PRELIMINARY ECONOMIC ASSESSMENT

on the

ROBERTSON PROPERTY

Lander County,
Nevada, USA

Prepared for

CORAL GOLD RESOURCES LTD.
Vancouver, B.C. Canada

Prepared by

Beacon Hill Consultants (1988) Ltd.

in conjunction with

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and
Kaehne Consulting Ltd.

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TABLE OF CONTENTS

List of Tables

List of Figures

SECTION 1.0 SUMMARY

1.1 SCOPE

A **Preliminary Economic Assessment** (**PEA**) has been prepared, to meet the NI 43-101 criteria for a technical report on the Robertson property by **Beacon Hill Consultants (1988) Ltd. (Beacon Hill)** on behalf of the property owners, **Coral Gold Resources Ltd. (Coral).**

1.2 PROJECT DESCRIPTION

The Robertson Property is an advanced-stage gold exploration project located in eastern Lander County, Nevada, 60 miles southwest of Elko. Coral Resources, Inc., a subsidiary of Coral Gold Resources Limited of Vancouver, B. C., acquired control of the Robertson Property in 1986. The property consists of 601 unpatented federal lode claims, mill sites, placer claims and nine patented lode claims covering over 7,300 acres of public lands administered by the Bureau of Land Management. Coral is record owner of 525 claims and controls an additional 76 claims through a series of mineral leases and option agreements.

Site location and claim map are shown on Figure 1-1 and 1-2 respectively.

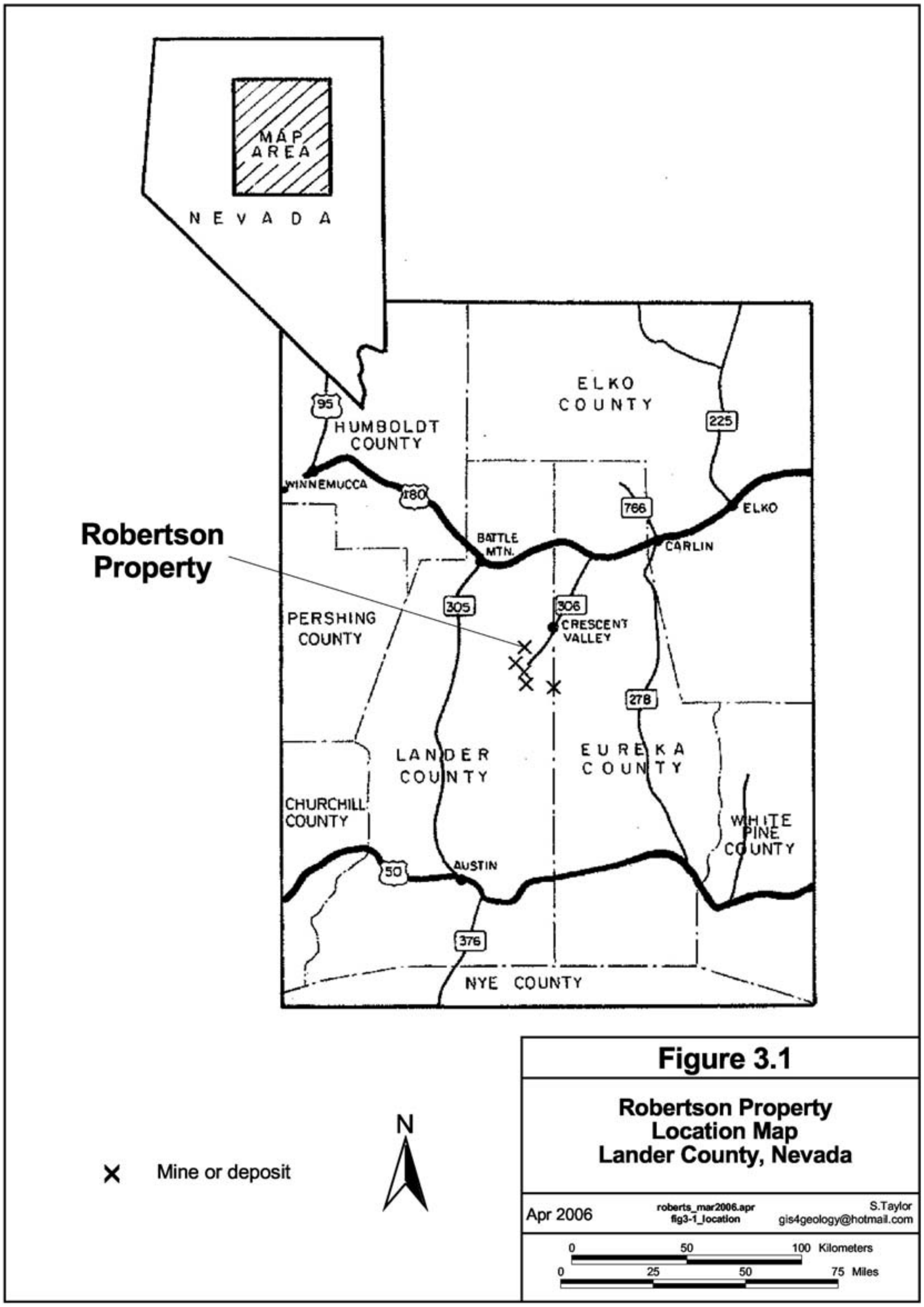

Figure 1-1: Location Map[1]

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[1] All figure numbers are shown at the bottom of each figure. The figure number contained within the figure should be disregarded.

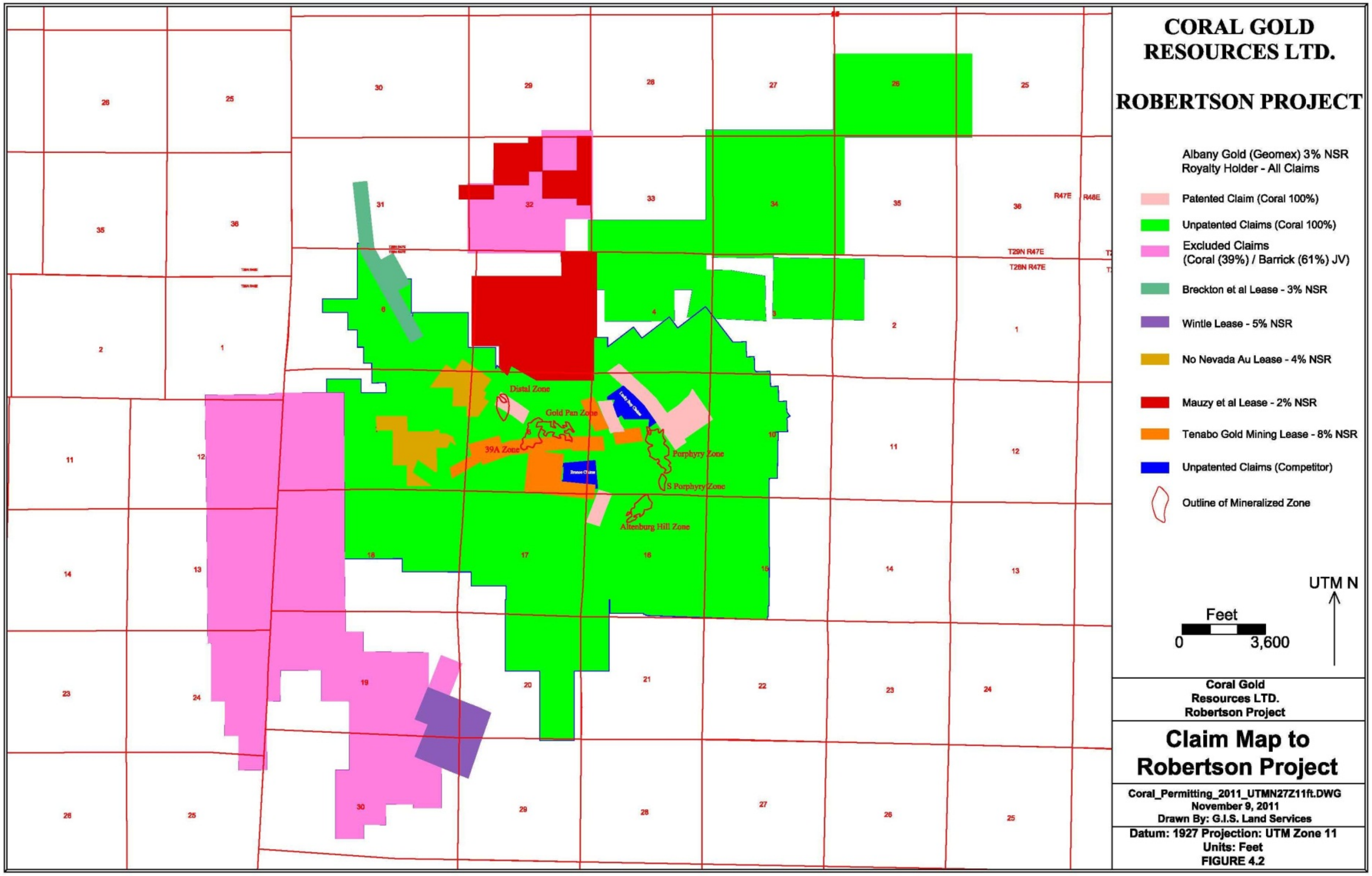


Figure 1-2: Claim Map

1.3 RESOURCES

The inferred resources estimated, as shown on Table 1.1, for the Altenburg Hill, Porphyry and Gold Pan zones is based upon a cut-off grade of 0.0147 ozAu/t for each zone termed as higher grade and 0.005 ozAu/t for lower grade material and upon the following open pit criteria;

- Ore mining cost of US$1.27 per ton[2];
- Waste mining cost of US$1.43 per ton;
- Processing cost of US$6.25 per ton;
- Metallurgical recovery of 70%;
- Gold price of US$1,000 per ounce;
- Tonnage factor of 12.2 cubic foot per ton;
- Pit slope angles of 45 degrees
- Bench interval 20 feet at double benches for a total of 40 feet
- Berm width 25 feet
- Slope angle bench 70 degrees
- Roadway gradient 8%
- Roadway width 105 feet.

Table 1.1: Inferred Resources for Robertson[3]

	Higher Grade			Lower Grade			Total		
Zone	**Quantity**	**Grade**	**Gold**	**Quantity**	**Grade**	**Gold**	**Quantity**	**Grade**	**Gold**
	Tons	**ozAu/t**	**Oz**	**Tons**	**ozAu/t**	**Oz**	**Tons**	**ozAu/t**	**Oz**
Altenburg	5,557,572	0.0189	105,038	13,810,677	0.0106	146,749	19,368,249	0.0130	251,787
Porphyry	17,337,872	0.0209	362,362	28,888,371	0.0103	296,479	46,226,243	0.0143	658,840
Gold Pan	3,613,310	0.0223	80,577	9,001,014	0.0100	89,717	12,614,324	0.0135	170,293
Total	**26,508,754**	**0.0207**	**547,976**	**51,700,062**	**0.0103**	**532,945**	**78,208,816**	**0.0138**	**1,080,921**

1.4 MINE PLAN

The mine plan as described in this report evaluates the open pit mining of three deposits of the Robertson property, the Altenburg Hill, Porphyry and Gold Pan zones. These deposits contain mineralization which is considered based upon metallurgical work to have cyanide heap leach potential. The estimated inferred resources at a cut-off of 0.005ozAu/t are some 78.2 million tons containing 1.08 million oz of gold. Based upon cut-off of 0.0147ozAu/t for a crushed higher grade heap leach recovery of 67% and a run-off-mine lower grade heap recovery of 45%, the recovered ozs are 604,100 oz. The mine life has been estimated at 10.5 years at a maximum production rate of approximately 40,000 tons per day for both mineral and waste. The waste to mineral strip ratio is 0.6:1.

The production rate was determined to establish a life of at least 8 years and no more than 15 years to minimize capital and operating costs and provide an acceptable yearly gold production and return on investment. It was determined that to meet these goals it was beneficial to leach the higher grade

[2] Since this is a property located in the USA all units are shown in imperial unless specifically noted as otherwise.

[3] Due to the uncertainty that may be attached to an inferred mineral resource, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured resource as a result of continued exploration.

separate from the lower grade. The schedule was optimized by mining the zones that would provide the lowest strip ratio, Altenburg Hill together with the larger Porphyry first and then Gold Pan. The production schedule for each deposit and the combined rates is shown on Table 1.2.

Table 1.2: Production Schedule

	Year												
Altenburg	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**Total**
Ore	375,042	1,003,340	1,554,611	1,363,935	990,942	252,602	17,100						**5,557,572**
Low Grade	2,488,073	3,014,970	2,387,223	3,106,209	2,121,969	600,277	91,957						**13,810,678**
Waste	655,810	1,810,154	2,436,546	2,249,406	665,707	188,525	133						**8,006,281**
Ore/day	8,180	11,481	11,262	12,772	8,894	2,437	312						**9,223**
Total/day	**10,054**	**16,653**	**18,224**	**19,199**	**10,796**	**2,975**	**312**						**13,035**
Strip Ratio	0.23	0.45	0.62	0.50	0.21	0.22	0.00						**0.41**
Insitu Oz	32,104	49,550	57,944	57,630	41,288	11,817	1,243						**251,577**
Recv oz	17,940	27,689	32,379	32,204	23,072	6,604	695						**140,581**
Grade HG	0.017	0.019	0.021	0.018	0.017	0.021	0.017						**0.019**
Grade LG	0.010	0.010	0.010	0.011	0.011	0.011	0.010						**0.011**
Porphyry													**Total**
Ore	64,150	432,602	1,132,510	834,539	1,885,615	2,650,810	3,511,097	4,511,743	2,114,437	200,369			**17,337,872**
Low Grade	952,827	2,525,769	2,076,814	3,581,568	3,934,335	5,652,520	5,372,632	3,500,143	1,117,418	174,345			**28,888,371**
Waste	5,583,903	4,120,102	3,243,186	3,042,152	4,872,920	5,122,511	3,963,411	1,679,417	357,562	154,651			**32,139,815**
Ore/day	2,906	8,452	9,169	12,617	16,628	23,724	25,382	22,891	9,234	1,071			**14,675**
Total/day	**18,860**	**20,224**	**18,436**	**21,309**	**30,551**	**38,360**	**36,706**	**27,689**	**10,255**	**1,512**			**24,878**
Strip Ratio	5.49	1.39	1.01	0.69	0.84	0.62	0.45	0.21	0.11	0.41			**0.70**
Insitu Oz	9,581	32,616	43,240	52,141	77,298	109,159	127,635	140,139	60,506	6,525			**658,840**
Recv oz	5,354	18,226	24,163	29,137	43,194	60,998	71,322	78,310	33,811	3,646			**368,160**
Grade HG	0.018	0.019	0.020	0.020	0.020	0.019	0.020	0.023	0.023	0.023			**0.020**
Grade LG	0.009	0.010	0.010	0.010	0.010	0.010	0.011	0.011	0.011	0.011			**0.010**
Gold Pan													**Total**
Ore								83,935	581,363	2,296,025	651,987		**3,613,310**
Low Grade								1,048,566	2,555,133	4,153,391	1,243,925		**9,001,015**
Waste								2,735,184	2,999,262	1,197,294	44,764		**6,976,504**
Ore/day								3,236	8,961	18,427	5,417		**8,009**
Total/day								**11,051**	**17,531**	**21,848**	**5,545**		**12,439**
Strip Ratio								2.42	0.96	0.19	0.02		**0.55**
Insitu Oz								11,273	35,641	97,174	26,555		**170,642**
Recv oz								6,299	19,916	54,301	14,839		**95,355**
Grade HG								0.019	0.018	0.024	0.021		**0.022**
Grade LG								0.009	0.010	0.010	0.010		**0.010**
Totals													**Total**
Ore	439,192	1,435,942	2,687,121	2,198,474	2,876,557	2,903,412	3,528,197	4,595,678	2,695,800	2,496,394	651,987		**26,508,754**
Low Grade	3,440,900	5,540,739	4,464,037	6,687,777	6,056,304	6,252,797	5,464,589	4,548,709	3,672,551	4,327,736	1,243,925		**51,700,064**
Waste	6,239,713	5,930,256	5,679,732	5,291,558	5,538,627	5,311,036	3,963,544	4,414,601	3,356,824	1,351,945	44,764		**47,122,600**
Ore/day	11,086	19,933	20,432	25,389	25,522	26,161	25,694	26,127	18,195	19,498	5,417		**21,281**
Total/day	**28,914**	**36,877**	**36,660**	**40,508**	**41,347**	**41,335**	**37,018**	**38,740**	**27,786**	**23,360**	**5,545**		**34,104**
Strip Ratio	1.61	0.85	0.79	0.60	0.62	0.58	0.44	0.48	0.53	0.20	0.02		**0.60**
Insitu Oz	41,686	82,167	101,184	109,772	118,586	120,977	128,878	151,412	96,146	103,698	26,555		**1,080,921**
Recv oz	23,294	45,915	56,542	61,340	66,266	67,602	72,017	84,609	53,727	57,947	14,839		**604,096**
Grade HG	0.017	0.019	0.021	0.019	0.019	0.019	0.020	0.023	0.022	0.024	0.021		**0.021**
Grade LG	0.010	0.010	0.010	0.010	0.011	0.010	0.011	0.010	0.010	0.010	0.010		**0.010**

1.5 PROCESSING AND METALLURGY

A considerable amount of metallurgical work has been completed over a number of years for forecasting leach recoveries for the Altenburg Hill, Porphyry and Gold Pan mineralized zones. In 2011 further laboratory test work was performed by McClelland Laboratories, Inc., Sparks, Nevada based upon samples from the Altenburg Hill and Gold Pan Deposits. No recent metallurgical test work has been done on the Porphyry deposit.

In this study, based on previous work and the recent work completed by McClelland Laboratories recoveries have been assumed as follows;

1. Mineralization that grades 0.0147ozAu/t or higher that will be crushed prior to being placed on the leach pad - 67% leach recovery;
2. Run-off-mine mineralization that has a grade lower than 0.0147ozAu/t and a cut-of 0.005ozAu/t – 45% recovery.

It should be noted that considerable further work is required to confirm the foregoing leach recoveries and they should be considered as indicative only.

1.6 INFRASTRUCTURE

The general arrangement drawing, which includes the open pits, haul roads, leach pads, pipelines, ponds, offices, dry, ADR plant, warehouse and maintenance facilities is shown on Figure 1-3.

1.7 ENVIRONMENTAL

The Project area consists of approximately 7300 acres, of which 169 acres are private lands, held as patented mining claims either owned or controlled by Coral Gold Resources, Inc. (Coral). The remaining 7131 acres are public lands administered by the BLM Mount Lewis Field Office (MLFO) in Lander County; Coral controls approximately 601 unpatented lode and placer claims on these public lands.

This mixed estate makes the MLFO the primary agency for authorizing mining activities on public and private land; the MLFO works with the Nevada Division of Environmental Protection – Bureau of Mining Regulation and Reclamation (BMRR) under a memorandum of understanding to authorize mining on projects on both public and private lands.

As a result of the foregoing there are a number of environmental evaluations, reports and approvals required to allow for work to be done on the property.

The first of these is an Environmental Assessment (EA) which upon approval by the regulatory authorities will provide the basis for work to commence on the property. It is expected that the EA, being performed by SRK Elko Nevada, will be completed by January 2012 and approval achieved during the third quarter 2012. Coral have indicated that they expect a number of drill holes to be approved prior to approval of the EA such that drilling can commence in the second quarter 2012. The EA approval will allow for the further drilling and the removal of samples for laboratory metallurgical test work and on-site bulk leach program to confirm laboratory results.

The second study will be site environmental work to provide data for an EIS based upon the proposed mine plan for the property.

These studies and associated approvals are the critical timeline for the project and any improvement to shorten this timeline will be beneficial to bringing the project on stream as soon as feasible. The schedule of these activities is shown on Figure 1-4.

1.8 DEVELOPMENT SCHEDULE

An overall development schedule of activities has been derived. The schedule provides a two stage approach to the development of the project; the first stage is the work program required to advance the project to prefeasibility. This program on the Altenburg Hill, Porphyry and Gold Pan deposits includes exploration and drill definition activities, metallurgical test work, environmental studies, geotechnical open pit and infrastructure test work and evaluations, infrastructure requirements, and capital and operating costs. The work is intended to upgrade the level of confidence in the resource estimates to measured and indicated categories and based on prefeasibility cost estimates move the resources into reserves.

It should be noted;

- there is no guarantee that resources will be designated as measured and indicated and that reserves will be designated on the Robertson property; and
- while the schedule indicates a timeline for the overall project, the period after the completion of a prefeasibility study is subject to the results of that study.

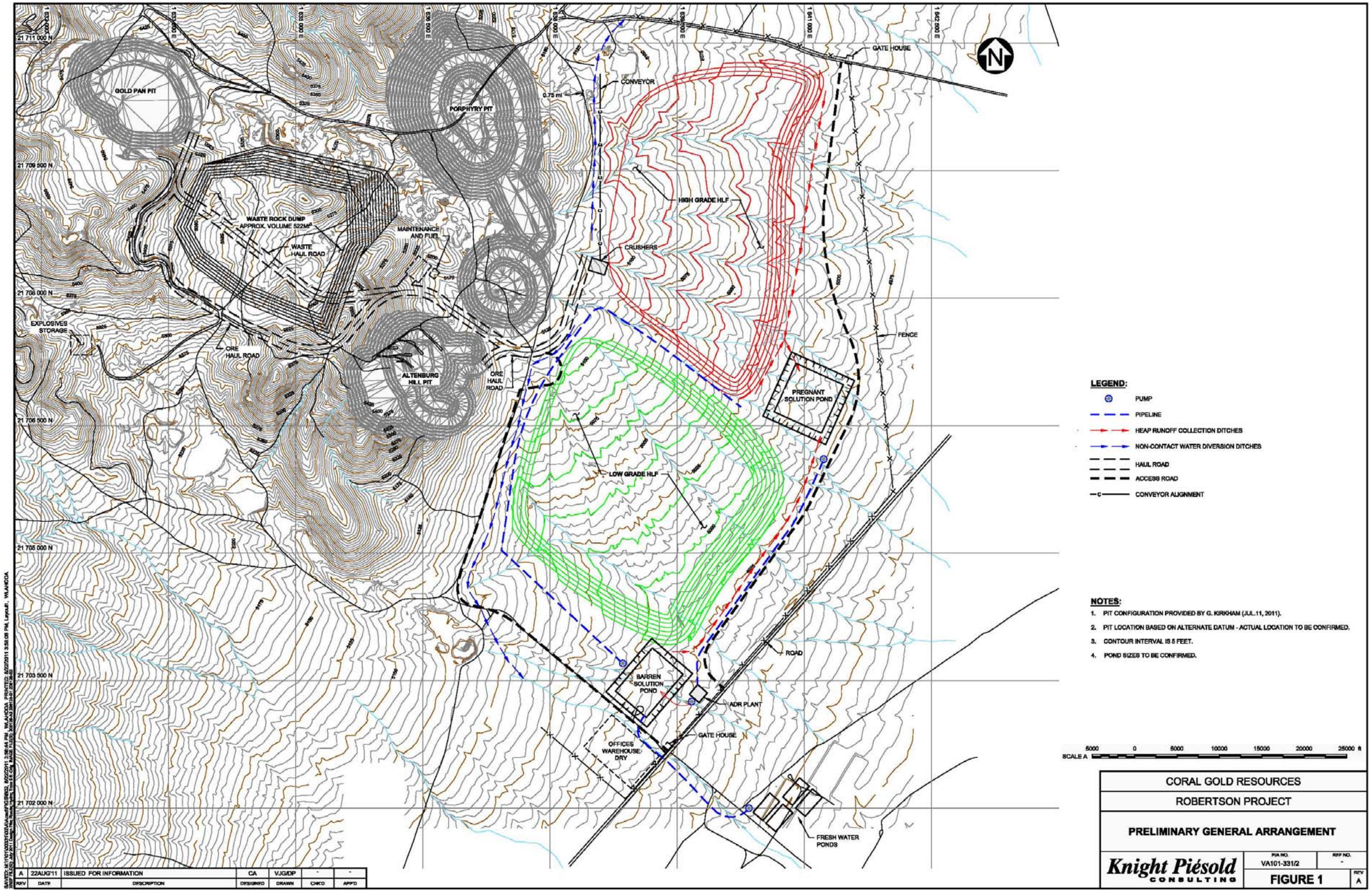


Figure 1-3: General Arrangement

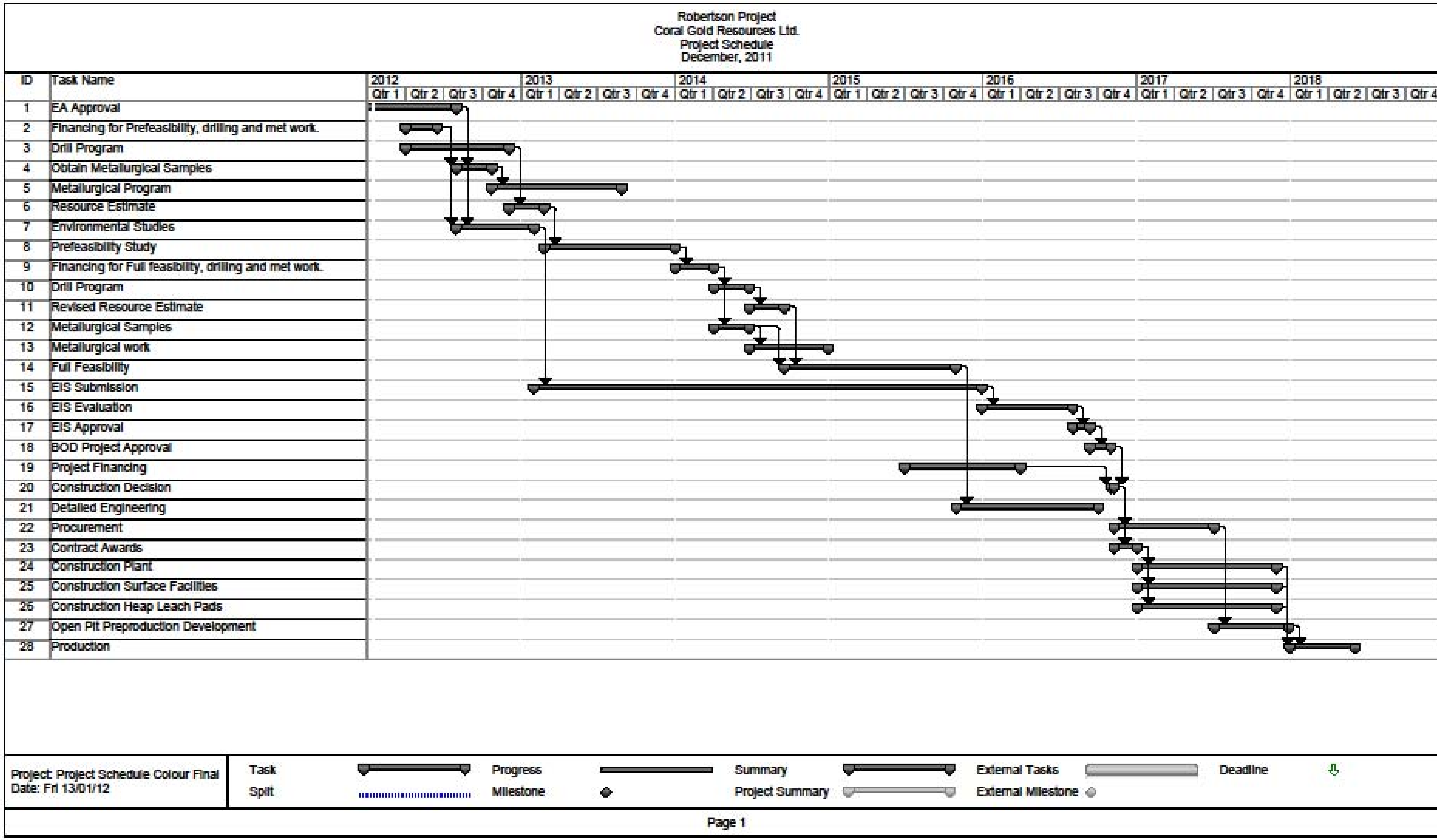


Figure 1-4: Development Schedule

1.9 CAPITAL COSTS

The project initial capital costs, based upon 2011 dollars and Owner operation, to construct the facilities have been estimated at $122.1 million with expenditures of $16.5 million to develop the property. Ongoing costs are estimated at $54.2 million. The initial capital cost schedule is shown in Table 1.3 and ongoing cost Table 1.4 and is based upon budget estimates from suppliers for major items, cost data from Mining Cost Service, InfoMine USA, Inc. and in-house data. These costs are preliminary and should be considered as indicative.

An alternative case has been developed using a Nevada Contractor to develop and mine the open pits with the Owner operating all other facilities. The initial capital cost for this approach is $97 million with ongoing capital of $26.1 million. The initial capital cost is shown in Table 1.5 and the ongoing cost Table 1.6. Under this alternative the Contractor provides all the mining associated equipment which reduces the initial capital expenditures.

The capital costs are intended to reflect those costs that Coral would incur to further explore the Robertson property, complete environmental and permitting studies, obtain permits, environmental bonding of the property, purchase of all equipment, buildings and supplies, construct all facilities, preproduction development, commissioning all facilities, construct all surface infrastructure including leach pads and waste disposal areas and commence production.

Coral have an agreement with Albany Gold Corp. on all the claims on which the Robertson property sits. This agreement calls for a 3% NSR to be paid or a purchase payment of $1.25 million paid at any time to purchase outright the 3% NSR on the claims. It is understood from Coral representatives that this purchase would be made upon Coral making a construction decision. Thus this cost has been added to the capital cost estimate. Tenabo Gold Mines have a royalty on a claim that slightly covers non mineable Gold Pan mineralization.

Environmental bonding will be required by the USA and Nevada regulators prior to obtain the permit(s) to commence construction of the project. An allowance of $7.5 million has been added to the capital cost of the project to cover this item, based upon bonding requirements to those experienced for similar projects in Nevada. At the end of the life of the mine reclamation will be required and this has been estimated at a similar cost to the bond, $7 million. In addition if the reclamation is completed then the bonding would be returned to the company and this is included in the financials.

Salvage of equipment can be expected for the mobile equipment, ARD and crusher plant. The capital cost for these are some $45 million. A salvage allowance of 15% has been used amounting to $6.75 million.

The capital costs include allowances for Engineering, Procurement, Construction Management, Construction Indirects, Freight, Start-up and Commissioning and First Fills and Capital Spares. A contingency of 15% has been added to the estimated capital cost.

Ongoing capital costs are shown separately and include ongoing construction of the leach pads and replacement equipment that are used for maintaining operations.

Table 1.3: Capital Cost Summary $(000)'s (Owner Operated)

Description		2012	2013	2014	2015	2016	2017
Project Development		$4,157	$3,012	$3,076	$2,525	$1,616	$1,334
Plant and Surface Facilities							$40,758
Mining							$31,102
Indirect Costs							$22,399
Contingency	15.00%	$624	$452	$461	$379	$242	$14,946
Total Initial Capital		**$4,780**	**$3,464**	**$3,538**	**$2,903**	**$1,859**	**$114,583**

In addition there is a Bond of $7.5 million which gives a total of $122.1 million.

Table 1.4: Capital Cost Summary $(000)'s (Contractor Operated)

Description		2012	2013	2014	2015	2016	2017
Project Development		$4,157	$3,012	$3,076	$2,525	$1,616	$1,334
Plant and Surface Facilities							$40,758
Mining							$14,159
Indirect Costs							$17,486
Contingency	15.00%	$624	$452	$461	$379	$242	$11,667
Total Initial Capital		**$4,780**	**$3,464**	**$3,538**	**$2,903**	**$1,859**	**$89,449**

In addition there is a Bond of $7.5 million which gives a total of $97 million.

Table 1.5: Ongoing Capital Cost Summary $(000)'s (Owner Operated)

		Year												
Description	**%**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**Total**
Leach Pads			$3,122	$3,122	$3,122	$2,943	$2,943	$1,472						**$16,724**
Replacement vehicles	5.00		$1,175	$1,175	$1,175	$1,175	$16,175	$1,175	$1,175	$1,175	$1,175			**$25,579**
Reclamation													$7,000	**$7,000**
Contingency	10.00		$430	$430	$430	$412	$1,912	$265	$117	$117	$117		$700	**$4,930**
Total			**$4,727**	**$4,727**	**$4,727**	**$4,530**	**$21,030**	**$2,912**	**$1,292**	**$1,292**	**$1,292**		**$7,700**	**$54,233**

Table 1.6: Ongoing Capital Cost Summary $(000)'s (Contractor Operated)

		Year												
Description	**%**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**Total**
Reclamation													$7,000	**$7,000**
Contingency	10.00		$312	$312	$312	$294	$294	$147					$700	**$2,372**
Total			**$3,434**	**$3,434**	**$3,434**	**$3,237**	**$3,237**	**$1,619**					**$7,700**	**$26,096**

1.10 OPERATING COSTS

The operating costs have been estimated based upon prevailing labour rates for Nevada and the expected costs for supplies and materials. Efficiencies of equipment have been based upon costs developed by the suppliers of the equipment which reflects their operational experience. Overall costs have been compared to adjacent operations where possible. No contingency has been added. The average operating cost estimated to mine and process the mineralization is $5.28/t mined for the Owner operated base case and $6.45/t for the Contractor operation. This includes mining of mineral and waste, leaching, processing, general and administration and owners cost. The costs are indicative only and are in 2011 US$.

Table 1.7 shows a summary of the estimated cost by year and area for the Owner operated and Table 1.8 the Contractor operated alternative.

Table 1.7: Operating Cost Summary $(000)'s (Owner Operated)

Description	Cost/t	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	Total
Processed Ore (000)'s Ton		3,880	6,977	7,151	8,886	8,933	9,156	8,993	9,144	6,368	6,824	1,896		78,209
Total		$25,644	$38,500	$40,236	$43,854	$45,011	$45,426	$44,543	$46,625	$35,455	$34,621	$12,744		$412,660
Yearly Cost/ton	$5.28	$6.61	$5.52	$5.63	$4.94	$5.04	$4.96	$4.95	$5.10	$5.57	$5.07	$6.72		$5.28

Table 1.8: Operating Cost Summary $(000)'s (Contractor Operated)

Description	Cost/t	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	Total
Processed Ore (000)'s Ton		3,880	6,977	7,151	8,886	8,933	9,156	8,993	9,144	6,368	6,824	1,896		78,209
Total		$33,741	$45,271	$48,182	$51,389	$59,206	$59,465	$56,309	$55,790	$41,635	$40,283	$13,554		$504,826
Yearly Cost/ton	$6.45	$8.70	$6.49	$6.74	$5.78	$6.63	$6.49	$6.26	$6.10	$6.54	$5.90	$7.15		$6.45

1.11 FINANCIAL ANALYSIS

The financial analysis has been prepared using standard discounted cash flow methods to determine the NPV and IRR of the project based upon 100% equity financing and metal price of US$1350/oz Au for the Base Case. This gold price is based on the trailing three year average. The analysis was performed in constant 2011 US dollars, excluding inflation. Various sensitivities have been completed to determine the effect of changes to metal prices, grade, capital costs, operating costs and leach recovery.

1.12 RESULTS

1.12.1 Base Case (Owner Operated)

Case 1 economic evaluation is are based on an initial construction capital expenditure of US$122.1 million operating cost of $5.28/ton mined and metal prices of US$1350/oz[4]. The project will generate an after-tax IRR of 15.44% and an NPV of US$180.6 million undiscounted and US$97 million discounted at 5%. Payback of initial capital can be achieved in 5.91 years. The inferred resources established in this study are 78.2 million tons grading 0.0138 oz/t of gold.

[4] It should be noted that the authors of this report have not and do not forecast the future price of gold. The alternatives shown in this report are intended to indicate the variation of the project financials based upon sensitivity criteria shown.

Case 1 financial analysis summary is shown on Table 1.9.

Case 5 reflects a gold price of US$1500/oz. Under this scenario the project will generate an after-tax IRR of 20.03% and an NPV of US$247.2 million undiscounted and US$147.1 million discounted at 5%. Payback of initial capital can be achieved in 4.72 years.

Case 7 reflects a gold price of US$1750/oz. Under this scenario the project will generate an after-tax IRR of 27.24% and an NPV of US$358.3 million undiscounted and US$230.7 million discounted at 5%. Payback of initial capital can be achieved in 3.91 years.

Sensitivity studies have been prepared varying the price of gold, the operating and capital cost, grade and leach recovery. The project is most sensitive to gold price, sensitive to operating cost, grade and recovery and least sensitive to capital cost. The sensitivity results are shown on Table 1.10.

It should be noted that the financial analysis has been evaluated on the basis of the construction activities being set against proceeds and those costs for the period to a construction decision have been included as sunk costs although these development costs have not been incurred at the time of writing this report. This is considered as an acceptable approach since any decision to construct will be based upon the financial analysis at the time following completion of the development activities. Thus the expenditures for construction only are set against the proceeds of the project and not the development costs. To include the development costs in the capital costs will not provide a basis for a fair evaluation of the project.

Table 1.9: Financial Analysis (Owner Operated)

ROBERTSON PROJECT

ALTENBURG, PORPHYRY AND GOLD PAN DEPOSITS

Financial Analysis Base Case

$(000)s

Resource Ton (000)'s **78,209** **YEAR**

Description	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**Total**
Production Ton/year HG				439	1,436	2,687	2,198	2,877	2,903	3,528	4,596	2,696	2,496	652		**26,509**
Au Grade oz/t				0.017	0.019	0.021	0.019	0.019	0.019	0.020	0.023	0.022	0.024	0.021		**0.021**
Recovery Au %				67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%		**67.00%**
Production Ton/year LG				3441	5541	4464	6688	6056	6253	5465	4549	3673	4328	1244		**51,700**
Au Grade oz/t				0.010	0.010	0.010	0.010	0.011	0.010	0.011	0.010	0.010	0.010	0.010		**0.010**
Recovery Au %				45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%		**45.00%**
Au ounces payable				20,277	42,619	57,438	58,180	65,106	66,406	73,156	90,776	55,848	59,364	14,942		**604,111**
Gross Revenue				$ 27,350	$ 57,487	$ 77,476	$ 78,477	$ 87,819	$ 89,572	$ 98,678	$ 122,444	$ 75,332	$ 80,074	$ 20,155		**342,699**
Operating Costs				$(25,644)	$(38,500)	$(40,236)	$(43,854)	$(45,011)	$(45,426)	$(44,543)	$ (46,625)	$(35,455)	$(34,621)	$(12,744)		**($412,660)**
Income Tax				$ (29)	$ (315)	$ (1,115)	$ (1,822)	$ (3,409)	$ (4,622)	$ (5,857)	$ (20,477)	$ (9,084)	$(11,167)	$ (1,608)		**($59,506)**
Revenue Before Capital Exp.				$1,677	$18,672	$36,125	$32,800	$39,399	$39,525	$48,278	$55,342	$30,793	$34,286	$5,803		$342,699
Capital Expenditures																
- Development/Construction			($114,583)													**($114,583)**
- On-Going Capital					($4,727)	($4,727)	($4,727)	($4,530)	($21,030)	($2,912)	($1,292)	($1,292)	($1,292)			**($46,528)**
Working Capital Change				($3,039)	($1,697)									$4,735		
Reclamation Bond			($7,500)												$7,500	
Reclamation															($7,700)	**($7,700)**
Salvage															$6,750	**$6,750**
Total Capital			($122,083)	($3,039)	($6,423)	($4,727)	($4,727)	($4,530)	($21,030)	($2,912)	($1,292)	($1,292)	($1,292)	$4,735	$6,550	($162,061)
Net Cashflow			($122,083)	($1,362)	$12,249	$31,399	$28,074	$34,869	$18,495	$45,366	$54,050	$29,500	$32,993	$10,538	$6,550	**$180,638**
Discounted NCF 5%			**($122,083)**	**($1,297)**	**$11,110**	**$27,123**	**$23,096**	**$27,321**	**$13,801**	**$32,241**	**$36,583**	**$19,016**	**$20,255**	**$6,161**	**$3,647**	**$96,976**
Discounted NCF 8%			**($122,083)**	**($1,261)**	**$10,501**	**$24,925**	**$20,635**	**$23,732**	**$11,655**	**$26,471**	**$29,201**	**$14,758**	**$15,282**	**$4,520**	**$2,601**	**$60,937**
Discounted NCF 10%			**($122,083)**	**($1,238)**	**$10,123**	**$23,590**	**$19,175**	**$21,651**	**$10,440**	**$23,280**	**$25,215**	**$12,511**	**$12,720**	**$3,694**	**$2,087**	**$41,164**

Rate of Return **15.44%**

Notes:

1. Metal Prices US $ Au/oz 1350.00

2. Capital requirements based on 100% equity.

3. All funds are in US$ except where noted.

4. Taxes are approximate.

Payback 5.91 years

Au first full production year 20,277 ozs

Average NSR/ton $10.42

Table 1.10: Sensitivities (Owner Operator)

Case	Description of Sensitivity	NPV Dis.0%	NPV Dis.5%	NPV Dis.8%	IRR
		US$(000)s	US$(000)s	US$(000)s	%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
Case 2	Gold Price $850/oz	($57,729)	($84,166)	($94,825)	-6.12%
CASE3	Gold Price $1100/oz	$79,451	$19,046	($6,612)	7.15%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE5	Gold Price $1500/oz	$247,156	$147,053	$103,770	20.13%
CASE6	Gold Price $1750/oz	$358,295	$230,661	$175,241	27.24%
CASE7	Gold Price $2000/oz	$464,197	$310,181	$243,159	33.41%
CASE8	Grade -10%	$124,051	$54,202	$24,232	11.07%
CASE9	Grade -5%	$151,823	$75,315	$42,414	13.29%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE10	Grade +5%	$209,974	$119,021	$79,779	17.55%
CASE11	Grade +10%	$240,183	$141,688	$99,127	19.62%
CASE12	Capital Cost -20%	$188,904	$103,569	$66,733	16.13%
CASE13	Capital Cost -10%	$184,771	$100,272	$63,835	15.79%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE14	Capital Cost +10%	$176,505	$93,679	$58,039	15.09%
CASE15	Capital Cost +20%	$172,372	$90,382	$55,141	14.75%
CASE16	Operating Cost -20%	$239,844	$142,662	$100,598	20.04%
CASE17	Operating Cost -10%	$209,805	$119,521	$80,533	17.74%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE18	Operating Cost +10%	$152,031	$74,793	$41,610	13.12%
CASE19	Operating Cost +20%	$124,961	$53,382	$22,744	10.80%
CASE20	Leach Recovery +10%	$240,183	$141,688	$99,127	19.62%
CASE 21	Leach Recovery +5%	$209,974	$119,021	$79,779	17.55%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE 22	Leach Recovery -5%	$151,823	$75,315	$42,414	13.29%
CASE23	Leach Recovery -10%	$124,051	$54,202	$24,232	11.07%

1.12.2 Results Alternative Case A1 (Contractor Operated)

Case A1 economic evaluation is are based on an initial construction capital expenditure of US$97 million operating cost of $6.45/ton mined and metal prices of US$1350/oz. The project will generate an after-tax IRR of 15.43% and an NPV of US$159.4 million undiscounted and US$85.2 million discounted at 5%. Payback of initial capital can be achieved in 5.94 years.

Case A1 financial analysis summary is shown on Table 1.11.

Case A5 reflects a gold price of US$1500/oz. Under this scenario the project will generate an after-tax IRR of 20.96% and an NPV of US$226.4 million undiscounted and US$135.9 million discounted at 5%. Payback of initial capital can be achieved in 4.86 years.

Case A7 reflects a gold price of US$1750/oz. Under this scenario the project will generate an after-tax IRR of 29.18% and an NPV of US$337.8 million undiscounted and US$219.7 million discounted at 5%. Payback of initial capital can be achieved in 3.82 years.

Sensitivity studies have been prepared varying the price of gold, the operating and capital cost, grade and leach recovery. The project most sensitive to gold price, sensitive to operating cost, grade and recovery and least sensitive to capital cost. The sensitivity results are shown on Table 1.12.

Table 1.11: Financial Analysis Alternative 1 (Contractor Operated)

ROBERTSON PROJECT
ALTENBURG, PORPHYRY AND GOLD PAN DEPOSITS
Financial Analysis Base Case
$(000)s

Resource Ton (000)'s **78,209** **YEAR**

Description	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	Total
Production Ton/year HG				439	1,436	2,687	2,198	2,877	2,903	3,528	4,596	2,696	2,496	652		**26,509**
Au Grade oz/t				0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02		**0.02**
Recovery Au %				67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%		**67.00%**
Production Ton/year LG				3441	5541	4464	6688	6056	6253	5465	4549	3673	4328	1244		**51,700**
Au Grade oz/t				0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01		**0.01**
Recovery Au %				45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%		**45.00%**
Au ounces payable				20,277	42,619	57,438	58,180	65,106	66,406	73,156	90,776	55,848	59,364	14,942		**604,111**
Gross Revenue				27,350	57,487	77,476	78,477	87,819	89,572	98,678	122,444	75,332	80,074	20,155		**268,219**
Operating Costs				(33,741)	(45,271)	(48,182)	(51,389)	(59,206)	(59,465)	(56,309)	(55,790)	(41,635)	(40,283)	(13,554)		**($504,826)**
Income Tax				($29)	($210)	($205)	($107)	($169)	($214)	($3,617)	($17,845)	($7,572)	($9,799)	($2,053)		**($41,819)**
Revenue Before Capital Exp.				($6,420)	$12,006	$29,090	$26,981	$28,445	$29,893	$38,752	$48,809	$26,125	$29,991	$4,548		$268,219
Capital Expenditures																
- Development/Construction			($89,449)													**($89,449)**
- On-Going Capital					($3,434)	($3,434)	($3,434)	($3,237)	($3,237)	($1,619)						**($18,396)**
Working Capital Change				($3,039)	($1,697)									$4,735		
Reclamation Bond			($7,500)												$7,500	
Reclamation															($7,700)	**($7,700)**
Salvage															$6,750	**$6,750**
Total Capital			($96,949)	($3,039)	($5,131)	($3,434)	($3,434)	($3,237)	($3,237)	($1,619)				$4,735	$6,550	($108,795)
Net Cashflow			($96,949)	($9,459)	$6,875	$25,656	$23,547	$25,207	$26,656	$37,132	$48,809	$26,125	$29,991	$9,283	$6,550	**$159,424**
Discounted NCF 5%			**($96,949)**	**($9,008)**	**$6,236**	**$22,162**	**$19,372**	**$19,751**	**$19,891**	**$26,389**	**$33,036**	**$16,840**	**$18,412**	**$5,428**	**$3,647**	**$85,207**
Discounted NCF 8%			**($96,949)**	**($8,758)**	**$5,895**	**$20,366**	**$17,308**	**$17,156**	**$16,798**	**$21,666**	**$26,370**	**$13,069**	**$13,892**	**$3,981**	**$2,601**	**$53,394**
Discounted NCF 10%			**($96,949)**	**($8,599)**	**$5,682**	**$19,275**	**$16,083**	**$15,652**	**$15,047**	**$19,055**	**$22,770**	**$11,079**	**$11,563**	**$3,254**	**$2,087**	**$35,999**

Rate of Return **15.43%**

Notes:
1. Metal Prices US $ Au/oz 1350.00
2. Capital requirements based on 100% equity.
3. All funds are in US$ except where noted.
4. Taxes are approximate.

Payback 5.94 years
Au first full production year 20,277 ozs
Average NSR/ton $10.42

Table 1.12: Sensitivities – Alternative 1 (Contractor Operated)

Case	Description of Sensitivity	NPV Dis.0%	NPV Dis.5%	NPV Dis.8%	IRR
		US$(000)s	US$(000)s	US$(000)s	%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
Case A2	Gold Price $750	($94,427)	($107,304)	($111,909)	-11.84%
CASE A3	Gold Price $1000	$50,380	$624	($20,123)	5.08%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A5	Gold Price $1500	$226,426	$135,895	$96,886	20.96%
CASE A6	Gold Price $1750	$337,781	$219,732	$168,591	29.18%
CASE A7	Gold Price $2000	$445,057	$300,377	$237,512	36.41%
CASE A8	Grade -10%	$106,068	$43,645	$17,154	10.42%
CASE A9	Grade -5%	$132,368	$64,225	$35,138	12.94%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A10	Grade +5%	$189,244	$107,787	$72,784	17.95%
CASE A11	Grade +10%	$219,453	$130,521	$92,229	20.37%
CASE A12	Capital Cost -20%	$162,796	$88,022	$55,930	15.80%
CASE A13	Capital Cost -10%	$161,071	$86,587	$54,640	15.62%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A14	Capital Cost +10%	$157,789	$83,836	$52,156	15.25%
CASE A15	Capital Cost +20%	$156,204	$82,502	$50,949	15.08%
CASE A16	Operating Cost -20%	$232,745	$142,161	$103,028	22.14%
CASE A17	Operating Cost -10%	$195,890	$113,650	$78,244	18.84%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A18	Operating Cost +10%	$127,084	$59,049	$30,099	12.14%
CASE A19	Operating Cost +20%	$95,153	$32,755	$6,515	8.88%
CASE A20	Leach Recovery +10%	$219,453	$130,521	$92,229	20.37%
CASE A21	Leach Recovery +5%	$189,244	$107,787	$72,784	17.95%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A22	Leach Recovery -5%	$132,368	$64,225	$35,138	12.94%
CASE A23	Leach Recovery -10%	$106,068	$43,645	$17,154	10.42%

1.13 CONCLUSIONS

The following is a list of the conclusions;

1. The Robertson property is one of merit upon which further work is warranted consisting of exploration and definition drilling, metallurgical test work, geotechnical test work and associated work that will allow the preparation of a project evaluation to a prefeasibility level.
2. The relative low capital cost provides an opportunity for a junior company such as Coral to raise those funds and operate the project.
3. The project is seen as one where a contractor operation would be beneficial due to the increased operating cost is offset by the lower capital cost expenditures.

4. The Nevada project approval system has elongated the development period for the project. Every effort should be made to attempt to reduce the time to achieve regulatory approval of the project and allow the project to be developed as quickly as is feasible.
5. The financial analysis results indicate that the project breaks even at US$950/oz gold for the Owner operation and US$1010/oz for the Contractor alternative.

1.14 RECOMMENDATIONS

It is recommended that the project be developed in two stages. The first is that work required to take the project to completion of a prefeasibility report and the second work required to complete a full "bankable" feasibility, i.e., one on which a construction decision could be made. Costs have been estimated for stage 1 and the cost for stage 2 have been included in the capital cost section for guidance only; stage 2 costs will be defined in the report derived from stage 1.

The work in stage 1 consists of;

1. Exploratory and definition drilling.
2. Metallurgical test work program.
3. Environmental program.
4. Geotechnical and associated work.
5. A prefeasibility study.

A summary of stage 1, estimated to cost $7.9 million, is shown on Table 1.13.

Cost breakdowns are shown on Tables 1.14, 1.15 and 1.16.

Table 1.13: Summary of Expenditures for Stage 1

Description	Estimated Cost $
Royalty and Regulatory Fees	$351,680
Exploratory and definition drilling	$2,817,000
Metallurgical test work program	$900,000
Environmental program	$1,826,138
Preliminary Feasibility Study	$1,495,000
Contingency	$510,182
Total	**$7,900,000**

1.14.1 Exploration and Definition Drilling

It is recommended that Coral conduct a two phase exploration program focused on expanding and up-grading the near-surface oxide and sulfide inferred mineral resources to measured and indicated.

The Phase I should consist of drilling 40 HQ diameter diamond core holes and 42 RC holes having an average depth of 400-500 ft and totaling about 40,000 ft in the;

- **Porphyry Zone:** "Twinning" 10 percent (20 holes) of the historic drill holes by diamond core drilling to determine if "historic" Amax drilling data can be used with confidence to upgrade the level of confidence in the resources. In addition an additional 17 RC holes, totaling about 7,600 ft, to be drilled along the west and south boundaries of the Porphyry Zones to test for possible extensions to mineralization.

- **Altenburg Hill/South Porphyry Area:** Twenty-five RC holes totaling 12,400 ft.

- **Gold Pan Zone:** Twenty wide-spaced diamond core holes totaling 10,000 ft to verify continuity and grade retuned in historic drilling.

- **Altenburg Hill/South Porphyry:** Base on results on the Phase I RC drilling follow up diamond core drilling (20 holes) is conducted in these areas.

Phase one drilling is expected to cost $2 million with the second Phase $800,000 for a total cost including contingency of $2.8 million.

1.14.2 Metallurgical Test Work Program

Variability testing will be performed on samples obtained both spatially and at depth for the oxide and transition to sulfide ore zones. This work will encompass;

- prepare composite material representing larger zones of each deposit to define the crush size and other process conditions;.
- crushing work index and abrasion testing;
- mineralogical evaluation of column feed and products;
- extensive column work to determine optimum crush size and other process conditions;
- similar testing as was performed on oxide materials to be done on sulfide and transition zone materials;
- additional processing parameters to be investigated including reagent use and concentrations;
- leach evaluation on below the cut-off grades of the various deposits was classified as waste based on dump leaching of run of mine low grade materials;
- laboratories testwork up to 10 tons of 100% minus 300 mm (~12") feed.

Table 1.14: Proposed Test Work to Prefeasibility

Description	Est Cost $
Oxide / Partially Oxidized High Grade Drill Core	$350,000
Sulfide and Transition High Grade Drill Core	$200,000
Low Grade Bulk Tonnage Samples	$350,000
Total	**$900,000**

1.14.3 Environmental Work

The proposed environmental work consists of the following as shown in Table 1.15. It is expected based upon present legislation to be complete and the project approved in 2017. It may be possible to reduce this period and it is recommended that every effort be made to achieve this.

Table 1.15: Summary of Proposed Environmental work

Description	Year 2012	2013	2014	2015	2016	Total
Baseline Data Collection						
Water	$50,000	$60,000	$60,000	$30,000	$30,000	$230,000
Geochemistry	$30,000	$20,000				$50,000
Eagle/Raptor Survey	$25,000					$25,000
Plan of Operation/Reclamation Permit Application	$35,000	$20,000	$15,000	$10,000	$20,000	$100,000
EIS (draft and final)	$90,000	$200,000	$300,000	$200,000	$210,000	$1,000,000
Water Pollution Control Permit			$40,000	$40,000	$40,000	$120,000
Air Permits			$40,000	$10,000	$10,000	$60,000
Water Appropriations	$10,000	$10,000		$10,000		$30,000
Stormwater Pollution Prevention Plan					$10,000	$10,000
Industrial Artificial Pond Permit					$125	$125
Potable Water					$10,000	$10,000
Septic					$20,000	$20,000
Class III Landfill					$5,000	$5,000
Contingency 10%	$24,000	$31,000	$45,500	$30,000	$35,513	$166,013
Total	**$264,000**	**$341,000**	**$500,500**	**$330,000**	**$390,638**	**$1,826,138**

1.14.4 Geotechnical and Associated Investigations

The proposed geotechnical and associated investigation costs are included in the feasibility costs estimates. It is planned to geotechnically log the core holes drilled as part of the exploration program. This information together with the list below will form the basis for this work.

- Evaluation of the allowable soil bearing pressures induced by plant site facilities for a range of different footing shapes and sizes, and consideration of potential settlements
- Estimation of the allowable bearing pressures of the bedrock
- Seismic design parameters for the plant site, and
- Test pits in the leach pad area to investigate foundation conditions, overburden materials, and depth to bedrock.
- Drill holes to investigate depth and quality of bedrock, for permeability testing, and for the installation of groundwater monitoring wells.
- Additional test pits to prove suitability, availability and quantity of borrow materials for earthworks construction.
- Additional laboratory index test works (including compaction tests) on potential borrow materials.
- Strength and permeability test work on potential borrow materials for pad foundation and embankment construction.
- Direct shear testing of the geosynthetic liner interfaces, to determine interface friction angles for stability assessment.
- Ore testing (including gradation, load-permeability, and load-density).

1.14.5 Prefeasibility Study

The expected cost to complete a prefeasibility study, based upon the results of the PEA as described in this report, is $1.495 million.

Table 1.16 shows the cost estimate breakdown for the Prefeasibility Study.

Table 1.16: Prefeasibility Cost Estimate

Task Description	Estimated Cost $
Project Management	$100,000
Geology	$150,000
Resource Estimate and Open Pit Optimization	$150,000
Mine Planning	$150,000
Metallurgy and Process Plant	$95,000
Geotechnical	$300,000
Surface Buildings	$50,000
Power supply/Electrical distribution/Communication	$70,000
Cost Estimates	$110,000
Financial Analysis	$75,000
Project Disbursements	$50,000
Contingency 15%	$195,000
Total Estimated Cost	**$1,495,000**

SECTION 2.0 INTRODUCTION

2.1 SCOPE OF WORK

Coral Gold Resources Ltd. (Coral) requested **Beacon Hill Consultants (1988) Ltd. (Beacon Hill)** to prepare a Preliminary Economic Assessment (PEA) on the Robertson Property containing a number of gold bearing zones, located in Lander County, Nevada, USA. The evaluation was to review the Altenburg Hill, Porphyry and Gold Pan zones only and include recent additional drilling and metallurgical work completed in 2010 and 2011. The PEA has been prepared to meet the NI 43-101 criteria for a technical report.

2.2 PROJECT CRITERIA

Unless otherwise indicated all references to dollars (US$) used in this report refer to currency of the United States.

Units of measure, conversion factors and abbreviation frequently used in this report include:

Units of measure and conversion factors used in this report (from SME-AIME):

1 inch = 2.54 centimeters
1 foot = 0.3048 meters
1 mile = 1.61 kilometers
1 acre = 0.4047 hectares
1 square mile = 640 acres
640 acres = 259 hectares
1 pound = 0.454 kilograms

10 parts per billion = 0.0029 troy ounces/short ton
1 part per million = 0.029 troy ounces/short ton
1 troy ounce/short ton = 34.2857 grams/metric ton
1 gram/metric ton = 0.02917 troy ounces/short ton
1 short ton = 2,000 pounds
1 short ton = 0.907 metric tons

Abbreviations used frequently in this report:

AA	Atomic absorption spectrometry
Au	Gold
Ag	Silver
As	Arsenic
Bi	Bismuth
Cu	Copper
DDH	Diamond drill hole (core)
ft	Feet
Hg	Mercury
ICP	Inductively coupled plasma spectrophotometry
i.d.	Inside diameter
ID	Identification
Ma	Million years ago
Mo	Molybdenum
NaCN	Sodium cyanide
NE	Northeast
NW	Northwest
NNW	North-northwest
Pb	Lead
ppb	Parts per billion
ppm	Parts per million
oz/t	Troy ounces/short ton
ROM	Run of mine
RC	Reverse circulation drilling
Sb	Antimony
Zn	Zinc

2.3 PROJECT MANAGEMENT

Beacon Hill Consultants (1988) Ltd. has compiled this report, providing the overall study co-ordination, as well as certain engineering elements within the study. The team completing the work is shown below.

2.4 TEAM MEMBERS

The team consisted of:

Beacon Hill Consultants (1988) Ltd. personnel and associates;

- **W. P. Stokes, P.Eng.**, Project Manager and Mining Engineer
- **G. Kirkham, P.Geo.**, Associate Geologist and Geoscientist
- **R.W. J Fox, P.Eng.,** Metallurgist

Independent Metallurgist

- **F. Wright, P.Eng.**

Independent Geological Consultant;

- **Robert T. McCusker, P.Geo.**

Knight Piésold Ltd.;

- **K. J. Brouwer, P.Eng. Managing Director**
- **C. Aurala, P.Eng., Project Manager**

Ledcor CMI Inc.

Nevada Contractor

2.5 DATA SUPPLIED BY CORAL

Coral has supplied the surface topography for the area; all the geological data, assay and associated data to allow for the resource modeling and estimation to be completed.

2.6 EVALUATION CRITERIA

The evaluation was based on two approaches;

Base Case;	The Owner was operator for all operations at the mine and mineral processing activities.
Alternative A;	A contractor operation to mine the mineral and waste and deliver that to the crusher, low grade stockpile and the waste deposition area; the Owner would be operator and responsible for all other activities.

SECTION 3.0 RELIANCE ON OTHER EXPERTS

The authors have relied on the experience and expertise of the on-site geology personnel for input with respect to the interpretation of recovery data and also the interpretation of geology, mineralization, and specific gravity data. The authors believe these interpretations to be a current and an accurate representation of the deposit.

SECTION 4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Robertson Property is located in eastern Lander County, Nevada, on public lands administered by the Bureau of Land Management (BLM). The property is situated in Sections 2-10, 15-18, 20, 21, Township 28 North, Range 47 East and Sections 26, 32 and 34, Township 29 North, Range 47 East, Mt. Diablo Baseline & Meridian (Figure 4-1).

4.2 MINING CLAIM DESCRIPTION

The Robertson Property consists of 510 unpatented federal lode, 90 placer claims, nine patented lode claims and one unpatented mill site, covering approximately 7,300 acres (Figure 4-2). The claim total includes two no-contiguous groups of claims totaling 82 claims that are also controlled by Coral. Coral Resources, Inc. owns or controls the surface and mineral estates within the project area through record title, or through mining lease and mining lease with option to purchase agreements of the patented and unpatented mining claims. At the end of 2010, Coral was record owner with 100 percent interest in 525 claims and controls an additional 76 claims through a variety of mineral leases. A comprehensive title search of these claims was completed in 2007 by Harris & Thompson, a Reno, Nevada law firm, which found no major title issues. The authors of this report have not completed a title search or verified the ownership of the claims and have relied upon the legal advice given by Coral.

4.3 AGREEMENTS AND ENCUMBRANCES

In 1986, Coral Gold Corporation and its US affiliate Coral Resources, Inc., acquired an option to acquire a 70 percent interest in the “mining properties” owned or controlled by Aaron Mining and its affiliates, in the Tenabo mining district, Nevada, under an Option and Joint Venture agreement. Coral was granted an undivided 70 percent interest in the Tenabo mining properties in 1987. Also in 1987, Coral and its affiliate, acquired an undivided 29 percent interest in the “Tenabo Properties” from Geomex Development, Inc. (predecessor-in-interest to Albany Gold Corp.). Geomex had acquired its interest from E & B Exploration, who explored the property under a 1980 Option Agreement with Aaron Mining Ltd. As part of Coral’s acquisition of the Geomex interest, Geomex reserved a 3 percent Net Smelter Return Royalty covering all of the “Tenabo Properties”, including all mining claims owned or controlled by Coral, or its affiliates, in Lander County, Nevada. This royalty has an option to purchase for $1,250,000.

In 1990, Coral and Amax Gold Exploration, Inc. (“Amax”) entered into an Amended and Restated Option and Earn-In Agreement in which Amax could earn a 60 percent interest in the property by producing a bankable feasibility study. A feasibility study was completed by Amax in November, 1994, but because of marginal economics of the project, the parties determined that additional exploration was necessary. Both companies agreed that Amax did not fulfill the requirements of the agreement and did not earn any interest in the property. Amax withdrew from the project in 1996.

In 1995, Coral and Amax agreed to “carve out” 219 claims (the “Excluded Claims”) from the original block of 755 claims and place them under a separate Earn-In and Mining Venture Agreement. In 1996, Amax withdrew from the 1990 agreement but maintained its interest in the agreement covering the “Excluded Claims”. Cortez acquired the Amax interest in the “Excluded Claims” and in 1998, Coral and Cortez signed an Option and Earn-In Agreement covering the Robertson Property (non-Excluded Claims). After completing an exploration program on the

Robertson Property in 1999, Cortez withdrew from the Option and Earn-In Agreement. Under terms of the agreement, Cortez did not earn an interest in the property.

In December 2004, Coral and Agnico-Eagle (USA), Ltd., then known as Nevada Contact, Inc., entered into an agreement covering 71 mining claims owned by Mauzy et al. and controlled by Coral through a 1989 Mineral Lease and Purchase Option (amended). Coral assigned and subleased the underlying 1989 Mauzy Agreement to Agnico-Eagle, along with the obligation to pay required lease payments under the Mauzy Agreement and minimum advance royalty payments to Coral. Coral also reserved a 2.5 percent net smelter return production royalty. In 2007, Agnico-Eagle withdrew from the agreement without earning an interest in the property.

As part of the Coral-Agnico agreement, the parties agreed to amend the underlying 1989 Mineral Lease and Purchase Option Agreement with Mauzy et al. The amendment included extending the term of the agreement, reducing the royalty to 2 percent of the net smelter return and adjusting the purchase price for all claims subject to the agreement.

In 2005, Coral acquired 39 unpatented lode claims and two association placer claims, covering approximately 960 acres, within the Robertson project area, from the Marcus Corporation. Under a Share Exchange Agreement dated July 12, 2005, Marcus shareholders were offered one common share of Coral stock for 4 common shares of Marcus stock and one warrant for additional common shares of Coral stock at a reduced fixed price for every 2 common shares of Marcus stock. As of December 2005, Coral had acquired 1,391,860 shares, or 99.98 percent, of the outstanding Marcus common shares. Certain of the Marcus claims cover portions of the Altenburg Hill inferred resource.

Twenty of the claims acquired as part of the Marcus acquisition include the Ruf unpatented lode claims which are subject to a 1995 Option and Joint Venture Agreement with Levon Resources Ltd. In 2002, after incurring a minimum $200,000 in exploration expenditures, as required under terms of the agreement, Levon was deemed to have earned an undivided one-third interest in the Ruf claims.

In 2008, Coral entered into a Mineral Lease and Option to Purchase Agreement with Dianne Breckon et al, owners of the June 1-6 unpatented lode claims. Terms of the agreement include annual rental payments of $25,000, renewable in successive 4 year terms, provided the rent increases by $5,000 every four years. The property is subject to a 3% NSR which can be purchased for $3,000,000 and the property for $1,000,000.

During 2007, Coral exercised its option to purchase 76 unpatented lode and placer claims and 6 patented mining claims owned by Elwood Wright and 16 unpatented lode claims owned by Florence Johnson.

Approximately 76 of the 601 claims that comprise the Robertson Property are controlled by Coral through six mining leases and option agreements. Total annual payments for the various leases and minimum advance royalties or rental payments are US$91,700. A summary compilation of the terms of these agreements are presented in Table 4.1. The location of claims subject to the various production royalties are shown in Figure 4-2. It should be noted that a 3% NSR royalty owned by Albany Gold Corp. applies to all Coral-owned mining claims within the Robertson project area, as well as throughout Lander County, Nevada.

Table 4.1: Tabulation of Mining Lease and Option Agreements.

Company/Date	Number of Claims	Option Pmt.	Production Royalty	Advance Royalty Pmt.
Tenabo Gold Mining Co Nov. 30, 1975	13	$2M	8% NSR	$12,000/yr
Northern Nevada Au, Inc Sept. 30, 1986	12	$0.3M	4% GSR	$9,600/yr
Albany Gold Corp. (Geomex)	All	$1.25M	3% NSR	None
Mauzy, et al Apr. 21, 1989	36	$1.15M	2% NSR	$23,500/yr
Jay Wintle Mar. 1, 1992	9		5% NSR	$21,600/yr
Breckon, et al Mar. 22, 2008	6	$1.0M	3% NSR	$25,000/yr

Annual federal rental fees of US$84,140 payable to the BLM and Notice of Intent to Hold Mining Claims fee of US$6,310.50 payable to Lander County have been paid for the 2011-2012 assessment year.

Effective September 2004, Coral changed its name from Coral Gold Corporation to Coral Gold Resources Limited. The company is listed on the TSX Venture Exchange.

4.4 ENVIRONMENTAL LIABILITIES

In 1988-89, Coral operated a small open pit gold mining operation and heap leach facility on the Robertson Property. The resulting disturbances include three small open pit mines, waste dumps, haul roads, drill roads, open drill holes, and a 350,000 ton heap leach facility and related recovery plant. In 1994, a reclamation plan was prepared by Amax and submitted to the Mount Lewis Field Office (formally the Battle Mountain office) of the BLM. The cost to perform the reclamation of the Robertson mine site was estimated at that time to be $2 million. In 2001, Coral began reclamation activities which were accelerated in 2002, with the recontouring of waste dumps, reclamation of the leach pad, haul roads and the filling of all open drill holes. As a result of this activity, in June, 2003, the BLM lowered the bonding requirements for the project to $406,000.

In March 2003, on behalf of Coral, SRK Consulting submitted a Final Plan for Permanent Closure with the BLM and Nevada Division of Environmental Protection (NDEP) which was approved by both agencies. The major component of the closure plan was the installation of a site fluid management system. During 2004, a terminal evapo-transpiration basin with a designed annual evapo-transpire capacity of 422,300 gallons per year was constructed. In 2007, the recovery plant was dismantled and removed from the property and additional road and drill site reclamation was completed. As a result of this work, the BLM lowered the bonding requirements to $352,934. Coral currently maintains a state wide performance bond totaling $63,650 with the Nevada State Office of the BLM covering its various drilling programs conducted under a Notice of Intent (TRY/View).

Coral’s current reclamation obligation at Robertson is $352,934, covering 209.5 acres of disturbance.

4.5 PERMITTING

In 2010, Coral submitted and was granted a five year renewal of Water Pollution Control Permit (NEV60035) by the NDEP. In addition, Coral continues to conduct reclamation and exploration activities under a Plan of Operation (NVN-067688) approved in 1989 by the BLM. In 2007, Coral conducted drilling operations in the extreme northwest part of the Robertson core property under a Notice of Intent NVN-083095 (TRY/View Notice).

During the period 2000 through 2003, no exploration activity was conducted on the Robertson Property. However, during that period a significant amount of surface reclamation was completed on the property. As a result, new exploration activities in reclaimed areas will require submission and approval of an Amendment to the Plan of Operation. Additionally, the National Historic Preservation Act requires that all operators on public lands conduct an archeological survey of the proposed sites of new disturbance. Much of the core Robertson Property has been previously cleared under various surveys conducted by Amax. Recent and planned future exploration activities by Coral have moved outside the area covered by previous archeological surveys. It is possible that future exploration will experience delays in receiving approval because additional surveys will be required by state and federal agencies. In 2004-06, Coral conducted exploratory drilling under a series of amendments to the Plan of Operation which were approved by the Mount Lewis Field office of the BLM and NDEP.

In 2007, NDEP requested that Coral prepare and submit a new Plan of Operation consolidating all of the past modifications and amendments to the previous plan into one document. The new plan was submitted in November 2007 and approval was granted in July 2008.

Coral submitted a new Amended Plan of Operation (APO) to the Mount Lewis Field office of the BLM in April 2010 to cover proposed drilling operations on approximately 5,169 acres of the Robertson core claims. After reviewing the APO, the BLM determined that Coral would be required under the National Environmental Policy Act (NEPA) to produce an Environmental Assessment (EA) to address various potential environmental issues associated with the proposed drilling activities. Approval of the new APO is expected in 2012.

There are no known environmental or threaten and endangered species issues at the Robertson Property that would provide grounds for denial of approval of an Amended Plan of Operation.

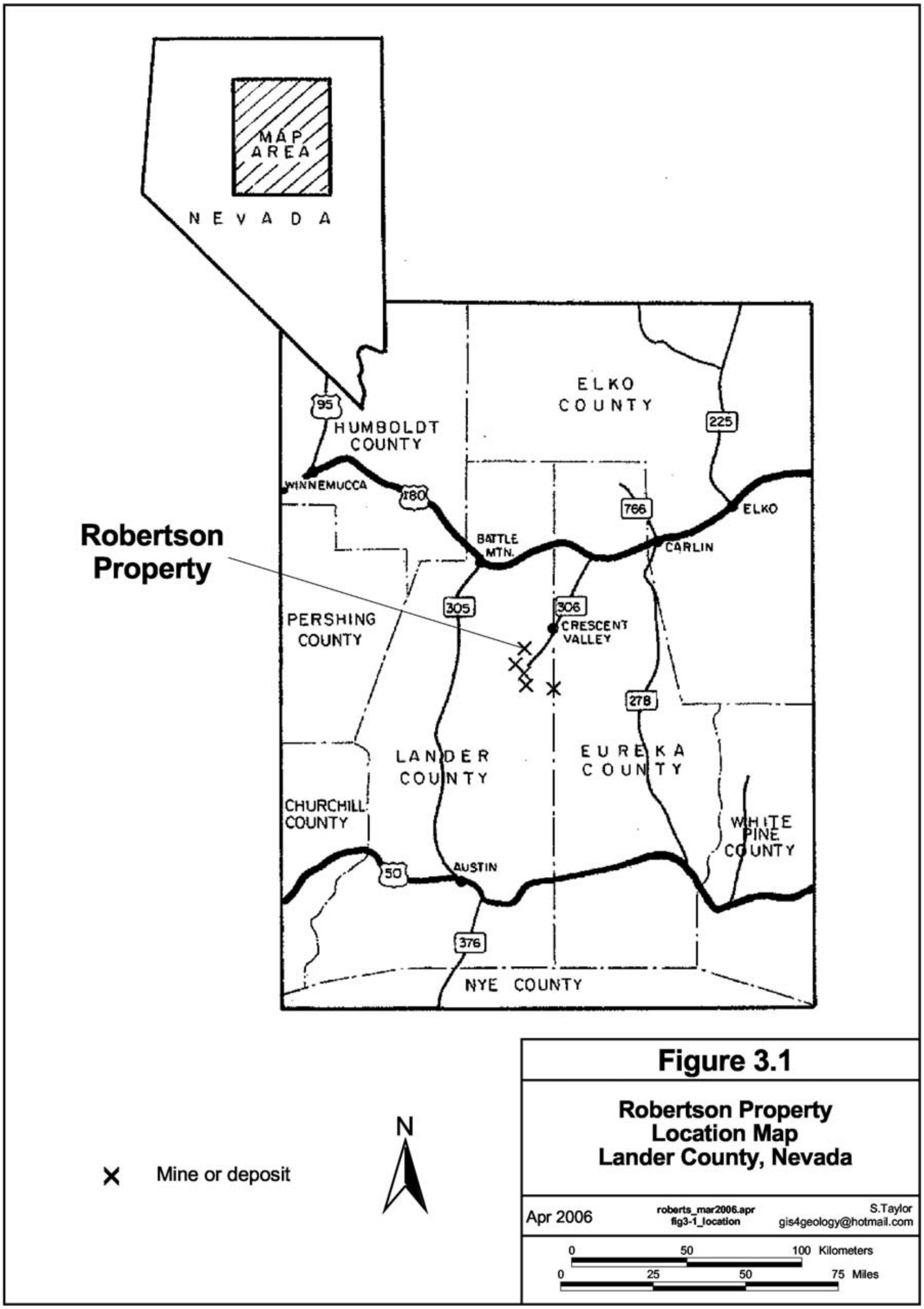


Figure 4-1: Robertson Property location map.

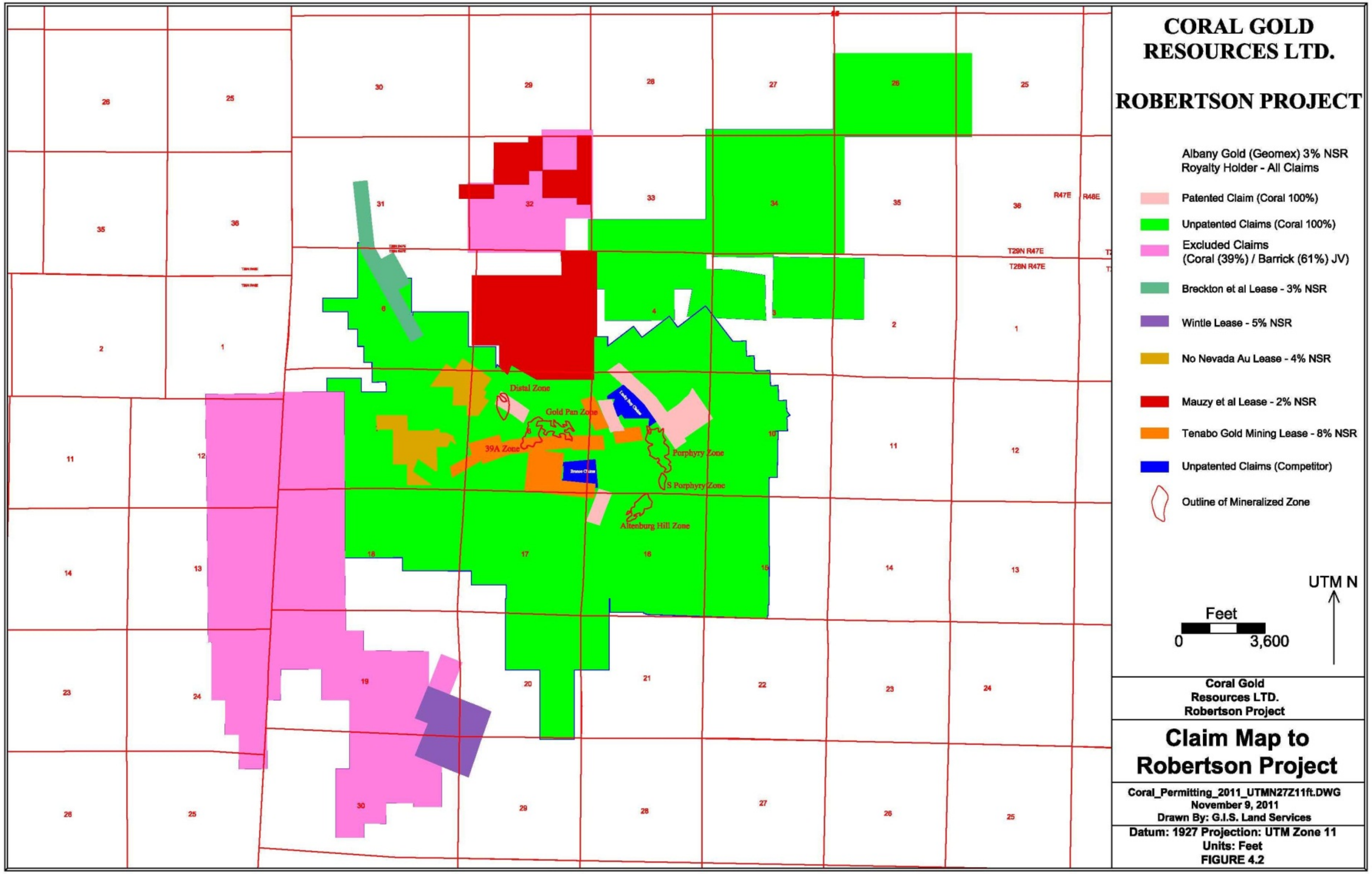


Figure 4-2: Claim Map

4.6 LIST OF CLAIMS

The following lists the claims owned or leased by Coral. This list was provided by Coral and the authors of this report have not confirmed the veracity of the claim list but have not reason to believe that the list represent those claims owned or leased by Coral.

There are two claim areas that are not held by Coral and are shown in dark blue on Figure 4-2 Claim Map. One of these claims "Bronko" is held by Newmont while the other "Lucky Boy" by the Filipini family. Coral are in ongoing negotiations to option both these group of claims.

