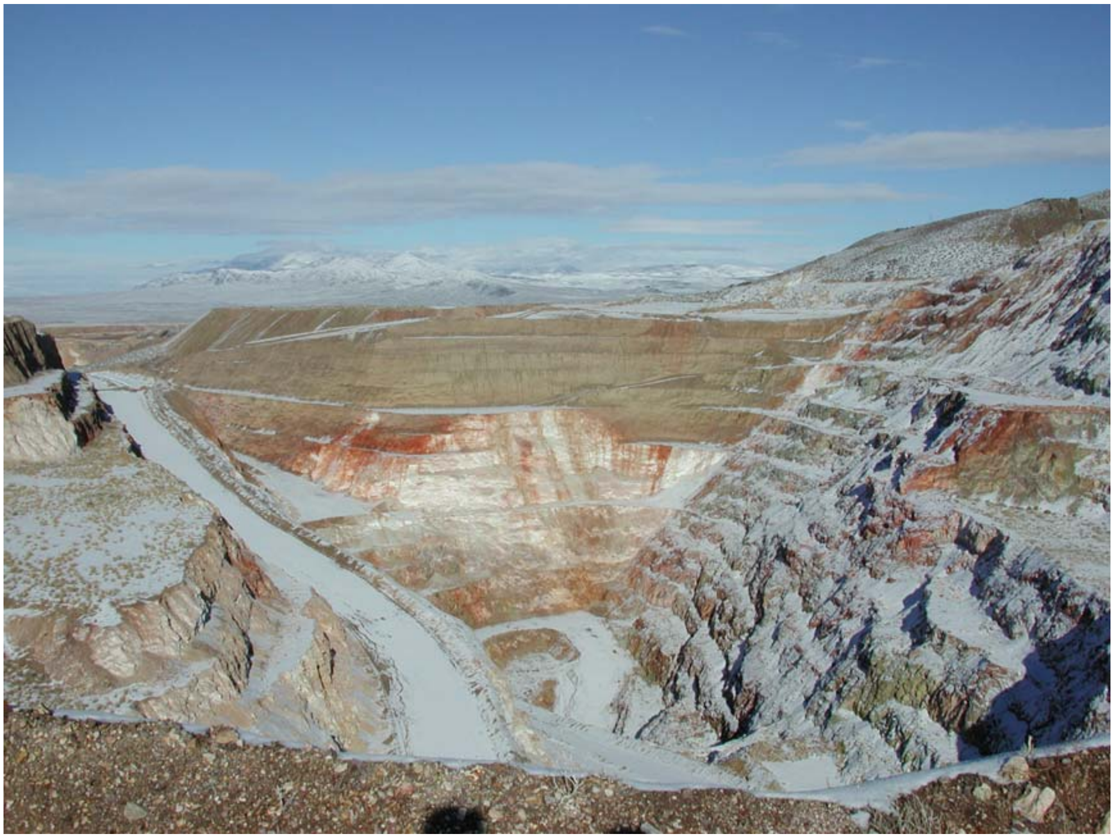


Figure 9.5 North Wall of the Brimstone Pit



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 Al2O3, 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.

10.0 EXPLORATION

10.1 Historic Exploration and Development

Between 1985 and 1999, HRDI drilled a total of 3,123 exploration drill holes, totaling 943,822 ft. Canyon completed 33 drill holes totaling 13,315 ft of reverse circulation drilling during 2005. The current Hycroft drill hole database consists of the former holes, plus 61 RC holes drilled by Homestake in 1982 and 29 rotary holes completed by Homestake in 1981. 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.

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,Cut4,Gap,North,SC
1988	RC	Hycroft	73	25,855	Alb,Brim,Cut4,North,SC
1989	RC	Hycroft	43	15,780	Alb,Brim,Cut4,North,SC
1990	DD	Hycroft	8	11,247	Cut 4,Sulfur
1990	RC	Hycroft	134	52,675	Alb,Brim,Cut4,North,SC
1991	RC	Hycroft	147	44,360	Cut 4, North,SC
1992	RC	Hycroft	265	83,030	Alb,Brim,Cut4,North,SC
1993	DD	Hycroft	6	2,318	Alb,Brim,SC
1993	RC	Hycroft	297	105,500	Alb,Brim,Cut4,North,SC
1994	DD	Hycroft	3	4,990	Brim
1994	RC	Hycroft	208	78,650	Alb,Brim,Cut4,Boneyard,SC
1995	RC	Hycroft	355	157,515	Alb,Brim,Cut4,Gap,Boneyard,SC
1996	DD	Hycroft	1	1,078	Brim
1996	RC	Hycroft	164	75,000	Alb,Brim,Cut4,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**	

A breakdown of the drill holes by type and orientation is found in Table 10.2.

Table 10.2 Exploration Drill Holes 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 - 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 zone is shown below in Table 10.3.

Table 10.3 Discovery Years of Hycroft Oxide Zones

Area	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.

An interesting statistic is that 95% of all Hycroft exploration holes are either within mined areas, or in areas to be mined with current reserves and resources.

10.2 History of Geologic Logging

The information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000.

A 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 relogged. 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 Prior to 1999
(after MRDI)

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 relogging. 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 comprised 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 relogging 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 relog 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 relogging, an additional field was added to record an estimate of the percent of sulfur. Samples that had already been relogged 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.

Between May and December 1999, approximately 410 Brimstone drill holes were relogged in accordance with the new classification system by the two geologists assigned to the project. Between December 1999 and January 2000, the Albert zone data was relogged. MRDI checked use of the logging system by comparing new logs against RC sample chips. Chip trays representing entire length of holes 99-1975, 99-1504 and 95-2648 were checked against drill logs. The logs were found to be accurate.

Table 10.5 Lithological, Structure, Alteration, Sulfur and Oxidation Codes
1999 RC Sample Logging
(after MRDI (developed by Vista))

CODE	**Lithology**	CODE	**Structure**	CODE	**Alteration**
0	Alluvium	0	No Structure	0	Unaltered
1	Gouge Material	1	Jig-Saw Breccia	1	Silicic; Quartz, Chalcedony, K-Feldspars
2	Und. Felsic Volcanics	2	Chaotic Breccia	2	Acid Leach
3	Rhyolite	3	Fractured Zone	3	Propylitic
4	Flow Banded Rhyolite	4	Fault, Gouge, Shear Zone	4	Argillic
5	Rhyolite Tuff	5	Voids	5	Calcite
6	Epiclastic Tuff	6	Stockwork	6	Clay
7	Crofoot Breccia	7			
8	Mafic Volcanics	8	Quartz Vein > 1'		
9	Auld Lang Syne Sedimentary Group Or Equivalent	9	Calcite Vein > 1'		
		9	Gypsum in Acid Leach		
-1	Data Missing	-1	Data Missing	-1	Data Missing
0	Alluvium	0	No Structure	0	Unaltered
CODE	Presence or Absence of Sulfur and or Sulfide **(Observed)**	CODE	**Oxidation State**	CODE	**% Native Sulfur (Observed)**
0	No Sulfur and Sulfide	0	< 25 % of the sulfide oxidized	0	Trace
1	Sulfide	1	> 25 % of the sulfide oxidized	1	<5%
2	Sulfur			2	> 5%
3	Sulfur and Sulfide			3	> 10%
-1	Data Missing	-1	Data Missing	-1	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 relogging. 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 relogging 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.

10.3 Surveying

The information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000.

A mine grid was established for all survey work at the Brimstone deposit. The grid is effectively based on magnetic north.

10.3.1 Drill Collar Surveys

Standard operating procedure is to lay out planned exploration drill-hole locations by surveying. After drilling was completed on a site, the actual drill-hole location was surveyed, and the survey data was then 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 was entered into the database

10.4 History of Drilling and Sampling in the Brimstone/Albert Area

Exploration drilling in the Brimstone/Albert's area began during 1986. Since then, a total of 269,396 feet have been drilled in 571 holes as shown in Table 10.6.

Table 10.6 Exploration in the Brimstone/Albert Area Since 1988
(after MRDI)

Year	RC	Diamond	Total	RC	Diamond	Total
1986	2,185		2,185	6		6
1988	5,925		5,925	16		16
1989	12,150		12,150	27		27
1990	29,575		29,575	60		60
1991	465		465	1		1
1992	28,098		28,098	67		67
1993	35,169	1,536	36,705	86	4	90
1994	34,680		34,680	64		64
1995	51,450		51,450	94		94
1996	58,495		58,495	127		127
1999	5,545	4,120	9,665	11	8	19
2005	13,315		13,315	33		33
Totals	**277,055**	**5,656**	**282,711**	**592**	**12**	**604**

11.0 DRILLING

Most of the information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000 and MRDI's Brimstone Restart Report, June 2002.

Exploration drilling in the Brimstone/Albert's area began during 1986. Since then a total of 269,396 ft have been drilled in 571 holes. All except twelve core holes were completed by reverse circulation drills.

11.1 Pre-1999 Drilling

Reverse-circulation drilling of the 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

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 drill hole 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 both 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 backreaming. 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.

Table 11.1 1999 Twin RC Drilling Campaign
(after MRDI)

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

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.

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 drill hole assays.

Table 11.2 Comparison of the Twin Drill Holes (Fire Assays)

1999 Drill Hole	Interval	Feet	Fire oz Au/ton	Cn Soluble oz Au/ton	Old Drill Hole	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	0-565	565	0.009		89-1419	0.008	
99-1432B	25-500	475	0.017		89-1432	0.009	
99-1504B	30-600	570	0.014		90-1504	0.012	
99-1523B	30-370	340	0.016		90-1523	0.017	
99-1944B	0-250	250	0.003	0.002	92-1944	0.005	0.003
99-1949B	0-410	410	0.013	0.013	92-1944	0.011	0.011
99-1950B	0-405	405	0.018	0.014	92-1950	0.014	0.010
99-1975B	75-545	470	0.027	0.022	92-1975	0.023	0.018
99-1976B	175-580	405	0.016	0.012	92-1976	0.020	0.007
99-2648B	100-305	205	0.004	0.002	95-2648	0.004	0.001
Totals		**4,325**	**0.015**			**0.013**	

Table 11.3 Comparison of the Twin Drill Holes (Cyanide Soluble Assays)

1999 Drill Hole	Interval	Feet	Fire oz Au/ton	Cn Soluble oz Au/ton	Old Drill Hole	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	330-565	235		0.010	89-1419		0.010
99-1432B	240-460	220		0.024	89-1432		0.014
99-1504B	125-600	475		0.009	90-1504		0.006
99-1523B	195-380	185		0.021	90-1523		0.027
99-1944B	0-250	250	0.003	0.002	92-1944	0.005	0.003
99-1949B	0-410	410	0.013	0.013	92-1944	0.011	0.011
99-1950B	0-405	405	0.018	0.014	92-1950	0.014	0.010
99-1975B	75-545	470	0.027	0.022	92-1975	0.023	0.018
99-1976B	175-580	405	0.016	0.012	92-1976	0.020	0.007
99-2648B	100-305	205	0.004	0.002	95-2648	0.004	0.001
Totals		**3,490**		**0.013**			**0.010**

11.3 Canyon Resource 2005 drilling

Canyon completed 33 reverse circulation drill holes 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 drill holes.

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 that 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 (Figure 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 1999 RC Sample Recovery Versus Fire Assay Gold

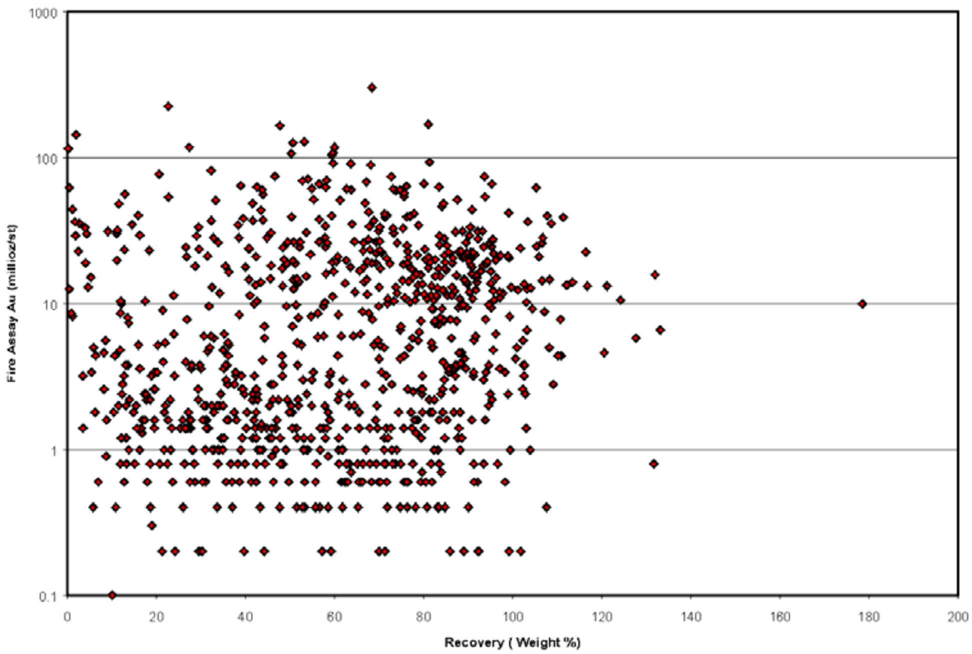


Figure 11.1a Recovery Vs. Chemex Fire Assay Au

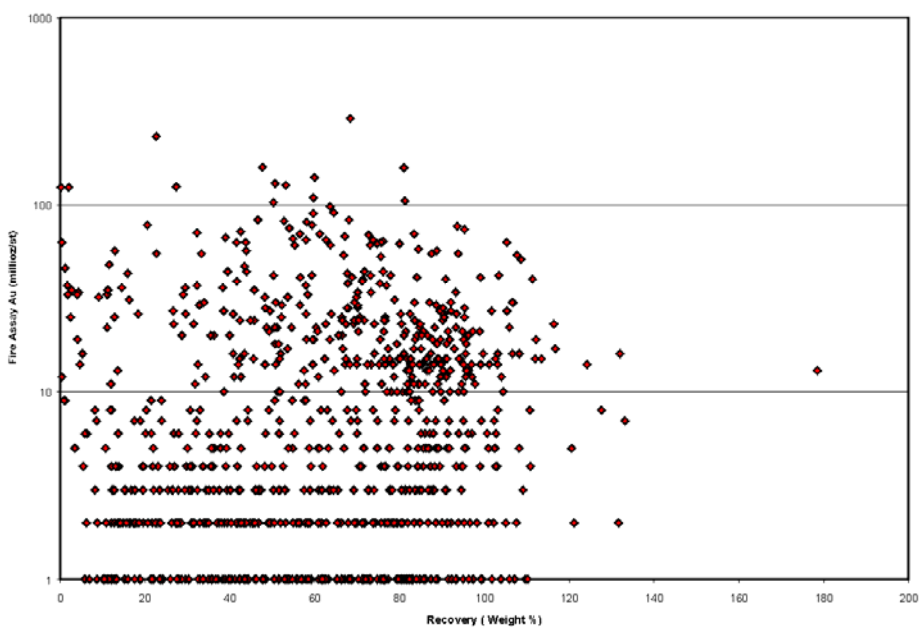


Figure 11.1b Recovery Vs. Hycroft Fire Assay Au

From Mineral Resources Development, Brimstone Restart Report, June 2002

Figure 11.2 1999 RC Sample Recovery Versus CN-Soluble Gold and Silver

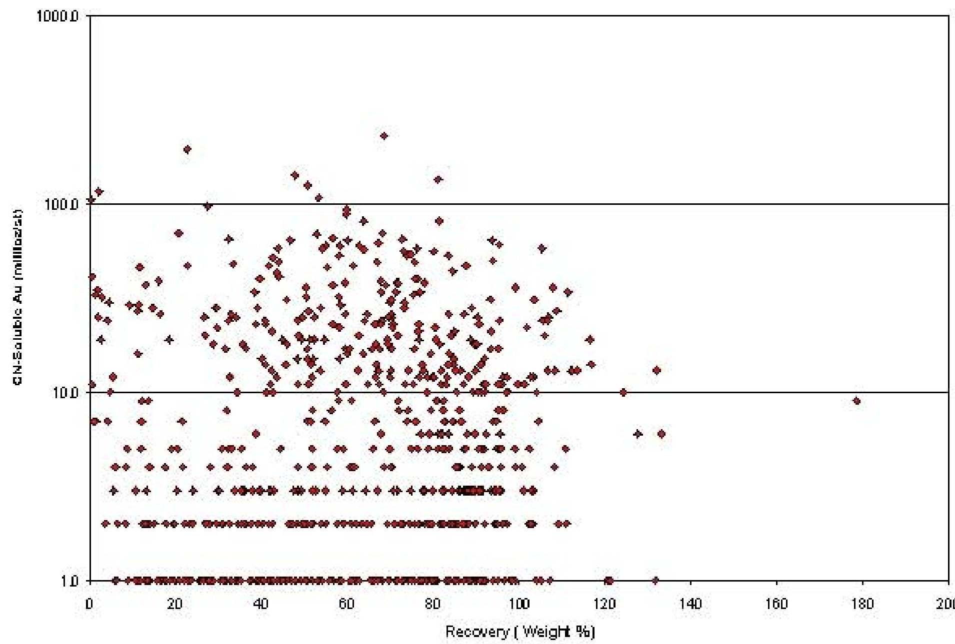


Figure 11.2a Recovery Vs. Hycroft CN-Soluble Au

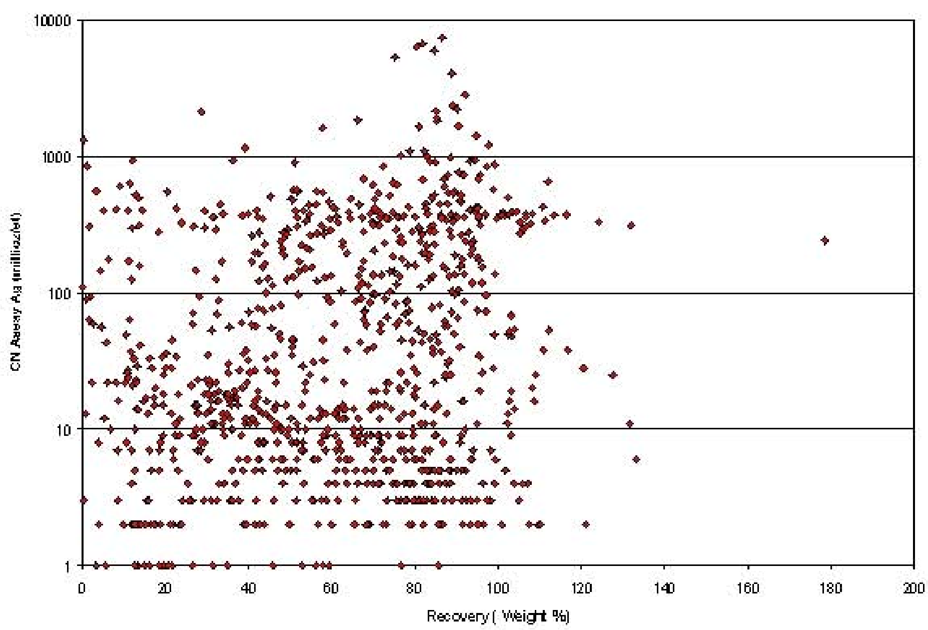


Figure 11.2b Recovery Vs. Hycroft CN-Soluble Ag

From Mineral Resources Development, Brimstone Restart Report, June 2002

11.4.3 Canyon 2005 Drilling

Drill hole recovery data was recorded for the 2005 drill program, however, the sample weights at 100% drill recovery are so variable, MDA believes this data is of little use unless consistent recovered weights for each rock type can be established.

12.0 SAMPLING METHOD AND APPROACH

Most of the information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000.

12.1 Reverse Circulation Sampling

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 drill holes (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.

12.1.1 2005 Canyon Reverse Circulation Sampling

The first 29 drill holes completed used a central return bit to improve sample recovery, however, the final 4 holes used a normal interchange after the center return hammer failed.

13.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

Most of the information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI. for Vista Gold Corp. in May, 2000. Other information is taken from a Vista Gold Corp. internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000.

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. This sample preparation method meets industry standards for preparation of Brimstone type ores.*

13.2 Drill Sample Preparation and Analysis

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 spliter 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 1999 Reverse Circulation Sample Preparation and Assaying

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
(after MRDI)

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:

1. *Weigh each sample as received. This weight was reported and recorded as the wet weight.*
2. *Oven-dry the samples at a temperature not to exceed 175^{o} F. This temperature was selected to minimize the volatilization of trace elements and sulfur.*
3. *Weigh each sample after drying. This weight was reported and recorded as the dry weight.*
4. *Crush the entire sample to 95% passing 10 mesh prior to any splitting.*
5. *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.*
6. *Pulverize the 400- to 800-gram split to 95% passing 150 mesh.*
7. *Riffle-split the pulp with one split retained by ALS Chemex for analysis and the other returned to Hycroft.*
8. *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:

1. *The sample pulps are blended on a roll cloth and 20 grams are stippled out and placed in 50-ml plastic centrifuge tubes.*
2. *20 grams of 20 lb. per ton NaCN solution containing 20 lb. per ton of NaOH are dispensed into each tube.*
3. *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 ^{o} F. The racks are placed so the centrifuge tubes are in a horizontal position.*
4. *The tubes are shaken at a moderately slow speed, approximately 60 rpm on the eccentric, for one hour.*
5. *The sample tubes are removed from the water bath, allowed to cool for several minutes, and then centrifuged.*
6. *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; and*

- *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.

13.2.3 2005 Reverse Circulation Sample Collection

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.

13.3 QA/QC Check Samples, & Check Assays

Up 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.*

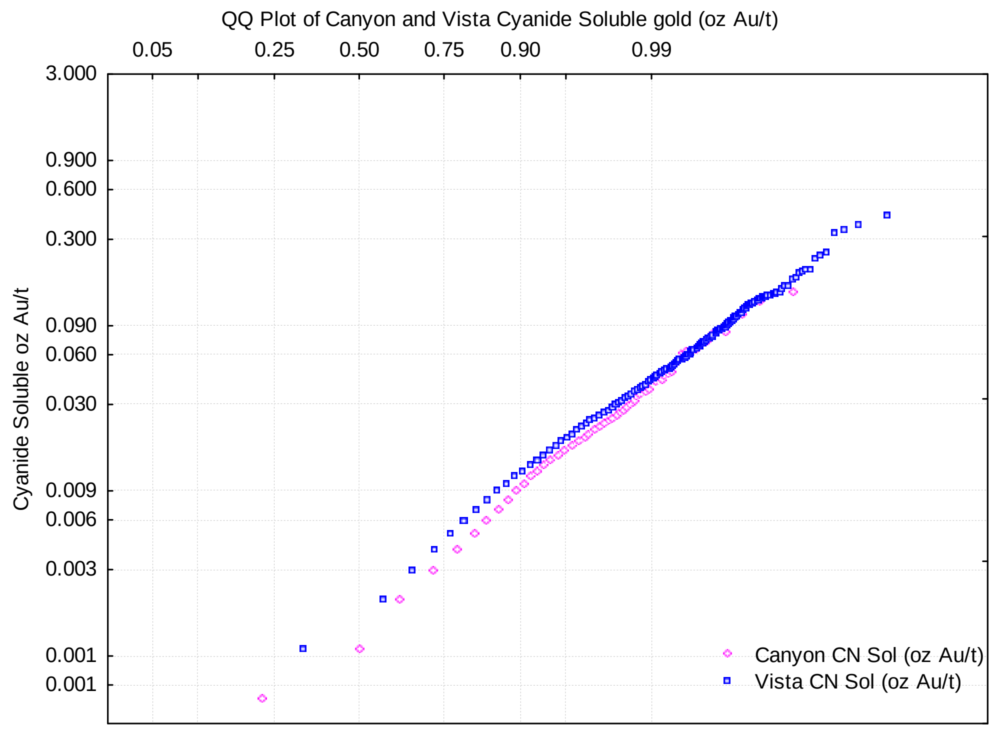
All exploration samples subsequent to 1991 that were assayed for cyanide-soluble gold and cyanide-soluble silver, were assayed at the Hycroft Mine laboratory. Fire assays were also performed. In most cases, if the fire assay was below detection, the cyanide-soluble assays were not performed. No decipherable 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 Drill Hole program were fire-assayed for gold by both 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.