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
Ajax Mine	NMC 108376	Lander	LODE	8	SW SE	28N	47E	MDB&M
Andy 10A Amd	NMC 185342	Lander	LODE	8	NW SW	28N	47E	MDB&M
Andy 11 Amd	NMC 194632	Lander	LODE	8	SE	28N	47E	MDB&M
Andy 12 Amd	NMC 280683	Lander	LODE	9	NW	28N	47E	MDB&M
Andy 13 Amd	NMC 304726	Lander	LODE	9	SW	28N	47E	MDB&M
Andy 14 Amd	NMC 345415	Lander	LODE	16	NW	28N	47E	MDB&M
Andy 1A Amd	NMC 185339	Lander	LODE	8	SW	28N	47E	MDB&M
Andy 2 Amd	NMC 185343	Lander	LODE	8	SW SE	28N	47E	MDB&M
Andy 3 Amd	NMC 185344	Lander	LODE	8	SE	28N	47E	MDB&M
Andy 4 Amd	NMC 185345	Lander	LODE	8	SE	28N	47E	MDB&M
Andy 5 Amd	NMC 185346	Lander	LODE	16	NW	28N	47E	MDB&M
Andy 6A Amd	NMC 185340	Lander	LODE	9	SW	28N	47E	MDB&M
Andy 7A Amd	NMC 185341	Lander	LODE	9	SW SE	28N	47E	MDB&M
Andy 8 Amd	NMC 185347	Lander	LODE	9	NW SW	28N	47E	MDB&M
Andy 9 Amd	NMC 185348	Lander	LODE	9	NW SW	28N	47E	MDB&M
April Fool	NMC 121043	Lander	LODE	7	NE	28N	47E	MDB&M
Blue Nugget	NMC 92283	Lander	LODE	33	NW SW	29N	47E	MDB&M
Blue Nugget 1	NMC 92282	Lander	LODE	33	NW	29N	47E	MDB&M
Blue Nugget 2	NMC 289435	Lander	LODE	32	NE	29N	47E	MDB&M
Blue Nugget 3	NMC 289436	Lander	LODE	32	NE	29N	47E	MDB&M
Blue Nugget 4	NMC 289437	Lander	LODE	32	NE	29N	47E	MDB&M
Blue Nugget 5	NMC 289438	Lander	LODE	32	NW	29N	47E	MDB&M
Blue Nugget 6	NMC 289439	Lander	LODE	32	NW	29N	47E	MDB&M
Blue Nugget 7	NMC 289440	Lander	LODE	32	NW	29N	47E	MDB&M
Blue Nugget 8	NMC 289441	Lander	LODE	31	NE	29N	47E	MDB&M
BLUE RIDGE	NMC 69209	Lander	LODE	20	SW	28N	47E	MDB&M
BLUE RIDGE 1	NMC 69210	Lander	LODE	20	SW	28N	47E	MDB&M
BLUE RIDGE 2	NMC 69211	Lander	LODE	29	NW	28N	47E	MDB&M
BLUE RIDGE 3	NMC 69212	Lander	LODE	30	NE	28N	47E	MDB&M
BLUE RIDGE 4	NMC 69213	Lander	LODE	30	NE	28N	47E	MDB&M
BLUE RIDGE 5	NMC 69214	Lander	LODE	19	SE	28N	47E	MDB&M
BLUE RIDGE 6	NMC 69215	Lander	LODE	30	NE	28N	47E	MDB&M
BLUE RIDGE 7	NMC 69216	Lander	LODE	30	NE	28N	47E	MDB&M
BLUE RIDGE 8	NMC 69217	Lander	LODE	19	SE	28N	47E	MDB&M
Blue Rock	NMC 121033	Lander	LODE	8	SW	28N	47E	MDB&M
Blue Rock 2	NMC 121034	Lander	LODE	8	SW	28N	47E	MDB&M
Bluebird 1	NMC 298637	Lander	LODE	7	NE	28N	47E	MDB&M
Bluebird 2	NMC 298638	Lander	LODE	6	SE	28N	47E	MDB&M
BO 1	NMC 618727	Lander	LODE	16	SW SE	28N	47E	MDB&M
BO 10	NMC 618736	Lander	LODE	15	NW SW	28N	47E	MDB&M
BO 11	NMC 618737	Lander	LODE	15	SW	28N	47E	MDB&M
BO 12	NMC 618738	Lander	LODE	15	NW SW	28N	47E	MDB&M
BO 13	NMC 618739	Lander	LODE	15	SW	28N	47E	MDB&M
BO 14	NMC 618740	Lander	LODE	15	NW SW	28N	47E	MDB&M
BO 15	NMC 618741	Lander	LODE	15	SW	28N	47E	MDB&M
BO 16	NMC 618742	Lander	LODE	15	NW SW	28N	47E	MDB&M
BO 17	NMC 618743	Lander	LODE	15	SW	28N	47E	MDB&M
BO 18	NMC 618744	Lander	LODE	15	NW SW	28N	47E	MDB&M
BO 19	NMC 618745	Lander	LODE	16	NE	28N	47E	MDB&M
BO 2	NMC 618728	Lander	LODE	16	NW SW	28N	47E	MDB&M
BO 20	NMC 618746	Lander	LODE	9	SE	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
BO 21	NMC 618747	Lander	LODE	16	NE	28N	47E	MDB&M
BO 22	NMC 618748	Lander	LODE	9	SE	28N	47E	MDB&M
BO 23	NMC 618749	Lander	LODE	16	NE	28N	47E	MDB&M
BO 24	NMC 618750	Lander	LODE	9	SE	28N	47E	MDB&M
BO 25	NMC 618751	Lander	LODE	15	NW	28N	47E	MDB&M
BO 26	NMC 618752	Lander	LODE	9	SE	28N	47E	MDB&M
BO 27	NMC 618753	Lander	LODE	15	NW	28N	47E	MDB&M
BO 28	NMC 618754	Lander	LODE	10	SW	28N	47E	MDB&M
BO 29	NMC 618755	Lander	LODE	15	NW	28N	47E	MDB&M
BO 3	NMC 618729	Lander	LODE	16	SE	28N	47E	MDB&M
BO 30	NMC 618756	Lander	LODE	10	SW	28N	47E	MDB&M
BO 31	NMC 618757	Lander	LODE	10	SW	28N	47E	MDB&M
BO 32	NMC 618758	Lander	LODE	10	SW	28N	47E	MDB&M
BO 33	NMC 618759	Lander	LODE	9	SE	28N	47E	MDB&M
BO 34	NMC 618760	Lander	LODE	10	SW	28N	47E	MDB&M
BO 35	NMC 618761	Lander	LODE	10	SW	28N	47E	MDB&M
BO 36	NMC 618762	Lander	LODE	10	SW SE	28N	47E	MDB&M
BO 37	NMC 618763	Lander	LODE	10	SW SE	28N	47E	MDB&M
BO 38	NMC 618764	Lander	LODE	10	SW SE	28N	47E	MDB&M
BO 39	NMC 618765	Lander	LODE	10	NW SW	28N	47E	MDB&M
BO 4	NMC 618730	Lander	LODE	16	NE SE	28N	47E	MDB&M
BO 5	NMC 618731	Lander	LODE	16	SE	28N	47E	MDB&M
BO 6	NMC 618732	Lander	LODE	16	NE SE	28N	47E	MDB&M
BO 7	NMC 618733	Lander	LODE	16	SE	28N	47E	MDB&M
BO 8	NMC 618734	Lander	LODE	16	NE SE	28N	47E	MDB&M
BO 9	NMC 618735	Lander	LODE	15	SW	28N	47E	MDB&M
Bolder Mine	NMC 108373	Lander	LODE	7	SE	28N	47E	MDB&M
BRIDGET 1	NMC 57891	Lander	LODE	9	SE	28N	47E	MDB&M
BRIDGET 2	NMC 57892	Lander	LODE	9	SW	28N	47E	MDB&M
BRIDGET 3	NMC 57893	Lander	LODE	9	SW SE	28N	47E	MDB&M
BRIDGET 4	NMC 57894	Lander	LODE	9	SE	28N	47E	MDB&M
BRIDGET 5	NMC 319536	Lander	LODE	9	SW SE	28N	47E	MDB&M
CGR 1	NMC 1041042	Lander	LODE	16	SWSE	28N	47E	MDB&M
CGR 2	NMC 1041043	Lander	LODE	16	NWSW	28N	47E	MDB&M
CGR 3	NMC 1041044	Lander	LODE	7	SE	28N	47E	MDB&M
CGR 4	NMC 1041045	Lander	LODE	8	SW	28N	47E	MDB&M
CGR 5	NMC 1041046	Lander	LODE	8	SW	28N	47E	MDB&M
CGR 6	NMC 1041047	Lander	LODE	8	SW	28N	47E	MDB&M
CGR 7	NMC 1041048	Lander	LODE	8	SWSE	28N	47E	MDB&M
CGR 8	NMC 1041049	Lander	LODE	8	NNWSE	28N	47E	MDB&M
CGR 9	NMC 1041050	Lander	LODE	8	NE	28N	47E	MDB&M
CGR 10	NMC 1041051	Lander	LODE	7	NE	28N	47E	MDB&M
CGR 11	NMC 1041052	Lander	LODE	8	NW	28N	47E	MDB&M
CGR 12	NMC 1041053	Lander	LODE	7	NE	28N	47E	MDB&M
CGR 13	NMC 1041054	Lander	LODE	7	NE	28N	47E	MDB&M
CGR 14	NMC 1041055	Lander	LODE	7	NE	28N	47E	MDB&M
CGR 15	NMC 1041056	Lander	LODE	7	NESE	28N	47E	MDB&M
CGR 16	NMC 1041057	Lander	LODE	6	NW	28N	47E	MDB&M
CGR 17	NMC 1047714	Lander	LODE	9	SE	28N	47E	MDB&M
Char	NMC 108452	Lander	LODE	16	NE	28N	47E	MDB&M
CORAL 111	NMC 388655	Lander	LODE	7	NW	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section	Township	Range	Meridian
CORAL 112	NMC 388656	Lander	LODE	7 NW	28N	47E	MDB&M
CORAL 113	NMC 388657	Lander	LODE	7 NW	28N	47E	MDB&M
CORAL 114	NMC 388658	Lander	LODE	7 NW SW	28N	47E	MDB&M
CORAL 115	NMC 388659	Lander	LODE	7 SW	28N	47E	MDB&M
CORAL 116	NMC 388660	Lander	LODE	7 SW	28N	47E	MDB&M
CORAL 117	NMC 388661	Lander	LODE	7 SW	28N	47E	MDB&M
CORAL 118	NMC 388662	Lander	LODE	7 SW	28N	47E	MDB&M
CORAL 119	NMC 388663	Lander	LODE	18 NW	28N	47E	MDB&M
CORAL 120	NMC 388664	Lander	LODE	18 NE NW	28N	47E	MDB&M
CORAL 121	NMC 388665	Lander	LODE	7 SW SE	28N	47E	MDB&M
CORAL 122	NMC 388666	Lander	LODE	7 SW SE	28N	47E	MDB&M
CORAL 123	NMC 388667	Lander	LODE	7 SW SE	28N	47E	MDB&M
CORAL 124	NMC 388668	Lander	LODE	7 SW SE	28N	47E	MDB&M
CORAL 125	NMC 388669	Lander	LODE	7 NW SW	28N	47E	MDB&M
CORAL 126	NMC 388670	Lander	LODE	7 NE NW	28N	47E	MDB&M
CORAL 127	NMC 388671	Lander	LODE	7 NE NW	28N	47E	MDB&M
CORAL 133	NMC 399063	Lander	LODE	9 NW	28N	47E	MDB&M
CORAL 134	NMC 399064	Lander	LODE	9 NE NW	28N	47E	MDB&M
CORAL 135	NMC 399065	Lander	LODE	4 SW	28N	47E	MDB&M
CORAL 136	NMC 538705	Lander	LODE	16 NE	28N	47E	MDB&M
CORAL 137	NMC 538706	Lander	LODE	16 NE	28N	47E	MDB&M
CORAL 138	NMC 538707	Lander	LODE	16 NE	28N	47E	MDB&M
CORAL 71	NMC 388615	Lander	LODE	18 NW	28N	47E	MDB&M
CORAL 72	NMC 388616	Lander	LODE	18 NW	28N	47E	MDB&M
CORAL 75	NMC 388619	Lander	LODE	18 NW SW	28N	47E	MDB&M
CORAL 76	NMC 388620	Lander	LODE	18 NW	28N	47E	MDB&M
CORAL 77	NMC 388621	Lander	LODE	18 NW SW	28N	47E	MDB&M
CORAL 78	NMC 388622	Lander	LODE	18 NE NW	28N	47E	MDB&M
CORAL 79	NMC 388623	Lander	LODE	18 NE NW	28N	47E	MDB&M
CORAL 80	NMC 388624	Lander	LODE	18 NE NW	28N	47E	MDB&M
CP 1	NMC 622393	Lander	PLACER	16 SE	28N	47E	MDB&M
CP 10	NMC 622402	Lander	PLACER	15 NW SW	28N	47E	MDB&M
CP 11	NMC 622403	Lander	PLACER	15 SW	28N	47E	MDB&M
CP 13	NMC 622405	Lander	PLACER	15 SW	28N	47E	MDB&M
CP 15	NMC 622407	Lander	PLACER	15 SW SE	28N	47E	MDB&M
CP 17	NMC 622409	Lander	PLACER	16 NE	28N	47E	MDB&M
CP 18	NMC 622410	Lander	PLACER	16 NE	28N	47E	MDB&M
CP 19	NMC 622411	Lander	PLACER	16 NE	28N	47E	MDB&M
CP 2	NMC 622394	Lander	PLACER	16 NE SE	28N	47E	MDB&M
CP 20	NMC 622412	Lander	PLACER	16 NE	28N	47E	MDB&M
CP 21	NMC 622413	Lander	PLACER	15 NW	28N	47E	MDB&M
CP 22	NMC 622414	Lander	PLACER	15 NW	28N	47E	MDB&M
CP 23	NMC 622415	Lander	PLACER	15 NW	28N	47E	MDB&M
CP 24	NMC 622416	Lander	PLACER	15 NW	28N	47E	MDB&M
CP 3	NMC 622395	Lander	PLACER	16 SE	28N	47E	MDB&M
CP 35	NMC 622427	Lander	PLACER	9 SW SE	28N	47E	MDB&M
CP 36	NMC 622428	Lander	PLACER	9 SE	28N	47E	MDB&M
CP 37	NMC 622429	Lander	PLACER	9 SE	28N	47E	MDB&M
CP 4	NMC 622396	Lander	PLACER	16 NE SE	28N	47E	MDB&M
CP 40	NMC 622432	Lander	PLACER	10 SW	28N	47E	MDB&M
CP 42	NMC 622434	Lander	PLACER	10 SW	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
CP 5	NMC 622397	Lander	PLACER	16	SE	28N	47E	MDB&M
CP 6	NMC 622398	Lander	PLACER	16	NE SE	28N	47E	MDB&M
CP 7	NMC 622399	Lander	PLACER	15	SW	28N	47E	MDB&M
CP 8	NMC 622400	Lander	PLACER	15	NW SW	28N	47E	MDB&M
CP 9	NMC 622401	Lander	PLACER	15	SW	28N	47E	MDB&M
Cracker Jack 1	NMC 121028	Lander	LODE	8	NW SW	28N	47E	MDB&M
Cracker Jack 2	NMC 121029	Lander	LODE	8	NW	28N	47E	MDB&M
Cracker Jack 3	NMC 298639	Lander	LODE	7	NE	28N	47E	MDB&M
Cracker Jack 4	NMC 298640	Lander	LODE	7	NE	28N	47E	MDB&M
Cracker Jack 5	NMC 121030	Lander	LODE	7	NE SE	28N	47E	MDB&M
Cresent	NMC 108400	Lander	LODE	16	NE NW	28N	47E	MDB&M
Crown	NMC 108398	Lander	LODE	16	NE NW	28N	47E	MDB&M
Crown 1	NMC 108399	Lander	LODE	16	NE NW	28N	47E	MDB&M
Crown 2	NMC 108396	Lander	LODE	16	NE	28N	47E	MDB&M
Crown 3	NMC 108397	Lander	LODE	16	NE NW	28N	47E	MDB&M
DR 1	NMC 108392	Lander	LODE	9	SW	28N	47E	MDB&M
DR 2	NMC 108393	Lander	LODE	9	SW	28N	47E	MDB&M
DR 3	NMC 108394	Lander	LODE	9	SW SE	28N	47E	MDB&M
DR 4	NMC 108395	Lander	LODE	9	SE	28N	47E	MDB&M
DR 5A	NMC 251744	Lander	LODE	9	NW SW	28N	47E	MDB&M
DR 6	NMC 251745	Lander	LODE	9	NW SW	28N	47E	MDB&M
DR 7	NMC 251746	Lander	LODE	9	SW SE	28N	47E	MDB&M
Eakin Fr	NMC 121032	Lander	LODE	8	SW SE	28N	47E	MDB&M
ELNA	NMC 145519	Lander	PLACER	10	NW	28N	47E	MDB&M
ELNA 1	NMC 145520	Lander	PLACER	10	SW	28N	47E	MDB&M
ELNA 2	NMC 145521	Lander	PLACER	10	SW	28N	47E	MDB&M
ELNA MS	NMC 145522	Lander	MILLSITE	10	NW	28N	47E	MDB&M
FANNIE K	NMC 145523	Lander	LODE	9	NE SE	28N	47E	MDB&M
FANNIE K 1	NMC 145524	Lander	LODE	10	NW SW	28N	47E	MDB&M
FANNIE K 10	NMC 186564	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 11	NMC 186565	Lander	LODE	4	SE	28N	47E	MDB&M
FANNIE K 12	NMC 186566	Lander	LODE	4	SE	28N	47E	MDB&M
FANNIE K 13	NMC 186567	Lander	LODE	4	SE	28N	47E	MDB&M
FANNIE K 14	NMC 186568	Lander	LODE	3	NW SW	28N	47E	MDB&M
FANNIE K 15	NMC 186569	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 16	NMC 186570	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 17	NMC 186571	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 18	NMC 186572	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 19	NMC 186573	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 2	NMC 145525	Lander	LODE	9	SE	28N	47E	MDB&M
FANNIE K 20	NMC 186574	Lander	LODE	3	SW	28N	47E	MDB&M
FANNIE K 21	NMC 186575	Lander	LODE	10	NW	28N	47E	MDB&M
FANNIE K 22	NMC 186576	Lander	LODE	10	NE NW	28N	47E	MDB&M
FANNIE K 23	NMC 186577	Lander	LODE	10	IE NW SW	28N	47E	MDB&M
FANNIE K 24	NMC 186578	Lander	LODE	9	NE	28N	47E	MDB&M
FANNIE K 25	NMC 186579	Lander	LODE	9	NE	28N	47E	MDB&M
FANNIE K 26	NMC 186580	Lander	LODE	4	SE	28N	47E	MDB&M
FANNIE K 27	NMC 186581	Lander	LODE	4	SW SE	28N	47E	MDB&M
FANNIE K 28	NMC 186582	Lander	LODE	4	SW SE	28N	47E	MDB&M
FANNIE K 3	NMC 145526	Lander	LODE	9	SE	28N	47E	MDB&M
FANNIE K 4	NMC 186558	Lander	LODE	10	NW SW	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
FANNIE K 5	NMC 186559	Lander	LODE	10	NW SW	28N	47E	MDB&M
FANNIE K 6	NMC 186560	Lander	LODE	10	NW	28N	47E	MDB&M
FANNIE K 7	NMC 186561	Lander	LODE	9	NE	28N	47E	MDB&M
FANNIE K 8	NMC 186562	Lander	LODE	9	NE	28N	47E	MDB&M
FANNIE K 9	NMC 186563	Lander	LODE	3	SW	28N	47E	MDB&M
Fannie K Ext 1	NMC 314753	Lander	LODE	3	SW	28N	47E	MDB&M
Fannie K Ext 2	NMC 314754	Lander	LODE	4	SE	28N	47E	MDB&M
Fannie K Ext 3	NMC 314755	Lander	LODE	4	SE	28N	47E	MDB&M
Fannie K Ext 4	NMC 314756	Lander	LODE	9	NE NW	28N	47E	MDB&M
Fannie K Ext 5	NMC 314757	Lander	LODE	4	SW SE	28N	47E	MDB&M
Fannie K Ext 6	NMC 314758	Lander	LODE	4	SW SE	28N	47E	MDB&M
Fannie K P 1	NMC 240336	Lander	PLACER	10	SW	28N	47E	MDB&M
Fannie K P 10	NMC 240345	Lander	PLACER	3	SW	28N	47E	MDB&M
Fannie K P 13	NMC 240346	Lander	PLACER	3	SW	28N	47E	MDB&M
Fannie K P 14	NMC 240347	Lander	PLACER	3	SW	28N	47E	MDB&M
Fannie K P 15	NMC 240348	Lander	PLACER	10	NW	28N	47E	MDB&M
Fannie K P 16	NMC 240349	Lander	PLACER	10	NW	28N	47E	MDB&M
Fannie K P 17	NMC 240350	Lander	PLACER	10	SW	28N	47E	MDB&M
Fannie K P 18	NMC 240351	Lander	PLACER	10	SW	28N	47E	MDB&M
Fannie K P 19	NMC 240352	Lander	PLACER	9	SE	28N	47E	MDB&M
Fannie K P 2	NMC 240337	Lander	PLACER	10	NW	28N	47E	MDB&M
Fannie K P 20	NMC 240353	Lander	PLACER	9	SE	28N	47E	MDB&M
Fannie K P 21	NMC 240354	Lander	PLACER	9	SE	28N	47E	MDB&M
Fannie K P 22	NMC 240355	Lander	PLACER	9	NE	28N	47E	MDB&M
Fannie K P 23	NMC 240356	Lander	PLACER	9	NW	28N	47E	MDB&M
Fannie K P 24	NMC 240357	Lander	PLACER	4	SE	28N	47E	MDB&M
Fannie K P 25	NMC 240358	Lander	PLACER	4	SE	28N	47E	MDB&M
Fannie K P 29	NMC 240359	Lander	PLACER	4	SE	28N	47E	MDB&M
Fannie K P 3	NMC 240338	Lander	PLACER	10	NE	28N	47E	MDB&M
Fannie K P 30	NMC 240360	Lander	PLACER	4	SE	28N	47E	MDB&M
Fannie K P 32	NMC 240361	Lander	PLACER	9	NE	28N	47E	MDB&M
Fannie K P 33	NMC 240362	Lander	PLACER	9	NE	28N	47E	MDB&M
Fannie K P 34	NMC 240363	Lander	PLACER	9	SE	28N	47E	MDB&M
Fannie K P 35	NMC 240364	Lander	PLACER	9	SE	28N	47E	MDB&M
Fannie K P 4	NMC 240339	Lander	PLACER	10	NE	28N	47E	MDB&M
Fannie K P 5	NMC 240340	Lander	PLACER	10	NE NW	28N	47E	MDB&M
Fannie K P 6	NMC 240341	Lander	PLACER	10	NE	28N	47E	MDB&M
Fannie K P 7	NMC 240342	Lander	PLACER	10	NW	28N	47E	MDB&M
Fannie K P 8	NMC 240343	Lander	PLACER	10	NW	28N	47E	MDB&M
Fannie K P 9	NMC 240344	Lander	PLACER	0	SW	28N	47E	MDB&M
FBA 101	NMC 244321	Lander	PLACER	9	SE	28N	47E	MDB&M
FBA 102	NMC 244322	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 103	NMC 244323	Lander	PLACER	9	SE	28N	47E	MDB&M
FBA 104	NMC 244324	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 105	NMC 244325	Lander	PLACER	9	SE	28N	47E	MDB&M
FBA 106	NMC 244326	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 107	NMC 244327	Lander	PLACER	9	SE	28N	47E	MDB&M
FBA 108	NMC 244328	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 109	NMC 244329	Lander	PLACER	9	NW	28N	47E	MDB&M
FBA 110	NMC 244330	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 111	NMC 244331	Lander	PLACER	9	SW	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
FBA 112	NMC 244332	Lander	PLACER	8	SE	28N	47E	MDB&M
FBA 113	NMC 244333	Lander	PLACER	8	SE	28N	47E	MDB&M
FBA 114	NMC 244334	Lander	PLACER	8	SE	28N	47E	MDB&M
FBA 115	NMC 244335	Lander	PLACER	9	SE	28N	47E	MDB&M
FBA 116	NMC 244336	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 117	NMC 244337	Lander	PLACER	9	SW	28N	47E	MDB&M
FBA 118	NMC 304727	Lander	PLACER	16	NW	28N	47E	MDB&M
FBA 119	NMC 304728	Lander	PLACER	16	NW	28N	47E	MDB&M
FBA 120	NMC 304729	Lander	PLACER	16	NW	28N	47E	MDB&M
FBA 121	NMC 304730	Lander	PLACER	16	NW	28N	47E	MDB&M
FBA 122	NMC 304731	Lander	PLACER	16	NW	28N	47E	MDB&M
FBA 123	NMC 304732	Lander	PLACER	16	NW	28N	47E	MDB&M
FBA 124	NMC 304733	Lander	PLACER	16	NE NW	28N	47E	MDB&M
FBA 125	NMC 304734	Lander	PLACER	16	NE	28N	47E	MDB&M
Fossil 1	NMC 485485	Lander	PLACER	16	NE SE	28N	47E	MDB&M
Fossil 2	NMC 485486	Lander	PLACER	16	NE SE	28N	47E	MDB&M
Gary	NMC 121031	Lander	LODE	7	NE SE	28N	47E	MDB&M
GIV 1	NMC 108476	Lander	LODE	16	SW SE	28N	47E	MDB&M
GIV 2	NMC 108477	Lander	LODE	16	SW SE	28N	47E	MDB&M
GIV 3	NMC 108478	Lander	LODE	16	SW SE	28N	47E	MDB&M
GIV 4	NMC 108479	Lander	LODE	16	NW SW	28N	47E	MDB&M
GIV 5	NMC 108480	Lander	LODE	16	NW	28N	47E	MDB&M
GIV 6	NMC 120965	Lander	LODE	16	NW SW	28N	47E	MDB&M
Gold Leaf 1 Ext	NMC 191669	Lander	LODE	2	NW	28N	47E	MDB&M
Gold Leaf 10	NMC 381907	Lander	LODE	3	SW	28N	47E	MDB&M
Gold Leaf 11	NMC 316600	Lander	LODE	4	SW SE	28N	47E	MDB&M
Gold Leaf 12	NMC 316601	Lander	LODE	4	SE	28N	47E	MDB&M
Gold Leaf 13	NMC 316602	Lander	LODE	4	SE	28N	47E	MDB&M
Gold Leaf 14	NMC 316603	Lander	LODE	4	SE	28N	47E	MDB&M
Gold Leaf 15	NMC 316604	Lander	LODE	4	SE	28N	47E	MDB&M
Gold Leaf 16	NMC 316605	Lander	LODE	4	SW SE	28N	47E	MDB&M
Gold Leaf 2 Ext	NMC 191670	Lander	LODE	3	NE	28N	47E	MDB&M
Gold Leaf 3 Ext	NMC 191671	Lander	LODE	34	SW	29N	47E	MDB&M
Gold Leaf 4 Ext	NMC 221410	Lander	LODE	3	NW	28N	47E	MDB&M
Gold Leaf 5	NMC 381902	Lander	LODE	3	NW SW	28N	47E	MDB&M
Gold Leaf 6	NMC 381903	Lander	LODE	3	SW SE	28N	47E	MDB&M
Gold Leaf 7	NMC 381904	Lander	LODE	3	SW SE	28N	47E	MDB&M
Gold Leaf 8	NMC 381905	Lander	LODE	3	NW SW	28N	47E	MDB&M
Gold Leaf 9	NMC 381906	Lander	LODE	3	SW	28N	47E	MDB&M
Gold Link	NMC 121036	Lander	LODE	8	SW SE	28N	47E	MDB&M
Gold Note 1	NMC 121453	Lander	LODE	9	SE	28N	47E	MDB&M
Gold Note Mine	NMC 108382	Lander	LODE	9	SE	28N	47E	MDB&M
Gold Note 1	NMC 108383	Lander	LODE	16	NW	28N	47E	MDB&M
Gold Note 1A	NMC 245899	Lander	LODE	8	SE	28N	47E	MDB&M
Gold Note 2	NMC 121454	Lander	LODE	9	NE SE	28N	47E	MDB&M
Gold Note 2	NMC 108384	Lander	LODE	17	NE	28N	47E	MDB&M
Gold Note 2A	NMC 245900	Lander	LODE	8	SW SE	28N	47E	MDB&M
Gold Pan 1	NMC 121016	Lander	LODE	8	NE SE	28N	47E	MDB&M
Gold Pan 2	NMC 121017	Lander	LODE	8	NE SE	28N	47E	MDB&M
Gold Pan 3	NMC 121018	Lander	LODE	8	NE	28N	47E	MDB&M
Gold Pan 4	NMC 121019	Lander	LODE	8	NE	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
Gold Pan 5	NMC 121020	Lander	LODE	8	NE	28N	47E	MDB&M
Gold Rock 1	NMC 121014	Lander	LODE	8	NE	28N	47E	MDB&M
Gold Rock 2	NMC 121015	Lander	LODE	8	NE	28N	47E	MDB&M
Gold Zone	NMC 121027	Lander	LODE	8	. NW SW	28N	47E	MDB&M
Gray Rock	NMC 121037	Lander	LODE	8	SW	28N	47E	MDB&M
Gylding 1	NMC 121021	Lander	LODE	8	IE NW SV	28N	47E	MDB&M
Gylding 2	NMC 121022	Lander	LODE	8	NW SW	28N	47E	MDB&M
Gylding 3	NMC 121023	Lander	LODE	8	NE NW	28N	47E	MDB&M
Gylding 4	NMC 121024	Lander	LODE	8	NE NW	28N	47E	MDB&M
Gylding 5	NMC 121025	Lander	LODE	5	SW SE	28N	47E	MDB&M
Hard Climb	NMC 121042	Lander	LODE	7	NE NW	28N	47E	MDB&M
HATTISBURG	NMC 121472	Lander	PLACER	9	NE	28N	47E	MDB&M
Hercules Amd	NMC 108375	Lander	LODE	8	SE	28N	47E	MDB&M
Jake 1	NMC 832390	Lander	LODE	9	SE	28N	47E	MDB&M
Jake 10	NMC 832399	Lander	LODE	15	NW	28N	47E	MDB&M
Jake 11	NMC 832400	Lander	LODE	15	NW SW	28N	47E	MDB&M
Jake 12	NMC 832401	Lander	LODE	15	NW SW	28N	47E	MDB&M
Jake 13	NMC 832402	Lander	LODE	15	NW SW	28N	47E	MDB&M
Jake 14	NMC 832403	Lander	LODE	15	SW	28N	47E	MDB&M
Jake 15	NMC 832404	Lander	LODE	15	SW	28N	47E	MDB&M
Jake 16	NMC 832405	Lander	LODE	15	SW	28N	47E	MDB&M
Jake 2	NMC 832391	Lander	LODE	10	SW	28N	47E	MDB&M
Jake 3	NMC 832392	Lander	LODE	10	SW	28N	47E	MDB&M
Jake 4	NMC 832393	Lander	LODE	10	SW	28N	47E	MDB&M
Jake 5	NMC 832394	Lander	LODE	10	SW	28N	47E	MDB&M
Jake 6	NMC 832395	Lander	LODE	10	SW	28N	47E	MDB&M
Jake 7	NMC 832396	Lander	LODE	15	NW	28N	47E	MDB&M
Jake 8	NMC 832397	Lander	LODE	15	NW	28N	47E	MDB&M
Jake 9	NMC 832398	Lander	LODE	15	NW	28N	47E	MDB&M
Jiant Lode Mine	NMC 108372	Lander	LODE	8	SW	28N	47E	MDB&M
JRT 1	NMC 584820	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 10	NMC 584829	Lander	LODE	34	NE SE	29N	47E	MDB&M
JRT 11	NMC 584830	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 12	NMC 584831	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 13	NMC 584832	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 14	NMC 584833	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 15	NMC 584834	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 16	NMC 584835	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 17	NMC 584836	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 18	NMC 584837	Lander	LODE	34	NE	29N	47E	MDB&M
JRT 19	NMC 584838	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 2	NMC 584821	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 20	NMC 584839	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 21	NMC 584840	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 22	NMC 584841	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 23	NMC 584842	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 24	NMC 584843	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 25	NMC 584844	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 26	NMC 584845	Lander	LODE	34	SW	29N	47E	MDB&M
JRT 27	NMC 584846	Lander	LODE	34	NW SW	29N	47E	MDB&M
JRT 28	NMC 584847	Lander	LODE	34	NW SW	29N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
JRT 29	NMC 584848	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 3	NMC 584822	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 30	NMC 584849	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 31	NMC 584850	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 32	NMC 584851	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 33	NMC 584852	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 34	NMC 584853	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 35	NMC 584854	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 36	NMC 584855	Lander	LODE	34	NW	29N	47E	MDB&M
JRT 37	NMC 584856	Lander	LODE	33	SE	29N	47E	MDB&M
JRT 38	NMC 584857	Lander	LODE	33	SE	29N	47E	MDB&M
JRT 39	NMC 584858	Lander	LODE	33	SE	29N	47E	MDB&M
JRT 4	NMC 584823	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 40	NMC 584859	Lander	LODE	33	SE	29N	47E	MDB&M
JRT 41	NMC 584860	Lander	LODE	33	SW	29N	47E	MDB&M
JRT 42	NMC 584861	Lander	LODE	33	SW	29N	47E	MDB&M
JRT 43	NMC 584862	Lander	LODE	33	SW	29N	47E	MDB&M
JRT 44	NMC 584863	Lander	LODE	33	SW	29N	47E	MDB&M
JRT 45	NMC 584864	Lander	LODE	33	SW	29N	47E	MDB&M
JRT 46	NMC 584865	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 47	NMC 584866	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 48	NMC 584867	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 49	NMC 584868	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 5	NMC 584824	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 50	NMC 584869	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 51	NMC 584870	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 52	NMC 584871	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 53	NMC 584872	Lander	LODE	26	SW	29N	47E	MDB&M
JRT 54	NMC 584873	Lander	LODE	26	NW SW	29N	47E	MDB&M
JRT 55	NMC 584874	Lander	LODE	26	NW SW	29N	47E	MDB&M
JRT 56	NMC 584875	Lander	LODE	26	NW	29N	47E	MDB&M
JRT 57	NMC 584876	Lander	LODE	26	NW	29N	47E	MDB&M
JRT 58	NMC 584877	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 59	NMC 584878	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 6	NMC 584825	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 60	NMC 584879	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 61	NMC 584880	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 62	NMC 584881	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 63	NMC 584882	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 64	NMC 584883	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 65	NMC 584884	Lander	LODE	26	SE	29N	47E	MDB&M
JRT 66	NMC 584885	Lander	LODE	26		29N	47E	MDB&M
JRT 67	NMC 584886	Lander	LODE	26		29N	47E	MDB&M
JRT 68	NMC 584887	Lander	LODE	26	NE	29N	47E	MDB&M
JRT 69	NMC 584888	Lander	LODE	26	NE	29N	47E	MDB&M
JRT 7	NMC 584826	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 70	NMC 584889	Lander	LODE	4	NE	28N	47E	MDB&M
JRT 71	NMC 584890	Lander	LODE	4	NE	28N	47E	MDB&M
JRT 72	NMC 584891	Lander	LODE	4	NE	28N	47E	MDB&M
JRT 73	NMC 584892	Lander	LODE	4	NE NW	28N	47E	MDB&M
JRT 74	NMC 584893	Lander	LODE	4	NW	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
JRT 75	NMC 584894	Lander	LODE	4	NW	28N	47E	MDB&M
JRT 76	NMC 584895	Lander	LODE	4	NW	28N	47E	MDB&M
JRT 77	NMC 584896	Lander	LODE	4	NW	28N	47E	MDB&M
JRT 78	NMC 584897	Lander	LODE	4	NW	28N	47E	MDB&M
JRT 8	NMC 584827	Lander	LODE	34	SE	29N	47E	MDB&M
JRT 9	NMC 584828	Lander	LODE	34	NE SE	29N	47E	MDB&M
Jumbo Mine	NMC 108374	Lander	LODE	8	SW	28N	47E	MDB&M
June #1	NMC 123439	Lander	LODE	6	NE	28N	47E	MDB&M
June #2	NMC 123440	Lander	LODE	6	SE	28N	47E	MDB&M
June #3	NMC 123441	Lander	LODE	6	NW	28N	47E	MDB&M
June #4	NMC 123442	Lander	LODE	31	SW	29N	47E	MDB&M
June #5	NMC 123443	Lander	LODE	31	SW	29N	47E	MDB&M
June #6	NMC 123444	Lander	LODE	6	NE	28N	47E	MDB&M
Lander Ranch	NMC 299862	Lander	LODE	5	NE	28N	47E	MDB&M
Lander Ranch 1	NMC 291643	Lander	LODE	5	NW	28N	47E	MDB&M
Lander Ranch 10	NMC 304592	Lander	LODE	5	NW SW	28N	47E	MDB&M
Lander Ranch 11	NMC 299867	Lander	LODE	5	NE	28N	47E	MDB&M
Lander Ranch 12	NMC 304593	Lander	LODE	5	NE SE	28N	47E	MDB&M
Lander Ranch 13	NMC 299868	Lander	LODE	5	NE	28N	47E	MDB&M
Lander Ranch 14	NMC 304594	Lander	LODE	5	NE SE	28N	47E	MDB&M
Lander Ranch 15	NMC 299869	Lander	LODE	5	NE	28N	47E	MDB&M
Lander Ranch 16	NMC 304595	Lander	LODE	5	NE SE	28N	47E	MDB&M
Lander Ranch 17	NMC 299870	Lander	LODE	5	NE	28N	47E	MDB&M
Lander Ranch 18	NMC 304596	Lander	LODE	5	NE SE	28N	47E	MDB&M
Lander Ranch 19	NMC 304597	Lander	LODE	5	SE	28N	47E	MDB&M
Lander Ranch 2	NMC 291644	Lander	LODE	5	NW SW	28N	47E	MDB&M
Lander Ranch 20	NMC 304598	Lander	LODE	5	SE	28N	47E	MDB&M
Lander Ranch 21	NMC 304599	Lander	LODE	5	SE	28N	47E	MDB&M
Lander Ranch 22	NMC 304600	Lander	LODE	5	SE	28N	47E	MDB&M
Lander Ranch 23	NMC 304601	Lander	LODE	5	SW SE	28N	47E	MDB&M
Lander Ranch 24	NMC 304602	Lander	LODE	5	SW	28N	47E	MDB&M
Lander Ranch 25	NMC 304603	Lander	LODE	5	SW	28N	47E	MDB&M
Lander Ranch 3	NMC 291645	Lander	LODE	5	NW	28N	47E	MDB&M
Lander Ranch 4	NMC 291646	Lander	LODE	5	NW SW	28N	47E	MDB&M
Lander Ranch 5	NMC 299864	Lander	LODE	5	NW	28N	47E	MDB&M
Lander Ranch 6	NMC 304590	Lander	LODE	5	NW SW	28N	47E	MDB&M
Lander Ranch 7	NMC 299865	Lander	LODE	5	NW	28N	47E	MDB&M
Lander Ranch 8	NMC 304591	Lander	LODE	5	NW SW	28N	47E	MDB&M
Lander Ranch 9	NMC 299866	Lander	LODE	5	NE NW	28N	47E	MDB&M
Lander Ranch ext	NMC 299863	Lander	LODE	5	NE	28N	47E	MDB&M
Little Gem Ext	NMC 121026	Lander	LODE	5	SW	28N	47E	MDB&M
Molly 1	NMC 121038	Lander	LODE	7	SE	28N	47E	MDB&M
Molly 2	NMC 121039	Lander	LODE	7	SW SE	28N	47E	MDB&M
Molly 3	NMC 121040	Lander	LODE	7	SE	28N	47E	MDB&M
Molly 4	NMC 121041	Lander	LODE	7	SE	28N	47E	MDB&M
New Ray	NMC 108377	Lander	LODE	8	SE	28N	47E	MDB&M
New Ray Fr	NMC 108378	Lander	LODE	9	NW SW	28N	47E	MDB&M
Our Faith	NMC 298636	Lander	LODE	5	SW	28N	47E	MDB&M
Phoenix Mine	NMC 108381	Lander	LODE	8	SE	28N	47E	MDB&M
Phoenix	NMC 121452	Lander	LODE	9	NE SE	28N	47E	MDB&M
Phoenix 1	NMC 108385	Lander	LODE	16	NW	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
Phoenix 2	NMC 108386	Lander	LODE	8	SE	28N	47E	MDB&M
Phoenix 3	NMC 108387	Lander	LODE	8	SE	28N	47E	MDB&M
Phoenix 4	NMC 108388	Lander	LODE	8	SE	28N	47E	MDB&M
Phoenix 5	NMC 108389	Lander	LODE	9	SW	28N	47E	MDB&M
Phoenix 6	NMC 108390	Lander	LODE	9	SW	28N	47E	MDB&M
Phoenix B	NMC 245898	Lander	LODE	8	SE	28N	47E	MDB&M
Rob 10A	NMC 943457	Lander	LODE	7	SW	28N	47E	MDB&M
Rob 1A	NMC 943456	Lander	LODE	8	NE	28N	47E	MDB&M
Rob 2A	NMC 943455	Lander	LODE	8	NE	28N	47E	MDB&M
Rob 5	NMC 936375	Lander	LODE	8	SW SE	28N	47E	MDB&M
Rob 6	NMC 936376	Lander	LODE	8	SW SE	28N	47E	MDB&M
Rob 7	NMC 936377	Lander	LODE	8	NW SW	28N	47E	MDB&M
Rob 8	NMC 936378	Lander	LODE	17	NW	28N	47E	MDB&M
Rob 9	NMC 936379	Lander	LODE	7	SE	28N	47E	MDB&M
Rod B 1	NMC 268011	Lander	LODE	7	SW SE	28N	47E	MDB&M
Rod B 2	NMC 268012	Lander	LODE	7	NW SW	28N	47E	MDB&M
Rod B 3	NMC 60952	Lander	LODE	7	SW SE	28N	47E	MDB&M
Rod B 4	NMC 60953	Lander	LODE	7	SW SE	28N	47E	MDB&M
Rod B 5	NMC 268013	Lander	LODE	7	NW SW	28N	47E	MDB&M
Rod B 6	NMC 268014	Lander	LODE	7	NE NW	28N	47E	MDB&M
Rod B 7	NMC 60954	Lander	LODE	7	NE SE	28N	47E	MDB&M
RUF 1	NMC 108404	Lander	LODE	16	SW	28N	47E	MDB&M
RUF 10	NMC 108413	Lander	LODE	20	NE	28N	47E	MDB&M
RUF 11	NMC 108414	Lander	LODE	17	SW SE	28N	47E	MDB&M
RUF 12	NMC 108415	Lander	LODE	17	SW SE	28N	47E	MDB&M
RUF 13	NMC 108416	Lander	LODE	20	NE NW	28N	47E	MDB&M
RUF 14	NMC 108417	Lander	LODE	20	NE NW	28N	47E	MDB&M
RUF 15	NMC 108418	Lander	LODE	20	NE NW	28N	47E	MDB&M
RUF 16	NMC 108419	Lander	LODE	20	NE SE	28N	47E	MDB&M
RUF 17	NMC 108420	Lander	LODE	20	SE	28N	47E	MDB&M
RUF 18	NMC 108421	Lander	LODE	20	SE	28N	47E	MDB&M
RUF 19	NMC 108422	Lander	LODE	20	SE	28N	47E	MDB&M
RUF 2	NMC 108405	Lander	LODE	17	SE	28N	47E	MDB&M
RUF 20	NMC 108423	Lander	LODE	20	SE	28N	47E	MDB&M
RUF 21	NMC 735751	Lander	LODE	16	SW	28N	47E	MDB&M
RUF 22	NMC 735752	Lander	LODE	17	SE	28N	47E	MDB&M
RUF 23	NMC 735753	Lander	LODE	17	SW SE	28N	47E	MDB&M
RUF 3	NMC 108406	Lander	LODE	20	NE	28N	47E	MDB&M
RUF 4	NMC 108407	Lander	LODE	17	SE	28N	47E	MDB&M
RUF 5	NMC 108408	Lander	LODE	20	NE	28N	47E	MDB&M
RUF 6	NMC 108409	Lander	LODE	20	NE	28N	47E	MDB&M
RUF 7	NMC 108410	Lander	LODE	20	NE	28N	47E	MDB&M
RUF 8	NMC 108411	Lander	LODE	20	NE	28N	47E	MDB&M
RUF 9	NMC 108412	Lander	LODE	20	NE	28N	47E	MDB&M
Sage	NMC 108401	Lander	LODE	16	NW	28N	47E	MDB&M
Silver Safe	NMC 121035	Lander	LODE	8	SW SE	28N	47E	MDB&M
S&M 1	NMC 4636	Lander	PLACER	8	SE	28N	47E	MDB&M
S&M 2	NMC 4637	Lander	PLACER	8	SE	28N	47E	MDB&M
S&M 3	NMC 4638	Lander	PLACER	8	SE	28N	47E	MDB&M
S&M 4	NMC 4639	Lander	PLACER	9	SW	28N	47E	MDB&M
Small Wonder Fr	NMC 121045	Lander	LODE	6	SW SE	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
Standard Mine	NMC 108380	Lander	LODE	8	SE	28N	47E	MDB&M
T&E 1	NMC 108481	Lander	LODE	8	SW SE	28N	47E	MDB&M
T&E 2	NMC 108482	Lander	LODE	9	SW	28N	47E	MDB&M
TENABO 1	NMC 87260	Lander	LODE	9	NE SW SE	28N	47E	MDB&M
TENABO 2	NMC 87261	Lander	LODE	9	SW SE	28N	47E	MDB&M
TENABO 3 Fr	NMC 57902	Lander	LODE	9	SE	28N	47E	MDB&M
TENABO 4	NMC 57903	Lander	LODE	9	NE SE	28N	47E	MDB&M
Tenabo Surprise Fr	NMC 57904	Lander	LODE	9	NW SW SE	28N	47E	MDB&M
TIGER 1	NMC 57889	Lander	LODE	9	NW SW	28N	47E	MDB&M
TIGER 2	NMC 57890	Lander	LODE	9	SW SE	28N	47E	MDB&M
Total Wreck	NMC 121044	Lander	LODE	6	SE	28N	47E	MDB&M
Try 100	NMC 330615	Lander	LODE	6	SE	28N	47E	MDB&M
TRY 37	NMC 2721	Lander	LODE	17	NE NW	28N	47E	MDB&M
TRY 38	NMC 2722	Lander	LODE	17	NE	28N	47E	MDB&M
TRY 39	NMC 2723	Lander	LODE	17	NE NW	28N	47E	MDB&M
TRY 40	NMC 2720	Lander	LODE	17	NE	28N	47E	MDB&M
TRY 41	NMC 2719	Lander	LODE	17	NE NW	28N	47E	MDB&M
TRY 42	NMC 2718	Lander	LODE	17	NE	28N	47E	MDB&M
TRY 43	NMC 2717	Lander	LODE	17	NW SW	28N	47E	MDB&M
TRY 44	NMC 2716	Lander	LODE	17	NE SE	28N	47E	MDB&M
TRY 45	NMC 2715	Lander	LODE	17	SW SE	28N	47E	MDB&M
TRY 46	NMC 2714	Lander	LODE	17	SE	28N	47E	MDB&M
TRY 47	NMC 2713	Lander	LODE	17	SW SE	28N	47E	MDB&M
TRY 48	NMC 2712	Lander	LODE	17	SE	28N	47E	MDB&M
TRY 49	NMC 2711	Lander	LODE	17	SW	28N	47E	MDB&M
TRY 50	NMC 2710	Lander	LODE	17	SW	28N	47E	MDB&M
TRY 51	NMC 2709	Lander	LODE	17	NW SW	28N	47E	MDB&M
TRY 52	NMC 2708	Lander	LODE	17	NW SW	28N	47E	MDB&M
TRY 53	NMC 2707	Lander	LODE	17	NW	28N	47E	MDB&M
TRY 54	NMC 2706	Lander	LODE	17	NW	28N	47E	MDB&M
TRY 55	NMC 2705	Lander	LODE	17	NW	28N	47E	MDB&M
TRY 56	NMC 2704	Lander	LODE	17	NW	28N	47E	MDB&M
TRY 57	NMC 2703	Lander	LODE	17	NW	28N	47E	MDB&M
TRY 58	NMC 2702	Lander	LODE	17	NW	28N	47E	MDB&M
TRY 63	NMC 2701	Lander	LODE	16	SW	28N	47E	MDB&M
TRY 64	NMC 2700	Lander	LODE	16	SW	28N	47E	MDB&M
TRY 65	NMC 2699	Lander	LODE	16	SW	28N	47E	MDB&M
TRY 66	NMC 2698	Lander	LODE	17	SE	28N	47E	MDB&M
TRY 67	NMC 2697	Lander	LODE	17	SW SE	28N	47E	MDB&M
TRY 68	NMC 2696	Lander	LODE	17	SW	28N	47E	MDB&M
TRY 70	NMC 2695	Lander	LODE	7	SE	28N	47E	MDB&M
TRY 74	NMC 2694	Lander	LODE	8	SE	28N	47E	MDB&M
TRY 75	NMC 2693	Lander	LODE	16	NW SW	28N	47E	MDB&M
TRY 80	NMC 2692	Lander	LODE	7	SE	28N	47E	MDB&M
TRY 81	NMC 2691	Lander	LODE	18	NE	28N	47E	MDB&M
TRY 82	NMC 2690	Lander	LODE	18	NE	28N	47E	MDB&M
TRY 83	NMC 2689	Lander	LODE	18	NE	28N	47E	MDB&M
TRY 84	NMC 2688	Lander	LODE	18	NE SE	28N	47E	MDB&M
TRY 85	NMC 2687	Lander	LODE	18	SE	28N	47E	MDB&M
TRY 86	NMC 2686	Lander	LODE	7	SE	28N	47E	MDB&M
TRY 88	NMC 2685	Lander	LODE	7	SE	28N	47E	MDB&M

Claim Name	BLM Serial No.	County	Type	Section		Township	Range	Meridian
TRY 89	NMC 2684	Lander	LODE	7	SE	28N	47E	MDB&M
TRY 90	NMC 2683	Lander	LODE	8	SW	28N	47E	MDB&M
TRY 91	NMC 245901	Lander	LODE	7	NW SW	28N	47E	MDB&M
TRY 92	NMC 245902	Lander	LODE	7	NW SW	28N	47E	MDB&M
TRY 93	NMC 245903	Lander	LODE	7	NW	28N	47E	MDB&M
TRY 94	NMC 280680	Lander	LODE	9	NW	28N	47E	MDB&M
TRY 95	NMC 280681	Lander	LODE	9	NW	28N	47E	MDB&M
TRY 96	NMC 280682	Lander	LODE	9	NW	28N	47E	MDB&M
TRY 97	NMC 330612	Lander	LODE	6	SE	28N	47E	MDB&M
TRY 98	NMC 330613	Lander	LODE	6	SE	28N	47E	MDB&M
TRY 99	NMC 330614	Lander	LODE	6	SE	28N	47E	MDB&M
View 1	NMC 928587	Lander	LODE	5	SW	28N	47E	MDB&M
View 10	NMC 928596	Lander	LODE	6	NW SW	28N	47E	MDB&M
View 11	NMC 928597	Lander	LODE	6	NW	28N	47E	MDB&M
View 13	NMC 935489	Lander	LODE	6	NW	28N	47E	MDB&M
View 14	NMC 935490	Lander	LODE	6	SW	28N	47E	MDB&M
View 15	NMC 935491	Lander	LODE	6	SW	28N	47E	MDB&M
View 16	NMC 935492	Lander	LODE	6	SW	28N	47E	MDB&M
View 17	NMC 980312	Lander	LODE	6	NW	28N	47E	MDB&M
View 18	NMC 980313	Lander	LODE	6	NW	28N	47E	MDB&M
View 19	NMC 980314	Lander	LODE	6	SW	28N	47E	MDB&M
View 2	NMC 928588	Lander	LODE	5	SW	28N	47E	MDB&M
View 20	NMC 980315	Lander	LODE	6	NE	28N	47E	MDB&M
View 21	NMC 980316	Lander	LODE	6	SE	28N	47E	MDB&M
View 22	NMC 980317	Lander	LODE	6	NW	28N	47E	MDB&M
View 23	NMC 980318	Lander	LODE	6	SW	28N	47E	MDB&M
View 24	NMC 980319	Lander	LODE	6	SW	28N	47E	MDB&M
View 3	NMC 928589	Lander	LODE	6	NE SE	28N	47E	MDB&M
View 4	NMC 928590	Lander	LODE	6	SE	28N	47E	MDB&M
View 5	NMC 928591	Lander	LODE	6	NE SE	28N	47E	MDB&M
View 6	NMC 928592	Lander	LODE	6	NE SE	28N	47E	MDB&M
View 7	NMC 928593	Lander	LODE	6	NE SE	28N	47E	MDB&M
View 8	NMC 928594	Lander	LODE	6	NW SW	28N	47E	MDB&M
White Cloud	NMC 108402	Lander	LODE	16	NW	28N	47E	MDB&M
White Cloud Fr	NMC 108403	Lander	LODE	9	SW	28N	47E	MDB&M
WYOD	NMC 57907	Lander	LODE	8	SE	28N	47E	MDB&M
WYOD 1	NMC 57908	Lander	LODE	8	SE	28N	47E	MDB&M
WYOD 2	NMC 57909	Lander	LODE	8	SE	28N	47E	MDB&M
WYOD 3	NMC 57910	Lander	LODE	8	SE	28N	47E	MDB&M
XRay	NMC 108379	Lander	LODE	9	SW	28N	47E	MDB&M

SECTION 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The property is 58 miles southeast of Battle Mountain and 70 miles southwest of Elko, NV (Figure 4-1). From Battle Mountain, the county seat, the property is reached by traveling 28 miles east on Interstate Highway 80, then 29 miles south on Nevada Highway 306 which passes through to the Coral property turn off. The property is reached by driving two miles west on a well maintained gravel road. A network of unimproved dirt roads and tracks provide access to the remainder of the property.

5.2 CLIMATE AND PHYSIOGRAPHY

The Robertson Property is situated in the Tenabo sub-district of the Bullion mining district, within a series of low foothills on the extreme eastern flank of the Shoshone Mountains near the abandon town site of Tenabo. The climate is arid to semi-arid, with high annual insolation, low annual precipitation and large daily temperature fluctuations. Altitude on the project varies from 5,000 ft to 6,281 ft above mean sea level. Vegetation is typical of the Great Basin Desert Shrub Steppe, comprised of communities of large sage brush, rabbit brush, Sandburg bluegrass and varieties of forbs. Average annual precipitation at the site is less than 7 inches per year, mostly from winter snowfall and sporadic summer afternoon thunderstorms. In the region, the average maximum temperature is 63°F and the average minimum temperature is 30°F. Mid-winter day-time temperatures average 24°F and mid-summer day-time temperatures average 90°F.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

Land ownership within the Robertson Property project area consists of federal surface and minerals within the Shoshone-Eureka Resource Area, administered by the Mount Lewis Field Office of the Bureau of Land Management. According to the BLM, principal land uses within the project area are mining, wildlife habitat and livestock grazing. Portions of the project area are subject to BLM livestock grazing permits.

Because the “Komp” well, the previous source of water for past mining and exploration activities at Robertson Property is no longer operational, water is available with permission, from the Barrick Cortez mining operation as a result of their de-watering operations at the nearby Pipeline mine. A series of infiltration ponds are located Section 22, T. 28 N., R. 47 E. which can provide sufficient water for future exploration activities. Water to support mining operations at Robertson likely will likely require new production wells.

A paved state highway provides access to within 2 miles of the property and a major power line that currently provides electrical power to a nearby operating mine crosses the east edge of the property. Surface rights at Robertson are sufficient to support a major mining operation. Experienced mining personnel are available in the nearby communities of Elko and Battle Mountain.

SECTION 6.0 HISTORY

6.1 INTRODUCTION

The Robertson property is located in the Tenabo area, a sub-district of the Bullion mining district. Historic lode mining in the sub-district dates from about 1905 with a total production of 20,000 ounces of gold credited to the mines of the area (Stager, 1977). Placer gold was discovered in many of the dry washes in the Tenabo area in 1916. During the period 1937-39, a small dragline dredge and washing plant operated in the district, and a dredge was reported by Humphrey (1945) to be operating in lower Mill Gulch in 1945. Placer production is estimated to have yielded $25,000 worth of gold and minor amounts scheelite (Johnson, 1973).

During the period 1966-70, a number of companies explored the district in search of porphyry copper-style mineralization. In 1968, while drilling a series of shallow rotary holes near the Gold Pan mine, Superior Oil discovered a small, but relatively high-grade zone of gold at shallow depths in what is now known as the Gold Pan Zone. However, with additional drilling, Superior quickly lost interest in the district. They were soon followed by a number of mainly Vancouver-based junior mining companies, including Placer Development (1974-75), Teck Corporation (1977), Aaron Mining Ltd. (1975-86), and E & B Exploration Ltd. (1980-81), all of whom sporadically explored the Tenabo area with limited success. A summary of the drilling completed by these companies prior to Coral's involvement (1986) is presented in Table 6.1. The locations of these holes are shown in Figure 5-1.

Table 6.1: Summary of Pre-Coral Drilling Activities at Robertson.

Company	Date of Activity	Number and Type of Holes Drilled	Drill Footage(ft)	Target
Superior Oil	1968-70	92(?)Conv. Rotary	c. 32,000	Gold Pan
Placer Dev.	1973-74	23(?)Conv. Rotary	c. 3,500	none
Teck Corp.	1977	none	none	none
Aaron	1977	7 Conv. Rotary	c.300	Gold Quartz
E & B Exploration	1980-81	148 RC(?)	30,807	Gold Pan
Totals		**270**	**65,407**	

Modern open pit mining and heap leaching began as early as 1974, when Aaron Mines, Ltd., initiated a pilot leach operation. Aaron placed about 15,000 tons of oxide material mined from the "glory hole" and waste dumps of the historic Gold Quartz mine on a small leach pad, from which 377 ounces of gold were recovered (Stevenson & Assoc., 1977). During the period 1978-80, Aaron placed 38,400 tons of ore on a second larger pad, from which they recovered 517 ounces of gold (Sampson, 1988). Also during that period Aaron continued exploration and began consolidating and acquiring claims in the district.

6.2 CORAL GOLD CORP. (1986-1989)

In 1986, Coral acquired Aaron's interest in the property and immediately began a series of major drilling programs beginning in 1986 and continuing until 1989. By 1988, Coral had reportedly defined a reserve of 11 million tons averaging 0.04 ozAu/t (NBMG, 1989). Mining operations commenced in 1988, but were suspended less than one year later. During the operating life of the

Robertson mine, approximately 350,000 tons of low-grade material was placed on leach pads from which about 6,200 ounces of gold were recovered.

During the period 1986 through 1989, Coral completed approximately 380 RC holes and 7 DDH, totaling about 109,377 ft. Much of this drilling was focused in four resources areas; Gold Pan, Gold Quartz, Gold Quartz extension (also called Gold Quartz West) and in the Triplet Gulch area. The purpose of this drilling was to determine the limits and continuity of mineralization within these zones. Nearly all of the RC holes were drilled vertically to an average depth of about 300 ft. The locations of the Coral drill holes and the four resource areas are shown in Figure 6-2.

During the later stages of Coral's exploration program, they completed two "deep" RC holes that reached depths of 1,400 ft and 1,810 ft, respectively. In addition to resource definition, Coral also embarked on a program of district-wide exploratory and follow-up drilling of numerous surface anomalies.

6.3 AMAX GOLD (1990-1996)

In 1990, Coral and Amax Gold Exploration entered into an Amended and Restated Option and Earn-In Agreement in which Amax could earn a 60 percent interest in the property by producing a bankable feasibility study. From 1990, until they withdrew from the venture in 1996, Amax completed a district-wide exploration program that included drilling 342 RC holes and 62 DDH, totaling over 176,000 feet. The locations of the Amax drill holes are shown on Figure 6-3.

During this time period, Amax discovered a number of mineralized zones which comprised a low-grade, drill-indicated resource of over 1 million ounces of gold. These resources are summarized below in Table 6.2. In 1994, Amax began close-spaced grid drilling of the Porphyry Zone resource to define a mineable reserve to serve as a basis for completing a bankable feasibility study (Amax, 1994). The resulting reserve of about 14 million tons averaging 0.019 and containing about 180,000 recoverable ounces of gold was deemed marginal at a gold price of $400/oz. **The historic mineral resource and mineral reserve estimates cited in Table 6.2 and discussed above are presented for the purpose of historic background only and do not represent defined mineral resources on the property. In addition, the classification of these historic mineral resources and reserves do not conform to the National Instrument 43-101.**

Table 6.2: Robertson Property Drill-Indicated Resources (1996).

Mineralized Zones	**Million Tons**	**Grade (ozAu/t)**	**Contained Ounces**
Altenburg Hill**	3.5	0.018	63,900
Porphyry Zone*	20	0.02	400,000
Gold Pan**	8.3	0.024	199,200
39A Zone	4.7	0.077	367,190
Total	**36.5**	**0.028**	**1,030,290**

* Includes AGI proven + probable reserve 13,556,125 tons at an average grade of 0.0188 (1994).
** Inferred mineral resources estimated by AGI (1996).

6.4 CORTEZ GOLD MINES (1998-1999)

In 1998, Cortez entered into an Option and Earn-In Agreement with Coral in which Cortez could earn a 70 percent interest in the Robertson property by producing a bankable feasibility study. The focus of their exploration was to expand the 39A Zone and test a number of outlying targets. During 1999, Cortez completed 46 RC drill holes and a single mud rotary hole, totaling 57,000 ft. Of the thirteen holes directed at expanding the 39A Zone only two holes, 99401 (130 ft/0.05 ozAu/t) and 99413 (80ft/0.163 ozAu/t), encountered significant mineralization. However, this drilling program did little to expand the resource. Of the remaining holes drilled by Cortez, only two holes (99406 and 99419) encountered significant mineralization. Both holes were designed to offset and (or) follow up existing drill intersections and surface gold anomalies. The locations of the Cortez drill holes are shown in Figure 5-4.

After completing this drilling program, Cortez declared its interest in renegotiating the terms of the Option Agreement. When Coral declined, Cortez subsequently terminated the agreement and did not earn an interest in the property.

6.5 CORAL GOLD RESOURCES LTD (2001-2010)

6.5.1 Coral Drilling Programs (2004-2010)

During 2004 and 2005, Coral conducted three drilling programs consisting of 32 RC holes totaling 24,020 ft on the Robertson Property (McCusker, 2004, 2005). The focus of this exploration was to expand and further define the 39A Zone, test the “deep” Gold Pan Zone for extensions of the 39A Zone and offset previous mineralized intersections in the “distal target area”. The locations of the Coral drill holes are shown in Figure 6-5

During 2006, Coral completed a two phase drilling program. Phase I consisted of drilling 14 RC holes totaling 11,355 ft, which were completed in the immediate vicinity of the existing 39A Zone indicated mineral resource. The purpose of Phase I drilling was to define the “economic margins” of the 39A mineralized zone and to test the continuity of higher grade intersections between wide spaced drill holes. Phase II drilling consisted of drilling 32 RC holes totaling 24,260 ft, which were completed in the Distal Zone, on the northeast flank of Altenburg Hill, in the gravel-covered area between the Altenburg Hill and the Porphyry Zone and along a northeast striking structural zone in the Porphyry Zone. Several of these holes were also used to test the north and northwest edges of 39A Zone (Coral Gold, 2007). The locations of the 2006 Coral drill holes are shown in Figure 6-6.

In 2007, the NDEP required Coral to consolidate previous amendments to the Plan of Operation before they would consider approving a new amendment. As a result no drilling was conducted on any of the defined mineral resources during 2007. Instead Coral was able to complete two deep flooded RC holes (TV07-1 and TV07-2) totaling 6,440 ft, in the extreme northwest part of the Robertson claim block under a Notice of Intent (TRY/View Notice). The purpose of the 2007 deep drilling program was to discover high-grade Carlin-type gold deposits on the Robertson Property hosted by favorable strata in the lower plate of the Roberts Mountains thrust fault. Drill targets were selected based on the coincidence of multi-element soil and rock chip geochemical anomalies with mapped faults having strong geophysical expressions that identified certain structures as major fluid conduits (McCusker, 2008). The locations of the 2007 deep drill holes are shown in Figure 6-6

The 2008 drilling program consisted of 34 RC holes totaling 22,835 ft to further define and expand the Altenburg Hill, South Porphyry, 39A and Distal mineralized zones. A total of 15 holes (7,270 ft) were completed on Altenburg Hill and 6 holes (3,000 ft) were drilled in the S. Porphyry zones to offset and infill known mineralization. At the 39A zone, eight holes (6,645 ft) were completed in an effort to expand the resource to the northwest and northeast and 5 offset holes (5,920 ft) were drilled in the Distal zone.

No exploration was conducted at Robertson in 2009.

In 2010, Coral conducted a 12 hole RC program to evaluate the lower Triplet gulch resource (also called the South Zone resource). The drilling program consisted of 12 vertical RC holes totaling 8,000 ft, drilled on a grid with holes spaced 500 ft apart on lines also spaced 500 ft apart. Hole depths ranged from 600 ft to 1,000 ft. Only two of the 12 RC holes, CR10-7 and -8, were completed to 1,000 ft (McCusker, 2010). The locations of the 12 RC holes drilled during 2010 are shown in Figure 6-7.

Also in 2010, Coral completed 14 HQ diameter diamond core holes totaling 6,350 ft and ranging from 400 ft to 550 ft deep. This limited program was designed to twin 12 existing RC holes in the Altenburg Hill (9 holes), S. Porphyry (1 hole) and Gold Pan (4 holes) zones in order to evaluate the reliability of the RC drilling and to provide samples for metallurgical testing. The location of the 14 diamond drill holes are shown in Figure 6-7.

6.5.2 Coral Resource Studies (2001-2010)

Prior to 2001, the earliest resource estimate that was made on any of the currently defined resources was in 1988 by Mintec Inc., using various cutoff grades for the Gold Pan Zone. An estimate of the 39A Zone resource was made by Amax in 1993, based on only 21,000 ft of drilling in 25 holes and using a 0.02 ozAu/t cutoff grade. In 1994, Amax and MRDI jointly completed a resource estimate for the Porphyry Zone as part of a feasibility study. The study concluded that a proven + probable mineral reserve was present, but was deemed “marginal” at the current gold price ($420/oz).

In 1996, after completing grid and in-fill drilling in the Gold Pan and Altenburg Hill zones, Amax completed resources estimates for these mineralized zones using a 0.01 ozAu/t cutoff grade. Results of these studies indicated that while a low-grade “drill indicated resource” was present, it did not represent a potential “mineable reserve which is of interest to Amax” (Candee, 1996).

In 2001, Coral Gold contracted R.T. McCusker Geological consulting to prepare a review of previous drilling activities and provide a resource summary. The study concluded that at the prevailing gold prices of less than $300/oz, a resource containing about 11 million short tons of mineralized material averaging 0.053ozAu/t, at an 0.02ozAu/t cutoff grade, were present on the Robertson property (Table 6.3).

In late 2005, Barnes Engineering was contracted by Coral Gold Resources to undertake a Preliminary Assessment of the currently defined mineral resources at the Robertson Property. The purpose for of this study was to update the 2001 resource estimate, include results of the 2004-2005 Coral drilling programs (24,000 ft) in the estimate and examine the effect of higher gold price on the economics of the existing resources.

Table 6.3: Robertson Property Indicated Resource Estimate (McCusker, 2001).

	Cutoff Grade ozAu/t	Million tons	Grade ozAu/t	Contained Ounces
Zone				
39A	0.02	2.8	0.101	282,800
39A*	0.05	1.54	0.13	200,000
Porphyry	0.02	3.95	0.04	158,000
Gold Pan**	0.02	2.97	0.038	112,900
Altenburg Hill	0.02	1.25	0.024	30,000
Total		**10.97**	**0.053**	**583,700**

*****Note 39A resource estimate at 0.05ozAu/t cutoff not used in total calculation.
**From Mintec Inc., 1988 estimate; includes oxide + sulfide material.

In 2006, the Robertson mineral resources were reported using gold cutoff grades calculated by Barnes Engineering is shown in Table 6.4. Inferred mineral resources were calculated by R.T. McCusker Geological consulting and are presented in Table 6.5.

Table 6.4: Robertson Property Measured and Indicated Mineral Resources (Barnes, 2006).

	Measured mineral resources			Indicated mineral resources			Total measured and indicated		
Zone	Tons (000's)	Grade ozAu/t	Contained ozs (000's)	Tons (000's)	Grade ozAu/t	Contained ozs (000's)	Tons (000s)	Grade ozAu/t	Contained ozs (000's)
Porphyry[1]	10,600	0.020	212	2,100	0.018	37	12,700	0.020	249
39A/Gold Pan[2]				10,200	0.044	450	10,200	0.044	450
Total	**10,600**	**0.020**	**212**	**12,300**	**0.040**	**487**	**22,900**	**0.031**	**699**

(1) Calculated using a 0.010 ozAu/t cutoff grade. (2) Calculated using a 0.015 ozAu/t cutoff grade.

Table 6.5: Robertson Property Inferred Mineral Resources (McCusker, 2006)

	Inferred Mineral Resources		
Zone	**Tons (000's)**	**Grade (ozAu/t)**	**Contained ozs (000's)**
Altenburg Hill[1]	3,500	0.018	63
39A/Gold Pan[2]	4,900	0.039	192
Distal Target[3]	1,008	0.178	179
Total	**9,408**	**0.046**	**434**

(1), (2), (3) Estimates calculated using 0.01 ozAu/t, 0.015 ozAu/t and 0.05 ozAu/t cutoff grades respectively.

In 2007 Coral contracted Beacon Hill Consultants to prepare an updated mineral resource estimate as part of a new NI 43-101 technical report. The new resource study included the 2006 drilling results (35,615 ft) and examined the effect of higher gold prices on the resources. In January 2008, Beacon Hill reported an inferred mineral resource estimated to contain 91.3 million tons averaging 0.0253 ozAu/t and containing about 2.3 million ounces of gold at a cutoff grade of 0.015 ozAu/t (Table 6.6).

Table 6.6: Inferred Mineral Resource Estimate for Robertson Property (Beacon Hill, 2008).

Zone	Tons	OzAu/t	Ounces
Distal	10,335,041	0.0335	346,224
39A	25,010,247	0.0287	717,794
South Zone	5,904,713	0.0269	158,837
Outside	2,187,500	0.0208	45,500
Gold Pan Oxide	7,049,181	0.0262	184,689
Altenburg Hill Oxide	4,558,402	0.0208	94,815
Porphyry Oxide	19,121,927	0.0213	407,297
Gold Pan Sulphide	12,053,279	0.0208	250,708
Altenburg Hill Sulphide	584,016	0.0176	10,279
Porphyry Sulphide	4,480,533	0.0223	99,916
TOTAL	**91,284,840**	**0.0253**	**2,309,506**

- Gold ounces were calculated on the basis of US$600/oz Au and 70% Au recovery.
- The 0.015 ozAu/t cut-off grade, utilized to report the resource, was derived from a mining cost of US$1.02/ton, process cost of US$5.00/ton and waste cost of US$1.14/ton.
- The mineral resources in the table above were estimated using the CIM Standards on Mineral Resources and Reserves. This resource is compliant with NI 43-101 regulations.

In October 2009, Coral announced a revised mineral resource estimate incorporating the 2008 drilling results (22,835 ft) and a higher gold price. Using a 0.0106 ozAu/t cutoff grade based on a three year rolling average gold price of $850/oz, Beacon Hill calculated an inferred mineral resource containing nearly 179 million tons averaging 0.0189 ozAu/t and containing nearly 3.4 million oz of gold.

Table 6.7: Inferred Mineral Resource Estimate for Robertson Property (Beacon Hill, 2009)

Zone	Tons	OzAu/t	Oz Au
Distal	13,310,451	0.0287	382,010
39A	38,945,698	0.0228	887,962
South Zone	9,993,853	0.0209	208,872
Outside	5,422,131	0.0156	84,585
Gold Pan Oxide	12,566,599	0.02	251,332
Altenburg Hill Oxide	12,873,976	0.0152	195,684
Porphyry Oxide	39,049,182	0.0167	652,121
Gold Pan Sulphide	32,524,592	0.0154	500,879
Altenburg Hill Sulphide	1,701,844	0.014	23,826
Porphyry Sulphide	12,535,861	0.0158	198,067
TOTAL	**178,924,188**	**0.0189**	**3,381,667**

Resource estimate parameters:

- Gold ounces were calculated on the basis of US$850/oz Au and 70% Au recovery.
- The 0.0106 ozAu/t cut-off grade utilized to report the resource was derived from a mining cost of US$1.02/ton, process cost of US$5.00/ton and waste cost of US$1.14/ton.
- The mineral resources in the table above were estimated using the CIM Standards on Mineral Resources and Reserves.

- The database comprised a total of 1,160 drill holes, 533,453 feet (162,638 meters) of drilling and 101,757 gold assays.
- The inferred resource covers 6 distinct and separate areas; Distal, 39A, Gold Pan, Porphyry, Altenburg Hill, Southern Area and then all remaining blocks outside these areas that warrant inclusion as an inferred resource. In addition, Gold Pan, Porphyry and Altenburg Hill were separated into oxide and sulphide zones for analysis and modeling.
- An interpreted mineralized envelope was modeled into a solid in MineSight 3DTM, with six area mineralized zones and then separated into oxide and sulphide zones.
- Block dimensions of 25 feet (7.6 m) north, 25 feet (7.6 m) East and 20 feet (6 m) vertically.
- Grade interpolation - 20 foot (6 m) composites.
- Composites greater than 0.075 ozAu/t (2.33 g Au/tonne) limited in influence to 100 feet (30.5 m).
- Tonnage estimates are based on 200 bulk historic density measurements carried out by previous operators. These were assigned to each block by zone. The resources are categorized as inferred since the amount and distribution of bulk tonnage factor data is sparse.

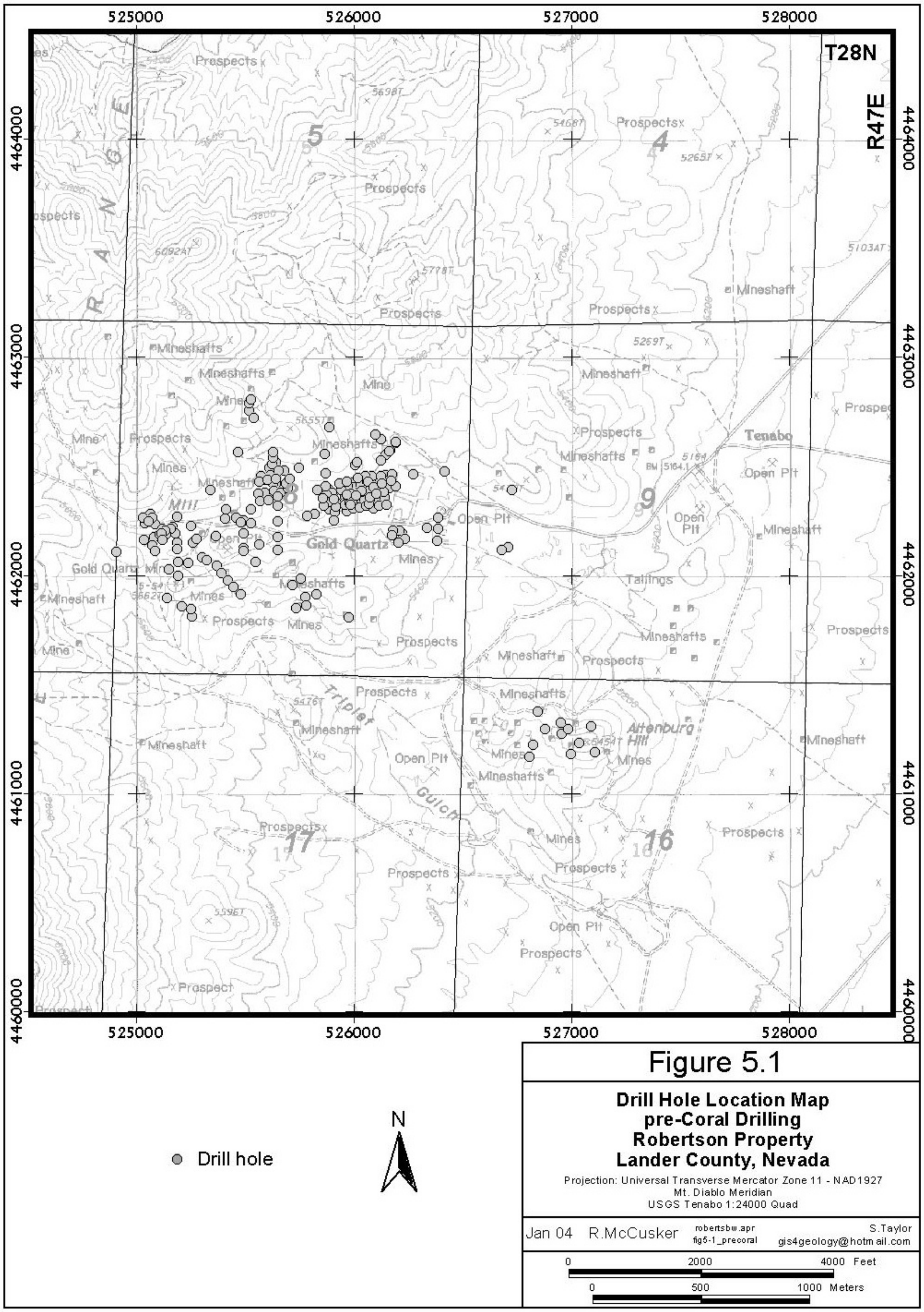


Figure 6-1: Drill Hole Location Map Pre-Coral Drilling

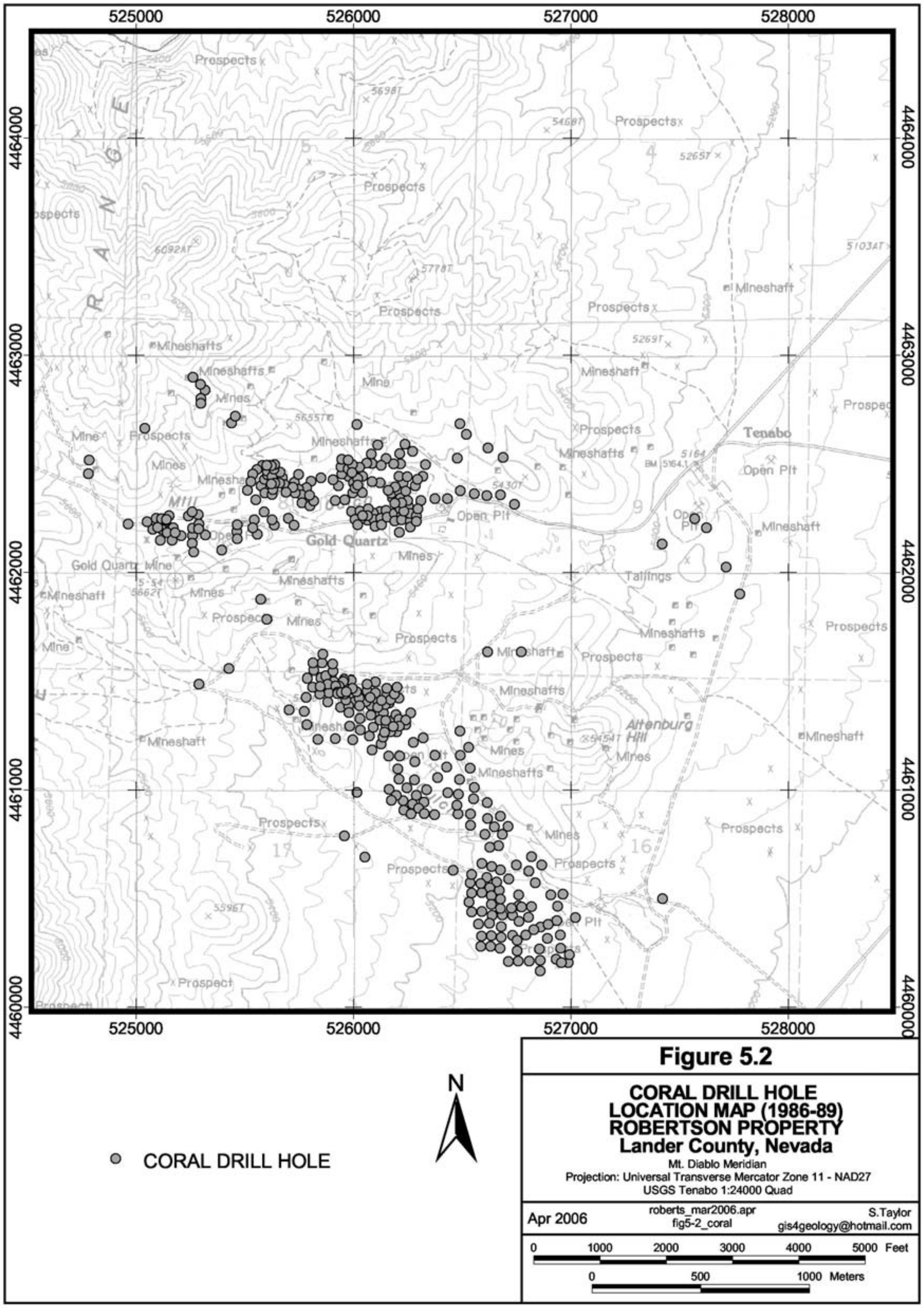


Figure 6-2: Coral Gold Corp. Drill Hole Location Map (1986-89)

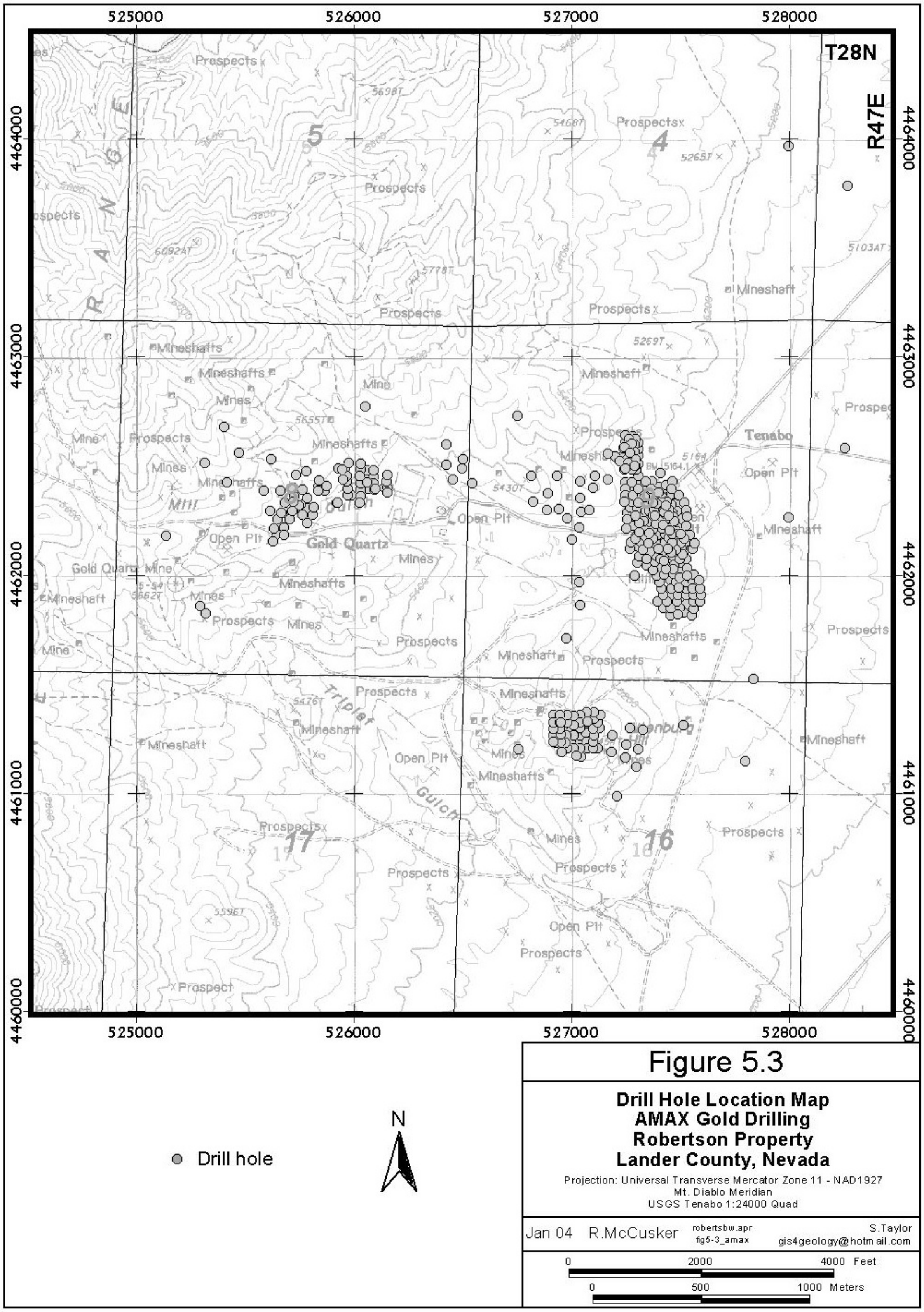


Figure 6-3: Drill Hole Location Map Amax Gold Drilling

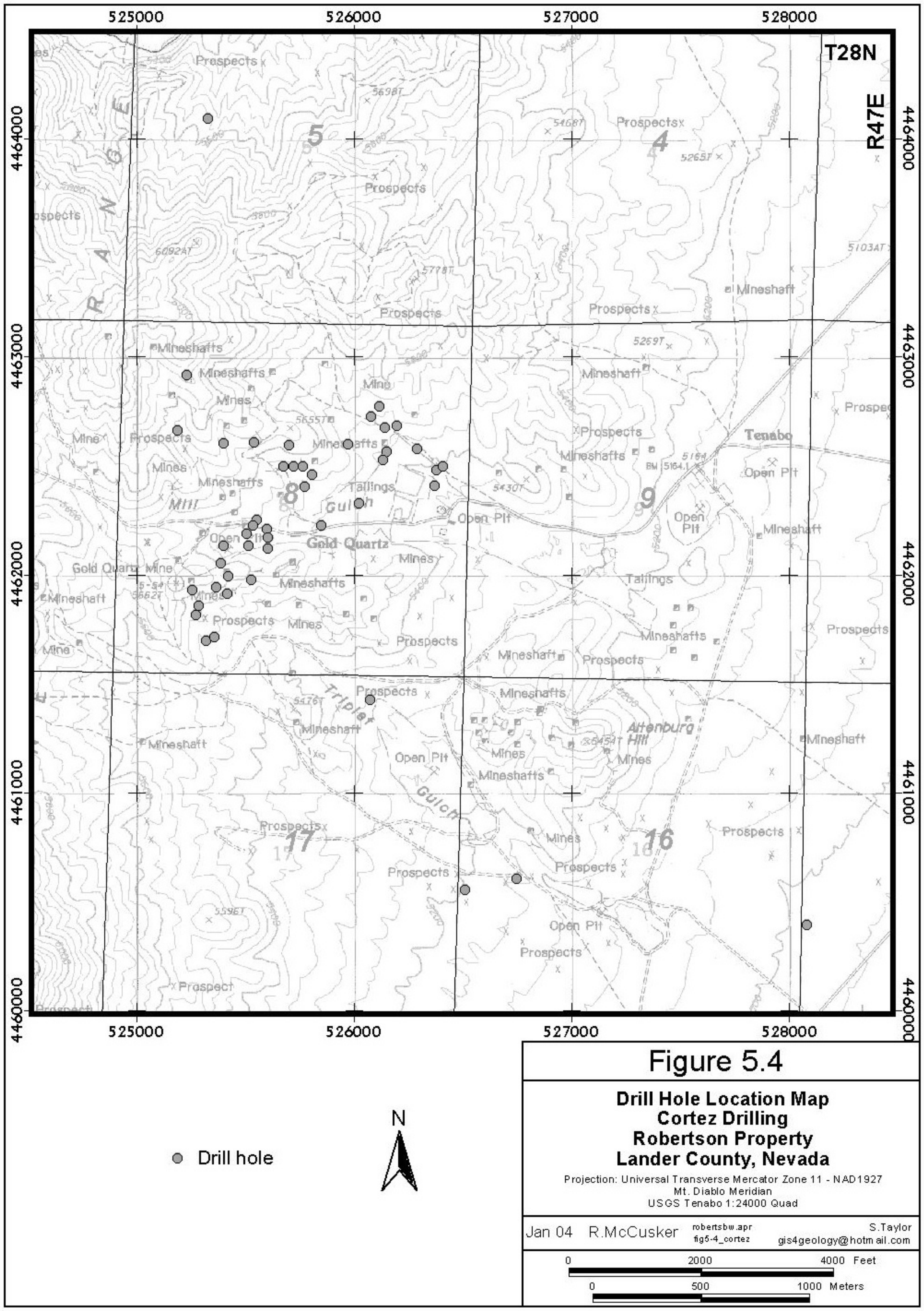


Figure 6-4: Drill Location Map Cortez Drilling

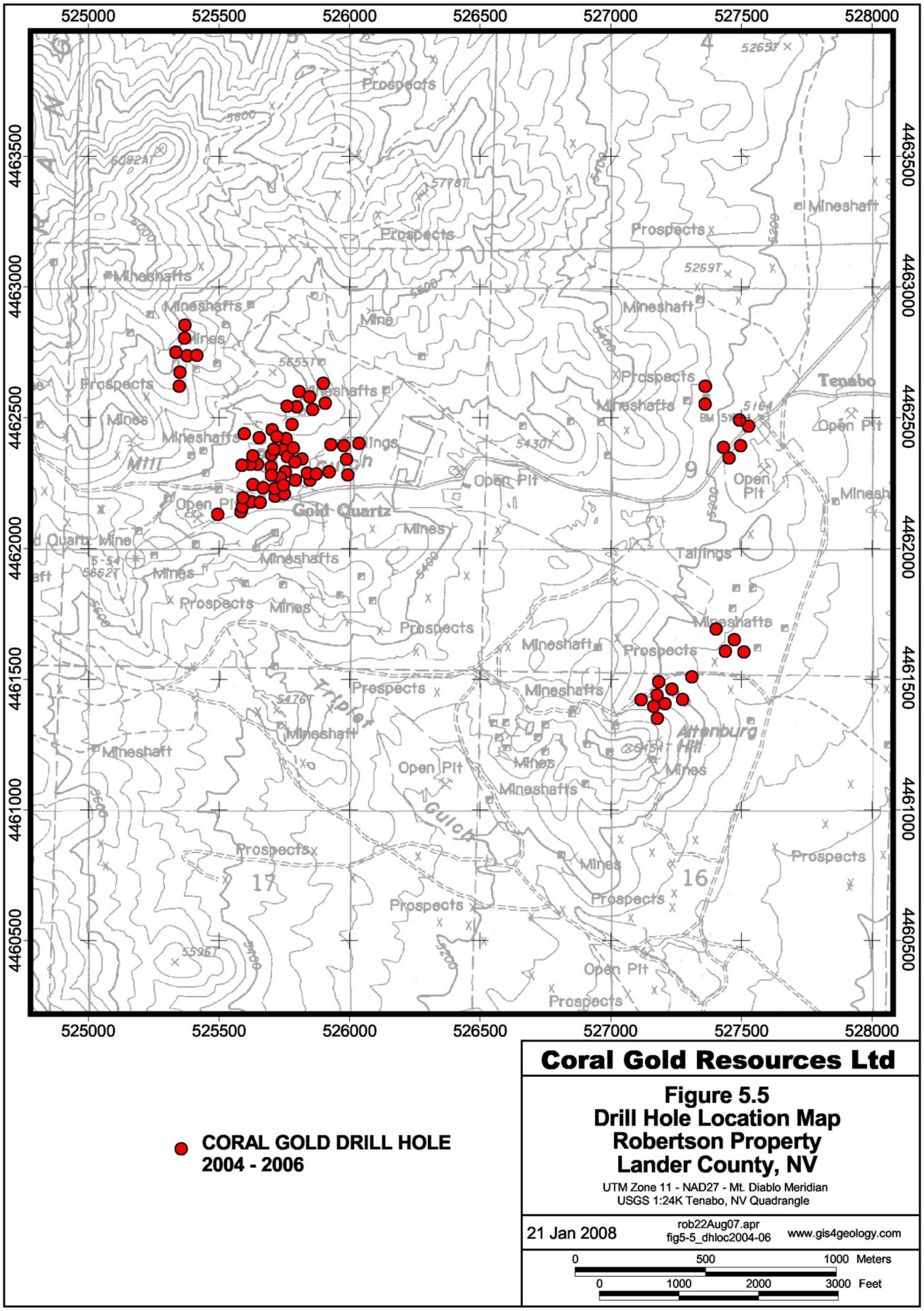


Figure 6-5: Drill Hole Location Map Robertson Property

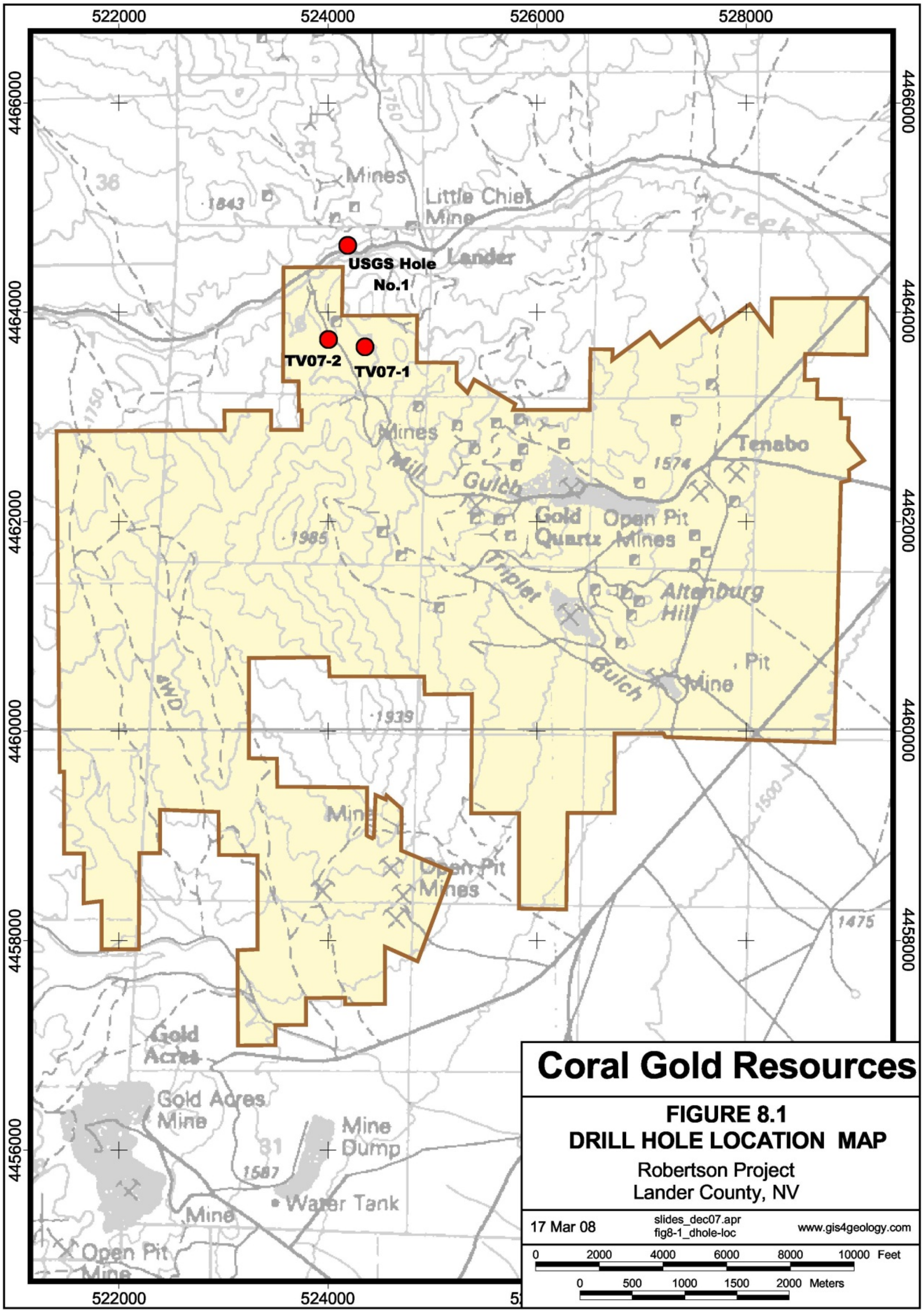


Figure 6-6: Map showing locations of the 2007 deep drill holes and USGS Hole No.1.