13.4 Comparison of Canyon and Vista Assay Data

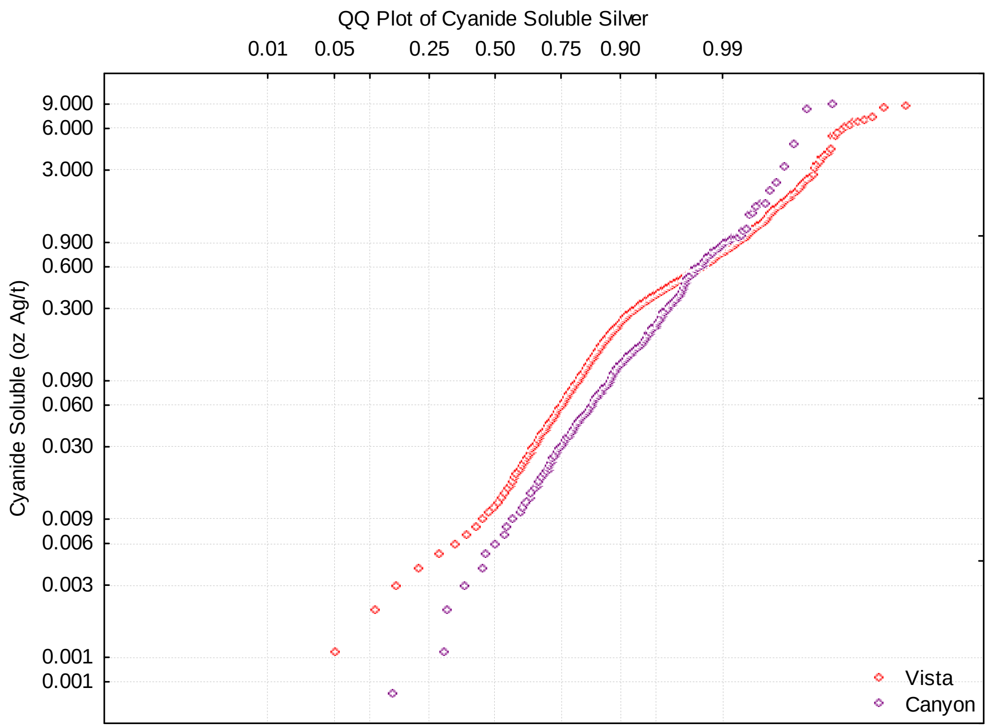
The assay data from the Canyon and Vista drilling programs were compared. The comparison is shown in Figure 13.1 for gold and Figure 13.2 for silver.

Figure 13.1 QQ Plot of Canyon and Vista Gold Assay Data



The Canyon drilling shows a higher grade distribution for assays below 0.05 oz Au/ton, likely due to the improved recovery of center return hammer drilling by Canyon.

Figure 13.2 QQ Plot of Canyon and Vista Silver Assay Data



The silver assay data for the Vista drilling is not complete. The comparison shows that Canyon found higher grades below 0.50 cyanide soluble oz Ag/ton and lower grades above 0.50 cyanide soluble oz Ag/ton. MDA suggests that areas with higher grade cyanide soluble silver be investigated in more detail.

14.0 DATA VERIFICATION

The information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista in May 2000. Other information is taken from a Vista internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000. The most recent evaluation was completed by Alan Noble in his report, Corrections for Bias in RC drilling and High-Sulfur Cyanide Assays, June 2005.

14.1 Integrity of Database

MRDI reviewed the database to establish its validity. This work and the results are described in the following sections.

14.1.1 Data Selections

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 were 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 Checks

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
(after MRDI)

Reason for Check	No. Samples	No. Checked	No. Errors	% Errors
Cyanide grade significantly greater than fire assay grade	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 grades	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
(after MRDI)

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 error

Table 14.3 Tabulation of Errors in Random Waste Category
(after MRDI)

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 error

Of the randomly selected samples, 0.8 percent 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 drill hole name and depths (from and to), there are six fields containing single digit integers corresponding to geologic *observation. Approximately five percent of the relogged drill hole intervals were selected at random. Nearly every relogged drill hole 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. Drill hole logs having errors were rechecked by Vista in their entirety. This led to detection and correction of some additional entry errors.*

Subsequent investigation by MRDI revealed entries where an interval had native sulfur observed, but no estimate of percentage (a separate field). This led to some additional relogging to correct these discrepancies. MRDI believes the corrected geology database has a sufficiently low incidence of entry errors for use in a resource model.

14.1.4 Collar Survey Checks

MRDI checked every drill hole 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 drill holes differing by one number were given the same collar coordinates. In the other case, the drill hole 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 Down-hole Survey checks

Very few drill holes had down-hole surveys. All drill holes 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, and

- 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/st:

  - Acid Leach Ore: Original assay x 1.40
  - Oxide Ore Original assay x 1.19

- Fire Assay gold: Adjustments were made to assays with gold ,0.08 oz/st:

  - Acid Leach Ore: Original assay x 1.39
  - Oxide Ore Original assay x 1.19

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

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*(CNsol Au) + 0.2944*(Fire)
Minor S (S=1)	
1988 - 1997, CNsol Au/Fire < 0.4	y = 1.450*(CNsol Au) + 0.160*(Fire)
1988 - 1997, CNsol Au/Fire 0.4 to 0.9	y = 0.3143*(CNsol Au) + 0.6143*(Fire)
1999	no adjustment
Other Oxide (not acid leach)	
Trace S (S=0) or Minor (S=1)	
CNsol Au/Fire < 0.33	y = 1.387*(CNsol Au) + 0.2157*(Fire)
CNsol Au/Fire 0.33 to 0.9	y = 0.2923*(CNsol Au) + 0.6788*(Fire)

Table 14.5 shows the correction factors applied to the cyanide soluble assays from the twin drill holes.

Table 14.5 Correction Factors Applied to the 1999 Twin Drilling
(after MRDI)

1999 Drill Hole	**Interval**	**Feet**	**Fire oz Au/ton**	**Cn Soluble oz Au/ton**	**Old Drill Hole**	**Original Fire oz Au/ton**	**Corrected Cn Soluble oz Au/ton**
99-1378B	180-410	230	0.015	0.012	88-1378	0.009	0.012
99-1419B	330-565	235		0.010	89-1419	0.010	0.011
99-1432B	240-460	220		0.024	89-1432	0.014	0.018
99-1504B	125-600	475		0.009	90-1504	0.006	0.007
99-1523B	195-380	185		0.021	90-1523	0.027	0.029
99-1944B	0-250	250	0.003	0.002	92-1944	0.003	0.004
99-1949B	0-410	410	0.013	0.013	92-1944	0.011	0.013
99-1950B	0-405	405	0.018	0.014	92-1950	0.010	0.011
99-1975B	75-545	470	0.027	0.022	92-1975	0.018	0.020
99-1976B	175-580	405	0.016	0.012	92-1976	0.007	0.010
99-2648B	100-305	205	0.004	0.002	95-2648	0.001	0.001
Totals		**3,490**		**0.013**		**0.010**	**0.012**

14.3 Correction of assays by 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 drill hole data contains codes identifying the quantity of sulfur in the sample based on visual examination of drill cuttings by the geologist. Sulfur categories are:*
>
> > *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:

$$Y = A \ln(X) + B,$$ *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.

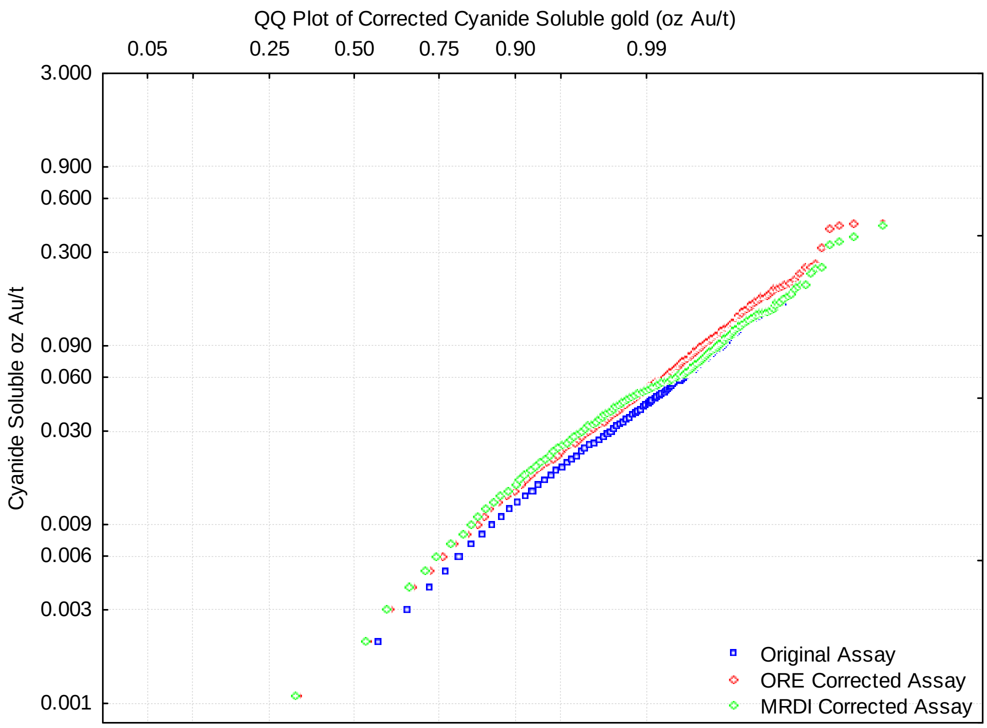
4) Sulfur corrections on the AuCN assay were then made by looking up the appropriate correction equation for the sulfur content category, calculating the uncorrected AuCN/AuFA ratio, calculating the corrected ratio from the correction curve, then multiplying the corrected ratio times the original AuFA 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.

MDA believes these adjustments are valid as there is considerable supporting evidence to indicate that the drilling introduced a sampling bias. The blasthole and production data clearly indicates higher grades than were indicated by the exploration drilling. MDA does not generally support the practice of applying correction factors to raw data, but in this case, the corrected data set does more closely model the deposit, as supported by past production records. A comparison of the original, MRDI's correction, and ORE's correction is shown in Figure 14.1

Figure 14.1 Comparison of Original and Corrected Assay Data



The corrections applied by ORE are generally lower than the MRDI corrections except for 1% of the higher grade assays for which a higher grade correction was applied.

15.0 ADJACENT PROPERTIES

The Rosebud gold mine, located in the Rosebud Mining District, in Pershing County, Nevada is approximately five miles west of the Hycroft Mine. Hecla and Newmont Gold Company each have a 50% interest in the mine. The information contained here was taken from Hecla's 10-K disclosures.

The Rosebud property consists of a 100% interest in three patented lode-mining claims, 618 unpatented lode-mining claims and four additional patented lode-mining claims currently under lease. The total 625 claims cover approximately 12,500 acres and collectively comprise the "Rosebud Mine." Mining activity was completed in July, 2000.

Total mine production through July 2000 averaged 714 tons per day of ore. Ore grades milled were 0.269 gold ounce per ton and 1.08 ounces of silver per ton. The ore produced from the mine was processed in a conventional carbon-in-leach circuit. The mill produced a high quality gold-silver dore. During 2000, 94.6% of the gold and 55.7% of the silver processed at the mill were economically recovered.

Currently, the Rosebud property is being reclaimed per the closure agreement with the Nevada Department of Environmental Protection.

The following table presents information with respect to Hecla's 50% share of production, the average cost per ounce of gold produced and Proven and Probable ore reserves for the Rosebud project as of the dates indicated:

Table 15.1 Rosebud Production
(after Hecla)

	2000	1999	1998	1997
Ore Milled	90,801	140,351	171,493	99,050
Gold Recovered (ounces)	23,926	56,329	65,496	46,974
Silver Recovered (ounces)	55,975	123,953	278,290	168,584

Table 15.2 Rosebud Average Cost per Ounce of Gold Produced
(after Hecla)

	2000	1999	1998	1997
Cash Operating Costs	$290	$184	$157	$137
Total Cash Costs	$301	$199	$176	$156
Total Production Costs	$392	$301	$274	$263

Table 15.3 Rosebud Proven and Probable Ore Reserves (1, 2, 3, 4)
(after Hecla)

	12/31/00	12/31/99	12/31/98	12/31/98
Total Tons	--	107,837	241,927	471,521
Gold (ounces per ton)	--	0.323	0.392	0.420
Silver (ounces per ton)	--	1.23	1.80	2.92
Contained Gold (ounces)	--	34,857	94,808	197,817
Contained Silver (ounces)	--	132,216	436,252	1,378,201

(1) For Proven and Probable ore reserve assumptions, including assumed metals prices, see Glossary of Certain Mining Terms.

(2) Ore reserves were depleted in 2000 due to depletion from mining and poor reserve performance in the North zone orebody. By mid-year, tons and grades of ore remaining were insufficient to sustain the project and thus were no longer termed ore reserves. Mining was completed in July 2000.

(3) The decrease in tons of Proven and Probable ore reserves in 1999 compared to 1998 is primarily attributable to production during 1999, a decrease in dilution applied to the East Zone, reestimation of the North Zone using 89 new drill holes, reestimation of the South Zone using 25 new drill holes and reclassification of reserve blocks that no longer meet Proven and Probable criteria. The decrease in tons of Proven and Probable ore reserves in 1998 compared to 1997 is attributable to production during 1998, reestimation of the East Zone using 108 new drill holes, an increase in cutoff grade from 0.150 oz./ton to 0.180 oz./ton, and reclassification of reserve blocks that no longer meet Proven and Probable criteria.

(4) Ore reserves represent in-place material, diluted and adjusted for expected mining recovery. Mill recoveries are expected to be 95% for gold and 65% for silver. Ore reserve estimates are performed by geostatistical methods in-house, based on drilling, sampling of mine openings and operations experience.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section documents the results of the mineral processing and metallurgical testing technical review of the Hycroft Project and constitutes an Independent Qualified Person's Review and Technical Report. Dr. Deepak Malhotra, President of Resource Development Inc. served as the Qualified Person responsible for the preparation of this section of the Technical Report as defined in National Instrument 43-101 (NUB-101), Standards of Disclosure for Mineral Projects and in compliance with Form 43-101F1 (the "Technical Report"). Information and data for the review and report were obtained from Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI for Vista Gold Corp. in May 2000 and Vista Gold Corp. internal report Brimstone Restart Study, Hycroft Mine, Nevada (June 2000) and other Vista Gold Corp. internal reports. The work entailed a thorough review of these reports and extracting relevant information for the preparation of the Technical Report. These reports are not included as part of this Technical Report due to the large volume of information contained therein.

16.1 Terms of Reference and Disclaimer

The analysis herein is based on the technical information provided in the above referred reports. This Technical Report provides pertinent information contained within the referred documents. The intent of this Technical Report is to provide substantive discussion regarding the processing plans and issues for use by public and private shareholders. Conclusions provided within the body of the above referred technical documents are preliminary in nature and further studies may be needed to confirm the findings.

16.2 Summary

Hycroft is an open pit gold-silver heap leach operation. Several deposits have been mined and placed on four leach pads, the most current being Brimstone (last ore placed on the pad in 1998). Brimstone was the first deposit to be completely placed on the pads as run-of-mine (ROM) ore instead of a combination of crushed and ROM ore (crushing was halted in 1995). Because the mine has not operated since 1998 due to low gold prices and leaching continued on the pads, excellent information on the expected gold recovery for the Brimstone ore is available. Current recovery for Pad 4 with only ROM Brimstone ore stands at 79.5% of cyanide soluble gold (about 57% of total gold by fire assay), the estimated recovery for the remaining ore at Brimstone is 78% of cyanide soluble gold. This value is arrived at from review of historic production results and supported by recent test work.

16.3 Brimstone Processing Facilities

16.3.1 Brimstone Leach Pad

The existing Pad 4 used for Brimstone ore is permitted for 9.1 million square feet to be constructed in phases with an ultimate capacity of 56 million tons of ore stacked to a height of 150 feet. Phase 1 (3 million square feet) currently holds 11.1 million tons of ore and has remaining capacity of 4 million tons.

A pad expansion will be required immediately to accommodate additional ore. The pad expansion may be constructed in phases as needed. The current permit for this pad will need to be modified to allow for higher ore stacking. This should be routinely approved since the original design contemplated stacking to 200 ft and a heap-stability evaluation for that design had been prepared. Run-of-mine ore is placed on leach pads by truck. Pad lifts are approximately 30 feet in height.

Upon placement of the ore on the heaps, the top surface is cross-ripped to enhance permeability following heap leaching by heavy equipment. A network of solution drip lines is laid out at 32” spacing. The ore is irrigated at a rate of 0.0025-0.0030 gpm/ft^2 using a buffered cyanide solution at 0.25 pounds of cyanide per ton of solution. Panels or sections of ore are allowed to leach for a period of 60 to 90 days. Occasionally, as solution management dictates, the surface of the pad is irrigated using Rainbird-type sprays. This type of irrigation generally is done to irrigate irregular surfaces (side-slopes) and to evaporate water. Return solution from the pad containing precious metal values is directed to the pregnant-solution pond.

Figure 16.1 Top of Brimstone Leach Pad

Figure 16.2 Brimstone Leach Pad and Recovery Pipes

16.3.2 Brimstone Plant

The Brimstone facility has a total of four solution-containment ponds. The two primary ponds are the pregnant and barren ponds, which have a capacity of 2.6 million gallons each. The third pond is the low-preg/emergency pond, and has a capacity of 2.8 million gallons. The last pond is the old Lewis-heap pregnant-pond and has a capacity of 4.0 million gallons. Cumulatively, Brimstone has a total of 12.0 million gallons of solution containment. An emergency generator/pump has been installed in the event of a power outage and solution can also be transferred to the Crofoot facilities via an installed 8” pipeline, if necessary.

The solution processing and precious metal recovery facility at Brimstone is a 2,800-gpm Merrill-Crowe zinc-precipitation plant. Pregnant solution is buffered and cyanide is added and then clarified using two 1,600 ft^2 Sparkler filters run in parallel. The clarified solution is de-aerated using a two-stage 75-hp vacuum pump and a packed vacuum tower. Zinc dust is metered into the clarified/de-aerated solution at a rate of 50 to 100 grams per minute using a standard submerged cone. Precipitate containing the precious metals is collected using three 48” recessed-plate filter-presses run in parallel. Collected precipitate is transported to the Crofoot refinery, retorted to remove mercury, and fire refined.

Barren solution from the process plant is contained in the barren pond and is re-circulated to the heap using a 500-hp pump.

Figure 16.3 Brimstone Recovery Plant and Ponds

Figure 16.4 Brimstone Recovery Plant

16.3.3 Brimstone Ore 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 78% represents a realistic estimate of recovery for the remaining ores in the Brimstone pit.

The historic production figures for Pad 4 and Pad 5 are presented in Table 16.1.

Table 16.1 Production Pad Loading and Recoveries

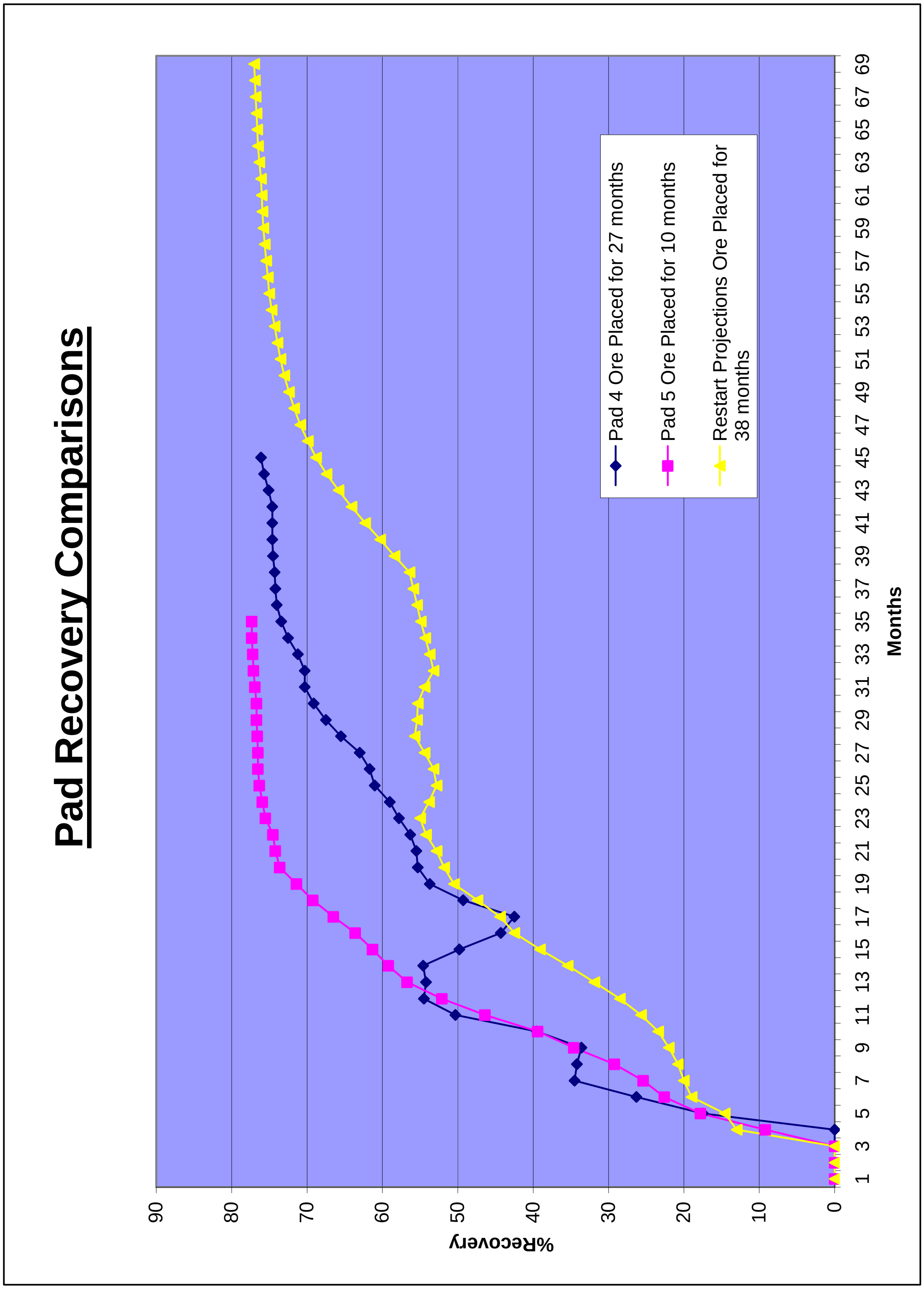
.

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 16.5 shows the historic production results of gold recovery versus time from the ROM heap leaching of the Brimstone deposit from pads 4 & 5. Also shown on Figure 16.5 is the estimated recovery

model showing gold recovery versus time for projected operations at Brimstone. This model shows a slower rate of gold recovery than that experienced for pads 4 & 5 which reflects the reduced mining rate and therefore a slower rate of heap building. Implementation, on restart of operations, of optimization procedures developed during the previous production period and additional testing of solution application rate and lift height optimization may improve future metallurgical performance.