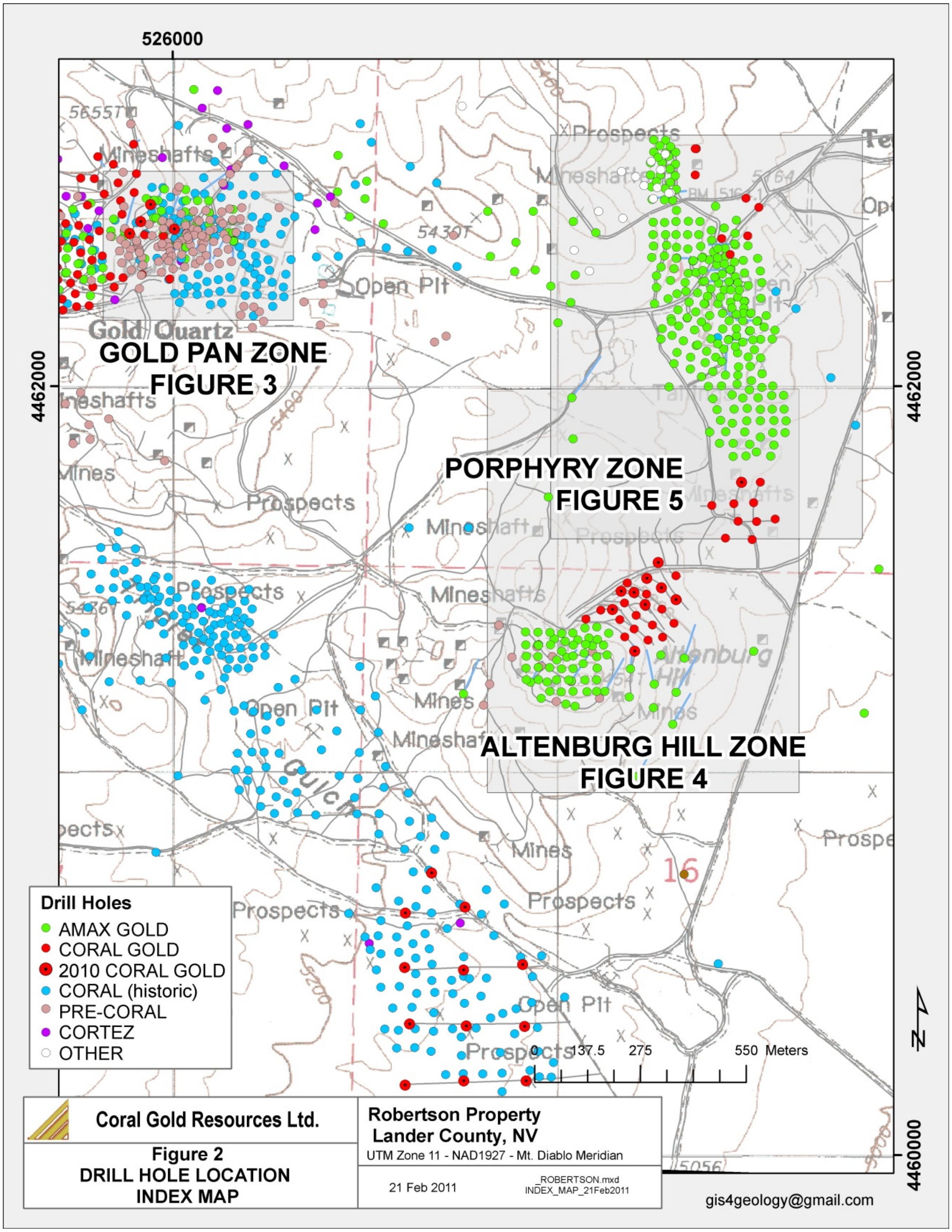


Figure 6-7: 2010 drill hole location map.

SECTION 7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

Published geological reports describing the regional geology of the Tenabo district include Gilluly and Gates (1965), Wrucke (1974), Wrucke and Armbrustmacher (1975), and Stewart (1980). The regional geology of the Tenabo area depicted in Figure 7-1 is modified from a map by Wrucke (1974).

The Robertson property lies along the far eastern flank of the Northern Shoshone Mountains in north-central Nevada within the Basin and Range physiographic province of western North America. The region lies at or very near the rifted margin of continental crust which was subjected to periodic contractual deformation starting in Middle Paleozoic and continuing until Late Cretaceous (Stewart, 1980, Oldow, 1984). The most important of these events affecting the region is a complex set of branching, low-angle faults that are part of the Roberts Mountains thrust fault (RMT). Forming the upper plate of this regionally significant structure are a series of thick, vertically stacked nappes of predominately dark-gray, fine-grained siliceous sedimentary and lesser submarine volcanic rocks of early to middle Paleozoic age. During latest Devonian or earliest Mississippian time, these eugeosynclinal rocks were transported eastward many tens of miles along segments of the RMT and structurally emplaced over a mostly carbonate sequence of similar age that comprise the lower plate. While siliceous rocks of the upper plate are widespread in the region, carbonate rocks of the lower plate are comparatively rare and are exposed in a few "structural windows" such as the Gold Acres and Cortez windows. Many of the regions gold deposits, including Gold Acres, Cortez, Cortez Hills and Pipeline are within or near to structural windows of lower plate carbonate rocks (Roberts, 1966).

Subduction-related calc-alkaline magmatism began as early as Middle Jurassic and continued through early Miocene time (Stewart, 1980). Beginning in the late Eocene, an important episode of magmatism gradually swept from northeast to southwest across northern Nevada which was accompanied by widespread extensional deformation (Christiansen and Yeats, 1992). Many of the regions largest gold deposits, including those situated along the Carlin and Battle Mountain-Eureka trends, are both spatially and temporally related to this period of calc-alkaline Eocene magmatism and extensional tectonics (Ressel and Henry, 2006). Starting about 17 Ma, regional extensional block faulting, accompanied by rift-related bimodal basalt-rhyolite magmatism, further modified the region resulting in the characteristic basin and range topography encountered in this part of Nevada (Dickinson, 2002).

7.2 LOCAL GEOLOGY

The Robertson property covers a series of low hills along the west side of Crescent Valley. The property, including the adjacent "Excluded" claims, lies approximately one mile north of the Pipeline open pit gold mine and reaches 4.5 miles north to Indian Creek. The geology of this area is depicted in Figure 7-2. Most of this area is underlain by a thick sequence of mostly fine-grained, siliceous sedimentary rocks that are part of the Roberts Mountains allochthon (RMA). These highly faulted and folded rocks are dominated by middle to late Devonian Slaven Chert composed mainly of argillite, chert, lesser shale, siltstone and mafic submarine volcanic rocks. Structurally overlying the Slaven Chert along the west, north and east sides of the property are a sequence of rusty brown weathering siltstone, sandstone and very minor limestone of the Silurian Elder Sandstone. Along the far west edge of property, many of these rocks are structurally overlain along the Lander thrust fault by massive quartzite of the Ordovician Valmy Fm.

Portions of the "Excluded" claims in the southwest part of the property cover the northern part of the Gold Acres structural window, which exposes a sequence of mostly carbonate rocks in the lower plate of the RMT. These strata consist of silty limestone, weakly to non-calcareous siltstone, calcareous mudstone and argillite of the Devonian-lower Mississippian Pilot Shale (Wrucke, 1974). However, this unit is remarkably similar to the Devonian-lower Mississippian Rodeo Creek Fm as described from the Carlin Trend (Lenardson and Rahn, 1996) where it overlies the Devonian Popovich Limestone. At Gold Acres, this unit conformably (?) overlies thin to thick bedded argillaceous to micritic limestone of the Middle Devonian Wenban Limestone which in turn conformably overlies a sequence of predominately finely laminated calcareous siltstone of the Silurian Roberts Mountains Fm.

Intruding a portion of the upper plate sequence, at the north end of the property, is an elliptical shaped granodiorite stock or lacolith (?), along with related dikes and sills that vary in composition from dacitic to rhyolitc. In its long dimension, the Tenabo granodiorite is exposed for 6,000 ft in an east-west direction and over 3,000 ft in a north-south direction. Its long axis is oriented at about 280°, whereas a set of prominent dike swarms strike approximately 300° to 340°. In addition, the district is transected by a series of E-W-striking sericite-altered feldspar-porphyry dikes that are exposed for at least 3.5 miles. Based on intrusive relationships the feldspar porphyry dike swarm appears to be intra-mineral, cutting early Au-Ag (Cu) mineralization but is cross cut by later Ag-Au ±As-base metal veining.

A series of recent K-Ar age dates indicate that emplacement of the stock took place about 39.35±0.07 Ma (Henry, 2011, written communication). A single Re-Os date on molybdenite from the Tenabo stock yielded an age of 39.0±1.4 Ma (Kelson, et al., 2005) and K/Ar age dates on adularia and sericite indicate that and the age of gold and base metal mineralization is 39.08±0.08 Ma (Henry, 2011, written communication). Based on two new K-Ar dates, emplacement of widespread post-mineral rhyolite porphyry dikes and sills is dated at 35.8±0.07 Ma (Henry, written communication). Gold mineralization at Robertson is closely associated with emplacement of the Tenabo stock.

7.3 ROBERTSON PROPERTY STRUCTURES

Mineralization at Robertson is strongly controlled by a system of low- and high-angle faults and related fracture zones. Locally, brecciation associated with the granodiorite and hornfels contact zones and to a lesser extent axial plane shears in isoclinal folds are also important hosts for mineralization. Although individual structures host ore-grade gold, higher grades commonly occur where one or more structures intersect.

7.3.1 Low-Angle Faults

Rocks of the district are cut by a myriad of low-angle thrust faults associated with the emplacement of the Roberts Mountains allochthon. Reactivation of these faults during emplacement of the Tenabo intrusive complex created an important fracture system that, in combination with younger high-angle faults and/or favorable lithology, strongly control the distribution gold at Tenabo. A major low-angle structural zone is thought to be the principal control of gold distribution in a broad area extending from hole AT-1 westward to 99445, a distance of over 3,000 ft, and containing the 39A and Distal zone mineral resources. Detailed logging has shown that portions of this low-angle structural zone are filled by a series of sill-like intrusions that form within or parallel to the broad envelope of mineralization and (or) alteration. In addition, at least two narrow and laterally persistent zones of open space quartz and sulfide filling and local sulfide replacement are developed within and along a series of low-angle faults with similar orientation as the principal fault system. The sulfide horizons are locally disrupted by post-mineral movement along these same low-angle faults. These relationships are summarized in Figure 7-3, a cross section oriented north-northeast and looking west. The location of C-C’ is shown in Figure 7-4.

7.3.2 NNW and East-West High-Angle Faults

In addition to the flat-lying thrust faults associated with reactivated segments of the RMT, rocks of the district are broken and offset by series of NNW-striking high-angle faults. Many of these faults extend (?) through the Eocene-age stock and are filled locally by younger dikes. Gold mineralization appears closely associated with NNW faults. A less obvious, but important east-west fault system occurs along the north side of the granodiorite stock which likely controlled emplacement of the granodiorite stock itself, as well as, the later feldspar porphyry dike swarm. At least two east-west and (or) north-northwest faults are postulated to have controlled mineralization within the 39A Zone. Where these faults intersect the principal low-angle fault zone, mineralization is typically thicker and higher grade. Movement along these faults is suspected to be, at least in part, sinistral. Within the Gold Pan zone, higher grade mineralization forms a series of en echelon ore shoots that trend generally east-west and are thought to be controlled by this fault zone.

7.3.3 Northeast and North-South High-Angle Faults/Dikes

A subtle NE-trending structural fabric is reflected mainly by a number of dike-filled faults and the apparent control exerted on the distribution of gold mineralization. In the Porphyry Zone, high-grade gold is commonly associated with a series of sub-parallel, NE striking faults locally filled by granodiorite and (or) igneous breccia dikes. The highest grades (0.94 ozAu/t) in the Porphyry Zone occur at or near the contacts of the dikes where they intersect prominent N-S faults. However, the overall trend of ore-grade (≥0.01 ozAu/t) mineralization in the Porphyry Zone is north-south.

7.3.4 Folds

Paleozoic-age upper plate rocks of the district are locally folded into small-scale isoclinal folds that typically strike NNW and plunge 5° to 15° to the NNW. This fold orientation is largely confined to the Devonian Slaven Chert. Strong shearing and brecciation are often well developed along many fold axial planes creating additional structural disruption in the rocks and sites for gold deposition.

In contrast, a series of broad, NE-striking open folds are developed in a nappe (?) of Silurian Elder Sandstone along the northeast part of the property. This variation in fold orientation and style suggests that these thrust sheets were emplaced separately and from different directions. A similar, broad anticlinal fold is developed in the Porphyry Zone that is more or less coincident with mineralization. This fold, which strikes N-S, is thought to have formed partly in response to emplacement of the Tenabo stock. Mineralization in this zone is strongly controlled by faulting in the footwall of a thick dike/sill. Any influence on the localization of the mineralization by this fold is unclear.

Recent geologic mapping and compilation has defined at least one prominent and potentially important fold along the west side of the Robertson Property (McCusker, 2007). The Shoshone antiform strikes about N-S and appears to plunge gently northward. The axis of this fold lies slightly west of the prominent ridge separating the Robertson claims from the “Excluded” claims. Because of the many small-scale folds, the Shoshone antiform is not well defined by bedding attitudes alone. However, the general pattern and “stratigraphic” position of certain units (greenstone and Elder Sandstone) provide convincing evidence for the existence of this structural feature. The fold is best defined in the northern half of the property even though it is disrupted by a series “major” NNW-striking high-angle faults. Many of these mapped faults coincide with major geophysical features and abrupt topographic breaks that interrupt the current ridge line.

This fold also affects both the plane of the RMT and lower plate strata where it is defined by bedding attitudes in outcrop, from measurements in drill cores and stratigraphic re-construction based on drill logs (McCusker, 2007). As defined in lower plate rocks, the N-S trending fold is broadly asymmetric with a shallow dipping west limb and steep or possibly overturned eastern limb. Because of disruption due to NE- and NNW-striking high-angle faults, the actual strike of the fold is unknown. However, its general trend appears to be N-S.

As a result of late Tertiary Basin and Range block faulting, rocks along the east side of the northern Shoshone Range, including the Tenabo area, have been tilted as much as 10° to 15° to the east (Gilluly and Gates, 1965). This post-mineral tilting resulted in the rotation of the large diorite dike in the Porphyry zone, leading to the original interpretation that this intrusion was a thick sill. The tabular mineralized zone associated with the footwall of the diorite dike was also rotated eastward, resulting in a shallow westward dip.

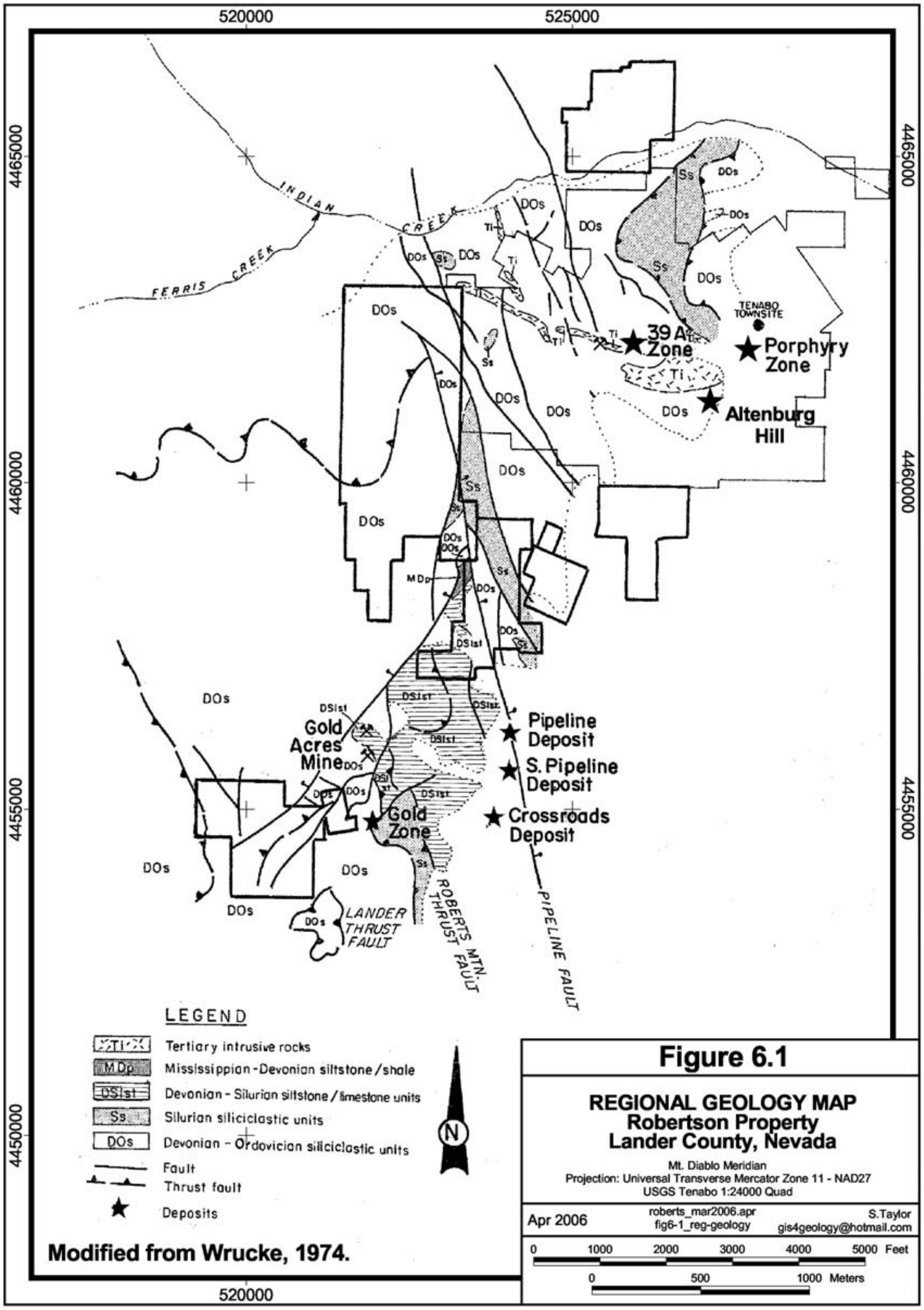


Figure 7-1: Regional Geology Map

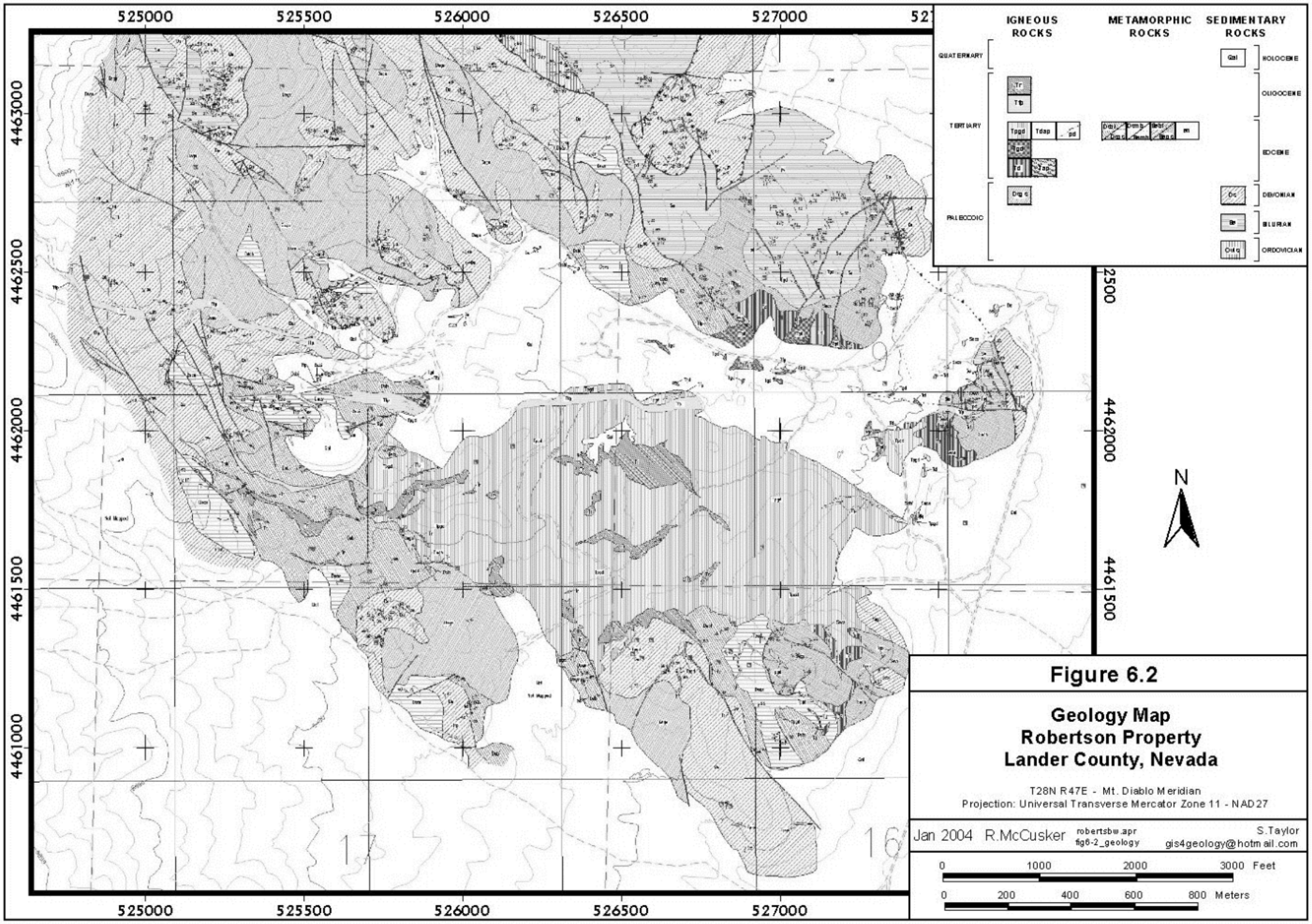


Figure 7-2: District Geology Map

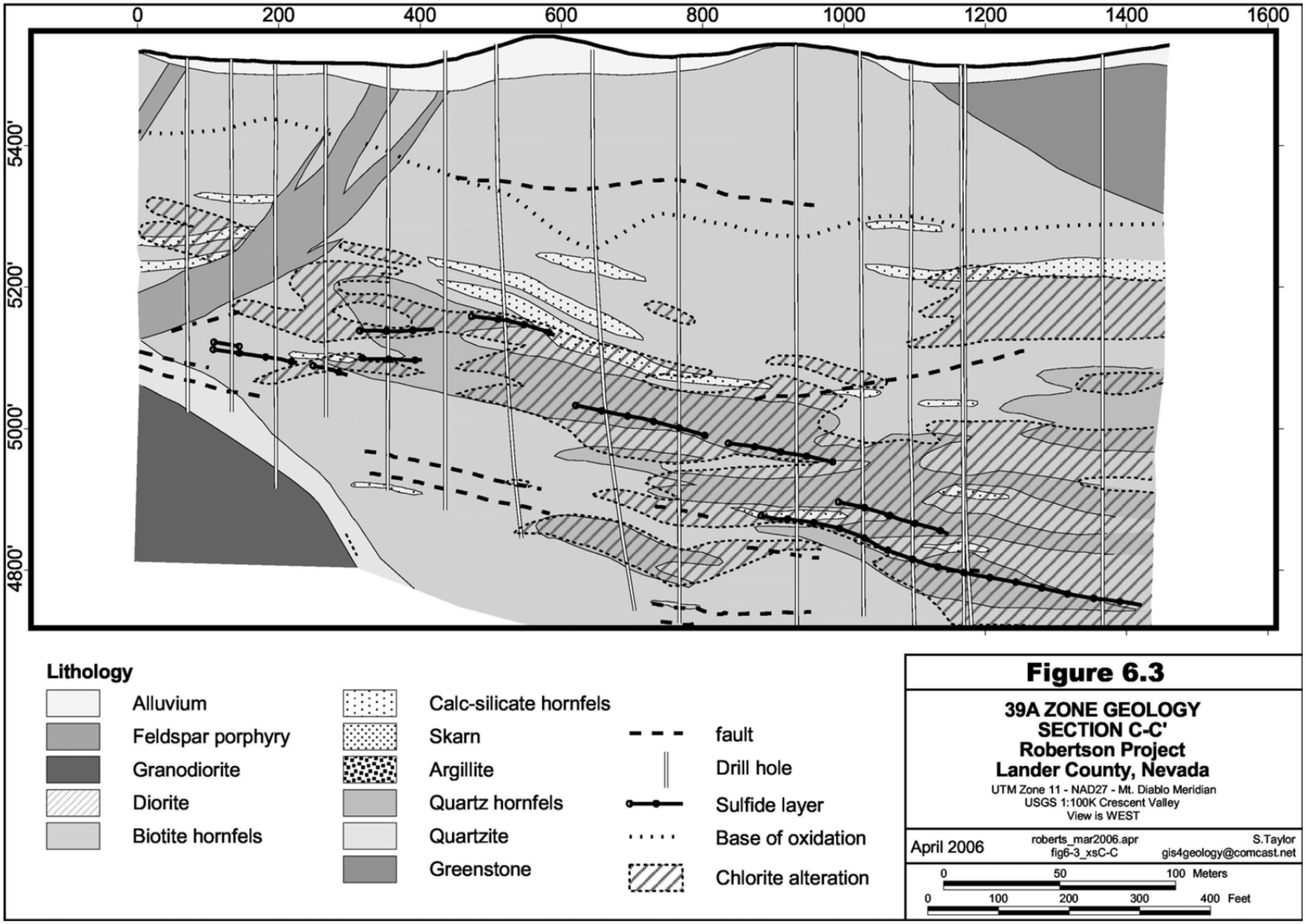


Figure 7-3: "Typical" cross section through 39A Zone looking west.

7.4 MINERALIZED ZONES

The currently identified mineral resources occur in five zones localized along the northern and eastern contact of the Tenabo granodiorite, forming a general east-west trend. These resources include Porphyry, Gold Pan, Altenburg Hill, 39A and Distal Zones (Figure 7-4). The Porphyry, Gold Pan and Altenburg Hill Zones occur in highly fractured hornfels and skarn units at the contact of the granodiorite stock, whereas the 39A Zone is localized at the intersection of a series of high-angle faults with a major low-angle structural zone in retrograde-altered hornfels. In addition, an emerging mineralized zone referred to as the Distal Zone occurs 1,500 ft northwest of the 39A Zone and at least 2,200 ft from the nearest exposure of Tenabo granodiorite. Gold in the Distal Zone occurs in weakly retrograde-altered quartz and calc-silicate hornfels and skarnoid.

7.4.1 Porphyry Zone

In 1994, Amax defined a proven + probable + inferred "mineable reserve" in the Porphyry Zone, using a 0.01 ozAu/t cutoff grade that contained 14 million short tons averaging 0.019 ozAu/t. **These historic reserves are presented for informational purposes only and do not represent current reserves at Robertson.**

The Porphyry Zone was re-evaluated in 2009 using a gold cutoff grade of 0.0106 ozAu/t and was estimated to contain a total inferred mineral resource of 51.6 Mt (oxide + unoxidized) containing about 850,188 ounces of gold. Approximately 156 RC and 51 diamond core holes define a tabular mineralized zone that dips 30º to 45º west, ranges from 50 ft to over 250 ft thick, reaches over 500 ft down dip and has a strike length of nearly 3,000 ft. Depth to the top mineralization varies from 5 ft to 30 ft in the south and 125 ft in the north. Mineralization is mainly fracture controlled with lesser amounts occurring as Approximately one-third of the Porphyry Zone gold resource is mainly fracture controlled hosted by a fine-grained margin phase of the Tenabo granodiorite (also called diorite) and two-thirds are hosted by biotite, quartz hornfels and retrograde skarn in Elder Sandstone. Locally, higher grade gold values occur as narrow replacement horizons in retrograde skarn. The Porphyry Zone is intruded at least two sets of dikes: a NE-striking set that is spatially associated with higher gold grades and an E-W-striking set of sericite-altered, post-mineral dikes that dip moderately to the south. The E-W dikes cut the zone and disrupt the continuity of mineralization. Overburden depth varies from none to 30-ft-thick. The intensity of surficial oxidation is controlled by the density of both high- and low-angle fracture zones and lithology. As a result the redox front forms a complex irregular interface that varies from 15 ft to as deep as 450 ft deep. It is estimated that 76 percent of the defined resources has undergone at least 50 percent oxidation. These relationships are summarized in section, 12,600 N (Figure 7-5) and section 11,500 E (Figure 7-6), which show the general geology and potential ore-grade mineralization at a cutoff grade of 0.01 ozAu/t. The locations of section 12,600 N and 11,500 E are shown in Figure 7-4.

7.4.2 39A Zone

The 39A zone is currently defined by 104 RC holes and 4 diamond core holes, totaling about 54,415 ft. In 2009 the 39A Zone was estimated to contain 38.9 Mt (sulfide) averaging 0.0228 ozAu/t and containing about 887,962 ounces of gold, using a 0.0106 ozAu/t cutoff grade. Using a 0.01 ozAu/t cutoff grade and a minimum of 20 ft thickness, the 39A mineralization forms a stratiform body roughly 1,500-ft-long in the principal north-northeast direction and 900-ft-long in the secondary west-northwest direction. Along these principal directions, mineralization averages about 129-ft-thick and 400-ft-wide. In the northeasterly direction, the zone plunges about 16° to the north. In the west-northwest direction, the zone plunges from 21-34° to the southeast. Depth to the top of mineralization varies from 300 ft at the south end of the zone to 605 ft at the extreme north end. Section D-D' (Figure 7-7) through the 39A Zone shows the general geology and distribution of potential ore-grade mineralization at a cutoff grade 0.015 ozAu/t. The location of section D-D' is shown in Figure 7-4.

The 39A Zone is characterized by a broadly and strongly developed zone of retrograde alteration consisting of fine-grained dark-green actinolite ± dark-green to black chlorite and accompanied by 5-25 percent sulfides as disseminations and local replacement. Semi-massive sulfide replacement form narrow (5-ft to 25-ft-thick), persistent layers that extend along strike up to 250 ft. The highest gold values are generally encountered at or near the base of these sulfide layers. Major ore controls are exerted by the intersection of NNE and WNW striking high-angle structural zones with a low-angle fracture system which are represented by the overall shape of the mineralized zone and distribution of higher gold grades. 39A mineralization is cut by a set of E-W-striking, sericite-altered post-mineral feldspar porphyry dikes that dip moderately to the south. These dikes disrupt continuity of mineralization in the southern part of the 39A Zone.

7.4.3 Gold Pan Zone

Gold mineralization in the Gold Pan Zone is confined to a series of high- and low-angle faults forming a number of narrow tabular to lenticular zones oriented generally to the northwest. Based on a gold cutoff grade of 0.01 ozAu/t, the zone is roughly 1,500-ft-long in an east-west direction, 500-ft-wide and at least 200-ft-thick. The depth to mineralization within the zone ranges from 10 ft to 200 ft. The Gold Pan resource is defined by approximately 135 RC holes spaced 100 ft or less apart and 7 diamond core holes. At least 90 percent of the holes were drilled vertical with an average depth of less than 300 ft.

The west half of the Gold Pan Zone resource is hosted mainly by a sequence biotite hornfels, quartz hornfels and locally abundant retrograde skarn replacing greenstone, whereas the eastern half is hosted largely by quartz, lesser biotite hornfels and abundant granodiorite sills extending from the north contact of the Tenabo granodiorite. Higher grades (≥ 0.05 ozAu/t) form a series en echelon shoots and veins aligned in an east-west direction that reach 600-ft-long, 100-ft-wide and 50-ft-thick (Candee, 1996). The depth of oxidation varies from up to 200-ft-deep in the east half to 10-75 ft in the west. The redox front in the Gold Pan Zone is complex particularly in areas of strong retrograde skarn development.

In 2009, the Gold Pan inferred mineral resources was estimated to contain a total of 45.0 Mt (oxide + sulfide) averaging 0.0167 ozAu/t and containing about 752,211 ounces of gold using a 0.0106 ozAu/t cutoff grade. Included in this resource are 12.6 Mt of material classified as oxide, with an average grade of 0.02 ozAu/t.

7.4.4 Altenburg Hill Zone

The Altenburg Hill resource occurs along the southeast contact of the Tenabo granodiorite in highly fractured and folded hornfels and intrusive rocks (Figure 7-8). Gold appears confined to fracture zones and vein zones cutting biotitized and calc-silicate altered siltstone and sandstone (Candee, 1996). Mineralization is also hosted by dikes and sills of granodiorite porphyry. Less commonly, gold distribution is controlled by fractures and breccia bodies developed along minor fold axes. The highest grade gold mineralization is controlled by the intersection of series of NW- and NE-striking dike-filled high-angle fault and fracture zones. The Altenburg Hill Zone is cut by at least one sericite-altered, post-mineral feldspar porphyry dike that locally disrupts continuity of mineralization.

As defined by a gold cutoff grade of 0.01 ozAu/t, mineralization forms a coherent tabular zone that extends from just south of the top of the Altenburg Hill (5,440 ft) northeastward approximately 1,500 ft along the north flank of the hill to the northern extent of current drilling. The mineralized zone averages about 500 ft wide. Depth to the top of mineralization varies from outcropping ore-grade material to less than 100 ft and generally occurs within 30 ft of the surface. Mineralization extends as deep as 495 ft and individual zones reach up to 145-ft-thick. Surficial oxidation is strongly developed in the Altenburg Hill Zone reaching as deep as 450 ft and affecting approximately 88 percent of the identified resource. Locally the redox front forms a complex interface due to persistent but thin layers of unoxidized calc-silicate hornfels. Figure 7-8 is a typical cross section through the Altenburg Hill resource showing the distribution of gold grading > 0.01 ozAu/t. The location of section C-C' is shown on Figure 7-4.

The Altenburg Hill inferred mineral resource is defined by approximately 57 RC holes and 10 diamond drill holes. Twenty-five RC and 10 core holes were drilled by Coral between 2006 and 2010. The remaining 32 RC holes that define the resource were drilled by Amax in 1991 and 1996. Total footage is estimated at 22,000 ft.

In 2009, Beacon Hill estimated the total Altenburg Hill inferred mineral resource to be 14.6 Mt (oxide + sulfide) averaging 0.0151 ozAu/t and containing 219,510 ounces of gold. Of this total resource, 12.9 Mt averaging 0.0152 ozAu/t are considered oxide.

7.4.5 Distal Zone

Mineralization in the Distal Zone is hosted by a thick sequence (>100-ft-thick) of locally retrograde-altered biotite, quartz, calc-silicate hornfels and a thin yellow-brown skarnoid (?) layer. The highest grade gold is within or immediately adjacent to the possible skarnoid layer. These rocks are cut by at least one prominent high-angle fault that strikes NNW and dips steeply southwest. This fault may exert significant control on the distribution of ore-grade mineralization. The Distal Zone mineralization is structurally overlain by a succession of hornfelsed and retrograde-altered greenstone followed by a locally hornfelsed sequence of carbonaceous argillite, shale, chert, minor siltstone and very minor silty limestone. The favorable mineralized horizon in The Distal Zone appears to be in the same "stratigraphic" position as the mineralized horizon in the 39A Zones, but in a more distal position with respect to the Tenabo stock contact. This is reflected in the higher average arsenic values (2,427 ppm) and lower average copper values (337 ppm) in the Distal Zone versus lower average arsenic values (577 ppm) and higher copper values (757 ppm) in the 39A Zone.

As shown in Figures 7-9 and 7-10 the "distal target" is currently defined by 14 vertical and one inclined RC holes collared from 160 ft to 625 ft apart and totaling 20,140 ft. The location of sections A-A' and B-B' are shown in Figure 7-4. At present, these holes outline an apparently flat-lying mineralized zone at least 1,200-ft-long in a northerly direction, 500-ft-wide east-west and ranging from 55-ft- to 170-ft-thick with an average thickness of 109 ft at a 0.01 oz Au/t cutoff grade. Except where disrupted by faulting, the continuity of mineralization between holes appears good especially for the N-S trending higher grades zones. Near its margin the zone begins to rapidly break up into a series of narrow mineralized intervals ranging from 15 ft to 55 ft thick. Depth to the top of mineralization ranges from 800 ft to 995 ft. A summary of assay results at a 0.05 oz Au/t cutoff grade is compiled in Table 7.3.

In 2009, the Distal Zone was estimated to contain an inferred mineral resource of 13.3 Mt averaging 0.0287 and containing about 383,010 ounces of gold using a 0.0106 ozAu/t cutoff grade.

7.5 HYDROTHERMAL ALTERATION

Hydrothermal alteration forms a broadly concentric halo surrounding the Tenabo stock and affecting certain rock units as far as 2,000 ft from the stock contact. Alteration patterns in the district are complex due to intense fracturing of the wall rocks, thin bedded nature and rapid compositional change that characterize the sedimentary sequence. Gold-related alteration consist of an early-stage intrusion-related potassic, a later “retrograde” skarn(?) stage and still younger quartz (silica)-sericite-(pyrite) and clay-chlorite-sulfide stages which appear to cut all earlier phases.

7.5.1 Potassic Alteration

Early potassic alteration is widespread and strongly developed in early phases of the intrusion, and is often spatially associated with gold mineralization, particularly in the Porphyry and Altenburg Hill Zones. In intrusive rocks, this alteration assemblage is characterized by widespread fine-grained secondary biotite replacing coarse-grained magmatic biotite, hornblende and pyroxene, and as veinlets accompanied by adularia (Honea, 1994). Fine secondary biotite also forms narrow envelopes surrounding quartz veinlets that cross-cutting early calc-silicate and biotite hornfels, and certain phases of the stock. In several localities, coarse-grained (≥ 5mm) adularia occurs as fracture fillings, accompanies quartz ± calcite veins and as cement in several pebble dikes. Potassic alteration is generally associated with Au-Ag (Cu) mineralization having an overall Au to Ag ratio of about 1.5:1.

7.5.2 Retrograde Skarn(?)

Gold mineralization at Robertson is closely associated with moderately to strongly retrograde-altered quartz ± calc-silicate and biotite hornfels. While not always obvious in deeply oxidized portions of mineralized zones both chlorite-actinolite and fine-grained quartz alteration form a strong fracture-controlled “stockwork” and local pervasive replacement of the biotite and calc-silicate hornfels. Locally, the chlorite-actinolite alteration is accompanied by semi-massive to massive sulfides (up to 50 percent) composed mainly of pyrrhotite, chalcopyrite, marcasite-pyrite, and arsenopyrite, which form 5-ft to 25-ft–thick semi-continuous replacement horizons and (or) open space fracture fillings. Gold grades exceeding 0.25 ozAu/t are sometimes, though not always, spatially associated with the strongest chlorite-actinolite-sulfide alteration. While the intensity of chlorite-actinolite alteration diminishes rapidly below the semi-massive sulfide layer, the highest gold grades are often encountered immediately beneath the sulfide replacement horizons and ore-grade gold values often persist well below the bottom of the chlorite-actinolite-sulfide alteration zone. This alteration is characterized by Au to Ag ratios of 1:<1.

7.5.3 Late Stage Quartz (Silica)-Sericite-(Pyrite)

The youngest (?) gold-related alteration consists of narrow envelopes of quartz (silica)-sericite-(pyrite) surrounding quartz-sulfide ±calcite veins and zones of fine-grained “jasperoidal” silicification carrying Au-Ag, arsenopyrite, tetrahedrite - tennantite, stibnite, galena, sphalerite and chalcopyrite. The age of the Q-S-(Py) alteration and related mineralization is indicated from its relationship with a series of intensely Q-S-(py-mc)-altered, post-mineral dikes that cut proximal Au-

Ag (Cu) skarn-type mineralization in the 39A and Porphyry Zones. Locally, Au-Ag, As-Sb ± Hg, Cu + Pb + Zn veins cut and are hosted by the same post-skarn dikes.

7.5.4 Chlorite-Clay

A late-stage, pale green chlorite ± clay (montmorillonite ?), accompanied by coarse-grained euhedral pyrite-marcasite and arsenopyrite with minor galena, sphalerite and chalcopyrite, forms narrow replacement envelopes along fractures cutting hornfels and, less commonly, intrusive rocks. This alteration assemblage occasionally carries moderate to high-grade gold. The position of the chlorite-clay assemblage within the alteration sequence is currently equivocal.

7.6 GOLD DISTRIBUTION

A number of unpublished petrographic (both thin and polished sections) and SEM studies to determine the distribution, locations and close mineralogical associations of gold in the various resource areas were conducted by Russ Honea from 1993 through 1996. Result of those studies indicates that gold occurs mainly as native particles of generally high fineness that range in size from 2 to 200 microns and average about 40 microns. Where oxidized gold and rare native silver occur mainly as liberated grains that are closely associated with goethite. In the deeper 39A zone (unoxidized), native gold (88-92 % Au) is often associated with arsenopyrite, pyrrhotite and chalcopyrite. When present, very minor bismuth tellurides, including tetradymite, hedleyite and tellurobismuthite, are closely associated with gold and electrum (minor), which occur as irregular blebs on telluride grain margins.

Locally, gold is encapsulated in fine silicate gangue and occurs as small grains hosted by pyrrhotite, arsenopyrite and loellingite. A summary of major and trace minerals identified during the various petrographic and SEM examinations are presented in Table 7.1. Samples used for these studies include metallurgical pan concentrates, selected intervals of RC cuttings and diamond drill core.

Table 7.1: Summary of Minerals Identified at the Robertson Property

native gold	native silver	electrum	pyrite	pyrrhotite
marcasite	arsenopyrite	stibnite	chalcopyrite	sphalerite
galena	bournonite	acanthite	loellingite	gersdorffite
tetradymite	petzite	hessite	hedleyite	tellurobismuthite
altaite	tetrahedrite	bornite	chalcocite	covellite
digenite	native copper	cuprite	chysocolla	azurite
goethite	magnetite	hematite	illmenite	scorodite

7.7 COPPER DISTRIBUTION

Copper mineralization is widespread and locally reaches economically significant concentrations on the Robertson Property. Copper is closely associated with ore-grade gold and is best developed in the Porphyry Zone, where Cu occurs mainly as secondary sulfides replacing chalcopyrite and pyrite, and as lesser oxides. Within selected five-foot assay intervals containing ore-grade gold (≥0.01 ozAu/t), copper values range from < 0.001 percent to over 9.5 percent, but averages less than 0.02 percent Cu. A tabulation of significant copper intercepts in the Porphyry Zone are presented in Table 7.2. The weighted average grade from this tabulation is 0.29 % Cu, over an average thickness of about 96 feet.

In the 39A Zone, copper occurs primarily as discrete grains of chalcopyrite closely associated with pyrrhotite, pyrite-marcasite, and less commonly with arsenopyrite. Within selected 20 ft composite assay intervals containing ore-grade gold (≥0.015 ozAu/t), copper values range from 0.009 percent to 8.28 percent, but averages only about 0.03% Cu.

Table 7.2: Summary of Significant Copper Intercepts in the Porphyry Zone*

Hole No.	Type	From (ft)	To (ft)	Thickness (ft)	Cu(%)
AT-123	RC	25	110	85	0.21
AT-150	RC	145	280	135	0.29
AT-151	RC	20	85	65	0.21
AT-167	RC	330	375	45	0.2
		395	465	70	0.28
AT-168	RC	320	465	145	0.22
AT-170	RC	215	400	185	0.15
		430	495	65	0.25
AT-171	RC	75	180	105	0.14
		205	290	85	0.2
		310	485	175	0.22
AT-174	RC	380	430	50	0.3
AT-187	RC	40	185	145	0.48
		215	250	35	0.25
AT-188	RC	175	265	90	0.14
		365	405	70	0.25
AT-190	RC	360	440	80	0.26
AT-191	RC	100	105	105	0.19
AT-198	RC	165	225	60	0.34
AT-199	RC	70	225	155	0.22
AT-209	RC	40	105	65	0.24
AT-211	RC	30	100	70	0.21
CAT-18	CORE	130	295	165	0.28
CAT-21	CORE	45	200	155	0.31
		230	275	45	0.25
CAT-22	CORE	95	155	60	0.21
		175	285	110	0.38
CAT-23	CORE	110	210	100	0.3
CAT-24	CORE	40	75	35	0.59
CAT-27	CORE	255	320	65	0.36
CAT-30	CORE	70	145	75	0.19
CAT-33	CORE	375	460	85	0.28
CAT-36	CORE	175	235	60	0.28
		255	325	70	0.7
		355	390	45	0.17
		410	480	70	0.22
CAT-37	CORE	115	250	135	0.18
		295	500(EOH)	205	0.64
CAT-40	CORE	110	175	65	0.12
		230	435	205	0.31
CAT-47	CORE	260	350	90	0.29
CAT-48	CORE	165	280	115	0.14
CAT-52	CORE	440	575	135	0.19
CAT-54	CORE	145	220	75	0.22
CAT-55	CORE	385	450	65	0.19
CAT-57	CORE	15	115	100	0.77

*Interval average grade calculated using 0.10% Cu cutoff.

Table 7.3: Summary of Distal Zone assay results using a 0.05 ozAu/t cutoff.

Hole No.	UTM East	UTM North	Collar Elev. (ft)	Hole Length (ft)	From (ft)	To (ft)	Thickness (ft)	Au Grade oz/t
CR05-1	525449	4462671	5741	1200	945	995	50	0.163
including					950	975	25	0.262
CR05-2	525375	4462734	5733	1200	855	865	10	0.105
including					905	940	35	0.128
					915	920	5	0.439
CR06-16	525347	4462618	5744	1200	905	985	80	0.085
including					960	965	5	0.241
					990	1045	55	0.113
including					1030	1035	5	0.295
CR06-17	525414	4462735	5739	1200	950	960	10	0.118
					1010	1020	10	0.057
CR06-18	525335	4462748	5780	1200	890	955	65	0.120
including					890	895	5	0.333
including					925	930	5	0.420
					1010	1045	35	0.120
CR06-19	525366	4462803	5800	1200		no significant values		
CR06-20	525369	4462851	5794	1500	1005	1015	10	0.123
					1070	1090	20	0.164
CR08-1	525324	4462896	5814	1120	930	940	10	0.0565
CR08-2	not drilled							
CR08-3	525299	4462769	5849	1200	1005	1010	5	0.127
CR08-4	525307	4462716	5821	1200	955	970	15	0.128
CR08-5	525314	4462659	5795	1200	860	870	10	0.054
					885	890	5	0.270
CR08-6	525378	4462595	5728	1200	910	920	10	0.354
					965	1000	35	0.084
					1015	1030	15	0.086
AT-3	525401	4462678	5678	2920	845	890	45	0.083
including					885	890	5	0.235
99406	525401	4462602	5680	1100	800	810	10	0.624 [1]
					900	910	10	0.061
					940	950	10	0.116
99445 [2]	525230	4462912	5890	1500	1080	1090	10	0.222
Total				**20140**				

(1) Average of two assays.
(2) Inclined hole drilled at -70° on an azimuth of 144°.

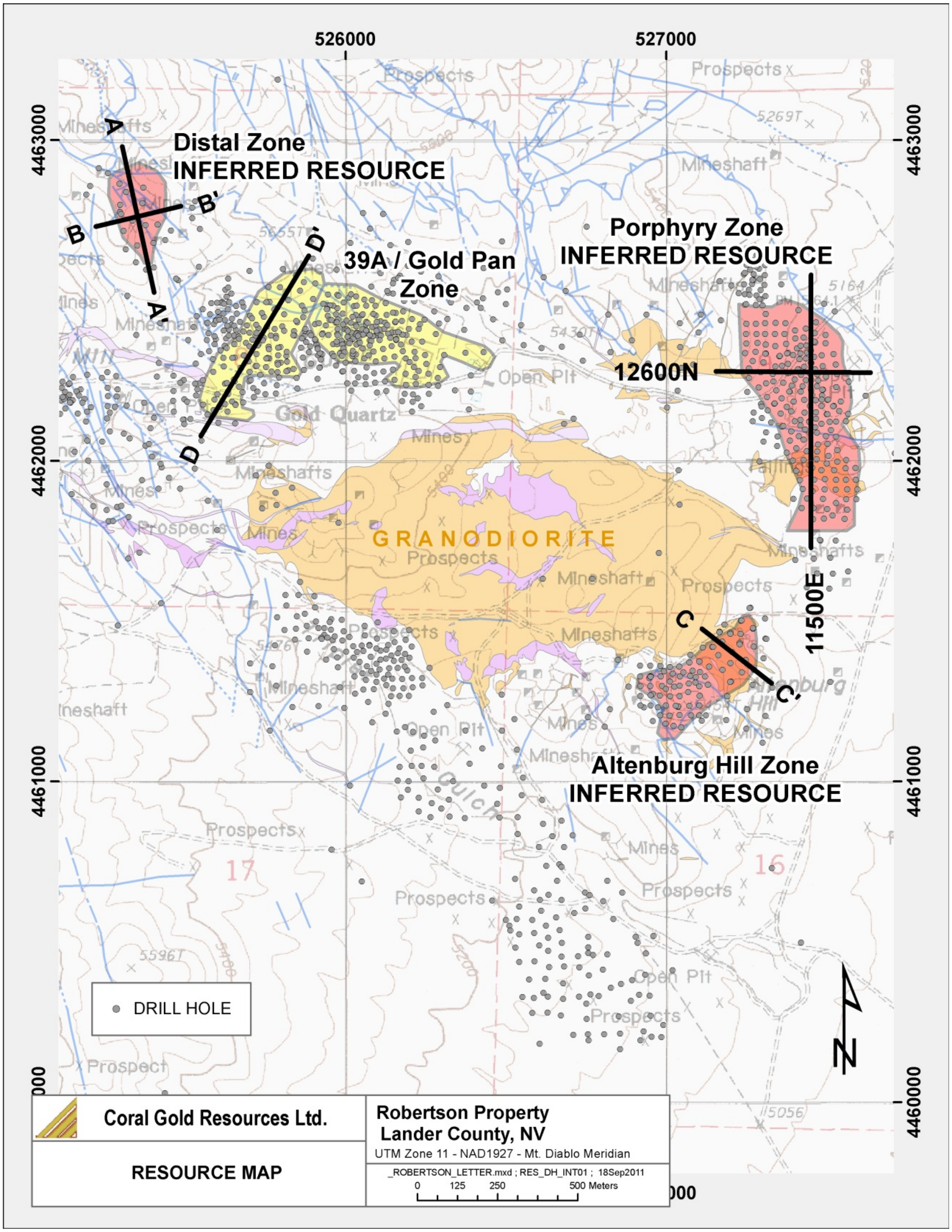


Figure 7-4: Resource Map

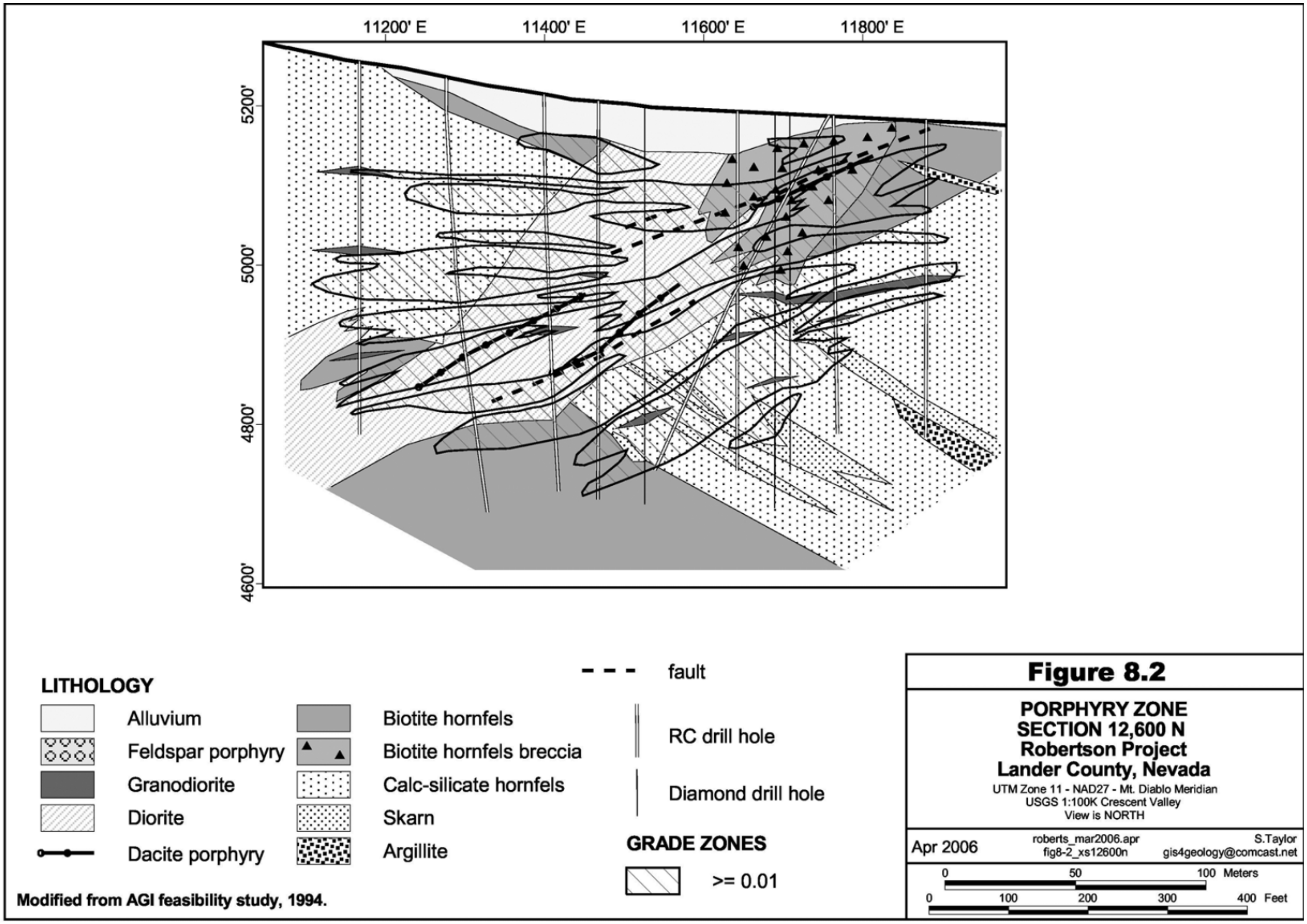


Figure 7-5: Porphyry Zone Section 12,600N Looking North

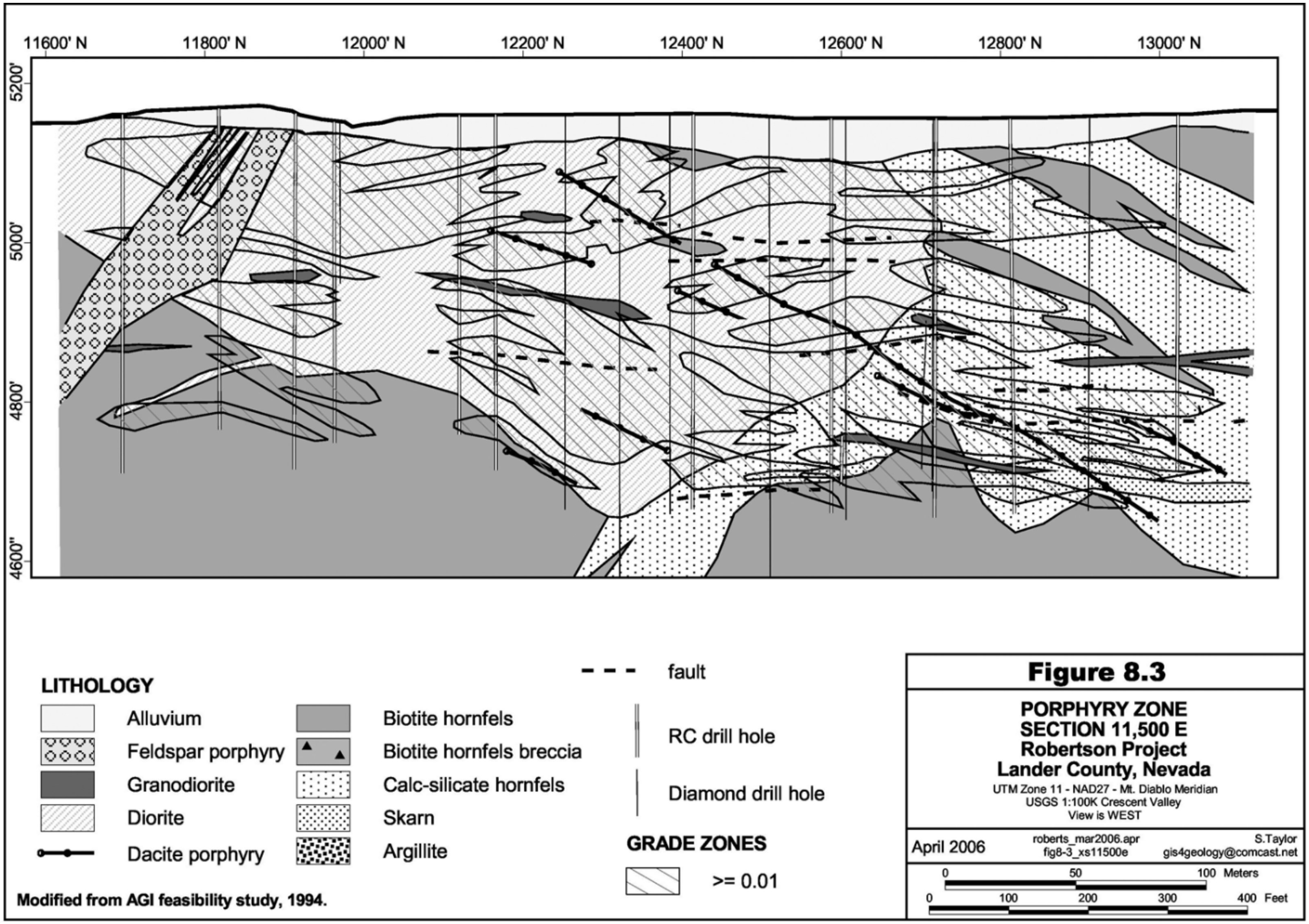


Figure 7-6: Porphyry Zone Section 11,500E Looking West

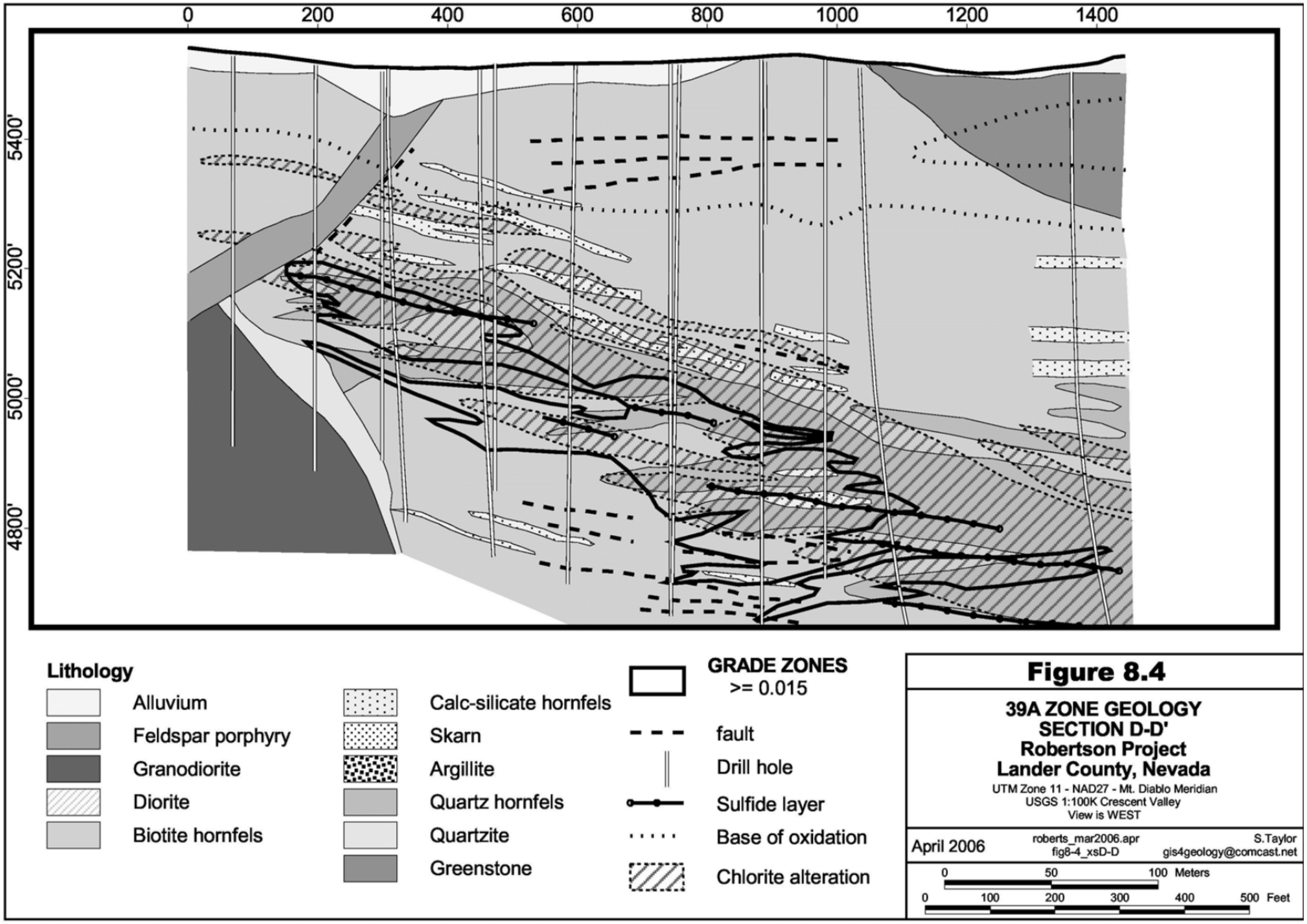


Figure 7-7: 39A Zone Geology section D-D[1] Looking West.

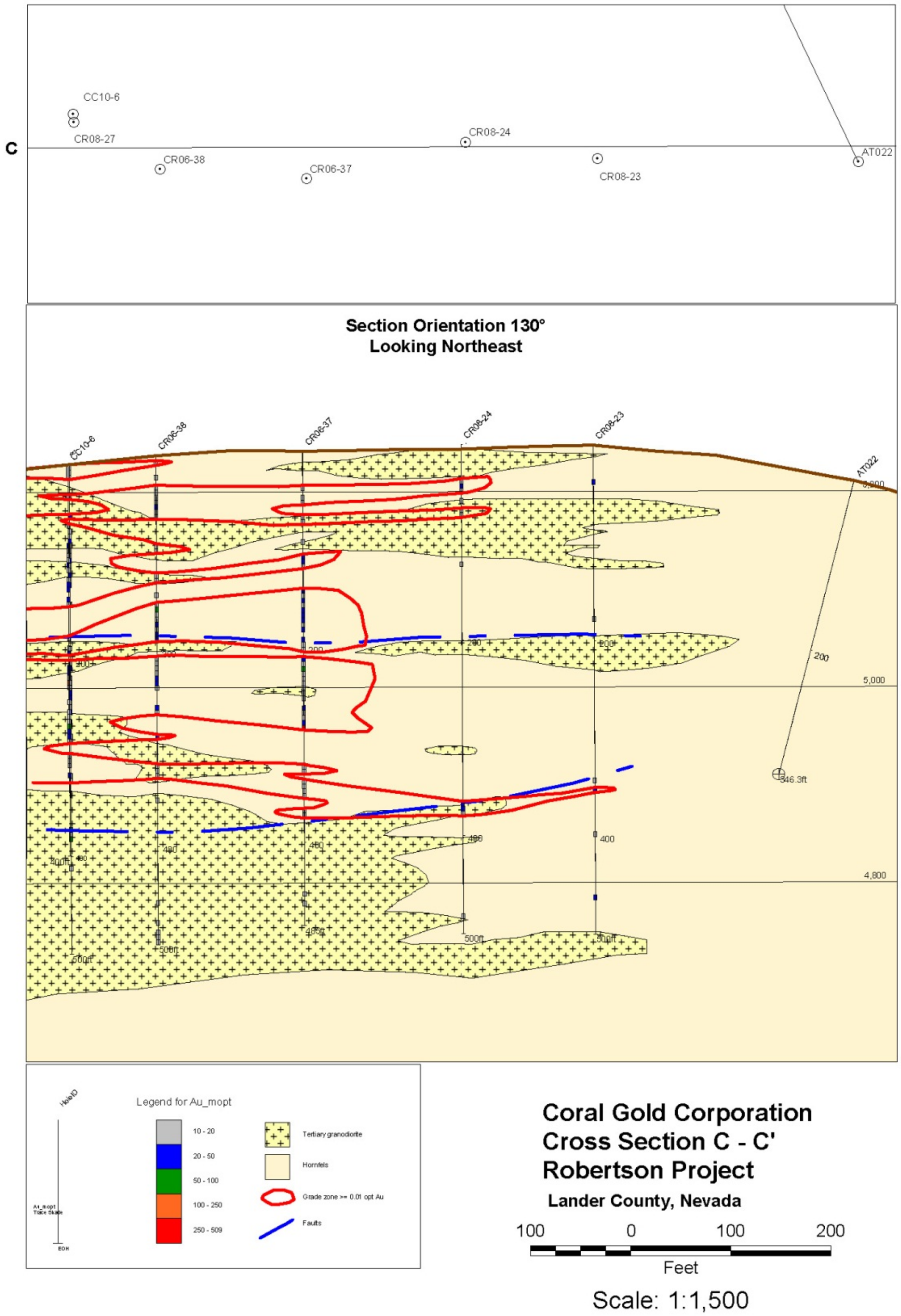


Figure 7-8: Typical cross section C-C' looking northeast through Altenburg Hill inferred mineral resource.

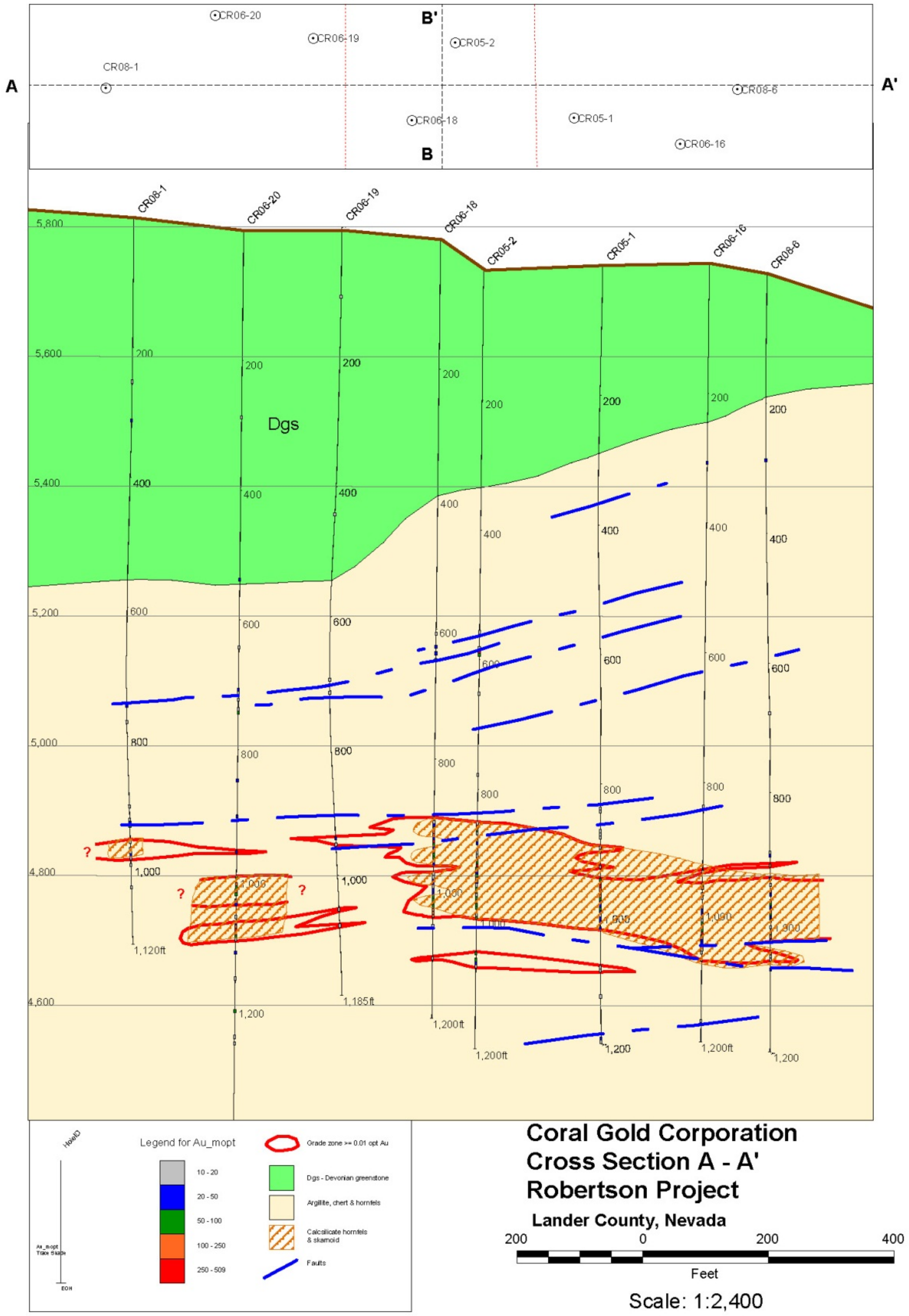


Figure 7-9: Cross section A-A' through Distal Zone mineral resource. Section looking east

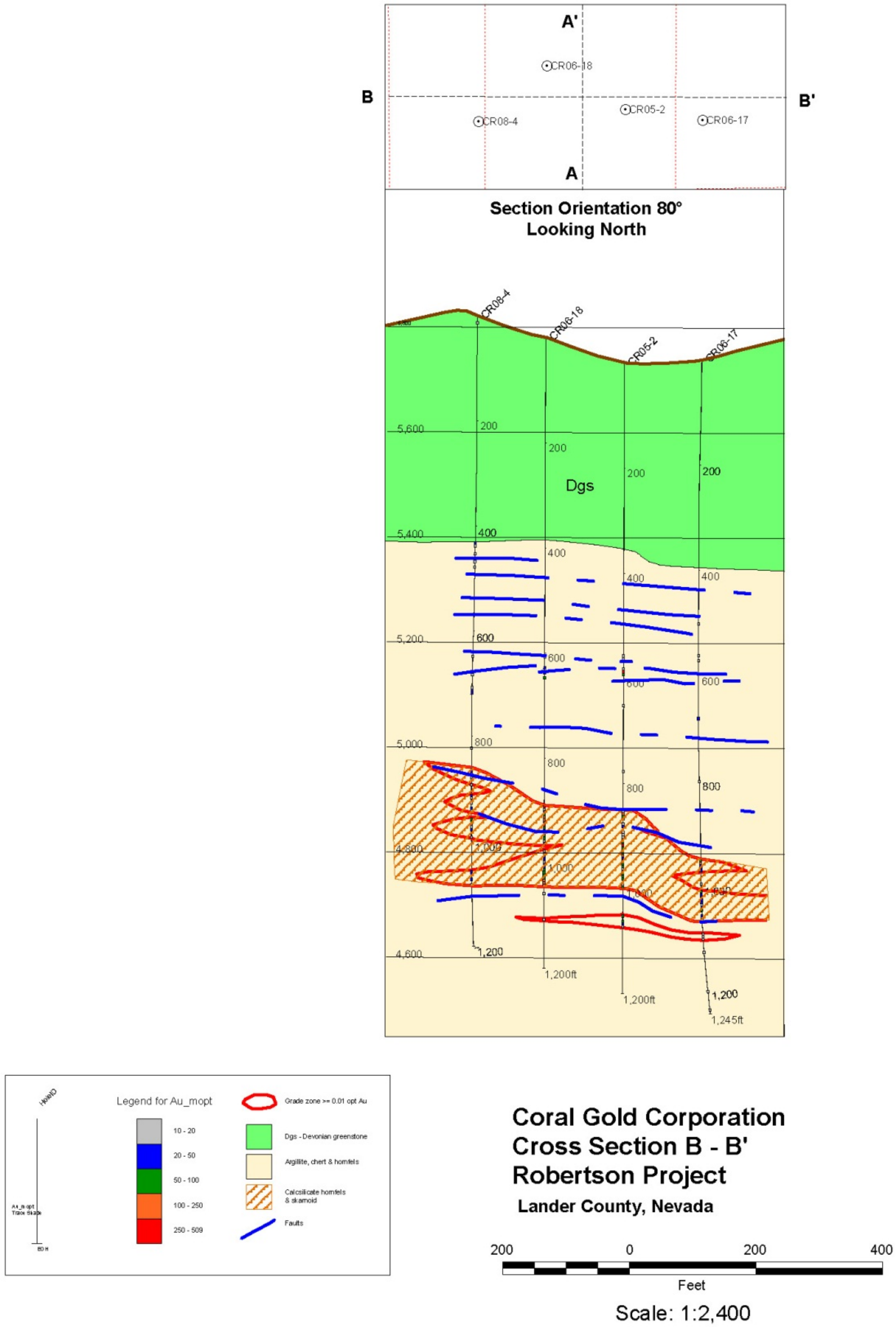


Figure 7-10: Cross section B-B' through Distal Zone inferred mineral resource. Section looking north.

SECTION 8.0 DEPOSIT TYPES

8.1 EXPLORATION TARGET CONCEPT

Economic gold mineralization at Robertson is both spatially and temporally associated with the Eocene-age Tenabo granodiorite. Gold mineralization in the Porphyry Zone occurs at the contact between a locally potassic-altered, fine-grained phase of the Tenabo granodiorite and calc-silicate hornfels and skarnoid units in the Elder Sandstone. These relationships suggest that this mineralization represents a proximal Au-Ag (Cu) skarn system as defined by Orris, et al. (1987) and Ray, et al. (1990). Gold in the Altenburg Hill and Gold Pan zones occur in a similar proximal position in biotite and calc-silicate hornfels at the faulted (?) contact of the granodiorite stock, related dikes, sills, and irregular diorite endoskarn bodies (Candee, 1996). The higher-grade 39A Zone is strongly controlled by intersecting high-angle fault zones developed in a highly fractured layered sequence of retrograde-altered biotite, calc-silicate hornfels and skarnoid. The deposit is developed 800 ft to 1,000 ft from the contact of the granodiorite stock and shares many mineralogical and geochemical characteristics of "distal" gold skarn, as defined by Ray et al., (1990).

The "Distal" Zone consists of a series of persistent, but narrow flat-lying mineralized zones, situated well outside the thermal metamorphic aureole of the stock that are postulated to represent a more distal position of the Tenabo Au-Ag (Cu) hornfels/skarn system. This mineralization is hosted locally by biotite, quartz, calc-silicate hornfels and skarnoid which developed in layered sequences of silty carbonaceous shale, siltstone and very minor silty limestone. Along with the lack of hornfels at the surface or the presences of intrusive rocks at depth, the highest grade gold mineralization is closely associated with an 80-ft-thick sequence of quartz and calc-silicate hornfels and a distinctive fine-grained yellow-brown skarnoid layer. Mineralization is often accompanied by weak retrograde actinolite-tremolite-chlorite alteration along with very strong enrichment in arsenic.

Also outside the metamorphic aureole are a series of locally mineralized, NNW-striking moderate to high-angle faults marked by discontinuous 5-ft- to +50-ft-thick, weakly to moderately silicified breccia zones in argillite, shale and chert. Surface rock chip sampling and RC drilling of these breccia outcrops have returned significant ore grade values. Many of the anomalous gold values from outcrop sampling have not been followed up and at least three ore-grade gold drill intercepts remain open along strike and down dip.

High-grade, east-west striking (?), quartz-sulfide ± calcite veins and (or) silicified zones with strong quartz (silica)-sericite-(pyrite) alteration envelopes, have been defined in the Gold Pan Zone and include a number of veins in the Phoenix and Silver Safe mines. The highest grades (+0.5 ozAu/t) occur at intersections with NW-striking faults and fracture zones. These veins and silicified structural zones carry significant values in Au-Ag, As-Sb ± Hg, Cu-Pb-Zn.

Deep drilling at Robertson in 2007 intersected Carlin-type (C-T) mineralization in hole TV07-2 hosted by variably altered limestone in the lower plate of the RMT. The gold values and trace element geochemistry returned in this hole, along with its geologic setting, host rock composition and hydrothermal alteration, meet the criteria for sediment hosted C-T mineralization as defined by Teal and Jackson (1997). These criteria include the following features:

- Single to multiple dike-filled faults, locally mineralized and showing evidence of protracted episodic movement.
- Multiple intrusive events (Cretaceous-Late Eocene).
- Significant gold mineralization in the district (>1million ounces of gold).

- Major NNW- to NW-trending fault zones with intersecting NE-trending faults.
- Map-scale anticlinal folds in lower plate carbonate rocks.
- Evidence that faults acted as major fluid conduits (positive geochemistry).
- Favorable carbonate host strata in the lower plate.

8.2 GEOLOGICAL MODEL FOR THE ROBERTSON PROPERTY

Gold mineralization within the Tenabo intrusive complex exhibits characteristics of an Au-Ag (Cu) hornfels/skarn system. Near-term exploration at Robertson will focus on expanding currently defined low-grade near surface resources (Porphyry, Altenburg Hill and Gold Pan Zones) and assess their potential to be brought into production. Longer term, the deeper resources (39A and Distal Zones) will be further evaluated for their potential to contribute to future production. Additional continued exploration for high-grade C-T Au mineralization hosted by favorable lower plate rocks should also be continued.

The geological setting for these deposit-types includes:

- structural intersections of high-and low-angle faults,
- replacement of relatively favorable lithologies for development of skarn,
- late-stage quartz-sulfide ± calcite “gash” veins within a strike-slip fault environment,
- proximity to the tenabo stock for proximal and distal au-ag (cu) hornfels/skarn,
- Igneous dike-filled faults.
- Deep high-grade Carlin-type mineralization hosted in lower plate carbonate rocks.

The Robertson Au-Ag (Cu) hornfels/skarn system contains a series of mineralized zones spatially associated with an intermediate-composition intrusive complex of late Eocene age. Gold occurs in highly fractured, fine-grained siliciclastic sedimentary and lesser volcanic rocks at or near contacts with intrusive rocks. Major fault zones controlled not only emplacement of the Tenabo stock and later dike swarms, but also provided pathways as well as sites of deposition for Au-Ag (Cu) mineralization. In northern Nevada, a number of Au-Ag (Cu) skarn systems have become significant producing gold mines and several identified mineral resources remain unexploited. Examples of deposits with similar geologic settings include McCoy/Cove mine, Phoenix Project (Battle Mountain district), Buffalo Valley mine and Redline prospect, all of which are on the Battle Mountain-Eureka trend.

At Robertson local Au-As-Hg ± Sb anomalies, thought to represent leakage of potential Carlin-type mineralization, occur in upper plate rocks outside or near the edge of Ag-base metal zones located southwest and north-northwest of the Tenabo stock (Figure 7-1). Similar relationships are described at Cove/McCoy where Au-Ag base metal retrograde skarn occurs at or near contacts with the Eocene age Brown stock, with fracture controlled sediment hosted Ag-Au base metal mineralization in a distal position and “true” Carlin-type mineralization located at the outer margin of the Ag-base metal mineralization (Johnston, 2000). Also, the close spatial and partly cross cutting relationship of a Carlin-type gold system in the East and West Archimedes deposits with Ag-base metal mineralization at Eureka, Nevada (Dilles et al., 1996), suggests a genetic association between the two assemblages that may also exist at Robertson.

SECTION 9.0 EXPLORATION

9.1 PRE-CORAL EXPLORATION ACTIVITY

The most important modern exploration activity prior to Coral's involvement in the property are those of Superior Oil (1968-70) and E & B Exploration (1980-81). The intersections of high-grade gold in holes T 29 (20 ft/1.154 ozAu/t) and T 83 (10 ft/0.525 ozAu/t), near the Gold Pan mine by Superior, along with at least six +100-ft-thick drill intercepts of lower grade mineralization, were the first indications that real potential for open-pit-mineable gold existed on the property. E & B Exploration advanced the property by undertaking the first district-wide mapping and sampling programs which succeeded in identifying a number of high priority gold anomalies. More importantly, they followed up the Superior drilling results in the Gold Pan Zone and provided a basis for defining a resource in that area.

9.2 CORAL GOLD EXPLORATION ACTIVITIES (1986-1989)

Exploration of the district by Coral during the period 1986-89, focused mainly on developing a resource base from which a mineable reserve could be defined. As a result, much of their activity was confined to drilling the known resource areas at Gold Pan, Gold Quartz and Gold Quartz Extension (West). At this time Coral also contracted two geophysical surveys (Barringer, 1987 and E-Scan, 1988) and an extensive geochemical orientation study (Barringer, 1987). None of these studies provided significant encouragement. During 1988, Coral decided to place the Gold Quartz West and poorly defined Gylding zones into production. But overly optimistic and aggressive resource estimates, and poor geological understanding resulted in less than expected head grades and poor heap leach recoveries. This ultimately led to the decision to halt mining less than a year later. In 1989, Coral focused its exploration in the upper Triplet Gulch area, where several widely scattered drill intercepts provided some encouragement. However, additional drilling has failed to define even a modest resource in this area.

9.3 AMAX GOLD ACTIVITIES (1990-1996)

From 1990, until they withdrew from the agreement, Amax completed a district-wide exploration program that included drilling of over 176,000 ft of combined RC and diamond core, detailed geological and geochemical reconnaissance of the entire claim block (including the Excluded Claims), rock chip and mine dump sampling, detailed geological mapping and site-specific geophysics. During that time period, Amax identified the 39A, Porphyry, and Altenburg Hill Zones and continued to define the previously identified Gold Pan Zone. As a result of the detailed reconnaissance stream sediment and follow-up rock chip sampling programs, a series of strong gold anomalies were identified within the claim block, but outside the main district. In 1999, Cortez drill-tested one of the Amax anomalies that was outlined on the Lander Ranch claims.