Figure 16.5 Hycroft Leach Pad Recovery (Cyanide Soluble) Comparisons



16.4 Metallurgical Testwork

Several metallurgical studies have been undertaken on the Hycroft ore. These studies 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 orebody. 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 testwork 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 the sustained low gold prices, mining in the Brimstone pit was halted in December 1998 and no further metallurgical testwork 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 orebody, mined to date and future reserves, was undertaken. During the course of this study, all the existing drill hole data was relogged, 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 represent. In addition, areas in the pit where specific ore types are now exposed were identified and new samples were collected for additional testwork

16.4.1 Previous Testwork

A significant amount of testwork 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 are representative.

Gold recoveries achieved from these column/barrel tests were as shown in Table 16.2.

Table 16.2 Column/Barrel Leach 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.

16.4.2 Recent Testwork

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 below:

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 testwork 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.

16.5 Comparison of Previously Mined Ore with Remaining 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

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

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		0.0150	75.7%

These results clearly 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 gold that is soluble in Cyanide 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.

Some of the information contained in this section of this report is taken from an Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada prepared by MRDI. for Vista Gold Corp. in May, 2000. Other information is taken from a Vista Gold Corp. internal report, Brimstone Restart Study, Hycroft Mine, Nevada, June 2000.

17.0 MINERAL RESOURCE ESTIMATE

17.1 Definitions

The resources stated for Hycroft Mine in this report conform to the definitions adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), August 20, 2000, and meet the criteria of those definitions, where:

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

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

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

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

17.2 Resource Estimate Background Information

The resource estimate quoted in this section was completed by ORE. All of the information contained in this section of this report is taken from <u>Brimstone and Boneyard Resource Estimates</u> prepared by ORE for Canyon in June, 2005.

17.2.1 Drill Data

Brimstone drill-hole data were provided by Vista Gold in Ascii format, including collar coordinates, down-hole directional survey, assay data, and geologic codes. The Vista data include 573 holes with 51,874 assay intervals totaling 262,205 feet for Brimstone.

Additional data were provided for the Brimstone area from the Canyon Resources due diligence drilling, including 14 holes with 1,015 assay intervals totaling 5,075 feet. Boneyard drill-hole data include 387 drill holes with 13,747 assay intervals totaling 72,455 feet. Since the Boneyard data were provided using a simple rectangular selection to create a subset of data from the Vista Gold Hycroft drill-hole database, a significant number of the provided drill holes are not actually in the Boneyard deposit but are in adjacent deposits that have largely been mined-out. No attempt was made to further filter these data to create a subset of data that is strictly from the Boneyard deposit.

17.2.2 Assay Corrections

Assay data were not corrected for the Boneyard Model. The Brimstone 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 more than 0.002 opt Au greater than AuFA, both assays were set to "missing".*

- *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.*

- *If the assay interval alteration was from a sulfur-bearing interval that did not also contain sulfides, the AuCN grade was corrected using the factors in Table 17.1.*

- *If the drill hole was a pre-1999 hole, and the alteration type was acid leach, the AuCN and AuFA grades were multiplied by 1.32 to correct for RC drilling bias. If the drill hole was a pre-1999 hole, and the alteration type was oxide ,the AuCN and AuFA grades were multiplied by 1.19 to correct for RC drilling bias.*

Table 17.1 Sulfur Correction Table for Brimstone Assays

		Parameters For High Ratio AuCN ÷ AuFA		Low Ratio Cutoff	Parameters For Low Ratio AuCN ÷ AuFA	
Oxidation	**Sulfur Content**	**"A"**	**"B"**	**oz CN Au/t**	**"A"**	**"B"**
>25 %	Trace	0.3101	0.9247			
>25 %	< 5%	0.2150	0.9561			
>25 %	5 to 10%	0.1947	1.0616	0.11	0.42	1.5283
>25 %	> 10%	0.2106	1.1953	0.90	0.50	1.8917
<25 %	Trace	0.4924	0.9353			
<25 %	>Trace	0.7211	1.1856			

Note: Corrected AuCN = AuFA - (A - ln (AuCn divided by AuFA) +B)

17.2.3 Blasthole Data

Blasthole data including XYZ coordinates and AuCN assays were available from production records for the Brimstone and Boneyard deposits. 71679 blastholes were available for the Brimstone deposit, and 12559 blastholes were available for the Boneyard deposit. No geologic codes or other data were available for the blasthole data, although a geologic interpretation based on pit mapping has been located for the Brimstone deposit but was not available for this study. Blasthole data were used as is except for a few blasts in the Brimstone, for which the bench elevation had to be corrected.

17.2.4 Topographic Data

Topographic data for the Brimstone deposit were extracted from previous MedSystem "VBM" files including current topography (as mined) and original topography (before mining).

The original topography consisted of digitized 20-foot contours up to the 5500 elevation, which left the southeast corner of the model with no topographic coverage. This was resolved by expanding the 5500 elevation upwards at a slope of 28 degrees to the 5960 contour.

The topographic data for the current topography contained no data above 5290 elevation and the southeast corner also was missing data, and in addition, a sliver of data was missing on the east edge of the Brimstone model area. Data was added to the blank areas based on contour data from the edited original topography map.

The only topographic data available for the Boneyard data were from "as-mined" AutoCAD drawings, that show the surface topography for the project at various stages of mine development. There are several problems with this topographic map for purposes of resource estimation and mine planning: First, it does not show the maximum depth of mining, since large portions of the pits were backfilled at the time of topographic mapping; Second, there is no detailed original topography data, although some maps with widely-spaced contour data appear to be available; Third, detailed topographic data did not extend to the north edge of the Boneyard model area..

These problems with the pre-mining and post-mining surfaces were resolved by manually editing the contours from the "as-mined topography" in conjunction with the blasthole data. Pre-mining contours were edited so that they were always above the top of blastholes, and post-mining contours were edited so that they were always below the bottom of blastholes. The missing contour data on the north edge of the model was extracted from a drawing file which contained 25-foot contours over the full Hycroft project area.

17.2.5 Brimstone Geologic Model

A digitized geologic model was extracted from the Vista/MRDI MedSystem "VBM" files. This model is based on closed polygonal outlines defining the major geologic units, as summarized in Table 3-3. These data were sorted so that the polygons with negative areas were ignored and the remaining polygons were arranged in order from largest to smallest area.

Block that had block centers inside each polygon were assigned the geologic code from that polygon. Since some of the polygons were overlapping, a block was ultimately assigned the code of the smallest polygon that contained the block (because the polygons were sorted by descending area). No geologic data were available above the 5290 elevation and those blocks were given a default code of 5 (Footwall Volcanics).

An extra code "6" was present in the geologic outline that defined the high-sulfur zones. These were recoded to code "100" and were used to create a separate "Sulfur Code" model with values of "0" for low sulfur and "100" for high sulfur.

The Vista/MRDI geologic model was edited to correct for slight changes in the geologic contacts, as defined by the Canyon Resources drilling through May 2005. The sulfur zone model was not modified.

17.2.6 Tonnage Factor

Tonnage factors for each geologic zone are based on the Vista June 2000 report, Section 5.2.3. Since the two types of oxide material, clay-bearing and siliceous, have been shown to have significantly different tonnage factors and heap-leach gold recoveries, the geologic model was modified to subdivide the oxide zone into the two types, as discussed later under clay-silica model. Where the clay/silica type could not be determined, the tonnage factor was estimated based on an assumption of 76% siliceous material. Table 17.2 shows the Brimstone tonnage factors. A tonnage factor of 14 ft

Table 17.2 Brimstone Tonnage Factors

Zone Code	Tonnage Factor (ft^3/ton)	Geologic Zone
1	18.00	Alluvium
2	17.50	Acid Leach
3	14.25	Oxide - Unknown Clay/Silica
13	16.00	Oxide - Clay Bearing
23	13.70	Oxide - High Silica
4	13.00	Sulfide
5	14.25	Footwall Volcanics

17.2.7 Drill Hole Compositing

Drill-hole assays were composited using standard, 30-foot bench composites at Brimstone and 20-foot bench composites at Boneyard. The tops and bottoms of benches were used to define the beginning and end of composite intervals. If the down-hole length of the composite was greater than 45 feet at Brimstone or 30 feet at Boneyard, however, the compositing method was changed to down-hole compositing with a fixed length of 30 feet at Brimstone and 20 feet at Boneyard. Missing assays were not used for computation of the length-weighted composite average, but at least one-half of the composite length had to be assayed before a composite value would be saved.

Geologic zone codes were added to the Brimstone composites using the same geologic model polygons 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.*

17.2.8 Blasthole Cyanide Soluble Gold Statistics

Geologic codes from the exploration geologic model were transferred from the Brimstone geologic model polygons to blastholes for compilation of basic statistics, which are summarized in Table 17.3. These statistics show that the majority of blastholes were in the acid-leach and oxide zones, and that those zones, which are the primary mineralized units at Brimstone, have the highest cyanide-soluble gold grades. Coefficients of variation range from about 1.0 to 1.26, which is moderate for Nevada gold deposits. It is noted that even though the AuCN grade in the sulfide zone is nearly as high as the grade in the acid-leach zone, very few blastholes were drilled in sulfides. It is believed that the relatively high AuCN grade in sulfide blastholes is primarily due to misclassification of zone codes because of the relatively coarse resolution of the geologic model, which was created from exploration drill-hole data. In addition, some weakly oxidized material slightly below the oxide/sulfide boundary contains significant AuCN grades.

Table 17.3 Blasthole Cyanide Soluble Gold Statistics

Zone	Geologic Zone	Number	Average oz CN Au/t	Standard Deviation	Coefficient of Variation
1	Alluvium	8,037	0.002	0.0021	1.26
2	Acid Leach	30,925	0.009	0.0103	1.17
3	Oxide	12,424	0.011	0.0121	1.12
4	Sulfide	723	0.008	0.0077	1.00
5	Footwall Volcani	4,499	0.003	0.0034	1.24

Lognormal cumulative frequency plots were compiled to evaluate the blasthole grade distributions. These graphs, which plot as straight lines if the distributions are lognormal, confirm generally lognormal AuCN distributions, particularly for the oxide zone. The acid-leach AuCN distribution is similar to the oxide distribution, but contains significantly more low-grade assays below 0.010 opt AuCN.

The sulfide grade distribution is significantly lower grade than the oxide and acid-leach distribution in all grade ranges, except below 0.005 opt AuCN where the sulfide distribution is slightly higher grade than the acid-leach zone. The AuCN grade distributions for the alluvium and footwall volcanics show that they have less than 3% of their samples above 0.01% AuCN and confirm that the grade distributions are very much lower than the oxide and acid-leach mineralization.

17.2.9 Exploration Drill Hole Statistics

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. Observations from these statistics include:

- *The highest average gold grade is in the oxide unit with an average fire-assay grade of 0.014 opt gold. The acid-leach and sulfide units have similar fire-assay gold grades with average fire-assay grades of 0.008 and 0.009 opt gold, respectively. Gold grade in the alluvium and footwall units is insignificant and fire-assay-gold grade averages 0.002 and 0.003 opt gold in those units.*

- *Cyanide-soluble gold grades follow the same pattern as fire-assay gold, except that cyanide-soluble gold grade is much lower in the sulfide unit because the majority of gold is still locked in sulfides and cannot be leached by the cyanide solution.*

- *The ratio, AuCN ÷ AuFA, is about 74% in the acid-leach and alluvium and 71% in oxide material. Discussions in previous reports suggest that the slightly lower cyanide solubility in the oxide unit is caused by silica encapsulation in high-silica portions of the oxide unit.*

- *The coefficient of variation (CV) for AuCN in exploration drill-hole composites is 0.43 in alluvium compared to 1.26 in blastholes. Over 70% of the alluvium composites have not been assayed, however, and this difference may be the result of preferential sampling of drill-hole intervals that appeared to be mineralized.*

- *The CV for AuCN in oxide zone composites is 1.18, which is similar to a CV of 1.12 in the blastholes. The CV for AuCN in acid-leach zone composites is 1.57, which is significantly higher than the CV of 1.17 in blastholes. The higher CV in the acid-leach zone composites appears to be caused by a significant subpopulation of lower-grade assays, below 0.005 opt AuCN, that is not present in the blasthole assays.*

- *The CV for AuFA in sulfide zone composites is 0.93, which is similar to the 1.00 CV for AuCN in blastholes. The CV for AuCN in sulfide zone composites is 1.58, which is much higher than the CV in blastholes. The higher CV for AuCN appears to be caused by high variability in the degree of oxidation of sulfide samples, and a resulting bimodal distribution of AuCN grade in sulfide samples.*

Table 17.4 Exploration Drill Hole Statistics

		Fire Assay					Cyanide Soluble Assay					
Zone Code	**Geologic Zone**	**No Value**	**Number**	**Average oz Au/t**	**Std Deviation**	**C.V**	**No Value**	**Number**	**Average oz Au/t**	**Std. Deviation**	**C.V.**	**Ratio Cyanide/Fire**
1	Alluvium	711	295	0.0015	0.0008	0.547	714	292	0.0011	0.0005	0.433	0.738
2	Acid-Leach	293	3,462	0.0081	0.0125	1.548	347	3,408	0.006	0.0094	1.571	0.739
3	Oxide	121	1,888	0.0137	0.0152	1.114	148	1,861	0.0097	0.0114	1.179	0.707
4	Sulfide	149	1,340	0.009	0.0083	0.926	172	1,317	0.0023	0.0035	1.537	0.251
5	Footwall	62	395	0.0031	0.0033	1.064	67	390	0.0017	0.0012	0.697	0.525

Lognormal cumulative frequency plots were compiled by geologic unit to further evaluate the gold grade distributions in drill-hole composites. The plots indicate nearly lognormal distributions at the higher grade ends of the curves as shown by the approximately straight lines at the upper ends of the curves. The lower ends of the plots curve downward, which indicates an excess of lower-grade material compared to a simple lognormal population. The higher-grade end of the acid-leach curve is similar to the oxide curve, but the acid-leach zone contains a much larger percentage of the lower-grade material *and the curves diverge below 0.01 opt AuCN. The sulfide curve has almost the same shape as the oxide curve, but the grades are only about 20% of the oxide zone, so the curve is shifted downward.*

The cumulative frequency plots of fire-assay gold grade are similar to the cyanide-soluble plots except that the sulfide curve has been shifted upwards and is much closer to the acid-leach and oxide curves. The upper end of the oxide curve is still about 30% higher grade than the sulfide curve, however, which may indicate a slight enhancement of gold grade due to enrichment during oxidation and/or loss of density.

It was observed previously that the distribution of gold grade in the acid-leach and oxide units were similar, but that the acid-leach unit contained a larger fraction of lower-grade material. Observations of the distribution of gold grades on plan maps show that the lower-grade and higher-grade portions in the two units are well-defined as distinct populations and some type of grade zoning is indicated for the resource model.

A quick check of ratio statistics showed that nearly 30% of sulfide composites had ratios above 50%. Accordingly, a thorough review of the geologic coding is recommended to ensure that oxide samples have not been classified as sulfides, and vice versa.

A few (about 1% of the total) composites are observed in these graphs with higher AuCN grades than AuFA grades. The source of this anomaly was investigated and was found to have two sources: First, the AuCN assay was missing in some samples that had low AuFA assays. When these were composited, the resulting AuCN composite grade was sometimes higher than the AuFA grade. Second, the assays from the May 2005 drilling were used without any adjustments, and some of the AuCN assays were higher than AuFA assays. Only 15 of the anomalous composites were above 0.005 opt Au, however, and this is not considered a significant problem.

17.2.10 Boneyard Statistics

Basic statistics for blasthole and 20-foot high bench composites of exploration data are summarized in Table 17.5. These statistics indicate average gold grades similar to Brimstone but slightly lower variability.

Considering that the average AuCN grade of blastholes is 44% higher grade than the average AuCN grade, it is tempting to look for a bias in the exploration data. Most of this difference is a high-grading bias, however, since the area mined was selectively higher grade than the entire deposit. Thus, the blastholes are taken from the higher-grade part of the deposit and are higher grade than the exploration holes, which cover the entire deposit including the lower grade extension to the north. When exploration composites are compared in the same area as blastholes, however, the blastholes are still 10% higher grade and a slight bias may be present. It is also noted that the average AuCN÷AuFA ratio is only 59% for the Boneyard deposit, compared to 70%, or better, for acid-leach and oxide material at Brimstone, so the AuCN grade may be biased low by the inclusion of sulfide material in the overall population.

Table 17.5 Boneyard Statistics

Assay	Data Type	Number Missing Assays	Number = 0.001	Number > 0.001	Average oz Au/t	Standard Deviation	Coefficient Variation
AuCN	Blastholes	0	842	11,406	0.013	0.0117	0.90
AuFA	Exploration Holes	140	152	895	0.012	0.0100	0.84
AuCN	Exploration Holes	165	351	671	0.009	0.0085	0.95

17.2.11 Variograms

Brimstone Blastholes

Variograms were computed for blastholes using log-transformed AuCN grades and only those samples above a minimum grade of 0.0035 opt AuCN in the oxide and acid leach zones. The minimum grade limit was used so the data were more nearly lognormal and to confine the variogram to the mineralized grade zone. The log-transformed variograms were converted to relative variograms for analysis and modeling. The first variograms were computed in the horizontal plane at 10-degree angular increments. These variograms indicate a strong anisotropy with best continuity in the north-northeasterly direction.

Variograms were then computed for the primary axes which were determined by inspection of the individual variograms at 10° increments to be 20° azimuth for the major axis, 110° for the secondary axis, and vertically for the tertiary axis. These variograms indicate a maximum range of 600 feet at an azimuth of 20°. Major axis range is almost two times the range at 110° azimuth and is about three times the range in the vertical direction. The nugget effect is 0.126, or less than 25% of the sill value, which is low relative to most gold deposits which have nugget effects that are between 40% to 70% of sill.

Brimstone Drill Hole Composites

Variograms were computed for composited, adjusted AuCN grades (with a minimum value of 0.0035 opt AuCN) from the exploration drill-holes for comparison with the blasthole variograms. These variograms are very similar to the blasthole variograms except that the variogram sill is slightly higher (0.601 vs 0.527), the nugget effect is slightly higher, the vertical variogram has a shorter range (140 feet vs 200 feet), and the horizontal directions appear to be nearly isotropic. The higher variance and nugget effect may be related to additional sampling variance from the RC drilling, but may also be a minor difference related to sampling different volumes. The shorter vertical variogram range from the exploration drill holes may be a better estimate of the true vertical range, since the angular search for the blasthole variogram was almost 40-feet square at the 200 feet maximum distance of the blasthole variograms. Thus, the blasthole vertical variogram contains a significant horizontal component, which is not present in the exploration drill-hole variogram. The nearly isotropic variogram at 110° azimuth is not easily explained, but may be related to a less well-defined anisotropy in the unmined area south of the Brimstone open pit.

Boneyard

The log-transformed variograms were converted to relative variograms for analysis and modeling. The first variograms were computed in the horizontal plane at 10-degree angular increments. These variograms indicate a strong anisotropy with best continuity in the north-northeasterly direction. The strongest continuity for these variograms is towards the north-northeast, much the same as at Brimstone, but with a much stronger (10:1) anisotropy perpendicular to the main axis compared to Brimstone (2:1).

Variograms were then computed for the primary axes at 13° azimuth for the major axis, 103° for the secondary axis, and vertically for the tertiary axis. These variograms indicate a maximum range of 900 feet at an azimuth of 13°. Major axis range is over 10 times the range at 103° azimuth and is about 3.6 times the range in the vertical direction, confirming the narrow, structurally controlled shape of the deposit. The nugget effect is 0.20, or about 25% of the sill value.

17.3 Resource Estimate

17.3.1 Resource Model Dimensions

The gold resource was estimated using two block models, each of which is oriented parallel to the mine survey grid. Model dimensions and sizes are summarized in Table 17.6.

Table 17.6 Resource Model Dimensions

	Brimstone			Boneyard		
Item	North (Rows)	East (Conlums)	Elevation (Levels)	North (Rows)	East (Conlums)	Elevation (Levels)
Maximum	46,000	25,100	5,980	52,500	21,425	4,800
Minimum	39,400	20,900	4,180	47,500	19,800	4,400
Number Blocks	264	168	60	200	65	20
Block Size	25	25	30	25	25	20

17.3.2 Brimstone Grade Model

17.3.2.1 Blasthole Grade Model

Blasthole AuCN grade was modeled to the same 25x25x30-foot block configuration as the resource model using ordinary kriging. The search parameters were varied according to the blasthole spacing so that gaps between blasthole patterns could be estimated without using a search that is too large in the well drilled areas. The 25x25x30-foot block is believed to be close to the selective mining unit (SMU) if the deposit is mined using equipment similar in size to the previous operation. Use of larger equipment, or wider blasthole spacing may increase the size of the SMU and add mining dilution. The procedure for the blasthole model was as follows:

- *Blasthole spacing was estimated based on the kriging variances from pointkriging. (To prevent problems from estimation of negative grades a flag variable was kriged that was set equal to 1* *for blastholes with any AuCN grade.) A zeronugget, linear variogram with a slope of ½ was used for kriging with a search radius of 45 feet and an isotropic search ellipse. A minimum of 1 point (blasthole) maximum of 16 points were used for estimation.*

- *Blasthole grades were kriged to blocks using a minimum of one and a maximum of 16 blastholes and a 6x6 grid for estimation of sample - block covariances. Search radii are summarized in Table 17.7. Variogram parameters are summarized in Table 17.8.*

Table 17.7 Blasthole Grade Model Search Distances

Blasthole Grid Code	Kriging Variance	Maximum Hole Grid	Horizontal Radius (ft)	Vertical Radius (ft)
1	5	18	20	15
2	10	36	30	15
3	15	54	45	45
4	20	71	Not Estimated	
99	>20	>71	Not Estimated	

Table 17.8 Blasthole Grade Model Variogram Parameters

Structure	Type	Value	Range		
			Major Axis	Secondary Axis	Tertiary Axis
0	Nugget	0.22			
1	Spherical	0.18	110	70	65
2	Spherical	0.127	900	350	225

Note: Major Axis is at 20^{o} Azimuth

17.3.2.2 Brimstone Grade Zones

After the low-grade and mineralized populations identified by the basic statistical analysis were confirmed as continuous features on plan-maps of NN models of gold grade, use of grade zones was indicated for resource estimation. Since there was insufficient time for manual grade zoning on plans or sections, a simple grade zone model was constructed using nearest-neighbor assignment and the adjusted AuCN composite grades as follows.

- *A composite search ellipse with radii of 250x175-feet horizontally and 100-feet vertically was used to find the nearest composite to each block;*
- *North of 48,850N (rows 115 to 264) the long axis of the search was oriented at an azimuth of 20°. South of 48,850N (rows 1 to 114) the long axis was oriented at 35°;*
- *Blocks were assigned NN grades using only composites from the same geologic unit, except for the acid-leach and oxide units, which were combined;*

- *Blocks with NN grades below 0.005 opt AuCN were assigned a grade-zone code of 10. Blocks above 0.005 opt AuCN were assigned a grade-zone code of 20. Blocks with no NN grade were assigned a grade-zone code of 900; and*

- *Grade-zone codes were added to the geologic unit code, thus 12 is lower-grade acid-leach, 23 is higher-grade oxide, and so on*

Grade zones were created for fire-assay gold using the above procedure, but with a grade threshold of 0.0075 opt AuFA instead of 0.005 opt AuCN.

17.3.2.3 Gold Grade Estimate

Gold grade was estimated using inverse-distance estimation but with gold grade selection ranges and capping parameters varying according to the grade zone that was estimated. The general procedure for creation of the gold-grade model was as follows:

- *A 250x175x100-foot radius search ellipse was used for the sample search. A maximum of nine points were used for estimation. All blocks were estimated with at least one point inside the search ellipse;*

- *The major axis of the search ellipse was oriented at 35° azimuth south of 48,850N and 20° azimuth north of 48,850N;*

- *The power was varied until the variance of IDP estimates was about 70% of the variance of nearest NN estimates for the same blocks. The 70% variance ratio was estimated based on the blasthole variogram;*

- *The composite grade-selection ranges and capping parameters were adjusted for each zone until the distribution of the estimated blocks matched the distribution of the kriged blasthole block grades; and*

- *Fire assay gold grades were estimated using the same procedure, but with slightly higher grade ranges. Fire assay estimation was not thoroughly optimized because all grade control and mine planning is done with cyanide-soluble assays.*

17.3.2.4 Comparison of Blasthole Model Grade to Grade Model

The blasthole grade distribution compared to the model grade distribution caused the smoothing factors to increase slightly in the higher grade zones and significantly in the lower-grade zones. In addition, the IDP grades were slightly higher than NN grades in the low-grade zones and slightly lower than NN grades in the high-grade zones. Overall, the difference between average IDP and NN grades is less than 2%, so no overall bias is indicated.

The same parameters were used for all other zones as were used for the acid-leach and oxide zones, since the acid-leach and oxide zones contain most of the resource.

The tonnage-grade distributions of the blasthole model and the IDP resource model were compared using only those blocks that were estimated in both models and were inside the existing open-pit. The results of this comparison, which are shown in Table 17.9, show that the IDP resource model provides a very close approximation to the tonnage-grade distribution of the blasthole model. Tonnage is estimated within 3.2% accuracy with cutoffs ranging from 0.003 opt AuCN to 0.018 opt AuCN; tonnage tends to be underestimated slightly. Cutoffs grades above 0.02 opt AuCN have very small tonnages, however, and small errors in the estimated tonnage cause large differences in the percentage error. Grade estimation accuracy is slightly better than tonnage on an overall basis, amounting to an approximate 2% underestimation for most cutoff grades. Because both tonnage and grade tend to be slightly underestimated, contained ounces of gold tend to be underestimated by slightly larger amounts, but are still better than 2% in the critical range.

This comparison shows that the resource model is slightly conservative relative to the actual resource and is about 2.0% less than actual contained ounces AuCN for cutoff grades that would be used for mining. This comparison uses all estimated blocks regardless of resource classification, however, and about 4% of the tonnage in the comparison is in the inferred resource class and would not be included in the measured and indicated resource. If the inferred resource is removed from the comparison, the resource model is about 3% to 4% more conservative, which provides a small safety factor in the model.

Table 17.9 Comparison of Cyanide Soluble Grade of Blasthole Model to Grade Model

	Blasthole Model			Grade Model			Difference to Blasthole Model		
Cutoff oz Au/t	Tons 000's	Grade oz Au/t	Ounces Au	Tons 000's	Grade oz Au/t	Ounces Au	Tons	Grade	Ounces
0.050	89	0.069	6.1	90	0.060	5.4	1.10%	-12.50%	-11.50%
0.040	253	0.053	13.3	225	0.050	11.3	-11.10%	-4.40%	-15.00%
0.030	841	0.039	33.1	707	0.040	27.9	-15.90%	0.30%	-15.70%
0.020	3,039	0.028	85.4	2,854	0.028	78.5	-6.10%	-2.10%	-8.10%
0.018	4,018	0.026	104.1	3,900	0.025	98.3	-2.90%	-2.70%	-5.60%
0.016	5,299	0.024	125.6	5,244	0.023	121.1	-1.00%	-2.50%	-3.50%
0.014	7,043	0.022	152.1	6,817	0.021	144.5	-3.20%	-1.90%	-5.00%
0.013	8,106	0.021	166.2	7,861	0.020	158.8	-3.00%	-1.50%	-4.40%
0.012	9,214	0.020	179.7	9,063	0.019	174.0	-1.60%	-1.50%	-3.20%
0.011	10,393	0.019	193.3	10,332	0.018	188.0	-0.60%	-2.20%	-2.70%
0.010	11,627	0.018	207.0	11,521	0.017	200.5	-0.90%	-2.20%	-3.10%
0.009	12,977	0.017	219.3	12,745	0.017	212.8	-1.80%	-1.20%	-3.00%
0.008	14,392	0.016	231.7	14,336	0.016	226.5	-0.40%	-1.90%	-2.20%
0.007	15,909	0.015	241.8	15,848	0.015	237.7	-0.40%	-1.30%	-1.70%
0.006	17,548	0.014	252.7	17,547	0.014	249.2	0.00%	-1.40%	-1.40%
0.005	19,355	0.014	263.2	19,343	0.013	259.2	-0.10%	-1.50%	-1.50%
0.004	21,461	0.013	272.6	21,495	0.013	268.7	0.20%	-1.60%	-1.40%
0.003	24,126	0.012	282.3	24,550	0.011	277.4	1.80%	-3.40%	-1.70%
0.002	28,207	0.010	290.5	29,178	0.010	288.9	3.40%	-3.90%	-0.60%
TOTAL	46,131	0.007	304.5	46,131	0.007	309.1	0.00%	1.50%	1.50%

17.3.2.5 Silica Clay Model

The silica-clay model was created after it was determined that the high-silica and high clay portions of the oxide zone had different tonnage factors and different leach recoveries. This model was created as follows:

- *The "ALTER", alteration field in the drill-hole data was recoded so the code "2" was set equal to 1 and all other codes were set equal to 0. Thus, the recoded value for ALTER is a 0,1 flag where "1" is siliceous and "0" is anything else;*
- *The recoded ALTER field was composited;*
- *A nearest-neighbor silica/clay model was constructed using the composited ALTER values. A search ellipse with 500x250x200-feet search radii was used for this model. The major axis of the search axis was oriented at 35° azimuth in the south part of the deposit and 20° in the north of the deposit. The composite geologic unit was strictly matched to the block geologic unit for this model.*
- *The clay/silica model was then coded into integer values for use as a geologic code. If the geologic unit was anything but oxide, the clay/silica code was set to zero. If the geologic code was oxide and the no value was estimated in the NN model, the clay/silica code was set to zero. Otherwise, the clay/silica code was set to 10 if the NN model value was less than 0.5, and the clay/silica code was set to 20 if the NN model value was greater than 0.5*

The final clay/silica model has 13% of the oxide unit blocks with undefined clay/silica type, 24% with clay-rich type, and 63% with silica-rich type.

17.3.2.6 Resource Classification

Resource classes were based on the drill-hole grid using categories derived from visual inspection of plan maps of the blasthole model and resource models. On these maps, it was observed that the +0.006 opt AuCN zones were very continuous along strike and were generally between 300-feet and 600-feet wide. Thus, a minimum 200-foot drillhole spacing provides two to three drill-hole intersections across the deposit, and will define indicated resource. Measured resource was defined as a 100-foot drill-hole grid, which is sufficient to define the smaller ore zones and included waste zones, and provides more accurate resource estimates for detailed mine planning. Approximately 96% of the mined-out pit is classified as "measured and indicated" using these parameters. Indicated resource was defined as those blocks with greater than 200-feet drill-hole spacing but less than 300-foot spacing. In addition to the area inside the drilling grid, a maximum extrapolation of approximately 28% of the maximum grid size was allowed for each resource class.

Drill-hole spacing was estimated based on the kriging variances from point-kriging a flag variable that was set equal to 1 for composites that had a non-missing AuCN grade. A zero-nugget, linear variogram with a slope of ½ was used for kriging. Data selection used a search ellipse with 400x400x75-foot radii, *a maximum of 12 composites, and no more than 1 composite from any drill hole. The kriging variances were converted to grid-spacing codes as summarized in Table 5-8.*

Table 17.10 Brimstone Oxide Resource Classification vs Drill Hole Grid

Resource Code	Resource Class	Max. Kriging Variance	Max. Hole Grid
1	Measured	28	100
2	Indicated	42	150
3	Indicated	52	200
4	Inferred	84	300
99	No Class	>84	>300

The sulfide resource is classified entirely as inferred because mining cutoffs would likely be over 0.02 opt AuFA, at which point the model is not as reliable as it is at lower cutoffs. In addition, the mineralized envelopes are much smaller and irregular at the higher cutoffs, so continuity is difficult to establish.

17.4 Brimstone Resource Summary

The remaining measured and indicated resource is summarized in Table 17.11. The total oxidized, inferred resource is summarized in Table 17.12. The total sulfide resource, all of which is classified as inferred, is summarized in Table 17.13.

Table 17.11 Brimstone Measured and Indicated Acid Leach + Oxide Resources

	Measured				Indicated				Measured + Indicated				
Cutoff oz CNAu/t	Tons 000,000's	oz CN Au/t	000's Ounces AuCN	oz Au/t	Tons 000,000's	oz CN Au/t	000's Ounces AuCN	oz Au/t	Tons 000,000's	oz CN Au/t	000's Ounces AuCN	oz Au/t	000's Contained Au Ounces
0.040	0.8	0.058	47	0.075	0.7	0.053	37	0.065	1.5	0.056	84	0.071	107
0.030	1.4	0.048	68	0.063	1.7	0.042	69	0.052	3.1	0.045	138	0.057	177
0.025	2.2	0.040	90	0.053	2.9	0.036	102	0.045	5.1	0.038	192	0.048	245
0.020	3.4	0.034	116	0.045	5.0	0.030	150	0.038	8.4	0.032	266	0.041	344
0.015	5.2	0.028	147	0.037	8.7	0.024	213	0.031	13.9	0.026	360	0.034	473
0.014	5.7	0.027	154	0.036	9.9	0.023	231	0.030	15.6	0.025	385	0.032	499
0.013	6.3	0.026	162	0.034	11.4	0.022	251	0.029	17.7	0.023	413	0.030	531
0.012	7.2	0.024	173	0.032	13.0	0.021	271	0.027	20.2	0.022	445	0.029	586
0.011	8.0	0.023	183	0.030	15.1	0.020	295	0.026	23.2	0.021	479	0.027	626
0.010	9.2	0.021	195	0.029	17.9	0.018	324	0.024	27.1	0.019	520	0.026	705
0.009	10.4	0.020	207	0.027	20.6	0.017	351	0.023	31.0	0.018	558	0.024	744
0.008	11.9	0.018	219	0.025	23.6	0.016	375	0.022	35.4	0.017	595	0.023	814
0.007	13.4	0.017	231	0.024	27.2	0.015	403	0.020	40.6	0.016	634	0.021	853
0.006	15.3	0.016	243	0.022	31.5	0.014	431	0.019	46.8	0.014	674	0.020	936
0.005	**17.2**	**0.015**	**254**	**0.020**	**35.5**	**0.013**	**453**	**0.018**	**52.7**	**0.013**	**707**	**0.019**	**1,001**
0.004	19.5	0.014	263	0.019	40.0	0.012	473	0.016	59.5	0.012	736	0.017	1,012
0.003	22.5	0.012	275	0.017	46.4	0.011	496	0.015	68.9	0.011	771	0.016	1,102
0.002	27.1	0.011	286	0.015	56.2	0.009	517	0.013	83.3	0.010	804	0.014	1,166
0.001	37.0	0.008	301	0.011	78.2	0.007	551	0.010	115.2	0.007	852	0.010	1,152
0.000	40.8	0.007	304	0.011	87.4	0.006	556	0.009	128.1	0.007	860	0.010	1,281

Note: Brimstone Resources are reported at a 0.005 cyanide soluble oz Au/ton cutoff grade.

Table 17.12 Brimstone Inferred Acid Leach + Oxide Resources

	Inferred Acid Leach				Inferred Oxide				Total Inferred				
Cutoff oz CNAu/t	Tons 000,000's	oz CN Au/t	000's Ounces AuCN	oz Au/t	Tons 000,000's	oz CN Au/t	000's Ounces AuCN	oz Au/t	Tons 000,000's	oz CN Au/t	000's Ounces AuCN	oz Au/t	000's Contained Au Ounces
0.040	0.0	0.065	1	0.091	0.1	0.046	3	0.055	0.1	0.049	4	0.059	6
0.030	0.0	0.041	1	0.050	0.2	0.038	7	0.044	0.2	0.038	8	0.045	9
0.025	0.1	0.030	4	0.040	0.2	0.035	8	0.042	0.4	0.033	13	0.041	16
0.020	0.2	0.027	6	0.037	0.5	0.028	13	0.034	0.7	0.028	19	0.035	25
0.015	0.3	0.023	8	0.031	0.9	0.023	21	0.027	1.3	0.023	29	0.028	36
0.014	0.4	0.022	8	0.030	1.1	0.022	23	0.026	1.4	0.022	31	0.027	38
0.013	0.4	0.022	8	0.029	1.2	0.021	25	0.025	1.6	0.021	34	0.026	42
0.012	0.5	0.020	10	0.027	1.5	0.019	29	0.024	2.0	0.019	38	0.025	50
0.011	0.7	0.018	12	0.024	1.9	0.018	33	0.022	2.6	0.018	45	0.023	60
0.010	0.9	0.016	14	0.022	2.4	0.016	39	0.020	3.3	0.016	53	0.021	69
0.009	1.1	0.015	16	0.020	3.0	0.015	44	0.019	4.2	0.015	61	0.019	80
0.008	1.3	0.014	18	0.019	3.7	0.014	50	0.018	5.0	0.014	68	0.018	90
0.007	1.5	0.013	20	0.018	4.4	0.013	55	0.017	6.0	0.013	75	0.017	102
0.006	1.9	0.012	22	0.016	5.4	0.011	62	0.016	7.2	0.012	84	0.016	115
0.005	**2.5**	**0.010**	**25**	**0.014**	**6.3**	**0.011**	**67**	**0.015**	**8.7**	**0.011**	**92**	**0.015**	**131**
0.004	3.2	0.009	28	0.012	7.0	0.010	70	0.014	10.2	0.010	98	0.013	133
0.003	4.6	0.007	33	0.010	8.2	0.009	74	0.013	12.8	0.008	107	0.012	154
0.002	7.3	0.005	40	0.008	9.6	0.008	77	0.012	16.9	0.007	117	0.010	169
0.001	16.5	0.003	52	0.004	12.0	0.007	81	0.010	28.5	0.005	133	0.007	200
0.000	21.6	0.003	55	0.004	13.7	0.006	82	0.009	35.3	0.004	137	0.006	212

Table 17.13 Brimstone Inferred Sulfide Resources

Cutoff oz Au/t	Tons 000,000's	oz Au/t	000's Contained Au Ounces	oz CN Au/t	000's Ounces AuCN
0.050	1.3	0.059	74	0.014	18
0.045	1.3	0.059	77	0.014	18
0.040	1.4	0.057	82	0.014	20
0.036	1.6	0.055	89	0.015	24
0.034	1.8	0.053	95	0.014	25
0.032	2.2	0.049	108	0.012	26
0.030	2.9	0.045	131	0.012	35
0.028	3.9	0.041	159	0.010	39
0.026	4.7	0.038	181	0.009	42
0.022	9.5	0.031	293	0.007	67
0.020	**13.5**	**0.028**	**379**	**0.006**	**81**
0.018	17.0	0.026	443	0.006	102
0.016	23.3	0.024	551	0.005	117
0.014	30.3	0.022	654	0.005	152
0.012	39.5	0.020	775	0.004	158
0.010	55.1	0.017	942	0.004	220
0.000	171.9	0.009	1548	0.002	344

17.5 Boneyard Resource Model

17.5.1 Boneyard Blasthole Model

The blasthole AuCN model was created using block kriging with a square search pattern measuring 30x30x20-feet, a maximum of 11 blastholes and a minimum of one blasthole. The variogram was a set of two nested spherical structures plus a nugget effect.

17.5.2 Boneyard Gold Grade Estimate

The gold resource for Boneyard was estimated using simple nearest-neighbor estimation with a search ellipse of 300x50x10-feet radius that was oriented with the long axis at 13° azimuth. No further estimation methods were tried after it was shown that the NN estimate provides excellent results for cutoff grades below 0.007 opt AuCN, as shown in Table 17.14. For cutoffs of 0.007 opt AuCN and above, the NN model does a very good job of predicting contained ounces of gold, but tonnage and grade must be corrected by addition of a dilution factor of 8% to 10% tonnage at zero grade.

Table 17.14 Comparison of NN AuCN Resource Model to Kriged Blasthole Model

	Blasthole Model			Grade Model (Nearest Neighbor)			Difference to Blasthole Model		
Cutoff oz Au/t	Tons 000's	Grade oz Au/t	Ounces Au	Tons 000's	Grade oz Au/t	Ounces Au	Tons	Grade	Ounces
0.020	562	0.027	15.3	655	0.029	19.1	17%	7%	25%
0.015	1,085	0.022	24.3	945	0.026	24.1	-13%	14%	-1%
0.014	1,212	0.022	26.2	1092	0.024	26.2	-10%	11%	0%
0.013	1,342	0.021	27.9	1193	0.023	27.6	-11%	11%	-1%
0.012	1,489	0.020	29.8	1292	0.022	28.8	-13%	12%	-3%
0.011	1,629	0.019	31.3	1429	0.021	30.4	-12%	11%	-3%
0.010	1,824	0.018	33.4	1581	0.020	31.9	-13%	10%	-4%
0.009	1,997	0.018	35.1	1812	0.019	34.2	-9%	7%	-3%
0.008	2,136	0.017	36.3	1987	0.018	35.6	-7%	5%	-2%
0.007	2,292	0.016	37.4	2107	0.017	36.5	-8%	6%	-2%
0.006	2,448	0.016	38.4	2481	0.016	39.0	1%	0%	1%
0.005	2,607	0.015	39.4	2622	0.015	39.9	1%	1%	1%
0.004	2,762	0.015	40.0	2753	0.015	40.5	0%	1%	1%
0.003	2,888	0.014	40.4	2,879	0.014	40.9	0%	1%	1%
0.002	2,984	0.014	40.6	2,977	0.014	41.1	0%	1%	1%
TOTAL	4,002	0.010	40.8	4,002	0.010	41.6	0%	2%	2%

17.5.3 Boneyard Resource Classification

The Boneyard resource was classified as "indicated" if the drill-hole grid was smaller than a 250x125-foot grid. The 250-foot dimension is parallel to the strike of the deposit and the 125-foot dimension is perpendicular to strike. Blocks inside wider-spaced drilling were classified as "inferred".

17.5.4 Boneyard Resource Estimate

The Boneyard resource is summarized in Table 17.15.

Table 17.15 Boneyard Resource

	Indicated					Inferred				
Cutoff oz CNAu/t	**Tons 000,000's**	**oz CN Au/t**	**000's Ounces AuCN**	**oz Au/t**	**000's Contained Au Ounces**	**Tons 000,000's**	**oz CN Au/t**	**000's Ounces AuCN**	**oz Au/t**	**000's Contained Au Ounces**
0.015	0.09	0.026	2.2	0.039	3.5	0.05	0.021	1.0	0.029	1.5
0.014	0.09	0.026	2.3	0.039	3.5	0.05	0.021	1.0	0.029	1.5
0.013	0.09	0.026	2.3	0.039	3.5	0.05	0.021	1.1	0.029	1.5
0.012	0.14	0.021	2.9	0.035	4.9	0.05	0.021	1.1	0.029	1.5
0.011	0.18	0.019	3.3	0.031	5.6	0.12	0.015	1.8	0.022	2.6
0.010	0.18	0.019	3.4	0.031	5.6	0.15	0.015	2.1	0.023	3.5
0.009	0.21	0.017	3.7	0.030	6.3	0.20	0.013	2.6	0.022	4.4
0.008	0.24	0.016	3.9	0.030	7.2	0.20	0.013	2.7	0.022	4.4
0.007	0.26	0.016	4.1	0.029	7.5	0.20	0.013	2.7	0.022	4.4
0.006	0.30	0.014	4.3	0.027	8.1	0.23	0.012	2.8	0.021	4.8
0.005	**0.39**	**0.012**	**4.8**	**0.025**	**9.8**	**0.29**	**0.011**	**3.1**	**0.020**	**5.8**
0.004	0.44	0.012	5.0	0.024	10.6	0.31	0.011	3.2	0.020	6.2
0.003	0.53	0.010	5.4	0.022	11.7	0.44	0.008	3.7	0.017	7.5
0.002	0.73	0.008	5.8	0.020	14.6	0.46	0.008	3.7	0.017	7.8
0.001	1.36	0.005	6.5	0.015	20.4	1.32	0.004	4.6	0.010	13.2
0.000	1.50	0.005	6.8	0.014	21.0	1.42	0.003	4.7	0.009	12.8

18.0 MINERAL RESERVE ESTIMATE

Mineral reserves for the Hycroft Mine were developed by applying relevant economic criteria in order to define the economically extractable portions of the resource model. The Boneyard deposit was considered too small at this time to be considered for mining. MDA developed the reserves for Hycroft to meet the NI 43-101 standards set for mineral reserves. The NI 43-101 standard uses the Canadian Institute of Mining, Metallurgy and Petroleum reserve definitions, which are:

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

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

The economic and design criteria used in determining the reserves in this report were derived from feasibility level economic studies completed by MDA in January 2006. Vista completed an internal feasibility study of restarting the Brimstone project in June 2000. MDA updated that study during 2004. MDA believes that there is enough information in both studies concerning the appropriate mining, processing, economic and other factors to support Proven and Probable reserves. A nominal mining rate of 2 million tons per month was used to plan the mine.

In the economic calculations, all costs for mining and processing were paid by both gold and silver revenues.

18.1 Applied Methodologies

The Brimstone reserves were derived from the resource model built by ORE. MDA used Medsystem/MineSight computer software to develop and report the reserves using the following procedure:

1. Update and adjust inputs as needed, then use inputs to generate multiple "pit shells" with Medsystem's Lerchs-Grossmann ultimate pit program;
2. Design an ultimate pit using the pit shells as guides. This design includes haul roads and eliminates any areas that could not be mined because of practical mining limitations; and
3. Tabulate Measured and Indicated resources inside the designed pit that meet the economic criteria for reserve classification and reporting.

18.2 Pit Design Parameters

Economic inputs used to develop the Lerchs-Grossmann pit shells and cutoff grades are listed in Table 18.1. Table 18.2 is a summary of the physical pit design parameters that were used in the ultimate pit program.

Table 18.1 Economic Parameters

Value	Description	Units (US$)
$450	Gold price	$/oz
$6	Silver price	$/oz
$0.80	Cost of mining	$/ton
$1.08	Cost of processing	$/ton ore
$0.22	Cost of administration, Jungo road, environmental, reclamation	$/ton ore
78%	CN gold recovery	%
3	Recovered oz silver per oz gold	oz

Table 18.2 Lerchs-Grossman Pit Design Parameters

Description	Value
Slope Angle	48 degrees
Bench Height	30 feet
Road Width	90 feet
Maximum Ramp Grade	10%
Minimum Mining Width	90 feet
Tonnage Factor (varies by rock type)	13-18

The ultimate economic pit was generally designed on the floating cones generated by the $450 gold prices. However, special consideration is given to the highwall stability near the East fault. As much as possible the highwall will be constructed into the competent footwall ground. However, to avoid additional stripping costs, the wall will move forward into the weaker hanging wall and the East fault splay in the lower levels of the pit. This design should limit the potential wall instability over most of the life of the mine. The limited height of the weaker materials should permit good ore recovery in those areas utilizing an aggressive exposure control program to limit mining risks.

The East fault zone is the primary geotechnical consideration of the pit design. Using the January 29, 1997 Call & Nicholas, Inc. Memorandum of the "Hycroft Crofoot & Lewis Mine: Brimstone East Wall Stability Study" as a guide, Table 18.3 shows the general guidelines for the pit design parameters.