9.3.1 39A Zone

Beginning in 1990, Amax drilled three "deep" RC holes totaling about 9,340 ft to test the concept that potential high-grade gold mineralized hosted by lower plate carbonate rocks existed at depth in the Tenabo district. All of the holes failed to reach the lower plate, but did provide substantial encouragement for near surface gold resources. These holes ultimately led to discovery of the higher grade 39A Zone near the west end of the district. As constrained by a 0.01 ozAu/t cutoff grade, the 39A Zone is currently defined by 104 RC and 4 core holes, totaling about 54,415 ft, including 13 RC holes drilled by Cortez in 1999 and 58 RC holes drilled by Coral Gold Resources in 2004-2008.

Table 9.1 is a compilation of significant gold intercepts from holes drilled in the 39A Zone, using a 0.01 ozAu/t cutoff grade and a 20 ft minimum thickness.

9.3.2 Porphyry Zone

Sampling a series of small bedrock exposures at the east end of the district in areas disturbed by historic placer mining, returned ore-grade results from highly fractured outcrops of potassic-altered, fine-grained diorite. Follow-up RC drilling encountered several +150-ft-thick intervals of low-grade Au mineralization beginning at the bedrock surface. Later, after a series of wide spaced RC holes, it was determined that a potentially large, shallow, low-grade oxidized resource was present. This led to grid drilling on approximately 120-ft-centers with 147, mostly vertical RC holes and 50 core holes, for a combined total of about 80,000 ft of drilling. The drilling defined a proven + probable mineral reserve estimated to contain 13,556,125 short tons of ore at a grade of 0.019 ozAu/t and containing about 255,000 ounces of gold (Amax, 1994). **The presentation of data form this historic mineral reserve is intended to provide context for the Amax exploration activities and should not be construed as representing a current mineable reserve**. A compilation of significant gold intercepts from holes drilled in the Porphyry Zone are presented in Table 9.2.

9.3.3 Altenburg Hill Zone

As a result of ore-grade gold values returned from several surface rock chip samples on Altenburg Hill and significant gold in follow-up grid soil sampling, a series of 15 shallow, vertical and inclined RC holes were drilled in 1993 by Amax. Results of this drilling suggested potential for shallow, structurally controlled, low-grade mineralized zone. In 1996, these intercepts were followed up by grid drilling on approximately 100-ft-centers with 38 vertical RC holes and one core hole, totaling 9,485 ft. The drilling defined an indicated resource of 3,500,000 short tons of mineralized material with an average grade of 0.018 ozAu/t, using a 0.01 ozAu/t cutoff grade (Candee, 1996). **The presentation of data form this historic mineral resource estimate is intended to provide context for the Amax exploration activities and should not be construed as representing a current mineable reserve.**

Table 9.1: Summary of significant gold intercepts in the 39A Zone*.

Hole No.	UTM East	UTM North	Collar Elev. (ft)	From (ft)	To (ft)	Thickness (ft)	Au Grade oz/t
AT-39A	525788	4462299	5506	670	765	95	0.134
AT-40	**525815**	4462279	5494	665	780	115	0.088
AT-44	525768	4462278	5505	575	730	155	0.047
AT-45	525757	4462320	5523	580	770	190	0.101
AT-49	525720	4462294	5512	500	700	200	0.124
AT-51	525726	4462359	5542	570	710	140	0.105
AT-56	525683	4462328	5529	505	635	130	0.084
AT-57	525673	4462288	5539	470	705	245	0.088
AT-58	525652	4462252	5545	350	485	135	0.056
AT-64	525646	4462184	5509	340	445	105	0.116
AT-65	525679	4462213	5507	420	455	35	0.072
				480	584	105	0.042
AT-68	525767	4462390	5516	570	730	160	0.054
CR04-8	525760	4462347	5523	600	735	135	0.074
CR04-12	525624	4462176	5515	295	375	80	0.063
CR04-15	525752	4462291	5513	540	575	35	0.078
				600	760	160	0.113
CR04-16	525734	4462268	5507	545	690	145	0.093
CR04-17	525698	4462309	5525	510	710	200	0.118
CR04-18	525847	4462258	5477	600	615	15	0.154
				745	845	100	0.074
CR05-3	525657	4462173	5505	385	450	65	0.089
CR05-8	525796	4462540	5582	480	540	60	0.039
				785	855	70	0.073
CR06-2	525712	4462226	5506	550	595	45	0.185
CR06-4	525700	4462276	5520	515	675	160	0.057
CR06-5	525792	4462258	5491	680	755	75	0.102
CR06-8	525783	4462381	5515	615	740	125	0.084
CR06-14	525821	4462500	5564	565	630	70	0.067
CR08-13	525808	4462404	5509	680	780	100	0.074
CR08-17	525868	4462447	5523	820	910	90	0.057
99413	525608	4462168	5514	300	380	80	0.163
CAT-5	525672	4462289	5538	470	595	125	0.112
CAT-6	525753	4462318	5524	620	770	150	0.163
CAT-58	525723	4462296	5516	495	700	205	0.059
CAT-59	525722	4462332	5536	530	585	55	0.059
				600	755	155	0.149

*Calculated using a 0.01 oz A/t cutoff grade.

9.3.4 Gold Pan Zone

Also in 1996, Amax drilled 25 RC holes in the previously identified and partly defined Gold Pan Zone resource area. The drilling was designed to provide further detail on the distribution and continuity of high-grade mineralization, as well as close off possible extensions to mineralization along the northwest edge of the zone. Upon completion of this program an indicated resource of 8,300,000 short tons of mineralized material with an average grade of 0.024 ozAu/t, using 0.01 ozAu/t cutoff grade, was estimated by Amax to be present in the Gold Pan Zone (Candee, 1996).

Table 9.2: Summary of significant gold intercepts in the Porphyry Zone*.

Hole No.	Type	From (ft)	To (ft)	Thickness (ft)	Grade ozAu/t
AT-72	RC	225	325	100	0.039
AT-123	RC	30	85	55	0.05
AT-150	RC	150	250	100	0.054
AT-151	RC	30	85	50	0.054
AT-165	RC	60	140	80	0.049
AT-167	RC	200	465	265	0.025
AT-171	RC	55	120	65	0.051
AT-187	RC	20	165	145	0.032
AT-188	RC	175	300	125	0.043
		365	405	40	0.035
AT-190	RC	360	450	90	0.068
AT-191	RC	100	130	30	0.155
		175	250	75	0.052
AT-195	RC	55	130	75	0.029
AT-199	RC	85	175	90	0.041
AT-209	RC	70	125	55	0.037
		175	210	35	0.052
AT-211	RC	25	70	45***	0.082
CAT-10	CORE	20	100	80	0.044
CAT-12	CORE	215	320	105	0.035
CAT-21	CORE	15	110	95	0.053
CAT-22	CORE	95	155	60	0.039
CAT-23	CORE	105	175	70	0.045
CAT-24	CORE	20	90	70	0.035
CAT-30	CORE	90	135	45	0.054
CAT-33	CORE	290	460	170	0.037
CAT-37	CORE	280	490	210	0.072
CAT-44	CORE	215	355	140	0.09
CAT-47	CORE	155	215	60	0.037
		250	345	95	0.127
CAT-49	CORE	270	410	140	0.036
CAT-52	CORE	210	285	75***	0.026
		360	480	120	0.067
CAT-54	CORE	140	215	75***	0.038
CAT-55	CORE	345	440	95	0.049
CAT-57	CORE	30	110	80	0.172

*Average grade calculated using 0.02 ozAu/t cutoff.
**Interval thickness from near vertical drill holes. ***Interval thickness from angle hole.

9.4 CORTEZ ACTIVITIES (1999)

In 1999, Cortez completed a 56,000 ft RC drilling program comprised of 47 holes that focused on extending the 39A mineralization, following up gold-anomalous surface samples mainly in upper Mill Gulch with a series of shallow trenching and testing a series of NW trending structures in upper Triplet Gulch (Hebert, 1999). Additionally, Cortez completed a grid enzyme leach survey over the pediment gravel east of Altenburg Hill, which was followed up by drilling a single 3,000-ft-deep mud rotary hole. A series of four RC holes were drilled in the vicinity of AT-3 to test a mineralized zones thought to represent a distal position of the porphyry system. Finally, single 1,500-ft-deep RC hole was drilled on the Lander Ranch claims, located about one mile north of the main district, to test a group of strong gold-anomalous surface samples taken from a moderately silicified fault breccia.

In the 39A Zone, Cortez focused 13 RC holes along an apparent NE-SW trend of mineralization testing for possible extensions of the zone. Only three of the holes encountered significant mineralization, but they failed to substantially expand the zone.

Four step-out holes were drilled by Cortez in the vicinity of AT-3, providing additional evidence that distal hornfels-skarnoid mineralization, hosted by a thick sequence of fine-grained siliciclastic rocks and forming a broad zone up to 800 ft in a NNW direction and 350 ft to the northeast.

A single 1,500-ft-deep RC hole was completed in the vicinity of a mineralized fault zone on the Lander Ranch claim block, which encountered significant gold values in two intervals from deep in the hole. These intersections suggest that ore-grade mineralization encountered in this drill hole at 1,100 ft may extend to the surface where ore-grade samples have been taken.

The deep mud-rotary-drill hole (99432) collared on an enzyme-leach-generated anomaly in the pediment east of Altenburg Hill, failed to cut any significant gold values and remained in siliceous upper plate rocks. The hole is located very close to the sites of USGS drill holes No. 2 and 3, diamond core holes that reached over 2,000-ft- and 1,250-ft-deep, respectively (Wrucke, 1974). Except for hole No. 3, which end in granodiorite of the Tenabo stock, these holes also encountered thick intervals of siliceous upper plate rocks.

9.5 CORAL GOLD RESOURCES ACTIVITIES (2004-2006)

During 2004 and 2005, Coral conducted three drilling programs consisting of 32 RC holes totaling 24,020 ft on the Robertson Property. The focus of this exploration was to expand and further define the 39A Zone, test the “deep” Gold Pan Zone for extensions of the 39A Zone and offset previous ore-grade intersections in the “distal target area”. Results of the 2004-2005 drilling programs succeeded in increasing the contained ounces in the 39A Zone by expanding the area of higher-grade gold within the defined resource area and by modestly expanding the resource to the north and east. A total of 22 holes were directed at expanding and testing continuity of the 39A Zone.

In 2006, Coral completed 48 RC holes totaling 35,615 ft on the Robertson Property. This drilling program focused on expanding and defining the 39A, Altenburg Hill and Porphyry Zone mineral resources, and offset drilling of the emerging Distal Zone.

A summary of Coral’s 2004-2006 drilling programs were previously reported in the NI 43-101 compliant technical report *Mineral Resource Estimate for the Robertson Property, Lander County, Nevada, USA* (Stokes et al., 2008) available on SEDAR.

Also during 2006, Coral completed a program designed to evaluate the potential of its Robertson Property to host deep CT-type gold deposits in favorable carbonate rocks in the lower plate of the RMT. The program consisted of geological mapping, 100 m x 100 m grid soil sampling (1,032 samples), collecting about 196 rock chip/dump samples and conducting a detailed gravity survey (300 stations/400m spacing).

The 2006 soil samples were submitted to ALS Chemex for sample prep and multi-element analysis. All samples were dry screened to +80 mesh-10 mesh. The -80 mesh and +10 mesh fractions were discarded. The samples were then pulverized to approximately 80 % passing -80 mesh and analyzed for Au by FA/AA, using a one-assay ton charge, and multi-element analyses using a combination of ICP-MS (inductive coupled plasma with mass spectroscopy) and ICP-AES (inductively coupled plasma with atomic emission spectroscopy) using a 5 g sample and aqua-regia digestion. The gold determination using FA/AA has a lower limit of detection (LLD) =5 ppb. The multi-element (61 elements) analysis using a combination ICP-MS and ICP-AES included Ag (LLD=0.01 ppm), As (LLD=0.2 ppm), Sb (LLD=0.05 ppm), Hg (LLD=0.01 ppm), Tl (LLD=0.02 ppm), Ba (LLD=10), Bi (LLD=0.01 ppm), Se (LLD=1 ppm), Cu (LLD=0.2 ppm), Mo (0.05 ppm), Pb (LLD=0.2 ppm), and Zn (LLD=2 ppm).

In 2006, Coral collected 196 rock chip and mine dump samples from the area of the recently acquired View claims and from the NW part of the Robertson property. Samples were analyzed by ALS Chemex for Au by FA/AA with a lower limit of detection (LLD) of 5 ppb. Multi-element determinations were made using a combination of inductively coupled plasma with mass spectroscopy and atomic emission spectroscopy for 61 elements including Ag (LLD=0.01 ppm), As (LLD=0.2 ppm), Sb (LLD=0.05 ppm), Hg (LLD=0.01 ppm), Tl (LLD=0.02 ppm), Ba (LLD=10), Bi (LLD=0.01 ppm), Se (LLD=1 ppm), Cu (LLD=0.2 ppm), Mo (0.05 ppm), Pb (LLD=0.2 ppm), and Zn (LLD=2 ppm).

Coral completed a detailed gravity survey during 2006 that covered all of the Robertson property as well as adjacent portions of the Excluded claims and claims owned by Cortez (with their permission). The survey was planned, supervised and the data interpreted by Bob Ellis, a consulting geophysicists. The acquisition and processing of the gravity data was completed by Magee Geophysical Services using state-of the-art gravity meters with high-accuracy GPS units to determine UTM coordinates and altitude of gravity stations. Gravity measurements were taken at approximately 300 stations spaced roughly 400 meters apart. In addition, Coral acquired detailed gravity data for the adjacent Lander Ranch and Blue Nugget claims as well as compiling all public domain gravity data

Result of these studies defined at least two target areas: The NNW-striking high-angle Try and Tomcat fault zones. Both are strongly developed structural zones that reach >5,000 ft along strike and up to 1,000 ft wide and show evidence of recurrent movement over a long time period. Certain fault segments are filled by multiple igneous dikes and contain local zones of strong silica-clay alteration that carry gold values up to 6.06 ppm and arsenic values >10,000 ppm. These structural zones are further defined by coincident linear gravity and magnetic features interpreted to represent deep expressions of the faults mapped at surface. Evidence that these structural zones served as important conduits for the transport of gold is provided by soil and rock geochemistry which define a series of strong gold-arsenic and silver-base metal anomalies that coincide precisely with the areal extent of the mapped faults zones and geophysical linear features (McCusker, 2007).

SECTION 10.0 DRILLING

10.1 CORAL DRILLING (2007)

Because of various permitting delays in 2007, Coral was unable to drilling targets in the “Core” area of the Robertson Property. Instead, Coral drilled two vertical flooded reverse circulation holes that tested deep targets which were identified along the west side of Coral’s Robertson claim block during the 2006 evaluation. Holes TV07-1 and TV07-2 were drilled to depths of 2,990 ft and 3,450 ft, respectively. TV07-1 intersected a thick sequence of fine grained siliceous sedimentary and volcanic rocks followed by biotite and quartz hornfels equivalents in the upper plate of the RMT. Although the hole failed to reach the lower plate rocks, it did intersect a number of narrow low-grade zones. TV07-2 was collared along a dike-filled splay of the Try fault zone and intersected limey mudstone in the lower plate of the RMT starting at 3,080 ft. The hole returned a 200-ft-thick interval of weakly to strongly anomalous gold values ranging from 0.031 to 2.190 ppm gold, hosted by altered lower plate carbonates rocks (McCusker, 2008). A summary of significant assay results for the 2007 deep drilling are presented in Table 10.1.

Table 10.1: Summary of significant gold intercepts in TV07-1 and TV07-2.

Hole ID	From (ft)	To (ft)	Thickness (ft)	Au, oz/t	Ag, oz/t	As, ppm
TV07-1	680	720	40	0.013	0.137	290
TV07-1	1640	1650	10	0.020	0.027	886
TV07-1	1740	1750	10	0.032	0.020	882
TV07-1	1930	2010	80	0.020	0.030	495
TV07-1	2460	2480	20	0.014	0.024	995
TV07-2	3080	3090	10	0.064	0.015	278
TV07-2	3120	3140	20	0.041	0.011	1560
TV07-2	3150	3160	10	0.010	0.003	333
TV07-2	3180	3200	20	0.008	0.003	344

10.2 CORAL DRILLING (2008)

During 2008, Coral completed 33 RC drill holes totaling 22,835 ft. Of the total drilling, 15 vertical holes were drilled on the north flank of Altenburg Hill mainly as infill and offset of the wide spaced 2006 drilling. Six holes were completed in the gravel-covered South Porphyry Zone, an extension to the Porphyry Zone, as follow up of the intercept in CR06-30 (140 ft/0.04 ozAu/t). The six vertical holes were offset up to 350 ft east and 420 ft north of CR06-30. Seven vertical holes were drilled in the 39A Zone to test the potential for expanding the current resource to the northeast and five vertical holes were completed in the Distal Zone to test for extensions north and west of the currently defined resource. Assay results for the 2008 drilling program are presented in Table 10.2.

10.2.1 Altenburg Hill Zone

The infill and offset drilling of the 2006 drilling in the Altenburg Hill Zone resulted in a modest expansion and increased confidence in the continuity of mineralization. The drilling also indicated that mineralization remains open for potential expansion northward and with limit expansion both to the east and west of the current resource. The most significant assay results were returned in holes CR08-21 (100 ft / 0.019 ozAu/t starting at surface), CR08-26 (130 ft/0.020 ozAu/t from 25 ft), CR08-30 (160 ft/0.023 ozAu/t from 5 ft) and CR08-31 (135 ft/0.018 ozAu/t from surface). These intervals were calculated using 0.01 ozAu/t cutoff grade.

10.2.2 South Porphyry

The results from offset drilling in the gravel-covered South Porphyry Zone indicate that mineralization diminishes rapidly to the east but is continuous to the north for at least 350 ft. Hole CR08-37 was collared 350 ft north of CR06-30 and intersected 135 ft averaging 0.020 ozAu/t from 135 ft, under 30 ft of gravel, and CR08-38 was located 125 ft north of CR06-30 and returned 50 ft averaging 0.039 ozAu/t from 160 ft, followed by 50 ft averaging 0.017. Mineralization in both holes is disrupted by a series of E-W striking post-mineral feldspar porphyry dikes.

10.2.3 39A Zone

Drilling in the 39A Zone resulted in a modest expansion to the northeast of CR06-8 (125 ft/0.084 ozAu/t from 615 ft). CR08-13 was collared 105 ft to the northeast and returned 100 ft/0.075 ozAu/t starting at 680 ft, CR08-17, collared 355 ft northeast intersected 90 ft/0.057 ozAu/t from 820 ft and CR08-16 located 560 ft northeast returned 130 ft/0.014 ozAu/t starting at 830 ft. These results, as well as other holes in this area, indicate that higher grade 39A mineralization is diminishing to the northeast but that lower grade gold values continue to define the zone.

10.2.4 Distal Zone

Drilling in the Distal Zone clearly identified the west edge of mineralization and a single hole CR08-1 drilled north of the zone returned only scattered low-grade gold values suggesting that mineralization is diminishing in that direction. However, offsetting CR06-16 (110 ft/0.068 ozAu/t from 955 ft), the southern-most hole in the Distal Zone 125 ft to the southeast, intersected 120 ft averaging 0.071 ozAu/t from 910 ft, including 10 ft that average 0.354 ozAu/t, in CR08-6. Drilling results indicate potential for expanding the Distal Zone resource farther south.

Table 10.2: Summary of 2008 Drilling

ALTENBURG HILL								
Hole No.	UTM north	UTM east	Collar elev. (ft)	Total depth (ft)	From (ft)	To (ft)	Thickness (ft)	Gold Grade, ozAu/t
CR08-10	4461369	527204	5265	500	95	170	75	0.014
CR08-19	4461311	527199	5293	500	40	115	75	0.027
					175	235	65	0.018
					265	280	15	0.071
CR08-20	4461399	527079	5278	500	35	65	30	0.010
					165	185	20	0.012
					210	230	20	0.037
CR08-21	4461421	527145	5259	500	0	100	100	0.019
					130	200	70	0.017
					270	305	35	0.017
					320	370	50	0.017
CR08-22	4461341	527236	5264	500			no significant values	
CR08-23	4461349	527273	5236	500			no significant values	
CR08-24	4461380	527248	5244	500	30	75	45	0.012
					360	380	20	0.024
CR08-25	4461428	527230	5209	500	60	150	90	0.019
					375	415	40	0.013
CR08-26	4461463	527201	5204	500	25	155	130	0.020
					290	315	25	0.018
CR08-27	4461475	527165	5209	500	0	15	15	0.013
					50	125	75	0.018
					200	265	65	0.021
					370	385	15	0.047
CR08-28	4461385	527303	5192	500	405	495	90	0.013
CR08-29	4461446	527311	5172	500	60	140	80	0.021
					435	450	15	0.034
CR08-30	4461477	527271	5172	425	5	165	160	0.023
					280	345	65	0.012
					365	385	20	0.024
					405	425	20	0.030
CR08-31	4461503	527234	5195	500	0	135	135	0.018
					330	365	35	0.018
					440	455	15	0.031
CR08-32	4461541	527260	5186	345	0	85	85	0.014
					305	320	15	0.023
Total				**7270**				

SOUTH PORPHYRY ZONE								
			Collar	Total				Gold
Hole No.	UTM north	UTM east	elev. (ft)	Depth (ft)	From (ft)	To (ft)	Thickness (ft)	Grade, ozAu/t
CR08-33	4461648	527518	5127	500	95	110	15	0.025
CR08-34	4461655	527565	5143	500			no significant values	
CR08-35	4461698	527511	5163	500	145	170	25	0.017
CR08-36	4461750	527530	5169	500	65	75	10	0.015
					220	230	10	0.032
					470	480	10	0.014
CR08-37	4461753	527479	5173	500	135	270	135	0.020
CR08-38	4461694	527458	5166	500	160	210	50	0.039
					370	420	50	0.017
Total				**3000**				

39A ZONE								
Hole No.	UTM north	UTM east	Collar elev. (ft)	Total depth (ft)	From (ft)	To (ft)	Thickness (ft)	Grade, ozAu/t
CR08-11	4462209	525782	5487	760	80	85	5	0.061
					165	170	5	0.057
					205	210	5	0.070
					660	685	25	0.034
					695	710	15	0.012
CR08-12	not drilled							
CR08-13	4462406	525809	5504	850	680	780	100	0.075
including					690	770	80	0.088
including					710	735	25	0.170
CR08-14	4462501	525821	5517	1000	80	155	75	0.017
					340	380	40	0.025
					555	580	25	0.012
					825	855	30	0.015
					905	925	20	0.022
CR08-15	4462482	525864	5551	1000	80	100	20	0.021
					385	395	10	0.064
					600	610	10	0.046
					835	900	65	0.037
					965	990	25	0.027
CR08-16	4462504	525898	5527	1040	55	75	20	0.014
					140	170	30	0.017
					265	280	15	0.018
					295	320	25	0.021
					380	390	10	0.054
					760	800	40	0.037
					830	965	135	0.014
CR08-17	4462445	525868	5515	970	95	110	15	0.025
					225	265	40	0.045
					820	910	90	0.057

including					820	830	10	0.327
CR08-18	4462472	525941	5487	1025	245	265	20	0.031
					285	350	65	0.039
					620	625	5	0.069
					710	735	25	0.022
					765	770	5	0.137
					965	1025	60	0.019
Total Footage				**6645**				

DISTAL ZONE								
Hole No.	**UTM north**	**UTM east**	**Collar elev. (ft)**	**Total depth (ft)**	**From (ft)**	**To (ft)**	**Thickness (ft)**	**Grade, ozAu/t**
CR08-1	4462897	525324	5804	1120	925	940	15	0.043
					960	1000	40	0.013
CR08-2	not drilled							
CR08-3	4462768	525301	5831	1200	680	730	50	0.027
					940	955	15	0.014
					980	1035	55	0.024
					1050	1080	30	0.026
CR08-4	4462714	525307	5806	1200	430	480	50	0.011
					705	720	15	0.021
					860	885	25	0.023
					910	945	35	0.015
					955	980	25	0.089
					1055	1085	30	0.018
CR08-5	4462659	525314	5791	1200	630	650	20	0.031
					820	835	15	0.028
					850	895	45	0.058
including					885	890	5	0.27
					925	945	20	0.022
					1080	1095	15	0.028
					1165	1175	10	0.029
CR08-6	4462596	525378	5728	1200	890	900	10	0.032
					910	1030	120	0.071
including					910	920	10	0.354
including					965	970	5	0.24
					1045	1060	15	0.016
Total Footage				**5920**				

Calculated using 0.010 ozAu/t cutoff grade.

10.3 CORAL DRILLING (2010)

During 2010, Coral completed 12 RC holes totaling 8,000 ft in the lower Triplet Gulch area where a previously identified historic resource was located. The purpose of this drilling was to verify the gold grade, reported thickness and continuity of mineralization in this zone. Also in 2010, 14 HQ-diameter diamond drill holes totaling 6,450 ft were completed as twins of existing RC holes in the

South Porphyry, Altenburg Hill and Gold Pan Zones to test the reliability and continuity of gold grades returned by RC drilling acquire specific gravity and geotechnical data.

10.3.1 Lower Triplet Gulch RC Drilling

The primary purpose of this drilling program was to verify the historic drilling results used to calculate the Triplet Gulch (also referred to as the "South Zone") inferred mineral resource in the 2008 NI 43-101 Technical Report (Stokes, et al., 2008). In addition, the drilling was also directed at expanding the resource.

To evaluate the validity of this resource, Coral conducted a 12-hole RC program drilled on a grid with holes spaced 500 ft apart on lines also spaced 500 ft apart (Figure 10-6). The program consisted of 12 vertical RC holes, CR10-1 through CR10-12, totaling 8,000 ft and ranging in depth from 600 ft to 1,000 ft. Only two of the 12 RC holes, CR10-7 and -8, were completed to 1,000 ft. A summary of assay results, using a 0.01 ozAu/t cutoff grade is shown in Table 10.3.

The Triplet Gulch drilling occurred in an alluvial covered area that lies between 2,000- 4,000 ft southeast of the exposed southern contact of Tenabo granodiorite. The area is also 3,000 ft south of the Altenburg Hill inferred resource. Rocks exposed in the immediate vicinity if the drilling are mainly thermally metamorphosed upper plate greenstone, chert and argillite of the Devonian Slaven Chert, which are converted locally to biotite and quartz hornfels. Based on mapping and drilling, the overburden ranges from a few feet to 125-ft-thick.

The 2010 drilling results indicate that the while broad zones of anomalous gold values ($\geq$0.1 ppm) are present in the Lower Triplet Gulch area, only widely scattered narrow and discontinuous intervals of mineralization exceeding 0.01 ozAu/t were encountered. These results cast serious doubt as to the validity of the historic drilling and assaying that are the basis of the "South Zone" inferred mineral resource. The South Zone resource represents about 5.6% of the total of the total Robertson inferred mineral resources.

10.3.2 Diamond Drilling (2010)

Also in 2010, Coral completed a 14-hole diamond drilling program (CC10-1 through CC10-10 and CC10-12 through CC10-15) aimed at accessing the reliability and continuity of mineralization returned in the 2008 RC drilling in the South Porphyry, Altenburg Hill and Gold Pan Zones. Core drilling also provided geological and geotechnical data (RQD), and bulk samples for metallurgical testing. Eleven of the 14 core holes were drilled as twins of existing RC holes and were collared between 4 ft and 17 ft from their respective twin RC holes and with less than one foot of collar elevation difference. The drilling produced 960 twinned sample pairs, each representing a "twinned" five foot assay interval. A summary of assay results for the 2010 diamond drilling is presented in Table 10.4.

To compare the core vs. RC results the global average grades returned by all the core and corresponding twin RC holes were calculated. The average for all twin RC holes is 0.011 ozAu/t whereas the average grade of all core holes is 0.0107 ozAu/t, a difference of 0.0003 ozAu/t. If the top one percentile of assays is removed from the data

Table 10.3: Summary of assay results for the 2010 Lower Triplet Gulch RC drilling

Hole No.	Total Depth(ft)	Angle	Azimuth	From (ft)	To (ft)	Thickness (ft)	Grade Au, oz/t
CR10-1	600	-90	0	0	10	10	0.023
				130	135	5	0.0128
				155	160	5	0.0112
				195	200	5	0.0194
				210	220	10	0.0139
				255	260	5	0.0114
				280	295	15	0.0142
				460	480	20	0.0169
				500	505	5	0.0132
				555	560	5	0.0148
				585	590	5	0.0107
				595	600	5	0.0166
CR10-2	600	-90	0	55	60	5	0.0151
				135	140	5	0.023
				215	245	30	0.0344
				275	280	5	0.0232
				320	340	20	0.0207
				360	370	10	0.0439
				380	385	5	0.0207
				460	465	5	0.0298
				515	520	5	0.0295
				580	585	5	0.0143
CR10-3	600	-90	0	20	25	5	0.0214
				285	295	10	0.0326
				350	360	10	0.0182
				490	495	5	0.0102
				515	535	20	0.0404
				585	595	5	0.0162
CR10-4	600	-90	0	10	15	5	0.0417
				320	340	20	0.0217
				500	505	5	0.0122
				580	600	20	0.0258
CR10-5	600	-90	0	105	110	5	0.0283
				125	135	10	0.016
				250	265	15	0.0118
				310	325	15	0.0184
				390	400	10	0.014
				430	440	10	0.0172
				480	485	5	0.0208
				500	515	15	0.0129
				585	595	10	0.0254
CR10-6	600	-90	0	320	325	5	0.0126
				335	340	5	0.0118
CR10-7	1,000	-90	0	80	85	5	0.0476
				95	100	5	0.418

				380	405	25	0.0125
				570	575	5	0.0123
				605	630	25	0.0119
				705	715	10	0.0313
				840	845	5	0.0146
				870	885	15	0.0142
				940	945	5	0.0389
				970	975	5	0.0205
				995	1,000	5	0.0516
CR10-8	1,000	-90	0	60	65	5	0.0463
				75	80	5	0.014
				185	210	25	0.0673
				365	380	15	0.0195
				525	530	5	0.0217
				540	550	10	0.0274
				560	565	5	0.011
				610	670	60	0.0182
				735	745	10	0.0172
				765	770	5	0.0167
				780	810	30	0.0164
				855	870	15	0.0193
				925	930	5	0.0152
				990	995	5	0.0174
CR10-9	600	-90	0	75	85	10	0.0257
including				75	80	5	0.4891
				180	185	5	0.0157
				385	390	5	0.0162
CR10-10	600	-90	0	120	125	5	0.0259
CR10-11	600	-90	0	385	395	10	0.0118
CR10-12	600	-90	0	100	105	5	0.0197
				500	505	5	0.0216

Calculated using a 0.010 ozAu/t cutoff grade

(≥2.244 ppm Au for RC and ≥3.296 ppm Au for core), the RC holes average 0.01 ozAu/t and core holes average 0.0095 ozAu/t, a difference of 0.0005 ozAu/t. In both cases the average grade for all core holes is slightly lower than the twinned RC holes by 2.7 percent and 5 percent, respectively. The correlation coefficient for the two data sets is 0.2705, indicating that short range correlation of the assay data is poor.

When compared on a grade zone basis, defined by the 0.01 ozAu/t cutoff grade and at comparable elevations, the RC holes averaged 0.0225 ozAu/t and the cores holes averaged 0.0232 ozAu/t, a difference of 0.007 ozAu/t (+3 percent higher in core). Individual grade zones in twinned RC holes, defined by a 0.01 ozAu/t cutoff grade, are well reproduced in the corresponding core holes. When the average grade and thickness of all intervals within the defined grade zones are compared between the twin core and RC, the difference is 0.0014 ozAu/t and less than one foot in thickness.

These analyses indicate that while individual assay intervals vary between the 2008 twinned RC holes and the 2010 core holes, the corresponding grade zones are well within the acceptable range of reproducibility. In addition, there is no evidence of down hole contamination in the twinned RC holes.

Table 10.4: Summary of assay results for the 2010 diamond drill holes.

Hole No.	Total Depth (ft)	UTM east	UTM north	Collar Elev. (ft)	From (ft)	To (ft)	Thickness (ft)	Grade ozAu/t
CC10-1	400	527480.02	4461750	5174.31	130	270	140	0.022
					350	365	15	0.011
CC10-2	400	527263.19	4461542	5189.43	20	55	35	0.010
					210	235	25	0.024
					285	290	5	0.071
CC10-3	500	527233.67	4461501	5200.03	20	140	120	0.023
					315	375	60	0.032
					390	395	5	0.037
					475	495	20	0.010
CC10-4	500	527271.35	4461476	5192.19	0	40	40	0.024
					60	170	110	0.022
					300	320	20	0.013
					365	380	15	0.044
CC10-5	450	527309.84	4461445	5185.79	45	150	105	0.029
including					65	70	5	0.223
					290	300	10	0.046
					325	340	15	0.017
CC10-6	400	527166.98	4461467	5228.34	0	35	35	0.011
					60	145	85	0.019
					175	320	145	0.021
CC10-7	400	527200.40	4461464	5227.10	5	15	10	0.021
					75	155	80	0.040
including					130	150	20	0.109
					190	195	5	0.031
					275	295	20	0.013
					325	330	5	0.040
CC10-8	500	527229.27	4461432	5227.19	95	175	80	0.019
					230	245	15	0.023
					290	305	15	0.021
					320	350	30	0.026
					390	405	15	0.022
					465	470	5	0.035
					490	495	5	0.11
CC10-9	500	527144.30	4461420	5260.98	50	145	95	0.014
					195	215	20	0.027
					270	285	15	0.029
					325	350	25	0.012
					370	385	15	0.015
					475	490	15	0.021
CC10-10	500	527202.24	4461311	5309.92	30	50	20	0.014
					80	115	35	0.012

					185	275	90	0.021
					315	325	10	0.042
CC10-12	450	525942.49	4462473	5501.46	130	165	35	0.022
					250	270	20	0.031
					285	345	60	0.024
CC10-13	450	525887.66	4462398	5486.56	0	45	45	0.045
including					0	15	15	0.106
					65	120	55	0.012
					245	255	10	0.019
					320	335	15	0.020
CC10-14	450	525916.60	4462428	5481.88	65	100	35	0.013
					145	235	90	0.019
					295	305	10	0.032
CC10-15	450	526004.09	4462409	5467.14	95	105	10	0.018
					150	180	30	0.019
					215	315	100	0.022
including					215	220	5	0.182
Total	**6350**							

Calculated using a 0.010 ozAu/t cuff grade

10.4 DRILLING PROCEDURES

The Robertson Property has had a long and varied drilling history during which time at least 1,238 drill holes totaling 512,737 ft have been completed. About 13 percent of the total footage drilled occurred prior to Coral's involvement in the property. As a result, much of the documentation regarding the early drilling programs is no longer available. What remains available is a digital compilation produced by Amax in the early 1990's, that includes collar coordinates in local mine and UTM grid systems, elevation in feet, assay interval, and gold assay value. The sources of this information were manual compilations by Coral and Amax taken from driller's logs, assay reports and assay certificates. In most cases no geologic logs were available.

The drilling methods employed by some early explorers are unknown (e.g. Aaron) or are not precisely known. It is known that the Superior and Placer Development drilling programs employed both conventional rotary drilling methods and "percussion" drilling methods. Conventional "open hole" rotary methods produce generally unreliable results because of chronic contamination problems and poor sampling technique. Except where twin RC or DDH have verified the original assays, much of the early results were not used in resource evaluations. Table 10.5 provides a summary of drilling in the district by companies known to have operated there since about 1968.

Table 10.5: Robertson Property Drilling Summary.

Company	Number of Holes	Drilling Method	Down Hole Survey	Collar Survey	Footage Drilled(ft)	Ave Hole Depth (ft)
Superior	92(?)	Conv. Rtry	no	yes	c.32,000	348
Placer Dev.	23(?)	Conv. Rrty	no	yes	c.3,200	150
Aaron	N/A	Percussion	no	some	<1,000	8-10
E & B Expl.	148	RC(?) "Percussion"	no	yes	30,800	200
Coral	380	RC	no	yes	105,877	278
	7	DDH	?	yes	3,500	500
Amax	338	RC	yes	yes	141,700	420
	62	DDH	yes	yes	34,300	553
Cortez	46	RC	yes	yes	54,000	1,174
	1	Mud Rtry	?	yes	3,000	
Coral Gold	125	RC	yes	yes	90,470	745
	10	DDH	no	yes	6,440	450
	2	Fldd RC	yes	yes	6,450	3,225
TOTAL	**1,238**				**512,737**	**398**

The E & B Exploration drilling is described in various reports by J. DeLeen (1980 and 1981) as "percussion", but it is believed to have been reverse circulation. Except for the Amax digital compilation, the only remaining record of the E & B drilling is a complete set of drill cuttings retained for each sample interval. In addition, approximately 20 chip boards have also been preserved. The cuttings are stored on site in 20-compartment plastic chip trays identified by hole number and footage. E & B routinely sampled the drill holes on even 5-ft intervals. Of the 148 "percussion" holes drilled by E & B, the majority of which were 200 ft or less in depth and all were drilled vertical.

In 1987, Coral developed a local north-south, east-west survey grid consisting of surveyed triangulation stations. All of the pre-Coral drill collars located in the field were subsequently tied to the local grid. Elevation control was determined from the USGS "Tenabo" bench mark (5,164.1 ft) situated at the east edge of the local survey grid.

10.4.1 Coral RC Drilling (1986-1989)

Documentation of the Coral RC and core drilling is incomplete. At the time of the Amax compilation (c.1991-92), the Coral drilling records were well organized and complete, and included driller's logs, sample interval, and assay report. Of the 380 RC holes drilled by Coral, the majority were less 250-ft-deep and fewer than 20 holes were inclined. Most holes had summary geologic logs and are on file in Coral's Crescent Valley office. The summary logs are included in the Amax digital compilation. A complete set of drill cuttings are retained for each RC sample interval and are stored on site in 20-compartment plastic chip trays identified by hole number and footage. Sampling was routinely done on even 5-ft-intervals over the length of the hole, including overburden. All drill collars were surveyed and tied to the local mine grid. No down-hole surveys were done on the Coral RC holes.

The majority of the Coral RC drilling was completed by two local drilling contractors: Eklund Drilling Company of Elko, NV and Rimrock Drilling also of Elko. The 7 core holes were drilled by Coates Drilling of Delta, B.C., Canada.

10.4.2 Amax RC Drilling

The methodology of the Amax RC drilling is well documented. Amax completed 342 RC holes, AT-1 through AT-341, totaling nearly 142,000 ft. Over 90 percent of the RC holes were drilled vertical to depths ranging from 400 ft to 2,830 ft (AT-3). Overburden depths varied from none to 60 ft. Where thin overburden was encountered, a minimum of 10 ft of steel surface casing was installed. In deeper overburden, 20 ft to 40 ft of steel casing was installed. Drill hole diameter averaged 5.5 inches and samples were collected on even 5-ft interval starting at the overburden-bedrock interface. Because of the low-angle orientation or flat-lying nature of most mineralized zones at Robertson, sample length (5 ft) is believed to represent true thickness.

Most RC holes drilled to depths of <1,000 ft, were completed using conventional down-the-hole hammer bits with conventional cross over assembly. Holes exceeding 1,000 ft feet were completed using standard tricone bits. All RC holes exceeding a depth of 400 ft had down-hole surveys using a multi-reading magnetic vertical deviation tool. Measured deflection in deeper hole ranged up to 105 ft, but averaged about 55 ft and is not considered serious. Surveys were conducted by Century Geophysics of Elko, NV. Prior drilling experience indicate that hole less than 400 ft deep had negligible deflection. Drill collars were surveyed and tied to the local grid system by registered surveyors from Desert Mountain Surveying of Winnemucca, NV.

Character samples for each 5-ft assay interval were collected from the reject port of the sample splitter and logged at the drill site. The drill cuttings are stored in 20-compartment plastic chip trays and stored on site. Geologic observations were recorded on standardized logging forms and included percentage oxidation, primary and secondary lithology, alteration and mineralization. The Amax drill logs are no longer in the possession of Coral.

The drilling contractor for 90 percent of the holes was Eklund Drilling Company of Elko, NV. The remaining holes were drilled by Lang Exploratory Drilling of Salt Lake City, UT and Becker Drills of Denver, CO. All of the contractors are well regarded and highly experienced.

10.4.3 Amax Diamond Drilling

Amax completed 62 DDH, CAT-1 through CAT-62, totaling over 34,000 ft. The principal drilling contractor was Longyear Drilling of Dayton, NV, a well-respected drilling company, who generally preformed above industry standard. The initial four core holes were completed by Tonto Drilling Company of Spokane, WA. At that time, their performance on the project was considered substandard due to their poor core recovery efforts.

Of the 62 DDH, 56 were drilled using HQ-diameter (2.5 inches) diamond impregnated bits, 5-ft core barrels and wire-line method. Six holes were inclined and nine were drilled using PQ-diameter (3.33 inches) tools in order to collect samples for metallurgical testing (column leach tests). Core orientation studies were attempted in four of the HQ core holes with only limited success. All hole collars were surveyed and tied to the local grid coordinate system. Down-the-hole surveys were completed on 90 percent of the core holes using a standard single shot camera survey tool. Shots were taken at regular 100 ft intervals.

Core was retrieved from the drill site in standard waxed cardboard core boxes (5 ft/box). Wooden blocks indicating footage were placed at the end of each core run by the driller. Prior to logging, the core was measured and marked into even 5-ft or less sample intervals. The percentage of core recovery was determined and RQD data collected and recorded on a standardized form. The core was cleaned of drilling mud and grease, and photographed. Detailed geologic observations of the core were made using a standardized logging form. Key observations included percent oxidation, primary and secondary lithology, alteration, mineralization and structure. In addition, 15 secondary fields were also visually estimated. Small lengths of representative core were collected at regular intervals over the entire length of the holes as a visual record. The "skeleton" core is stored on site in standard cardboard core boxes identified with hole number and footage interval. Individual core is identified by hole number and footage in permanent marker.

10.4.4 Cortez RC Drilling

The Cortez drilling consisted of 46 RC holes totaling 54,000 ft and one "mud rotary" hole drilled to a depth of 3,000 ft. The RC drilling contractor was Eklund Drilling Company of Elko, NV, a well-respected and experienced drilling company. The mud rotary hole was completed by the Lang Division of Boart Longyear of Salt Lake City, UT, a well-respected and experienced drilling company.

Of the 46 RC holes 34 were drilled to a depth of between 1,000 ft to 2,000 ft and 12 were drilled to a depth of 500 ft to 900 ft. Twelve holes were inclined from 52° to 62°. All of the RC holes had down-hole surveys using a multi-reading magnetic vertical deviation tool. The surveys were conducted by Silver State Surveys, Inc., of Elko, NV. None of the measured deflection is considered serious.

Cortez routinely sampled the RC holes at 10-ft intervals over the entire length of the hole. Because of the low-angle orientation or flat-lying nature of most mineralized zones at Robertson, sample length (10 ft) is believed to represent true thickness. No geologic logging was completed and no other details, except assay results were provided. Character samples were collected for each 10-ft assay interval in 20-compartment plastic chip trays which are stored off-site.

10.4.5 Coral RC Drilling (2004-2006, 2008 and 2010)

The methodologies employed during the period 2004 through 2010 for the various Coral RC drilling programs are well documented. During that time period Coral completed 125 RC holes totaling 90,470 ft. A summary of the Coral RC drilling from 2004 through 2010 is presented in Table 10.6. Depths of the holes varied from 345 ft to 1,500 ft. Overburden depths ranged from none to 125 ft. A total of 122 of the RC holes were drilled vertical to depths ranging from 345 ft to 1,500 ft. Three holes were inclined at angles of between 60° and 70° and drilled to depths of 600 ft to 725 ft. Where thin overburden was encountered, a minimum of 10 ft of steel surface casing was installed. In deeper overburden, 20 ft to 40 ft of steel casing was installed. Drill hole diameter averaged 5.5 inches and samples were collected on even 5-ft interval starting at the overburden-bedrock interface. Because of the low-angle orientation or flat-lying nature of most mineralized zones at Robertson, sample length (5 ft) is believed to represent true thickness.

The drilling was conducted using truck-mounted TH-75 Ingersoll Rand and Drill Tech D40K drills with 900 cfm/350 psi compressors. These drills employ air-cyclone sample collection equipment that meet current industry standard. Auxiliary air compressors were employed when extremely hard rock conditions were encountered. Standard care was taken to minimize contamination during all parts of sampling procedure. In addition, a geologist was at the drill site during all phases of the drilling and sampling procedure.

Most RC holes drilled to depths of <1,000 ft, were completed using conventional down-the-hole hammer bits with conventional cross over assembly. Holes exceeding 1,000 ft feet were completed using standard tricone bits.
All RC holes exceeding a depth of 600 ft had down-hole surveys using a multi-reading gyroscopic vertical deviation tool. Measured horizontal deflection at the bottom of holes with total depth between 600 ft and 1,000 ft ranged from 9 ft to 24.5 ft on azimuths varying from nearly due north to east-southeast. Within ore-grade intersections, deflection varied from 7 ft to 21.5 ft along similar azimuths. The measured deflections are considered negligible. Measured horizontal deflections in deeper holes (>1,000 ft) ranged up to 67 ft, but averaged about 22 ft and is not considered serious. Surveys for holes drilled in 2004 were conducted by Wellbore Navigation, Inc of Elko, NV. Surveys of holes drilled in 2005 and later were done by International Directional Services of Elko, NV. Prior drilling experience indicated that holes less than 600 ft deep had negligible deflection. Drill collars were surveyed and tied to the UTM and local grid systems by registered surveyors from Summit Engineering of Elko, NV.

Character samples for each 5-ft assay interval were collected from the reject port of the sample splitter and logged at the drill site. Drill cuttings are stored in 20-compartment plastic chip trays and stored on site. Geologic observations were recorded on standardized logging forms and included percentage oxidation, primary and secondary lithology, alteration and mineralization.

The drilling contractor for the 2004-2010 drilling programs was Lang Exploratory Drilling, a division of Boart Longyear, Inc. of Elko, NV. Lang is a well-regarded and highly experienced RC drilling company.

Table 10.6: Summary of Coral RC drilling from 2004 through 2010.

Year Drilled	Hole No.	Holes Drilled	Hole Depth (ft)	Total Footage	Drilling Contractor	Down Hole Survey	Collar Survey
2004	CR04-1 to -20	20	485-850	13,600	Lang	>600ft	UTM
2005	CR05-1 to -12	12	500-1,200	10,420	Lang	>600ft	UTM
2006	CR06-2 to -48A	48	445-1,500	35,615	Lang	>600ft	UTM
2007	TV07-1, -2	2	3,450	6,440	Lang	>600ft	UTM
2008	CR08-1, -3 to -6 CR08-10 to -11 CR08-13 to -38	33	345-1,200	22,835	Lang	>600	UTM
2010	CR10-1 to -12	12	600-1,000	8,000	Lang	>600	UTM
Total		**127**		**96,910**			

10.4.6 Coral Flooded RC Drilling (2007)

The 2007 deep drilling was conducted using an LM 120 drill capable of reaching depths exceeding 3,000 ft and which employs a flooded RC method. This method uses conventional RC drilling starting at the surface with a 9 inch hammer bit and conventional crossover assembly. The conventional RC drilling is advanced as deep as possible, typically 600-900 ft deep, in order to reach at least 250 ft below the local water table. Once the hammer bit has reached its maximum depth, the hole is converted to flooded RC. This method employs a 9 inch tricone bit followed by a series of stabilizers, several hundred feet of drill collars (for added weight), a series of crossover assemblies and finally conventional 20-ft-long dual-walled RC drill rods. Drilling fluid is introduced into the hole through the casing between the drill rods and the wall of the hole. A portion of the fluid reaches the drill bit and working face while the rest is captured by the crossover assembly. At the same time high pressure compressed air is introduced through the outer annulus of the dual-walled RC rods and mixes with the drilling fluid that enters through the crossovers, creating a highly buoyant air-charged fluid that rises at high velocity (>1,000 ft/sec) through the center tube of the RC rods to the surface. This rapid upward movement creates a significant vacuum at the bit face where the sample, entrained in the fluid, also passes rapidly upward through holes in the bit face, stabilizers and collars, mixes with the buoyant air-charged fluid and ascends at high velocity to the surface. The sample and fluid are discharged into a cyclone where, once their velocity is slowed, they empty on to a series of stacked vibrating screens. At this point the drilling fluid and some very fine sample are separated from the bulk sample and sent to a de-sander where the fine rock material is available for sampling and discharged into the mud sump. The drilling fluid is returned to a holding tank to be reused. The entire sample interval is discharged into a Gilson-type sample splitter directly from the vibrating screens.

Down-the-hole directional surveys were conducted on both holes by International Directional Services, Inc. of Elko, Nevada, using a gyroscopic directional survey tool that measures horizontal and vertical deflection in the hole. TV07-1 had a total deviation of 500 ft in a northeasterly direction at the bottom of the hole. TV07-2 had a total deviation of 118 ft in a due east direction at 3,200 ft.

The drilling contractor for this program was Lang Exploratory Drilling, a division of Boart Longyear, Inc., of Elko, Nevada. Lang is a highly experienced drilling company specializing in reverse circulation and is an industry leader in deep exploration. Lang's performance on this project was excellent and met current industry standard.

10.4.7 Coral Diamond Drilling (2010)

In 2010, Coral completed 14 vertical, HQ diameter diamond core holes, CC10-1 through CC10-15 (CC10-11 abandon) totaling 6,450 ft. The drilling contractor for this program was Major Drilling Inc. of Salt Lake City, Utah. Major Drilling is a highly experienced core drilling company. Their performance on this project was plagued by inexperienced drilling crews, equipment breakdowns, crew absenteeism and generally poor core recovery in critical mineralized areas.

All 14 core holes were drilled using a truck mounted LF-70 diamond core drill using conventional HQ-diameter (2.5 inch core) diamond impregnated bits, 5-ft core barrels and standard wire-line method. Overburden depth ranged from none to 30 ft. A minimum of 10 ft of casing was installed in all holes. Hole CC10-1, which was collared on the gravel-covered South Porphyry Zone, employed 35 ft of casing. Hole depths varied from 400 ft to 500 ft. Drill collars were surveyed by Summit Engineering of Elko, Nevada. No down-the-hole surveys were conducted.

Core was retrieved from the drill site in standard waxed cardboard core boxes (10 ft/box). All core boxes were marked with hole number, footage interval and box number. Wooden blocks indicating footage were placed at the end of each core run by the driller. Prior to logging, the core was measured and marked into even 5-ft or less sample intervals. The percentage of core recovery was determined and RQD data collected and recorded on a standardized form. The core was cleaned of drilling mud and grease, and photographed. Detailed geologic observations of the core were made using a standardized logging form. Key observations included percent oxidation, primary and secondary lithology, alteration, mineralization and structure. In addition, 15 secondary fields were also visually estimated. Small lengths (3-6 inches) of representative core were collected at regular intervals over the entire length of the holes for specific gravity measurements and as a visual record. A total of 279 samples were collected and measured for specific gravity. The "skeleton" core is stored on site in standard cardboard core boxes identified with hole number and footage interval. Individual core is identified by hole number and footage in permanent marker.

Because the core was to be used for metallurgical testing, it was not split or sawed prior to bagging in pre-numbered heavy 17 in x 28 in canvas bags.

Core recovery for individual 5-ft assay intervals varied 0 to 100 percent. However, core recovery in the top 10 ft to 30 ft of most holes and within certain mineralized intervals was considered to be poor (<50 percent recovery). Overall core recovery for entire holes ranged from 87.4 to 97.5 percent and averaged 93.4 percent.

SECTION 11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 INTRODUCTION

The authors have no first-hand knowledge of the sampling procedures employed during the exploratory drilling programs prior to Coral (pre-1986) (85 holes), as well as the procedures used by Coral (1986-1989) (403 holes). In addition, no written documentation or description of the sampling procedures employed by these companies was found in the Coral files. It cannot be assumed that the sampling procedures used by these companies would meet industry standard of today. However, the authors also have no reason to believe the procedures used in the past were, in any way, suspect or unprofessional. For the purpose of the inferred resource estimate, all holes were utilized due to confidence factors. These holes are distributed throughout the Gold Pan, Porphyry, Altenburg Hill and in particular the Triplet Gulch areas. Further validation and verification is required in order to utilize these holes in an upgraded resource however the grade distribution with and without these holes is within acceptable limits. Therefore, the inclusion or exclusion of these holes will not adversely affect the resource with the exception of Triplet Gulch were a much more rigorous validation program is required.

Additionally, the authors have no first-hand knowledge of the sampling procedures employed by Cortez (1999). No written documentation or verbal description of the sampling procedures used at the Robertson Property was provided by Cortez. However, it can be assumed that Cortez employed the same or similar sampling protocol on this project as is used at their nearby mining operation and that it meets current industry standard.

The sampling procedures employed by Amax and Coral Gold during their RC and the core drilling programs are well documented and, as Senior Project Geologist for Amax and as the supervising Qualified Person for Coral at the Robertson Property, the author has first-hand knowledge of their sampling methodologies. The sampling protocols used by Amax and Coral Gold on this project meet current industry standards.

11.2 AMAX RC AND CORE DRILLING

A summary of sampling methods employed by Amax during RC and core drilling operations at Robertson were previously reported in the NI 43-101 compliant technical report *Mineral Resource Estimate for the Robertson Property, Lander County, Nevada, USA* (Stokes et al., 2008) available on SEDAR.

11.3 CORAL GOLD RESOURCES RC SAMPLING (2004-2006)

A summary of sampling methods employed by Coral during their 2004-2006 RC drilling operations at Robertson were previously reported in the NI 43-101 compliant technical report *Mineral Resource Estimate for the Robertson Property, Lander County, Nevada, USA* (Stokes et al., 2008) available on SEDAR.

11.4 CORAL GOLD RESOURCES RC SAMPLING (2007)

The 2007 deep drilling was conducted using an LM 120 drill which employs a flooded RC method. This method uses conventional RC drilling starting at the surface until the hole is advanced as deep as possible, typically 600-900 ft deep. Once the hammer bit has reached its maximum depth, the hole is converted to flooded RC.

Since the two deep holes were drilled from the surface using conventional RC methods, samples were initially collected using an air-cyclone collection system. From the cyclone, samples were discharged into a radial rotating wet splitter. The split assay sample is collected in 5 gallon plastic buckets. Samples were decanted of excess water and transferred to pre-numbered 11 x 17 inch micro-pore sample bags. The reject material from the splitter is discharged into the mud sump. The sample collection buckets were cleaned after every sample using high pressure water hose. The splitter was routinely cleaned using a high pressure hose at every 20 ft rod change.

While using flooded RC, samples were collected in a similar air-cyclone collection system that discharged on to a series of stacked vibrating screens. Samples are discharged directly from the vibrating screens into a Gilson-type sample splitter. The split assay sample was collected in a 2.5 gallon pan. The split assay sample is transferred to a pre-numbered 11 x 17 inch micro-pore sample bag. The reject material from the splitter is discharged into the mud sump. The vibrating screens, sample splitter and sample collection pans are cleaned using a high pressure water hose after every sample.

Samples for assay were collected on even 10 ft intervals starting at the surface. During sampling, pre-numbered "rig duplicate" samples were routinely collected from the reject outlet of the splitter. These samples were used to assess the effectiveness of the sampling procedure and the natural distribution of gold in the rock by comparing them with the "original" sample.

Once the samples were dry, all drill samples were weighed at the lab. The average dry weight for a 10-ft sample was about 6.7 kg. Together with observations at the drill site, monitoring sample weights provided data from which sample recovery could be estimated.

11.5 CORAL GOLD RESOURCES RC SAMPLING (2008, 2010)

The 2008 and 2010 drilling programs were conducted using a truck-mounted Drill Tech D40K drill with 900 cfm/350 psi compressor. These drills employ air-cyclone sample collection equipment that meets current industry standard. Standard care was taken to minimize contamination during all phases of the sampling procedure. In addition, a geologist was at the drill site during the drilling and sampling procedure.

All drill holes were drilled wet beginning at the surface. Under these conditions, a small amount of water is injected during the drilling process until the water table is encountered, which varied from 220-ft- to 475-ft-deep. Samples for gold assay were collected on even 5-ft intervals starting at the surface. During wet drilling conditions, drill cuttings were spit on site using a radial rotating wet splitter. The purpose of splitting the sample was to obtain a consistent and representative sample weighing 3-4 kg. When drilling wet, samples were collected using a series of 5 gallon plastic buckets. The samples were decanted of excess water and transferred to pre-numbered heavy 10 inch x 17 inch sample bags. The sample collection buckets were cleaned using a high pressure water hose after each sample. The wet splitter was routinely cleaned with a high pressure water hose at every 20 ft rod change. In zones of high water flow, the 5-gallon sample buckets filled and overflowed with water and fine-grained material. These "fines" were not captured. Previous sampling studies of this material indicated that the gold content was negligible in the overflow material.

Character samples for each 5-ft assay interval were collected from the reject port of the sample splitter and logged at the drill site. The drill cuttings are stored in 20-compartment plastic chip trays that are stored on site. Geologic observations were recorded on standardized logging forms and included percentage oxidation, primary and secondary lithology, alteration and mineralization.

During RC drilling, "rig duplicate" samples were routinely collected from the reject outlet of the splitter. These samples were used to assess the effectiveness of the sampling procedure and were compared with the "original" sample. Additionally, specially collected coarse "blank" (<0.005 ppm) and very low-level material (<0.025 ppm Au) were inserted into the sample sequence at pre-determined intervals to monitor possible lab contamination such as smearing of coarse gold on sample preparation equipment.

Once the samples were dry, all RC samples were weighed at the lab. The average dry weight for a 5-ft sample was about 4.0 kg. Together with observations at the drill site, monitoring sample weights provided data from which sample recovery could be estimated.

11.6 CORAL GOLD RESOURCES DRILL CORE SAMPLING (2010)

Drill core was delivered to Coral's logging facility directly from the drill site at the end of each shift in core boxes containing approximately 10 ft of 2.5 inch diameter core. Each box is marked with hole number, interval footage and box number. Wooden core blocks are inserted at the end of each drill run with the measured depth marked on the block. At the logging facility the core is measured and core recovery was determined for each 5-ft assay interval. Individual assay intervals were marked by wooden blocks. Core recovery averaged 93.4 percent. After cleaning, photographing and detailed logging, the entire unsplit, 5-ft assay interval was placed in a pre-labeled heavy canvas bag. Each bag was labeled with hole number and a unique sample number. The samples remained in the logging facility until they were picked by the assay lab.

A total of 279 representative core samples (3-6 inches long) were collected and measured for specific gravity, which are stored on site.

Specially collected coarse barren material were inserted into the sample sequence at pre-determined intervals to monitor possible lab contamination and smearing of coarse gold on sample prep equipment.

11.7 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.7.1 Introduction

The authors have no first-hand knowledge of the sample preparation protocols, analytical methodology and sample security that were employed at the Robertson Property by companies prior to Coral (pre-1986), as well as the procedures used by Coral (1986-1989). No written documentation or description of these procedures was found in the Coral files. Further, it cannot be assumed that the sample preparation, analytical procedures and sample security measures used by the early companies would meet industry standard of today. However, the authors also have no reason to believe the procedures used in the past were, in any way, suspect or unprofessional. For the purpose of the inferred resource estimate, all holes were utilized due to confidence factors. These holes are distributed throughout the Gold Pan, Porphyry, Altenburg Hill and in particular the Triplet Gulch areas. Further validation and verification is required in order to utilize these holes in an upgraded resource however the grade distribution with and without these holes is within acceptable limits. Therefore, the inclusion or exclusion of these holes will not adversely affect the resource with the exception of Triplet Gulch were a much more rigorous validation program is required.

The authors have no first-hand knowledge of the sample preparation protocols, analytical procedures or sample security measures employed by Cortez (1999) during their evaluation of the Robertson

Property. It should also be noted that no written or oral description of these procedures was provided by Cortez. However, it may be assumed that the sample preparation, analytical procedures and sample security used by Cortez on this project would meet industry standard of today.

The sample preparation protocol, analytical procedures and sample security measures employed by Amax (1990-1996) and Coral Gold (2004-2010) during their RC and core drilling programs are well documented and, as Senior Project Geologist for Amax and as the supervising Qualified Person for Coral Gold at the Robertson Property, the author also has first-hand knowledge of these methodologies. The sample preparation protocols, analytical procedures and sample security measures used by Amax and Coral Gold on this project meet current industry standard practices.

11.7.2 Amax Drill Sample Security Measures

All RC and core samples remained on the project site until they were picked up by the analytical lab. No sample preparation occurred on-site. Access to the samples prior to pick up by the lab, by non-Amax or Coral personnel was controlled during both daylight and night-time hours. While it is the opinion of the author that no tampering with samples occurred during this period and, within the context of the entire drilling program, large-scale tampering would have been impossible to achieve.

11.7.3 Analyses of Amax Drill Samples

During the period 1990-91 and 1993-94, Amax employed Monitor Geochemical Laboratory, with sample prep and analytical facilities in Elko, NV, as its principal analytical lab for the Robertson Property. In 1992, the principal laboratory was Bondar-Clegg, with sample prep facilities in Reno, NV and analytical laboratories in Vancouver, B.C. In 1993-94, during the Porphyry Zone evaluation, Chemex Labs, Inc. was chosen as the secondary (umpire) lab and used primarily to verify sample prep and check assaying.

The Amax sample preparation protocol was used at the Monitor, Bondar-Clegg and Chemex sample prep facilities and is summarized below:

- Entire drill sample was dried in the sample bag and weighed;
- Entire 25-35 pound (11-16 kilograms) sample jaw-crushed to 95% -10 mesh;
- Crushed material reduced to 6 kilograms (13 pounds) with Jones-type splitter;
- Reject material saved and stored at lab;
- Entire 6 kilogram sub sample disk-pulverized to 95% -80 mesh;
- Pulverized material reduced to 350 gram with Jones-type splitter
- 350 gram sub sample passed through rotary mill to 95% -200 mesh;
- nominal 350 gram pulp sample sent for analysis;
- Jaw crusher, rotary mill and pulverizer were cleaned by passing barren gravel and sand between samples; Jones splitter cleaned by compressed air.

The gold assay procedure used at both Monitor and Bondar-Clegg facilities is summarized below:

- Re-homogenization of the 350 gram pulp;
- One-assay ton (nominal 29.1 grams) weighed from 350 gram pulp;
- One-assay ton fire assay followed by bead digestion in aqua regia and AA determination;
- All values exceeding 10 ppm gold are re-assayed using one-assay ton with gravimetric finish.

11.7.4 Metallic Screen Analyses

During the Porphyry Zone evaluation, concern over the presence of "coarse" gold prompted Amax to initiate "screen fire" analysis, considered the most accurate assay method for assessing coarse gold. The evaluation consisted of 132 screen fire assays (about 1 percent of all sample in the Porphyry Zone). The screen fire assay procedure employed at Monitor is summarized below:

- Split a 1 kilogram (2.2 pounds) sub sample from the –10 mesh reject material;
- 1 kilogram sub sample pulverized in ring and puck mill to nominal –150 mesh;
- Pulverized material weighed (weight recorded);
- Entire sample wet sieved through 150 mesh screen;
- Entire +150 mesh (oversize) collected, dried, weighed and fire assayed;
- Entire –150 mesh (undersize) collected, dried and a 2-assay ton split (58.3 grams) fire assayed;
- Assay results of the two size fractions are combined by weighted averaging to determine the reported assay.

11.7.5 Cyanide-Soluble Gold and Copper Analyses

During evaluation of the Porphyry Zone resource, cyanide soluble gold and copper analyses were routinely performed on all samples with FA/AA gold values $\geq$ 0.01 ozAu/t (over 2,100 samples). The cyanide soluble analytical procedure consisted of leaching 10 grams of –200 mesh pulp material in a flask for 24 hours in a 100 ml solution of one gram/liter NaCN solution. The leach solution was analyzed for gold and copper by atomic absorption spectrometry.

11.7.6 Use of Standards and Blanks

In order to assess the Monitor and Bondar-Clegg sample prep and analytical performance, Amax developed a QA/QC program consisting of submitting "blind" coarse blank material (3.5 %), rig duplicates (4 %) and Amax prepared standard pulps (9.3 %) into the sample stream. All reference materials were pre-numbered in sequence with the drill samples and submitted blind to the labs without unique identifiers. Standard pulps were chosen to match as closely as possible the oxidation state, expected grade and sulfide content of the accompanying drill samples.

Over 900 samples, representing coarse reject material and pulps, were randomly selected for sample preparation and gold assay determinations by Chemex. These materials were shipped by the primary lab. Sample prep and analytical procedure used by the secondary lab were same as those used by the primary lab facilities.

11.7.7 Coral Gold RC Drill Sample Security Measures (2004-2010)

All RC samples remained on the project site until they were picked up by the analytical lab. No sample preparation occurred on-site. No Coral personnel were involved in any aspect of sample preparation. Access to the samples prior to pick up by the lab by non- Coral personnel was controlled during daylight hours only. While it is the opinion of the author that no tampering with samples occurred during this period and, within the context of the entire drilling program, large-scale tampering would have been impossible to achieve.

11.7.8 Analyses of Coral Gold RC Drill Samples (2004-2010)

During the period 2004-2010, Coral employed ALS Chemex with sample prep facilities in Elko, NV and analytical laboratories in Reno, NV to perform sample preparation and gold assays. ALS Chemex laboratories in North America are registered to ISO 9001:2000 by QMI Quality Registrars.

The sample preparation protocol used at the ALS Chemex sample prep facilities and is summarized below:

- Entire drill sample was dried in the sample bag and weighed;
- Entire 7-15 pound (3.2-6.7 kilograms) sample jaw-crushed to 70% -2mm;
- Crushed material reduced to 1 kilogram (2.2 pounds) with riffle-type splitter;
- Reject material saved and stored at lab;
- Entire 1 kilogram sub sample disk-pulverized to 85% -200 mesh;
- Pulverized material reduced to 250 gram with riffle-type splitter;
- nominal 250 gram pulp sample sent for analysis;
- Jaw crusher and pulverizer were cleaned after every sample by passing barren gravel and sand.

The gold assay procedure used at ALS Chemex facilities is summarized below:

- Re-homogenization of the 250 gram pulp;
- One-assay ton (nominal 29.1 grams) weighed from 250 gram pulp;
- One-assay ton fire assay followed by bead digestion in aqua regia and AA determination;
- All values exceeding 10 ppm gold are re-assayed using one-assay ton with gravimetric finish.

11.7.8.1 Use of Standards and Blanks

In order to assess the ALS Chemex sample prep and analytical performance, Coral developed a QA/QC program consisting of submitting “blind” coarse blank and low-level gold material (4.3 %), rig duplicates (2.1 %) and standard reference pulps (1.2 %) into the sample stream. All reference materials were pre-numbered in sequence with the drill samples and submitted blind to the labs without unique identifiers. Standard pulps were chosen to match as closely as possible the oxidation state, expected grade and sulfide content of the accompanying drill samples.

In 2004-2005, over 146 samples, representing coarse reject material and pulps, were selected for additional sample preparation and gold assay determinations by ALS Chemex. In 2006, a total of 604 samples, representing coarse reject material, were selected for additional sample preparation, compositing and gold assay determinations by McClelland Labs and Rocky Mountain Geochemical, both located in Sparks, NV, USA.

The sample preparation protocol, analytical procedures and sample security measures employed by Coral Gold during its 2004-2010 RC drilling programs meet current industry standard practices. As the supervising Qualified Person for Coral Gold at the Robertson Property, the author also has first-hand knowledge of these methodologies.

11.7.9 Coral Gold Core Drilling Sample Security Measures (2010)

At the end of each drill shift, core samples were delivered to a secure Coral storage facility in Crescent Valley, Nevada. No sample preparation occurred on-site. No Coral personnel were involved in any aspect of sample preparation. Access to the samples prior to delivery to Coral personnel was controlled at all time during drilling operations (24 hrs). While it is the opinion of the author that no tampering with samples occurred during this period and, within the context of the entire drilling program, large-scale tampering would have been impossible to achieve.

11.7.10 Analyses of Coral Gold Core Drilling Samples (2010)

During the 2010 drilling program, Coral employed ALS Chemex with sample prep facilities in Elko, NV and analytical laboratories in Reno, NV to perform sample preparation and gold assays. ALS Chemex laboratories in North America are registered to ISO 9001:2000 by QMI Quality Registrars.

Because one of the objectives of the 2010 core drilling was to supply samples for metallurgical testing, a special sample preparation was designed to provide nominal 3/4 inch crushed material for column leach tests. The modified sample protocol used during the core drilling is summarized below:

- Entire drill sample was dried in the sample bag and weighed;
- Entire 24.2 pound (average sample weight 11.0 kilograms) sample jaw-crushed to -3/4 inch;
- 1 kilogram (2.2 pounds) split from -3/4 inch crushed sample with rotary-type splitter;
- Entire -3/4 inch coarse reject material saved and stored at the lab;
- Entire 1 kilogram sub sample disk-pulverized to 85% -200 mesh;
- Pulverized material reduced to 250 gram with riffle-type splitter;
- nominal 250 gram pulp sample sent for analysis;
- Jaw crusher and pulverizer were cleaned after every sample by passing barren gravel and sand.
-

The gold assay procedure used at ALS Chemex facilities is summarized below:

- Re-homogenization of the 250 gram pulp;
- One-assay ton (nominal 29.1 grams) split and weighed from 250 gram pulp;
- One-assay ton fire assay followed by bead digestion in aqua regia and AA determination;
- All values exceeding 10 ppm gold are re-assayed using one-assay ton with gravimetric finish.

11.7.10.1 Use of Standards and Blanks

In order to assess the ALS Chemex sample prep and analytical performance, Coral developed a QA/QC program consisting of submitting “blind” coarse blank and low-level gold material (5.3 %) and standard reference pulps (3.7 %) into the sample stream. All reference materials were pre-numbered in sequence with the drill samples and submitted blind to the labs without unique identifiers. Standard pulps were chosen to match as closely as possible the oxidation state, expected grade and sulfide content of the accompanying drill samples.

The sample preparation protocol, analytical procedures and sample security measures employed by Coral Gold during its 2010 core drilling program meet current industry standard practices. As the supervising Qualified Person for Coral Gold at the Robertson Property, the author also has first-hand knowledge of these methodologies.

SECTION 12.0 DATA VERIFICATION

12.1 INTRODUCTION

Verifying data collected by companies prior to Coral Resources (1986-1989) is nearly impossible due to a lack of documentation. However, it is possible to make a judgment regarding the E & B Exploration and portions of the Coral Resources databases. While details regarding drilling, sampling, sample prep and assay procedures are unavailable, the fact that both companies employed reputable assay labs for their analytical work suggests that the assays data is of good quality. An exception is the assay data of Coral Resources reported during a portion of 1988 and all of 1989, when Coral prepared and analyzed their own samples on site. The Coral assays produced during this period lack documentation and have been shown to be unreliable and should be excluded from use in resource estimates.

Data verification was undertaken by Amax during evaluation of the Porphyry Zone resource. MRDI (Mineral Resource Development, Inc., of San Mateo, CA) was responsible for verification of Amax's methodology used to verify the quality of analytical data used in their evaluation at the Robertson Property (1994 Amax feasibility study). It was the opinion of MRDI that the methodology used is appropriate in the context of the evaluation being considered and that the work performed by Amax meets or exceeds industry standards.

12.2 DATA VERIFICATION OF AMAX DRILL SAMPLE RESULTS

The Amax primary labs were the Monitor facility in Elko, NV and the Bondar-Clegg facility in Vancouver, B. C. The Chemex Labs of Vancouver, B. C. was chosen as the secondary (umpire) lab facility.

The primary labs prep facilities were responsible for preparing course "rig" duplicate samples, coarse blank samples and for inserting the Amax supplied standard reference materials into the sample stream. The standard reference material was prepared by Amax as part of an internal, company-wide program that supplied reference material for all Amax sampling programs. The standards are highly reliable and were routinely monitored by Amax. Once the drill samples were reduced to pulp samples ready for assay, the Amax standards (pulps) were inserted by the lab in pre-numbered sequence with the other pulps and assayed sequentially.

In addition, Amax was supplied with and monitored the analytical results of the primary and secondary labs own internal QA/QC controls for sample batches.

Amax routinely (6% of all samples) submitted pulp samples prepared by the primary lab to the umpire lab for gold assay. Less frequently, coarse (-10 mesh) reject material was submitted to the umpire lab for preparation and analysis. The primary lab was responsible for shipping all samples to the umpire lab.

A complete summary of the verification of analytical data developed by Amax during their evaluation of the Porphyry Zone inferred mineral resource (1994 feasibility study) was previously reported in the NI 43-101 compliant technical report *Mineral Resource Estimate for the Robertson Property, Lander County, Nevada, USA* (Stokes et al., 2008) available on SEDAR.

12.3 CORAL GOLD DATA VERIFICATION OF RC DRILL SAMPLE RESULTS (2004-2006)

The Coral primary lab was ALS Chemex with sample prep facilities in Elko, NV and assay facilities in Reno, NV. McClelland Metallurgical Labs, Sparks, NV was used as the secondary (check) lab.

The ALS Chemex prep facilities was responsible for preparing coarse "rig" duplicate samples, coarse blank samples, coarse low-level gold samples, and for inserting the Coral supplied standard reference materials into the sample stream. The commercially available certified reference materials were supplied by CANMET and MEG. The standards are highly reliable and were routinely monitored by Coral. Once the drill samples were reduced to pulp samples ready for assay, the certified reference pulps were inserted by the lab in pre-numbered sequence with the other pulps and assayed sequentially.

In addition, Coral was supplied with and monitored the analytical results of ALS Chemex own internal QA/QC controls for sample batches.

The following statistical analysis were calculated by substituting the assay value <0.005 ppm Au (below analytical detection) with the value 0.004 ppm Au.

12.3.1 Summary of Verification-Drill Sample Results

To assess the ability of the primary and umpire labs to provide consistently accurate analytical results, their performance was monitored using standards, blanks and rig duplicate samples. A total of 11,918 drill samples were submitted to ALS Chemex for initial gold determination from 2004-2006. In addition Coral submitted 1,727 control (14.5%) samples to monitor lab performance.

12.3.2 Certified Reference Pulps

Analytical results for the 178 (1.5 % of all samples) certified reference pulps submitted to ALS Chemex indicate that the average for all "certified" values was 2.535 ppm Au, versus the primary labs average value of 2.530 ppm Au. This is a relative difference of 0.005 ppm Au, or <1 percent lower than the mean of the certified values. A correlation coefficient of 0.973 was calculated for the pair. These results indicate that relative bias between the two data sets is absent. However, it should be noted that assay results of certified blank samples by ALS Chemex returned gold values ranging from <0.005 ppm Au up to 0.085 ppm Au. Of the 19 certified blank samples submitted, six samples returned values >0.005 ppm Au. This suggests a contamination or precision problem at the assay lab

12.3.3 Check Assays

Check assays on the 2004-2005 pulps prepared by the primary lab were performed on 207 (1.7% of all samples) samples. The average value from the initial assay was 1.544 ppm Au versus the average value from the check assay of 1.544, essentially no difference between the mean of both values. However on a sample by sample comparison, results between the two data sets (average values) varied from 0.027 ppm to 0.116 ppm. A correlation coefficient of 0.998 was calculated for the data pairs. These results indicate very good analytical reproducibility by the primary lab.

Check assays on the 2006 assays were performed on 604 (5.1% of all samples) coarse reject samples which were made into 25 composite samples for metallurgical testing. The average value from the initial assay of the 604 samples was 1.94 ppm Au versus the average value from check assays which was 1.65 ppm Au, a difference of 0.29 ppm (14.9%). A correlation coefficient of 0.9802 was calculated for the data pairs. A comparison of the initial (ALS) assay versus the check assay (RMG) results indicates that on average the initial assays are higher than the corresponding check samples. On a sample by sample comparison, results between the two data sets (average values) varied from 0.10 ppm to 0.84 ppm. These results suggest that either the initial determinations are biased high or the check sample assays are biased low. In addition problems with analytical reproducibility (precision) or coarse gold may also be present. To assess these issues, assay results from both labs were compared with the head grades calculated from 2,000 g cyanide leach tests. The average initial assay value for the individual 25 composite samples were compared with the average grade for the calculated head grade and plotted on a scatter diagram shown in Figure 12-1. With the exception of a single sample pair, the data are in excellent agreement and show no systematic bias. The average for the initial assays is 1.94 ppm versus 1.92 ppm for the calculated head grade, a difference of 0.02 ppm or 1.03% of the average initial assay value. However, a comparison of the average check assay value for each of the 25 composite samples with the calculated head grade, shown in Figure 12-2, indicates a systematic low bias for the check assays versus the calculated head grade. The average value for all check assays is 1.65 ppm versus 1.92 ppm for the calculated heads, a difference of 0.27 ppm or 16.4% of the average check assay value. These comparisons support the conclusion that no apparent bias exists between the initial assays performed by the primary lab and the calculated head grades. They also indicate that the check assays performed by a secondary lab are on average lower than either the initial assay or the calculated head grade. A correlation coefficient of 0.9658 was calculated for the initial assays versus the calculated head grade and 0.9389 was calculated for the check assays versus the calculated head grade. These correlations also suggest bias between the check assays and the calculated head grades.

12.3.4 Rig Duplicate Samples

The “rig” duplicate samples taken from reject material at the drill rig, and submitted blind to the lab, were prepared and analyzed along with the routine drill samples. The primary lab prepared and analyzed 245 ‘rig’ duplicate samples (2.1% of all samples). The average value for the primary assay sample was 0.528 ppm Au and the average value for the “rig” duplicate assay was 0.544 ppm Au, a relative difference of 0.016 ppm Au, or 3.0 percent higher than the mean value for the primary sample. However, in several instances significant differences between 12 assay pairs were noted. The relative difference between these assay pairs ranged from 0.752 ppm to 10.1 ppm. Despite the high variance between these samples, a correlation coefficient of 0.932 was calculated for the data pairs. These results indicate an overall very good reproducibility of the sample preparation and analytical procedures by the primary lab. While the variance between the four sample pairs is relatively large, it probably reflects the erratic nature of the gold distribution in the sample rather than poor sample preparation or analytical precision.

12.3.5 Coarse Blank and Low-Level Gold Samples

A total of 493 samples (4.1% of all samples) containing coarse blank material (<0.005 ppm Au) or coarse material containing trace levels of gold (< 0.005 to 0.025 ppm Au), were systematically introduced into the sample stream to assess the potential for introducing contamination during sample preparation. The average assay value for the 250 blank samples (sample CO-1 + CO-2) was 0.006 ppm Au and varied from <0.005 to 0.282 ppm Au. The average value for the 225 trace-level gold material (sample CV-1) was 0.01 ppm Au and ranged from <0.005 ppm to 0.646 ppm Au. An examination of the expected values for the blank and low-level samples as compared with assay results of the immediately preceding 3 or 4 samples indicates that no discernible contamination occurred during sample preparation. The 19 detectable gold values (>0.005 ppm) from the coarse blank samples might indicate low level contamination or a lack of precision at low level determinations. A more likely explanation is that the control blank samples occasionally contain low-level detectable gold. However, if two samples are removed (0.275 ppm Au and 0.282 ppm Au), the average grade for the blank control samples is <0.005 ppm Au. The low-level sample that returned 0.646 ppm Au, as well as the three samples above and below this sample, were re-assayed. Results indicate that the analyses were correct, but that the control sample was mixed up with the preceding sample on the assay report. This error was corrected on the certificate of assay and in the Coral database.

12.4 CORAL GOLD DATA VERIFICATION OF DEEP RC DRILL SAMPLE RESULTS (2007)

The Coral primary lab for the 2007 RC drilling was ALS Chemex with sample prep facilities in Elko, NV and assay facilities in Reno, NV.

The ALS Chemex prep facilities was responsible for preparing coarse “rig” duplicate samples, coarse blank samples, coarse low-level gold samples, and for inserting the Coral supplied standard reference materials into the sample stream. The commercially available certified reference materials were supplied by MEG. The standards are highly reliable and were routinely monitored by Coral. Once the drill samples were reduced to pulp samples ready for assay, the certified reference pulps were inserted by the lab in pre-numbered sequence with the other pulps and assayed sequentially.

In addition, Coral was supplied with and monitored the analytical results of ALS Chemex own internal QA/QC controls for sample batches.

12.4.1 Summary of Verification-Drill Sample Results

To assess the ability of the primary lab to provide consistently accurate analytical results, their performance was monitored using certified reference pulps and rig duplicate samples. A total of 644 drill samples were submitted to ALS Chemex for initial gold determination from the 2007 drilling program. In addition Coral submitted 26 control (4.0%) samples to monitor lab performance.

12.4.2 Certified Reference Pulps

Analytical results for the 20 (3.1% of all samples) certified reference pulps submitted to ALS Chemex indicate that the average for all “certified” values was 1.047 ppm Au, versus the primary labs average value of 1.043 ppm Au. This is a relative difference of 0.004 ppm Au, or <1 percent lower than the mean of the certified values. A correlation coefficient of 0.997 was calculated for the pair. These results indicate that relative bias between the two data sets is absent.

12.4.3 Rig Duplicate Samples

The “rig” duplicate samples taken from reject material at the drill rig, and submitted blind to the lab, were prepared and analyzed along with the routine drill samples. The primary lab prepared and analyzed 13 ‘rig’ duplicate samples (2.0% of all samples). The average value for the primary assay sample was 0.111 ppm Au and the average value for the “rig” duplicate assay was 0.087 ppm Au, a relative difference of 0.024 ppm Au, or 21.6 percent lower than the mean value for the primary sample. However, in several instances a significant difference between 3 assay pairs was noted. The relative difference between these assay pairs ranged from 0.048 ppm to 0.350 ppm. Despite the high variance between these samples, a correlation coefficient of 0.993 was calculated for the data pairs. These results indicate an overall very good reproducibility of the sample preparation and analytical procedures by the primary lab. While the variance between the three sample pairs is relatively large, it probably reflects the erratic nature of the gold distribution in the sample rather than poor sample preparation or analytical precision.

12.5 CORAL GOLD DATA VERIFICATION OF RC DRILL SAMPLE RESULTS (2008, 2010)

The Coral primary lab for the 2008-2010 RC drilling was ALS Chemex with sample prep facilities in Elko, NV and assay facilities in Reno, NV.

The ALS Chemex prep facilities was responsible for preparing coarse "rig" duplicate samples, coarse blank samples, coarse low-level gold samples, and for inserting the Coral supplied standard reference materials into the sample stream. The commercially available certified reference materials were supplied by MEG. The standards are highly reliable and were routinely monitored by Coral. Once the drill samples were reduced to pulp samples ready for assay, the certified reference pulps were inserted by the lab in pre-numbered sequence with the other pulps and assayed sequentially.

In addition, Coral was supplied with and monitored the analytical results of ALS Chemex own internal QA/QC controls for sample batches.

12.5.1 Summary of Verification-Drill Sample Results

To assess the ability of the primary lab to provide consistently accurate analytical results, their performance was monitored using certified reference pulps and rig duplicate samples. A total of 6,160 drill samples were submitted to ALS Chemex for initial gold determination from the 2008 and 2010 drilling programs. In addition Coral submitted 620 control samples (10.1%) to monitor lab performance.

12.5.2 Certified Reference Pulps

Analytical results for the 95 (1.5% of all samples) certified reference pulps submitted to ALS Chemex in 2008 indicate that the average for all "certified" values was 1.314 ppm Au, versus the primary labs average value of 1.262 ppm Au. This is a relative difference of 0.052 ppm Au, or 4.0 percent lower than the mean of the certified values. This large difference is due to a single sample pair for which the lab sample was 0.068 ppm Au versus 4.5 ppm Au for the certified value. A correlation coefficient of 0.950 was calculated for the set of pairs. If this sample pair is removed for the data, the average certified value is 1.280 versus 1.2075 for the average lab value, a difference of 0.005 ppm Au and a correlation of 0.999.

Analytical results for the 40 (<1% of all samples) certified reference pulps submitted to ALS Chemex in 2010 indicate that the average for all "certified" values was 1.426 ppm Au, versus the primary labs average value of 1.402 ppm Au. This is a relative difference of 0.024 ppm Au, or 1.6 percent lower than the mean of the certified values. A correlation coefficient of 0.999 was calculated for the set of pairs.

These results indicate that relative bias between the two data sets is absent.

12.5.3 Rig Duplicate Samples

In 2008, the primary lab prepared and analyzed 125 'rig' duplicate samples (2.0% of all samples). The average value for the primary assay sample was 0.404 ppm Au and the average value for the "rig" duplicate assay was 0.319 ppm Au, a relative difference of 0.085 ppm Au, or 21 percent lower than the mean value for the primary sample. However, in one instance a significant difference between a single assay pairs was noted. The relative difference between these assay pairs 4.69 ppm Au. Because of the high variance between these samples, a correlation coefficient of 0.831 was calculated for the data pairs. These results indicate an overall very good reproducibility of the sample preparation and analytical procedures by the primary lab. While the variance between the single sample pairs is relatively large, it probably reflects the erratic nature of the gold distribution in the sample rather than poor sample preparation or analytical precision.

Forty "rig" duplicate samples were analyzed by the primary lab in 2010. The average value for the primary assay sample was 0.265 ppm Au and the average value for the "rig" duplicate assay was 0.199 ppm Au, a relative difference of 0.071 ppm Au, or 26.8 percent lower than the mean value for the primary sample and a correlation coefficient of only 0.454. These results indicate a large variance between samples possibly due to poor sampling at the splitter. However, much of this large variance is mainly due to a single assay pair having a relative difference of 3.63 ppm Au. With these samples removed from the data, the relative difference between the two data sets is 0.028 ppm Au, or an 18 percent higher average for the duplicate assays over the initial assay sample and a correlation coefficient of 0.918.