Table 18.3 Economic Pit - East Wall Parameters

Description	Value
Interramp Slope Angle	50 degrees
Bench Height	30 feet
Triple Bench Design allows safety catch benches every third bench	12 – 24 feet
Assumed Bench Face Angle	55-60 degrees
Road Width	90 feet
Maximum Ramp Grade	10%
Minimum Mining Width	90 feet

The assumed bench face angles are shallow and are reflective of those mentioned in the Call and Nicholas report for areas near the East fault. Good wall control blasting practices should achieve a much steeper angle and give wider safety catch benches. However, the design will show the catch benches at 24 ft and assume a face angle of 60 degrees.

Wherever possible, interim roads not included as part of the final pit design, will be designed at 8% and may be wider than 90 ft to promote faster truck haulage times.

18.3 Dilution

MDA believes that the combination of inverse distance weighting, model block size and averaging of grades across multiple zones adequately accounts for dilution in the reserve. However, it will be necessary to practice grade control, specifically at material boundaries. Another area of concern will be the internal waste. Grade control will have to be practiced to avoid contaminating the ore with non-economic mineralized material. As in any mine, there will also be invisible losses due to misclassification of ore and waste near the cutoff grade. The model created from blastholes compared well to the resource grade model.

18.4 Cutoff Grade

MDA constructed a dollar value block model on which to run Medsystem's Lerchs-Grossmann ultimate pit program. MDA used a cutoff grade of 0.0047 cyanide soluble oz Au/ton. The final designed pit was based on a $450 gold price Lerchs-Grossman pit.

Figure 18.1 Brimstone Annual Pit Design – Yr. 1

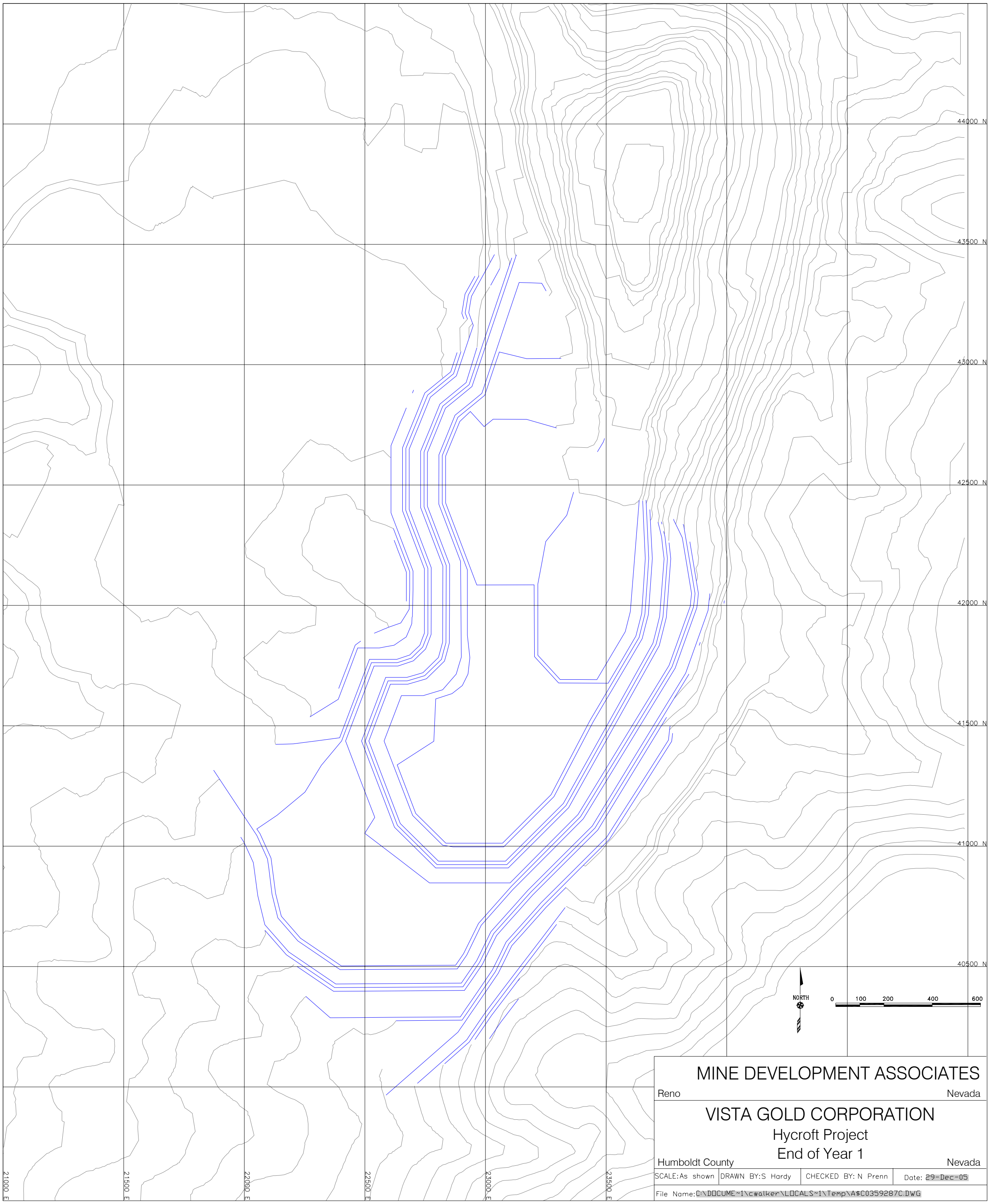


Figure 18.2 Brimstone Annual Pit Design – Yr. 2

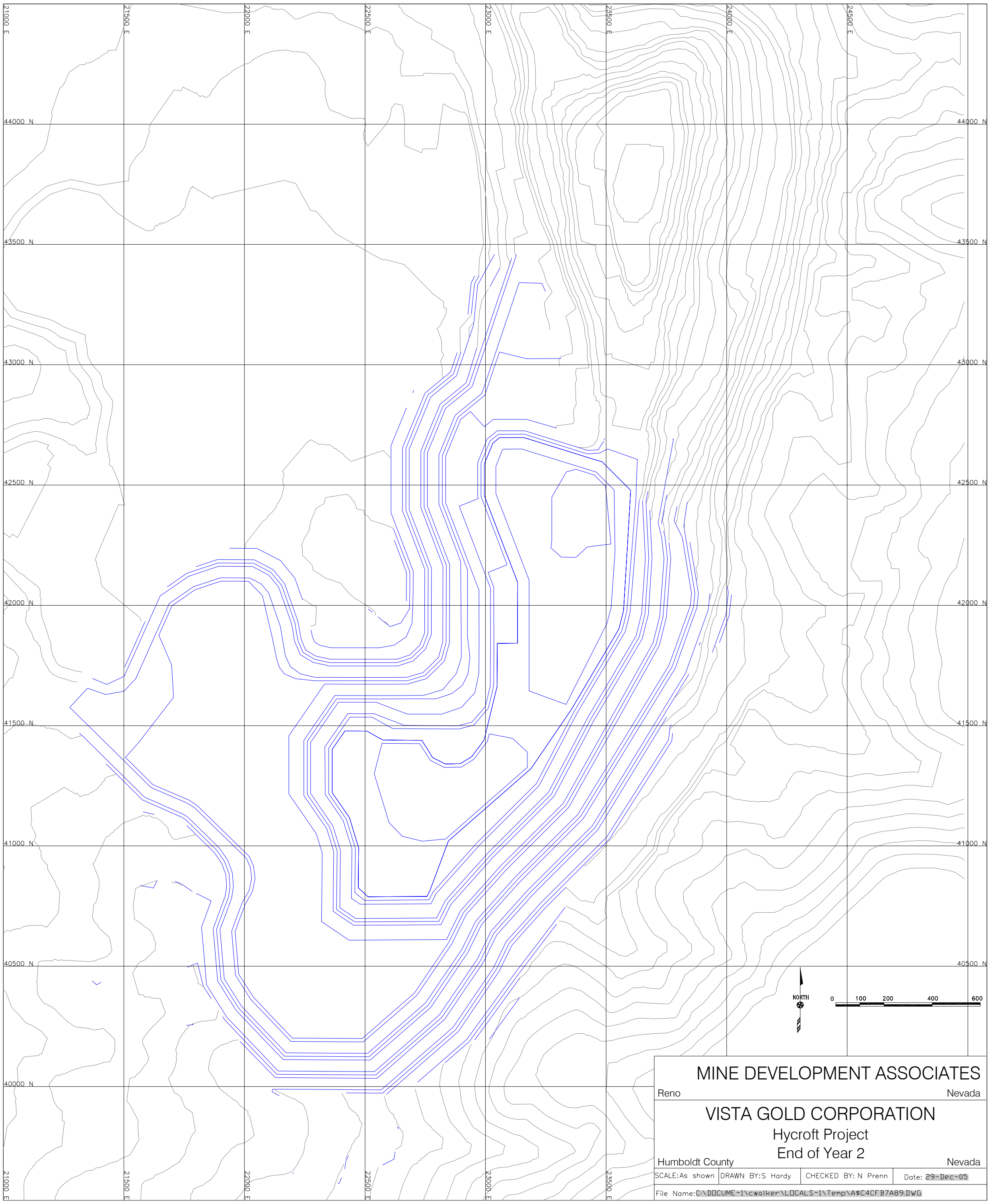


Figure 18.3 Brimstone Annual Pit Design – Yr. 3

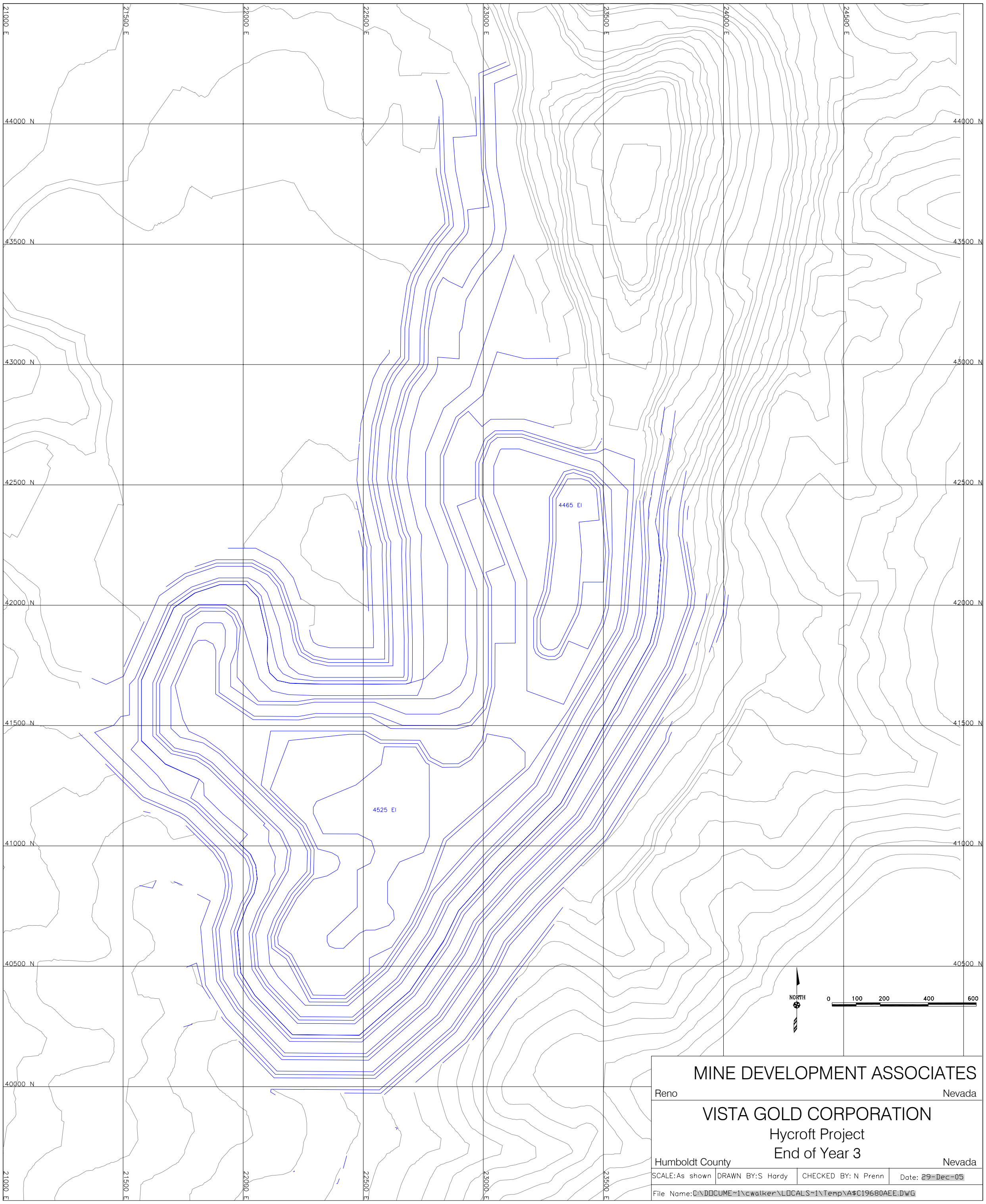
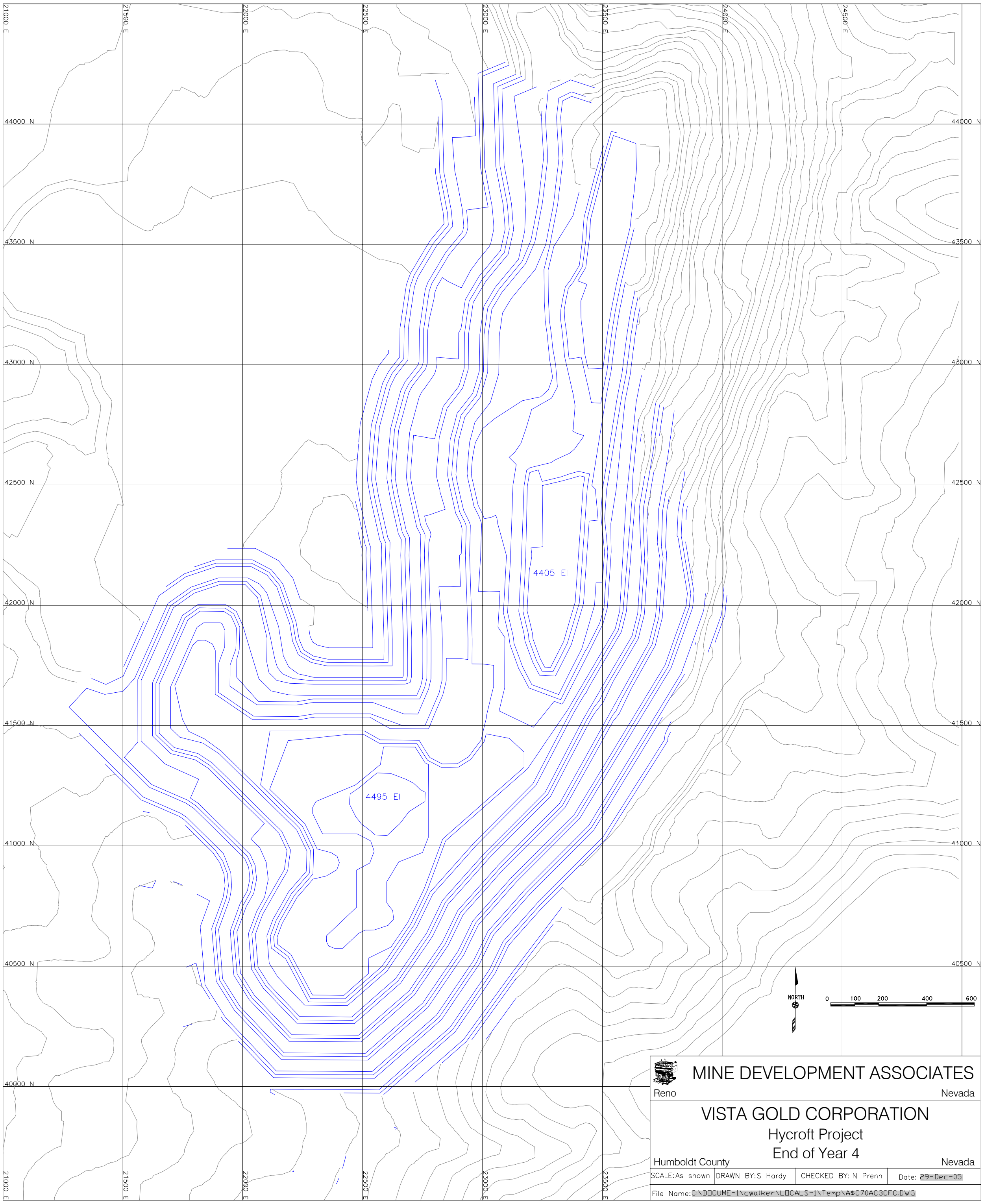


Figure 18.4 Brimstone Final Pit Design



18.5 Mineral Reserve Estimate

The mineral reserve estimate is given in Table 18.4.

Table 18.4 Hycroft Mineral Reserve Estimate

Category	Tons 000's	oz Au/t Cyanide Soluble	Contained oz Au Cyanide Soluble	oz Au/t Fire Assay	Contained oz Au	Waste Alluvium 000's Tons	Waste Rock 000's Tons	Total Waste 000's Tons	Total Pit 000's Tons	Strip Ratio t waste/t ore
Proven	11,954	0.016	188,600	0.022	261,000					
Probable	21,366	0.014	292,700	0.019	401,800					
Totals	**33,320**	**0.014**	**481,300**	**0.020**	**662,800**	**4,975**	**45,833**	**50,808**	**84,128**	**1.52**

MDA also prepared a quarterly mining schedule for these reserves and the economic viability was confirmed. MDA also compared the planned costs with Hycroft historical costs and found them to be reasonable, allowing for annual inflation and changes in fuel pricing and explosives costs. The long term fuel price was estimated to average $2.00/gallon.

Table 18.5 Revenue and Cost Summary

Item	Value
Tons Ore Processed (000"s)	33,320
Grade oz Au/ton (Fire)	0.020
Grade oz Au/ton (Cyanide Soluble)	0.014
Gold Recovered (000's)	375.4
Silver Recovered (000's)	1,501.7
Total Revenue (000's)	179,451.1
Revenue per ton processed (assumes $450/oz Au & $7/oz Ag)	**$5.39**
Costs (per ton processed)	
Mining	$1.85
Equipment Lease	$0.96
Processing	$0.78
Administration	$0.21
Jungo Road Maintenance	$0.02
Royalties	$0.01
Off Site Treatment	$0.04
Total Cost per Ton Processed	**$3.88**
Gross Profit (per ton processed)	**$1.51**
Capital Payments	**$0.63**
Profit Before Income Tax (per ton proce	**$0.88**

19.0 OTHER RELEVENT DATA AND INFORMATION

A little over eight million tons of pre-stripping will be required prior to the start of mining. This will require that sufficient funds be available to complete the stripping, allow routine placement of ore on the leach pads and allow time for the leaching process to begin to return gold production in sufficient quantities to cover operating expenses.

There are geotechnical concerns associated with mining along the East fault and the north end of the Brimstone pit. The current Call and Nicholas recommendation for the fault splay at the north end of the pit is a 33 degree slope angle. This design increases the stripping in that area to such a degree that it makes mining the area uneconomic. However, the north end of the pit also contains inferred tons and the potential for additional ore tons. Additional drilling may assist in defining problem areas. Mining along the East fault will need to be handled carefully, with wall designs being adjusted as the fault is better defined through mining.

Reclamation for the entire project has been included in the economic analysis for the new reserves. The inclusion of the remaining reserves in the reclamation plan does not change current reclamation requirements for Hycroft. If mining the Brimstone pit proceeds, reclamation bonds will need to be increased to cover the costs associated with disturbing additional areas. Currently, reclamation costs are estimated to be approximately $6.8 million. An additional $2.233 million in bonding is estimated to be required for the Brimstone expansion.

20.0 INTERPRETATION AND CONCLUSIONS

The gold mineralization in the Sulfur District, in which the Hycroft Mine is located, is broadly controlled by four major north-northeast-trending, west-dipping normal fault zones. From west to east, these fault zones are referred to as the Central, Boneyard, Albert and East faults. In the past, the mineralized areas of the Central and Boneyard faults have been mined. From 1983 through 1997, 877,460 ounces of gold were produced from these areas. From 1996 to 1998, the north portion of the Brimstone deposit adjoining the East fault was mined and produced 175,965 ounces of gold.

The remaining areas of interest lie in the Albert and Brimstone deposits. The Brimstone deposit has been tested by a total of 412 drill holes (181,828 ft.). Mineralization in the Albert deposit is defined by a total of 163 drill holes (81,473 ft.). Drilling prior to 1999 had poor sample recovery with all drilling techniques because of the soft, friable nature of acid-leach and oxide ores. There was also the problem when mining the Brimstone deposit of recovering more gold than the resource model was predicting.

In 1999, Vista relogged 410 drill holes in the Brimstone deposit and approximately 160 drill holes in the Albert deposit. A comprehensive system for logging lithology, structure, alteration, oxidation, the presence of sulfur and percent sulfur was used. This better data allowed major improvements in the interpretation of the geometry and location of acid-leach, oxide, sulfide and foot-wall units. MRDI found that this work was done professionally and their interpretation honored the revised information. Eleven twin holes were also drilled in 1999 to test the hypothesis that the previous RC drilling had underestimated gold grades. The new holes returned higher fire assay and cyanide-soluble gold grades than the original holes did. Using information derived from the blast hole model and the relogging and twinning programs, MRDI was able to arrive at correction factors for fire-assay and cyanide-soluble gold in the model. MRDI then constructed a new resource model using multiple indicator kriging. This resource model was validated by comparing it with an ordinary kriged cyanide-soluble-gold model of blast holes in the mined area of North Brimstone. The excellent agreement between the models validated the new resource model.

MDA used the resource model created by ORE to determine the resources and reserves of the Hycroft Mine. Pit optimization studies at various gold prices using the entire resource model, including inferred resources, indicated that there is potential for pit expansion if drilling can upgrade the inferred material to measured and indicated status. Vista has proposed an exploration program whose first priorities include infill drilling and extending the oxide resource at the north end of the Brimstone pit. Vista is also proposing to drill other oxide targets in the Sulfur District outside of the Brimstone area.

To date, Vista has concentrated on the oxide portions of the deposits, but there are also plans to examine the sulfide material for the possibility of higher grade veins at depth.

21.0 RECOMMENDATIONS

Currently, Hycroft's Brimstone deposit contains only enough reserves to permit mining for 3.25 years. The restart project at a $450 gold price shows favorable economics. Drilling to increase the proven or probable reserves may improve the project economics. MDA recommends that the Hycroft project project restart be given serious consideration and that additional drilling be completed.

During the exploration and development phases of the Hycroft Mine property, several areas of oxide gold mineralization have been identified. At that time, these areas were deemed to have low potential for generating large deposits (in excess of several million tons) and consequently were not drilled off in detail.

Review of the Brimstone deposit which showed a 19% cyanide-soluble gold-grade increase in the clay-siliceous oxide ore and a 37% cyanide-soluble gold-grade increase in the acid leach ore, indicates that the exploration reserve for this deposit has been underestimated. This is partly due to poor drill-hole sample quality and poor geological modeling. Since similar drilling and geological modeling techniques were used throughout the property, a re-evaluation of all known remaining oxide mineralization zones encountered at Hycroft is suggested.