12.5.4 Coarse Blank and Low-Level Gold Samples

In 2008-2010, a total of 320 samples (4.1% of all samples) containing coarse blank material (<0.005 ppm Au) or coarse material containing trace levels of gold (< 0.005 to 0.025 ppm Au), were systematically introduced into the sample stream to assess the potential for introducing contamination during sample preparation. The average assay value for the 233 blank samples (sample CO-1) was 0.006 ppm Au and varied from <0.005 to 0.246 ppm Au. Of the 233 "blank" control samples, 65 samples returned gold values between 0.005 ppm and 246 ppm with an average gold value of 0.013 ppm. The average value for the 87 trace-level gold material (sample CV-1) was 0.008 ppm Au and ranged from <0.005 ppm to 0.056 ppm Au. An examination of the expected values for the blank and low-level samples as compared with assay results of the immediately preceding 3 or 4 samples indicates that no discernible contamination occurred during sample preparation. The 65 detectable gold values (≥0.005 ppm) from the coarse blank samples might indicate low level contamination or a lack of precision at low level determinations. A more likely explanation is that the control blank samples occasionally contain low-level detectable gold. However, if a single sample is removed (0.246 ppm Au), the average grade for the blank control samples is <0.005 ppm Au.

12.6 CORAL GOLD DATA VERIFICATION OF DDH SAMPLE RESULTS (2010)

The Coral primary lab for the 2010 core drilling was ALS Chemex with sample prep facilities in Elko, NV and assay facilities in Reno, NV.

The ALS Chemex prep facilities was responsible for preparing coarse blank samples, coarse low-level gold samples, and for inserting the Coral supplied standard reference materials into the sample stream. The commercially available certified reference materials were supplied by MEG. The standards are highly reliable and were routinely monitored by Coral. Once the drill samples were reduced to pulp samples ready for assay, the certified reference pulps were inserted by the lab in pre-numbered sequence with the other pulps and assayed sequentially.

In addition, Coral was supplied with and monitored the analytical results of ALS Chemex own internal QA/QC controls for sample batches.

12.6.1 Summary of Verification-Drill Sample Results

To assess the ability of the primary lab to provide consistently accurate analytical results, their performance was monitored using certified reference pulps and coarse blank and low-level control samples. A total of 1,224 core samples were submitted to ALS Chemex for initial gold determination from the 2010 core drilling program. In addition Coral submitted 113 control samples (9.2%) to monitor lab performance.

12.6.2 Certified Reference Pulps

Analytical results for the 43 (3.5% of all samples) certified reference pulps submitted to ALS Chemex indicate that the average for all "certified" values was 0.982 ppm Au, versus the primary labs average value of 0.974 ppm Au. This is a relative difference of 0.008 ppm Au, or <1 percent of the mean of the certified values. A correlation coefficient of 0.997 was calculated for the set of pairs.

12.6.3 Check Assays

Check assays of the 2010 initial assay were performed using 17 composite samples composed of 281 coarse reject samples used for metallurgical testing. The average value from the initial (ALS) assay of the 17 composite samples was 0.89 ppm Au versus the average value from check (RMG) assays which was 0.86 ppm Au, a difference of 0.03 ppm (3.4%). A correlation coefficient of 0.7510 was calculated for the data pairs. A comparison of the initial assay versus the check assay results indicates that on average the initial assays are higher than the corresponding check samples (Figure 12-3). On a sample by sample comparison, results between the two data sets (average values) indicate that the initial assay values varied from 0.39 ppm lower to 0.72 ppm higher than the corresponding check assay. These results suggest that either the initial determinations are biased high or the check sample assays are biased low. In addition problems with analytical reproducibility (precision) or coarse gold may also be present. To assess these issues, assay results from both labs were compared with the head grades calculated from 1,000 g cyanide leach tests. The average initial assay values for the individual 17 composite samples were compared with the average grade for the calculated head grade and plotted on a graph shown in Figure 12-4. The average for the 17n initial assays was 0.89 ppm Au compared to 0.82 ppm Au for the calculated head grade, a difference of 0.07 ppm (7.9%). The 17 sample pairs have a correlation coefficient of 0.8314. A comparison of the average check assay value (0.86 ppm Au) for each of the 17 composite samples with the calculated head grade based on results from 17 bottle roll cyanide leach tests is shown in Figure 12-5. The two data sets have an average difference of 0.04 ppm (4.6%) and a correlation coefficient of 0.9162, indicating relatively good agreement. Although the check assays are on average higher that the calculated head grades, the variance ranges from 0.18 ppm lower to 0.22 ppm higher values for the check assays. These comparisons support the conclusion that assay results from ALS exhibit an overall slightly higher bias verses either the check assays or the calculated head grades for the 17 composite samples.

12.6.4 Coarse Blank and Low-Level Gold Samples

A total of 70 samples (5.7% of all samples) containing coarse blank material (<0.005 ppm Au) or coarse material containing trace levels of gold (< 0.005 to 0.040 ppm Au), were systematically introduced into the sample stream to assess the potential for introducing contamination during sample preparation. The average assay value for the 36 blank samples (sample CO-1) was 0.005 ppm Au and varied from <0.005 to 0.015 ppm Au. Of the 36 "blank" control samples, 14 samples returned gold values between 0.005 ppm and 0.015 ppm with an average gold value of 0.007 ppm. The average value for the 34 trace-level gold material (sample CV-1) was 0.011 ppm Au and ranged from <0.005 ppm to 0.040 ppm Au. An examination of the expected values for the blank and low-level samples as compared with assay results of the immediately preceding 3 or 4 samples indicates that no discernible contamination occurred during sample preparation. The 14 detectable gold values (≥0.005 ppm) from the coarse blank samples might indicate low level contamination or a lack of precision at low level determinations. A more likely explanation is that the control blank samples occasionally contain low-level detectable gold.

12.6.5 Comparison of Twin DDH and RC Assay Results

An objective of the 2010 core drilling program was to evaluate the potential for down hole contamination and verify assay results from a series of RC holes drilled in 2008. This was accomplished by twinning 11 existing RC holes with HQ-diameter core which produced 960 sample pairs (1,920 samples) from the two sets of twinned holes. To test the effectiveness of RC drilling to return representative gold assays, the global average grades from the twinned RC holes were compared with the global average grades returned from the corresponding DDH. The results are shown in Table 12.1 and a plot of results is in Figure 12-1. The global average grade for all twinned RC holes is 0.011 ozAu/t compared with a global average grade for all corresponding core holes of 0.0107 ozAu/t, relative difference of 0.0003 ozAu/t (<1% of both mean values). Correlation between the twinned RC and cores ranged from 0.0271 to 0.7116, indicating mostly poor short range correlation between the holes which were spaced between 4 ft and 17 ft apart. By removing the top one percentile from the data, an average global average grade of 0.010 ozAu/t for the RC holes and 0.0095 ozAu/t for the core holes results, which is a difference of 0.0005 ozAu/t.

Table 12.1: Comparison of global average grades between twin core and RC holes

		Average Grade	
Core Hole	**RC Hole**	**Core, ozAu/t**	**RC, ozAu/t**
CC10-1	CR08-37	0.0098	0.0102
CC10-2	CR08-32	0.0061	0.0074
CC10-3	CR08-31	0.0128	0.0112
CC10-4	CR08-30	0.0122	0.014
CC10-5	CR08-29	0.0101	0.0089
CC10-6	CR08-27	0.0134	0.0125
CC10-7	CR08-26	0.0135	0.011
CC10-8	CR08-25	0.011	0.0084
CC10-9	CR08-21	0.0099	0.0126
CC10-10	CR08-19	0.0084	0.0119
CC10-12	CR08-18	0.0095	0.0117
All core	**All RC**	**0.0107**	**0.011**

When the twinned RC hole and corresponding core holes are compared on a grade zone basis, as defined by 0.010 ozAu/t cutoff grade and at equal elevation, the average grades are 0.0225 ozAu/t and 0.232 ozAu/t, respectively. Table 12.2 is a summary comparing selected 0.01 ozAu/t grade zones in the twinned RC and core holes. As shown in Table 14.2, the average difference between the twinned RC and DD holes is 0.0027 ozAu/t. While the thickness of grade zones varies from 5 ft to 110 ft, the average thickness indicates that the RC grade zones are on average 5 ft thicker.

Table 12.2: Comparison of twinned RC/DDH ≥ 0.01 ozAu/t grade zones.

Twin Hole ID	Interval thickness (ft)	Grade ozAu/t	DD grade diff. (ozAu/t)	DD thickness diff. (ft)
CC10-1	140	0.022	0.002	+5
CR08-37	135	0.020		
CC10-3	180	0.026	0.008	+10
CR08-31	170	0.018		
CC10-4	150	0.0225	-0.0005	-10
CR08-30	160	0.023		
CC10-5	105	0.023	0.002	+25
CR08-29	80	0.021		
CC10-6	265	0.019	0.0002	+110
CR08-27	155	0.0188		
CC10-7	90	0.038	0.016	-40
CR08-26	130	0.020		
CC10-8	80	0.019	0	-10
CR08-25	90	0.019		
CC10-9	125	0.0184	0.0004	-80
CR08-21	205	0.018		
CC10-10	145	0.0179	-0.0096	-10
CR08-19	155	0.0275		
CC10-12	80	0.0258	0.0112	-5
CR08-18	85	0.037		
Ave diff.			**0.0027**	**-5**

To evaluate potential for down hole contamination a comparison was made of the depth at which individual grade zones start and end in the twinned RC and corresponding core holes. The elevation difference between the top of grade zones in the twinned RC versus the core holes ranges from 0 to 40 ft with grade zones in the core starting an of average 20 ft higher. It should be noted that that all of the twinned RC holes had 10 ft of casing installed using open-hole rotary methods which results in generally poor sample quality. Also poor core recovery in the upper 20 ft of most the 2010 core holes resulted in small or no samples.

12.7 AUTHORS DATA VERIFICATION

As project manager for Amax during the time period 1990 through 1994 and a supervising Qualified Person for Coral Gold (2004-2010), the author was primarily responsible for developing, implementing and monitoring the QA/QC program employed by Amax and Coral Gold at the Robertson Property. These results verify the adequacy and effectiveness of these QA/QC programs, and justify a high level of confidence in the analytical data generated by Amax and Coral Gold used in the evaluation of the Robertson project.

It should be noted that although the overall quality of the assay data from the primary lab appears to be good, results reported by the check labs are, on average, lower than either the initial assay or calculated head grade for the same sample. This appears to be the result of a systematic low bias introduced during the check assay process or during the AA determination. A comparison of all (2004-2010) assays of certified reference pulp with the primary lab assay results indicates a consistent low bias in the primary lab assays. In addition, the primary lab reported a number of inconsistent gold determinations for samples at or below the lower limit of detection (0.005 ppm), possibly indicating a lack of analytical precision at lower levels of gold detection.

These results indicate that down hole contamination during RC drilling is minimal. This is confirmed by the twin core holes drilled by both Amax (1994 feasibility study) and Coral (2010) that there is no evidence of down hole contamination and the differences in global average grades between the twinned RC and core holes are well within acceptable limits.

An additional concern is the high variability between the initial lab assay and the "rig" duplicate assay. This is usually due to significant differences between just one or two sample pairs. However, when compared on a sample by sample basis the gold values from both data sets are in "relative" agreement. Elevated gold values in the initial assay data set are usually mirrored by similarly elevated values in the duplicate sample set and low values are likewise mirrored. The variance is likely due to natural short range variability in gold distribution and sampling methodology.

While the author has no first-hand knowledge of the QA/QC program employed by Cortez during their 1999 evaluation of the Robertson Project, it is the author's opinion that as a mining industry leader, they employed a program which was adequate to justify a high level of confidence in the analytical data generated.

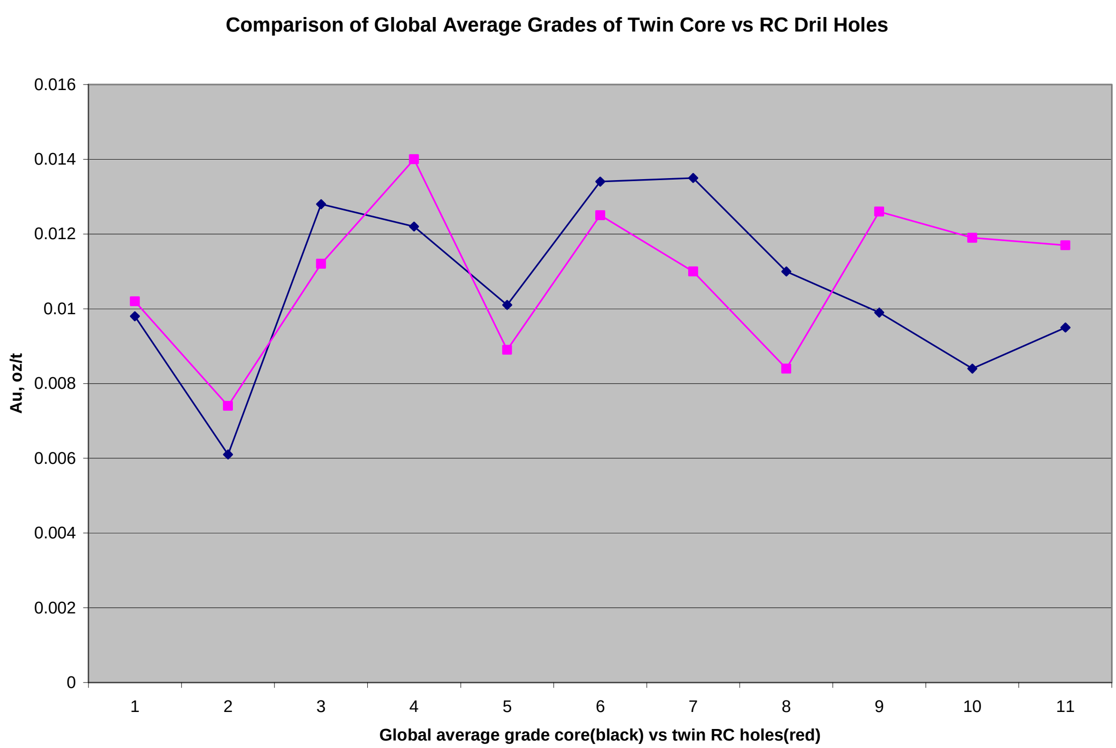


Figure 12-1: Comparison of global average grades for twinned core holes (black) vs. RC holes (red)

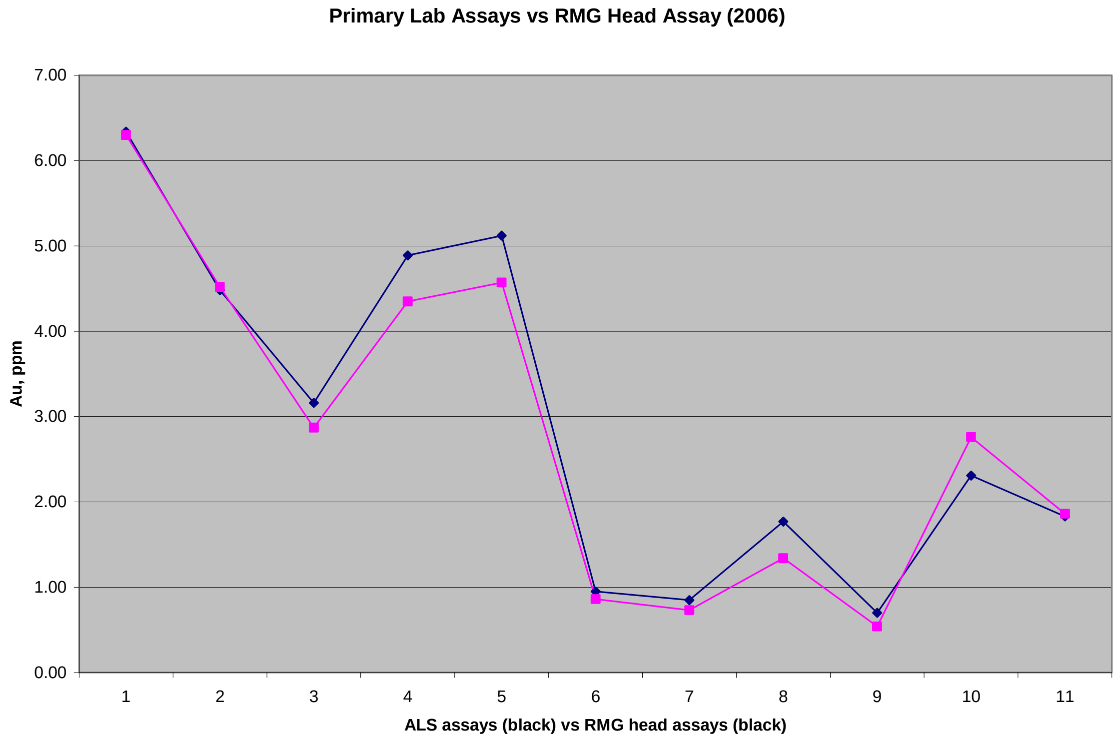


Figure 12-2: Comparison of initial lab assays (black) vs. RMG head assays (red)

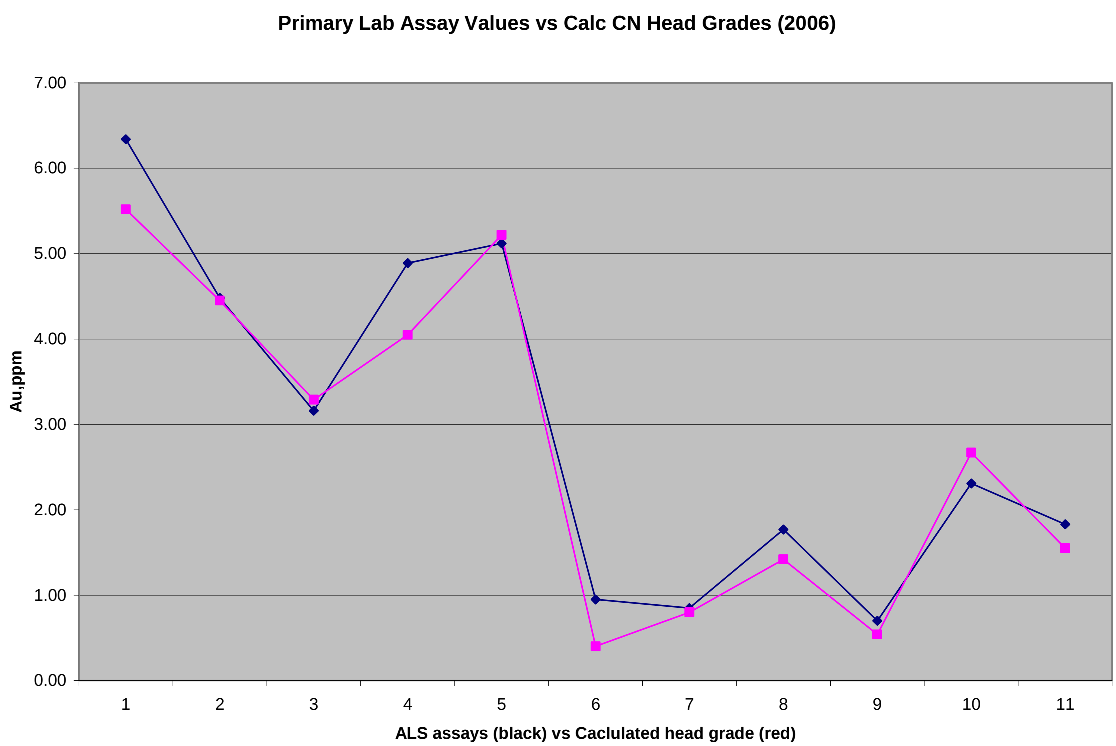


Figure 12-3: Comparison of initial assays (black) vs. calculated CN head grade (red)

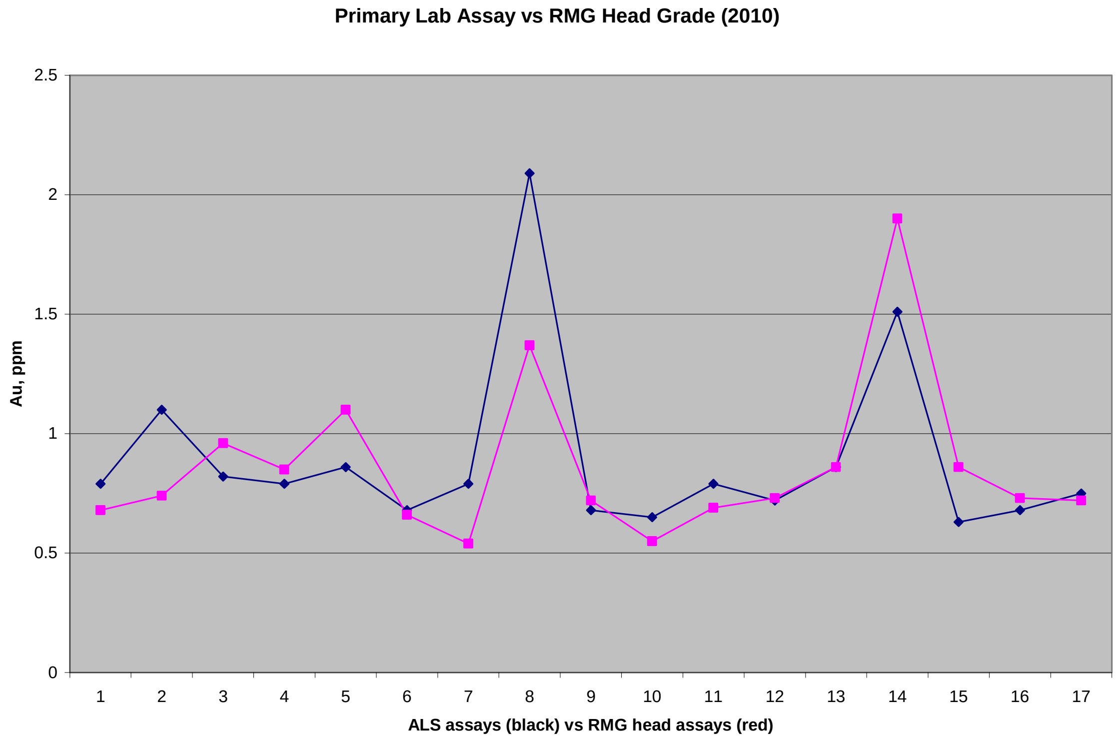


Figure 12-4: Comparison of initial assay (black) vs. RMG head grade (red)

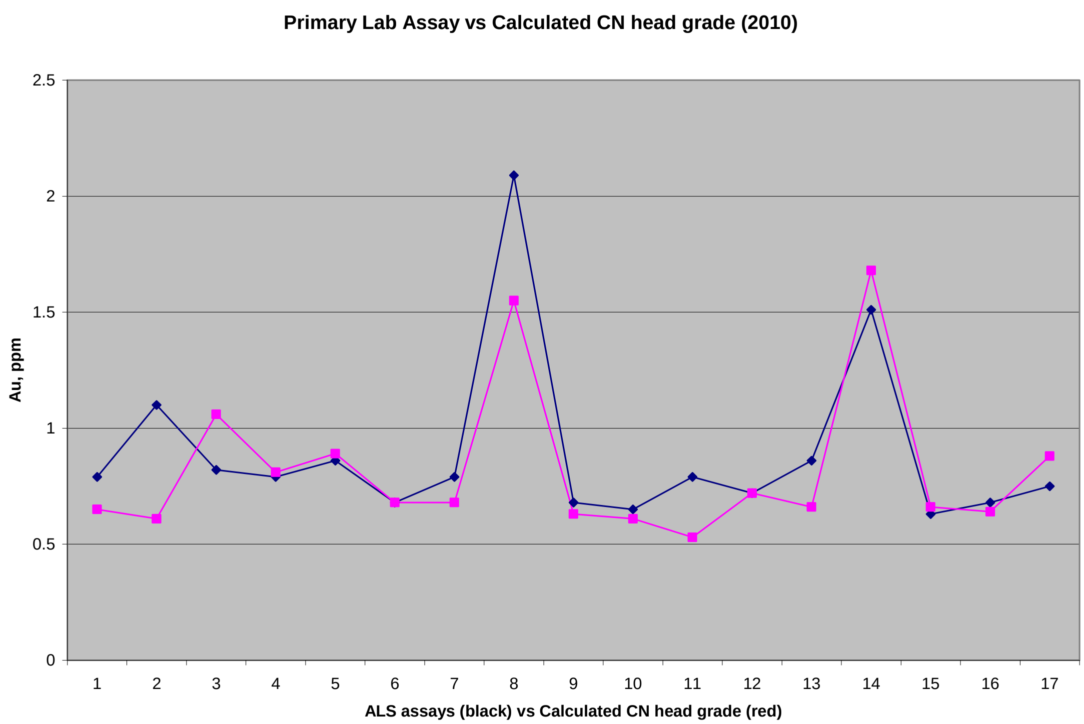


Figure 12-5: Comparison of initial assays (black) vs. calculated CN head grade (red)

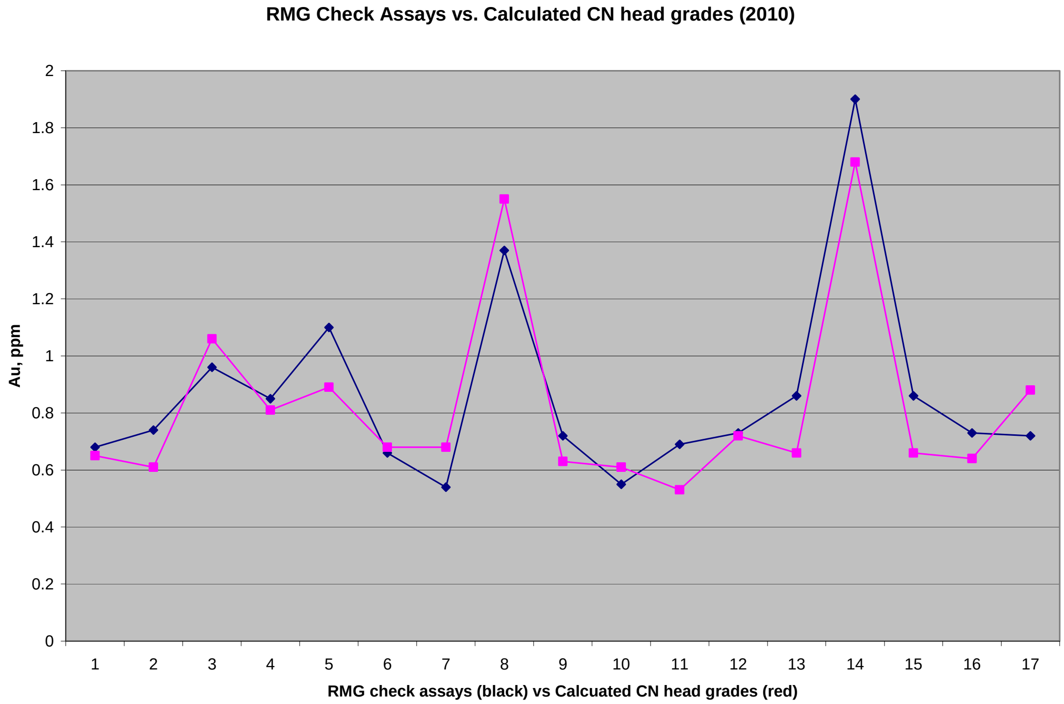


Figure 12-6: Comparison of check assays (black) vs. Calculated CN head grade (red)

SECTION 13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The Robertson Property has six mineralized zones (resources) or deposits and one mined out deposit with the following names:

- Porphyry;
- 39A;
- Distal;
- Altenburg Hill;
- Gold Pan;
- Triplet Gulch;
- and the mined out Tenabo Gold Quartz Deposit.

Historical metallurgical studies have indicated that the Porphyry, Altenburg Hill and the near surface zone of the Gold Pan and 39A are amenable to heap leach treatment. The Porphyry deposit process response is the best understood. In 2011 a laboratory program was undertaken at McClelland Laboratories in Sparks, Nevada to provide further leaching data for samples originating from the Altenburg Hill and Gold Pan deposits, and included some low grade samples which were labeled as overburden.

An overview of the historical test work performed on the property, followed by a description and results summary of the 2011 laboratory program are provided below.

13.1.1 Historical Studies

Of the six resources and deposits identified at Robertson, the Porphyry has had most of the previous test work performed on it as it was the subject of a feasibility study by Amax Gold in 1994. Its metallurgical characteristics are better defined than the other deposits. Its test program can be classed as detailed, while the others are more preliminary in nature. No documentation on the metallurgical characteristics of the Triplet Gulch Deposit was available and it is not known if such work was carried out in the past.

Details of the metallurgical testwork performed on each of the remaining mineralized zones or deposits together with the names of the laboratories that undertook the work are provided below summarizing work done from the late 1980’s to 2007.

13.1.1.1 Gold Quartz Deposit

According to Mr. Bob McCusker, the Tenabo Gold Quartz Deposit has been mined out and is on longer relevant to the current evaluation of the Property but the historical metallurgical testwork and process information have been included for the record of this report as this deposit is close to the Gold Pan Deposit.

In 1981, the Miller-Kappes Company performed 14 cyanide leach tests. Two of the samples were from the small open pit (glory hole) and waste dumps in the original Gold Quartz mine. One was taken near the iron stained area of the north rim and the other from a low grade zone from the west end of the pit. The other twelve were from an existing ore heap and were taken by back-hoe trenching.

All samples were crushed to minus 5/8" and a 50lb split was used for testing. Leach solution ranging from 0.3 to 0.9 g/L cyanide and pH 9.5 to 10.7 was slowly dripped from a head tank to a column containing the crushed ore sample. Pregnant solution draining from the column reported to a bottle containing activated carbon to adsorb the leached gold and silver values. The pit samples were leached for 101 days and the ore heap samples for 75 days. Lime and cyanide consumptions averaged 2.7 and 3lb/ton respectively.

Results of the tests are presented in the table 13.1 below:

Table 13.1: Cyanide Leach Tests

	Gold Quartz Pit	Ore Heap	Weighted Average
Feed Grade (oz Au/t)	0.026	0.050	0.046
Leach Residue (oz Au/t)	0.004	0.009	0.008
Recovery to day 15		57.5	
Recovery to day 27	83.6		
Final Gold Recovery	87.9	77.3	78.8

In 1986, an additional column test by Kappes Cassidy and Associates using a 50kg of bulk ore sample of Tenabo ore was tested. The sample was crushed to minus 1 inch and placed in a 6 inch diameter column. Height of ore in the column was about 63 inches. Cyanide solution strength of 0.6g/L was dripped into the column for a period of 24 hours. The column was then left dormant for the following 24 hours. This 24 hour leach and rest cycle was maintained for the test duration of 40 days. The collected pregnant solution was passed through a bottle containing activated carbon before being returned to leach.

From the calculated head grade of 0.033oz Au/t, 75.7% was recovered in the first 7 days and 84.8% gold recovered in the 40 day test period. Lime consumption was 2.56lb/ton ore and the cyanide consumption was 1.35lb/ton.

Analysis of the screen fractions of the column feed showed the gold to be concentrated in the minus 1/4 inch fraction which comprised 35% of the weight and 86% of the gold. Recoveries of gold from the various fractions range from 50% in the plus ½ inch to 94% from the minus 65 mesh fraction.

An agitated cyanide leach test was also performed on a sample ground to 100% passing 100mesh. Based on a head grade of 0.036ozAu/t, the recovery was 94.7% in 24 hours of leach time. It was also reported that solution fouling by soluble copper would be minor.

Reference: Kilborn Engineering (B.C.) Ltd Coral Gold Corp. dated January 1989, Evaluation Report.

13.1.1.2 Porphyry Oxide and Sulfide Resource (from McCusker 2006 NI 43-101 report)

As part of a 1994 feasibility study of the Porphyry Zone resource, Amax conducted detailed metallurgical testing in three phases summarized below. Phase I testing was performed by AMAX Research and Development in Golden, CO and Phase II and Phase III testing were performed by McClelland Laboratories of Sparks, NV.

Phase I:	Preliminary bottle roll leach tests on 32 composite RC cuttings (assay rejects) at 10 mesh and 80 mesh. Composite samples represent a variety of oxidation states, gold grade and copper content.
Phase II:	Detailed column and bottle roll leach testing on 13 composite samples from PQ-diameter (3.3 inches) core, representing variable gold grade, oxidation state and cyanide soluble copper content. Used crush sizes of 1 1/2 inches and ½ inch and grind sizes ranging from 10 mesh, 100 mesh and 200 mesh.
Phase III:	Bottle roll testing on 13 composite samples at ½ inch and 100 mesh. Composite material is from 3 PQ-diameter core and RC cuttings from a new area within the Porphyry resource.

Phase I

Phase I results indicates gold extractions from thirty 10 mesh samples ranged from 60.9 percent to 96.9 percent. Extractions for the two 80 mesh samples achieved 35.3 percent and 50 percent. A summary of Phase I metallurgical testing is shown in Table 13.2.

Table 13.2: Summary of Phase I Metallurgical Test Results (AMAX R&D,1993)

Sample Description	**--------Head Grades-------**			**Au Extn**	**Ag Extn**	**Cu Extn**	**Consumptions (lbs/ton)**	
	Au,opt	**Ag,ppm**	**Cu,,ppm**	**(%)**	**(%)**	**(%)**	**NaCN**	**Lime**
Oxide (no Cu)	0.028	2.77	971	82.8	33.9	11.5	1.92	1.1
Oxide (with Cu)	0.027	N/A	2,253	83.9	56.7	11.8	3.0	1.1
Part Oxide (no Cu)	0.022	N/A	1,229	82.5	27.4	27.2	3.33	2.1
Sulfide (no Cu)	0.023	1.1	323	77.3	46.7	19.0	2.2	0.4
Sulfide (with Cu)	0.015	1.4	788	69.1	30.1	21.8	3.6	2.4*
High grade	0.263	3.5	1,550	76.3	41.4	36.2	3.6	N/A

*Average of 7 samples, average of 6 samples 0.63 lbs/ton.

Phase II

The purpose of Phase II testing was to obtain data for process design. Core composites were prepared using Amax interval compositing instructions. The 13 composites for columns, bottle roll and crushing tests were prepared by blending, coning and quartering each specified interval. All rejects from the coning and quartering of 1 ½ inch material were crushed to 80 percent passing ½ inch and were blended and split to obtain samples for testing the 10 mesh and finer sizes. A total of 337 intervals of core from five PQ-diameter diamond drill holes were used to develop the composite samples. Results of Phase II are summarized in Table 13.3.

Table 13.3: Summary of Phase II Metallurgical Test Results (McClelland, 1994)

Comp. No.	Calc Head Grade	Oxid. %	CN Sol Cu(lbs/t)	Gold Extractions (%) 1 ½ "	½ "	10m	100m	200m	Consumptions(1/2" feed) NaCN(lbs/t)	Lime(lbs/t)
1	0.018	100	0.2	56.3	72.2	76.5	89.5	89.5	1.69	4.5
2	0.023	99	0.2	8.2	82.6	75.0	84.0	84.0	1.51	4.3
3	0.017	86	0.2	55.	75.0	73.7	88.9	89.5	1.36	4.5
4	0.028	99	0.4	51.7	N/A	78.3	92.6	93.1	N/A	N/A
5	0.024	58	1.53	38.1	54.5	72.0	92.3	92.0	2.57	5.0
6	0.035	100	1.27	44.8	N/A	81.3	93.9	97.1	N/A	N/A
7	0.031	98	0.3	N/A	72.4	83.3	93.9	93.9	2.92	5.3
8	0.038	89	7.33	36.4	51.2	31.6	26.3	29.7	12.0	111.3
9	0.032	97	2.07	24.1	50.0	3.1	82.1	79.2	3.33	5.2
10	0.017	6	0.4	33.3	53.8	75.0	93.3	93.8	1.58	3.8
11	0.040	0	0.4	N/A	39.1	61.9	86.5	92.3	1.17	2.4
12	0.111	100	0.2	N/A	N/A	68.3	92.6	94.0	N/A	N/A
13	0.119	58	2.20	N/A	N/A	60.0	79.0	80.5	N/A	N/A

Composite head grades were determined by conventional fire assays for gold and silver, triplicate gold metallic screen fire assays, multi-element ICP analysis, sulfur speciation, carbon speciation and cyanide soluble gold and copper shaker tests.

Bottle roll leach tests were conducted on 13 composites using 1 ½ inch, ½ inch, 10 mesh, 100 mesh and 200 mesh material to determine gold, silver and copper extractions and reagent sensitivity to ore feed size. Bottle roll tests were conducted over a period of 96 hours.

Column percolation leach tests were performed on 9 composites at feed sizes of 80 percent passing 1 ½ inch and ½ inch. The ore was charged into 10-ft-high columns with inside diameters ranging from 4 inches to 12 inches. The column leach tests were conducted over a period of 39 days to 53 days.

Results from Phase II testing indicate that gold extractions are highly sensitive to ore crush size. Since recoveries were markedly increased with a finer crush size, Amax chose a nominal ½ inch crush size as the basis for their heap leach process design.

13.1.2 Phase III

The purpose of Phase III testing was to obtain additional data on extensions of the Porphyry resource that were identified after Phase II testing had begun. Because of time constraints imposed by the November 1994 deadline for producing the feasibility study, only bottle roll leach tests were conducted during Phase III. The core intervals were prepared and assayed using the same protocols as were the Phase II samples. A total of 234 intervals from three PQ-diameter diamond drill holes, were made into 13 composite samples at crush sizes of 80 percent passing ½ inch and 80 percent passing 100 mesh. Bottle roll procedures were the same as those used in Phase II and test results are summarized in Table 13.4.

Table 13.4: Summary of Phase III Metallurgical Test Results (McClelland, 1994)

Comp.	Calc Head	Oxid.	CN Sol	Gold Extractions (%)					Consumptions(1/2" feed)	
No.	Grade	%	Cu(lbs/t)	1 ½ "	½ "	10m	100m	200m	NaCN(lbs/t)	Lime(lbs/t)
14	0.020	100	0.3	N/A	66.7	N/A	83.3	N/A	0.15	7.3
15	0.019	100	0.2	N/A	53.3	N/A	93.8	N/A	0.85	4.1
16	0.025	100	0.2	N/A	72.4	N/A	96.0	N/A	0.28	6.3
17	0.025	91	0.4	N/A	78.3	N/A	96.6	N/A	0.92	9.3
18	0.018	98	0.2	N/A	65.0	N/A	85.7	N/A	0.24	7.6
19	0.034	100	0.2	N/A	51.3	N/A	92.5	N/A	0.90	4.7
20	0.131	100	0.2	N/A	51.7	N/A	93.4	N/A	0.21	10.2
21	0.023	88	0.6	N/A	50.0	N/A	94.7	N/A	0.63	5.9
22	0.800	95	6.0	N/A	41.2	N/A	57.5	N/A	9.12	8.3
23	0.174	50	0.6	N/A	20.9	N/A	30.8	N/A	27.02	109.3
24	0.076	73	1.8	N/A	30.8	N/A	94.0	N/A	2.6	4.3
25	0.038	5	6.0	N/A	43.2	N/A	89.4	N/A	4.97	19.0
26	0.051	8	0.2	N/A	20.0	N/A	92.2	N/A	1.79	4. 3

The culmination of the Amax and McClelland metallurgical testwork studies resulted in MRDI proposing a 17000tpd heap leach operation with relocated crushing, screening and conveying equipment from the Sleeper Mine in the feasibility study. Ore was to be crushed to minus ½" and conveyed to the leach pad. The projected gold recovery at this crush size was 67% for both the oxide and sulfide resources. Carbon columns were proposed for treating the gold bearing pregnant solution from the heap and a copper strip using cyanide to remove copper from the loaded carbon was included in the flowsheet. The resulting loaded carbon after copper removal was then shipped to the Sleeper mine for toll processing of the gold.

Following the Amax feasibility study, there has been no additional drilling or metallurgical testwork on the Porphyry deposit. Optimization of the crush size ie 10 mesh in a column test and other means for dealing with the soluble copper minerals and the resulting high cyanide consumption needs further work. In the Amax feasibility study, the conclusion was made that the tonnage of ore with high soluble content was small compared to the overall tonnage of the mineable ore, hence it was not considered to be a big operating problem for the plant.

13.2 GOLD PAN, ALTENBURG HILL AND 39A RESOURCES

In 1996, McClelland Labs of Sparks, NV conducted metallurgical studies on shallow oxide and mixed oxide/sulfide material from the Gold Pan Zone (composites 1-7, 17-19), oxide material from Altenburg Hill Zone (composites 8-11) and ore-grade sulfide material from the 39A Zone (composites 12-16, 20-22). These studies included 40 direct agitation cyanide leach tests (bottle rolls) using various grind sizes on all the samples; 11 bulk sulfide flotation tests on the Gold Pan and the 39A sulfide resource and 5 column percolation leach tests on the Gold Pan and Altenburg Hill samples. Composites samples were prepared and analyzed in the same manner as describe above in Phase II of the Amax study.

Samples for testing consisted of 9 composites (1-9) derived from 135 five-foot-intervals from drill core, 10 composites (10-19) from RC cuttings and 3 composites (20-22) from pulverized drill core assay reject samples. Bulk sulfide flotation tests were conducted on selected composites at a nominal 35 mesh. Results from these tests are summarized in Table 13.5. Direct agitated cyanidation (bottle roll) tests were conducted on all 22 composites at feed sizes of 80% -11/2 ", 80% -1/2", 80% -35M, 80% -10M and 80-95% -200M. Results of these tests are reported in Table 13.5. Column percolation leach tests were conducted on drill core composites 2, 4, 6, 8, and 9 at a feed size of 80% -1/2". Results from these tests are summarized in Table 13.7.

As shown in Table 13.5, the selected composites tested generally respond well, except for composite 1, to beneficiation using conventional flotation methods at 35 mesh grind size. Gold recovery in the cleaner concentrates achieved by flotation for the remaining composites at 35 mesh feed size ranged from 74.3 to 88.1 percent. A single flotation test was conducted using –200M feed size resulted in lower gold recovery probably due to the oxidation products formed during grinding prior to flotation. In general the flotation results improved with depth for the Gold Pan and the 39A in fill samples. Flotation was not performed on the Altenburg Hill samples.

Table 13.5: Summary of Bulk Sulfide Flotation Test Results (McClelland, 1996)

Comp.	Interval	Feed	-----Cleaner Concentrates-----				Recovery			Concentration			
			Wt.	Assay, oz/ton			Percent (1) %			Ratio			
No.	Ft.	Size	%	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Wt
1	60-70	35M	1.7	2.263	157.68	6.60	26.1	30.1	8.8	15:1	18:1	5:1	59:1
1	60-70	35M	0.7	1.375	183.39	6.10	7.1	16.7	3.6	10:1	24:1	5:1	143:1
1	60-70	200M	1.3	0.823	76.21	2.83	9.9	16.0	3.1	8:1	12:1	2:1	77:1
3	135-155	35M	1.7	5.691	3.09	2.72	54.9	28.7	22.9	32:1	17:1	14:1	59:1
6	160-200	35M	1.6	0.831	0.98	1.68	77.0	25.5	70.3	49:1	16:1	42:1	62:1
7	200-205	35M	2.1	3.000	0.42	1.49	77.9	18.3	67.8	37:1	8.1	30:1	48:1
13	295-300	35M	6.4	1.326	0.33	0.20	76.5	34.8	54.1	12:1	5:1	10:1	16:1
14	465-550	35M	1.6	3.116	1.44	1.45	74.3	26.0	64.7	46:1	46:1	36:1	62:1
20	545-665	35M	0.6	19.060	4.79	0.62	76.6	45.8	24.9	128:1	80:1	62:1	167:1
21	600-775	35M	2.4	2.725	0.61	0.98	80.0	53.0	62.0	33:1	20:1	25:1	42:1
22	605-665	35M	4.4	6.343	1.06	1.55	88.1	77.0	77.4	20:1	18:1	17:1	23:1

(1) Recovered in the cleaner concentrate.

As summarized in Table 13.6, bottle roll tests conducted on the drill core composite samples (1-9) averaged only 45.0 percent gold recovery using 80 percent –1/2 inch feed size. Crushing these composites to 80 percent –10 mesh improved gold recoveries significantly, averaging 71.2 percent. Also a comparison of composite samples used in the column leach tests with the corresponding –1/2 inch feed bottle roll composites, indicates that the column tests achieved significantly higher gold recoveries than the bottle roll tests (60.8 % vs. 45%). The higher column recoveries were probably on account of the 40 or more days of leach time vs 4 days for the bottle roll tests.

RC cuttings composites (10-19) from the Gold Pan, Altenburg Hill and 39A were amenable to direct agitation cyanidation treatment at the “as received” nominal 10M feed size. Gold recoveries varied from 42.9 to 78.8 percent and averaged 59.9 percent.

Drill core assay reject composite samples (20-22) from the 39A were readily amenable to direct agitated cyanidation treatment at the “as received” approximately 95 percent –35M feed size. Gold recoveries from these samples were 94.7, 91.2 and 93.2 percent, respectively and were only slightly improved by grinding to 80 percent –200M.

Table 13.6 Summary of Bottle Roll Test Results (McClelland, 1996)

Comp.	Hole	Interval	Feed	Au Recov.	-----ozAu/t ore-			Reagent Consump. lb/ton ore	
No.	No.	Feet	Size	%	Ext'd	Tail	Calc. Head	NaCN	Lime
1	CAT-60	60-70	½”	40.1	0.059	0.088	0.147	10.02	19.6
1	CAT-60	60-70	10M	45.2	0.061	0.078	0.135	11.71	22.2
2	CAT-60	80-135	1 ½”	70.6	0.012	0.005	0.017	0.47	5.2
2	CAT-60	80-135	½”	78.9	0.015	0.004	0.019	0.40	8.1
2	CAT-60	80-135	10M	83.3	0.015	0.003	0.018	0.53	10.0
3	CAT-60	135-155	1 ½”	52.1	0.061	0.056	0.117	.35	38.8
3	CAT-60	135-155	½”	48.5	0.065	0.069	0.134	5.93	30.4
3	CAT-60	135-155	10M	70.3	0.116	0.049	0.165	12.28	73.8
4	CAT-60	180-235	1 ½”	33.3	0.006	0.012	0.018	0.30	3.1
4	CAT-60	180-235	½”	30.4	0.007	0.016	0.023	0.15	4.0
4	CAT-60	180-235	10M	64.3	0.018	0.010	0.028	1.02	4.8
5	CAT-61	130-140	½”	46.7	0.014	0.016	0.030	8.19	52.7
5	CAT-61	130-140	10M	39.6	0.019	0.029	0.048	7.64	45.9
6	CAT-61	160-200	1 ½”	59.1	0.013	0.009	0.022	0.99	4.6
6	CAT-61	160-200	½”	56.3	0.009	0.007	0.016	0.57	4.8
6	CAT-61	160-200	10M	76.0	0.019	0.006	0.025	1.94	6.6
7	CAT-61	205-210	½”	35.4	0.067	0.112	0.189	0.29	3.6
7	CAT-61	205-210	10M	75.7	0.131	0.042	0.173	0.87	4.5
8	CAT-62	0-35	1 ½”	39.3	0.011	0.017	0.028	0.16	4.9
8	CAT-62	0-35	½”	37.5	0.009	0.015	0.024	0.25	5.4
8	CAT-62	0-35	10M	63.3	0.019	0.011	0.030	0.05	4.7
9	CAT-62	35-110	1 ½”	20.7	0.006	0.023	0.029	0.15	3.0
9	CAT-62	35-110	½”	31.3	0.005	0.011	0.016	0.26	2.7
9	CAT-62	35-110	10M	70.0	0.021	0.009	0.030	0.01	3.2
10	AT-289*	15-85	10M	60.9	0.014	0.009	0.023	0.19	4.9
11	AT-289*	85-155	10M	61.1	0.011	0.007	0.018	0.01	4.9
12	AT-302*	295-300	10M	78.8	0.078	0.021	0.099	0.61	5.0
13	AT-302*	395-405	10M	46.9	0.043	0.049	0.092	0.41	3.7
14	AT-302*	465-550	10M	62.7	0.059	0.035	0.094	0.31	3.0
15	AT-302*	575-640	10M	52.3	0.023	0.021	0.044	0.15	2.9
16	AT-302*	640-650	10M	42.9	0.024	0.032	0.056	0.38	3.0
17	AT-335*	65-90	10M	68.8	0.011	0.005	0.016	0.61	12.4
18	AT-335*	140-165	10M	53.8	0.043	0.037	0.080	2.38	12.0
19	AT-335*	165-265	10M	70.8	0.017	0.007	0.024	0.95	6.2
20	CAT-58**	545-665	35M	94.7	0.270	0.015	0.285	0.18	2.0
20	CAT-58	545-665	35M	94.2	0.131	0.008	0.139	0.01	3.7
21	CAT-59**	600-775	35M	91.2	0.135	0.013	0.148	2.69	5.4
21	CAT-59	600-775	200M	92.6	0.126	0.010	0.136	2.89	5.3
22	CAT-59*	605-665	35M	93.2	0.440	0.032	0.472	2.51	5.2
22	CAT-59	605-665	200M	95.5	0.321	0.015	0.336	2.20	4.1

Leach time for all bottle roll tests 4 days.
* Feed size as received drill cuttings nominally 35M.
** Feed size 95% -35M.

As summarized in Table 13.7, composite 1 was readily amenable to simulated heap leach cyanidation treatment at 80 percent –1/2 inch feed size, achieving 84.3 percent gold recovery in 40 days. Composite samples 6, 8, and 9 were moderately amenable to heap leach treatment at 80 percent –1/2 inch feed size. Gold recoveries from those composites were 65.0, 53.8, and 60.0, respectively. Longer leaching cycles are likely to improve gold recoveries slightly. Cyanide consumptions were generally low and lime added to ore charges prior to leaching was sufficient to maintain protective alkalinity. The column tests were performed on the Gold Pan and the Altenburg Hill samples only.

Table 13.7: Summary of Column Leach Test Results (McClelland, 1996)

				Leach	Au	-------ozAu/t ore------			Reagent Consump.	
Comp.	**Hole**	**Interval**		**Feed**	**Time**	**Recov.**			**Calc.**	
	Lb/ton ore									
No.	**No.**	**Feet**	**Size**	**days**	**%**	**Ext'd**	**Tail**	**Head**	**NaCN**	**Lime**
1	CAT-60	80-135	-1/2"	40	84.2	0.016	0.003	0.019	0.52	6.5
4	CAT-60	180-235	-1/2"	40	40.9	0.009	0.013	0.022	0.71	3.5
6	CAT-61	160-200	-1/2"	44	65.0	0.013	0.007	0.020	1.02	4.0
8	CAT-62	0-35	-1/2"	41	53.8	0.014	0.012	0.026	0.75	4.5
9	CAT-62	35-110	-1/2"	43	60.0	0.012	0.008	0.020	0.66	2.5

13.3 LEACH TESTS RESULTS

Results from the cyanide leach tests carried out by Amax (1993-94) indicate that oxidation and cyanide soluble copper content are the most important geological variables affecting process operating costs. Generally, greater oxidation of ore-grade material results in a higher gold extraction. The cyanide soluble copper content (average 185 ppm) does not present gold recovery problems, but does slow the rate of gold extraction and results in higher reagent consumption. From the process side, gold extractions are highly influenced by crush size. For the purposes of process scoping studies used in the 1994 Feasibility Study, a nominal ½ inch crush size, 1.1 pounds/ton of NaCN and 3.0 pounds/ton of lime were used to design optimal heap leach recovery. Heap recoveries were estimated to be 72 percent for oxide ore, 67 percent for mixed oxide ore, and 60 percent for sulfide ore, with an overall recovery estimated to be 67 percent.

Results from preliminary metallurgical testing carried out on behalf of Amax (1996) on drill samples from the Gold Pan, Altenburg Hill and the 39A Zone mineral resources indicate that most of the composite samples tested respond well to bulk sulfide flotation at 35 mesh and direct agitation cyanide leach tests at 10M feed size. In addition, four of the five composite samples used in the column leach tests are amenable to heap leaching at 80 percent –1/2 inch. Three of 10 composite samples from CAT-60 drilled in the Gold Pan Zone respond poorly to direct agitation at all feed sizes suggesting possible encapsulation and (or) mineralogical problems. Similar results were returned from CAT-62 drilled on the far west edge of the Altenburg Hill resource. The poor results retuned from these two holes negatively impacted the overall metallurgical studies. It should also be noted that 63 percent of all 1996 metallurgical tests were on composite samples from only three core holes, CAT-60 through CAT-62. These holes are unlikely to be representative of the mineral resources from which they were obtained and results should be viewed as very preliminary in nature.

13.3.1 Distal, 39A and Altenburg Hill

The 2007 laboratory metallurgical test program was also conducted by McClelland Labs of Sparks, Nevada. This program was started in late 2006 and completed in 2007.

The samples tested were assay rejects from the 2006 RC drill cuttings. The 2006 exploration program focused on the 39A, Distal and Altenburg Hill deposits.

The 39A composites tested consisted of high and low grade composites. The low grade composites were further broken down into shallow near surface mineralization and deep low grade mineralization.

The Distal composites were considered deep while the Altenburg Hill composites shallow for the laboratory testing methods.

The shallow composites were tested for heap leach amenability whiles the deep for whole ore milling followed by cyanidation and whole ore milling followed by flotation. For the high grade composites, gravity was included in the test procedure in addition to whole ore milling and flotation.

In total, 25 RC drill cuttings composites were tested. There were 5 high grade samples (designated HG1 to 5), 2 low grade shallow (LG1 and LG2) and 4 low grade deep (LG3, 5,6 and 7) from the 39A; 5 from the Distal and 9 from the Altenburg Hill deposits.

13.3.2 Sample Selection and Composite Make-up Information,

The sample selection and composite make up was based on processing criteria. For the shallow near surface deposits, the preferred process specification for testing was its amenability to heap leaching while for the deeper mineralized zones, testing was focused on whole ore treatment.

The mineralized zones were identified from the drill logs and if the grade and location from surface were on interest for testwork, the drill hole and intercepts were noted for those sample intervals to be set aside for metallurgical testing. In total there were 25 composites selected for this program. The composite make-up information together with the drill hole numbers, the lab received weights and the sample preparation procedure are provided in McClelland 2007 report. The drill intercepts used can be found in the drill logs and assay report.

A composite from Hole CR-06-26 was selected as an extra for testing as it represented the northern-most hole in the 39A zone to return interesting grade material.

Another composite from Drill Hole CR-06-09 was proposed for testing but the wrong sample numbers were shipped. Hence testing of this composite was omitted from this program.

13.3.3 Laboratory Test Conditions

13.3.3.1 Gravity Procedure

This procedure was used on the five high grade composites only. Each sample (10 kg) was stage ground, using a lab ball mill to 80%-75um in size. Each milled sample was subjected to gravity concentration by passing once through a Knelson (KC-MD3) concentrator. The resulting gravity rougher concentrate was cleaned once by hand panning to produce a gravity cleaner concentrate and gravity cleaner tailings. The gravity cleaner concentrate was dried, weighed and assayed to determine gold content. The gravity cleaner tailings and the gravity rougher tailings were each dried and weighed, and the tailings weight distributions were calculated. Each of the gravity cleaner tailings samples and the gravity rougher tailings samples were blended and split to obtain properly weighted samples for reconstituting a 2.5kg recombined gravity tailings sample for flotation testing. The splits from each sample were recombined, and used as feed for a batch flotation test.

13.3.3.2 Bulk Sulfide Flotation Test Procedures and Conditions

The bulk sulphide flotation test procedure was used on the recombined gravity tailings and on splits of whole ore from the 39A deep composites and the Distal composites. The whole ore samples were staged ground to 80% -75 microns in a laboratory steel ball mill while the recombined gravity tailings were subjected to a polishing grind of 2 minutes in a lab steel mill.

Flotation was conducted using a Denver laboratory scale flotation unit at 1200rpm. The ground pulp was slurried with water to achieve 30 weight percent solids and conditioned for 10 minutes with 0.25kg/t $CuSO_4$. Flotation was conducted in 5 stages with incremental additions of 0.005kg/t of each PAX, Aero 208 and 3477 promoters. Total addition of each reagent was 0.025kg/t ore. Aerofroth 65 was used as a frother. The pulp was floated at natural pH. The 5 stages of concentrate were combined into a rougher concentrate. The rougher concentrate was cleaned once to produce a cleaner concentrate and cleaner tail. No additional reagents were added during cleaner flotation. The two float products were dried, weighed and assayed to determine precious metal content. The rougher tails were dried, weighed and assayed directly in triplicate to determine residual precious metal content.

13.3.3.3 Direct Agitated Cyanidation Test Procedures and Conditions

Direct agitated cyanidation (bottle roll) tests were conducted on all 25 composites to determine gold recovery, recovery rate and reagent requirements. Tests were conducted on the 39A low grade shallow composites (LG1 and LG2) and on all nine Altenburg Hill composites at the received (nominal 1.7mm) feed size to determine amenability to heap leach cyanidation treatment. Tests were conducted on the 39A high grade composites, 39A low grade deep composites and the Distal composites at an 80% - 75 micron feed size to determine amenability to milling/cyanidation treatment.

The 75 micron feeds were stage ground using laboratory steel ball mills. Ore charges were mixed with water, or settled in grinding water, to achieve 40 weight percent solids. Natural pulp pHs were measured. High calcium hydrated lime was added to adjust the pH of the pulps to 11.0 before adding the cyanide. Sodium cyanide, equivalent to 1.0g NaCN/L of solution was added to the alkaline pulps.

Leaching was conducted by rolling the pulps in bottles on the laboratory rolls for either 96 hours (as-rec'd feeds) or 72 hours (75 micron feeds). Rolling of the as-received slurries was suspended briefly after 2, 6, 24, 48, and 72 hours to allow the pulps to settle so samples of pregnant solution could be taken for gold and silver analysis by AA methods. Pregnant solution volumes were measured and sampled. Cyanide concentration and pH were determined for each pregnant solution. Make-up water, equivalent to that withdrawn, was added to the pulps. Cyanide concentrations were restored to initial levels. Lime was added, when necessary, to maintain the leaching pH at between 10.8 and 11.2. Rolling was then resumed.

Leaching procedures for the 75 micron feeds were essentially the same, except that a 12 hour sampling interval was added and the 96 hour sampling interval was deleted. Slurry dissolved oxygen levels were also measured at each sampling interval. Measured D.O. levels generally were above 6.0 mg/L, and never were lower than 4.0 mg/L.

After 72 or 96 hours, the pulps were filtered to separate liquids and solids. Final pregnant solution volumes were measured and sampled for gold and silver analysis. Final pH and cyanide concentrations were determined. Leached residues were washed, dried, weighed, and assayed in triplicate to determine residual precious metal content.

13.3.4 Results

13.3.4.1 Gravity/ Flotation Results

Summary results of the gravity and flotation results are presented in Table 13.8:

Table 13.8: Summary Metallurgical Results, Gravity and Flotation Tests, 80%-75um Feed Size

	Weight Distribution %				Assays, gAu/mt					Au Distribution %			
	Gravity	Flotation			Gravity	Flotation			Calc'd	Gravity	Flotation		
Composite	Cl Conc	Cl Conc	Cl Tail	Ro Tail	Cl Conc	Cl Conc	Cl Tail	Ro Tail	Head	Cl Conc	Cl Conc	Cl Tail	Ro Tail
HG1 Comp	0.25	2.02	2.73	95	1087	60	4.15	0.2	4.23	64.3	28.5	2.7	4.5
HG2 Comp	0.16	0.91	2.86	96.07	1529	68	14.7	0.13	3.61	67.7	17.2	11.6	3.5
HG3 Comp	0.16	0.59	2.36	96.89	905	133	0.66	0.14	2.38	60.8	32.8	0.7	5.7
HG4 Comp	0.22	2.2	2.18	95.4	1135	80	0.6	0.2	4.47	55.9	39.5	0.3	4.3
HG5 Comp	0.33	4.63	1.74	93.33	562	49	8.67	0.42	4.48	37.7	50.2	3.4	8.7
LG3 Comp	n/a	1.8	3.7	94.5	n/a	49	1.83	0.12	1.06	n/a	82.9	6.4	10.6
LG5 Comp	n/a	3	6.2	90.8	n/a	18	1.38	0.08	0.71	n/a	77.8	12.1	10.3
LG6 Comp	n/a	1.7	2	96.3	n/a	123	10.83	0.14	2.44	n/a	85.6	8.9	5.5
LG7 Comp	n/a	1.2	2.9	95.9	n/a	144	7.12	0.08	2.01	n/a	85.9	10.3	3.8
Distal Comp CR06-16	n/a	2.9	5.8	91.3	n/a	41	1.01	0.17	1.41	n/a	84.8	4.2	11
Distal Comp CR06-17	n/a	6.1	5.7	88.2	n/a	11	1.23	0.12	0.82	n/a	78.6	8.5	12.9
Distal Comp CR06-18	n/a	4.4	5	90.6	n/a	53	0.99	0.21	2.58	n/a	90.7	1.9	7.4
Distal Comp CR06-20	n/a	6.4	6.8	86.8	n/a	29	0.58	0.33	2.2	n/a	85.2	1.8	13
Distal Comp CR06-20	n/a	4.2	3	92.8	n/a	42	1.06	0.11	1.88	n/a	92.9	1.7	5.4

The five high grade composites from 39A responded very well to whole ore gravity concentration followed by flotation at a grind size of 80% passing 75 microns. Gold recovery by gravity after cleaning ranged from 38 to 68% and the combined gravity/flotation recovery ranged from 91 to 96%.

Gold values reporting to the flotation rougher concentrate represented between 29% and 54% (additional to gravity recoveries) of gold values contained in the ore.

Gravity cleaner concentrates ranged from 562 - 1,529gAu/t (16 - 45 ozAu/t). Flotation cleaner concentrate grades ranged from 49 - 133gAu/t (1.4 - 3.9 ozAu/t). Flotation rougher (final) tail grades ranged from 0.13 to 0.42gAu/t.

Gravity cleaner concentrate weights ranged from 0.16% to 0.30% of the ore weight. Flotation cleaner concentrate weights ranged from 0.59% to 4.63% of the ore weight. Flotation cleaner tails ranged from 1.7% to 2.8%. Combined concentrate weights ranged from 3.9% to 6.7% of the ore weight.

The low grade deep composites from the 39A and Distal deposits also responded well to whole ore milling with bulk sulfide flotation treatment at a grind size of 80% passing 75 microns.

The four from 39A provided rougher concentrate grades between 7 and 62 g Au/t with 3.7% to 9.2% of the feed weight and represented gold recoveries of between 89.4% and 96.0%. The five from Distal provided similar results with mass recoveries to the rougher flotation concentrate of 7.2% to 13.2%, assayed between 6 and 25g Au/t and represented gold recoveries of between 87% and 95%. Flotation cleaner concentrate grades ranged from 11 to 53g Au/t.

13.3.4.2 Cyanidation Bottle Roll Results

These results are summarized and presented in Table 13.9:

Table 13.9: Summary Metallurgical Results - Bottle Roll Tests

Composite	Feed Size	Au Recovery %	Extracted	Tail	Calc'd Head	Assayed Head	NaCN Cons.	Lime Added
39A HG1 Comp	P8075um	98.7	6.22	0.08	6.3	5.47	0.08	0.9
39A HG2 Comp	P8075um	98.7	4.46	0.06	4.52	4.38	0.15	1
39A HG3 Comp	P8075um	97.9	2.81	0.06	2.87	3.17	0.12	1
39A HG4 Comp	P8075um	98.4	4.28	0.07	4.35	4.11	0.39	1
39A HG5 Comp	P8075um	97.4	4.45	0.12	4.57	5.09	0.31	1.2
39A LG1 Comp	As Rec'd (1.7mm)	93	0.8	0.06	0.86	0.4	<0.07	1.6
39A LG2 Comp	As Rec'd (1.7mm)	91.8	0.67	0.06	0.73	0.8	0.08	2.2
39A LG3 Comp	P8075um	96.3	1.29	0.05	1.34	1.41	0.15	1.9
39A LG5 Comp	P8075um	92.6	0.5	0.04	0.54	0.54	0.08	1.3
39A LG6 Comp	P8075um	97.8	2.7	0.06	2.76	2.67	0.18	1.3
39A LG7 Comp	P8075um	97.8	1.82	0.04	1.86	1.55	0.1	1.1
Distal Comp CR06-16	P8075um	78.9	1.38	0.37	1.75	1.29	0.37	3.2
Distal Comp CR06-17	P8075um	76.9	0.6	0.18	0.78	0.58	0.65	2.8
Distal Comp CR06-18	P8075um	93.7	4.17	0.28	4.45	1.85	0.51	2.6
Distal Comp CR06-18	P8075um	91.6	4.38	0.4	4.78	1.85	0.38	2
Distal Comp CR06-20	P8075um	87.7	2.22	0.31	2.53	1.83	0.39	2.3
Distal Comp CR06-26	P8075um	92.5	1.47	0.12	1.59	1.88	0.46	2
Altenburg Hill Comp CR 06-30	As Rec'd (1.7mm)	76.4	0.68	0.21	0.89	0.52	0.26	2.1
Altenburg Hill Comp CR 06-34	As Rec'd (1.7mm)	60.8	0.31	0.2	0.51	0.26	0.3	1.9
Altenburg Hill Comp CR 06-35	As Rec'd (1.7mm)	73.3	0.33	0.12	0.45	0.23	0.15	2.1
Altenburg Hill Comp CR 06-36	As Rec'd (1.7mm)	68.8	0.33	0.15	0.48	0.5	0.08	1.8
Altenburg Hill Comp CR 06-37	As Rec'd (1.7mm)	66.7	0.5	0.25	0.75	0.42	0.23	1.8
Altenburg Hill Comp CR 06-38	As Rec'd (1.7mm)	70.7	0.29	0.12	0.41	0.27	0.21	1.9
Altenburg Hill Comp CR 06-39	As Rec'd (1.7mm)	58.9	0.99	0.69	1.68	1.18	0.1	1.7
Altenburg Hill Comp CR 06-40	As Rec'd (1.7mm)	73.9	0.51	0.18	0.69	0.41	0.23	1.8
Altenburg Hill Comp CR 06-41	As Rec'd (1.7mm)	54.3	0.25	0.21	0.46	0.24	0.15	2.3

The 39A high grade composites were readily amenable to whole ore milling followed by cyanidation at a grind of 80% passing 75 microns. Gold recoveries ranged from 97% to 99% in 72 hours of leaching. Gold recovery rates were rapid and extraction was substantially complete in 24 hours. Reagent requirements were low. These results are comparable to the 1996 met test results from two core holes drilled in the 39A Zone. Head assay results showed that all the high grade composites contained less than 7 gAg/t ore.

The 39A low grade deep composites also responded favourably to whole ore milling followed by cyanidation at a grind of 80% passing 75 microns. Gold recoveries ranged from 93% to 98% in 72 hours of leaching. The behaviour of these composites is similar to the high grade composites where gold recovery rates were rapid and extraction was substantially complete in 24 hours. The reagent requirements were also low and there was good agreement between the calculated and the assayed head assays.

The 39A near surface composites were readily amenable to direct cyanidation treatment on the as received (approx. 1.7mm) feed size. Gold recoveries for LG-1 and LG-2 were 93% and 92% respectively in 96 hours of leaching. Gold recovery rates were rapid and the reagent requirements were low.

The Altenburg Hill composites also responded favourably to direct cyanidation treatment on the approximately 1.7mm feed size but the gold recoveries were lower than the 39A low grade composites. They ranged from 54% to 76%, and averaged 67% in 96 hours of leaching based on the calculated head. Gold recovery rates were moderate and a longer leaching cycle would improve recoveries. Reagent requirements were low. These results are similar to those obtained in 1996 met test results on the core and RC samples at the test size of 10mesh.

The Distal composites varied in their response to whole ore milling followed by cyanidation treatment at an 80% passing 75 microns feed size. Gold recoveries ranged from 77% to 94%, and averaged 87%, in 72 hours of leaching. Gold recovery rates were rapid, and extraction was substantially complete after 24 hours. Reagent requirements were low but were higher than for the 39A composites.

13.3.5 Summary of Historical Laboratory Studies

Metallurgical testwork completed to date suggests that the Porphyry, Altenburg Hill and the near surface deposits of the Gold Pan and 39A deposits are amenable to heap leach treatment. Optimization of the crush size would be required in future programs together with a column test on an overall or representative composite once the optimum crush size has been defined and selected.

No soluble copper problems were noted or identified in the 2007 test program as most reagent requirements in the cyanidation series of tests were low. The work suggests the Altenburg Hill and the 39A mineralized zones may not have as much soluble copper content as the Porphyry.

There are more processing options available for the deeper deposits of 39A and Distal. In addition to whole ore cyanidation where the best gold recoveries were achieved, these deposits are also amenable to flotation as well. Should the process selection lean towards heap leaching for the treatment of these two deep deposits, then there is the need to optimize the crush size in future test programs.

The 1996 test program on the Gold Pan deposit suggests that flotation is a viable process option for this deposit at depth with an increase in sulphide content. Further flotation work would be required for this deposit and the zone of oxidation from the sulphides better defined for the selection of test composites.

There were several unusual observations noted when testing the Distal composites. Composite CR-06-18 used in the bottle roll cyanidation test had to be repeated due to an unusually high calculated head grade when compared to the assayed head grade and the calculated head grade form the flotation test. This test was repeated and provided a similar result with poor head assay agreement.

Composite CR-06-16 had a very high silver head assay of 8gAg/t compared to the other Distal composites of less than 5gAg/t. This abnormality was noted when the solution assays from the bottle roll tests were reviewed and compared to the other pregnant solution assays.

These two occurrences on the Distal composites suggests that there may be coarse gold and native silver in this deposit. In addition, the reagent requirements for the Distal were slightly higher than for the 39A. Mineralogy on the Distal is therefore highly recommended in any future test program and comparison should be made with those of 39A as these are deep deposits.

On the Altenburg Hill bottle roll cyanidation tests, the calculated heads in general were higher than the assayed heads. It is not known if this is a lab procedure issue or an assaying issue. Confirmatory testing for this series of tests would be recommended together with the longer leach time.

The flotation recoveries in general were slightly lower than the whole ore cyanidation results for the deep composites tested. Since the flotation tests were performed on RC drill cuttings, these flotation results could be slightly bias to the low side on account of surface oxidation of the sulphides. Confirmatory flotation tests with core drilling are better and should be considered in future test programs.

There is no documentation or reports on the metallurgical characteristics of the Triplet Gulch Deposit. Any future test program should include some preliminary work should this deposit become of interest for development.

13.3.6 2011 Laboratory Study

A laboratory program was performed in 2011 at McClelland Laboratories undertaken on samples obtained from the Altenburg Hill and Gold Pan deposits. The samples originated from continuous drill core intervals obtained during the 2010 exploration program. Portions of the samples were initially composited for use in bottle roll studies, followed by further re-blending of coarser material for column leach testing. A laboratory report by McClelland labs was not available at the time of issuing this report, but the interim resulting data is discussed below. A final laboratory report is expected by year end 2011 for this program.

13.3.6.1 Initial Composites

The initial samples were composited for use in bottle roll testing based on their location and head characteristics. Sample identification was based on a prefix AH for Altenburg Hill and GP for the Gold Pan deposit, followed by the drill hole number. Drill hole numbers and intervals used to make up the various composites will be included in the McClelland report expected to be issued in late December 2011. Based on head assays the material was classified as either oxide, partially (potentially) oxidized, or as sulfide material and identified respectively with a suffix OX, POX and SUL.

There were eight oxide composites originating from Altenburg Hill, and two composites classified as partially oxidized from the Gold Pan deposit. Each was subjected to bottle roll testing. The corresponding head assays of these composites are provided in Table 13.10.

Table 13.10: Head Analyses Oxide and Partial Oxidized Composites

Composite	Au		Ag	Cu	S_T	S^{2-}
ID	g/tonne	oz/ton	ppm	ppm	%	%
AH-CC10-3-OX	0.68	0.020	1.5	460	0.02	0.02
AH-CC10-4-OX	0.96	0.028	5.5	760	0.03	0.02
AH-CC10-5-OX	1.1	0.032	2.4	636	0.02	0.01
AH-CC10-6-OX	0.66	0.019	1.4	396	0.01	0.01
AH-CC10-7-OX	1.37	0.040	5.1	1,425	0.02	0.01
AH-CC10-8-OX	0.72	0.021	3.1	548	0.01	0.01
AH-CC10-9-OX	0.69	0.020	1.7	372	0.01	0.01
AH-CC10-10-OX	0.73	0.021	3.7	1,495	0.18	0.14
GP-CC10-13-POX	1.9	0.055	7.5	577	0.60	0.03
GP-CC10-15-POX	0.34	0.010	<0.5	351	0.29	0.19

The gold grades for these composites varied from 0.34 g/t to 1.9 g/t (0.010 oz/t to 0.055 oz/t). There was low associated silver, which increased slightly with a higher gold content. The copper content ranged from 350 to 1500 ppm. With one exception (AH-CC10-10-OX) the total sulfur content (S_T) for the oxide material was less than 0.03%. Total sulfur content in the POX material was significantly higher, and a portion of these sulfides had been oxidized based on comparing total sulfur to the sulfide sulfur (S^{2-}) analyses.

The initial metallurgical composites classified as sulfides are listed along with corresponding head analyses in Table 13.11.

Table 13.11: Head Analyses Sulfide Composites

Composite	Au		Ag	Cu	S_T	S^{2-}
ID	g/tonne	oz/ton	ppm	ppm	%	%
AH-CC10-3-SUL	0.74	0.022	2.0	1,110	0.63	0.50
AH-CC10-4-SUL	0.85	0.025	3.5	1,055	0.24	0.14
AH-CC10-6-SUL	0.54	0.016	2.0	532	0.44	0.32
AH-CC10-8-SUL	0.55	0.016	1.5	316	0.18	0.13
GP-CC10-12-SUL	0.86	0.025	2.6	638	1.89	1.65
GP-CC10-13-SUL	0.73	0.021	0.6	977	3.88	3.25
GP-CC10-14-SUL	0.63	0.018	1.0	507	1.88	1.68
GP-CC10-15-SUL	0.72	0.021	1.1	626	1.78	1.55

Precious metal head grades for sulfide composites varied mid range as compared to the previous oxide samples with gold contents of 0.54 g/t to 0.86 g/t (0.016 oz/t to 0.025 oz/t). Copper grades were similar for both oxides and sulfides. For the sulfide composites the Gold Pan had significantly higher sulfur content than those of Altenburg Hill. Gold Pan and Altenburg composites had similar extent of oxidation based on the sulfide to total sulfur ratio.

A final set of three composites was prepared on material with low gold grades and originating from rock that would be stripped or mined to access the ore. The sub-samples used for these composites were blended together based on anticipated gold grade (low, medium, and high). These samples were labeled as overburden (OB) for the lab program but are not organic or alluvial and are in competent rock, originating from drill holes overlying or adjacent to the higher grade ore zones at both Altenburg Hill, and Gold Pan, along with one drill hole located in the south Porphyry area.

The gold content in these OB composites is currently below that envisioned to be suitable for crushing and leaching on prepared pads. The material may be suitable for dump leaching and consequently was evaluated for cyanidation response in the testing program. A summary of the head assays for the OB composites is provided in Table 13.12.

Table 13.12: OB Composites Head Analyses

Composite	Au		Ag	Cu	S_T	S^{2-}
ID	g/tonne	oz/ton	ppm	ppm	%	%
OB-Comp-Low	0.28*	0.008	3*	668	0.30	0.22
OB-Comp-Med	0.28*	0.008	3*	377	0.98	0.76
OB-Comp-High	0.28*	0.008	2*	416	0.10	0.08
* Average of triplicate direct assay.						

Despite an attempt to segregate the three samples based on grade from the exploration geochemical gold analyses, the three samples had very similar gold and silver content, as averaged from triplicate assay splits. The medium grade sample had higher sulfide content.

13.3.6.2 Bottle Roll Testing

A series of thirty bottle roll tests were performed on the initial composites, with some of the composites subjected to alternate procedures including the use of carbon in leach (CIL) methods to evaluate any potential preg robbing effect. Baseline bottle roll procedures including crushing to a particle size of 80% passing 1.7 mm (10 Tyler mesh). Bottle roll testing was conducted under protective alkalinity of pH 10.5 to 11 for 96 hours maintaining 1.0 g/L cyanide at approximately 40% solids.

The first set of 8 bottle rolls were performed on the Altenburg Hill oxide composites, with the results summarized in Table 13.13

Table 13.13: Altenburg Hill Oxide Composites –Bottle Roll Gold Leach Response

Composite	Recovery,	Extracted	Tail	Head (Au g/t)	
ID	% Au	Au, g/t	Au, g/t	Calc.	Assay
AH-CC10-3-OX	60.0	0.39	0.26	0.65	0.68
AH-CC10-4-OX	58.5	0.62	0.44	1.06	0.96
AH-CC10-5-OX	73.0	0.65	0.24	0.89	1.1
AH-CC10-6-OX	63.2	0.43	0.25	0.68	0.66
AH-CC10-7-OX	72.3	1.12	0.43	1.55	1.37
AH-CC10-8-OX	66.7	0.42	0.21	0.63	0.72
AH-CC10-9-OX	88.7	0.47	0.06	0.53	0.69
AH-CC10-10-OX	63.9	0.46	0.26	0.72	0.73

Gold extractions for the various samples ranged from 58% to 88% within the 96 hour leach retention time. Detailed results the majority of the gold went into solution within the first 24 hours with dissolution nearing the maximum when leaching was terminated. The calculated and assayed heads for gold appear to be reasonably close. Tailing gold grades were variable between 0.06 g/t to 0.44 g/t. Higher losses roughly correlated to the increased head grades, indicating there may be a benefit to increase leach retention times for higher head grades.

Silver recoveries were 26% to 55% on heads of 2-7 g/t Ag. Soluble copper was typically less than 10%, except for composite AH-CC10-10-OX, which had 17% copper dissolution. This was likely due to an elevated sulphate content as compared to the other oxide samples. Soluble copper did not appear to adversely affect precious metal recovery, but cyanide consumption increased to 0.74 kg/tonne for sample AH-CC10-10-OX as compared to an average of 0.17 kg/tonne for the other seven oxide composites. Lime consumption was reasonably uniform for all samples in a range of 1.5 to 2.3 kg/tonne.

The next set of bottle roll leach tests discussed are on the two partially oxidized composites Gold Pan composites as shown in Table 13.14. Composite GP-CC10-15-POX was repeated in triplicate due to disagreement between the assayed and calculated heads.

Table 13.14: Gold Pan Deposit Partially Oxidized Composites
Bottle Roll Gold Leach Response

Composite	Rerun	Recovery,	Extracted	Tail	Head (Au g/t)	
ID		% Au	Au, g/t	Au, g/t	Calc.	Assay
GP-CC10-13-POX	n/a	85.1	1.43	0.25	1.68	1.9
GP-CC10-15-POX	1	81.5	0.66	0.15	0.81	0.34
GP-CC10-15-POX	2	78.4	0.58	0.16	0.74	0.34
GP-CC10-15-POX	3	90.6	0.58	0.06	0.64	0.34

If averaging the 3 tests for GP-CC10-15-POX, the bottle roll results gave gold extractions in the mid eighty percent range. The three tests repeated on GP-CC10-15-POX all gave calculated heads close to double the assay head. The tailing grade also showed considerable fluctuation resulting in a 12% variation in gold recovery using the same sample and conditions. This sample also showed elevated sulfide content and approximately 25% copper dissolution during cyanidation that may be contributing to the variability of the sample. The average lime and cyanide consumptions of the three tests performed on GP-CC10-15-POX were similar respectively averaging ~3.2 kg/t and 0.4 kg/t. Leach profiles showed minor increases to precious metal dissolution when the testing was terminated at 96 hours. Further work is required to better understand the reasoning behind the variability in composite 15. GP-CC10-13-POX had a gold recovery of 85% in part owing to the higher head of 1.68 g/t (0.049 oz/t). It showed elevated lime consumption of 7.1 kg/tonne, but a moderate cyanide consumption of 0.26 kg/t.

The results for bottle roll testing on the Altenburg Hill sulfide samples is provided in Table 13.15. Each of the four composites was run using standard baseline bottle roll procedures, as well as a second method consisting of carbon in leach (CIL) as a means of investigating preg robbing potential.

Table 13.15: Altenburg Hill Sulfide Composites –Bottle Roll Gold Leach Response

Composite	Method	Recovery,	Extracted	Tail	Head (Au g/t)	
ID		% Au	Au g/t	Au g/t	Calc.	Assay
AH-CC10-3-SUL	CN	70.5	0.43	0.18	0.61	0.74
AH-CC10-3-SUL	CIL	66.1	0.39	0.20	0.59	0.74
AH-CC10-4-SUL	CN	70.4	0.57	0.24	0.81	0.85
AH-CC10-4-SUL	CIL	68.8	0.53	0.24	0.77	0.85
AH-CC10-6-SUL	CN	66.2	0.45	0.23	0.68	0.54
AH-CC10-6-SUL	CIL	60.8	0.31	0.20	0.51	0.54
AH-CC10-8-SUL	CN	57.4	0.35	0.26	0.61	0.55
AH-CC10-8-SUL	CIL	53.2	0.33	0.29	0.62	0.55

The gold extractions for the Altenburg Hill sulfide composites varied between 53% to 70%, which are slightly lower than the oxide samples discussed previously. This is likely in part due to the lower head grades and slower leach kinetic response. Tailing grades were in a similar, but narrower range than shown for the oxide composites. The use of CIL procedures did not show improved gold extraction, indicating a preg robbing component due to gold re-adsorption is not present. Inexplicably cyanide and lime consumption were consistently elevated in CIL verses the standard cyanide procedures. Cyanide consumptions with standard procedures ranged from 0.3 to 0.8 kg/t, which is higher than with the Altenburg Hill oxides. Lime consumption for the Gold Pan sulfides is in a range of 1.4 to 2.2 kg/t, similar to the oxides.

Bottle roll tests were also performed on four Gold Pan sulfide composites comparing CIL to standard leach procedures. The results as outlined in Table 13.16.

Table 13.16: Gold Pan Deposit Sulfide Composites
Bottle Roll Gold Leach Response

Composite	Method	Recovery,	Extracted	Tail	Head (gAu/t)	
ID		% Au	Au g/t	Au g/t	Calc.	Assay
GP-CC10-12-SUL	CN	56.1	0.37	0.29	0.66	0.86
GP-CC10-12-SUL	CIL	58.1	0.43	0.31	0.74	0.86
GP-CC10-13-SUL	CN	50.0	0.32	0.32	0.64	0.73
GP-CC10-13-SUL	CIL	50.0	0.32	0.32	0.64	0.73
GP-CC10-14-SUL	CN	59.4	0.38	0.26	0.64	0.63
GP-CC10-14-SUL	CIL	56.4	0.31	0.24	0.55	0.63
GP-CC10-15-SUL	CN	44.3	0.39	0.49	0.88	0.72
GP-CC10-15-SUL	CIL	51.2	0.44	0.42	0.86	0.72

The use of CIL did not show a significant improvement to gold recovery with one possible exception on sample GP-CC10-15. This gave a 0.07gAu/t lower tailing grade, transposing into a 7% increase in extraction. Most of the leaching was completed within 96 hours with the possible exception of composite 14. The final gold recoveries on the sulfide composites were lower as compared to the two partially oxidized composites evaluated for Gold Pan, and discussed previously. Again this is partly due to the lower head grades, but the tailing also contained higher corresponding gold losses. Cyanide consumptions were elevated ranging from 0.9 to 2.3 kg/t. Lime consumption for the Gold Pan sulfide composites varied from 1.7 to 2.2 kg/t, with one notable exception in GP-CC10-13 which was 6.3 kg/t.

The three overburden (OB) samples were run under the standard bottle roll conditions and results are summarized in Table 13.17.

Table 13.17: OB Composites –Bottle Roll Gold Leach Response

Composite	Recovery,	Extracted	Tail	Head (Au g/t)	
ID	% Au	Au, g/t	Au, g/t	Calc.	Assay
OB-Comp-Low	63.0	0.17	0.10	0.27	0.28
OB-Comp-Med	54.2	0.13	0.11	0.24	0.25
OB-Comp-High	59.3	0.16	0.11	0.27	0.28

Original head assay for OB-Comp-Med, showed 0.25 gAu/t, but follow-up analyses with triplicate fire assay gave 0.28 g/t for all three composites (see Table 13.12 previously). This sample also has the highest sulfide content of the three composites. The results indicate all three composites had a similar leach profile with relatively fast kinetics and with similar tailing grades. Calculated heads varied slightly resulting in a gold recovery range of 54% to 63%. Reagent consumption for the three composites was also similar ranging from 0.30 g/t to 0.53 kg/t for cyanide, and 1.9 kg/t to 2.5 kg/t for lime.

13.3.6.3 Bottle Roll Comparisons

Plots of gold recovery verses head grade, and verses sulfide content are shown respectively in Figures 13-1 and 13-2 for all of the oxide (OX), partially oxidized (POX) and sulfide (SUL) composites that were tested.

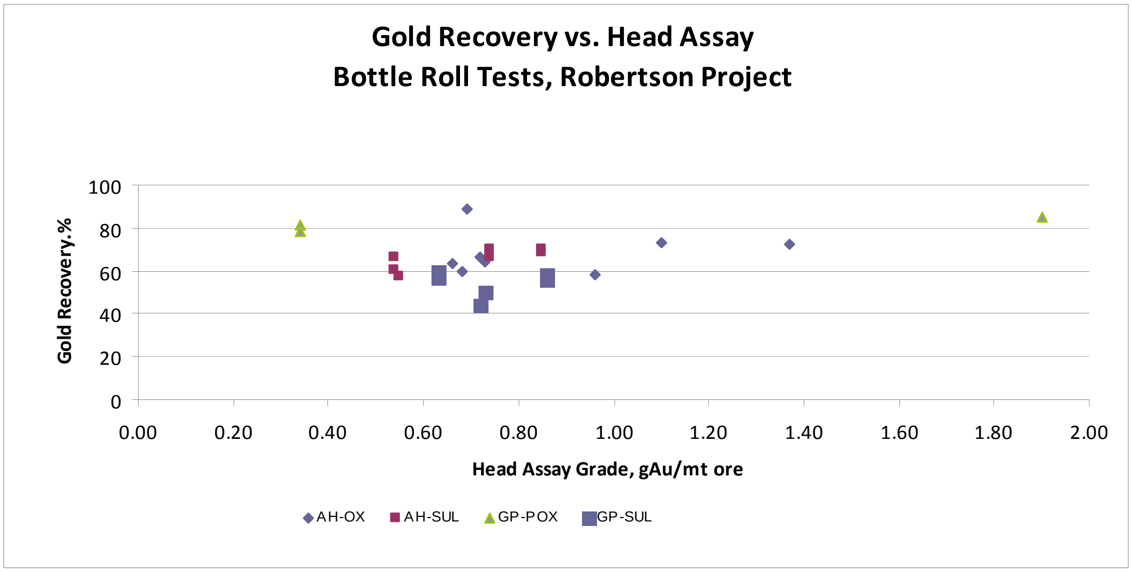


Figure 13-1*: Gold Recovery vs Head Grade Bottle Roll Tests

*from McClelland Laboratories Inc.

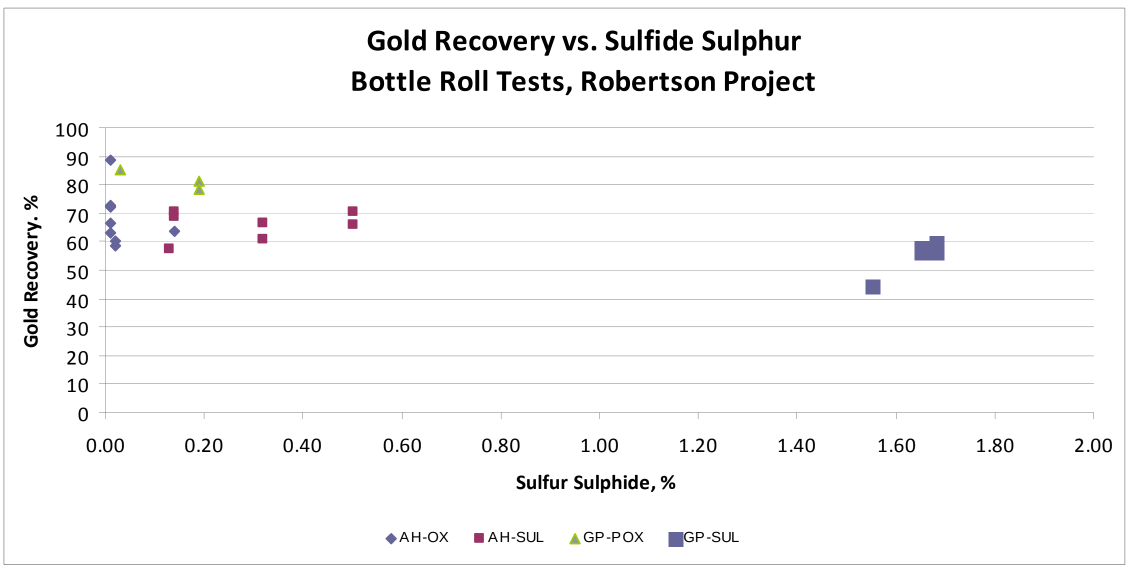


Figure 13-2*Gold Recovery vs Sulfide Sulphur Bottle Roll Tests

*from McClelland Laboratories Inc.

Comparisons within the various composite types indicate that head grade has a minor effect on recovery; and that the sulfide content likely plays a more significant roll. Higher head grade within composite types results in modestly higher recovery, and with higher tailing losses that may benefit from continued leach retention time. Generally higher sulfide content corresponds to lower gold recovery. Evaluation of the detailed lab report indicates that the organic carbon content and preg robbing factor may have played a related but minor role with the sulfide content. There does not appear to be a strong preg robbing characteristic for these samples.

Cyanide consumption was significantly influenced by extent of copper dissolution and the sulfide content in the feed. Increased copper dissolution and higher sulfide content translated into a marked increase in cyanide consumption as shown in Figures 13.3 and 13.4

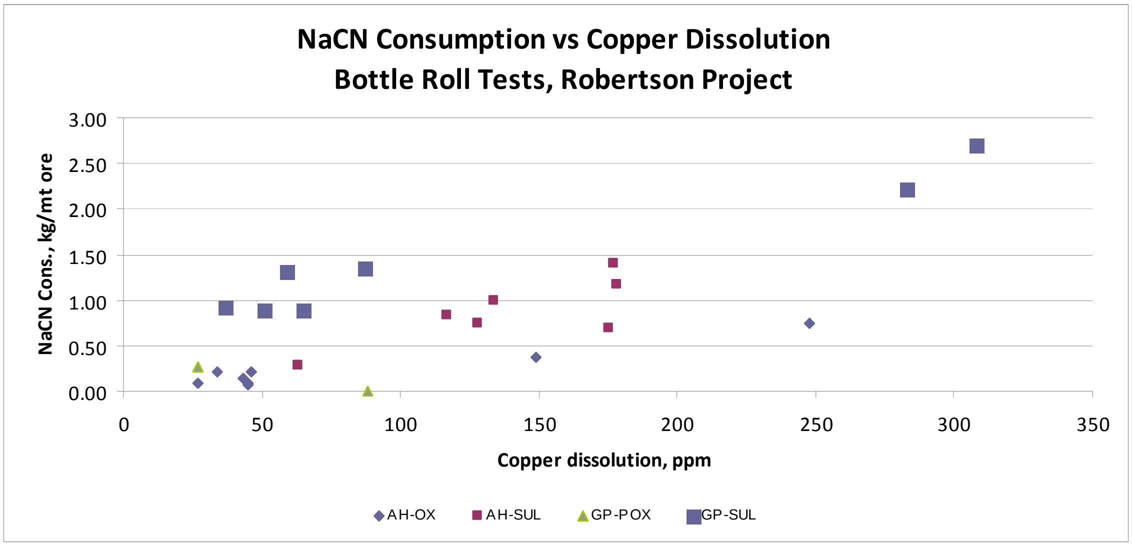


Figure 13-3*: NaCN Consumption vs Copper Dissolution

*from McClelland Laboratories Inc.

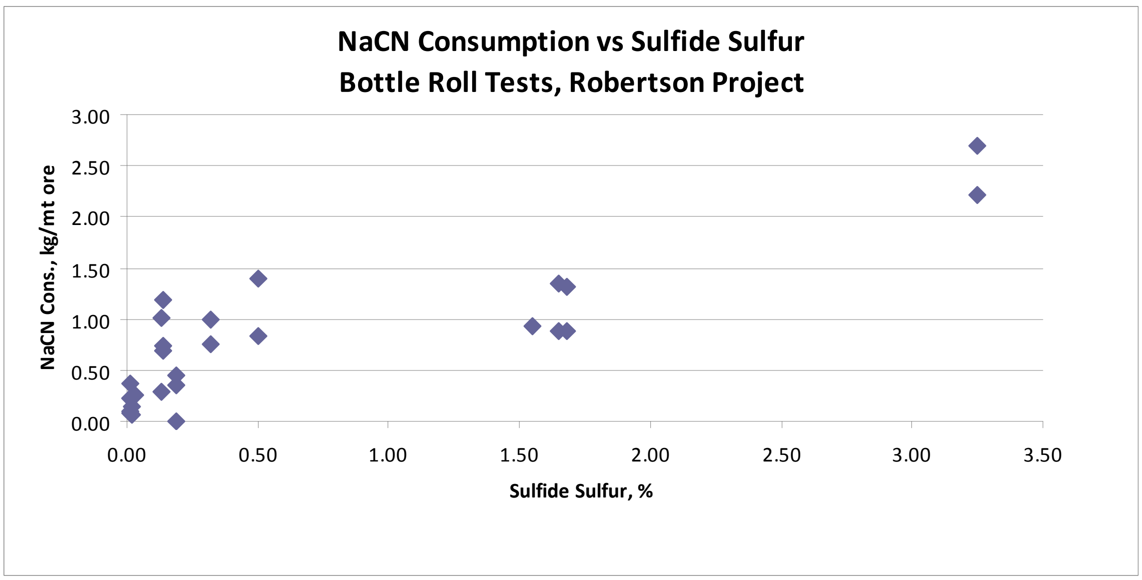


Figure 13-4*: NaCN Consumption vs Sulphide Sulphur

*from McClelland Laboratories Inc.

13.3.7 Column Leaching

13.3.7.1 Procedure

Following evaluation of the bottle roll test data, seven column leaching tests were performed on minus 19 mm (-3/4”) composites. The study included separate column tests on each of the three overburden samples (OB) composites, using splits of the same material that had been crushed to -10 Tyler mesh for bottle roll testing. The remaining composites representing oxidized (OX), partially oxidized (POX) and sulfide (SUL) ore samples (that were also used for bottle roll tests) were re-blended into new master composites for column leaching. Table 13.18 below provides a summary of the feed material used in each of the columns, as well as the height and dry weight of the feed, and the inside diameter of the column. A more detailed explanation of the methods follows the table.

Table 13.18: -3/4" Column Leach Composites

Column	Composite	Material	Head	Weight	Height	Dia
#	ID	Description	Au, g/t	Kg	m	cm
P1	OB-Comp-Low	same as bottle roll	0.28	127.3	2.59	20
P2	OB-Comp-Med	same as bottle roll	0.28	126.6	2.63	20
P3	OB-Comp-High	same as bottle roll	0.28	123.9	2.52	20
P4	RP-CC10-OX/POX	AH oxide plus GP partial oxide	0.92	146.2	2.96	20
P5	AH-CC10-SUL	Altenburg Hill (AH) sulfides	0.67	71.3	2.43	15
P6	GP-CC10-SUL	Gold Pan (GP) sulfides	0.74	73.3	2.63	15
P7	RP-CC10-OX/SUL	mixed oxide, POX, sulfides	0.87	71.5	2.59	15

Each of the columns were loaded with >95% passing 19 mm (3/4") rock. Based on the initial loaded weight and column volume the respective bulk density of the solids was calculated to range from 1.53 to 1.57 for the overburden, oxide and partially oxidized samples. The bulk density of the Altenburg Hill and Gold Pan sulphides were 1.64, and 1.56 respectively. Upon commencement of leaching the solution uptake to saturation ranged from 9.4% to 12.9% for the various columns. Particle size distribution was performed on each feed material and assayed head grades for the column were based on the weight and analyses of the screened fractions. Lime was added with the dried blended solids to the column and protective alkalinity was maintained throughout the leach. Agglomeration with cement was not incorporated.

The column leaching tests (P1, P2, P3) were respectively performed on the three overburden (OB) samples Low, Med, High, which effectively all had similar head grades of 0.28 gAu/t.

Test P4 was done on the eight oxidized composites from Altenburg Hill blended with the two partially oxidized composites from the Gold Pan resulting in an estimated grade of 0.92 gAu/t. Each of these 10 composites of P4 made up roughly equal mass (~10 wt.% ea.) for the 146 kg loaded into the column.

The remaining columns had less material available and used smaller diameter columns of 15 cm. Test P5 consisted of the same four Altenburg Hill sulphide composite samples that had been used in bottle roll testing. These were blended together for the column leach in approximately equal amounts of 25 wt.% each and totalled 71 kg with an estimated grade of 0.67 gAu/t. Similarly, the four Gold Pan sulphide composites were blended in roughly equal amounts to produce 73 kg of feed grading 0.74 gAu/t for test P6.

The seventh column (P7) used of combination of the eight Altenburg Hill oxide composites in equal portions of 7.5 wt.% each, accounting for 60% of the mass total loaded into the column. The remainder was made up from the two partially oxidized Gold Pan composites in equal portions accounting for an additional 15% of the loaded mass and the Gold Pan sulphides which made up the remaining 25%. The blended material weighed 71.5 kg was calculated to grade 0.87 g/t Au.

Each column was percolated with 1.0 g/L NaCN solution and routinely monitored to record flow and cyanide consumption. Hydrated lime provided protective alkalinity. The loaded pregnant leachate solutions (PLS) from the column underflow were passed through activated carbon. Gold and silver dissolution was routinely measured in the PLS, as well as the barren recycle solution, and ultimately balanced against the loaded carbon. Rest and rinse cycles were employed as deemed appropriate. At termination of leaching the column had a final wash and leached solids analyzed as part of the precious metal balance.

13.3.7.2 Results

The seven column tests (P1 to P7) were performed on -19 mm (-3/4”) feed material that consisted of;

- P1, P2, P3: three columns of low grade gold mineralization, labelled as overburden (OB).
- P4: consisting of oxide composites from Altenburg Hill (80% of weight) mixed with partially oxidized material from Gold Pan (20% of the weight).
- P5, P6: two columns on sulfide material composited respectively from the Altenburg Hill and Gold Pan deposits.
- P7: a blended column of Altenburg Hill oxide composites (60 wt.%) with Gold Pan composites that were partially oxidized (15 wt.%), with Gold Pan sulphides (25 wt.%).

The feeds were subjected column cyanide leaching with a summary of the results provided in Table 13.19 below.

Table 13.19: -3/4” Column Leach Results

Test #	Sample ID	Time, days	Leach Soltn. mt/mt ore	Gold %Rec.	Gold Grade (Au, g/mt)			Silver % Rec.	Ag g/mt Calc. Hd.	Copper % Rec.	Screen Hd Cu, ppm	NaCN kg/mt ore	Lime kg/mt ore
					Tail Screen	Calc. Hd.	Screen Hd.*						
P1	OB-Comp-Low	109	3.3	48.1	0.14	0.27	0.25	n/a	n/a	n/a	733	1.29	1.50
P2	OB-Comp-Medium	109	3.3	44.0	0.14	0.25	0.21	36.4	1.1	16.9	398	1.24	2.00
P3	OB-Comp-High	109	3.4	53.1	0.15	0.32	0.25	20.0	1.5	0.7	364	1.21	1.70
P4	RP-CC10-OX/POX	98	4.8	61.5	0.42	1.09	1.07	28.1	3.2	22.4	735	1.84	1.60
P5	AH-CC10-SUL	59	2.9	39.7	0.38	0.63	0.98	18.5	2.7	15.2	806	1.17	1.40
P6	GP-CC10-SUL	59	2.8	24.0	0.57	0.75	0.81	26.7	1.5	23.1	735	1.30	2.70
P7	RP-CC10-OX/SUL	59	3.7	57.7	0.33	0.78	0.87	30.8	2.6	20.1	698*	1.81	2.00

The columns were run for between 59 to 109 days and were terminated based on the leach profile showing that gold dissolution was nearing completion. The three OB composites (tests P1, P2, P3) showed slightly different extents of leaching although the profiles remained similar. For all three tests gold leaching proceeded rapidly for the first 2 weeks to approximately 1/3 extraction. Slower kinetics then prevailed with OB Comp High (P3) having the best results with 53% gold extraction. This was followed by OB Comp. Low (P1) and OB Comp. Medium (P2) which had 48% and 44% respective gold extractions. The calculated and screened heads ranged from 0.21 g/t Au to 0.32 g/t Au, but were generally lower than assayed heads that were performed for bottle roll testing that gave 0.28 gAu/t.

Silver and copper extraction was not followed for test P1. Silver showed 20% recovery for P2, and 36% recovery for P3, respectively on head grades of 1.1 g/t and 1.5 g/t. Copper dissolution was significantly higher on test P2 verses test P3, but did not appear to significantly affect cyanide consumption which was approximately 1.2 kg/t. Lime consumption ranged between 1.5 to 2.0 kg/t for the three OB columns.

Significantly, the precious metal leach curves, continued to show modest increases at termination as shown in Figure 13-5 and 13-6 below for the low and high OB composites.

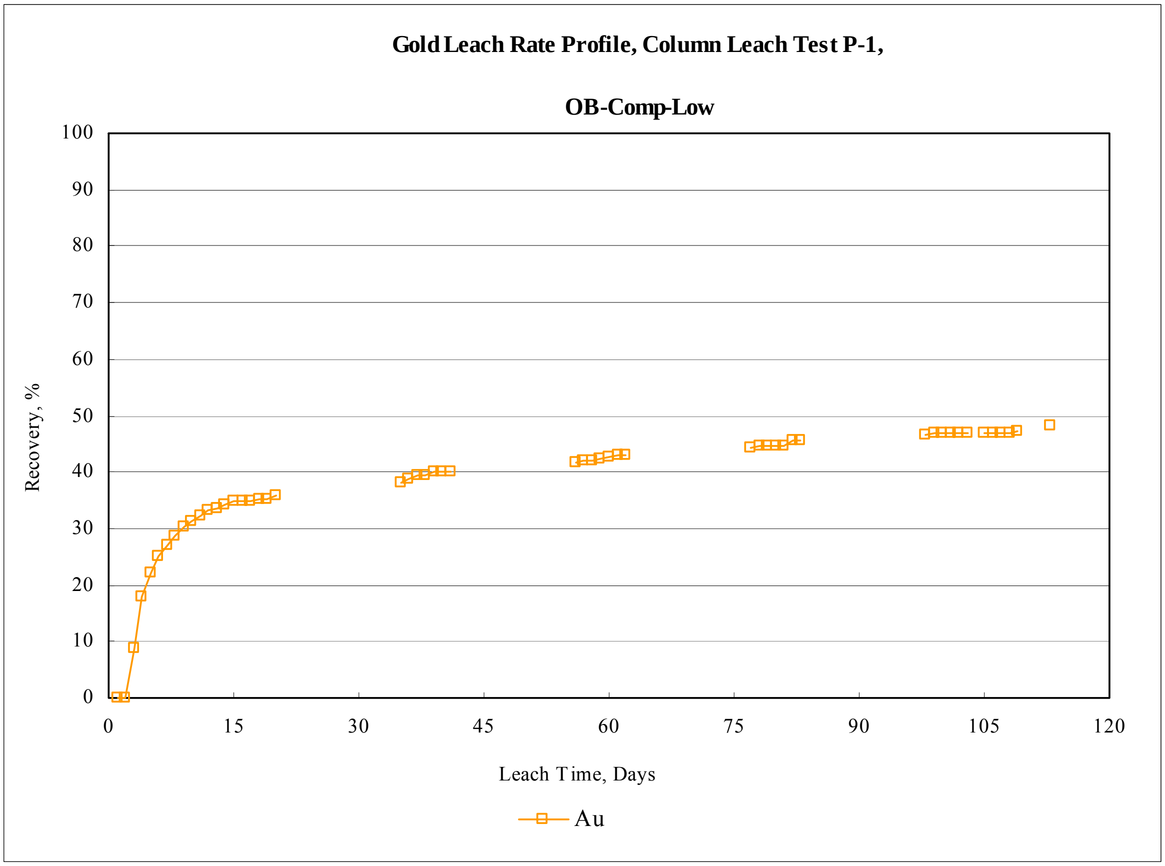


Figure 13-5*: Leach Rate Profile Column Leach Test F-1 OB –Comp - Low

*from McClelland Laboratories Inc.

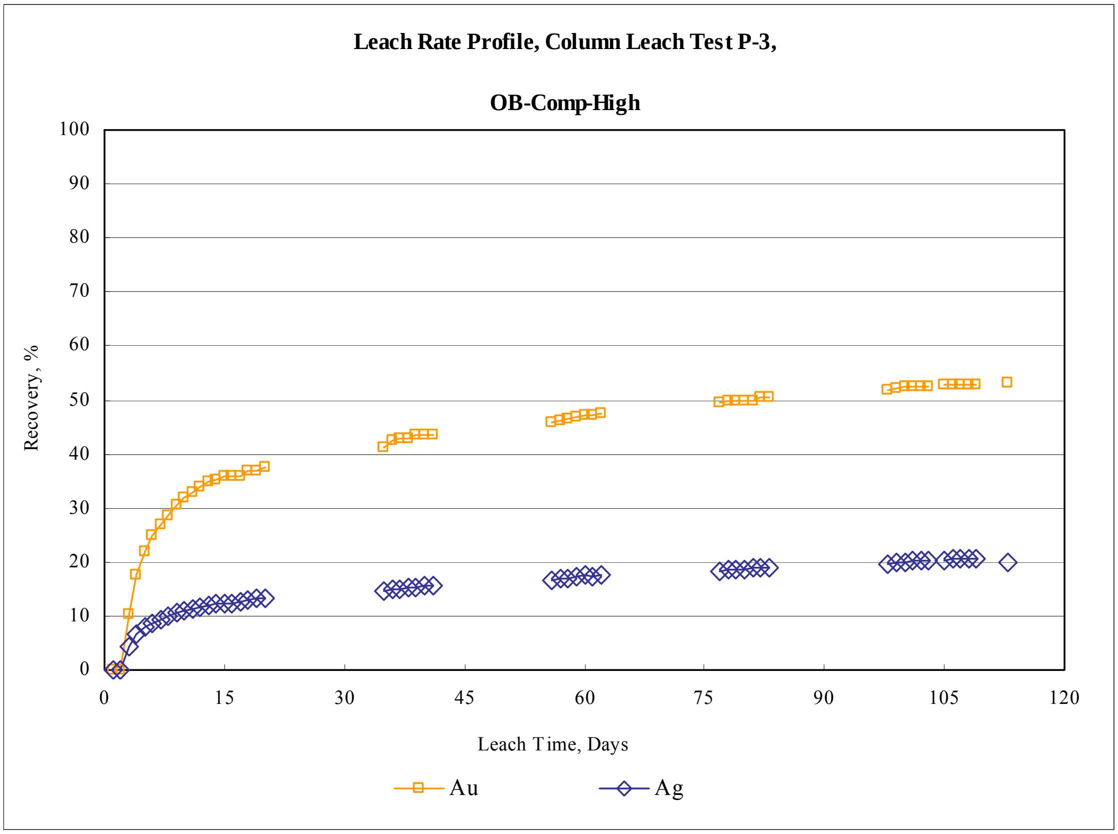


Figure 13-6* Leach Rate Profile Column Leach Test F-1 OB –Comp - High

*from McClelland Laboratories Inc.

The leach profile indicates relatively slow kinetics with minor improvements to overall precious metal recoveries expected by further extending the leach times. Column tailing grades averaged 0.14 g/t Au and were modestly higher than corresponding -10 mesh bottle roll tests that averaged 0.11 gAu/t. This indicates the OB material gold recoveries can be expected to hold up well with larger crush sizes.

Test P4 was performed on the eight oxide composites from Altenburg Hill blended with the two partially oxidized composites from Gold Pan in equal ratios. Each of these composites had previously been subjected to bottle roll testing. The final column gold extraction after 98 days of leaching is calculated at 62% on a calculated head of 1.09 g/t. Silver extraction was 28% on a calculated head of 3.2 g/t. The corresponding leach profile is provided in Figure 13-7.

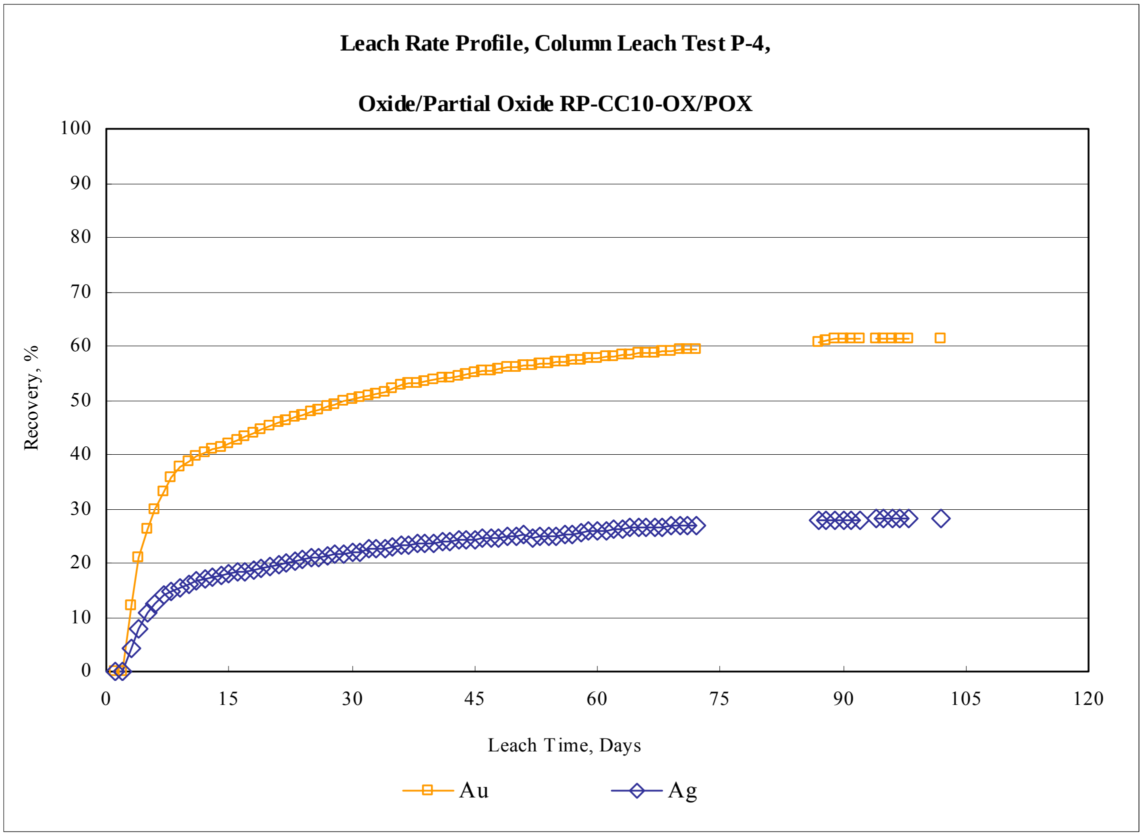


Figure 13-7*: Leach Rate Profile Column Leach Test F-4

*from McClelland Laboratories Inc.

The final tailing grade for test P4 analyzed 0.42 gAu/t. This compares to the tailing grades in the 10 corresponding bottle roll tests that ranged from 0.06 g/t to 0.44 g/t, with an un-weighted average tailings grade of 0.21 gAu/t. The overall column performance provided lower gold recoveries and higher tailing analyses as compared to the bottle roll testing done on -10 mesh material. Looking at the screen analyses data by particle size fraction of the spent column shows poor extractions were achieved in the 40% of material weight above 12.5 mm (1/2”). Material less than 12.5 mm gave correspondingly decreasing tailing grades with finer particle size. This suggests further investigation into using a finer crush size than the 19 mm used is warranted.

Column tests P5 and P6 were respectively performed on sulfide composites from the Altenburg Hill and Gold Pan deposits. Gold analyses for test P5 is being rechecked due to poor agreement between the assayed and calculated heads. Leach profiles for these two tests are provided in Figure 13-8 for Altenburg Hill sulfides and in Figure 13-9 for Gold Pan sulfides.

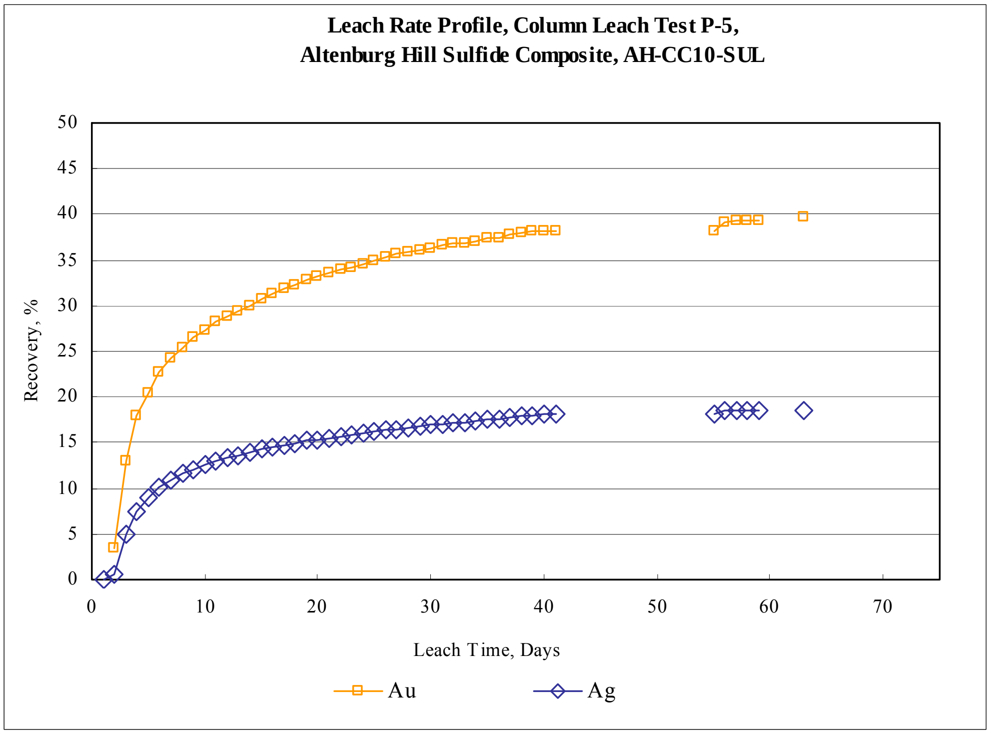


Figure 13-8*: Leach Rate Profile Column Leach Test F-5

*from McClelland Laboratories Inc.

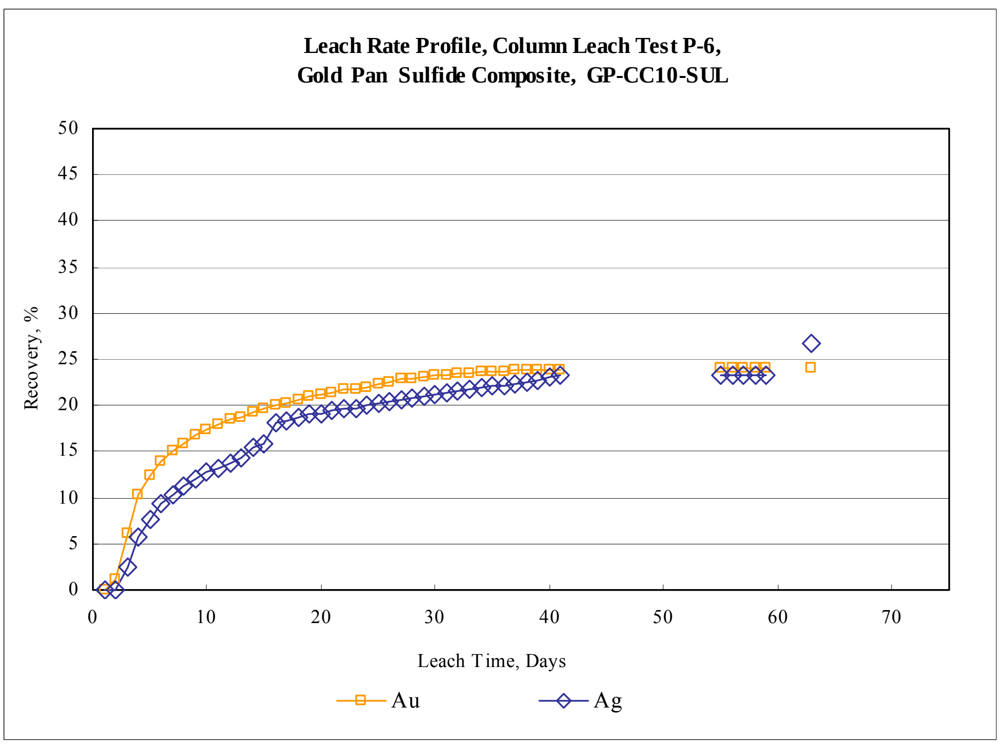


Figure 13-9*: Leach Rate Profile Column Leach Test F-6

***from McClelland Laboratories Inc.**

The column data shows a relatively poor leach response as compared to the corresponding bottle roll tests done on sulfide composites from each of the two deposits. The four Altenburg Hill bottle rolls done by standard cyanide procedures (not including CIL) gave an average un-weighted gold recovery of 66% based on 0.23 g/t tailing and 0.52 g/t average calculated head. Column P5 had 0.38 g/t Au with 0.63 g/t Au in the calculated head resulting in 40% recovery.

Similarly for Gold Pan sulfides the un-weighted average bottle roll gold recovery of 52%, based on 0.34 g/t tailing from 0.71 g/t averaged calculated head. This compares to a column (P6) tailing grade of 0.57 g/t on a calculated head of 0.75 g/t, Au resulting in 24% recovery. Silver recoveries generally followed the gold dissolution profile for test P6.

The lower gold recovery in columns P5 and P6 are likely primarily due to poor permeability of sulfide particles in the coarser crushed material. Lower oxygen availability and minor preg robbing characteristics may have also contributed to the lower column leach response. Cyanide consumption for the sulfide material was in a similar range as the other columns as previously summarized in Table 13.19.

The final column test P7 was operated on mixed oxide and sulfide materials, with the leach profiles plotted in Figure 13-10.

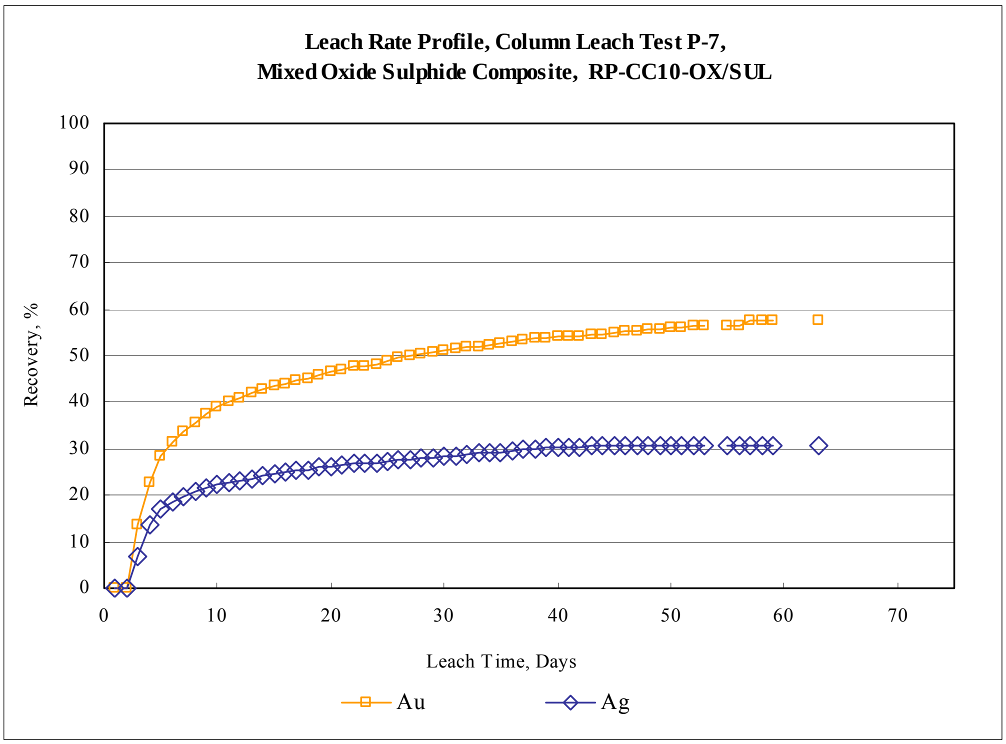


Figure 13-10* Leach Rate Profile Column Leach Test F-7

*from McClelland Laboratories Inc.

The blended sulfide and non-sulfide composites provided 58% gold recovery based on a calculated head of 0.78 g/t that resulted in 0.33 g/t tailing. This appears to offer a relatively good correlation to the expected recovery based on the ratio of sulfide and non-sulfide feed materials used.

A final comparison of the bottle roll data to the corresponding column extractions is plotted in Figure 13-11.

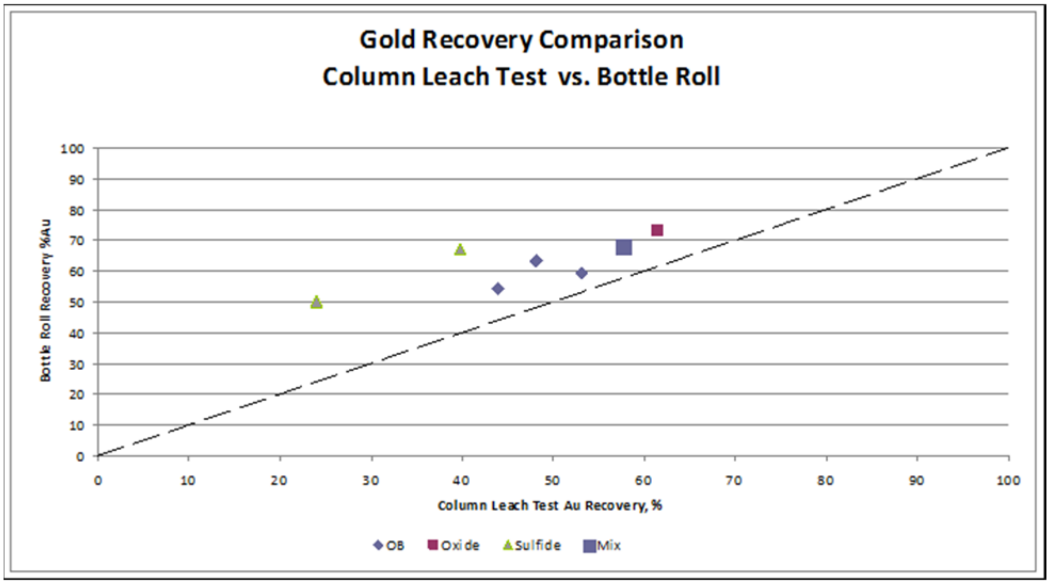


Figure 13-11*: Gold Recovery Comaprison

*from McClelland Laboratories Inc.

The plot outlines the lower recoveries from the column results verses the bottle roll data. The largest variation is graphically shown for the sulfide material (tests P5, P6).

13.4 CONCLUSIONS

The 2011 laboratory program was performed on 2010 drill core samples composited from the Robertson Property to evaluate heap leaching potential. The leaching studies included testing on both oxide and sulfide material from the Altenburg Hill deposit, and on partially (potentially) oxidized and sulfide material from the Gold Pan deposit. The program also included testing of low grade gold composites (~0.28 g/t Au) that were classified as overburden from areas that would need to be mined in order to access higher grade zones. Initial test work included bottle roll cyanidation performed on 30 composites from these areas and deposits. Bottle rolls were undertaken for 96 hours on composites that had been crushed to minus 1.7 mm (-10 Tyler mesh). This was followed by column leach test work on seven re-blended master composites that had been crushed to 95% passing minus 19 mm (-3/4") and operated for periods between 59 to 109 days.

Three overburden (OB) composites with ~0.28 g/t head grades showed bottle roll recoveries averaging 59% recovery, with fixed tailing grades of ~0.11 g/t. The three corresponding column leach tests had lower average calculated head grades and gave an average recovery of 48% with tailing grades of 0.14 g/t. Considering the head grade variation the overall coarser crush size in the columns compared well to bottle roll data. The column leach profiles suggest that extending the leach time may improve recoveries slightly. Future evaluation at coarser crush sizes is required to determine how the OB material would respond to dump leaching techniques. A preliminary estimate would suggest a gold recovery of 40% to 50% could be expected for OB type material.

A column leach for non-sulfide ore was performed on a blend of Altenburg Hill oxide material (80% by weight), blended with partially oxidized Gold Pan material (~20% by weight). The results provided a gold recovery of 62%. This is considerably lower than the average bottle roll testing gold recovery of 71%, which was done on the sub-composites that together made up the column feed material. Examination of the results of the screened column tailing suggest that the majority of the losses were in the +12.5 mm (+1/2") fraction. Historic studies suggested that a minus 1/2" crush size appeared to be optimum for the property. The earlier studies also included more detailed investigation on the Porphyry deposit and reported that a 67% gold recovery might be applied with a -1/2" crush size. The most recent test data suggests a similar recovery might be applicable to oxide composites from Altenburg Hill and Gold Pan, if crushed to ½".

There were two column leach studies performed on sulfide material, one from Altenburg Hill composites and the other on Gold Pan composites. Both tests experienced significantly lower recoveries than the corresponding bottle rolls. A third column used sulfides blended with oxide material. Based on these three overall column results, heap leach recoveries can be expected to decrease with the addition of higher ratios of sulfide ore. Additional study is required to better determine this relationship.

Depending on the schedule of project advancement, on-site testing can be considered for both heap and dump leaching evaluation to supplement and /or replace portions of a laboratory program. This would involve bulk mining and building of cribs and/or test heaps (with pads and ponds) in order to better evaluate leaching of low grade run of mine rock as well as higher grade crushed material. Such a program would be costly as it involves contract bulk mining of several thousand tons of rock, mobile crushing and stacking, and long term monitoring. Costs would need to be developed with local contractors and suppliers.

SECTION 14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The following sections detail the methods, processes and strategies employed in creating the resource estimate for the Robertson Deposit. Table 14.1 lists some conventions and abbreviations that are encountered throughout the resource estimation section of this report.

Table 14.1: Report Conventions and Abbreviations

Abbreviation	*Description*
Au	Gold
OzAu/t	Ounce Gold per Ton
% or pct	Percent
m	Meters
QA/QC	Quality Assurance / Quality Control
X, Y, Z	Cartesian Coordinates, also "Easting", "Northing", and "Elevation"
DDH	Diamond Drill Holes.
N, S, E, W	Cardinal points, North, South, East, and West, respectively, and combinations thereof.
CV	Coefficient of Variation.
Coral	Coral Gold Resources Ltd.

14.2 DATA EVALUATION

A total of 1,266 drillholes were supplied for the Robertson Property in Lander County, Nevada which are the combined drillholes for the Gold Pan, 39A, Porphyry, Altenburg Hill and Lower Triplett Gulch zones in addition to areas that have drilling but lie outside the main areas of interest. The drillholes within the database included collars, downhole surveys, assays, and lithology.

The drillhole database (US Imperial) was delivered in electronic format which was supplied by Coral. This included collars, downhole surveys, lithology data and assay data (i.e. ozAu/t) with downhole from and to intervals in imperial units. It should be noted that some surface data was supplied in metric units and then converted to imperial. It is considered that this conversion has been done correctly and the author has verified that the data is valid with respect to location.

Simple statistics for the assay data are shown below in Table 14.2 and 14.3, which shows statistics for gold assays weighted and unweighted by assay interval, respectively.

Table 14.2: Statistics for Gold Assays for the Robertson Deposit
Unweighted

	Count	Maximum	Mean	Std. Devn.	Co. of Variation
Gold Pan	24,760	2.886	0.009	0.033	3.896
Porphyry	22,463	0.959	0.011	0.026	2.403
Altenburg Hill	6,562	0.847	0.007	0.020	2.711
39A	12,703	1.524	0.013	0.048	3.580
Distal	1,982	0.439	0.009	0.032	3.398
Triplet Gulch	12,991	2.000	0.003	0.022	6.690
Outside	9,660	1.006	0.004	0.024	5.634
All	91,121	2.886	0.009	0.031	3.672

Table 14.3: Statistics for Gold Assays for the Robertson Deposit
Weighted by Assay Interval

	Ft.	Maximum	Mean	Std. Devn.	Co. of Variation
Gold Pan	132,113	2.886	0.009	0.033	3.797
Porphyry	108,980	0.959	0.011	0.026	2.412
Altenburg Hill	33,844	0.847	0.007	0.020	2.749
39A	69,735	1.524	0.013	0.046	3.639
Distal	11,134	0.439	0.009	0.031	3.305
Triplet Gulch	70,160	2.000	0.003	0.022	6.424
Outside	56,792	1.006	0.004	0.023	5.895
All	482,758	2.886	0.008	0.031	3.695

The assay Au database (91,121 Au values) shows that gold distributions are very well behaved (in comparison with other deposits), still with a some samples in each case representing a outlier populations. The mean overall for Au grade (weighted by sample length) is 0.008 ozAu/t with standard deviation of 0.031. However, within the individual zones the mean grade is 0.009, 0.013, 0.011, 0.007, 0.009 and 0.003 ozAu/t for the Gold Pan, 39A, Porphyry, Altenburg Hill, Distal and Triplet Gulch areas, respectively. In addition, the assays that lay outside these specific areas are listed with the average grade being 0.004 ozAu/t.

Gold assays have a relatively high coefficient of variation (CV) of 3.797, 3.639, 2.412, 2.749, 3.305 and 6.424, respectively (weighted by sample length in Table 14.3). This indicates a relatively modest scatter of the raw data values. The coefficient of variation is defined as $CV=\sigma/m$ (standard deviation/mean), and represents a measure of variability that is unit-independent. This is a variability index that can be used to compare different and unrelated distributions.

Figure 14-1 shows the histogram weighted by assay interval length, of all Au assays along with the corresponding probability plots in Figures 14-2. The histograms and cumulative probability plots shows that the assay data all demonstrate log normal distributions. Figure 14-3 and 14-4 shows a plan and long section views of the drillholes.

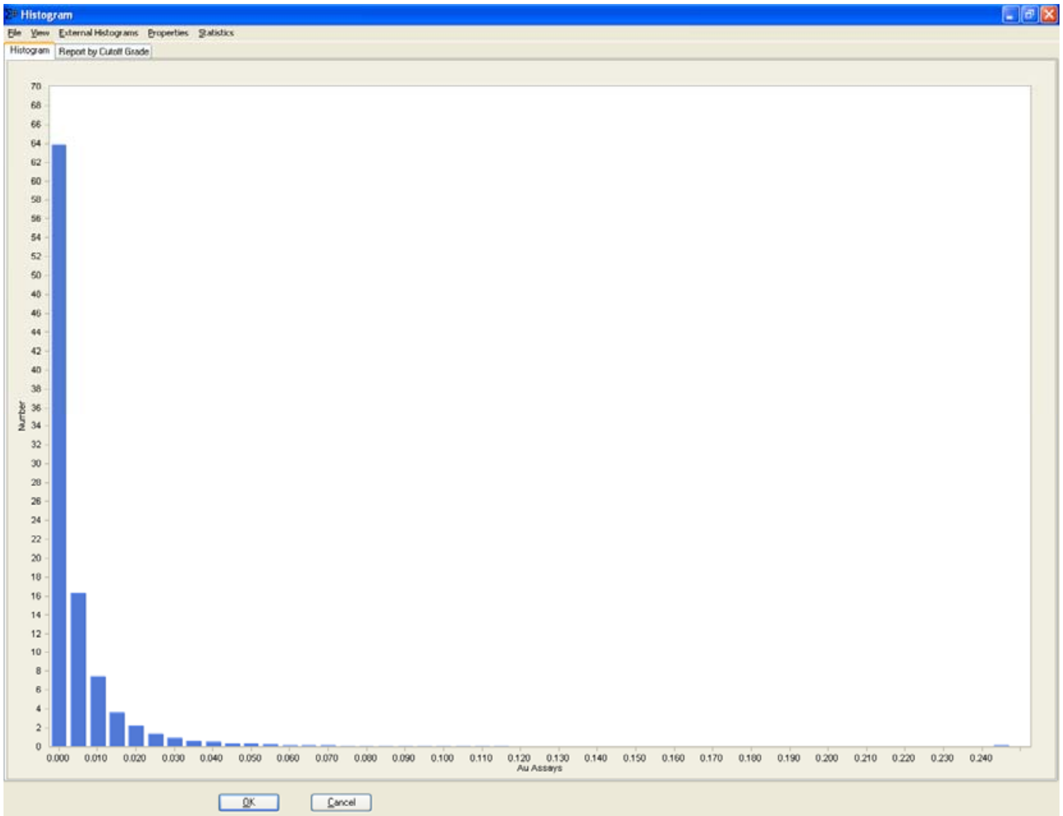


Figure 14-1: Au Assays Weighted

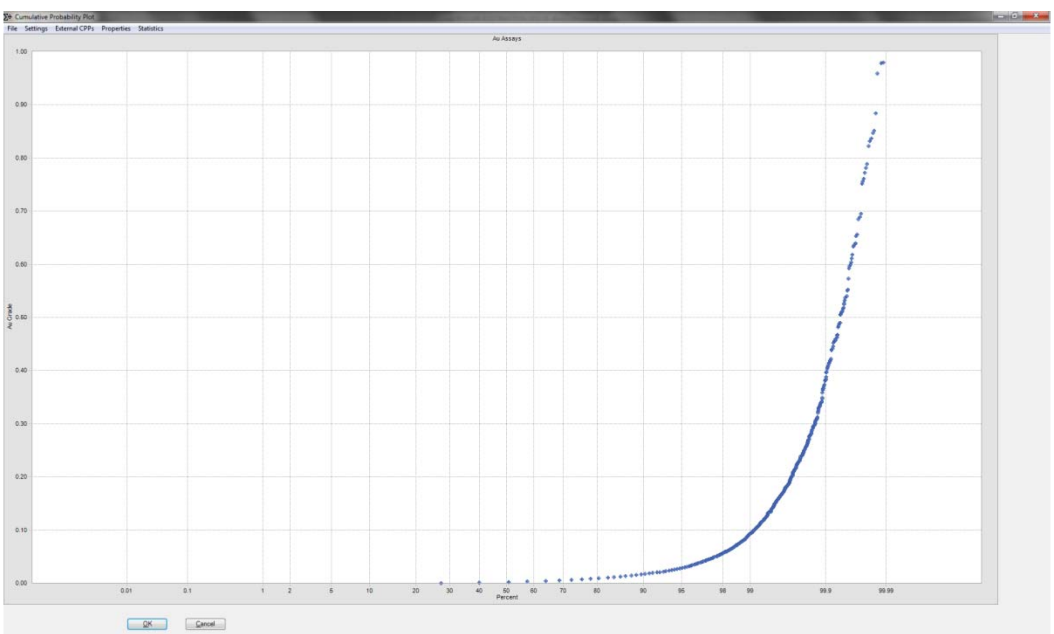


Figure 14-2: Cumulative Distribution Plot of Au Assays.

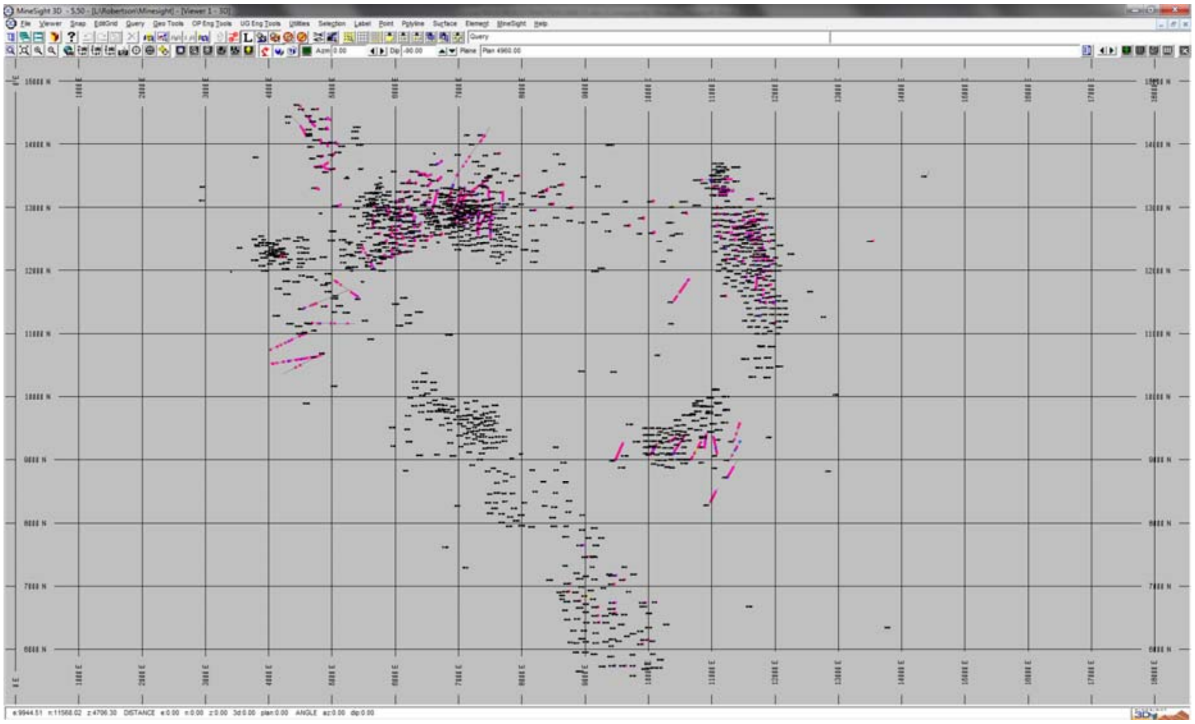


Figure 14-3: Plan view showing drill holes used in resource estimate.

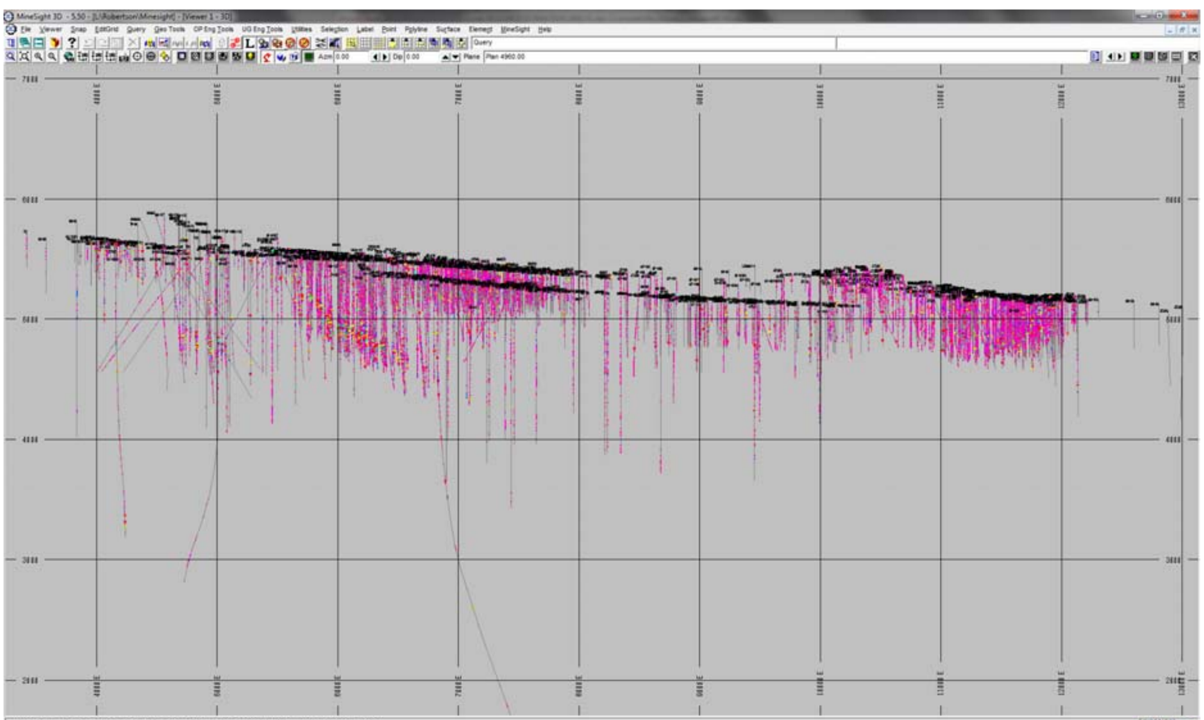


Figure 14-4: Long Section Perspective view (looking north) showing all drill holes used in Resource Estimate.

14.3 TOPOGRAPHY

Topography was imported from an AutoCAD topographic map supplied by Coral in DXF format. The topography was surveyed and is believed to be accurate. Checks against drillhole collars illustrate and accuracy to within 1 meter with the exception of one drillhole which was identified and corrected. In addition, in an effort to delineate the top of mineralization, a model of bottom contact of the Quaternary Alluvium layer (QAL) was triangulated from drillhole intercepts. This surface is shown in Figure 14-6.

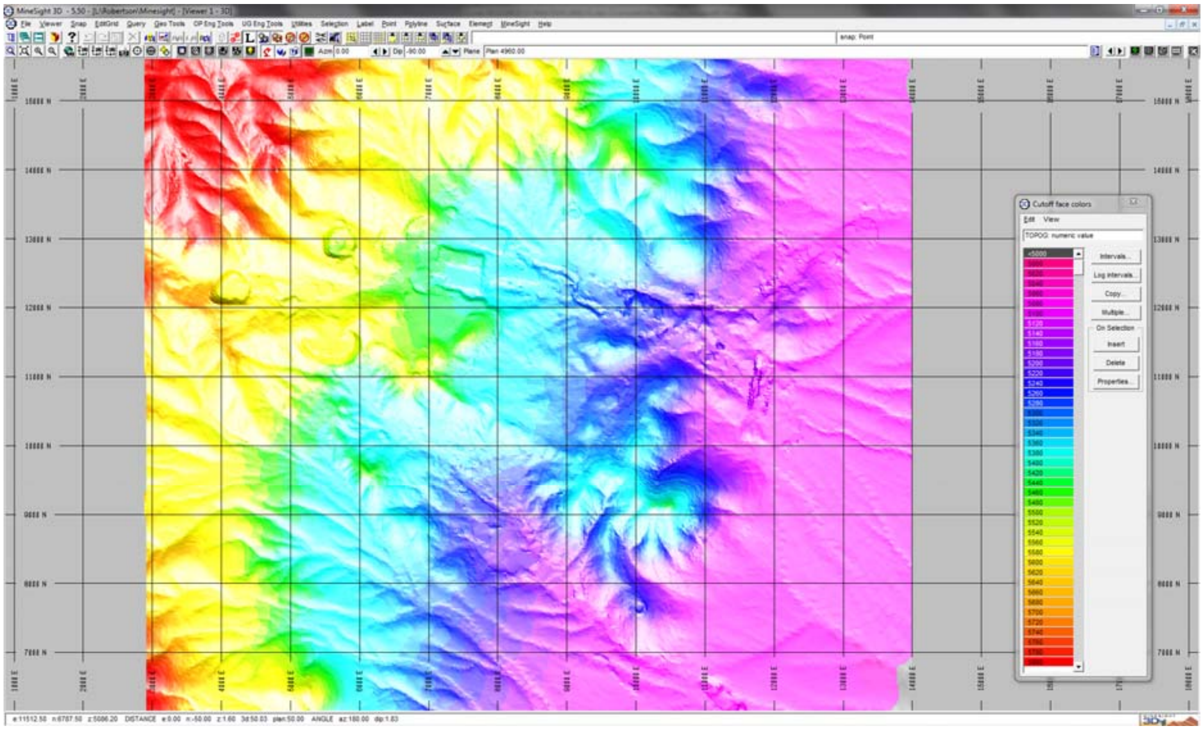


Figure 14-5: Plan view of the 3D gridded topographic.

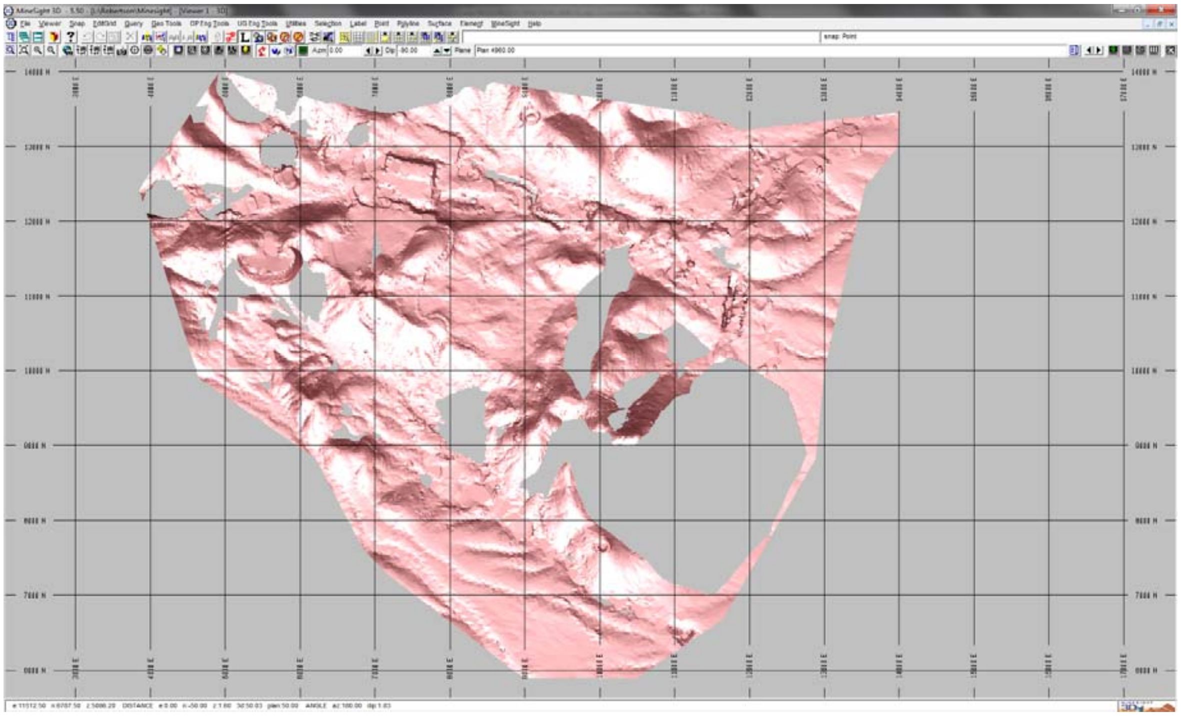


Figure 14-6: Plan view of the 3D gridded model of Quaternary Alluvium layer (north to top of screen).

14.4 COMPUTERIZED GEOLOGIC MODELING

Solids models of the main ore zones within the Robertson Deposit were created that encompass the Gold Pan, 39A, Porphyry, Altenburg Hill, Distal and Triplet Gulch deposit areas. The ore zones to be included within the solids model and then to be used for constraining the interpolation procedure are split into an Oxide Zone and a Sulphide Zone where sufficient data existed to do so which included that Gold Pan, 39A, Porphyry and Alteberg Hill areas. Due to its depth, the Distal zone is considered to be sulphide material.

Once the solids models were created, they were used to then code the drillhole assays and composites for subsequent geostatistical analysis. For the purpose of the resource model, the individual solid zones were utilized to constrain the block model by matching assays to those within the zones in a process called geologic matching so that only composites that lie within a particular zone are used to interpolate the blocks within that zone. The orientation and ranges (distances) utilized for search ellipsoids used in the estimation process were derived from previous reports, site observations and geostatisical analysis.

Figure 14-7 shows the solids created that encompass the Gold Pan, 39A, Porphyry, Altenburg Hill, Distal and Triplet Gulch deposit areas. Figure 14-9 illustrates the oxide solid that identifies the oxide/sulphide interface. All material lying outside the oxide solid is considered sulphide by default.

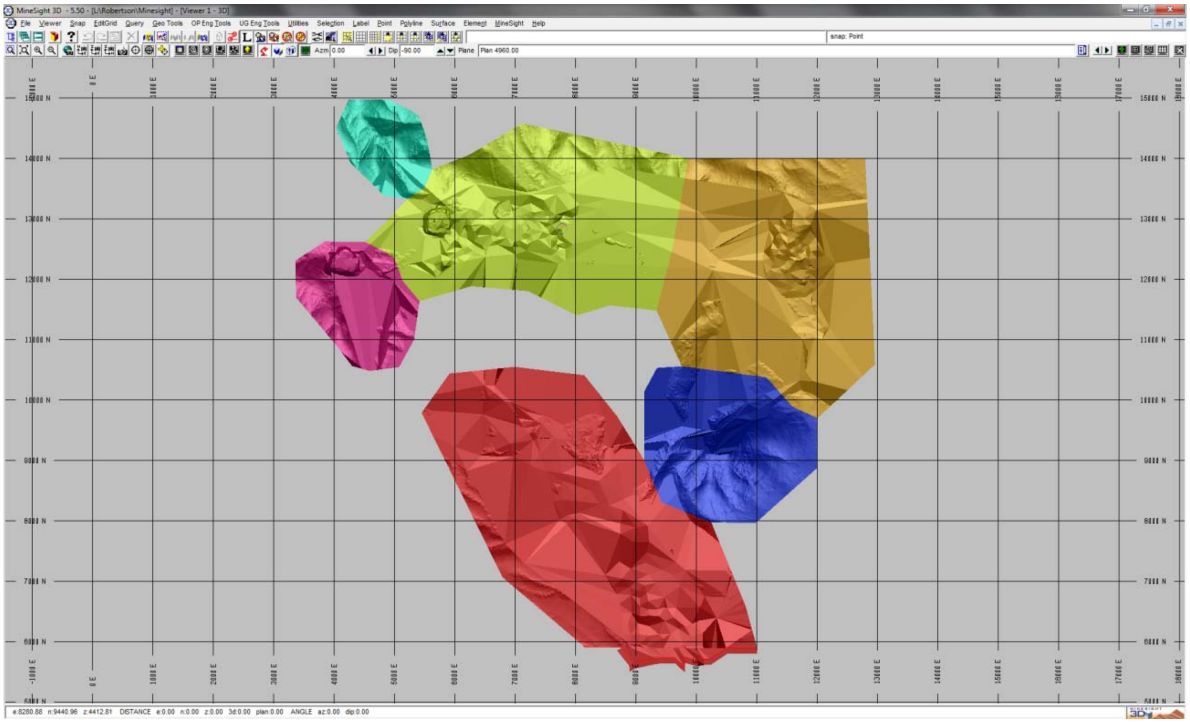


Figure 14-7: Plan view of showing drill holes with interpretation of the mineralized zones (Gold Pan-yellow, Distal-green, Porphyry- orange, Altenburg Hill-blue and Triplet Gulch-red) clipped to topography.

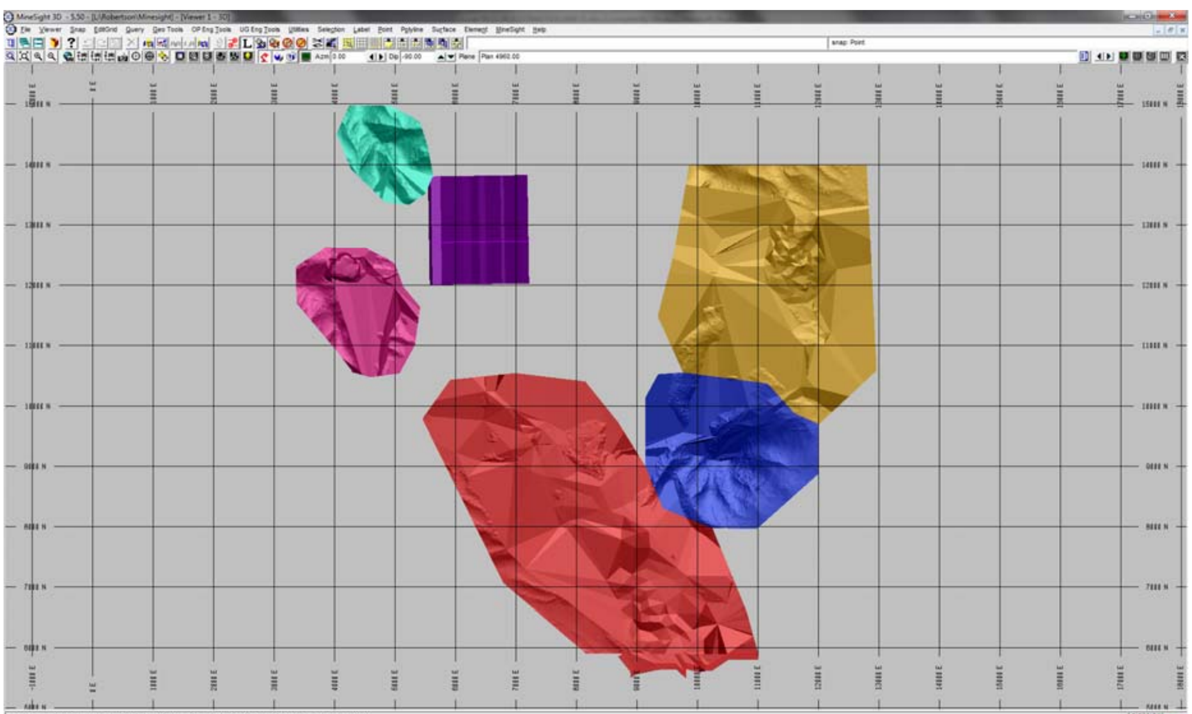


Figure 14-8: Plan view of showing drill holes with interpretation of the mineralized zones (39A-purple, Distal-green, Porphyry- orange, Altenburg Hill-blue and Triplet Gulch-red) clipped to topography.

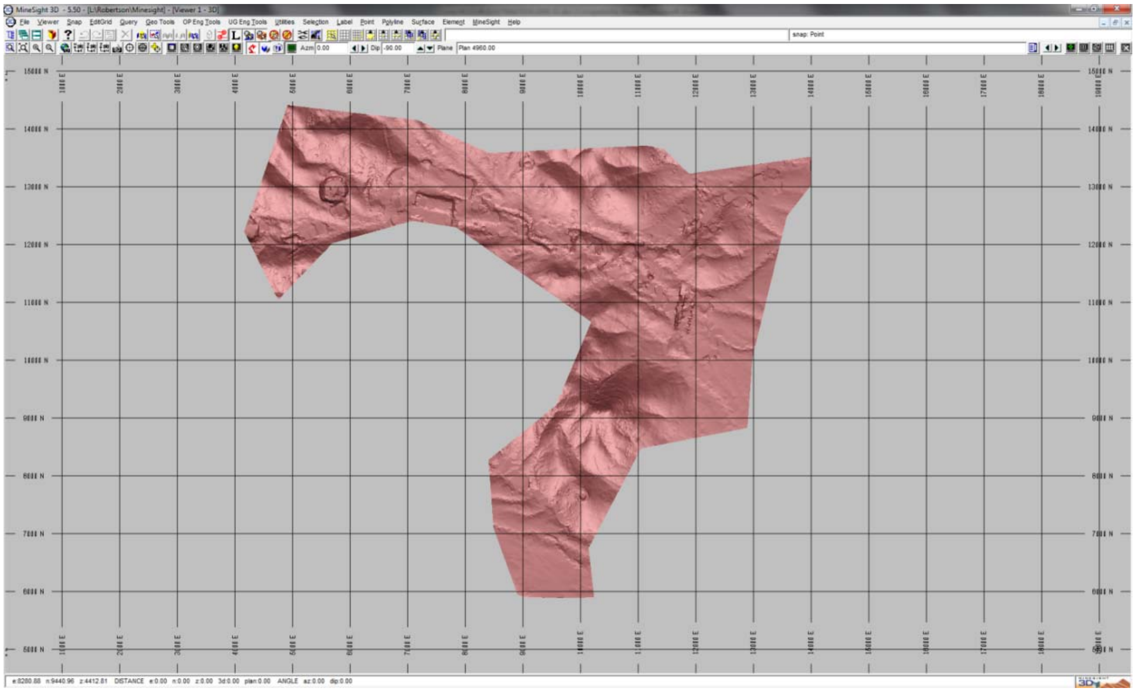


Figure 14-9: Plan view of showing drill holes with interpretation of the oxide zone clipped to topography.

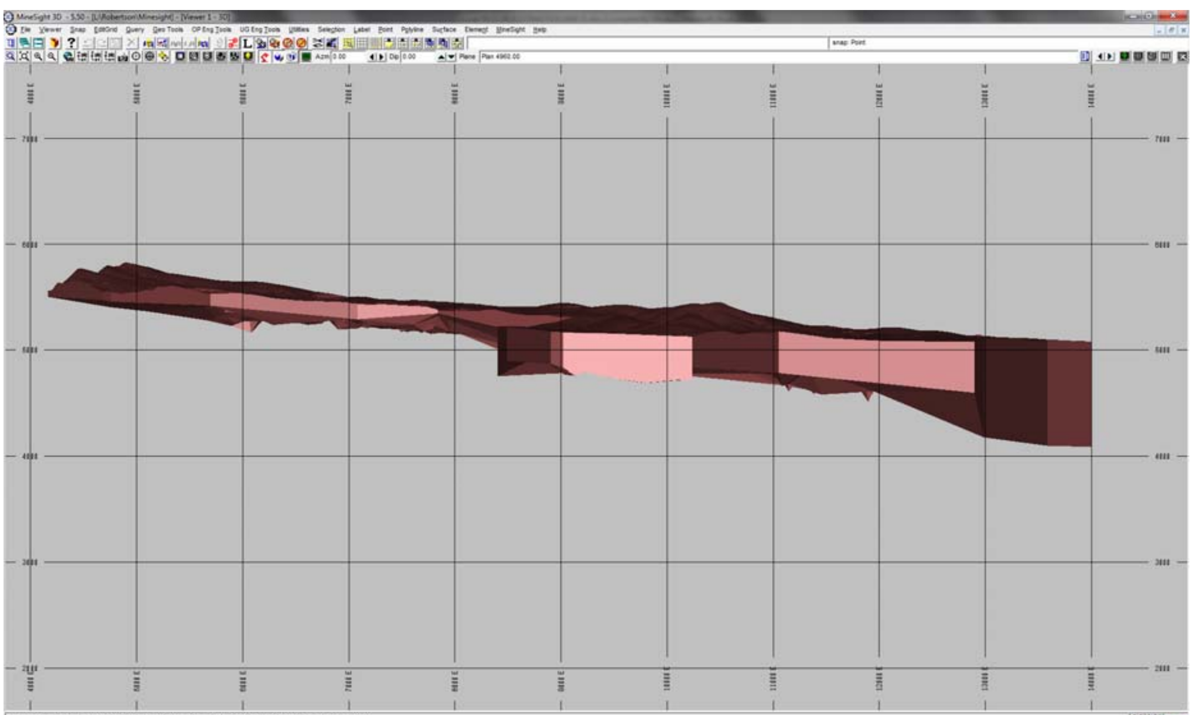


Figure 14-10: Section view of showing drill holes with interpretation of the oxide zone clipped to topography.

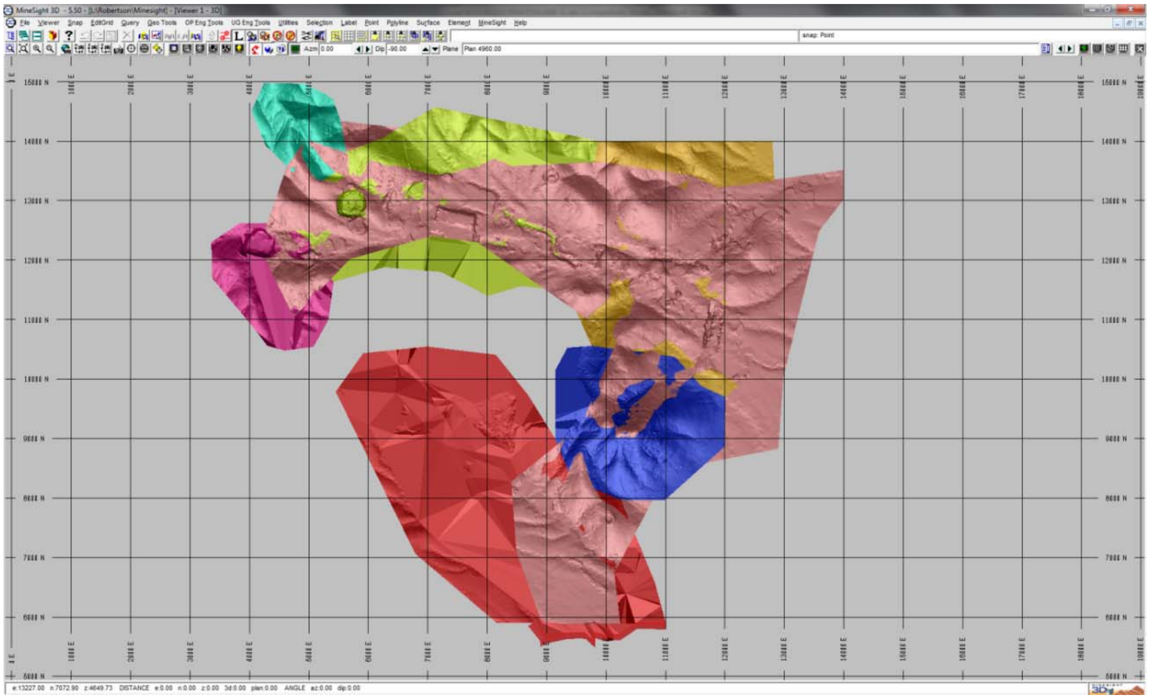


Figure 14-11: Section view of showing drill holes with interpretation of the mineralized zones (Gold Pan-yellow, Distal-green, Porphyry- orange, Altenburg Hill-blue and Triplet Gulch-red) overlain with the oxide solid in red and yellow solid clipped to topography.

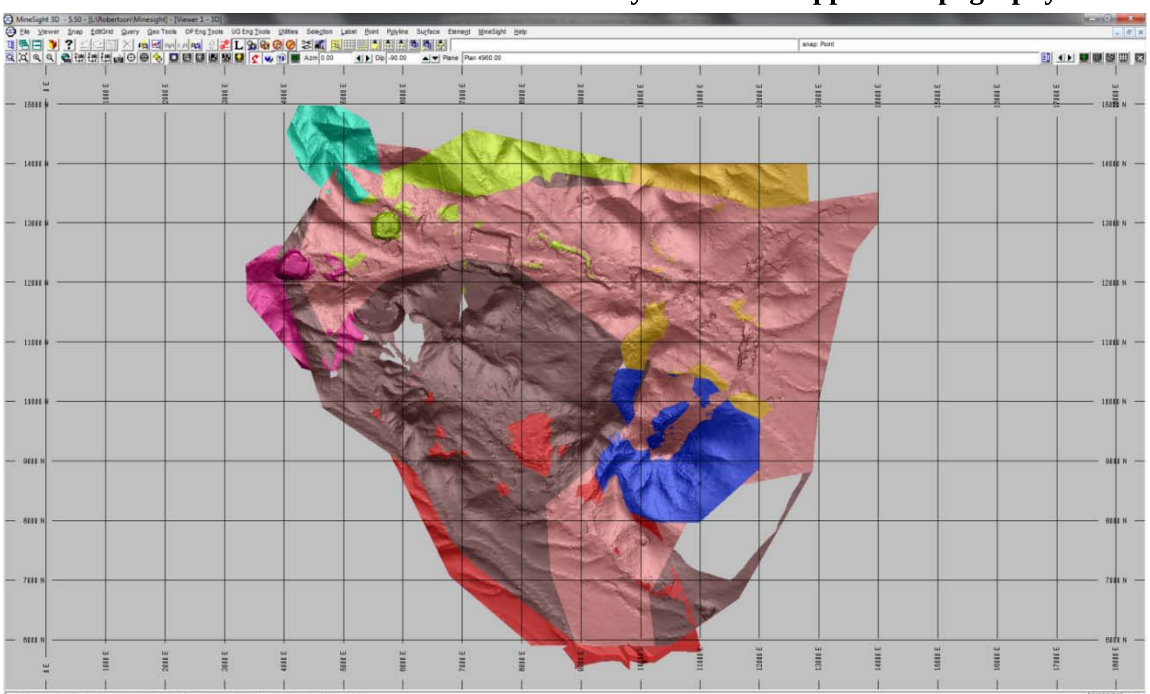


Figure 14-12: : Plan view of showing drill holes with interpretation of the mineralized zones (Gold Pan-yellow, Distal-green, Porphyry- orange, Altenburg Hill-blue and Triplet Gulch-red) overlain with the oxide solid in red and yellow solid clipped to topography.

14.5 COMPOSITES

It was determined that the 20 foot composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of the grades in addition to reducing the effect of high grades to a small extent. However, the primary reason for the choice of 20 feet for the composite length was that bench heights are likely to be 20 feet and this will be the selective mining unit. Table 14.4 shows the basic statistics for the 20 foot composites for Au listed by zone and by oxide/sulphide sub-zone. Figures 14-13 and 14-14 illustrate the histogram, grade-tonnage curve and cumulative distribution plots for gold composite grade intervals. Figure 14-15 shows the box plots for each of the zones split by oxide and sulphide with the numeric codes as listed in Table 14.4. It is important to note that the oxide and sulphide for Gold Pan, Altenburg Hill and Triplet Gulch are markedly similar and that separation by domaining is not necessary whilst the oxide and sulphide within the Porphyry zone so illustrate differing populations and presents a case for domaining. The same could be said for Distal however only a small percentage of the assays are in oxide and are of very low grade therefore Distal is considered to be sulphide. In addition, 39A illustrates distinctively higher grades and character which confirms the need to segregate 39A into an individual domain and interpolate separately.

Table 14.4 Composite Statistics

	ZONE	OX/SUL	Length	Max	Mean	1st Q	Median	3rd Q	SD	CV
11	**Distal**	**Oxide**	1,695	0.055	0.003	0.000	0.000	0.001	0.009	3.540
12	**Distal**	**Sulphide**	25,109	0.196	0.006	0.000	0.001	0.004	0.017	3.019
22	**39A**	**Sulphide**	31,215	0.452	0.023	0.004	0.008	0.021	0.043	1.891
31	**Gold Pan**	**Oxide**	81,724	0.367	0.008	0.001	0.004	0.008	0.017	2.186
32	**Gold Pan**	**Sulphide**	95,506	0.781	0.008	0.002	0.004	0.009	0.021	2.599
41	**Porphyry**	**Oxide**	99,773	0.418	0.011	0.002	0.007	0.014	0.018	1.575
42	**Porphyry**	**Sulphide**	11,359	0.118	0.005	0.000	0.002	0.007	0.009	1.749
51	**Alt Hill**	**Oxide**	34,160	0.218	0.009	0.002	0.005	0.011	0.012	1.418
52	**Alt Hill**	**Sulphide**	6,530	0.063	0.007	0.002	0.005	0.008	0.007	1.097
61	**Triplett Gulch**	**Oxide**	22,655	0.502	0.005	0.000	0.002	0.005	0.018	3.868
62	**Triplett Gulch**	**Sulphide**	51,230	0.335	0.003	0.000	0.001	0.003	0.010	3.080
72	**East Zone**	**Sulphide**	22,660	0.311	0.004	0.000	0.001	0.004	0.016	3.562
Total			483,616	0.781	0.009	0.001	0.004	0.009	0.020	2.343
All			527,962	0.781	0.008	0.001	0.003	0.008	0.020	2.442

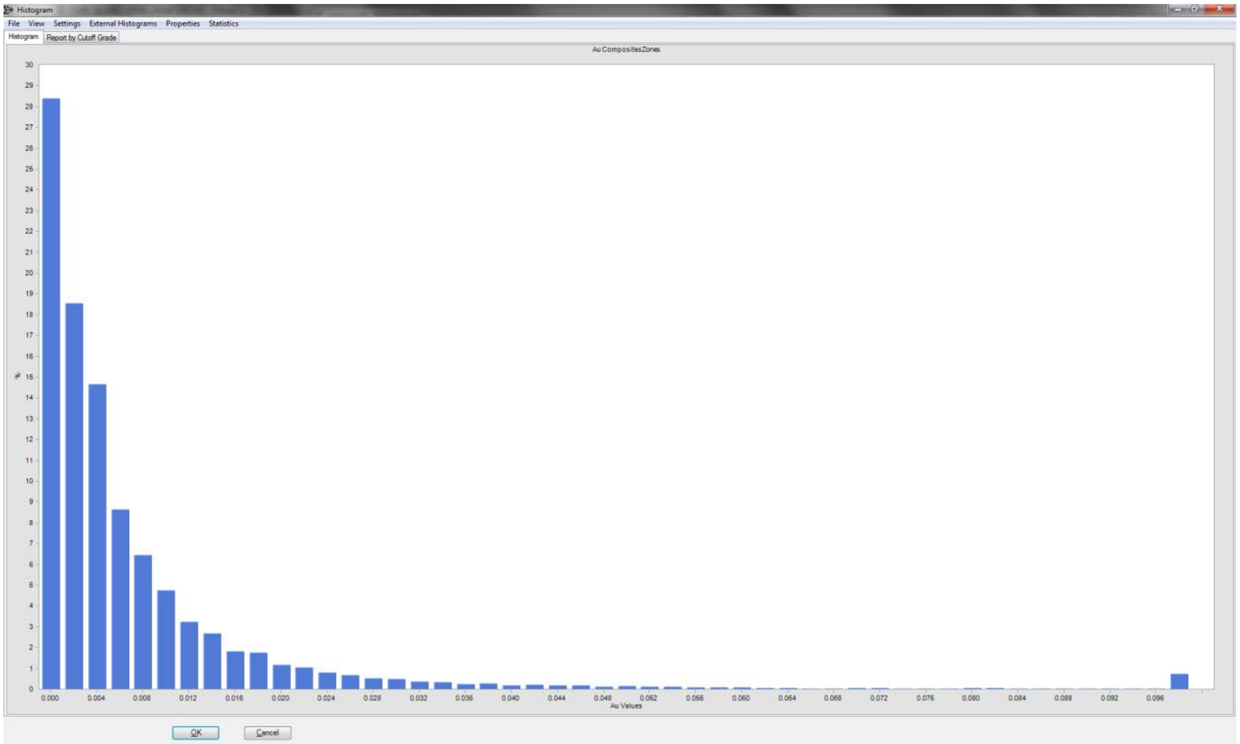


Figure 14-13: Histogram for Au, 20 foot composites.