The oxide resource potential of these zones, recommended drilling programs, and the potential for new oxide gold mineralization targets were prepared by Vista. Canyon drilled 33 holes during 2005. A total of 24 holes were completed in locations proposed by Vista in a detailed report A Proposed Exploration Program for the Hycroft Mine (Bates, 2001). Nine of the Canyon holes were more than 150 ft from any proposed location. The remaining Phase 1 exploration program from the 2001 proposed program is outlined below:

- **Brimstone Oxide Resource Extension;** Canyon completed 3 of the 35 proposed holes for the Brimstone Oxide extension. The remaining 32 holes total 18,335 of drilling that were considered first priority drilling. This program is estimated to cost $330,000.

- **Brimstone Infill Drilling;** Canyon completed 21 of the 50 proposed drill holes. The remaining 29 priority 1 and 2 drill holes total 18,065 ft of drilling. This program is estimated to cost $325,000.

- **Geophysical Program**; A pole-dipole IP program is proposed for the southern and northern part of the district, where alluvial cover prevents prospecting and sampling. The program would entail about 27 line miles of geophysics, and cost approximately US$ 65,000.

The estimated cost of the Phase 1 program is shown in Table 21.1, while the drill hole locations are shown in Table 21.2 for the Brimstone oxide extension and 21.3 for the Brimstone infill drilling.

Table 21.1 Phase 1 Program

Program	# Holes	Footage	Drill Type	Cost
Brimstone Oxide Extension	32	18,335	RC	330,000
Brimstone Infill Drilling	29	18,065	RC	325,000
Geophysics				65,000
Totals	61	36,400		720,000

Table 21.2 Phase 1 Brimstone Oxide Extension Suggested Holes

Hole	East	North	Elevation	Azmuth	Dip	Depth	Target	Drill Type	Priority
05-3053	Completed by Canyon						Brimstone Oxide Extension	RC	1
05-3054	Completed by Canyon						Brimstone Oxide Extension	RC	1
05-3064	Completed by Canyon to about 50% of recommended depth					320	Brimstone Oxide Extension	RC	1
Prop-01	22600	40625	5060	110	-55	370	Brimstone Oxide Extension	RC	1
Prop-02	22850	39600	5180	110	-55	370	Brimstone Oxide Extension	RC	1
Prop-03	22665	39400	5130	0	-90	500	Brimstone Oxide Extension	RC	1
Prop-04	22675	39000	5130	0	-90	450	Brimstone Oxide Extension	RC	1
Prop-05	22570	38625	5110	0	-90	425	Brimstone Oxide Extension	RC	1
Prop-06	22430	38400	4990	0	-90	500	Brimstone Oxide Extension	RC	1
Prop-07	22700	37750	5110	0	-90	500	Brimstone Oxide Extension	RC	1
Prop-08	21770	40000	4980	125	-60	675	Brimstone Oxide Extension	RC	1
Prop-09	21660	40100	4955	125	-60	675	Brimstone Oxide Extension	RC	1
Prop-10	21560	39930	4935	125	-60	625	Brimstone Oxide Extension	RC	1
Prop-11	21870	39930	5015	125	-60	710	Brimstone Oxide Extension	RC	1
Prop-13	23900	45000	4855	110	-45	600	Brimstone Oxide Extension	RC	1
Prop-14	24250	45650	4860	110	-45	350	Brimstone Oxide Extension	RC	1
Prop-16	22665	39600	5110	0	-90	500	Brimstone Oxide Extension	RC	1
Prop-17	22260	39825	5030	0	-90	650	Brimstone Oxide Extension	RC	1
Prop-18	21700	39200	4965	125	-60	650	Brimstone Oxide Extension	RC	1
Prop-19	17500	42950	4400	90	-60	400	Brimstone Oxide Extension	RC	1
Prop-30	26600	51500	4800	110	-50	600	Brimstone Oxide Extension	RC	1
Prop-31	26300	51000	4800	110	-50	600	Brimstone Oxide Extension	RC	1
Prop-32	26100	50500	4800	110	-50	600	Brimstone Oxide Extension	RC	1
Prop-33	25900	50000	4800	110	-50	600	Brimstone Oxide Extension	RC	1
Prop-34	25700	49500	4800	110	-50	600	Brimstone Oxide Extension	RC	1
Prop-35	22425	39000	4990	0	-90	475	Brimstone Oxide Extension	RC	1
Prop-36	22425	38800	5070	0	-90	475	Brimstone Oxide Extension	RC	1
Prop-37	21800	40200	4990	125	-60	690	Brimstone Oxide Extension	RC	1
Prop-38	21930	40165	5010	125	-60	700	Brimstone Oxide Extension	RC	1
Prop-39	21800	40725	4965	0	-90	665	Brimstone Oxide Extension	RC	1
Prop-41	21400	41450	4870	0	-90	470	Brimstone Oxide Extension	RC	1
Prop-42	22135	41650	4940	0	-90	640	Brimstone Oxide Extension	RC	1
Prop-43	21700	39500	4935	125	-60	650	Brimstone Oxide Extension	RC	1
Prop-44	21860	39400	4965	125	-60	650	Brimstone Oxide Extension	RC	1
Prop-45	21550	39300	4935	125	-60	650	Brimstone Oxide Extension	RC	1
	32	Drill Holes				18,335	Brimstone Oxide Extension	RC	1

Table 21.3 Phase 1 Brimstone Infill Drilling Suggested Holes

Hole	East	North	Elevation	Azmuth	Dip	Depth	Target	Drill Type	Priority
05-3047	Completed by Canyon						Brimstone Oxide Infill	RC	1
05-3048	Completed by Canyon						Brimstone Oxide Infill	RC	1
05-3056	Completed by Canyon						Brimstone Oxide Infill	RC	1
05-3049	Completed by Canyon to about 60% of recommended depth					240	Brimstone Oxide Infill	RC	2
05-3038	Completed by Canyon to about 60% of recommended depth					200	Brimstone Oxide Infill	RC	2
05-3039	Completed by Canyon to about 60% of recommended depth					200	Brimstone Oxide Infill	RC	2
05-3041	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3042	Completed by Canyon to about 60% of recommended depth					240	Brimstone Oxide Infill	RC	2
05-3043	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3044	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3045	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3046	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3050	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3051	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3057	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3058	Completed by Canyon to about 60% of recommended depth					250	Brimstone Oxide Infill	RC	2
05-3059	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3061	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3062	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3063	Completed by Canyon						Brimstone Oxide Infill	RC	2
05-3066	Completed by Canyon to about 60% of recommended depth					300	Brimstone Oxide Infill	RC	2
Prop-21	23800	44725	4910	110	-65	550	Brimstone Oxide Infill	RC	1
Prop-22	23600	44725	4900	110	-65	600	Brimstone Oxide Infill	RC	1
Prop-23	23600	44725	4900	0	-90	700	Brimstone Oxide Infill	RC	1
Prop-24	23500	44325	4540	90	-65	300	Brimstone Oxide Infill	RC	1
Prop-25	23390	42525	4630	0	-90	700	Brimstone Oxide Infill	RC	1
Prop-26	23520	42125	4955	0	-90	560	Brimstone Oxide Infill	RC	1
Prop-27	21900	41850	4960	0	-90	800	Brimstone Oxide Infill	RC	1
Prop-46	23325	42025	4960	0	-90	650	Brimstone Oxide Infill	RC	2
Prop-47	23170	41825	4960	0	-90	800	Brimstone Oxide Infill	RC	2
Prop-48	23500	41825	4960	0	-90	500	Brimstone Oxide Infill	RC	2
Prop-50	23500	41625	4960	0	-90	300	Brimstone Oxide Infill	RC	2
Prop-51	22265	41625	4985	0	-90	400	Brimstone Oxide Infill	RC	2
Prop-56	22200	41525	4930	0	-90	580	Brimstone Oxide Infill	RC	2
Prop-57	22500	41425	4950	0	-90	700	Brimstone Oxide Infill	RC	2
Prop-58	23140	41425	4955	0	-90	550	Brimstone Oxide Infill	RC	2
Prop-59	22600	41425	4950	90	-72	700	Brimstone Oxide Infill	RC	2
Prop-60	22900	41225	4985	90	-70	700	Brimstone Oxide Infill	RC	2
Prop-61	22000	41225	4920	0	-90	500	Brimstone Oxide Infill	RC	2
Prop-62	22000	41225	4920	90	-65	475	Brimstone Oxide Infill	RC	2
Prop-68	22520	40725	4975	0	-90	700	Brimstone Oxide Infill	RC	2
Prop-69	22210	40725	4995	0	-90	570	Brimstone Oxide Infill	RC	2
Prop-70	22000	40625	5025	0	-90	600	Brimstone Oxide Infill	RC	2
Prop-72	22675	40625	5025	90	-60	400	Brimstone Oxide Infill	RC	2
Prop-73	22735	40525	5050	0	-90	400	Brimstone Oxide Infill	RC	2
Prop-75	21935	40525	4990	0	-90	700	Brimstone Oxide Infill	RC	2
Prop-76	22590	40325	5050	0	-90	500	Brimstone Oxide Infill	RC	2
Prop-79	22600	40225	4970	0	-90	500	Brimstone Oxide Infill	RC	2
Prop-81	22330	40225	4960	0	-90	700	Brimstone Oxide Infill	RC	2
Prop-83	22530	40125	5085	0	-90	500	Brimstone Oxide Infill	RC	2
	29	Drill Holes				18,065			

The proposed Phase 2 program is outlined below:

- **Oxide targets outside Brimstone Area;** A total of five RC holes totaling 2,450 ft were proposed; the areas explored by these holes are the east wall of Cut 4 and the area south of Silver Camel on the Hades lineament. This program would cost $44,100. This program is third priority.

- **Brimstone sulfide;** This program is a test of high-grade targets on the south end of the Central fault, and a test of the Brimstone system at depth. The Central fault drilling will require about 2,000 ft of RC drilling and 2,100 ft of Diamond drilling, in 7 holes and the Brimstone bulk tonnage sulfide test would require about 8,720 ft of RC drilling in 7 holes. The estimated cost of this program is $276,960.

- A further follow-up drill program to bring resources found in (1) above is estimated to involve 45,000 ft of drilling in 75 drill holes. The estimated cost for this program is US$ 810,000. This program could be carried out after completing Phase 1. Drill positions are dependent on results of Phase 1.

The Phase 2 program will follow up on results from the Brimstone oxide exploration in Phase 1 above, and address targets under Brimstone and along the Central fault for high-grade potential. The estimated cost of the Phase 2 program is shown in Table 21.4. A first attempt at finding oxide resources should also be performed along the Hades lineament and a test drilling program should be performed in the Cut 5 east wall.

Table 21.4 Phase 2 Program Drilling Oxide Reserve Extensions-Bulk Tonnage Sulfide/High Grade

Program	# Holes	Footage	Drill Type	Cost
Brimstone Oxide Extensions (6)	75	45,000	RC	810,000
Sulfide Bulk Tonnage, High-Grade (4)	14	10,720	RC-DD	276,960
Oxide Outside Brimstone, Cut5 (3)	5	2,450	RC	44,100
Total Phase 2	**94**	**58,170**	**RC-DD**	**1,131,060**

MDA has reviewed the proposed program and recommends that it be completed to properly evaluate the material remaining at the property.

22.0 ADDITIONAL INFORMATION ON THE PROPOSED HYCROFT OPERATION

22.1 Mining

Mining will be a conventional shovel truck operation. Trucks will be approximately 150 ton size and the shovel will load them in 4-5 passes. A large front end loader will act as a backup to the shovel. Drills will be capable of drilling 7 7/8" diameter holes with a single pass on an 18-20 ft staggered pattern.

It has not yet been determined whether Hycroft will hire personnel and purchase or lease equipment to do the mining themselves or whether the mining portion of the operation will be contracted. In either case, Hycroft personnel will operate the leaching and processing facilities. The planned operation will operate two ten hour shifts per day, six days per week.

Table 22.1 Planned Primary Mine Equipment

Equipment Description	Size	Quantity
Blasthole Drill	6.5 - 9 in	3
Hydraulic Front Shovel	27 cy	1
Front End Loader	24 cy	1
Trucks	150 ton	11
CAT D11 Bulldozer		3
Wheel Dozer		1
CAT 16H Motor Grader		1
Water Truck		1

All of this major equipment is rented for the life of the mine. MDA derived equipment productivities, haul profiles and cost estimates based on the designed pit. Estimated owner mining costs averaged $0.81 per ton mined, not including equipment rental, as shown in Table 22.2. Equipment rental costs totals $0.42 per ton mined.

Table 22.2 Owner Mining – Estimated Costs

Item	Cost/ton mined	Cost/ton processed
General Mine Expense	$0.073	$0.167
Drilling	$0.097	$0.221
Blasting	$0.110	$0.249
Loading	$0.168	$0.382
Hauling	$0.244	$0.555
Support	$0.122	$0.277
Totals	**$0.814**	**$1.850**

Note: Not including equipment rental, preproduction stripping

Table 22.3 shows the mine production schedule including pre-stripping. The gold ounces recovered from the heap leach operation are shown as recovered ounces.

Table 22.3 Production Schedule

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Totals
Production Statistics							
Ore Mined, 000's Tons	800.0	8,312.0	10,639.0	11,514.0	2,055.0		33,320.0
Waste Mined, 000's Tons	7,554.0	15,688.0	13,361.0	12,486.0	1,719.0		50,808.0
Total Mined, 000's Tons	8,354.0	24,000.0	24,000.0	24,000.0	3,774.0		84,128.0
Ore Grade (oz Au/ton)	0.013	0.020	0.021	0.020	0.016		0.020
Ore Grade (cyanide soluble oz Au/ton	0.011	0.016	0.015	0.014	0.011		0.014
Contained Ounces Au (000's)	10.5	169.8	221.4	228.7	32.4		662.8
Contained Soluble Ounces Au (000's)	8.5	129.5	164.0	156.3	23.0		481.3
Gold Sales (000's oz Au)		66.1	108.4	131.7	60.6	8.6	375.4
Silver Sales (000's oz Ag)		264.2	433.8	526.9	242.4	34.4	1,501.7
Strip ratio		1.89	1.26	1.08	0.84		1.52

Figures 22.1 and 22.2 are views of the current Brimstone pit.

Figure 22.1 Brimstone Pit Looking North

Figure 22.2 Brimstone Pit Looking South

Waste Dumps

Depending on the location that the waste exits the designed pit, some waste will be placed on pads to the west of the pit, but the majority of the waste will be used as backfill in the previously mined Central fault pit. A small amount of waste may be dumped in the north end of the designed pit after that area has been mined out.

22.2 Processing

The processing costs were developed from past experience from the Brimstone deposit. Anticipated reagent consumption is shown in Table 22.4, based on 1998 actual data and recent reagent quotes from vendors.

Table 22.4 Reagent Consumption

Reagent	Rate (pounds/ton)	Cost $/pound
Lime (3/8" x 1/8")	2.50	$0.04
Cyanide (33-34% Soln)	0.20	$0.65
Anti-Scalent (Antiprex)	0.02	$0.55
Filter Aid (D.E)	0.15	$0.17
Zinc Dust	0.03	$0.86

Table 22.5 shows the estimated processing costs over the life of the mine. The estimated costs assume 75% of the cyanide consumption occurs during the first year of operation. Gold recovery during year six is assumed to be from rinsing during closure activities.

Table 22.5 Processing Operating Cost Estimate

Item	Year 1 000's	Year 2 000's	Year 3 000's	Year 4 000's	Year 5 000's	Totals 000's	$/ ton processed
Tons Processed (000's)	9,112.0	10,639.0	11,514.0	2,055.0		33,320.0	
Ounces Recovered (000's)	66.1	108.4	131.7	60.6	8.6	375.4	
Leaching	$2,985.3	$3,328.5	$3,525.2	$1,803.3	$1,277.9	$12,920.1	0.388
Merrill Crowe	$1,293.5	$1,364.0	$1,403.9	$1,082.4	$957.7	$6,101.5	0.183
Refining	$463.8	$463.8	$463.8	$463.8	$301.8	$2,157.1	0.065
Assay Lab Operations	$435.7	$435.7	$435.7	$435.7	$204.3	$1,946.9	0.058
General	$607.3	$607.3	$607.3	$607.3	$382.4	$2,811.6	0.084
Totals	**$5,785.5**	**$6,199.3**	**$6,435.9**	**$4,392.5**	**$3,124.2**	**$25,937.3**	**0.778**

22.3 Personnel

Table 22.6 shows the anticipated personnel requirements for owner mining.

Table 22.6 Anticipated Hycroft Personnel – Owner Mining

Item	Number
Mine Operations	
Mine Manager	1
Mine Superintendent	1
Mine Ops Supervisor	3
Blasting Foreman	1
Mine Clerk	1
Drill Operators	6
Blasters	2
Blaster Helpers	2
Loader Operators	12
Haul Truck Drivers	25
Dozers Operators	6
Grader Operators	3
Utility Operators	1
Trainee/Spare	1
Subtotal Mine Operations	**65**
Mine Maintenance	
Maintenance Superintendent	1
Maintenance General Foreman	1
Maintenance Supervisor	3
Maintenance Planner	1
Mechanics	10
Electricians	2
Welders/Millwright	4
Lt. Veh. Mechanics	1
Apprentice	1
Servicemen	2
Tireman	2
Subtotal Mine Maintenance	**28**
Mine Engineering and Geology	
Chief Mining Engineer	1
Exploration Manager	1
Mining Engineer	1
Geologist	2
Surveyor	2
Subtotal Engineering and Geology	**6**
Processing	
Plant Manager	1
Metallurgist	1
Leach Supervisor	2
Leach Operator	5
Leach Utility	1
Assayors	5
Plant Operators	8
Refinery Operators	1
Mechanics and Welders	5
Electrician	2
Subtotal Processing	**31**
Administration	
General Manager	1
Human Resources Manager	1
Human Resources Clerk	1
Mine Controller	1
Information Systems Coordinator	1
Purchasing Agent	1
Accountants	1
Accounting Clerks	1
Warehouse Supervisor	1
Warehouse Clerk	1
Safety and Environmental	
Safety Director	1
Environmental Manager	1
Environmental Specialist	1
Subtotal Administration	**13**

22.4 Environmental Considerations

In January 2004, Vista announced that HRDI had reached an agreement to replace their existing reclamation bonds. The new bond package allows for future increases when Hycroft moves back into production. Reclamation at the property is on-going. In 2003, the Mines Group, Inc. of Reno, Nevada estimated the cost of the current reclamation to total $6.767 million. Additional bonding may be required for the new leach pad area and other disturbed areas. Vista provided a previous estimate of total bonding costs of $9 million. MDA has included an additional $2.233 million in bonding as a capital item that may be required, bringing the total bonding estimate to $9 million for the project.

22.5 Taxes

MDA's economic evaluation is a pre-tax evaluation, however the estimated Nevada Net Proceeds tax is included in the economic evaluation, which is expected to total $2.77 million over the life of the mine. Anticipated federal, state and local property taxes are not included in the evaluation.

22.6 Capital and Operating Costs

Table 22.7 shows the estimated capital costs for mine equipment based on equipment rental. Estimated equipment rental cost is shown in Table 22.8. Preproduction stripping of a total of 8.35 million tons is estimated to cost a total of $10.2 million, including equipment rental charges. Construction of the new 3 million square ft leach pad area is estimated to cost $3.4 million. The initial inventory of consumables and spare parts is estimated at $1.15 million.

Table 22.7 Hycroft Mine Equipment Estimated Capital Cost

Item	Cost	Frt/Erect	Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Totals
	$000's	$000's	$000's	$000's	$000's	$000's	$000's	$000's	$000's
Loading Equipment									
Spare 27.5 cm Bucket	$425.0	$0.0	$425.0		$425.0				$425.0
Cranes for Assembly	$40.0	$0.0	$40.0	$40.0					$40.0
EX-3600 Parts On-site Inventory	$40.0	$0.0	$40.0	$40.0					$40.0
Haul Trucks									
Spare truck bed	$145.0	$5.0	$150.0		$150.0				$150.0
Support Equipment									
Light Plant	$23.0	$0.6	$23.6	$70.7					$70.7
Mechanics Truck	$100.0	$2.5	$102.5	$205.0					$205.0
Pickup Trucks	$35.0	$0.9	$35.9	$287.0		$287.0			$574.0
Welding Truck/Crane	$60.0	$1.5	$61.5	$61.5					$61.5
45 T Hydraulic Crane	$440.0	$11.0	$451.0	$451.0					$451.0
200 HP Integrated Tool Carrier	$300.0	$7.5	$307.5	$307.5					$307.5
1 CM Loader/Backhoe	$85.0	$2.1	$87.1	$87.1					$87.1
Ambulance and Fire Equipment	$150.0	$3.8	$153.8	$153.8					$153.8
Flatbed Truck	$60.0	$1.5	$61.5	$61.5					$61.5
Crew Vans	$50.0	$1.3	$51.3	$102.5		$102.5			$205.0
Forklift	$40.0	$1.0	$41.0	$82.0					$82.0
Total Equipment Capital				**$1,949.6**	**$575.0**	**$389.5**	**$0.0**	**$0.0**	**$2,914.1**

Table 22.8 Hycroft Mine Equipment Rental Cost

Item	Unit Cost $000's	Frt/Erect $000's	Unit Total $000's	Yr -1 $000's	Yr 1 $000's	Yr 2 $000's	Yr 3 $000's	Yr 4 $000's	Totals $000's
Drills									
IR DML Blasthole Drill	$880.0	$22.0	$902.0	$259.8	$844.3	$844.3	$844.3	$140.7	$2,933.3
Loading Equipment									
27.5 cy Front Shovel - Hitachi EX-3600	$4,450.5	$0.0	$4,450.5	$640.9	$1,281.7	$1,281.7	$1,281.7	$213.6	$4,699.7
24 CY Wheel Loader - Cat 994	$2,950.0	$132.0	$3,082.0	$443.8	$887.6	$887.6	$887.6	$147.9	$3,254.6
Haul Trucks									
150 t Truck- Cat 785	$1,665.0	$41.6	$1,706.6	$1,228.8	$4,392.9	$5,038.0	$5,406.6	$778.2	$16,844.4
Dozer - Cat D11R	$1,565.0	$39.1	$1,604.1	$462.0	$1,443.7	$1,443.7	$1,443.7	$240.6	$5,033.7
Rubber Tire Dozer Cat 834	$680.0	$17.0	$697.0	$100.4	$200.7	$200.7	$200.7	$33.5	$736.0
Grader 16H	$650.0	$16.3	$666.3	$95.9	$191.9	$191.9	$191.9	$48.0	$5,769.8
Water Truck - Cat 777	$1,150.0	$28.8	$1,178.8	$169.7	$339.5	$339.5	$339.5	$56.6	$3,602.3
Totals				**$3,401.3**	**$9,582.3**	**$10,227.4**	**$10,596.0**	**$1,659.1**	**$35,466.1**

Owner mining with rental equipment is estimated to cost $0.81 per ton mined, not including the rental equipment cost, based on a long term average fuel cost of $2.00 per gallon. Equipment rental adds about $0.42 per ton mined to the cost. The estimated processing cost of $0.78 per ton of run of mine material is shown in detail in Table 22.5. Administrative costs are estimated to average $0.22 per ton of run of mine material, including $0.03 per ton for Jungo road maintenance. Transportation, insurance, and refining of the dore produced at the mine is estimated to cost $0.02 per ton placed on the pad. Total costs are estimated to be $3.86 per ton of mine run material, including an estimate of $0.96 per ton ore for equipment rental. Silver credits for the project total $28 per oz Au. Royalties and the Nevada Net Proceeds tax add about $0.10/ton placed on the pad. The total cash cost to produce an ounce of gold is $257 per ounce prior to rental costs and $343 including the rental costs. The Nevada Net Proceeds tax and royalties bring the total to $351 per ounce.

22.7 Hycroft Cash Flow Estimates

The Hycroft cash flow is most sensitive to gold price followed by operating cost and mining cost. The cash flow was extended through year six even though mining will be completed in year four to include the delayed revenues from a heap leach operation. Figure 22.3 shows the project sensitivity of the pre-tax internal rate of return, while Figure 22.4 illustrates the net present value (5% discount) sensitivity. The total cash cost of the restart project is estimated to be $257 per ounce ($343 with equipment rental), as shown in Table 22.9. The estimated pre-tax internal rate of return of the project is 29.5% using a gold price of $450 per ounce and silver price of $7.00 per ounce. The pre-tax net present value at a 5% discount rate is $18.9 million.

Table 22.9 Restart Project Economics – Base Case – Mine Equipment Rental

Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Totals	$/Ounce Au
Production Statistics								
Ore Mined, 000's Tons	800.0	8,312.0	10,639.0	11,514.0	2,055.0		33,320.0	
Waste Mined, 000's Tons	7,554.0	15,688.0	13,361.0	12,486.0	1,719.0		50,808.0	
Total Mined, 000's Tons	8,354.0	24,000.0	24,000.0	24,000.0	3,774.0		84,128.0	
Ore Grade (oz Au/ton)	0.013	0.020	0.021	0.020	0.016		0.020	
Ore Grade (cyanide soluble oz Au/ton	0.011	0.016	0.015	0.014	0.011		0.014	
Contained Ounces Au (000's)	10.5	169.8	221.4	228.7	32.4		662.8	
Contained Soluble Ounces Au (000's)	8.5	129.5	164.0	156.3	23.0		481.3	
Gold Sales (000's oz Au)		66.1	108.4	131.7	60.6	8.6	375.4	
Silver Sales (000's oz Ag)		264.2	433.8	526.9	242.4	34.4	1,501.7	
Strip ratio		1.89	1.26	1.08	0.84		1.52	
Revenue								
Gold Revenue		$29,727.9	$48,798.3	$59,273.5	$27,265.8	$3,873.9	$168,939.3	$450
Silver Revenue		$1,849.7	$3,036.3	$3,688.1	$1,696.5	$241.0	$10,511.8	$28
Gross Revenues	**$0**	**$31,577.6**	**$51,834.6**	**$62,961.7**	**$28,962.3**	**$4,115.0**	**$179,451.1**	$478
Cash Costs								
Mining (excludes cap. pre-strip)	$0.0	$19,057.6	$19,416.2	$19,975.8	$3,207.7	$0.0	$61,657.3	$164
Equipment Rental	$0.0	$9,582.3	$10,227.4	$10,596.0	$1,659.1	$0.0	$32,064.8	$85
Processing	$0.0	$5,785.5	$6,199.3	$6,435.9	$4,392.5	$3,124.2	$25,937.3	$69
Refining, Freight	$0.0	$231.2	$379.5	$461.0	$212.1	$30.1	$1,314.0	$4
Administration	$0.0	$1,701.5	$1,761.3	$1,784.1	$1,014.2	$716.3	$6,977.4	$19
Jungo road	$0.0	$216.0	$216.0	$216.0	$108.0	$0.0	$756.0	$2
Direct Operating Costs	$0.0	$36,574.2	$38,199.7	$39,468.8	$10,593.5	$3,870.6	**$128,706.8**	$343
Royalty and Nevada Net Proceeds								
Crofoot Royalty - 4% net profit	$0.0	$120.0	$120.0	$120.0	$120.0	$0.0	$480.0	$1
Nevada Net Proceeds			$675.7	$1,168.6	$912.4	$8.6	$2,765.5	$7
Total Cash Costs	**$0.0**	**$36,694.2**	**$38,995.5**	**$40,757.4**	**$11,625.9**	**$3,879.3**	**$131,952.3**	**351**
Capital Expenditures								
Mine pre-stripping	$11,154.9	$0.0	$0.0	$0.0	$0.0	$0.0	$11,154.9	$30
Mine Equipment	$1,949.6	$575.0	$389.5				$2,914.1	$8
Leach Pad	$2,000.0	$1,400.0	$0.0	$0.0	$0.0	$0.0	$3,400.0	$9
Inventory	$1,150.0	$0.0	$0.0	$0.0	$0.0	$0.0	$1,150.0	$3
Total Capital Expenditures	**$16,254.5**	**$1,975.0**	**$389.5**	**$0.0**	**$0.0**	**$0.0**	**$18,619.0**	$50
Other Capital								
Sale of Assets					($437.1)		($437.1)	($1)
Reclamation & Severance (Additional Bondi	$2,233.0	$0.0	$0.0	$0.0	$200.0	$50.0	$2,483.0	$7
Net Cash Flow	**($18,487.5)**	**($7,091.6)**	**$12,449.6**	**$22,204.2**	**$17,573.5**	**$185.7**	**$26,833.9**	$71
Cumulative Cashflow	**($18,487.5)**	**($25,579.1)**	**($13,129.5)**	**$9,074.7**	**$26,648.2**	**$26,833.9**		
Net Present Value and Internal Rate of Ret	Gold Price	$450						
	Silver Price	$7			$ 000's			
			NPV	0%	$26,833.9			
				3%	$21,802.3			
				5%	$18,890.3			
				10%	$12,868.3			
			IRR	29.47%				
			Payback =	31.1	months			

Figure 22.3 Hycroft IRR Sensitivity

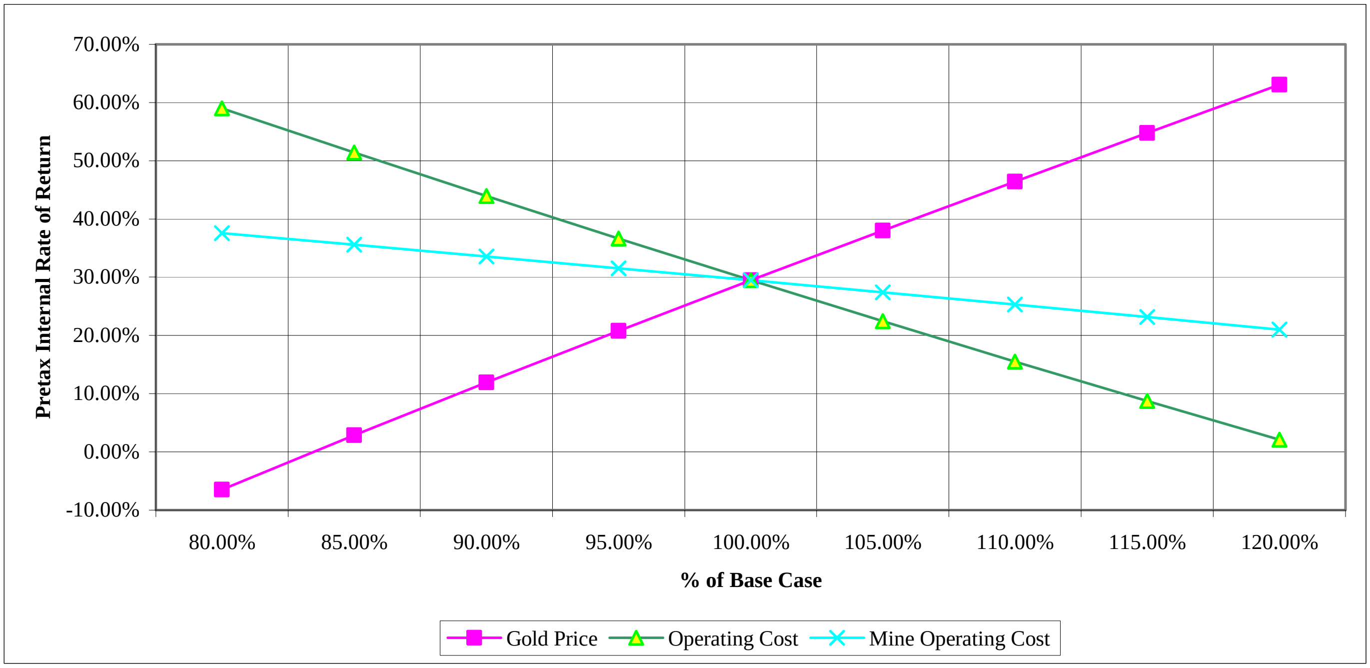
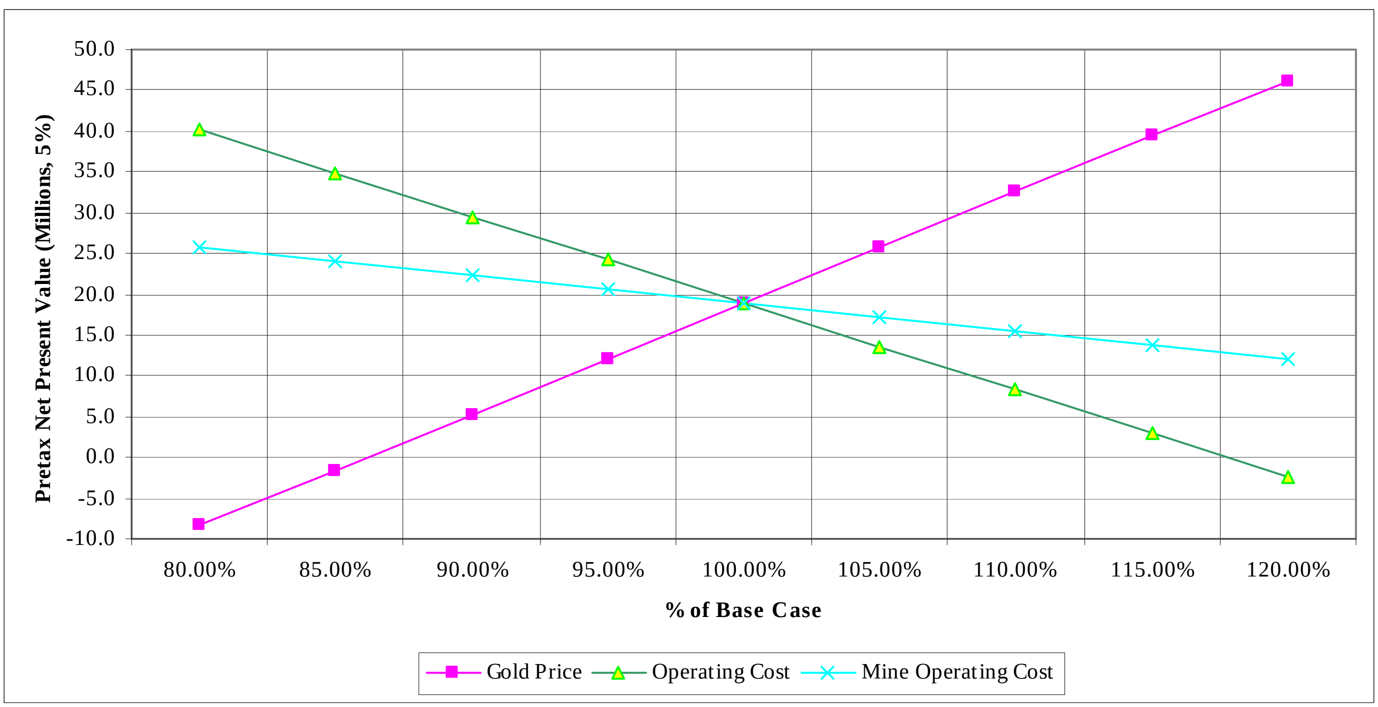


Figure 22.4 Hycroft NPV (5%) Sensitivity



22.8 Mine Life and Exploration Potential

Exploration should concentrate first on upgrading the inferred material within and immediately adjacent to the final pit. This would have a substantial positive impact on the existing cash flow. Any additional reserves identified immediately adjacent to the pit could also significantly increase the mine life and have a positive financial effect.

Recommendations for development of resources and reserves are discussed in more detail in Section 21.

23.0 REFERENCES

Agterberg, F.P., 1974, Geomathematics; Developments in Geomathematics 1, Elsevier Scientific Publ. Co., Amsterdam, 596 p.

Bailey, E.H, and Phoenix, D.A., 1944 , Quicksilver Deposits in Nevada, University of Nevada Bulletin, v.38, no.5

Bates, W.R., 2000. Hycroft Exploration Update, unpublished Vista Gold report

Bates, W.R., 2001. A proposed Exploration Program for the Hycroft Mine, unpublished Vista Gold report

Call & Nicholas, Inc. January 29, 1997, "Hycroft Crofoot & Lewis Mine: Brimstone East Wall Stability Study"

Clark. I.C., 1918, Recently Recognized Alunite Deposits at Sulphur, Humboldt County, Nevada, engineering and Mining Journal, v. 106 , no.4

Couch, B.F and Carpenter, J. A., 1943, Nevada's metal and mineral production (1859-1940 inclusive)Univ. Nevada Bulletin, v.37,no.4 Geology and Mining Services no.38

Davis, J.C., 1986, Statistics and Data Analysis in Geology (2nd ed.); John Wiley and Sons Inc., New York, 646 p.

Ebert, S.W., 1996 The anatomy and Origin of the Crofoot/Lewis Mine, a low-sulfidation hot-spring type gold-silver deposit located in northwest Nevada; Ph.D. thesis, University of Western Australia, Nedlands West Australia, (unpublished)

Friberg, R.S., 1980, Detailed Evaluation Report of the Sulphur gold-silver prospect, Humboldt and Pershing Counties, Nevada. Unpublished Homestake Mining Company Report , 32 p.

Fulton and Smith, 1932, Nevada Bureau of Mines and Geology File Manuscript

Hycroft Memorandum: July 21, 1994, "Brimstone R.O.M. testwork conducted on blast round B.C. 002 material"

Hycroft Memorandum: August 11, 1994, "Final Report on Brimstone Core Column Leaching"

Hycroft Memorandum: August 24, 1994, "Results of Barrel Leaching the Residues from Brimstone Core Column Leach Samples"

Hycroft Memorandum: September 20, 1994 "Brimstone Test Heap Results"

Hycroft Memorandum: December 22, 1994, "Final Report on Brimstone Ore from Bench 4935, Test Pad #4"

Hycroft Memorandum: July 19, 1995, "Brimstone Recoveries"

Jones, J.C. 1921 , Report on the Property of the Silver Camel Mining and Development Company, Sulphur, Nevada. Unpublished Silver Camel Mining and Development Company Report, 6 p.

McLean, D.A., 1991 Geology of the Crofoot Mine; unpublished Hycroft Resources and Development Inc Report 11 p.

MRDI, May 2000-Evaluation of Sampling Biases, Resource Model and Reserves, Brimstone Gold Deposit, Nevada, Volume 1-Report, unpublished report for Vista Gold Corp. completed by Mineral Resources Development Inc.

MRDI, June 2002, Brimstone Restart Report, unpublished report prepared for Vista Gold Corp. completed by Mineral Resources Development Inc.

Noble, Alan C., 2005, Brimstone and Boneyard Resource Estimates, prepared for Canyon Resources Corporation.

Vandenburg, W.O., 1938, Reconnaissance of Mining Districts in Humboldt County, Nevada; U.S. Bureau of Mines Circular 6995, 47 p.

Wallace, A.B., 1980, Geology of the Sulphur District, Southwestern Humboldt County, Nevada, unpublished report for the Soc. Econ. Geol. Field Trip, 1984

Ware,G.H., 1989, Surface Mapping, Sampling and Selected Cross Sections, unpublished report for Hycroft Resources and Development

Willden R., 1964, Geology of Mineral Deposits of Humboldt County, Nevada, Nevada Bureau of Mines Bulletin 59, 154 p.

24.0 AUTHOR'S CERTIFICATE

I, Neil B. Prenn, of Reno, Nevada, do hereby certify:

1. I am currently employed as Principal Engineer by:
   Mine Development Associates, Inc.
   210 South Rock Blvd.

   Reno, Nevada 89502.

2. I graduated with an Engineer of Mines degree from the Colorado School of Mines in 1967.

3. I am a Registered Professional Mining Engineer in the state of Nevada (#7844) and a member of the Society of Mining Engineers and councilor-at-large for the Mining and Metallurgical Society of America.

4. I have worked as an engineer for a total of 35 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 am responsible for the preparation of the technical report titled Technical Report – Vista Gold Corp., Hycroft Mine, Winnemucca, Nevada, USA dated January 25, 2006 (the "Technical Report") relating to the Hycroft Property. I visited the Hycroft property on January 8, 2004 for one day.

7. Prior to the visit in January 2004 I had no involvement with the Hycroft Project. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. That I have read National Instrument 43-101 and Form 43-101 and that this technical report was prepared in compliance with National Instrument 43-101.

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

10. 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 this 25th day of January 2006.

____
Signature of Qualified Person

Neil B. Prenn
Print Name of Qualified Person

I, Deepak Malhotra, of Wheat Ridge, Colorado, do hereby certify:

1. I am currently employed as President of Resource Development Inc.,
   11475 W I-70 Frontage Road N.
   Wheat Ridge, CO 80033

2. I graduated with a Masters of Science in Metallurgical Engineering in 1973 and a PhD in Mineral Economics in 1977 from Colorado School of Mines.

3. I am a member of the Society of Mining, Metallurgy, and Exploration Inc. (SME) and Canadian Institute of Mining, Metallurgy, and Petroleum (CIM).

4. I have worked as an engineer for a total of 33 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 am responsible for the preparation of the technical report titled Technical Report-Vista Gold Corp., Hycroft Mine, Winnemucca, Nevada, USA dated January 25, 2006 (the "Technical Report") relating to Hycroft Property. I have not visited the Hycroft property.

7. I have had no previous involvement with the Hycroft Project. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. That I have read National Instrument 43-101 and form 43-101 and that this technical report was prepared in compliance with National Instrument 43-101.

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

10. 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 this 28th day of January 2006.

Signature of Qualified Person

APPENDIX A

CLAIMS LIST

Claim Name	NMC Number	Owner	County
AIRSTRIP # 1	88292	Crofoot	Humboldt
AIRSTRIP # 2	88293	Crofoot	Humboldt
AIRSTRIP # 3	88294	Crofoot	Humboldt
AIRSTRIP # 4	88295	Crofoot	Humboldt
AIRSTRIP # 5	88296	Crofoot	Humboldt
AIRSTRIP FRAC	88297	Crofoot	Humboldt
CKC # 1	88348	Crofoot	Humboldt
CKC # 2	88349	Crofoot	Humboldt
CKC # 3	88350	Crofoot	Humboldt
CKC # 4	88351	Crofoot	Humboldt
CKC # 5	88352	Crofoot	Humboldt
CKC # 6	88353	Crofoot	Humboldt
CKC # 7	88354	Crofoot	Humboldt
CKC # 8	88355	Crofoot	Pershing
CKC # 9	88356	Crofoot	Pershing
TRIPLE L # 1	127534	Lewis	Humboldt
TRIPLE L # 2	127535	Lewis	Humboldt
TRIPLE L # 3	127536	Lewis	Humboldt
TRIPLE L # 4	127537	Lewis	Humboldt
TRIPLE L # 5	127538	Lewis	Humboldt
RFG #104	141664	Crofoot	Humboldt
RFG #105	141665	Crofoot	Humboldt
RFG #106	141666	Crofoot	Humboldt
RFG # 107	141667	Crofoot	Pershing
RFG #108	141668	Crofoot	Humboldt
RFG #109	141669	Crofoot	Pershing
RFG #110	141670	Crofoot	Humboldt
RFG #111	141671	Crofoot	Pershing
RFG #112	141672	Crofoot	Humboldt
RFG # 113	141673	Crofoot	Pershing
RFG # 114	141674	Crofoot	Pershing
RFG # 115	141675	Crofoot	Pershing
RFG # 116	141676	Crofoot	Pershing
RFG # 117	141677	Crofoot	Pershing
RFG # 118	141678	Crofoot	Pershing
RFG # 119	141679	Crofoot	Pershing
RFG #120	141680	Lewis	Pershing
RFG #121	141681	Lewis	Pershing
RFG #122	141682	Lewis	Pershing
RFG #123	141683	Lewis	Pershing
RFG #124	141684	Lewis	Pershing
RFG #125	141685	Lewis	Pershing
RFG #127	141686	Lewis	Pershing
RFG #129	141687	Lewis	Pershing

Claim Name	NMC Number	Owner	County
RFG #131	141688	Lewis	Pershing
RFG #132	141689	Lewis	Pershing
RFG #133	141690	Lewis	Pershing
RFG #134	141691	Lewis	Pershing
RFG #135	141692	Lewis	Pershing
RFG #136	141693	Crofoot	Humboldt
RFG #137	141694	Lewis	Pershing
RFG #138	141695	Crofoot	Humboldt
RFG #139	141696	Lewis	Pershing
RFG #140	141697	Crofoot	Humboldt
RFG #141	141698	Lewis	Pershing
RFG # 142	141699	Crofoot	Pershing
RFG #143	141700	Lewis	Pershing
RFG # 144	141701	Crofoot	Pershing
RFG #145	141702	Lewis	Pershing
RFG # 146	141703	Crofoot	Pershing
RFG #147	141704	Lewis	Pershing
RFG #148	141705	Lewis	Pershing
RFG #149	141706	Lewis	Pershing
RFG #150	141707	Lewis	Pershing
RFG #151	141708	Lewis	Pershing
RFG #152	141709	Lewis	Pershing
RFG #153	141710	Lewis	Pershing
RFG #154	141711	Lewis	Pershing
RFG #155	141712	Lewis	Pershing
RFG #156	141713	Lewis	Pershing
RFG #157	141714	Lewis	Pershing
RFG #158	141715	Lewis	Pershing
RFG #159	141716	Lewis	Pershing
RFG #160	141717	Lewis	Pershing
RFG #161	141718	Lewis	Pershing
RFG #162	141719	Lewis	Pershing
RFG #163	141720	Lewis	Pershing
RFG #164	141721	Lewis	Pershing
RFG #165	141722	Lewis	Pershing
RFG #166	141723	Lewis	Pershing
RFG #167	141724	Lewis	Pershing
RFG #200A	141725	Lewis	Pershing
RFG #201A	141726	Lewis	Pershing
RFG #202A	141727	Lewis	Pershing
RFG #203A	141728	Lewis	Pershing
RFG #204A	141729	Lewis	Pershing
RFG #205A	141730	Lewis	Pershing
RFG #206A	141731	Lewis	Pershing

Claim Name	NMC Number	Owner	County
RFG #207A	141732	Lewis	Pershing
RFG #208A	141733	Lewis	Pershing
RFG #209A	141734	Lewis	Pershing
RFG #210A	141735	Lewis	Pershing
RFG #211A	141736	Lewis	Pershing
RFG #212A	141737	Lewis	Pershing
RFG #213A	141738	Lewis	Pershing
RFG #214A	141739	Lewis	Pershing
RFG #215A	141740	Lewis	Pershing
RFG #216A	141741	Lewis	Pershing
RFG #217A	141742	Lewis	Pershing
RFG #218A	141743	Lewis	Pershing
RFG #219A	141744	Lewis	Pershing
RFG #220A	141745	Lewis	Pershing
RFG #221A	141746	Lewis	Pershing
RFG #222A	141747	Lewis	Pershing
RFG #223A	141748	Lewis	Pershing
RFG #224A	141749	Lewis	Pershing
RFG #225A	141750	Lewis	Pershing
RFG #226A	141751	Lewis	Pershing
RFG #227A	141752	Lewis	Pershing
RFG #228	141753	Lewis	Pershing
RFG #228A	141754	Lewis	Pershing
RFG #229	141755	Lewis	Pershing
RFG #229A	141756	Lewis	Pershing
RFG #230	141757	Lewis	Pershing
RFG #230A	141758	Lewis	Pershing
RFG #231	141759	Lewis	Pershing
RFG #231A	141760	Lewis	Pershing
RFG #232A	141761	Lewis	Pershing
RFG #233	141762	Lewis	Pershing
RFG #233A	141763	Lewis	Pershing
RFG #234	141764	Lewis	Pershing
RFG #234A	141765	Lewis	Pershing
RFG #235	141766	Lewis	Pershing
RFG #235A	141767	Lewis	Pershing
RFG #236	141768	Lewis	Pershing
RFG #236A	141769	Lewis	Pershing
RFG #237	141770	Lewis	Pershing
RFG #237A	141771	Lewis	Pershing
RFG #238A	141772	Lewis	Pershing
RFG #239A	141773	Lewis	Pershing
RFG #240A	141774	Lewis	Pershing
RFG #241A	141775	Lewis	Pershing