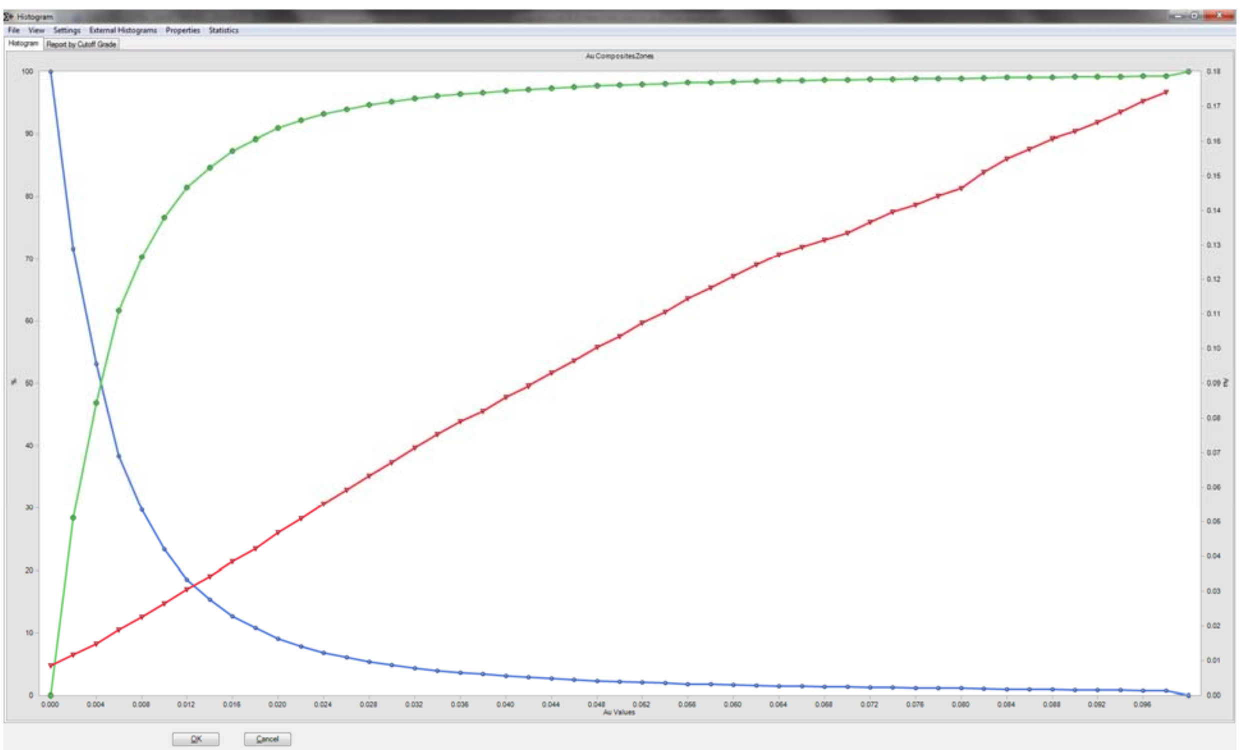
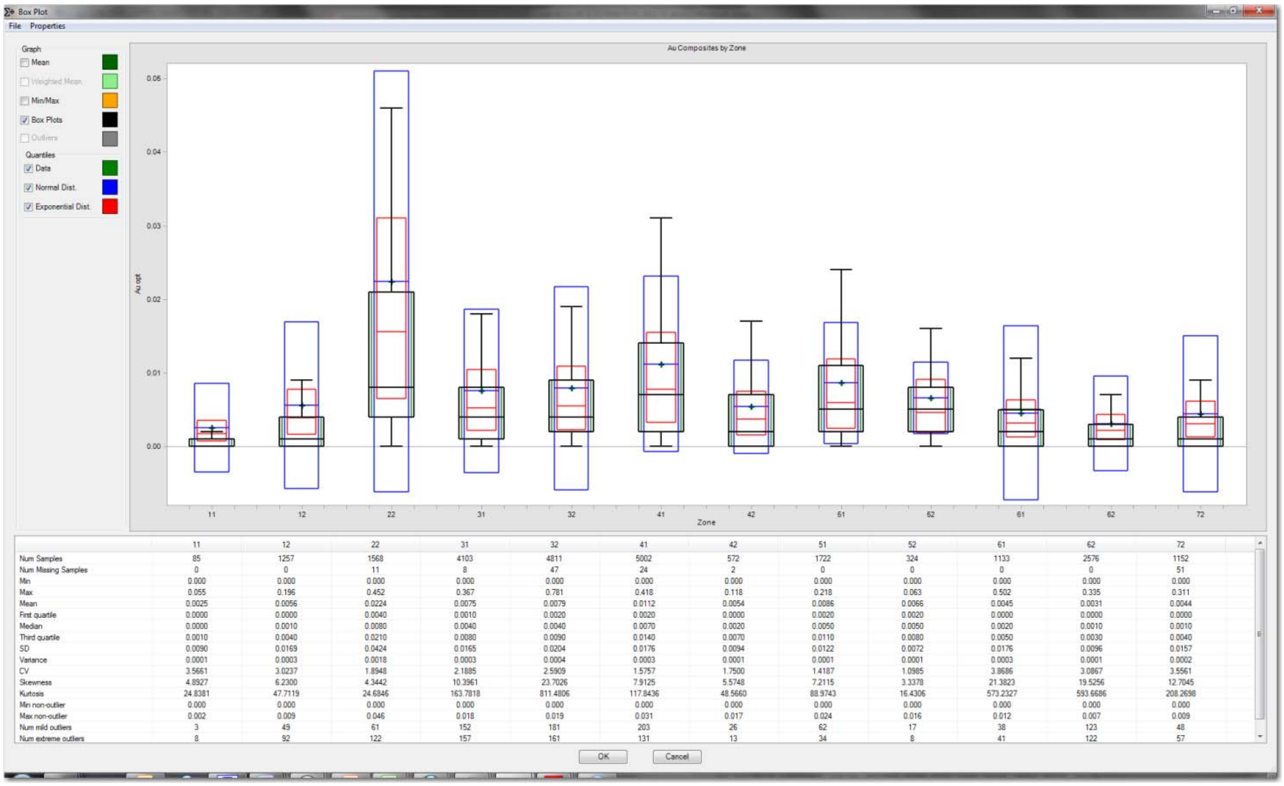


Figure 14-14: Grade-Tonnage Curves for Au, 20 foot composites.



	11	12	22	31	32	41	42	51	52	61	62	72
Num Samples	85	1257	1568	4103	4811	5002	572	1722	324	1133	2576	1152
Num Missing Samples	0	0	11	8	47	24	2	0	0	0	0	51
Min	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Max	0.055	0.196	0.452	0.367	0.781	0.418	0.118	0.218	0.063	0.502	0.335	0.311
Mean	0.0025	0.0056	0.0224	0.0075	0.0079	0.0112	0.0054	0.0086	0.0066	0.0045	0.0031	0.0044
First quartile	0.0000	0.0000	0.0040	0.0010	0.0020	0.0020	0.0000	0.0020	0.0020	0.0000	0.0000	0.0000
Median	0.0000	0.0010	0.0080	0.0040	0.0040	0.0070	0.0020	0.0050	0.0050	0.0020	0.0010	0.0010
Third quartile	0.0010	0.0040	0.0210	0.0080	0.0090	0.0140	0.0070	0.0110	0.0080	0.0050	0.0030	0.0040
SD	0.0090	0.0169	0.0424	0.0165	0.0204	0.0176	0.0094	0.0122	0.0072	0.0176	0.0096	0.0157
Variance	0.0001	0.0003	0.0018	0.0003	0.0004	0.0003	0.0001	0.0001	0.0001	0.0003	0.0001	0.0002
CV	3.5661	3.0237	1.8948	2.1885	2.5909	1.5757	1.7500	1.4187	1.0985	3.8686	3.0867	3.5561
Skewness	4.8927	6.2300	4.3442	10.3961	23.7026	7.9125	5.5748	7.2115	3.3378	21.3823	19.5256	12.7045
Kurtosis	24.8381	47.7119	24.6846	163.7818	811.4806	117.8436	48.5660	88.9743	16.4306	573.2327	593.6686	208.2698
Min non-outlier	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Max non-outlier	0.002	0.009	0.046	0.018	0.019	0.031	0.017	0.024	0.016	0.012	0.007	0.009
Num mild outliers	3	49	61	152	181	203	26	62	17	38	123	48
Num extreme outliers	8	92	122	157	161	131	13	34	8	41	122	57

Figure 14-15: Box Plots for Au, 20 foot composites.

14.6 OUTLIERS

Limiting the influence as opposed to capping of gold assays were performed for although the distribution of grades followed a normal distribution the probability plots showed "breaks" which indicated multiple populations. The author used the commonly accepted method of determining metal-at-risk, which employs the utilization of cumulative frequency plots and identifying "breaks" along the plot trend. The grade threshold chosen was determined to be 0.18 ozAu/t.

However, it is important to note the method employed for this study is not to cut the high grade outliers but to limit their influence. The range chosen at which to limit grades greater than the outlier cutoff was determined to be 50 feet. In other words, composite grades greater than 0.18 ozAu/t would not influence blocks greater than 50 feet from where that composite is located.

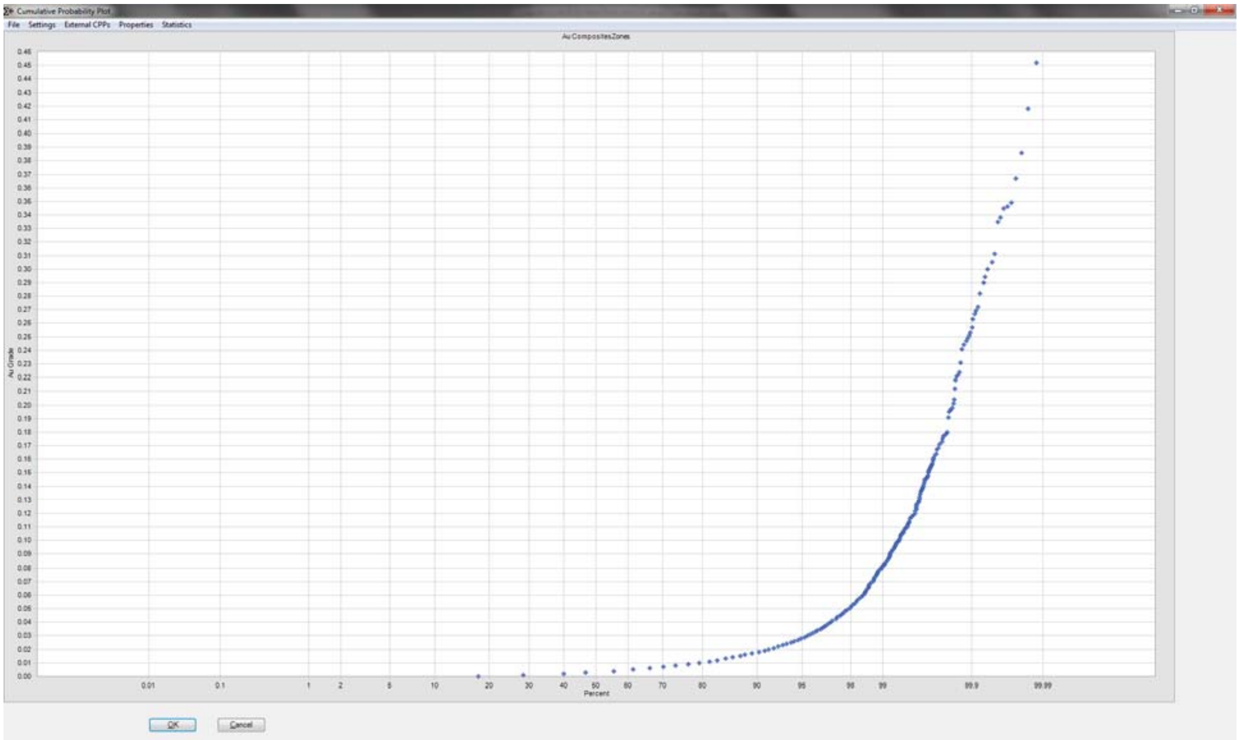


Figure 14-16: Probability Plot for Au, 20 foot composites.

14.7 SPECIFIC GRAVITY DETERMINATIONS

Approximately 215 historic core samples were analyzed for dry bulk density using the volume displacement method. Densities for both ore and waste are primarily related to lithology, argillization, calc-silicate content and sulfide content. The average density for country rock was determined to be 12.2 cu ft/ton and 15.5 cu ft/ton for alluvium (2 determinations). These are historic determinations, are located for the most part, within the Porphyry Zone. In addition, 279 specific gravity measurements were derived for samples from Altenburg Hill and Gold Pan in 2010.

14.8 VARIOGRAPHY

The author carried out geostatistical analysis on the composites to evaluate the search parameters to be used in the grade estimate.

Downhole correlograms were generated in order to make an estimate of the nugget effect as that is the direction in which there is most abundant data. The downhole correlogram for the complete dataset within the zones indicates that the nugget effect is in the order of 55% of the sill for Au as shown in Figure 14-17. Although this is relatively high, it is to be expected to some extent and within acceptable ranges for a typical gold deposit.

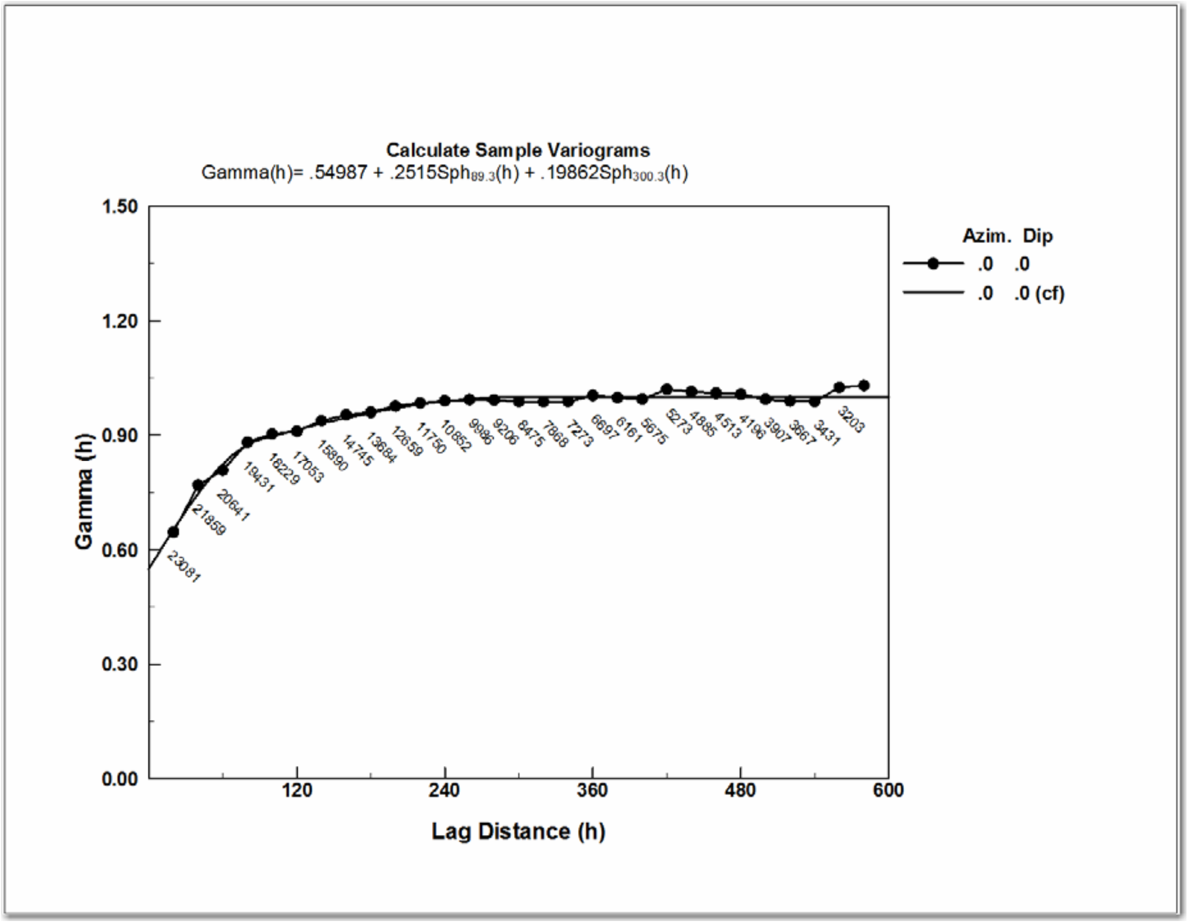


Figure 14-17: Downhole correlogram for Au within all zones.

Geostatistical analyses were also performed on the composites in all directions using no constraints in addition to the coded intervals within the zone solids split out by oxide/sulphide.

The ellipsoid direction chosen for the estimation process was chosen to be 0 degrees azimuth and 0 degrees dip for the major axis and -90 degrees for the vertical axis as there appears to be no predominant direction of mineralization. The 39A Zone appears to dip 270 degrees at an azimuth of 30 degrees so this direction was utilized for the 39A Zone. The Porphyry Zone appears to strike at 45 degrees at an azimuth of -30 degrees and a rotated vertical orientation of -20 degrees.

The spatial continuity estimator chosen for this study was the correlogram, which has been shown in previous work to be more robust with respect to drift and data variability, allowing a better estimation of the observed continuity (Parker and Srivastava, 1988). Note that the sill of the variograms has been standardized to one, and therefore they are in fact relative variograms.

14.9 BLOCK MODEL DEFINITION

The Block Model used for calculating the resources was defined according to the following limits:

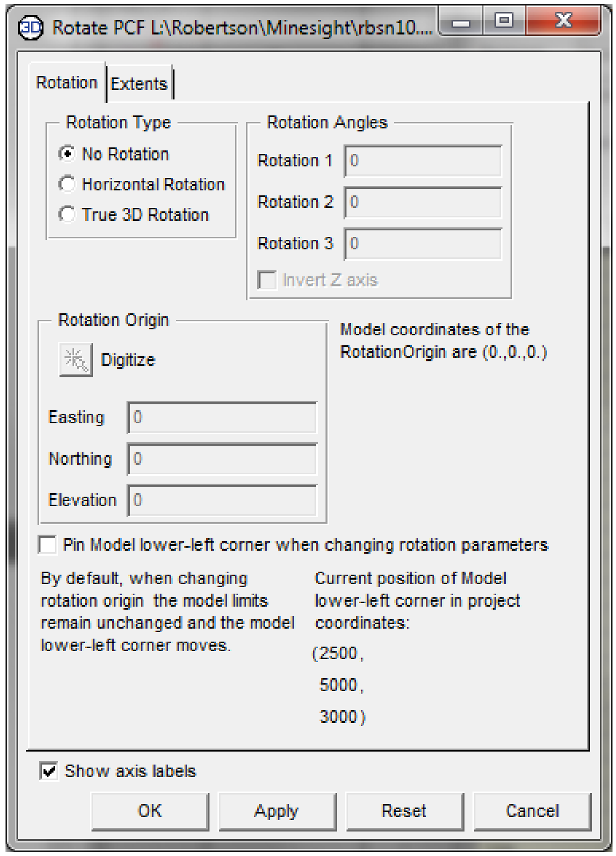


Figure 14-18: Block Model Origin

The block model is non-rotated as there appears to be no predominant orientation that reflected regional or local mineralization trends. Figure 14-18 shows the position and orientation of the block model used for the study. The block size chosen was 25 x 25 x 20 foot to roughly reflect drill hole spacing available and to adequately descretize the deposit.

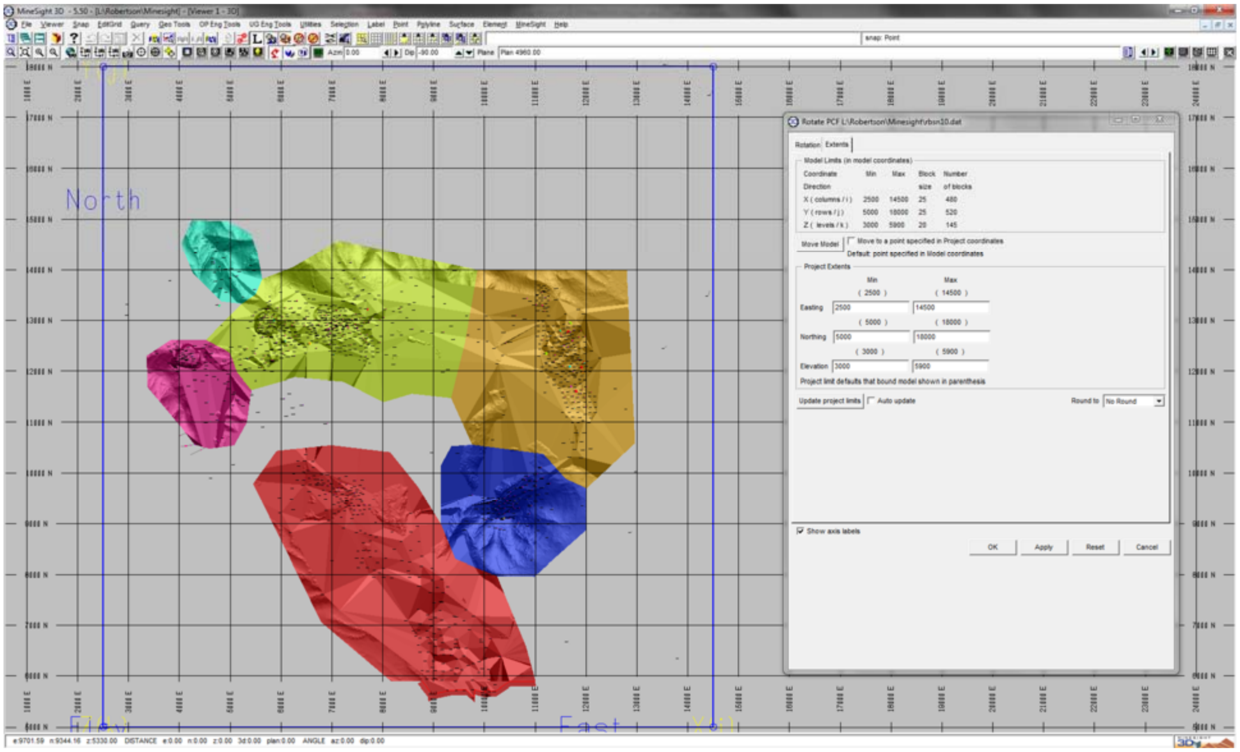


Figure 14-19: Block Model Limits.

Of the potential 36,946,000 blocks to be estimated (490 columns, 520 rows, 145 levels); less than 1,327,655 blocks or 3.6% have estimated values in them (weighted against topography). This is primarily due the geologic constraints applied to the estimation process in addition to the limited search distances applied, search ellipsoid direction and the use of ordinary kriging as the modeling method.

14.10 INDICATOR ESTIMATION

To further domain the mineralized material within the large, broad zones the author created iso-volumes of ordinary kriged probability indicators at a 0.005 ozAu/t cut-off grade within which to constrain the grade estimation. This methodology entails coding all grades above 0.005 ozAu/t with 1 and those below with a zero. Variography is then performed and the values are then kriged using the correlogram parameters which results in the blocks being assigned with values between 0 and 1 which represents the probability that the block will have a grade greater than 0.005 ozAu/t (i.e. 0.5 represents essentially 50% probability). Those blocks that have an indicator greater than 0.5 are then clustered into iso-volumes which are edited in order to exclude isolated blocks and portions of the iso-volumes that have tails.

These iso-volumes as shown in Figure 14-19, are then used to further domain the zones and constrain the resource interpolation

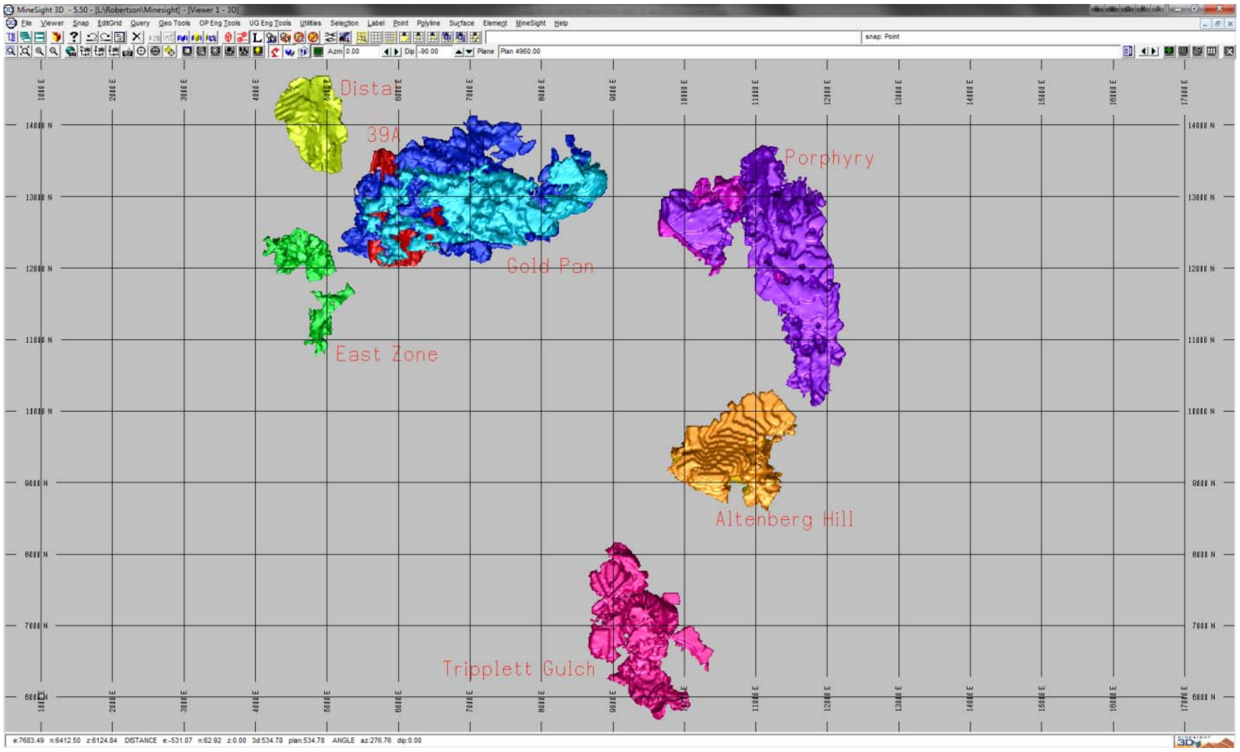


Figure 14-20: 0.005 ozAu/t Indicator Isovolumes

14.10.1 Resource Interpolation

The estimation plan includes the following items:

- Storage of the mineralized zone and iso-volume code and percentage of mineralization.
- Application of density based on limited SG measurements.
- Estimation of the grades for Au using ordinary kriging.
- Ellipsoid orientation was orthogonal and ranges were set to 300 feet in the northing and easting whilst 200 feet in elevation.

The estimation strategy employed a minimum of four composites and a maximum of 15 with a maximum of two from any one drillhole.

Also, an octant search was used as it aids in declustering the estimate. This means that it helps to avoid over-influence of individual drill holes or sectors being overly informed, avoiding the use of samples that clustered together and thereby redundant. The maximum number of composites allowed in any one octant was two.

The resultant block model is illustrated in plan view in Figure 14-20 at a 0.0147 OzAu/t cut-off grade through one level.

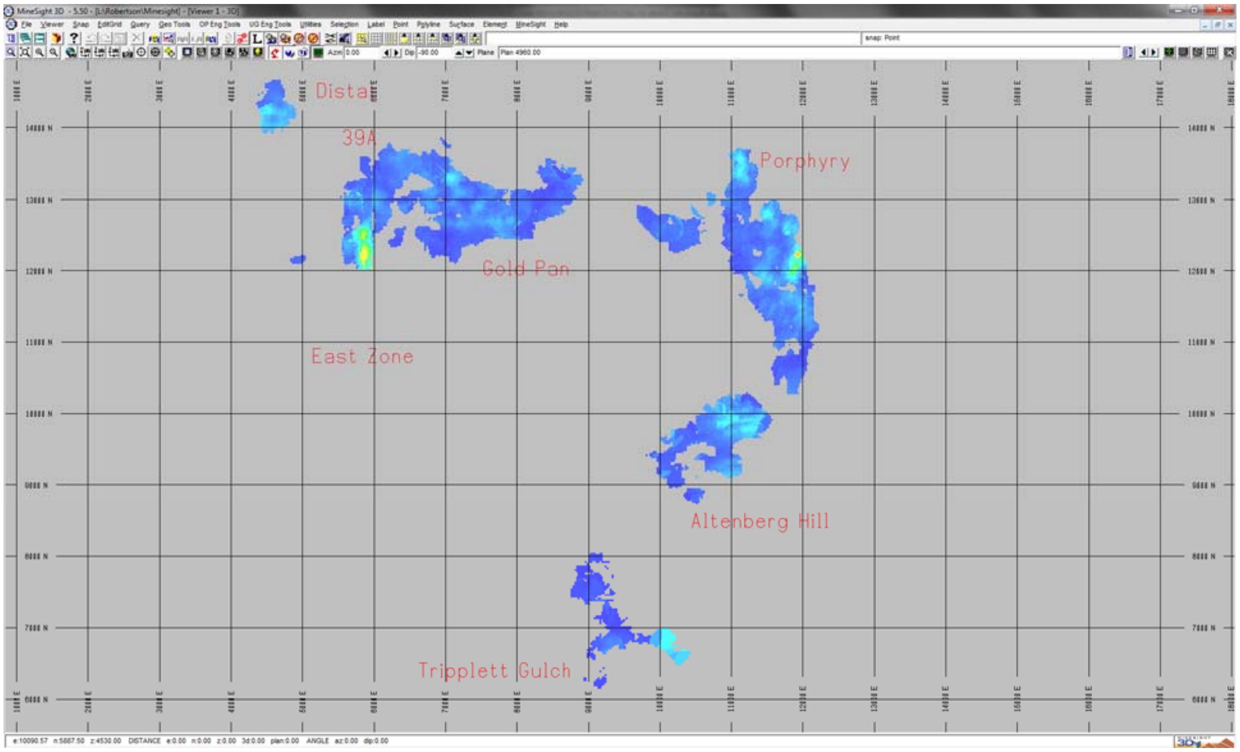


Figure 14-21: Plan View of Grade Model Displaying Block Model with Drillholes.

14.11 RESOURCES

The inferred resources as defined by the parameters detailed above are listed in the Table 14.5 for Au. These resources are listed at a cut-off grade of 0.0147 ozAu/tAu for each zone and split out by oxide and sulphide.

Table 14.5: Inferred Resources for Robertson

ZONE	TONS	ozAu/t	Ounces
39A	26,779,714	0.0230	615,933
GP Oxide	21,939,550	0.0127	278,632
GP Sulphide	48,759,224	0.0119	580,235
Porphyry Oxide	59,707,994	0.0137	818,000
Porphyry Sulphide	9,817,623	0.0132	129,593
Altenberg Hill Oxide	23,170,083	0.0131	303,528
Altenberg Hill Sulphide	178,279	0.0087	1,551
Totals	190,352,467	0.0143	2,727,472

ZONE	TONS	ozAu/t	Ounces
Triplett Gulch Sulphide	678,279	0.0152	10,310
East Zone Sulphide	694,672	0.0171	11,879
Totals	1,372,951	0.0162	22,189

Grand Totals	191,725,418	0.0143	2,741,673

- Gold ounces were calculated on the basis of US$1,350/oz Au and 70% Au recovery.
- The 0.0067 ozAu/t cut-off grade utilized to report the resource was derived from a mining cost of US$1.02/ton, process cost of US$5.00/ton and waste cost of US$1.14/ton.
- The mineral resources in the table above were estimated using the CIM Standards on Mineral Resources and Reserves.

14.11.1 Model Validation

A graphical validation was done on the block model. This graphical validation serves several purposes:

- Checks the reasonableness of the estimated grades, based on the estimation plan and the nearby composites.
- Checks that the general drift and the local grade trends of the block model, compared to the drift and local grade trends of the composites.
- Insure that all blocks that should be filled in, are in fact filled in.
- Checks that topography has been properly accounted for.
- Checks against manual "ballbark" estimates of tonnage to determine reasonableness.
- Inspection of and an explanation for high grade blocks created as a result of outliers.

A full set of cross sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composite. No evidence of any block being wrongly estimated was found: it appears that every block grade can be explained as a function of the surrounding composites, the correlogram models used, and the estimation plan applied.

Table 14.6 lists the total inferred resources for all zones by cut-off grade.

Table 14.6: Inferred Resources for Robertson

ZONE	CUTOFF	TONS	ozAu/t
39A	0.005	27,269,468	0.0227
	0.0067	**26,779,714**	**0.0230**
	0.0105	22,062,501	0.0260
	0.0113	20,399,591	0.0273
	0.0123	18,926,230	0.0285
	0.0134	17,553,279	0.0297
	0.0147	16,243,853	0.0309
	0.0163	13,967,214	0.0334
	0.0184	12,081,968	0.0359
	0.021	10,609,632	0.0382
GP Oxide	0.005	22,850,411	0.0124
	0.0067	**21,939,550**	**0.0127**
	0.0105	11,684,427	0.0161
	0.0113	9,389,345	0.0174
	0.0123	7,649,591	0.0186
	0.0134	6,397,541	0.0197
	0.0147	5,360,656	0.0208
	0.0163	3,685,451	0.0233
	0.0184	2,665,984	0.0255
	0.021	1,907,787	0.0279
GP Sulphide	0.005	52,093,240	0.0115
	0.0067	**48,759,224**	**0.0119**
	0.0105	25,806,354	0.0149
	0.0113	20,863,730	0.0158
	0.0123	16,542,009	0.0168
	0.0134	13,094,263	0.0178
	0.0147	10,054,304	0.0189
	0.0163	5,872,951	0.0214
	0.0184	3,391,394	0.0243
	0.021	2,052,254	0.0275
Porphyry Oxide	0.005	62,360,659	0.0134
	0.0067	**59,707,994**	**0.0137**
	0.0105	37,720,289	0.0166
	0.0113	32,082,993	0.0175
	0.0123	27,148,567	0.0186
	0.0134	23,005,124	0.0196
	0.0147	19,389,345	0.0206

	0.0163	13,703,894	0.0227
	0.0184	9,735,656	0.0249
	0.021	6,929,304	0.0271
Porphyry Sulphide	0.005	10,214,140	0.0130
	0.0067	**9,817,623**	**0.0132**
	0.0105	6,383,197	0.0158
	0.0113	5,652,664	0.0164
	0.0123	4,943,648	0.0170
	0.0134	4,128,074	0.0178
	0.0147	3,343,238	0.0187
	0.0163	2,022,541	0.0209
	0.0184	1,299,180	0.0228
	0.021	764,344	0.0253
Altenberg Hill Oxide	0.005	23,582,993	0.0129
	0.0067	**23,170,083**	**0.0131**
	0.0105	15,528,689	0.0151
	0.0113	12,985,656	0.0159
	0.0123	10,571,722	0.0167
	0.0134	8,464,140	0.0177
	0.0147	6,661,886	0.0187
	0.0163	4,015,369	0.0208
	0.0184	2,408,812	0.0230
	0.021	1,395,492	0.0256
Altenberg Hill Sulphide	**0.0067**	**178,279**	**0.0087**
	0.0105	13,320	0.0115
	0.0113	6,148	0.0120
Triplett Gulch Sulphide	**0.0067**	**678,279**	**0.0152**
	0.0105	498,975	0.0174
	0.0113	362,705	0.0198
	0.0123	320,697	0.0209
	0.0134	296,107	0.0215
	0.0147	294,057	0.0216
	0.0163	225,410	0.0233
	0.0184	203,893	0.0239
	0.021	146,516	0.0255
East Zone Sulphide	0.0105	664,959	0.0175
	0.0113	641,393	0.0177
	0.0123	612,705	0.0180
	0.0134	570,697	0.0183

	0.0147	533,811	0.0186
	0.0163	479,508	0.0190
	0.0184	440,574	0.0191
	0.021	11,270	0.0213
TOTALS	0.005	199,963,124	0.0140
	0.0067	**191,725,418**	**0.0143**
	0.0105	120,362,710	0.0177
	0.0113	102,384,226	0.0188
	0.0123	86,715,168	0.0201
	0.0134	73,509,225	0.0214
	0.0147	61,881,150	0.0227
	0.0163	43,972,338	0.0257
	0.0184	32,227,460	0.0287
	0.021	23,816,599	0.0320

Validation techniques are as follows:

- Visual inspections on a section-by-section and plan-by-plan basis.
- Histogram at varying cut-off grades demonstrate a very uniform, normal distribution.
- The use of Grade Tonnage Curve as shown in Figures 14-20.
- Swath Plots showing the comparison of the Ordinary Kriged blocks vs. Inverse Distance and Nearest Neighbor estimates.
- An inspection of histograms of distance of first composite to nearest block, average distance to blocks for all composites.
- Analysis of Relative Variability Index which quantifies variability within the deposit.

It is important to note the nearest distance to composite, average distance to composite and RVI as useful tools for categorizing resources may be utilized for future resource classification as a measure of quantifying risk and qualifying resources.

14.12 PIT OPTIMIZATION

A series of pit optimizations were run at varying price scenarios. For the purposes of this report, the following parameters were used as the basis for the optimization which was the basis for the preliminary pit design:

- Ore mining cost of US$1.27 per ton
- Waste mining cost of US$1.43 per ton
- Processing cost of US$6.25 per ton
- Metallurgical recovery of 70%
- Gold price of US$1,000 per ounce
- Tonnage factor of 12.2 cubic foot per ton
- Pit slope angles of 45 degrees

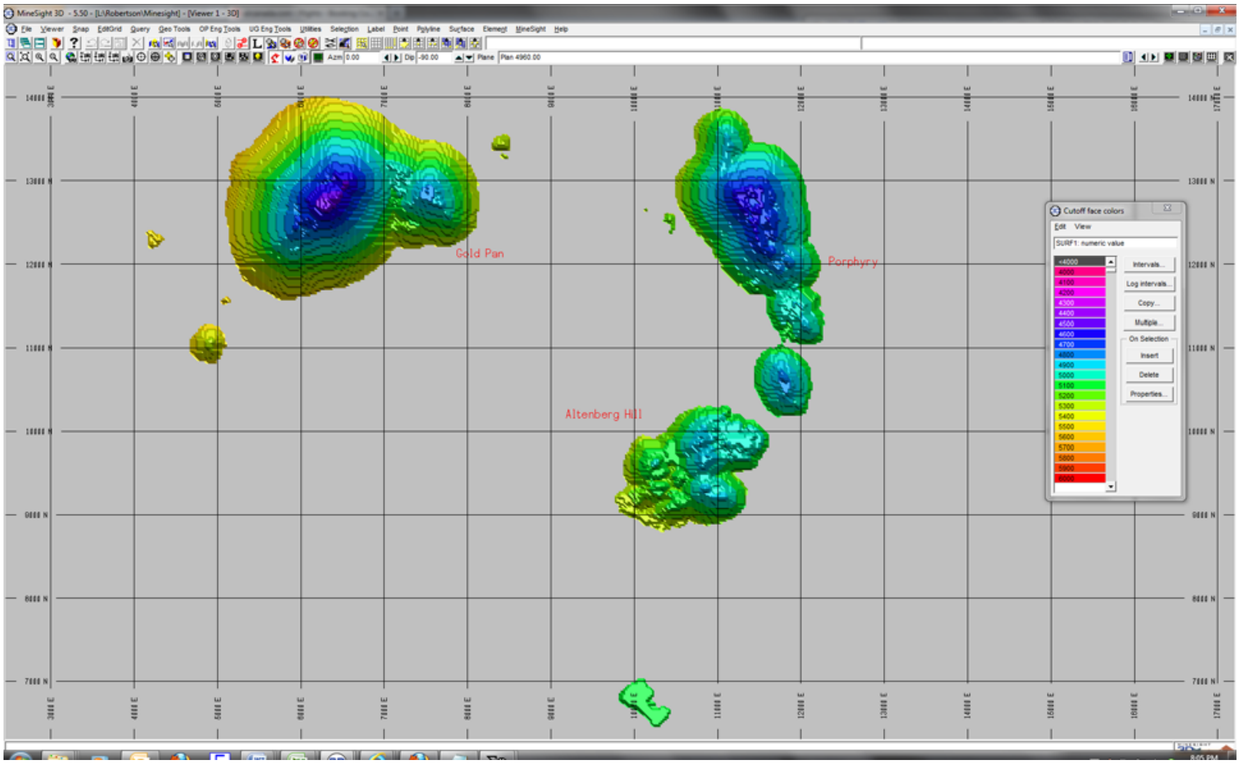


Figure 14-22: Pit Shells derived from Lersch-Grossman Pit Optimization

14.13 PIT DESIGN AND RESOURCES

Based on the pit optimization study as described above, the preliminary pit design was created using the following parameters:

- Bench interval — 20 feet at double benches for a total of 40 feet
- Berm width — 25 feet
- Pit slope angle — 45 degrees
- Slope angle bench — 70 degrees
- Roadway gradient — 8%
- Roadway width — 105 feet

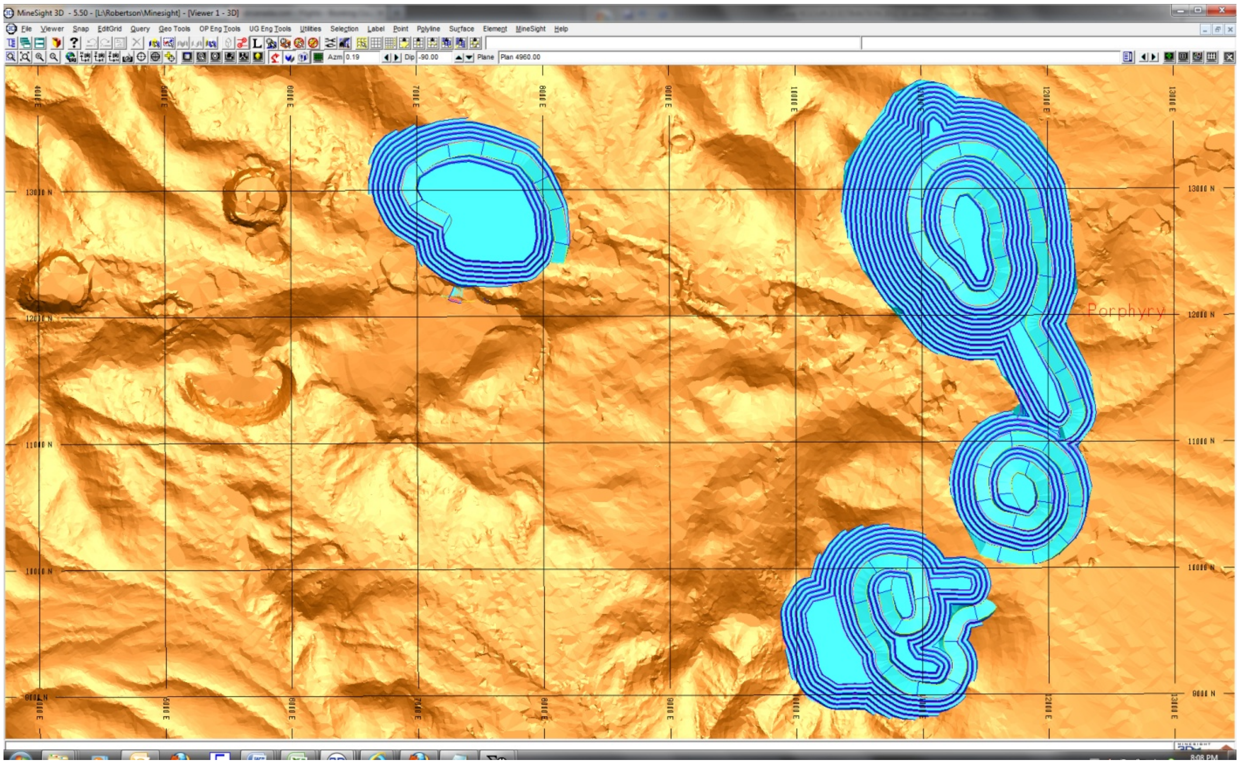


Figure 14-23: Benched Pit Designs with Haul Roads

The tables below list the inferred resources within the Altenburg Hill, Porphyry and Gold Pan design pits, respectively.

Table 14.7: Inferred Resources Altenburg Hill

AH										
BENCH TOE	ORE TONS	WASTE TONS	S/R	AUOK	ORE TONS	WASTE TONS	S/R	AUOK	LOW GRADE TONS	LOW GRADE AUOK
5440	-	4,355	-1	-1	-	4,355	-1	-1	-	
5420	28,945	21,568	0.8	0.0174	43,576	6,936	0.2	0.0156	14,631	0.012
5400	31,650	122,121	3.9	0.0166	143,340	10,430	0.1	0.0122	111,690	0.011
5380	23,648	310,635	13.1	0.0154	310,697	23,586	0.1	0.0102	287,049	0.010
5360	42,295	464,160	11.0	0.0158	443,873	62,582	0.1	0.0103	401,578	0.010
5340	86,619	584,180	6.7	0.0170	529,180	141,619	0.3	0.0116	442,561	0.011
5320	84,764	704,426	8.3	0.0165	616,168	173,023	0.3	0.0114	531,404	0.011
5300	77,121	932,439	12.1	0.0165	776,281	233,279	0.3	0.0113	699,160	0.011
5280	94,928	1,050,256	11.1	0.0178	845,953	299,232	0.4	0.0112	751,025	0.010
5260	169,242	1,214,703	7.2	0.0172	938,146	445,799	0.5	0.0115	768,904	0.010
5240	229,180	1,292,121	5.6	0.0178	1,036,363	484,939	0.5	0.0115	807,183	0.010
5220	509,990	1,268,043	2.5	0.0194	1,197,848	580,184	0.5	0.0145	687,858	0.011
5200	624,160	1,319,365	2.1	0.0221	1,248,555	694,969	0.6	0.0162	624,395	0.010
5180	561,762	1,643,781	2.9	0.0228	1,350,400	855,143	0.6	0.0155	788,638	0.010
5160	368,689	1,860,625	5.1	0.0178	1,342,879	886,434	0.7	0.0125	974,190	0.010
5140	180,953	1,808,391	10.0	0.0157	1,156,527	832,818	0.7	0.0116	975,574	0.011
5120	309,744	1,444,641	4.7	0.0170	1,143,299	611,086	0.5	0.0123	833,555	0.011
5100	453,463	1,176,045	2.6	0.0186	1,123,822	505,686	0.5	0.0137	670,359	0.010
5080	419,775	926,537	2.2	0.0189	1,046,496	299,816	0.3	0.0141	626,721	0.011
5060	356,486	888,812	2.5	0.0172	1,016,158	229,139	0.2	0.0133	659,672	0.011
5040	255,881	733,494	2.9	0.0175	828,709	160,666	0.2	0.0133	572,828	0.011
5020	278,893	612,080	2.2	0.0174	721,271	169,703	0.2	0.0139	442,378	0.012
5000	99,682	553,289	5.6	0.0181	546,773	106,199	0.2	0.0123	447,091	0.011
4980	128,053	384,488	3.0	0.0206	413,330	99,211	0.2	0.0135	285,277	0.010
4960	69,344	232,326	3.4	0.0223	245,133	56,537	0.2	0.0145	175,789	0.011
4940	55,205	171,988	3.1	0.0202	194,416	32,777	0.2	0.0138	139,211	0.011
4920	17,100	92,090	5.4	0.0168	109,057	133	0.0	0.0114	91,957	0.010
TOTAL	**5,557,572**	**21,816,957**	**3.9**	**0.0189**	**19,368,249**	**8,006,281**	**0.4**	**0.0130**	**13,810,677**	**0.011**

Table 14.8: Inferred Resources Porphyry

PORPHYRY										
BENCH TOE	**ORE TONS**	**WASTE TONS**	**S/R**	**AUOK**	**ORE TONS**	**WASTE TONS**	**S/R**	**AUOK**	**LOW GRADE TONS**	**LOW GRADE AUOK**
5360	-	47,008	-1	-1	-	47,008	-1	-1		
5340	-	193,596	-1	-1	-	193,596	-1	-1		
5320	-	350,809	-1	-1	-	350,809	-1	-1		
5300	-	584,672	-1	-1	-	584,672	-1	-1		
5280	-	787,961	-1	-1	25,133	762,828	30.4	0.0082	25,133	0.008
5260	-	1,077,264	-1	-1	66,414	1,010,850	15.2	0.0082	66,414	0.008
5240	2,910	1,345,236	462.3	0.0155	127,387	1,220,758	9.6	0.0088	124,477	0.009
5220	24,498	1,757,254	71.7	0.0171	313,115	1,468,637	4.7	0.0096	288,617	0.009
5200	36,742	2,342,695	63.8	0.0189	484,928	1,894,508	3.9	0.0097	448,186	0.009
5180	111,721	3,420,502	30.6	0.0181	1,034,836	2,497,387	2.4	0.0107	923,115	0.010
5160	320,881	4,225,369	13.2	0.0188	1,923,535	2,622,715	1.4	0.0112	1,602,654	0.010
5140	485,164	5,320,000	11.0	0.0196	2,561,978	3,243,186	1.3	0.0119	2,076,814	0.010
5120	647,346	5,323,719	8.2	0.0197	2,928,914	3,042,152	1.0	0.0121	2,281,568	0.010
5100	834,539	5,005,502	6.0	0.0201	3,144,150	2,695,891	0.9	0.0125	2,309,611	0.010
5080	895,922	4,351,752	4.9	0.0202	3,070,646	2,177,029	0.7	0.0131	2,174,724	0.010
5060	989,693	4,058,064	4.1	0.0197	3,003,637	2,044,119	0.7	0.0134	2,013,944	0.010
5040	851,014	3,598,617	4.2	0.0198	2,845,277	1,604,355	0.6	0.0132	1,994,263	0.010
5020	874,150	3,368,350	3.9	0.0185	2,768,463	1,474,037	0.5	0.0129	1,894,313	0.010
5000	925,646	2,757,234	3.0	0.0190	2,485,461	1,197,418	0.5	0.0137	1,559,815	0.011
4980	893,996	2,416,055	2.7	0.0198	2,159,283	1,150,768	0.5	0.0145	1,265,287	0.011
4960	879,980	1,973,832	2.2	0.0198	1,984,262	869,549	0.4	0.0149	1,104,282	0.011
4940	897,039	1,776,516	2.0	0.0202	1,927,879	745,676	0.4	0.0151	1,030,840	0.011
4920	840,082	1,434,355	1.7	0.0203	1,752,490	521,947	0.3	0.0151	912,408	0.010
4900	857,193	1,242,920	1.5	0.0221	1,627,039	473,074	0.3	0.0164	769,846	0.010
4880	785,174	938,525	1.2	0.0231	1,455,297	268,402	0.2	0.0171	670,123	0.010
4860	839,160	788,064	0.9	0.0234	1,433,269	193,955	0.1	0.0181	594,109	0.011
4840	719,365	673,545	0.9	0.0224	1,274,918	117,992	0.1	0.0176	555,553	0.011
4820	700,820	602,182	0.9	0.0233	1,198,955	104,047	0.1	0.0180	498,135	0.011
4800	610,031	479,672	0.8	0.0233	1,022,408	67,295	0.1	0.0182	412,377	0.011
4780	555,369	448,156	0.8	0.0227	916,875	86,650	0.1	0.0181	361,506	0.011
4760	545,021	258,607	0.5	0.0217	749,795	53,832	0.1	0.0189	204,774	0.011
4740	452,777	258,781	0.6	0.0220	657,469	54,088	0.1	0.0187	204,692	0.011
4720	343,391	197,838	0.6	0.0240	509,898	31,332	0.1	0.0197	166,507	0.011
4700	217,879	244,303	1.1	0.0246	397,818	64,365	0.2	0.0186	179,939	0.011
4680	131,998	171,004	1.3	0.0226	239,857	63,145	0.3	0.0177	107,859	0.012
4660	68,371	157,992	2.3	0.0228	134,857	91,506	0.7	0.0169	66,486	0.011
TOTAL	**17,337,870**	**63,977,952**	**3.7**	**0.0209**	**46,226,242**	**35,089,580**	**0.8**	**0.0143**	**28,888,372**	**0.010**

Table 14.9: Inferred Resources Gold Pan

GP										
BENCH TOE	**ORE TONS**	**WASTE TONS**	**S/R**	**AUOK**	**ORE TONS**	**WASTE TONS**	**S/R**	**AUOK**	**LOW GRADE TONS**	**LOW GRADE AUOK**
5540		1,393	-1.0	1.0000	-	1,393	1.0	-1.0000		
5520	-	32,213	-1.0	1.0000	-	32,213	1.0	-1.0000		
5500	-	53,914	-1.0	1.0000	-	53,914	1.0	-1.0000		
5480	-	29,477	-1.0	1.0000	13,740	115,738	8.4	0.0080	13,740	0.008
5460	5,748	267,090	46.5	0.0202	115,543	157,295	1.4	0.0088	109,795	0.008
5440	18,863	545,574	28.9	0.0222	242,654	321,783	1.3	0.0099	223,791	0.009
5420	5,799	1,122,684	193.6	0.0219	302,357	826,127	2.7	0.0097	296,558	0.009
5400	53,525	1,631,404	30.5	0.0177	458,207	1,226,721	2.7	0.0105	404,682	0.010
5380	190,420	1,789,836	9.4	0.0190	836,209	1,144,047	1.4	0.0118	645,789	0.010
5360	181,824	1,982,807	10.9	0.0178	1,098,361	1,066,270	1.0	0.0111	916,537	0.010
5340	209,119	1,781,752	8.5	0.0173	1,201,926	788,945	0.7	0.0113	992,807	0.010
5320	360,697	1,534,375	4.3	0.0201	1,321,639	573,432	0.4	0.0129	960,942	0.010
5300	359,570	1,286,168	3.6	0.0231	1,396,486	249,252	0.2	0.0133	1,036,916	0.010
5280	433,504	1,118,402	2.6	0.0249	1,385,113	166,793	0.1	0.0148	951,609	0.010
5260	555,420	766,219	1.4	0.0255	1,205,584	116,055	0.1	0.0174	650,164	0.010
5240	586,834	645,523	1.1	0.0239	1,140,594	91,762	0.1	0.0176	553,760	0.011
5220	422,725	592,172	1.4	0.0223	984,764	30,133	0.0	0.0154	562,039	0.010
5200	229,262	696,516	3.0	0.0197	911,148	14,631	0.0	0.0125	681,886	0.010
TOTAL	**3,613,310**	**15,977,521**	**4.4**	**0.0223**	**12,614,324**	**6,976,506**	**0.6**	**0.0135**	**9,001,014**	**0.010**

SECTION 15.0 MINERAL RESERVE ESTIMATES

There are no mineral reserves.

SECTION 16.0 MINING METHODS

16.1 MINING

16.1.1 Resources and Production Rates

The estimated open pit resources for the Altenburg Hill, Porphyry and Gold Pan deposits contained within the Robertson property are described in Section 14. These resources were used to derive the yearly mineral production rate based upon mining a maximum 40,000tpd (14 million tons per year). This production rate was chosen based on the configuration of the mineralized zones and a mine life that balances capital expenditures against operating costs to provide the optimum return on investment. The open pit design incorporates mining the Altenburg Hill and Porphyry deposits first as a combined operation. This is based upon the following;

- size;
- location and proximity; and
- strip ratios.

Size

The resources estimated for the Altenburg Hill and the configuration of the deposit is relatively small and to mine this independently would not allow for the production rate developed within this plan to be achieved. Thus the Porphyry commences production at the same time as the Altenburg Hill and by combining the production from each open pit the required production rate is achieved while meeting practicable mining criteria and acceptable operating costs.

Location and Proximity

The Altenburg Hill and Porphyry deposits lie adjacent to each other and close to the proposed location for the leach pads to the south and the proposed waste rock disposal area. The location of the ramp entrance to each pit can be accessed by a single haul road located between the pits. This allows for an interchange between open pit mining equipment since it can be considered as a single pit with two access ramps and provides for a flexible and efficient mining operation.

Strip Ratios

When starting a mine operation such as envisaged at Robertson it is advantageous to minimize the strip ratio and maximize grade; often a starter pit will be established to enable this to occur. Unfortunately the present resource estimate does not allow for this to be achieved since mining the higher grade material only in year 1 and 2 amounts to approximately 1.9 million tons. The grade difference between HG and LG is only 0.008 ozAu/t and the extra cost involved in mining this material as a starter pit would be more than offset by mining lower grade material in subsequent years. Also the Porphyry pit does not readily lend itself to a starter pit.

The Altenburg Hill deposit is basically a mineralized hill and thus has a low strip ratio, on average 0.4: 1 waste to ore with a ratio of 0.35:1 for the initial two years whereas the Porphyry averages 2.44 for the first two years. Thus mining the two deposits together reduces the initial strip ratio significantly.

These mineral resource estimates are designated as inferred[5].

Table 16.1 shows the yearly production schedule.

16.1.2 Pit Design

Open pit operations are planned as a conventional truck-and-shovel operation, utilizing diesel-powered equipment to maintain flexibility and minimize capital costs. Several combinations of truck/shovel size ranges were evaluated, with the most cost effective being a 40 cuft excavator in combination with100 ton haul trucks. Overall pit design was based on an overall slope angle of 45° for all walls. This is considered conservative since the geotechnical evaluation indicated 50°. Ore and waste will be mined on 20 ft benches. Bench width on the northwest and south walls will be 25 ft; on the east and west it will be 30 ft. The overall height between benches will be 40 ft, with a bench face angle of 70°.

The crusher will be located at the lower elevation below and to the south east of the Porphyry pit. and adjacent to the HG leach pad. These locations have been chosen to minimize haul distances for waste and rock (see Figure 1.2). The haul road within the pit has a maximum grade of 10%.

16.1.3 Mining Operations

Mining activities at the pits will be conducted on 7 days per week, two 10-hour shifts per day, 350 days per year. (based upon 52 weeks, 7 days per week gives 365 days less 15 statutory holidays). The study has been based upon owner-purchased and owner-operated equipment. Budget costs for a contractor operated open pit operation were obtained from a USA contractor who has experience in Nevada

Ore and waste will be drilled and blasted using 6.5 in. diameter holes on a 15 ft x 15 ft pattern.

Blasting operations will use a combination of ammonium nitrate and fuel oil (ANFO) and water-resistant emulsion explosives. Estimated powder factors range between 0.31 and 0.33 kg/t material. There should be no water concerns with the pit envelopes since the pit bottoms are above the water table. Precipitation, due to the low rain/snow fall, is not considered a concern thus it has been assumed that about 20% of the holes will be "wet."

[5] Due to the uncertainty that may be attached to an inferred mineral resource, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured resource as a result of continued exploration.

Table 16.1: Production Schedule by Year

	Year												
Altenburg	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**Total**
Ore	375,042	1,003,340	1,554,611	1,363,935	990,942	252,602	17,100						**5,557,572**
Low Grade	2,488,073	3,014,970	2,387,223	3,106,209	2,121,969	600,277	91,957						**13,810,678**
Waste	655,810	1,810,154	2,436,546	2,249,406	665,707	188,525	133						**8,006,281**
Ore/day	8,180	11,481	11,262	12,772	8,894	2,437	312						**9,223**
Total/day	**10,054**	**16,653**	**18,224**	**19,199**	**10,796**	**2,975**	**312**						**13,035**
Strip Ratio	0.23	0.45	0.62	0.50	0.21	0.22	0.00						**0.41**
Insitu Oz	32,104	49,550	57,944	57,630	41,288	11,817	1,243						**251,577**
Recv oz	17,940	27,689	32,379	32,204	23,072	6,604	695						**140,581**
Grade HG	0.017	0.019	0.021	0.018	0.017	0.021	0.017						**0.019**
Grade LG	0.010	0.010	0.010	0.011	0.011	0.011	0.010						**0.011**
Porphyry													**Total**
Ore	64,150	432,602	1,132,510	834,539	1,885,615	2,650,810	3,511,097	4,511,743	2,114,437	200,369			**17,337,872**
Low Grade	952,827	2,525,769	2,076,814	3,581,568	3,934,335	5,652,520	5,372,632	3,500,143	1,117,418	174,345			**28,888,371**
Waste	5,583,903	4,120,102	3,243,186	3,042,152	4,872,920	5,122,511	3,963,411	1,679,417	357,562	154,651			**32,139,815**
Ore/day	2,906	8,452	9,169	12,617	16,628	23,724	25,382	22,891	9,234	1,071			**14,675**
Total/day	**18,860**	**20,224**	**18,436**	**21,309**	**30,551**	**38,360**	**36,706**	**27,689**	**10,255**	**1,512**			**24,878**
Strip Ratio	5.49	1.39	1.01	0.69	0.84	0.62	0.45	0.21	0.11	0.41			**0.70**
Insitu Oz	9,581	32,616	43,240	52,141	77,298	109,159	127,635	140,139	60,506	6,525			**658,840**
Recv oz	5,354	18,226	24,163	29,137	43,194	60,998	71,322	78,310	33,811	3,646			**368,160**
Grade HG	0.018	0.019	0.020	0.020	0.020	0.019	0.020	0.023	0.023	0.023			**0.020**
Grade LG	0.009	0.010	0.010	0.010	0.010	0.010	0.011	0.011	0.011	0.011			**0.010**
Gold Pan													**Total**
Ore								83,935	581,363	2,296,025	651,987		**3,613,310**
Low Grade								1,048,566	2,555,133	4,153,391	1,243,925		**9,001,015**
Waste								2,735,184	2,999,262	1,197,294	44,764		**6,976,504**
Ore/day								3,236	8,961	18,427	5,417		**8,009**
Total/day								**11,051**	**17,531**	**21,848**	**5,545**		**12,439**
Strip Ratio								2.42	0.96	0.19	0.02		**0.55**
Insitu Oz								11,273	35,641	97,174	26,555		**170,642**
Recv oz								6,299	19,916	54,301	14,839		**95,355**
Grade HG								0.019	0.018	0.024	0.021		**0.022**
Grade LG								0.009	0.010	0.010	0.010		**0.010**
Totals													**Total**
Ore	439,192	1,435,942	2,687,121	2,198,474	2,876,557	2,903,412	3,528,197	4,595,678	2,695,800	2,496,394	651,987		**26,508,754**
Low Grade	3,440,900	5,540,739	4,464,037	6,687,777	6,056,304	6,252,797	5,464,589	4,548,709	3,672,551	4,327,736	1,243,925		**51,700,064**
Waste	6,239,713	5,930,256	5,679,732	5,291,558	5,538,627	5,311,036	3,963,544	4,414,601	3,356,824	1,351,945	44,764		**47,122,600**
Ore/day	11,086	19,933	20,432	25,389	25,522	26,161	25,694	26,127	18,195	19,498	5,417		**21,281**
Total/day	**28,914**	**36,877**	**36,660**	**40,508**	**41,347**	**41,335**	**37,018**	**38,740**	**27,786**	**23,360**	**5,545**		**34,104**
Strip Ratio	1.61	0.85	0.79	0.60	0.62	0.58	0.44	0.48	0.53	0.20	0.02		**0.60**
Insitu Oz	41,686	82,167	101,184	109,772	118,586	120,977	128,878	151,412	96,146	103,698	26,555		**1,080,921**
Recv oz	23,294	45,915	56,542	61,340	66,266	67,602	72,017	84,609	53,727	57,947	14,839		**604,096**
Grade HG	0.017	0.019	0.021	0.019	0.019	0.019	0.020	0.023	0.022	0.024	0.021		**0.021**
Grade LG	0.010	0.010	0.010	0.010	0.011	0.010	0.011	0.010	0.010	0.010	0.010		**0.010**

In addition to the 35 cu ft hydraulic loading shovel and 100 ton rigid body haul trucks, several pieces of auxiliary and service equipment will be required to maintain haul roads and waste dumps, and for general pit operations. A list of all major equipment is shown in Table 16.2.

Table 16.2: List of Major Equipment

Description	# Req'd
Atlas Copco Viper Drill	2
Komatsu PC1800 Shovel	1
Komatsu WA90	1
Komatsu HD785-7	8
Komatsu HD155AX-6	2
Komatsu GD655	1
Komatsu 600-6	1
Air trac/compressor	1
Komatsu HD325-7	2
Service truck	2
Crane truck	1
Blaster's truck	2
Light plant	4
Pick ups	5
Pump truck	1

16.2 MANPOWER

The manpower estimates for operations are based upon similar mining operations, the type and number of pieces of equipment, and the general operations of the facilities.

The estimate of persons employed directly by the project is 145 persons. A breakdown of personnel is shown in Tables 16.3, 16.4, 16.5.16.6 and 16.7.

In addition to direct employment, the project will create approximately 3 to 4 times more indirect jobs through associated service providers and mine suppliers.

Table 16.3: Administration Staff

Description	No. Req'd
General Manager	1
Mine Superintendent	1
Maintenance Superintentent	1
GM Secretary	1
Chief Accountant	1
Purchasing Agent	1
Personnel/Office Manager	1
Environmental Co-ordinator	1
Safety Director	1
Safety Trainers	2
Acct/purch/admin assistants	4
Total	**15**

Table 16.4: Surface Operations

Description	No. Req'd
Equipment Operators	2
Utility Personnel	2
Maintenance	4
Misc. Labourers	4
Sub-total	12
Total	**27**

Table 16.5: Mine Staff

Description	Mine Staff Required
Mine Foreman	1
D&B Foreman	1
Mech Foreman	1
Sr. Engineer	1
Engineer	2
Surveyor	2
Sr Geologist	1
Geologist	2
Sampler	4
Clerical	4
Labourers	4
Total	**23**

Table 16.6: Mine Operations

Mine Operations – Labour		
Activity	**Category**	**# Required**
Drilling	Drill Operator	2
	Helper	2
Blasting	Blaster	1
	Helper	2
Loading	Operator	2
	Helper	2
Haulage	Truck Drivers	12
Roads / Dumps	Dozer Operator	2
	Grader Operator	2
Maintenance	Lead Mechanic	2
	Mechanics	2
	Serviceman	2
	Labourers	2
Total		**35**

Table 16.7: Process Plant and Leach Pad

Description	# Required
Plant Operations	
Plant operator	8
Plant Helper	8
Refiner Operator	2
Labourer	6
Leach Pad	
Pad Operator	3
Pad Helper	3
Assay Laboratory	
Assayer	2
Sample Preparation	4
Plant Maintenance	
Mechanic/Welder	4
Mechanic Helper	4
Electrician	1
Total	45

SECTION 17.0 RECOVERY METHODS

17.1 PROCESS DESCRIPTION

17.1.1 Introduction

The process description as indicated below is based upon the production rates and proposed mine plan. The general approach is to utilize two heap leach pads one for crushed ore and the other for run-off-mine material. The run-off-mine material will required a closer blast pattern than that required for the ore to be crushed to reduce the size of material to be placed on the leach pad.

17.1.2 Recovery Methods

Ore will be loaded onto two separate heap leach pads based on gold grade, and at a combined rate of 20,000 tons per day (tpd). A high grade pad will be will be employed for ore above a cutoff grade of 0.0147 oz/ton (0.50 g/mt) following crushing to a nominal ½ inch (12.5 mm) particle size. A second pad will operate on run of mine low grade ore projected to be minus 6 inch (150 mm) and using a cut-off grade of 0.0050 oz/ton (0.17 g/mt). Pregnant leachate solution (PLS) from both pads will be directed to a PLS storage pond and pumped to the adsorption desorption recovery (ADR) plant for recovery of gold, along with minor silver credits with on-site Dore production. A simplified flowsheet is provided in Figure 17-1.

The high grade (HG) run of mine ore will be hauled by ore trucks from the open pits to the crusher at an average expected rate of 7,500 tpd. The HG ore will be direct dumped into a 150 t capacity dump pocket through a 20 inch (500 mm) static grizzly screen. Closed circuit crushing will consist of a 200 HP primary 42” X 48” jaw crusher feed feeding a reclaim feeder and conveying to a 200 HP 5.5 foot standard cone crusher for secondary crushing. Secondary crushed product would be screened with the oversize directed to final crushing using a 200 HP 5.5 foot shorthead cone crusher. Final crushed and screened product will be transported to the heap leach by an overland conveyor and a series of portable conveyors to a radial stacker that will place the HG ore onto the pad. There is no agglomeration. The final pad design consists of five lifts, each approximately 10 m high. The run of mine low grade (LG) ore would be trucked at an average rate of 12,500 tpd and transferred to the pad by direct dump methods. ARD plant plan and elevation views are provided in Figures 17-2. Crushing plant plan and elevations are shown on Figures 17-3 and 17-4.

The heap leach pads consist of an underlying compacted layer of low permeability soil or clay overlaid with a 2 mm thick HPDE/LLDPE synthetic liner. Overlying material for liner protection will consist of crushed and screened HG ore. Heap irrigation will be by drip emitters for the HG ore pad and sprinklers for the LG ore pad at a projected rate of ~12 L per h/m^2. Collected PLS will be directed to the pregnant solution ponds. The process solution ponds will be double HDPE lined and equipped with leak detection pumps.

PLS will be pumped to the ADR plant where gold adsorption will be accomplished in two trains, each consisting of five 10 ft (3 m) diameter carbon tanks. Barren solution from the ADR plant will be directed to a barren storage pond and reapplied to the heaps. Gold will be stripped from the collected loaded carbon, followed by standard electrowinning and smelting techniques to produce Dore. Operations will be 365 days per year.

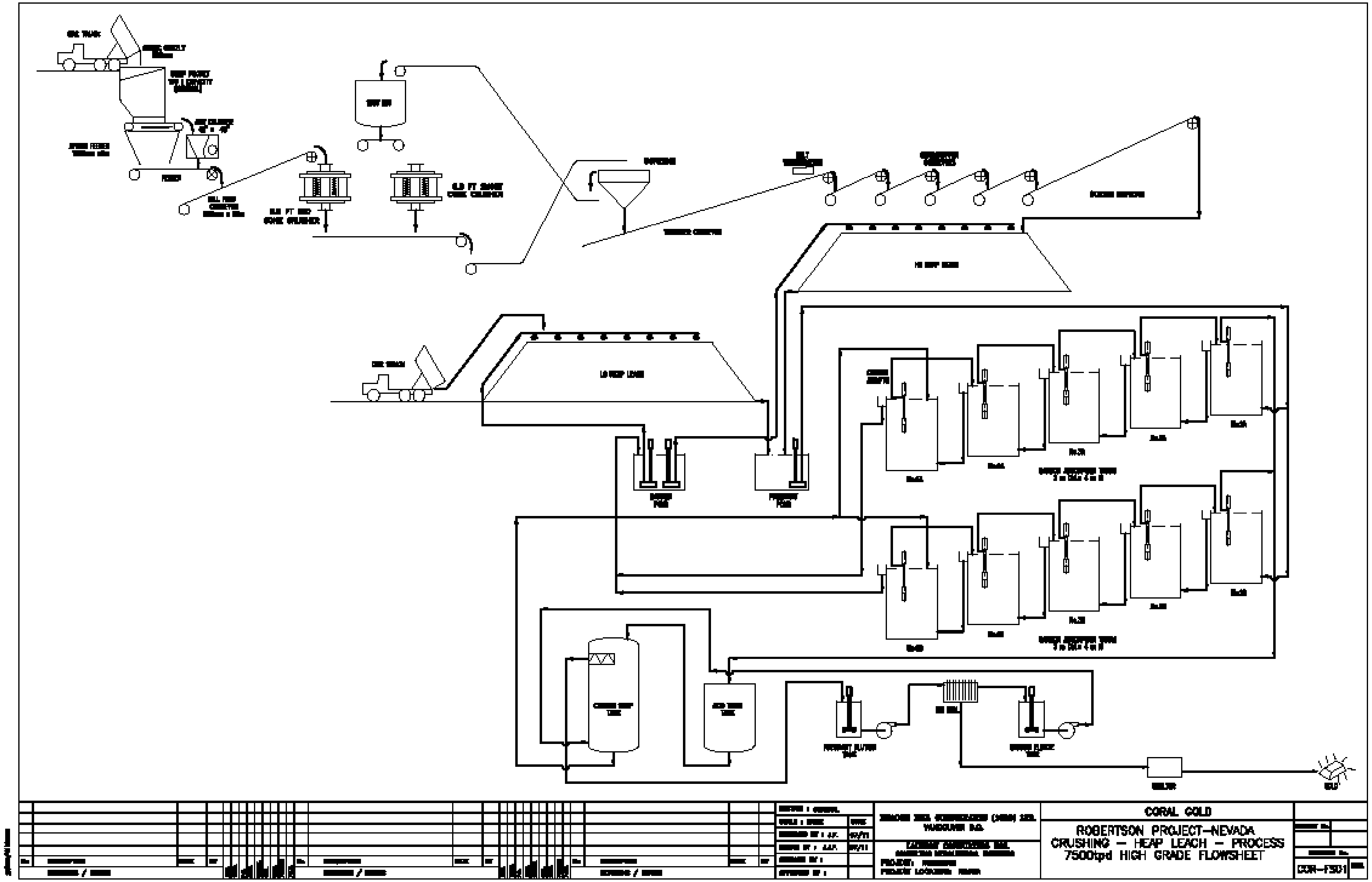


Figure 17-1: Crushing Heap Leach Process 7500tpd

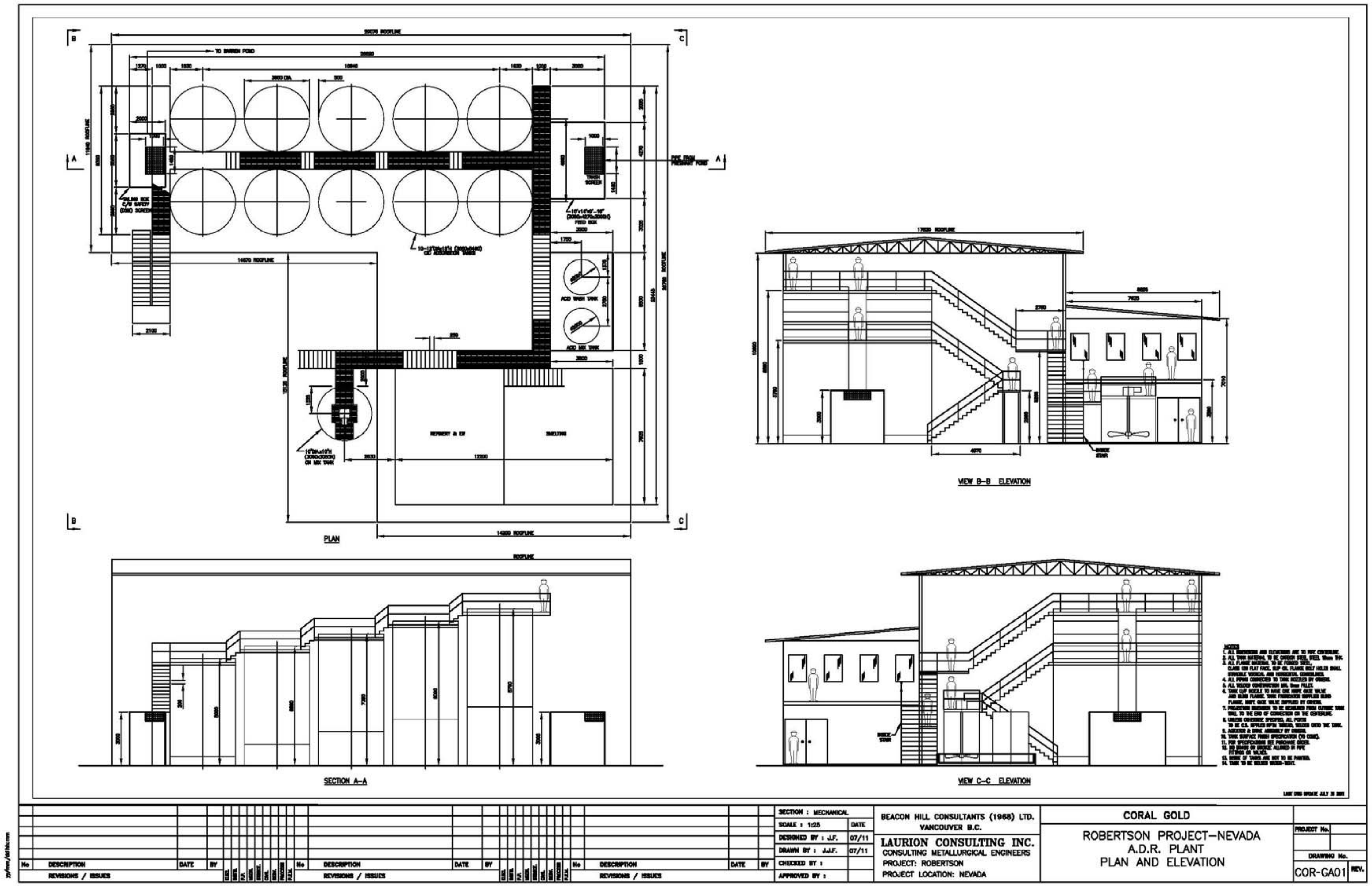


Figure 17-2: ARD Plan and Elevation

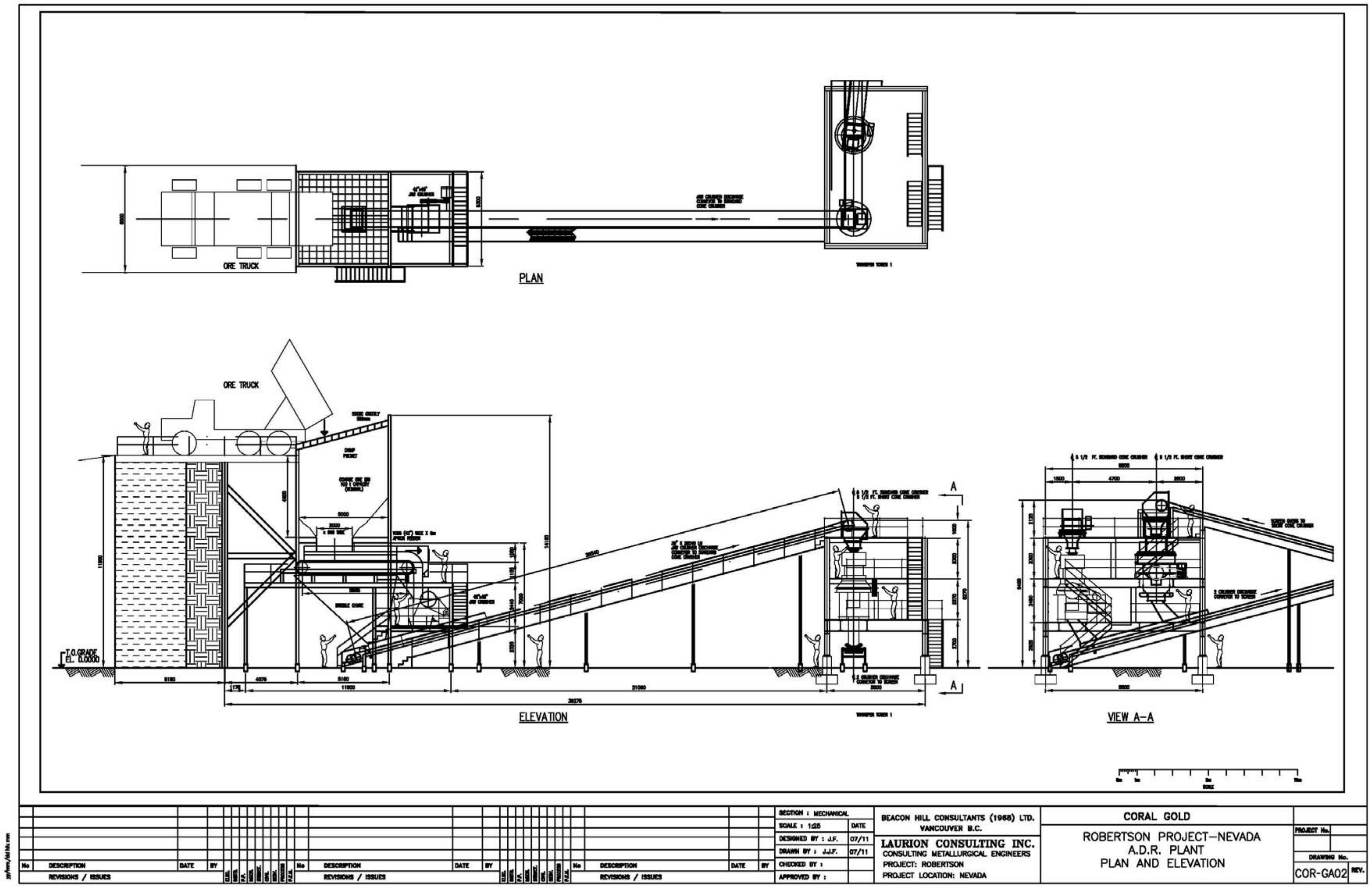


Figure 17-3: ARD Plan and Elevation

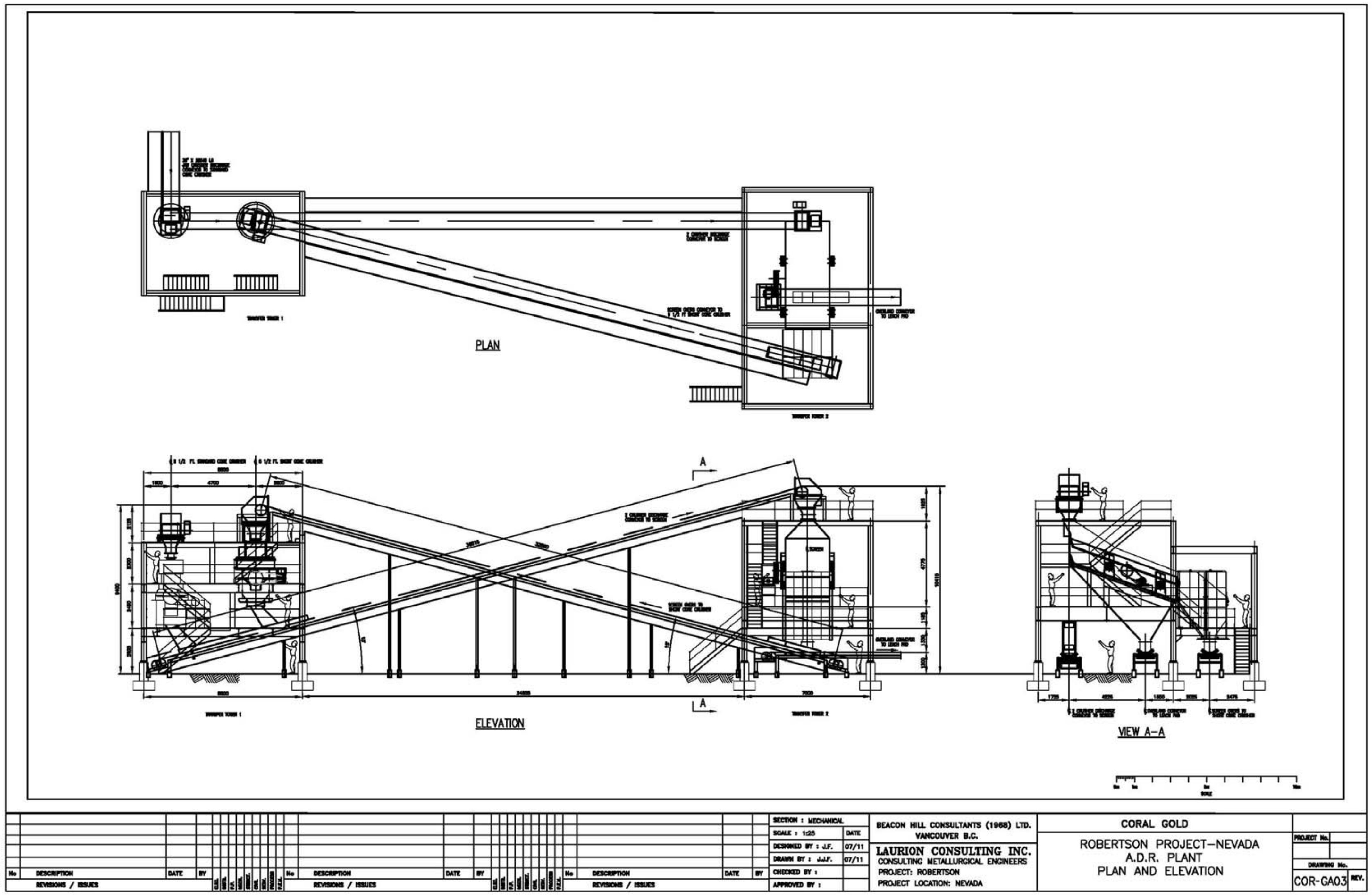


Figure 17-4: ARD Plan and Elevation

SECTION 18.0 PROJECT INFRASTRUCTURE

18.1 GENERAL

18.1.1 Introduction

In order to provide leach pads for the different grades of mineralization two separate heap leach pads will be constructed, one for the low-grade ore and another for high-grade ore respectively. Low-grade run-of-mine ore will be stacked on the low-grade heap while high-grade ore will pass through a crushing circuit prior to being stacked on the high-grade heap.

The heap leach facilities will be constructed on a gently sloping hillside that will drain freely under the effect of gravity. The heap leach pads will be dosed with weak cyanide solution to dissolve gold contained within the ore. Lined collection channels placed along the downstream toe of the leach pads will drain gold-bearing solution to the pregnant solution pond. The cyanide solution that has passed through the heaps is referred to as “pregnant” solution. Pregnant solution is then pumped to an Adsorption-Desorption-Recovery (ADR) plant. The ADR will strip gold from the solution, rendering the solution “barren”. The barren solution is contained in a second pond (termed the barren solution pond) where it will be re-circulated to the heap leach pads.

The retained gold will be refined and cast into gold doré bars using facilities located on site.

This report addresses the general civil and geotechnical design conditions that affect the development of major mine site earthworks (open pit development, foundation design), heap leach pad design, waste management, and site wide surface water management.

18.1.2 Site conditions

18.1.2.1 Climate

Limited climate data is available for the Robertson Property project; however, the project area can be described as cold, semi-arid (Köppen climate classification: type Bsk). Regions of this type often exhibit hot and dry summers and very cold winters. Snowfall often occurs in winter months, albeit at much lower quantities than more humid areas at similar latitudes.

The site exhibits a climate that is typical of Northern Nevada high desert. The area is characterized by a short warm to hot summer season, relatively long and cold winters and moderate springs and falls.

Mean monthly temperatures (Fahrenheit) are estimated in the low to mid-20s during December through February; low to mid-30s in November and March, 50s in May and September and low to upper-60s from June through August. Summer time highs can reach nearly 90 degrees and winter lows can fall below zero. The following table summarizes the monthly climate data that is available for the region.