Claim Name	NMC Number	Owner	County
RFG #250	141776	Lewis	Pershing
RFG #251	141777	Lewis	Pershing
RFG #252	141778	Lewis	Pershing
RFG #253	141779	Lewis	Pershing
RFG #254	141780	Lewis	Pershing
RFG #255	141781	Lewis	Pershing
RFG #256	141782	Crofoot	Humboldt
RFG #257	141783	Lewis	Pershing
RFG #259	141784	Lewis	Pershing
RFG #261	141785	Lewis	Pershing
RFG #263	141786	Lewis	Pershing
RFG # 1	143252	Lewis	Humboldt
RFG # 2	143253	Lewis	Humboldt
RFG # 3	143254	Lewis	Humboldt
RFG # 4	143255	Lewis	Humboldt
RFG # 5	143256	Lewis	Humboldt
RFG # 6	143257	Lewis	Humboldt
RFG # 7	143258	Lewis	Humboldt
RFG # 8	143259	Lewis	Humboldt
RFG # 9	143260	Lewis	Humboldt
RFG # 10	143261	Lewis	Humboldt
RFG # 11	143262	Lewis	Humboldt
RFG # 12	143263	Lewis	Humboldt
RFG # 13	143264	Lewis	Humboldt
RFG # 14	143265	Lewis	Humboldt
RFG # 15	143266	Lewis	Humboldt
RFG # 16	143267	Lewis	Humboldt
RFG # 17	143268	Lewis	Humboldt
RFG # 18	143269	Lewis	Humboldt
RFG # 19	143270	Lewis	Humboldt
RFG # 20	143271	Lewis	Humboldt
RFG # 21	143272	Lewis	Humboldt
RFG # 22	143273	Lewis	Humboldt
RFG # 23	143274	Lewis	Humboldt
RFG # 24	143275	Lewis	Humboldt
RFG # 25	143276	Lewis	Humboldt
RFG # 26	143277	Lewis	Humboldt
RFG # 27	143278	Lewis	Humboldt
RFG # 28	143279	Lewis	Humboldt
RFG # 29	143280	Lewis	Humboldt
RFG # 30	143281	Lewis	Humboldt
RFG # 31	143282	Lewis	Humboldt
RFG # 32	143283	Lewis	Humboldt
RFG # 34	143285	Lewis	Humboldt

Claim Name	NMC Number	Owner	County
RFG # 36	143287	Lewis	Humboldt
RFG # 40	143291	Lewis	Humboldt
RFG # 41	143292	Lewis	Humboldt
RFG # 55	143306	Lewis	Humboldt
RFG # 56	143307	Lewis	Humboldt
RFG # 69	143320	Lewis	Humboldt
RFG # 70	143321	Lewis	Humboldt
RFG #168	143347	Lewis	Humboldt
RFG #169	143348	Lewis	Humboldt
RFG #170	143349	Lewis	Humboldt
RFG #171	143350	Lewis	Humboldt
RFG #172	143351	Lewis	Humboldt
RFG #173	143352	Lewis	Humboldt
RFG #174	143353	Lewis	Humboldt
RFG #175	143354	Lewis	Humboldt
RFG #176	143355	Lewis	Humboldt
RFG #177	143356	Lewis	Humboldt
RFG #178	143357	Lewis	Humboldt
RFG #179	143358	Lewis	Humboldt
RFG #180	143359	Lewis	Humboldt
RFG #181	143360	Lewis	Humboldt
RFG #182	143361	Lewis	Humboldt
RFG #183	143362	Lewis	Humboldt
RFG #184	143363	Lewis	Humboldt
RFG #185	143364	Lewis	Humboldt
RFG #186	143365	Lewis	Humboldt
RFG #187	143366	Lewis	Humboldt
RFG #188	143367	Lewis	Humboldt
RFG #189	143368	Lewis	Humboldt
RFG #190	143369	Lewis	Humboldt
RFG #191	143370	Lewis	Humboldt
RFG #192	143371	Lewis	Humboldt
RFG #193	143372	Lewis	Humboldt
RFG #194	143373	Lewis	Humboldt
RFG #195	143374	Lewis	Humboldt
RFG #196	143375	Lewis	Humboldt
RFG #197	143376	Lewis	Humboldt
RFG #198	143377	Lewis	Humboldt
RFG #199	143378	Lewis	Humboldt
RFG #200	143379	Lewis	Humboldt
RFG #201	143380	Lewis	Humboldt
RFG #202	143381	Lewis	Humboldt
RFG #203	143382	Lewis	Humboldt
RFG #204	143383	Lewis	Humboldt

Claim Name	NMC Number	Owner	County
RFG #205	143384	Lewis	Humboldt
RFG #206	143385	Lewis	Humboldt
RFG #207	143386	Lewis	Humboldt
RFG #208	143387	Lewis	Humboldt
RFG #209	143388	Lewis	Humboldt
RFG #210	143389	Lewis	Humboldt
RFG #211	143390	Lewis	Humboldt
RFG #212	143391	Lewis	Humboldt
RFG #213	143392	Lewis	Humboldt
RFG #214	143393	Lewis	Humboldt
RFG #215	143394	Lewis	Humboldt
RFG #216	143395	Lewis	Humboldt
RFG #217	143396	Lewis	Humboldt
RFG #218	143397	Lewis	Humboldt
RFG #219	143398	Lewis	Humboldt
RFG #220	143399	Lewis	Humboldt
RFG #221	143400	Lewis	Humboldt
RFG #222	143401	Lewis	Humboldt
RFG #223	143402	Lewis	Humboldt
RFG #224	143403	Lewis	Humboldt
RFG #225	143404	Lewis	Humboldt
RFG #226	143405	Lewis	Humboldt
RFG #227	143406	Lewis	Humboldt
RFG #239	143407	Lewis	Humboldt
RFG #240	143408	Lewis	Humboldt
RFG #241	143409	Lewis	Humboldt
RFG #242	143410	Lewis	Humboldt
RFG #243	143411	Lewis	Humboldt
RFG #244	143412	Lewis	Humboldt
RFG #245	143413	Lewis	Humboldt
RFG #246	143414	Lewis	Humboldt
RFG #247	143415	Lewis	Humboldt
RFG #248	143416	Lewis	Humboldt
RFG #264	143417	Lewis	Humboldt
RFG #265	143418	Lewis	Humboldt
RFG #266	143419	Lewis	Humboldt
RFG #267	143420	Lewis	Humboldt
RFG #268	143421	Lewis	Humboldt
RFG #269	143422	Lewis	Humboldt
RFG #270	143423	Lewis	Humboldt
RFG #271	143424	Lewis	Humboldt
RFG #286	143425	Crofoot	Humboldt
RFG #287	143426	Crofoot	Humboldt
RFG #289	143428	Crofoot	Humboldt

Claim Name	NMC Number	Owner	County
RFG #291	143430	Crofoot	Humboldt
RFG #293	143432	Crofoot	Humboldt
RFG #295	143434	Crofoot	Humboldt
RFG #297	143436	Crofoot	Humboldt
RFG #299	143438	Crofoot	Humboldt
RFG #301	143440	Crofoot	Humboldt
RFG #303	143442	Crofoot	Humboldt
RFG #305	143444	Lewis	Humboldt
RFG #306	143445	Lewis	Humboldt
RFG #307	143446	Lewis	Humboldt
RFG #328	143453	Lewis	Humboldt
RFG #330	143455	Lewis	Humboldt
RFG #332	143457	Lewis	Humboldt
RFG #334	143459	Lewis	Humboldt
RFG #336	143461	Lewis	Humboldt
RFG #338	143463	Lewis	Humboldt
RFG #340	143465	Lewis	Humboldt
RFG #342	143467	Lewis	Humboldt
RFG #358	143469	Lewis	Humboldt
RFG #359	143470	Lewis	Humboldt
RFG #360	143471	Lewis	Humboldt
RFG #361	143472	Lewis	Humboldt
RFG #362	143473	Lewis	Humboldt
RFG #363	143474	Lewis	Humboldt
RFG #364	143475	Lewis	Humboldt
RFG #365	143476	Lewis	Humboldt
RFG #366	143477	Lewis	Humboldt
RFG #367	143478	Lewis	Humboldt
RFG #368	143479	Lewis	Humboldt
RFG #102	143481	Crofoot	Humboldt
RFG #126	143482	Crofoot	Humboldt
RFG #128	143483	Crofoot	Humboldt
RFG #130	143484	Lewis	Humboldt
RFG #258	143485	Crofoot	Humboldt
RFG #260	143486	Crofoot	Humboldt
RFG #262	143487	Lewis	Humboldt
RFG #0BF	143488	Lewis	Humboldt
RFG #1FS	143489	Lewis	Humboldt
RFG # 12A	143490	Lewis	Humboldt
RFG # 13A	143491	Lewis	Humboldt
RFG # 22A	143492	Lewis	Humboldt
RFG # 29A	143493	Lewis	Humboldt
RFG # 29B	143494	Lewis	Humboldt
RFG # 30A	143495	Lewis	Humboldt

Claim Name	NMC Number	Owner	County
RFG # 36A	143496	Lewis	Humboldt
RFG # 36B	143497	Lewis	Humboldt
RFG # 94A	143503	Crofoot	Humboldt
RFG #201A	143504	Lewis	Humboldt
RFG #215B	143505	Lewis	Humboldt
RFG #217B	143506	Lewis	Humboldt
RFG #218A	143507	Lewis	Humboldt
RFG #218B	143508	Lewis	Humboldt
RFG #219B	143509	Lewis	Humboldt
RFG #238F	143510	Lewis	Humboldt
RFG #239A	143511	Lewis	Humboldt
RFG #362A	143512	Lewis	Humboldt
RFG #364	143513	Lewis	Humboldt
RFG #366A	143514	Lewis	Humboldt
RFG #368A	143515	Lewis	Humboldt
RFG #241A	143596	Lewis	Humboldt
RFG #240	143597	Lewis	Humboldt
RFG #239	143598	Lewis	Humboldt
RFG #400	175062	Lewis	Humboldt
RFG #401	175063	Lewis	Humboldt
RFG #402	175064	Lewis	Humboldt
RFG #403	175065	Lewis	Humboldt
RFG #404	175066	Lewis	Humboldt
RFG #405	175067	Lewis	Humboldt
RFG #406	175068	Lewis	Humboldt
RFG #407	175069	Lewis	Humboldt
RFG #408	175070	Lewis	Humboldt
RFG #409	175071	Lewis	Humboldt
RFG #410	175072	Lewis	Humboldt
RFG #411	175073	Lewis	Humboldt
RFG #412	175074	Lewis	Humboldt
RFG #413	175075	Lewis	Humboldt
RFG #414	175076	Lewis	Humboldt
RFG #415	175077	Lewis	Humboldt
RFG #416	175078	Lewis	Humboldt
RFG #417	175079	Lewis	Humboldt
RFG #418	175080	Lewis	Humboldt
RFG #419	175081	Lewis	Humboldt
RFG #420	175082	Lewis	Humboldt
RFG #421	175083	Lewis	Humboldt
RFG #422	175084	Lewis	Humboldt
RFG #423	175085	Lewis	Humboldt
RFG #424	175086	Lewis	Humboldt
RFG #425	175087	Lewis	Humboldt

Claim Name	NMC Number	Owner	County
RFG #426	175088	Lewis	Humboldt
RFG #427	175089	Lewis	Humboldt
PACIFIC	181010	Lewis	Humboldt
SULPHATE	181011	Lewis	Humboldt
ALUNITE	181012	Lewis	Humboldt
ALUNITE # 2	181013	Lewis	Humboldt
DIA # 1	284248	Lewis	Humboldt
DIA # 2	284249	Lewis	Humboldt
DIA # 3	284250	Lewis	Humboldt
DIA # 4	284251	Lewis	Humboldt
DIA # 5	284252	Lewis	Humboldt
RFG #328X	307553	Lewis	Humboldt
RFG # 39	436884	Lewis	Humboldt
RFG # 72	436912	Lewis	Humboldt
CKC # 12	444109	Crofoot	Humboldt
CKC # 15	444112	Crofoot	Humboldt
BLACKROCK # 2	545996	Crofoot	Humboldt
MAYO	545997	Crofoot	Humboldt
ANITA	545998	Crofoot	Humboldt
ASHLODE	545999	Crofoot	Humboldt
ALBERT	546000	Crofoot	Humboldt
CKC # 10	546001	Crofoot	Humboldt
CKC # 11	546002	Crofoot	Humboldt
CKC # 13	546003	Crofoot	Humboldt
CKC # 14	546004	Crofoot	Humboldt
RFG # 33	546005	Crofoot	Humboldt
RFG # 35	546006	Crofoot	Humboldt
RFG # 37	546007	Crofoot	Humboldt
RFG # 38	546008	Crofoot	Humboldt
RFG # 39A	546009	Crofoot	Humboldt
RFG # 42	546010	Crofoot	Humboldt
RFG # 43	546011	Crofoot	Humboldt
RFG # 44	546012	Crofoot	Humboldt
RFG # 45	546013	Crofoot	Humboldt
RFG # 46	546014	Crofoot	Humboldt
RFG # 47	546015	Crofoot	Humboldt
RFG # 48	546016	Crofoot	Humboldt
RFG # 49	546017	Crofoot	Humboldt
RFG # 50	546018	Crofoot	Humboldt
RFG # 51	546019	Crofoot	Humboldt
RFG # 52	546020	Crofoot	Humboldt
RFG # 52A	546021	Crofoot	Humboldt
RFG # 53	546022	Crofoot	Humboldt
RFG # 54	546023	Crofoot	Humboldt

Claim Name	NMC Number	Owner	County
RFG # 57	546024	Crofoot	Humboldt
RFG # 58	546025	Crofoot	Humboldt
RFG # 59	546026	Crofoot	Humboldt
RFG # 60	546027	Crofoot	Humboldt
RFG # 61	546028	Crofoot	Humboldt
RFG # 62	546029	Crofoot	Humboldt
RFG # 63	546030	Crofoot	Humboldt
RFG # 64	546031	Crofoot	Humboldt
RFG # 65	546032	Crofoot	Humboldt
RFG # 66	546033	Crofoot	Humboldt
RFG # 67	546034	Crofoot	Humboldt
RFG # 67A	546035	Crofoot	Humboldt
RFG # 68	546036	Crofoot	Humboldt
RFG # 68A	546037	Crofoot	Humboldt
RFG # 71	546038	Crofoot	Humboldt
RFG # 73	546039	Crofoot	Humboldt
RFG # 74	546040	Crofoot	Humboldt
RFG # 75	546041	Crofoot	Humboldt
RFG # 76	546042	Crofoot	Humboldt
RFG # 77	546043	Crofoot	Humboldt
RFG # 78	546044	Crofoot	Humboldt
RFG # 79	546045	Crofoot	Humboldt
RFG # 80	546046	Crofoot	Humboldt
RFG # 81	546047	Crofoot	Humboldt
RFG # 81A	546048	Crofoot	Humboldt
RFG # 82	546049	Crofoot	Humboldt
RFG # 83	546050	Crofoot	Humboldt
RFG # 84	546051	Crofoot	Humboldt
RFG # 85	546052	Crofoot	Humboldt
RFG # 86	546053	Crofoot	Humboldt
RFG # 87	546054	Crofoot	Humboldt
RFG # 88	546055	Crofoot	Humboldt
RFG # 89	546056	Crofoot	Humboldt
RFG # 90	546057	Crofoot	Humboldt
RFG # 91	546058	Crofoot	Humboldt
RFG # 92	546059	Crofoot	Humboldt
RFG # 93	546060	Crofoot	Humboldt
RFG # 94	546061	Crofoot	Humboldt
RFG # 95	546062	Crofoot	Humboldt
RFG # 97	546063	Crofoot	Humboldt
RFG # 99	546064	Crofoot	Humboldt
RFG #101	546065	Crofoot	Humboldt
RFG #103	546066	Crofoot	Humboldt
RFG #288	546067	Crofoot	Humboldt

Claim Name	NMC Number	Owner	County
RFG #290	546068	Crofoot	Humboldt
RFG #292	546069	Crofoot	Humboldt
RFG #294	546070	Crofoot	Humboldt
RFG #296	546071	Crofoot	Humboldt
RFG #298	546072	Crofoot	Humboldt
RFG #300	546073	Crofoot	Humboldt
RFG #302	546074	Crofoot	Humboldt
RFG #304	546075	Crofoot	Humboldt
RFG #322	546076	Crofoot	Humboldt
RFG #323	546077	Crofoot	Humboldt
RFG #324	546078	Crofoot	Humboldt
RFG #325	546079	Crofoot	Humboldt
RFG #326	546080	Crofoot	Humboldt
RFG #327	546081	Crofoot	Humboldt
RFG #329	546082	Crofoot	Humboldt
RFG #331	546083	Crofoot	Humboldt
RFG #333	546084	Crofoot	Humboldt
RFG #335	546085	Crofoot	Humboldt
RFG #337	546086	Crofoot	Humboldt
RFG #339	546087	Crofoot	Humboldt
RFG #341	546088	Crofoot	Humboldt
RFG #343	546089	Crofoot	Humboldt
WRC-1	714252	Lewis	Pershing
WRC-2	714253	Lewis	Pershing
WRC-3	714254	Lewis	Pershing
WRC-4	714255	Lewis	Pershing
WRC-5	714256	Lewis	Pershing
WRC-6	714257	Lewis	Pershing
WRC-7	714258	Lewis	Pershing
WRC-8	714259	Lewis	Pershing
WRC-9	714260	Lewis	Pershing
WRC-10	714261	Lewis	Pershing
WRC-11	714262	Lewis	Pershing
WRC-12	714263	Lewis	Pershing
WRC-13	714264	Lewis	Pershing
WRC-14	714265	Lewis	Pershing
WRC-15	714266	Lewis	Pershing
WRC-16	714267	Lewis	Pershing
WRC-17	714268	Lewis	Pershing
WRC-18	714269	Lewis	Pershing
WRC-19	714270	Lewis	Pershing
WRC-20	714271	Lewis	Pershing
WRC-21	714272	Lewis	Pershing
WRC-22	714273	Lewis	Pershing

Claim Name	NMC Number	Owner	County
WRC-23	714274	Lewis	Pershing
WRC-24	714275	Lewis	Pershing
WRC-25	714276	Lewis	Pershing
WRC-26	714277	Lewis	Pershing
WRC-27	714278	Lewis	Pershing
WRC-28	714279	Lewis	Pershing
WRC-29	714280	Lewis	Pershing
WRC-30	714281	Lewis	Pershing
WRC-31	714282	Lewis	Pershing
WRC-32	714283	Lewis	Pershing
WRC-33	714284	Lewis	Pershing
WRC-34	714285	Lewis	Pershing
WRC-35	714286	Lewis	Pershing
WRC-36	714287	Lewis	Pershing
WRC-37	714288	Lewis	Pershing
WRC-38	714289	Lewis	Pershing
WRC-39	714290	Lewis	Pershing
WRC-40	714291	Lewis	Pershing
WRC-41	714292	Lewis	Pershing
WRC-42	714293	Lewis	Pershing
WRC-43	714294	Lewis	Pershing
WRC-44	714295	Lewis	Pershing
WRC-45	714296	Lewis	Pershing
WRC-46	714297	Lewis	Pershing
WRC-47	714298	Lewis	Pershing
WRC-48	714299	Lewis	Pershing
WRC-49	714300	Lewis	Pershing
WRC-50	714301	Lewis	Pershing
WRC-51	714302	Lewis	Pershing
WRC-52	714303	Lewis	Pershing
WRC-53	714304	Lewis	Pershing
WRC-54	714305	Lewis	Pershing
WRC-55	714306	Lewis	Pershing
WRC-56	714307	Lewis	Pershing
WRC-57	714308	Lewis	Pershing
WRC-58	714309	Lewis	Pershing
WRC-60	714311	Lewis	Pershing
WRC-82	714313	Lewis	Pershing
WRC-84	714315	Lewis	Pershing
WRC-87	714317	Lewis	Pershing
WRC-88	714318	Lewis	Pershing
WRC-89	714319	Lewis	Pershing
WRC-90	714320	Lewis	Pershing
WRC-91	714321	Lewis	Pershing

Claim Name	NMC Number	Owner	County
WKM-1	780688	Lewis	Humboldt
WKM-2	780689	Lewis	Humboldt
WKM-3	780690	Lewis	Humboldt
WKM-4	780691	Lewis	Humboldt
WKM-5	780692	Lewis	Humboldt
WKM-6	780693	Lewis	Humboldt
WKM-7	780694	Lewis	Humboldt
WKM-8	780695	Lewis	Humboldt
WKM-9	780696	Lewis	Humboldt
WKM-10	780697	Lewis	Humboldt
WKM-11	780698	Lewis	Humboldt
WKM-12	780699	Lewis	Humboldt
WKM-13	780700	Lewis	Humboldt
WKM-14	780701	Lewis	Humboldt
WKM-15	780702	Lewis	Humboldt
WKM-16	780703	Lewis	Humboldt
WKM-17	780704	Lewis	Humboldt
WKM-18	780705	Lewis	Humboldt
WKM-19	780706	Lewis	Humboldt
WKM-20	780707	Lewis	Humboldt
WKM-21	780708	Lewis	Humboldt
WKM-22	780709	Lewis	Humboldt
WKM-23	780710	Lewis	Humboldt
WKM-24	780711	Lewis	Humboldt
WKM-25	780712	Lewis	Humboldt
WKM-26	780713	Lewis	Humboldt
WKM-27	780714	Lewis	Humboldt
WKM-28	780715	Lewis	Humboldt
WKM-29	780716	Lewis	Humboldt
WKM-30	780717	Lewis	Humboldt
WKM-31	780718	Lewis	Humboldt
WKM-32	780719	Lewis	Humboldt
WKM-33	780720	Lewis	Humboldt
WKM-34	780721	Lewis	Humboldt
WKM-35	780722	Lewis	Humboldt
WKM-36	780723	Lewis	Humboldt
WKM-37	780724	Lewis	Humboldt
WKM-38	780725	Lewis	Humboldt
WKM-39	780726	Lewis	Humboldt
WKM-40	780727	Lewis	Humboldt
WKM-41	780728	Lewis	Humboldt
WKM-42	780729	Lewis	Humboldt
WKM-43	780730	Lewis	Humboldt
WKM-44	780731	Lewis	Humboldt

Claim Name	NMC Number	Owner	County
WKM-45	780732	Lewis	Humboldt
WKM-46	780733	Lewis	Humboldt
WKM-47	780734	Lewis	Humboldt
WKM-48	780735	Lewis	Humboldt
WKM-50	780736	Lewis	Humboldt
WKM-51	780737	Lewis	Humboldt
WKM-52	780738	Lewis	Humboldt
WKM-53	780739	Lewis	Humboldt
WKM-54	780740	Lewis	Humboldt
WKM-55	780741	Lewis	Humboldt
WKM-56	780742	Lewis	Humboldt
WKM-57	780743	Lewis	Humboldt
WKM-58	780744	Lewis	Humboldt
WKM-60	780745	Lewis	Humboldt
WKM-62	780746	Lewis	Humboldt
WKM-64	780747	Lewis	Humboldt

APPENDIX B

DRILL HOLE DATABASE (CD)