Table 18.1: Monthly Climate Data[1]

Month	Temperature		Precipitation	Evaporation[2]	Snowfall
	Min. Average [ºF]	Max. Average [ºF]	[in]	[in]	[in]
January	14.7	39.9	0.84	n/a	5.5
February	20.5	25.3	0.67	n/a	2.6
March	25.3	53.6	0.76	n/a	1.7
April	29.8	62.5	0.86	n/a	0.7
May	37.3	72.4	1.17	8.55	0.1
June	44.0	82.4	0.84	9.95	0
July	50.3	91.9	0.29	12.8	0
August	47.5	89.9	0.44	11.3	0
September	38.6	80.6	0.51	8.12	0
October	28.7	67.5	0.59	4.90	0.2
November	21.6	51.0	0.83	n/a	1.1
December	15.7	40.8	083	n/a	3.4
Annual	31.2	64.8	8.62	n/a	15.2

[1] Monthly climate data from Western Regional Climate Center, Beowawe Station –
[2] Evaporation data from Rye Patch Dam Station (7192) from NOAA-EDIS.

18.1.3 Seismicity

A preliminary review of the regional seismicity has been carried out to enable selection of an appropriate design earthquake event for seismic stability assessment of the heap leach facility.

According to maps generated by the US Geological Survey (Figure 18-1), this area has a peak horizontal ground acceleration ranging between 0.10g to 0.15g corresponding to the 475-year event, defined as an event which has a 10% probability of exceedance in 50 years.

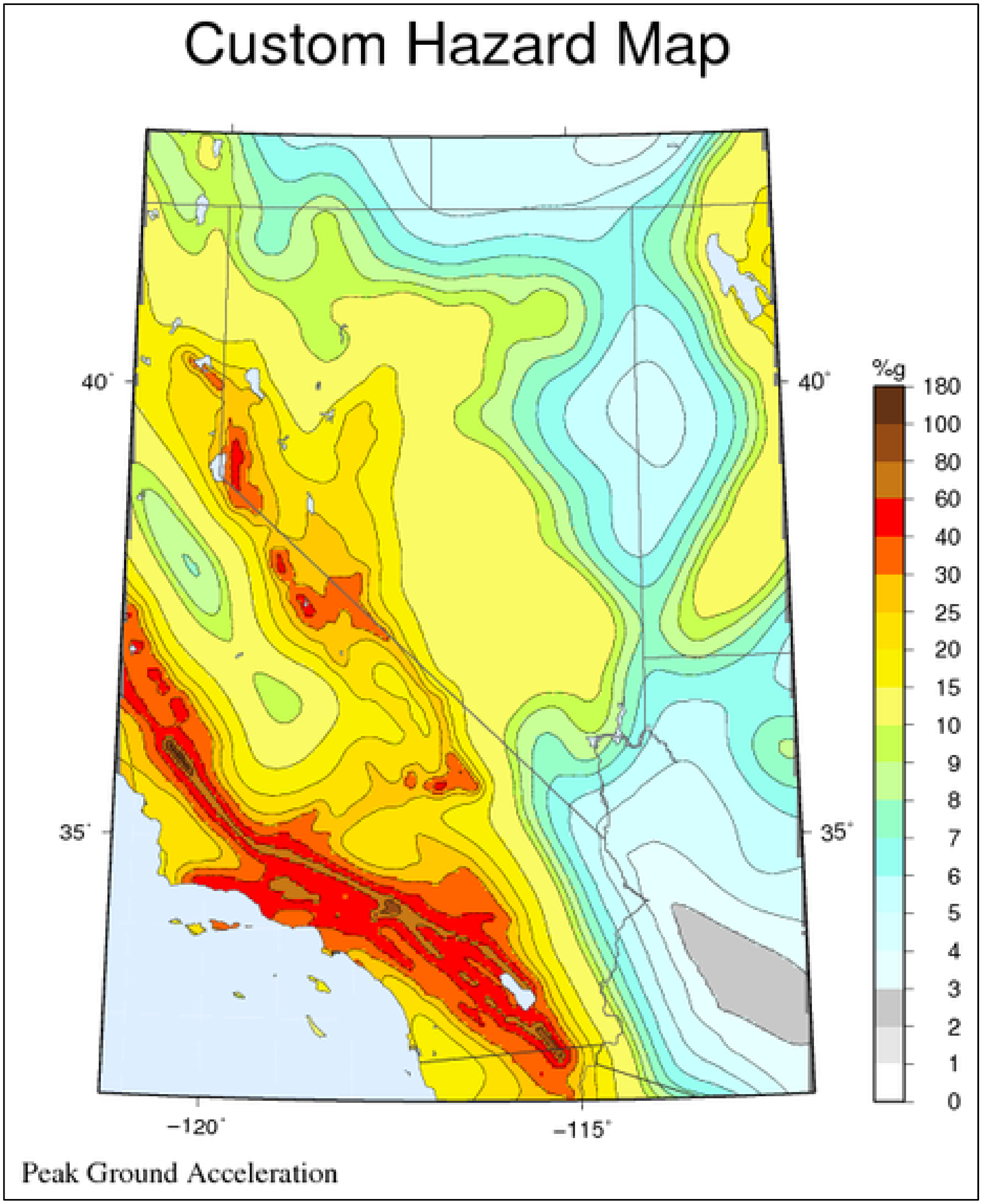


Figure 18-1 : USGS Seismic Hazard Map – Peak Ground Acceleration (1:475)

18.1.4 Mine Development

Proposed Layout of Mining Facilities

The preliminary general arrangement is shown on Figure 1-3. The general arrangement shows the full build out condition of the proposed mine development including the following components:

- Site access from Nevada State Route 306.
- On site light-duty access roads.
- On site haul routes.
- Surface water drainage diversions.
- Storm water pond.
- Ancillary facilities, including:
- Administrative offices
    - Warehouse
    - Dry facilities
    - Maintenance shop
    - Fuel depot
    - Explosives storage, and
    - Gatehouses.
- Proposed open pits, including:
    - Porphyry Pit
    - Altenburg Hill Pit, and
    - Gold Pan Pit.
    - Crushing facilities and high-grade ore conveyor.
- Fresh water ponds.
- Heap leach circuit infrastructure, including:
    - High-grade ore heap leach facility
    - Low-grade ore heap leach facility
    - Solution transfer channels
    - Pregnant solution pond
    - ADR process plant
    - Barren solution pond, and
    - Event (storm water) pond.

Design criteria for the open pit, heap leach facilities, waste rock storage area, water management facilities, and foundations of critical structures are discussed in the following sections.

18.2 OPEN PIT DESIGN CRITERIA

18.2.1 Information Review

A slope stability study to determine pit slope angles at the Porphyry Open Pit was conducted in 1994 for Amax Gold Inc. (Watters R., 1994). The study was based on information derived from diamond drill core and core logs. Four simplified geological domains were delineated, including diorite, skarn/endoskarn, Calcite-silicate hornfels and biotite-hornfels.

The 1994 geotechnical study indicated that the East Wall had unfavourable discontinuity orientations where 'discontinuities may dip towards the slope face' based on limited outcrop stereographic analyses. The adverse structural features were found to be less significant for the rest of the pit walls.

Rock mass characterization was conducted during the 1994 geotechnical study using the Rock Mass Rating (RMR, Bieniawski, 1976) classification system. Rock mass within the East Wall was identified to be weaker based on unfavourable discontinuity orientations and resulted in the lowering of the calculated RMR. The average values of the equivalent rock mass shear strength parameters for the 1994 Porphyry Pit are below:

East Walls – Friction Angle = 23 degrees, Cohesion = 5100 psf, and
West, North and South Walls – Friction Angle = 29 degrees, Cohesion = 7400 psf.

The conclusions of this study identified six sectors of the preliminary pit walls for overall slopes ranging from 50 to 56 degrees for the West, North and South Sectors and 45 degrees for the East Sector.

18.2.2 Slope Stability Analysis

The Porphyry Pit has been expanded and two additional smaller pits, Altenburg Hill and Gold Pan, have been included in the current PEA study. The maximum slope heights for the expanded Porphyry Pit, Altenburg Hill Pit, and Gold Pan Pit will be in the order of 700 ft, 500 ft and 300 ft, respectively (Garth Kirkham, January 2011). The geological information for the new pits is very limited.

A series of limit equilibrium stability sensitivity analyses were carried out to determine appropriate overall slope angles for the proposed open pits. The targeted Factor of Safety (FOS) for the overall slopes was 1.3 and the following assumptions were made in the analyses:

- Average rock mass strength parameters determined from the 1994 study were applied in the analyses.
- It is assumed that the rock mass structural features and rock mass qualities are similar in these proposed pits (i.e., weaker rock mass presents in the East Sector).
- Groundwater table was considered to be below the pit bottom, as the existing drillhole data suggests that the regional groundwater table is below the extent of all open pits.
- The sensitivity analyses cover the following variations:
- Overall slope angles: 40, 45, 50 and 55 degrees
- Total slope heights: 300, 400, 500, 600 and 700 ft, and
- Rock mass strength parameters: Lower bound (1994 Porphyry Pit East Wall data) and upper bound (1994 Porphyry Pit West, North and South Wall average data).

The results of the analyses are illustrated as an FOS vs. slope height chart in Figure 18-2. The currently proposed maximum slope heights, preliminary overall slope angles, and resulted FOS for each pit design sector are highlighted as follows:

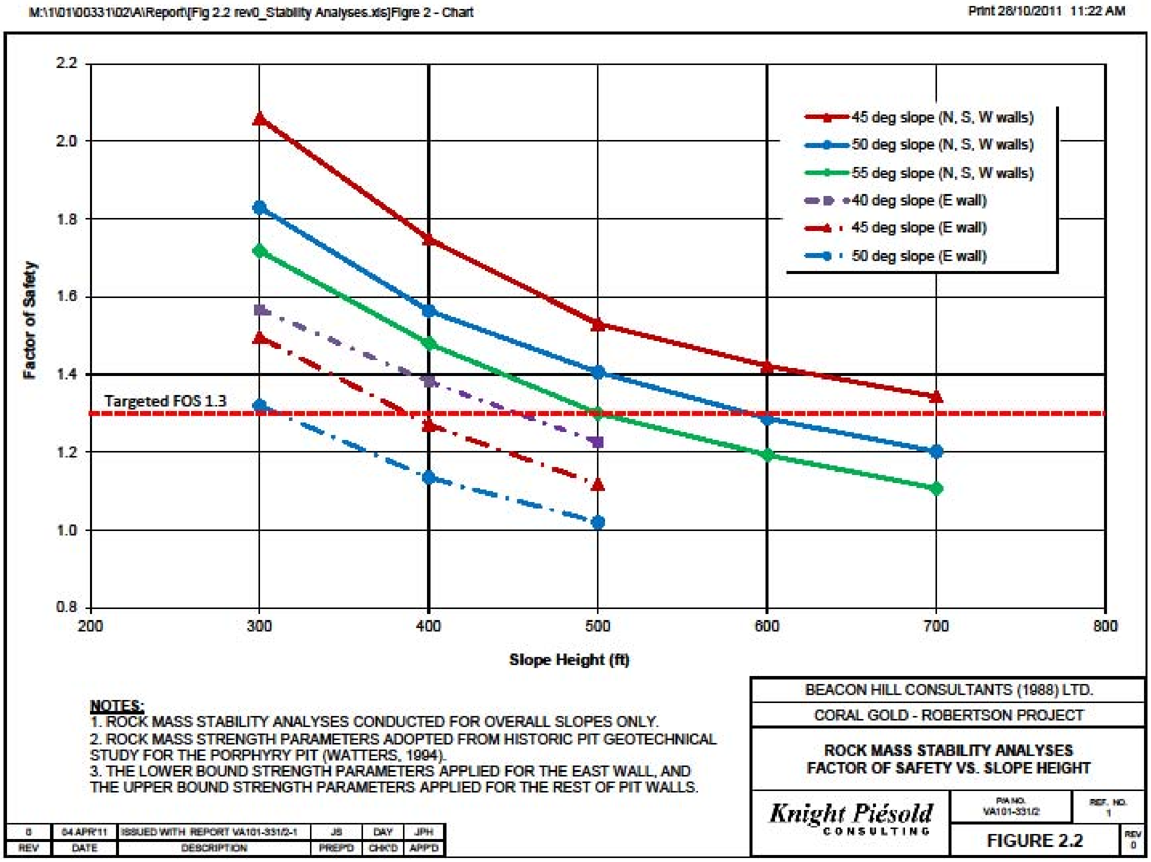


Figure 18-2: Rock Mass Stability Analysis Factor of Safety VS Slope Height

Porphyry Open Pit

West Wall: Total slope height = 625 ft, 50º overall slope angle to achieve a FOS of 1.3.
North Wall: Total slope height = 700 ft, 45º overall slope angle to achieve a FOS of 1.3.
South Wall: Total slope height = 400 ft, 55º overall slope angle to achieve a FOS of 1.5.
East Wall: Total slope height = 500 ft, 40º overall slope angle to achieve a FOS of 1.2.

Altenburg Hill Open Pit

West Wall: Total slope height = 525 ft, 55º overall slope angle to achieve a FOS of 1.3.
North Wall: Total slope height = 325 ft, 55º overall slope angle to achieve a FOS of 1.7.
South Wall: Total slope height = 400 ft, 55º overall slope angle to achieve a FOS of 1.4.
East Wall: Total slope height = 325 ft, 50º overall slope angle to achieve a FOS of 1.3.

Gold Pan Open Pit

West Wall: Total slope height = 325 ft, 55º overall slope angle to achieve a FOS of 1.7.
North Wall: Total slope height = 275 ft, 55º overall slope angle to achieve a FOS of 1.7.
South Wall: Total slope height = 275 ft, 55º overall slope angle to achieve a FOS of 1.7.
East Wall: Total slope height = 225 ft, 50º overall slope angle to achieve a FOS of 1.3.

It should be noted that these analyses considered the overall rock mass stability only. Rock mass structures were not taken into account. Coarse assumptions of geotechnical parameters (rock mass including were made to cover the variation of site uncertainty. Further geotechnical site investigations should be carried out to verify these assumptions.

18.2.3 Summary

Based on limited geotechnical data and the corresponding stability analysis results, the recommended overall pit slope angles for the PEA study are summarized below:

Porphyry Pit: 40º (East Wall), and 50º for the rest of pit walls
Altenburg Hill Pit: 50º, and
Gold Pan Pit: 50º.

It should be noted that these preliminary recommendations were based on limited information and a geotechnical site investigation program should be performed to collect rock mass structural and quality data for detailed slope stability assessment. It is reported that Barrick operations at Cortez, located a short distance to the Robertson Project, has achieved overall slope angles varying from 45 to 50 degrees. There may be potential to optimize slope angles when additional geomechanical data becomes available.

18.3 LEACH PAD DESIGN CRITERIA

18.3.1 Summary

The proposed heap leach facilities will include fully contained pads capable of treating gold ore. Preliminary plans involve the extraction of gold by heap leaching. The heap leaching process will involve the irrigation of weak cyanide solution over successive lifts of heaped ore. A total of 78 million US short tons (Mt) of ore is assumed for the design with consideration for potential expansion. Of this ore, it is assumed that 26 Mt will be routed to the high-grade heap leach pad via a mechanical crushing circuit and 52 Mt will be routed to the low-grade heap leach pad as run-of-mine ore. Geotechnical site investigations have not been carried out. Site investigations, design studies and ore characterization will be required to confirm preliminary design assumptions and the feasibility of the project as the project advances.

The site selection, general site conditions, geotechnical implications and design requirements for the heap leach pads are discussed in the following sections.

18.3.2 Site Selection

The selection of the locations for the heap leach pads was based on the following criteria:

- Proximity to the proposed open pits
- Minimizing the overall project footprint
- Storage capacity requirements
- Potential for expansion
- Satisfying stability requirements, and
- Mitigating environmental impact.

While other locations were considered, the preferred site for the heap leach facilities is located between the eastern limits of the proposed Porphyry and Altenburg Hill open pits and Nevada State Route 306. This site can accommodate both the low-grade and high-grade heap leach facilities and their associated infrastructure. The gently sloping terrain (1-2%) is conducive to heap leach operations, allowing pregnant solution to drain via gravity to solution transfer channels located along the downstream toes of each of the heap leach pads.

The proposed heap leach pads will be sited on relatively-uniform gently sloping terrain directly to the west of Nevada State Route 306. The east-facing slope runs down gradient from the eastern limits of the Porphyry and Altenburg Hill open pits at approximately 1%. The site was chosen for its proximity to the proposed open pits and the fact that the area has been previously disturbed by historic placer mining operations. Vegetation is comprised primarily of hearty shrubs and grasses with no apparent tree cover.

A number of surface expressions exist along the slope. It is likely that the channels were formed by both natural (erosive effects of surface runoff) and anthropogenic means (placer mining activities). The channels are ephemeral and run dry for most of the year. It is anticipated that the channels may be used to further facilitate gravity drainage of the heap leach pads.

18.3.3 Design Objectives

The primary design objectives for the proposed heap leach pads are as follows:

- Provide a stable and cost effective configuration for staged heap development
- Effectively collect and convey leachate solutions to the pregnant solution pond while ensuring maximum recovery
- Provide secure containment of events solutions while monitoring and minimizing losses due to leakage
- Minimize surface runoff entering the leach pad area while accounting for the collection of direct runoff from the heap area
- Sequential, staged development and leaching operations, and
- Effective decommissioning and reclamation of all heap leach facility components.

The following sections discuss the storage requirements and preliminary design assumptions and parameters for the proposed leach pads. Design features for each component are discussed below.

18.3.4 Design Basis

The following sections outline the criteria for the engineering design of the Robertson Property heap leach facilities, associated ponds and drainage controls.

18.3.5 Foundation Preparation

Foundation preparation includes the removal of vegetation, existing structures and unsuitable materials, and site grading as follows:

- Vegetation: clear and grub any vegetation.

- Structures: remove any existing structures or infrastructure that interferes with the construction or operation of the proposed facilities.
- Surface soils: strip organic soil cover to a maximum of 10 feet beyond the leach pad limits; organic soil should be placed in a temporary topsoil stockpile for use in subsequent reclamation; the exposed ground surface should be scarified, conditioned and re-compacted in at-grade and fill areas.
- Site grading: site grading cut and fill according to design; fill to be compacted as directed.

18.3.6 Leach Pads

The two leach pads will be constructed to process run-of-mine (low grade) and crushed (high grade) ore. Components of the pads are described below:

- Arrangement: two single-cell pads (high grade and low grade).
- Grade: native hillside slopes not to exceed 2.5H: 1V in lined areas; nominally 2.5% in the pad base near the toe; minimum of 1% along pregnant pipeline route from pad to pond.
- Perimeter berm: bench graded along perimeter for liner installation; heap to be kept within pad perimeter and 2 feet minimum below perimeter elevation.

18.3.7 Pad Liner Systems

The leach pad liner systems provide an impervious boundary to contain leach solution and to prevent solution loss. The components of the pad liner systems are listed below:

- Underliner: geosynthetic clay liner (GCL) deployed on prepared surface; final re-graded subsurface or fill surface to be compacted with a smooth drum roller to form a smooth, firm and unyielding surface with no protruding particles exceeding 1 inch; GCL installed permeability at 1 x 10^{-6} cm/sec or less.
- Geomembrane: 80 mil textured HDPE.
- Geomembrane anchor trench: 3 feet minimum depth.

18.3.8 Overliner and Solution Drainage System

The leach pad overliner protects the geomembrane from exposure to the elements and from vehicle traffic during ore loading. The free-draining overliner, supplemented by drain pipes, also reduces the hydraulic head on the geomembrane and speeds solution recovery. Components of the overliner system are listed below:

Overliner: 36 inch minimum thickness of minus 2-inch nominal well-graded, free-draining granular material with less than 10% by weight particles passing the No. 200 ASTM sieve size; placed in a single lift with no compaction; permeability of 1 x 10^{-3} cm/sec or higher.

Solution drain pipes: 4 inch diameter perforated (Type SP) corrugated polyethylene tubing (CPT) lateral pipes on 30-foot centers to collector pipes and to solution collection channel; perforated CPT collector pipes of varying diameters placed across the pad as designed; all CPT lateral and collector pipes to be ADS N-12 dual wall smooth interior pipes, or approved equivalent.

Drain pipe capacity: solution flow at not more than 80% of pipe-full capacity for spare capacity to handle storm and other upset conditions.

18.3.9 Pregnant Solution Transfer

The pregnant solution transfer allows leach solution and storm runoff to pass from the leach pad to the pregnant solution pond. The pregnant solution transfer system components are listed below:

- Location: the solution transfer channels will run along the downstream toe of each heap leach pad.
- Depth: at grade.
- Bottom Width: 6 feet.
- Slopes: 2H:1V.
- Grade: maximum of 2.5%.
- Liner: transfer channel connected to leach pad geomembrane liner.
- Capacity: Application solution flow rate at less than 80% bank-full capacity to accommodate surges due to storm and other upset conditions.

18.3.10 Pregnant Solution Pond

Solution and storm runoff flows conveyed by the pregnant solution transfer channel drain into the pregnant solution pond. The pond components consist of the following:

- Depth: 12 ft.
- Crest width: 25 ft.
- Slopes: 2.5H:1V liner interior; 2H:1V unlined exterior.
- Bottom grade: graded to drain towards the south corner to facilitate transfer to the ADR facility.
- Liner: 60 mil smooth HDPE bottem (secondary) geomembrane; 60 mil smooth HDPE top (primary) geomembrane (surface may be textured for traction); leak detection system consisting of geonet between geomembranes on pond slopes and bottom to corner leak detection sump and riser/monitor system; 3 foot minimum depth for geomembrane anchor trench.
- Capacity: total: 370,800 ft^3; operationally: 286,500 ft^3 with upset capacity storage to contain 24-hour process solution draindown plus 25-year, 24-hour storm from leach pad catchment area plus freeboard.
- Freeboard: 2 feet minimum.

18.3.11 Barren Solution Pond

Barren solution is delivered to the barren solution pond from the ADR facility. The pond components consist of the following;

- Depth: 12 ft.
- Crest width: 25 ft.
- Slopes: 2.5H:1V liner interior; 2H:1V unlined exterior.

- Bottom grade: graded to drain towards the southeast corner to facilitate transfer to the ADR facility.
- Liner: 60 mil smooth HDPE bottom (secondary) geomembrane; 60 mil smooth HDPE top (primary) geomembrane (surface may be textured for traction); leak detection system consisting of geonet between geomembranes on pond slopes and bottom to corner leak detection sump and riser/monitor system; 3 foot minimum depth for geomembrane anchor trench.
- Capacity: total: 370,800 ft^3; operationally: 286,500 ft^3 with upset capacity storage to contain 24-hour process solution draindown plus 25-year, 24-hour storm from leach pad catchment area plus freeboard.
- Freeboard: 2 feet minimum.

18.3.12 Solution Application

Solution application involves the uniform application of barren solutions, reagents, low grade pregnant solution recycle and mark-up water to the heap surfaces for controlled infiltration and leaching of the ore. The solution application parameters are listed below:

- Flow rate: provided by others.
- Application rate: 56,500 ft^3/hr (provided by others).
- Application method: emitters and/or wobblers.
- Leach cycle: variable.
- Active leach surface: 365 acres (low-grade: 188 acres / high-grade: 177 acres) at maximum application rate.

18.3.13 Event Pond

The event pond provides addition (dry) capacity to prevent excess storm water runoff from overwhelming the pregnant solution pond. An emergency spillway, sized for the 1 in 25 year event, will allow overflow from the pregnant solution pond to drain directly into the event pond. As the overflow solution will likely contain cyanide, the event pond will require lined construction and leak detection similar to the primary solution ponds. The event pond design criteria are listed below:

- Depth: 22 ft.
- Crest width: 25 ft.
- Slopes: 2.5H:1V liner interior; 2H:1V unlined exterior.
- Bottom grade: graded to drain towards the south corner to facilitate transfer to the ADR facility.
- Liner: 60 mil smooth HDPE bottom (secondary) geomembrane; 60 mil smooth HDPE top (primary) geomembrane (surface may be textured for traction); leak detection system consisting of geonet between geomembranes on pond slopes and bottom to corner leak detection sump and riser/monitor system; 3 foot minimum depth for geomembrane anchor trench.
- Capacity: Operationally, zero storage. Upset capacity storage to contain 24-hour process solution draindown plus 25-year, 24 hour storm from leach pad area catchments. 7,050 gpm x 24 hrs = 10,137,600 gallons draindown storage plus 14,219,000 gallons storm runoff for a total of 24,356,600 gallons plus freeboard.

- Freeboard: 2 feet minimum.

18.3.14 Diversion System

Diversion ditches will be constructed up gradient of the heap leach pads to divert storm runoff from uphill catchments into natural drainage courses. Ditches concurrently serve as pad liner installation benches. The diversion system components are listed below:

- Design: trapezoidal channel.
- Depth: varies.
- Bottom grade: 1% minimum.
- Side slopes: varies.
- Freeboard: 1 foot minimum.
- Armour: pad liner, riprap or grouted riprap as needed.
- Capacity: peak flows from the 100-year, 24 hour storm from upstream catchments.

18.3.15 Leach Pad Design

18.3.15.1 General

The two leach pads will be constructed on the east-facing slope of the foothills located to the west of Crescent Valley. The leach pads will extend over an area of approximately 365 acres, filling much of the eastern extent of the property (as shown on Figure 1-3). This area will be re-graded, lined and configured to drain to the pregnant solution pond located downslope to the east of the two pads. This section contains additional description of the leach pad features and their design.

18.3.15.2 Grading Plan

The grading plan involves development of a lower floor area for the pads to facilitate heap stability and controlled solution drainage from the pads to the pregnant solution pond. The majority of the pad area is presently situated over mildly sloping terrain with occasional undulations. The natural slopes tend to lie within the range of 1%-2% so significant regrading will not be required; however, local undulations may need to be smoothed out.

Perimeter roads or benches may be required to facilitate the construction of the pads. These features may also serve as external storm water diversion. Drill and blast work is not anticipated in the heap leach pad areas. Benches will be retained operationally for pad perimeter access and functionally for storm water diversion.

The pad floors will be backfilled and built up to form surfaces upon which the pad drainage systems will be constructed. The built up floor area occupies the flatter terrain near the toe of the pads. The floor area will be graded to a consistent 2.5% gradient, draining the toe limit of the pads.

18.3.15.3 Lining System and Overliner

The lining system is designed to prevent seepage from within the leach pads from escaping into the underlying subsurface and groundwater environment. The liner system will be comprised of a composite liner and an overliner containing a leachate collection pipe network.

The composite liner system consists of a geosynthetic clay liner (GCL) and an 80 mil textured HDPE geomembrane. The GCL will be placed directly on the prepared subgrade using conventional installation, overlapping and sealing techniques. This material is designed to have a hydraulic conductivity of 10^{-6} cm/s or lower. The 80 mil textured HDPE geomembranes will be placed directly on the GCL using conventional deployment, installation, and seaming techniques.

A protective layer of coarse crushed ore will be required over the liner system. This layer will protect the geomembrane from damage during ore placement and will act as a drainage layer to maximize solution recovery while reducing hydraulic head on the liner. The overliner will be a 3 feet (approx.) thick lift over the full extent of each pad. This is required to cover the main solution collection pipes. For an 80 mil (2 mm) HDPE geomembrane, maximum particle size for overliner material should be minus 1.5 inch with low fines content. The solution collection pipe network system will consist of 4 inch HDPE slotted lateral pipes that will drain by gravity into 12 inch HDPE slotted trunk lines or secondary headers. These secondary headers will flow by gravity into 20 inch HDPE slotted primary headers which will drain to the solution transfer channels located at the toe of each heap. The spacing of the pipes should be designed to limit the potential head expected on the liner system.

Sections and details of the proposed liner and overliner system are shown on Figures 18-3 and 18-4.

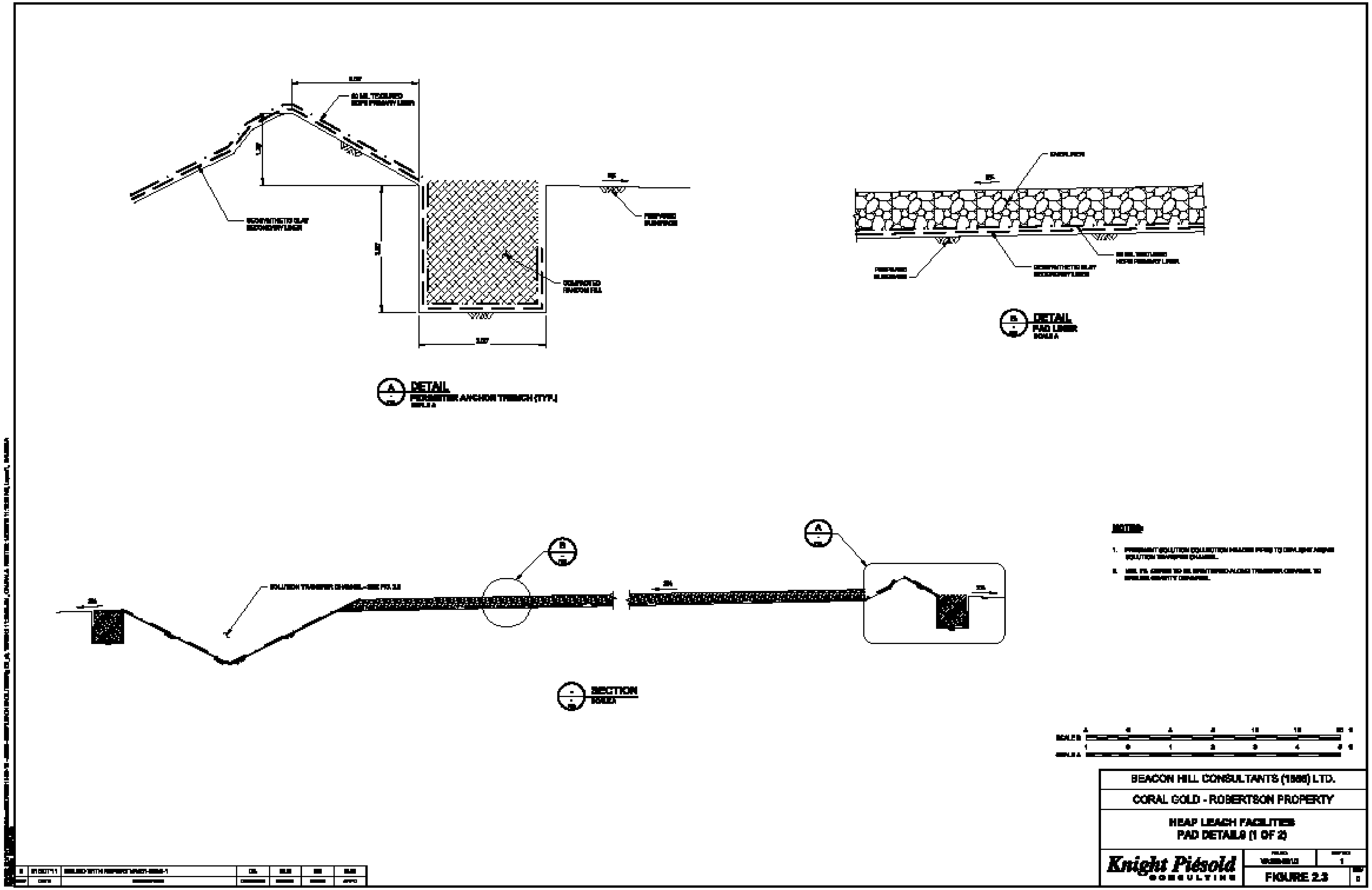


Figure 18-3: Heap Leach Facilities Pad Details (1 of 2)

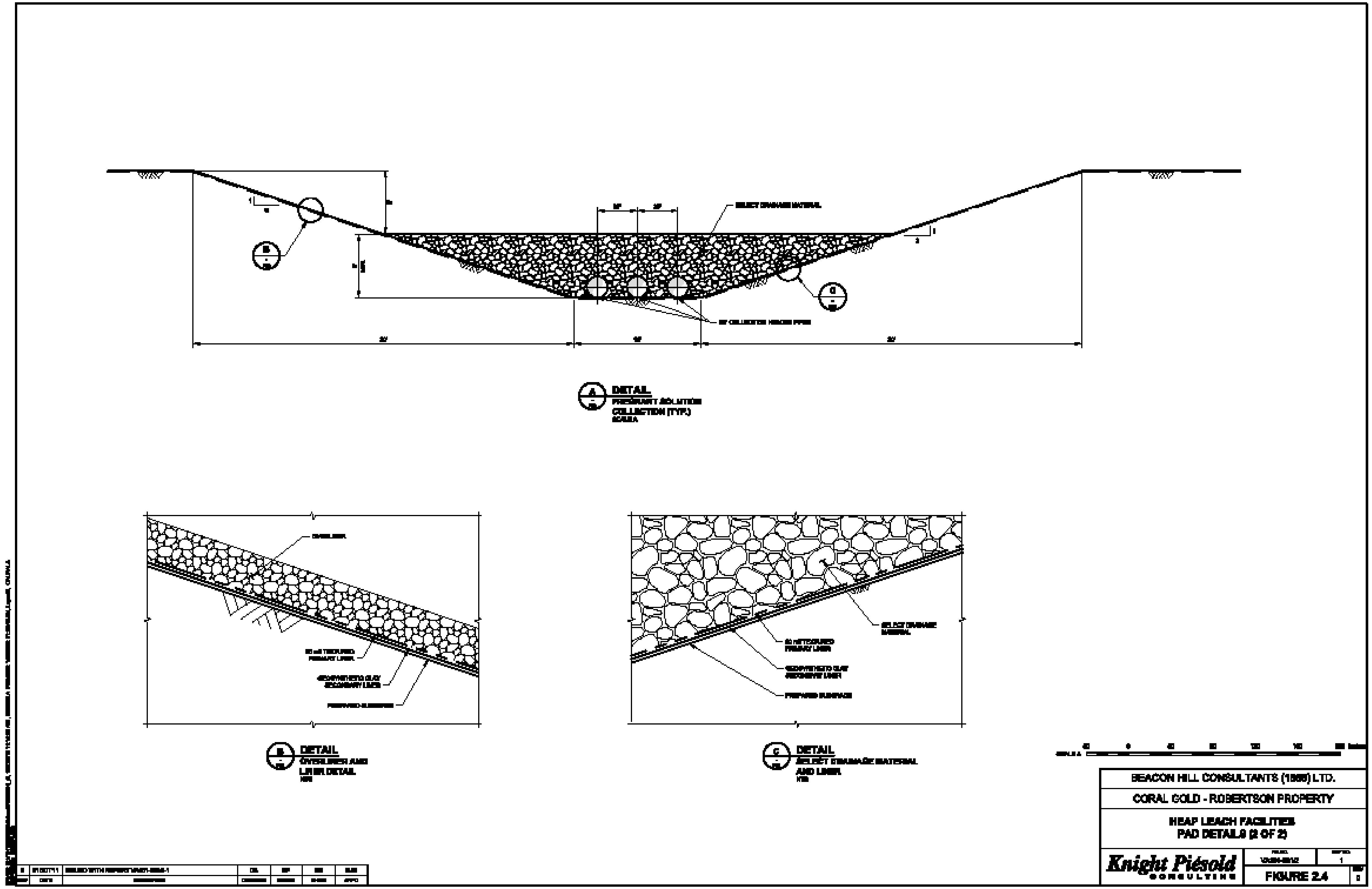


Figure 18-4: Heap Leach Facilities Pad Details (2 of 2)

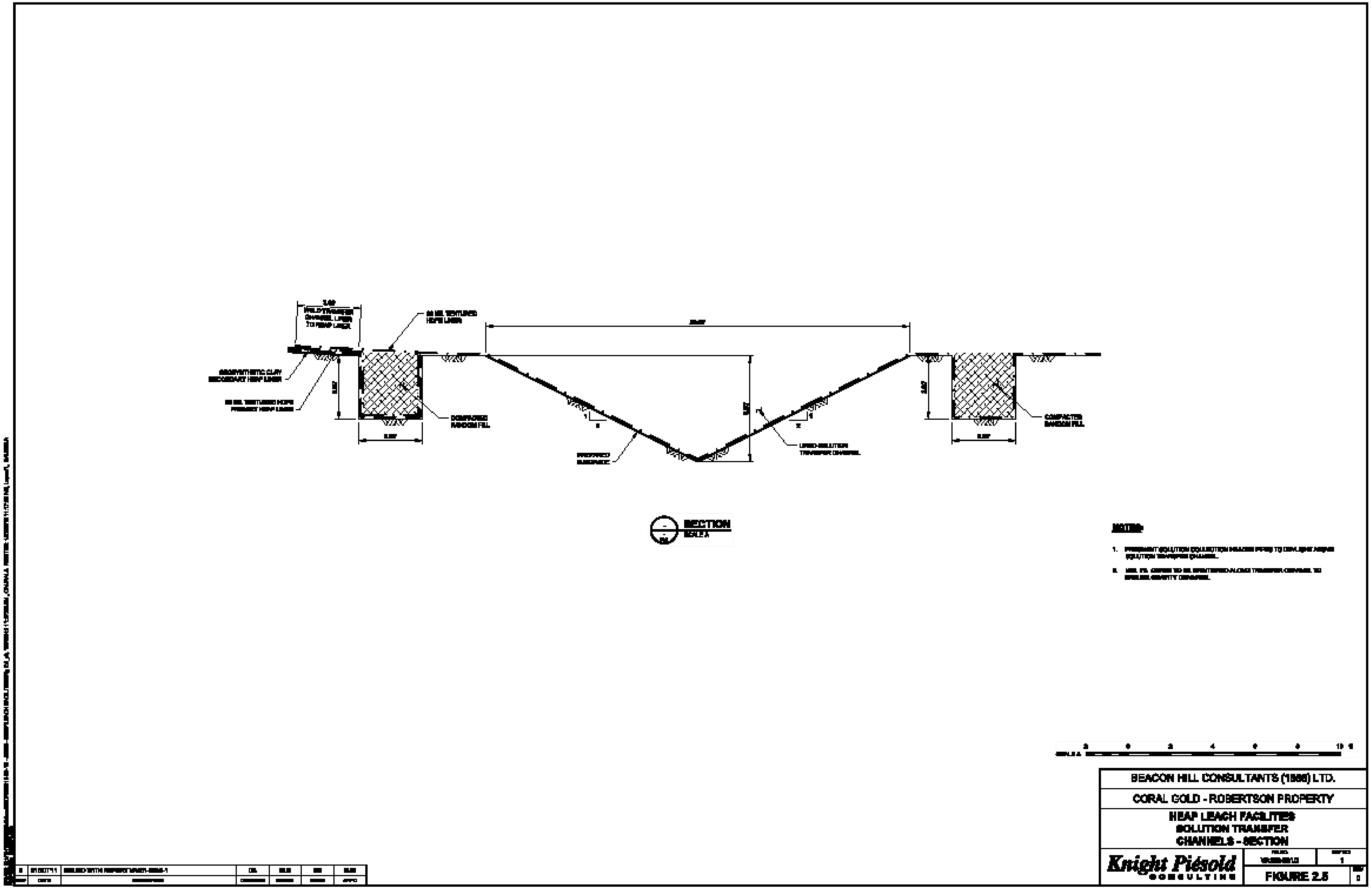


Figure 18-5: Heap Leach Facilities Solution Transfer Channels - Section

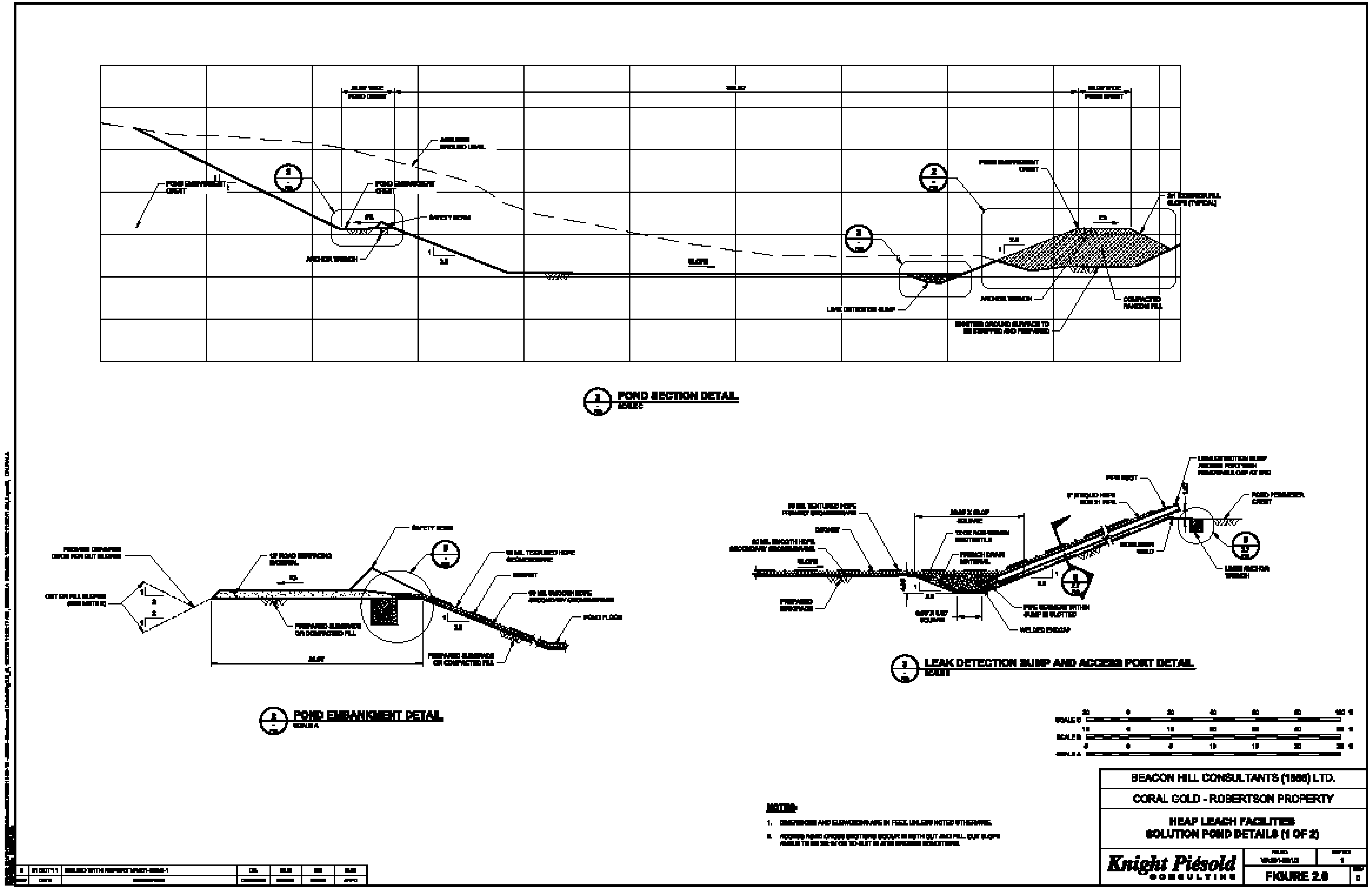


Figure 18-6: Heap Leach Facilities Solution Pond Details (1 of 2)

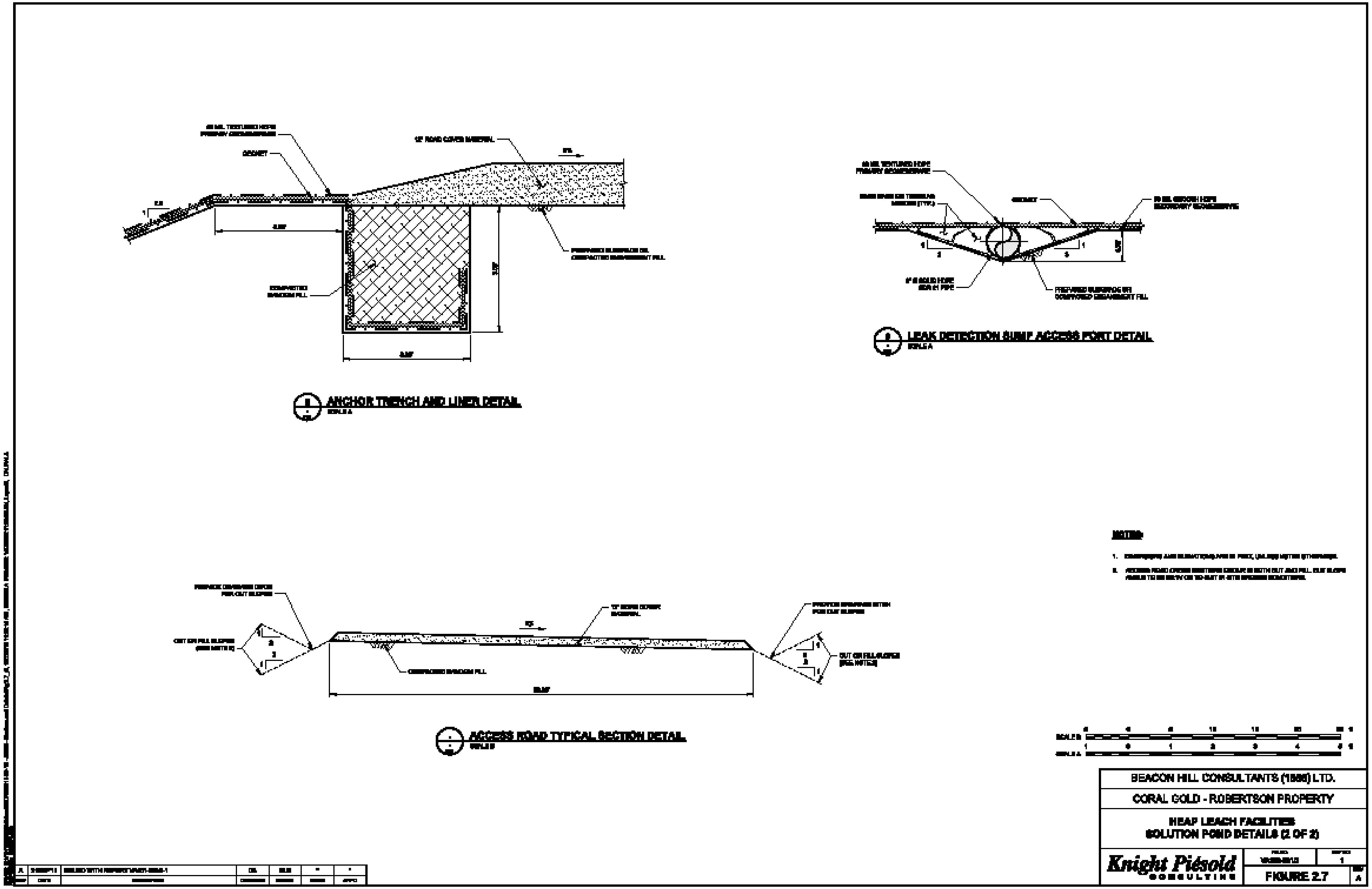


Figure 18-7: Heap Leach Facilities Solution Pond Details (2 of 2)

18.3.15.4 Solution Transfer Channels

The downstream toe of each heap leach pad will include a lined pregnant solution transfer channel. The channels will be lined with 80 mil textured HDPE geomembranes. These liners will be integrated into the pad liner systems using conventional installation, overlapping and sealing techniques. The downstream end of the liner will be anchored into soil to prevent detachment.

The solution transfer channels will drain via gravity to the pregnant solution pond. A minimum gradient of 1% should be maintained to ensure gravity drainage.

Sections and details of the proposed solution transfer channels are shown on Figure 18-5.

18.3.15.5 Solution Ponds

Two separate solution ponds are required for the operation of the heap leach facilities. The pregnant solution pond will receive gold-bearing solution from the two leach pads via the solution transfer channels. Solution from the pregnant solution pond will be pumped directly to the adsorption-desorption-recovery plant for gold extraction. The barren solution would then be pumped to the barren solution pond.

Each pond will be designed to contain the same volume of solution. The design criteria are consistent with best practices to provide dual-liner protection to semi-permanent cyanide solution storage areas.

Figures 18-6 and 18-7 show common details shared between the two ponds.

18.4 FRESHWATER SUPPLY

It is anticipated that fresh water will be procured from Barrick's Cortez Operation located immediately to the south of the Robertson Property. Barrick operates a collection of ponds that are located directly across State Route 306 from the proposed plant facilities.

This water should provide a suitable source and quantity to sustain heap leach operations; however, this component of the project is being assessed by others.

18.5 WASTE ROCK STORAGE AREA

18.5.1 Summary

A preliminary location and layout has been developed for the storage of non-reactive mine waste rock. The waste dump is to be located in the centre of the mine site, approximately equidistant from each of the three open pits.

A total of 50 Mt of mine waste rock will be mined from the three open pits over a 13 year period. Preliminary waste characterization indicates that it is unlikely that the waste rock is potentially reactive.

18.5.2 Geotechnical Conditions

The Waste Rock Storage Area is situated on a north-facing slope south of the deposit area. Vegetation is comprised primarily of scrub brush and grasses.

Geotechnical conditions at the proposed Waste Rock Storage Area have not been assessed.

18.5.3 Waste Rock Storage Area struction and Operation

The following are recommended methods of construction and operation to ensure on-going stability and performance of the Waste Storage Area. These methods may be updated and revised, as necessary, based on field observations and performance monitoring during the initial stages of waste pile construction.

Pre-Production:

- Strip the foundations of the Waste Rock Storage Area prior to loading to remove the topsoil and other deleterious foundation materials
- Clear vegetated areas prior to placement of waste materials, and
- Construct diversion ditches and runoff collection ditches where required.

Operations:

- Waste/ore material will be transported from the pit using haul trucks. The material will be end dumped and spread by dozers over the crest of the pile.
- Load the waste/ore pile in lifts starting at the toe and working progressively upslope. Each lift may be developed at the angle of repose for the material, however, benches should be left along the toe of each successive lift to establish a maximum overall slope angle of 2H:1V.
- The lifts should be loaded and developed in a direction parallel to the slope wherever possible to avoid stress concentrations associated with “nose” configurations where the axis of the pile extends outward and perpendicular to the slope.
- Trial sections shall be constructed in the field during the initial stages of development to monitor pile stability and foundation performance. The various waste/ore materials should be sampled for characterization and for durability test work to confirm the design parameters.
- Waste rock material shall be end dumped over the crest to allow for maximum segregation of the coarser material at the base of each bench.
- The construction of sediment control structures consisting of a series of small embankments will likely be required downstream of the waste dump to act as sediment entrapment ponds for runoff from stripped foundation areas and dump surfaces.

Additional site investigations will confirm if excavating down to competent bedrock is required. Bedrock is anticipated at shallow depths throughout the Waste Rock Storage Area with an assumed average depth of approximately 10 feet, and will provide a suitable foundation. In areas where the depth to competent bedrock is deeper it may be preferable to stage the waste pile configuration to provide suitable stability by buttressing the toe of the waste pile.

18.5.4 Stability Assessment

Stability analyses have not been carried to examine the stability of the proposed waste rock pile. The stability analyses should be carried out using the limit equilibrium modelling application such as SLOPE/W. The program utilizes a systematic search to obtain the minimum factor of safety from a number of potential slip surfaces.

The minimum acceptable factor of safety for the waste rock pile under static conditions is 1.3 for short-term operating conditions and 1.5 for long-term conditions (after closure). The consequences of failure of the waste rock pile during an earthquake event are likely to be minimal and restricted to some displacement of the waste pile slopes. There would be a negligible impact on the integrity of the waste pile and little, if any, impact on other Mine site facilities. However, for design of the waste pile a conservative design earthquake corresponding to the 1 in 475 year earthquake event has been adopted with a corresponding mean average maximum acceleration on rock of 0.10-0.15 g.

The seismic stability assessment of the waste rock pile should include an estimation of potential seismically induced deformations from the design earthquake. Potential deformations under earthquake loading may be estimated using the simplified methods of Newmark (1965) and Makdisi-Seed (1977).

Detailed site investigations and stability analyses will be required for future design studies and prior to development of the Waste Rock Storage Area.

18.5.5 Reclamation

Preliminary closure requirements for the Waste Rock Storage Area will include on-going monitoring of surface and groundwater quality and flow rates, and periodic inspection of the waste rock dump slopes.

Reclamation will be carried out in conjunction with on-going environmental monitoring to ensure that sediment control and water quality objectives are met. The final waste rock bench crests will be rounded to provide long-term stability. The bench tops of the final waste rock dump will be covered with a suitable topsoil layer and re-vegetated.

18.6 OPERATIONAL STRATEGY

The proposed heap leach pad will be developed in stages. This will minimize initial capital costs while allowing mine operators to maximize efficiency and recovery during leaching. The general operational strategy will involve placing the ore in successive lifts followed by irrigation with leach solutions. The lifts will be placed by conveyor or truck, as appropriate, and spread with dozers. The ore will then be placed in strips parallel to the slope and loaded upslope to ensure ongoing stability and to prevent trafficking over the liner. Proposed bench lifts of 25 feet should be constructed at bench face angles of approximately 1.4H:1V. Berm widths of 36 feet will be left at the toe of each lift resulting in an overall slope of 2.5H:1V. Ongoing pad development will involve extending the composite liner upslope to allow for continued lift placement.

Leachate solution will be collected by a series of drainage pipes within the coarse drainage layer at the base of the pad. The drain pipes will connect to main leachate collector pipes for transport to the vertical riser pipe within the potential ponding area.

The sequence of lift placement and leaching will likely be as follows:

- Place 1.5 inch minus overliner material on the surface of the geomembrane liner to provide liner protection and base drainage.
- Place 25 foot lift of ore, as required, by moveable ore transport conveyors or haul trucks. Spread with dozers to establish an evenly graded surface for leaching.
- Layout irrigation lines for drip leaching. Sprinkler leaching may also be possible during summer months as part of a rotational, cell-type leach operation.
- Cover irrigation lines with ore having a thickness greater than the depth-of-freeze in the fall to prevent freezing during winter operations.
- Leach the lift of ore above the irrigation lines prior to loading with the next 25 foot lift.

18.7 MONITORING AND RECLAMATION

Monitoring and reclamation will be carried out on an ongoing basis to ensure the safe and effective operation of the heap leach facility while minimizing impacts to the surrounding environment. Preliminary recommendations for monitoring and reclamation are summarized below:

18.7.1 Monitoring:

Surface water quality sampling in surrounding creeks downstream of the facility.

Installation of monitoring wells around the facility to monitor groundwater quality during operations and at closure. These wells would be installed prior to development to obtain baseline information for comparative assessment.

Installation of an LDRS to monitor and recover any leakage through the liner systems within the solution transfer channels and process water ponds.

Slope movement monuments and survey control points will be installed and monitored to ensure the integrity and stability of the ore heap.

18.7.2 Reclamation:

Grading and re-vegetation of final heap slopes to provide adequate drainage and erosion protection from surface runoff. This may be carried out during operations as the final slope of the heap is developed.

Detoxification of the ore at the end of operations.

Removal of geosynthetic liners from the heap leach pad areas including pregnant and barren solution ponds as required.

Decommissioning of the pregnant solution recovery system.

Removal of pregnant solution pond pumps and pipe works.
Application of topsoil and seed mixture to all disturbed areas, as required.

Details of the monitoring and reclamation plans will be developed in conjunction with the appropriate regulatory authorities.

18.8 SITE WATER MANAGEMENT

18.8.1 General

Storm water diversions are required to redirect surface runoff away from mining-related infrastructure and disturbances.

18.8.2 Design Storm

The Robertson Property is located in central Nevada. The average rainfall intensity for the design of storm water diversion structures utilized data provided by the National Oceanic and Atmospheric Administration's Point Precipitation Frequency Estimates and the National Weather Service's Hydrometeorological Design Studies Center. Table 18.2 lists the annual return interval (ARI) events for the region and Figure 18-9 depicts the depth-duration-frequency curves for the project area.

Table 18.2: Design Storm Events[1]

	Annual Return Interval [years]							
	1	2	5	10	25	50	100	200
24-hr Precip. Depth [in]	0.873	1.09	1.39	1.63	1.96	2.22	2.50	2.79

[1] Design storm events developed using NOAA Point Precipitation Frequency Estimates

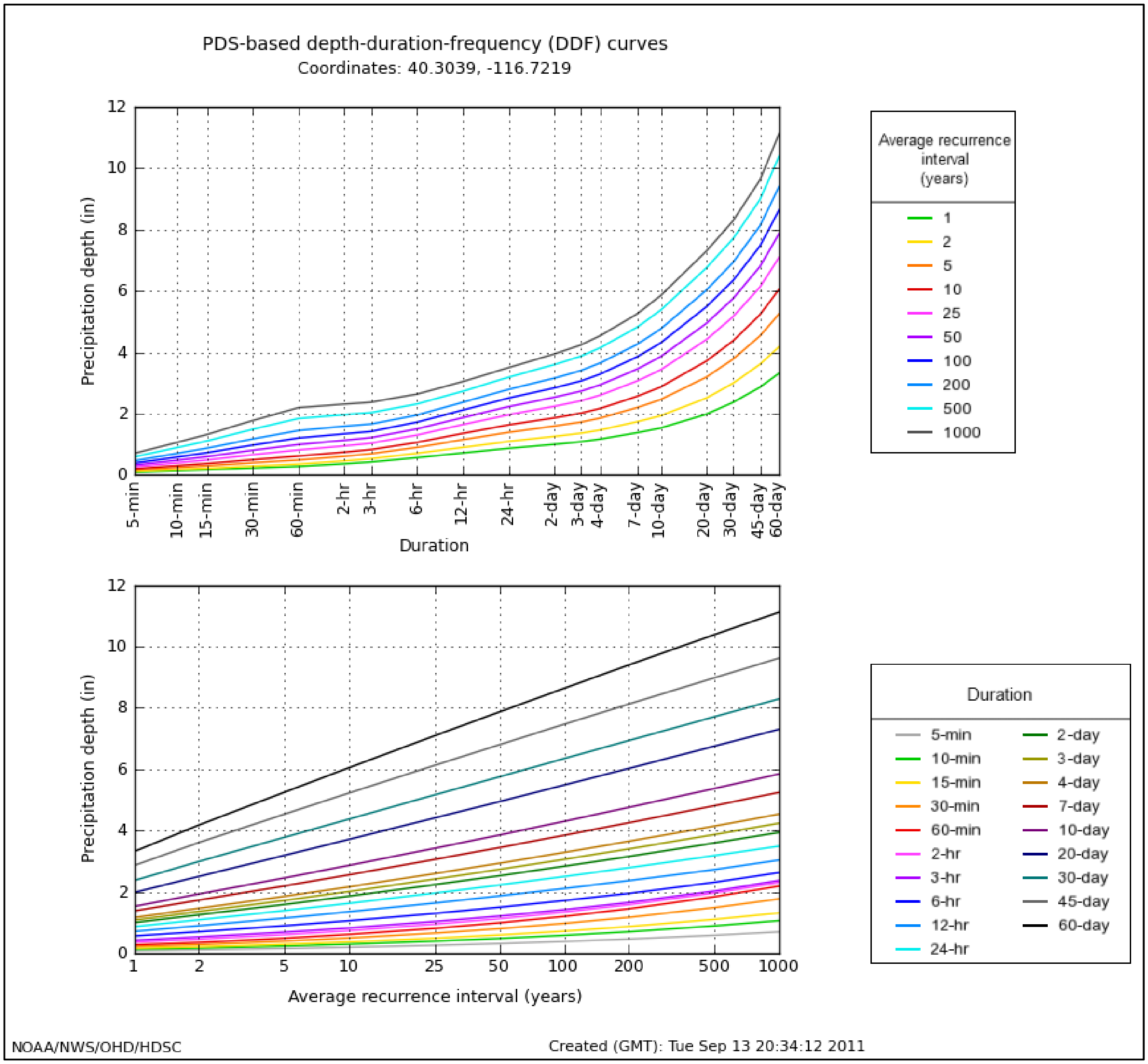


Figure 18-8: NOAA Depth Duration Curves for Project Area[1]

[1] Generated by NOAA Website – Sep. 13, 2011

18.8.3 Design Methodology

A hydraulic analysis was carried out for mine site to determine the storm water runoff volumes and flow rates for sizing on site diversion channels. The SCS curve number method presented in the United States Department of Agriculture (USDA) Technical Release 55 (TR-55) Urban Hydrology for Small Catchments was used for the analysis.

18.8.3.1 Catchment Characteristics

It is anticipated that the mine site footprint will be approximately 1,500 acres. The hydrologic characteristics of the on-site soils are currently undefined; therefore a curve number (CN) of 70 was assumed for analysis based on the following assumptions:

- Soils in the site belong to hydrologic soil group D.
- Vegetation at the site consists of sagebrush with grass understory in fair condition.

Disturbed areas were assessed a CN value of 80, while the waste rock storage area and heap leach pads were given CN values of 70.

18.8.3.2 Channel Geometry

In order to determine the appropriate size of the storm water diversion channels the peak runoff from the hydrologic analyses was used to determine the capacities. For analysis it was assumed that all channels will be designed with 2H:1V side slopes and minimum longitudinal slopes of 1%. Channel roughness was assumed to correspond to a Manning's coefficient of 0.025 based on relatively smooth constructed channel conditions.

18.8.3.3 Design Flow and Channel Capacity

Table 18.3: Channel Design

Channel	Design Flow by Return Period [cfs]							Design Depth [ft][1]
	2	5	10	25	50	100	200	
Waste Rock Dump Toe	0.2	2	5	15	26	40	57	1.5 + 1.0 = 2.5
South Diversion	0.2	0.9	3	8	15	23	33	1.5 + 1.0 = 2.5
North Diversion	0.1	0.7	3	8	15	23	32	1.5 + 1.0 = 2.5
LG Solution Collection[2]	0.3	1.5	4	11	17	26	37	2.0 + 3.0 = 5.0
HG Solution Collection[2]	0.3	1.7	5	13	21	31	44	2.0 + 3.0 = 5.0

[1] All conveyance ditches sized to convey the 100 year discharge plus 1.0 ft freeboard.
[2] Solution transfer channels sized to convey 100 year discharge plus 3.0 ft freeboard.

18.8.3.4 Event Pond Design

The event pond is hydraulically connected to the pregnant solution pond via a broad-crested overflow located along the east side of the pregnant solution pond. The overflow is sized to convey flows in excess of the 25-year storm event of 24 hours duration.

The pond will be maintained in dry condition to provide contingency storage in the event of a large storm water event or to accommodate planned or unplanned draindown of either or both heap leach pads. Once the given event has passed, the solution will be pumped to the ADR plant for gold recovery and cyanide detoxification if required.

18.9 ELECTRICAL SUPPLY AND DISTRIBUTION

18.9.1 Summary

This portion of the project Preliminary Economic Analysis (PEA) covers electrical power, control and communications aspects and includes both capital and operating costs.

This electrical report is based on the development of a 40,000 ton per day surface mine and processing facility. The cost estimates are expressed in US dollars and are based on constant 2011 prices.

The proposed facilities covered by the electrical PEA include the mine (mining equipment, dewatering pumping (if required) and portable lighting will be diesel powered) and surface facilities including a crushing plant, ore conveyors, leach pads, ponds, ADR plant, fresh water pumping and ancillary buildings consisting of a gate house, offices, warehouse, dry, workshop/service garage and fuel storage.

Electrical power will be supplied by NV Energy (formerly Sierra Pacific Power Company). In 2017, it is planned to establish a 25 kV service via a tap from an existing nearby 120 kV line, a 120 - 25 kV substation and two miles of 25 kV distribution line.

The Robertson Project's service point will be located at a main site substation to be built and owned by Coral Gold adjacent to the ADR Plant. The main substation will include NV Energy's revenue metering, two step-down transformers and the main secondary 4.16kV distribution switchgear. This switchgear will then distribute power to the various load centers around the site via buried cable to nearby loads and overhead lines to more distant loads, e.g. crushing, pregnant solution and fresh water pumping and the mine workshop. To provide power during a utility outage, an emergency generator will be installed to allow an orderly shutdown of the plant, ongoing communications, and freeze protection.

Electrical services will be generally designed and built to current North American best available technology standards and will include grounding, lighting, security, small power, welding, heating, process power, process control, instrumentation, fire protection, site and off-site communications systems. For this study, all materials and equipment have been assumed to be new.

18.9.2 Introduction

This section of the report has been prepared by Kaehne Consulting Ltd. (KCL) of North Vancouver, BC under the direction and at the request of Beacon Hill Consultants (1988) Ltd., consultants to the project, who are conducting the overall Preliminary Economic Analysis (PEA) of the Robertson Project near Crescent Valley, Nevada on behalf of Owners, Coral Gold Resources Ltd. (Coral).

The purpose of this report is to provide a PEA of the design, capital and operating cost of electrical power supply, distribution and services for the Robertson project based on development of a surface mine, crushing, conveying, leach pads, pumping, processing plant and ancillary buildings.

18.9.3 Electrical Power Supply

The Robertson project is estimated to have a peak electrical load demand (15 minute thermal average) of 2 MW, a load factor of 0.8 and an average annual energy consumption of 14 GWh.

Average power factor will be 0.95 lagging or better, achieved with power factor correcting capacitors.

Electrical power supply will be provided by NV Energy who will develop a new tap from a nearby 120 kV transmission line, install a 120 - 25 kV step-down substation near the tap, and run two miles of new 25 kV distribution line to a new main substation near the project main entrance and gate house. NV Energy has provided a preliminary capital estimate of $1 million to cover this work.

Coral will build and own a 25 - 4.16 kV main site substation and the downstream 4.16 kV three phase distribution system from the service point at the main substation to various load centres around the property.

NV Energy has indicated that the addition of the Coral load of 2 MW to the existing 120 kV transmission system is most likely possible under present system loading conditions. They do advise, however, that before their offer can be considered firm, a system study would be required at a possible cost to Coral of about $20,000. In our opinion, this is normal utility practice and is considered reasonable.

Correspondence has been done with NV Energy to determine the details of the GS3 tariff applicable to Coral and their calculation of the present average unit cost of energy that would apply. Note that the GS3 tariff is a “time of use” tariff, meaning that the rate changes with summer and winter seasons and on-, mid-, and off-peak daily periods.

18.9.3.1 Main Site Substation

Coral will build and own a new main site substation consisting of twin 1.5/2.0 MVA 25 - 4.16 kV three phase outdoor power transformers complete with primary isolation and protective devices. NV Energy’s revenue metering will be located here and check metering may be installed if desired. Either one of the main transformers will have sufficient capacity to carry the plant load at its forced cooling rating. A line-up of secondary 4.16 kV switchgear will provide the source for various power feeders to the mine, crushing, ADR plant, pumping, ancillary buildings and miscellaneous loads.

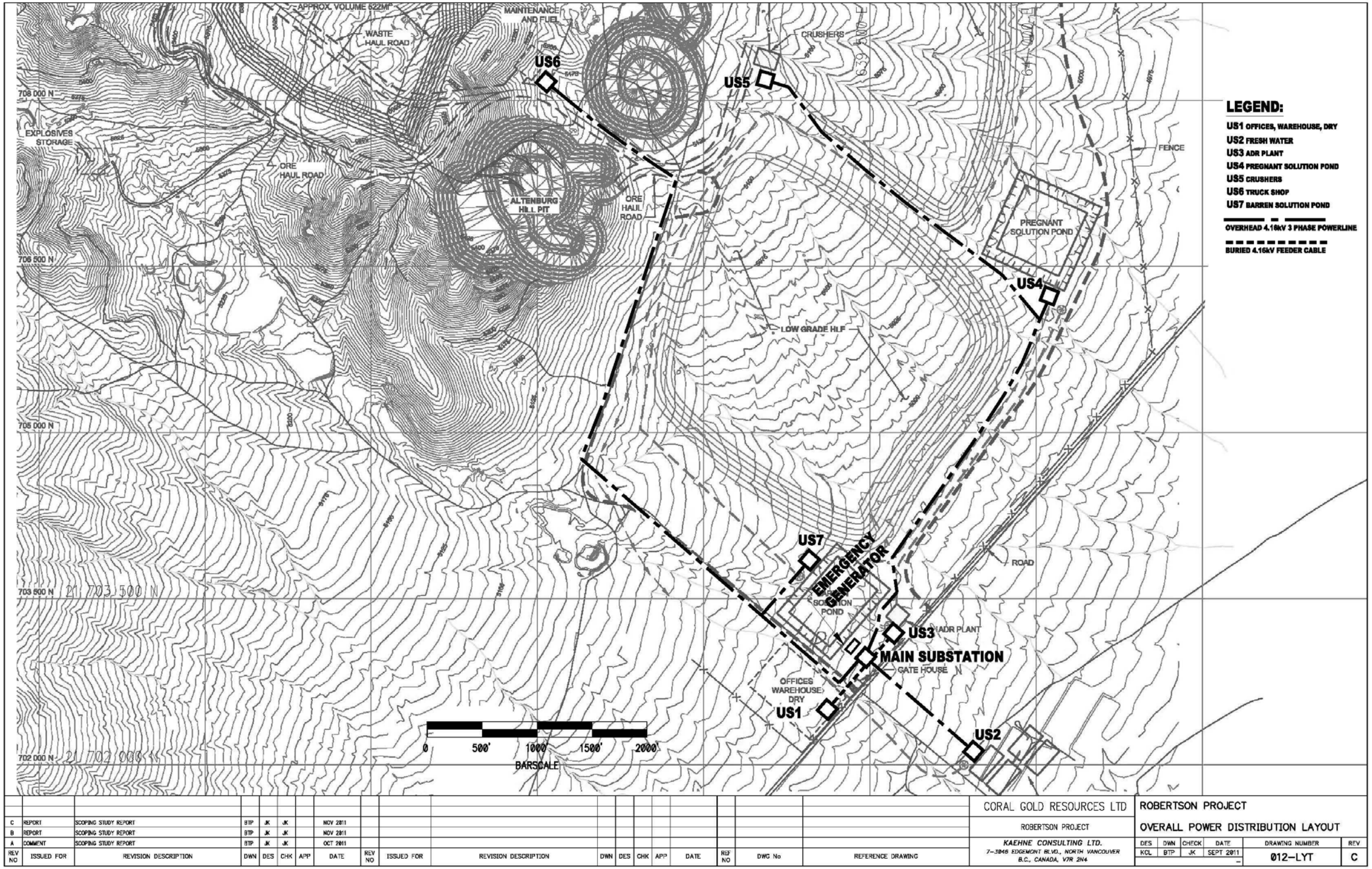


Figure 18-9: Overall Power Distribution Layout

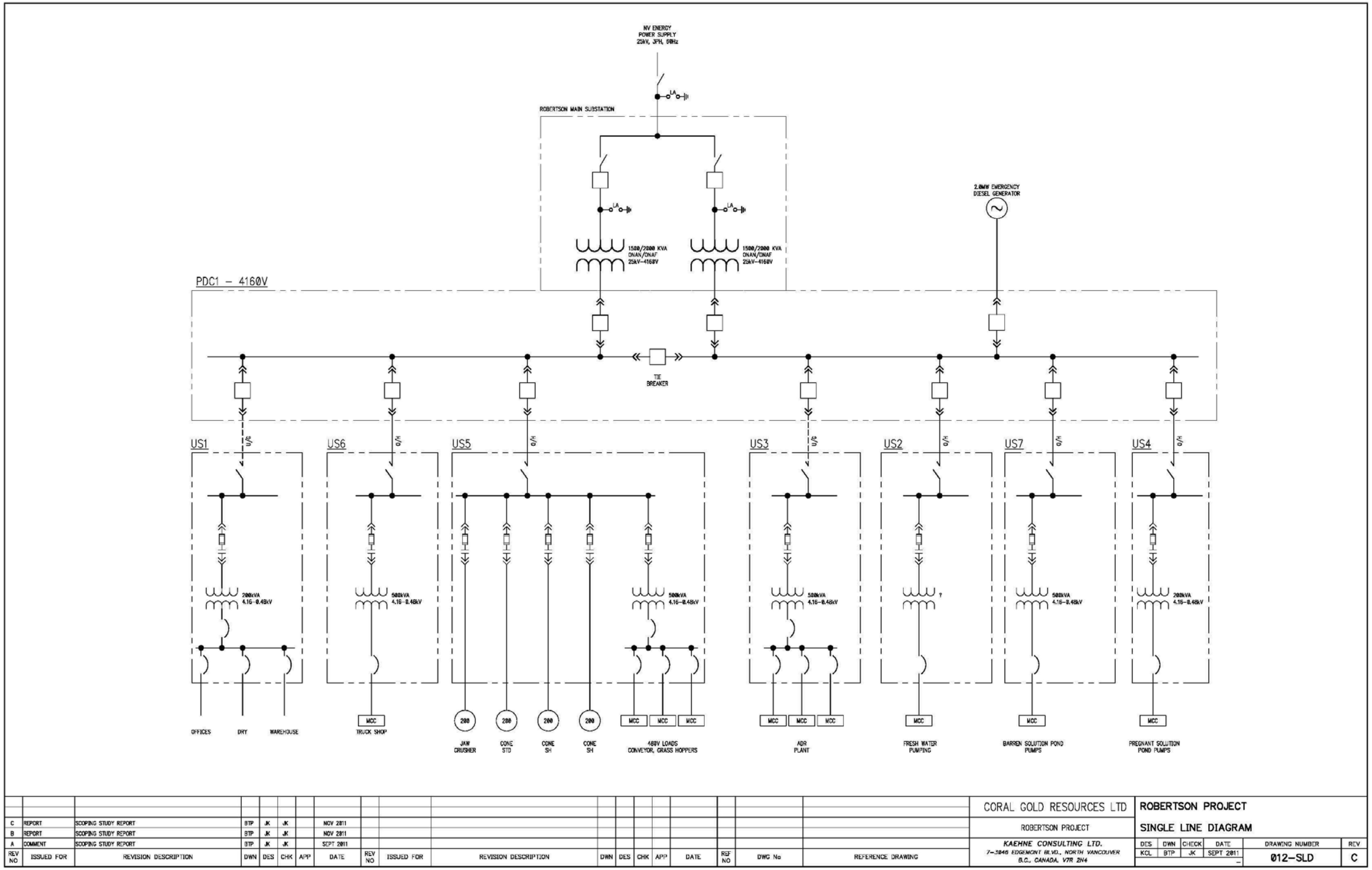


Figure 18-10: Electrical Single Line Diagram

18.9.4 Electrical Distribution

Power will be distributed from the main site substation at 4.16 kV, three phase, 60 Hertz to load centres via overhead power lines or cable feeders mounted on ladder trays or buried underground, as appropriate. Local to each load centre, a unit substation will step power down to 480 volts and lower for general consumption. Overhead power lines at 4.16 kV will extend to the mine area, crushing and freshwater pumping.

18.9.5 Process Drive Motors

Process drive motors will be totally enclosed fan cooled (TEFC), high efficiency. These motors have been priced as part of the mechanical equipment they drive and are not included in this portion of the cost estimates. With the exception of the 200 HP crusher motors which will be 4 kV, all other process motors will be 460 volt three phase or, for small auxiliary motors, 120/208 or 120/240 volts. Larger drives will include power factor correction via static capacitors.

18.9.6 Grounding

The main substation will be provided with an overall ground grid, gradient control mats where required and structure and fence grounding systems. Overall ground resistivity is limited to 1 ohm to ensure acceptable touch-and-step potentials. If necessary, a remote ground electrode will be used. All buildings and major outdoor structures will be connected to a buried grounding system with a maximum resistance to ground of 5 ohms. This may be achieved with grounding rods or few deep holes with copper piping installed.

18.9.7 Lighting

Lighting will be provided to meet safety and statutory requirements. Highbay discharge lighting will be used in process and service buildings with a mounting height of 20 feet or more. Lowbay vapourtight discharge lighting will be provided in other process and service areas, fluorescent lighting in offices, control rooms and similar. Exit lights will be provided where required. Emergency lighting will provide minimal lighting for safe egress in event of a power failure. Outdoor lighting will be provided where required from fixtures mounted on building exteriors. Pit lighting will be provided by mobile trailer-mounted units.

18.9.8 Fire Detection & Suppression

POC (products of combustion) and ROR (rate of rise) detector heads will be installed in transformer rooms, electrical rooms and control rooms. No sprinkling is permitted in these areas. Suppression systems will be either CO_2 or halon replacement, manually operated.

18.9.9 Security

An allowance has been made for surveillance/security systems consisting of cameras and monitoring equipment. It is expected that all site security will be located at the guardhouse located at the main property entrance.

18.9.10 Process Control / Instrumentation

The main elements of the process control equipment will be controlled/monitored through a fully redundant PC based HMI (Human-Machine Interface) similar to Wonderware's Factory Suite and will be centralized at a control room in the ADR building. Control will be PLC based and will not have manual override.

Conveyors will have standard control devices: emergency pull cord switches, belt misalignment switches, speed and plugged chute switches where applicable.

The primary ore conveyor will have a weightometer. Process water tanks will have continuous level monitoring with ultrasonic level transmitters.

18.9.11 Plant Site Communications

An allowance has been made for an in-plant intercom system. This will be a self-contained system with multi-channel selective calling.

Allowance has been made for telephone, data and internet communications to and from the Robertson project site.

18.10 OTHER SURFACE INFRASRUCTURE

18.10.1 Transportation of Doré Bars

The transportation of the gold doré bars does not pose any concerns. It is likely that they would be transported to Bratislava by a security truck specially designed for carrying small value loads. Security personnel trained for this purpose will travel with the truck.

18.10.2 Fuel Storage

Fuel storage requirements at the site have been estimated at 500,000 liters based upon an average one-month consumption. Access to the site is available at all times, thus delivery of fuel would be on a regular schedule.

18.10.3 Explosive Storage

Minimal explosives will be stored on site since access to explosives in the area on a regular basis is feasible.

18.10.4 Warehouse/Shops

The warehouse will be 67ft (20 m) wide x 140ft (42 m), and will include offices. The maintenance facilities will have two service bays, a wash bay, a tire repair bay, small truck repair bays, offices, and a shop area.

18.10.5 Administration Building / Security and Gatehouse

The administration building consists of a two-storey building designed to provide sufficient area for the projected staff and associated persons.

The security building will have sufficient facilities to store emergency vehicles and serve as a gatehouse. Figures 18-9 through 18-15 outline the proposed surface ancillary facilities.

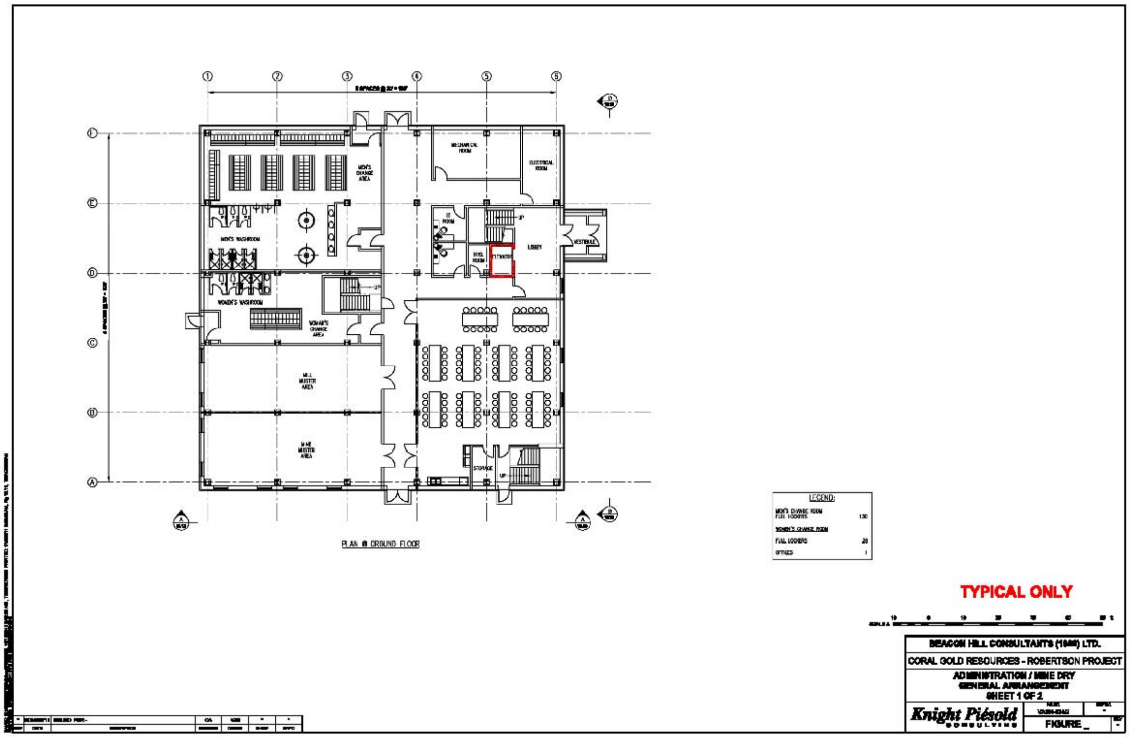


Figure 18-11: Administration / Mine Dry – General Arrangement Plan (Sheet 1 of 2)

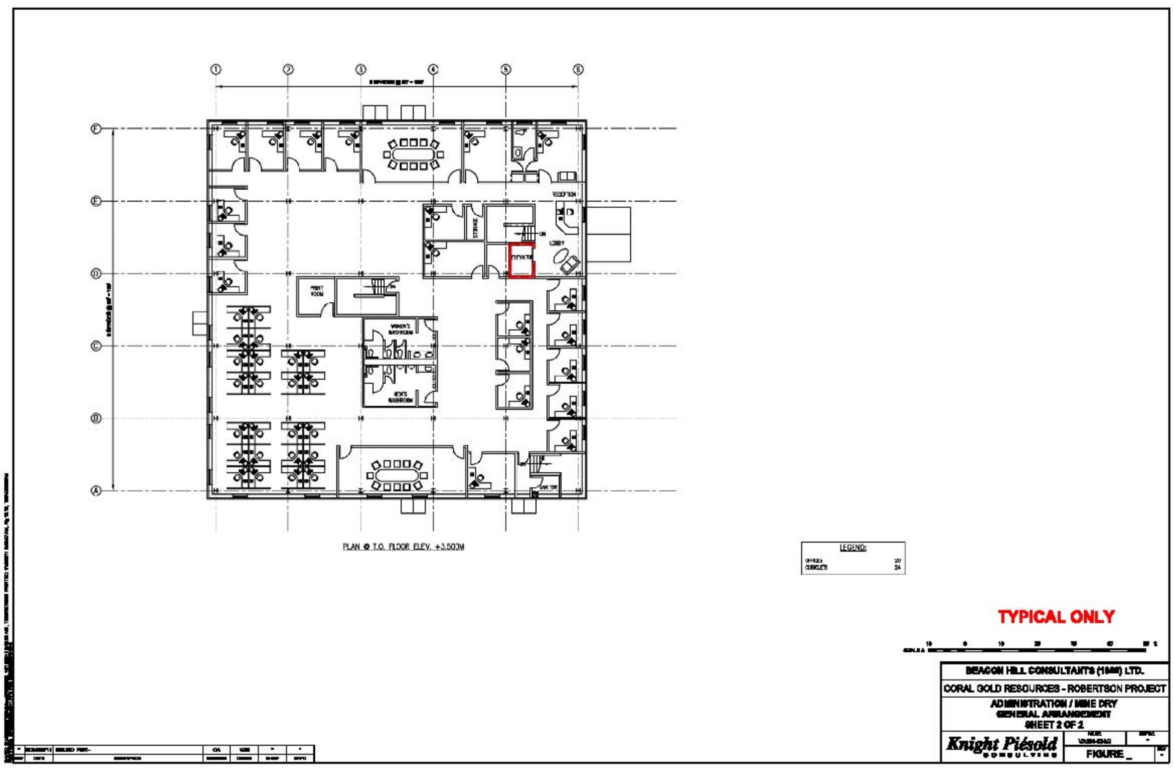


Figure 18-12: Administration / Mine Dry – General Arrangement Plan (Sheet 2 of 2)

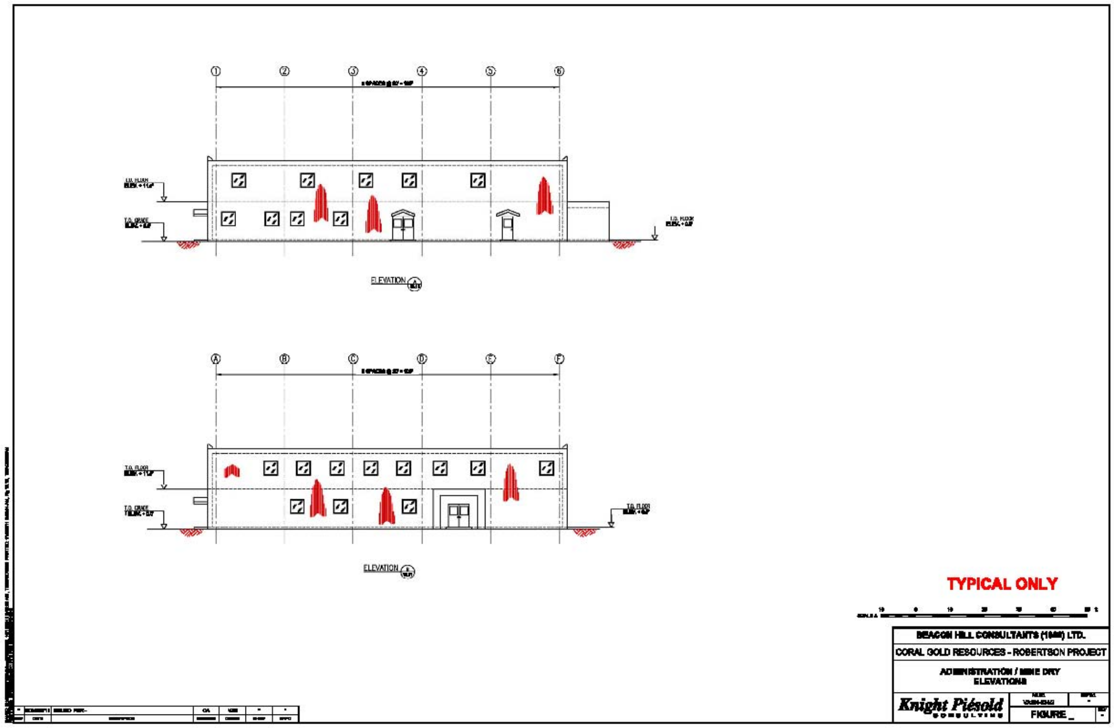


Figure 18-13: Administration / Mine Dry – General Arrangement Elevations

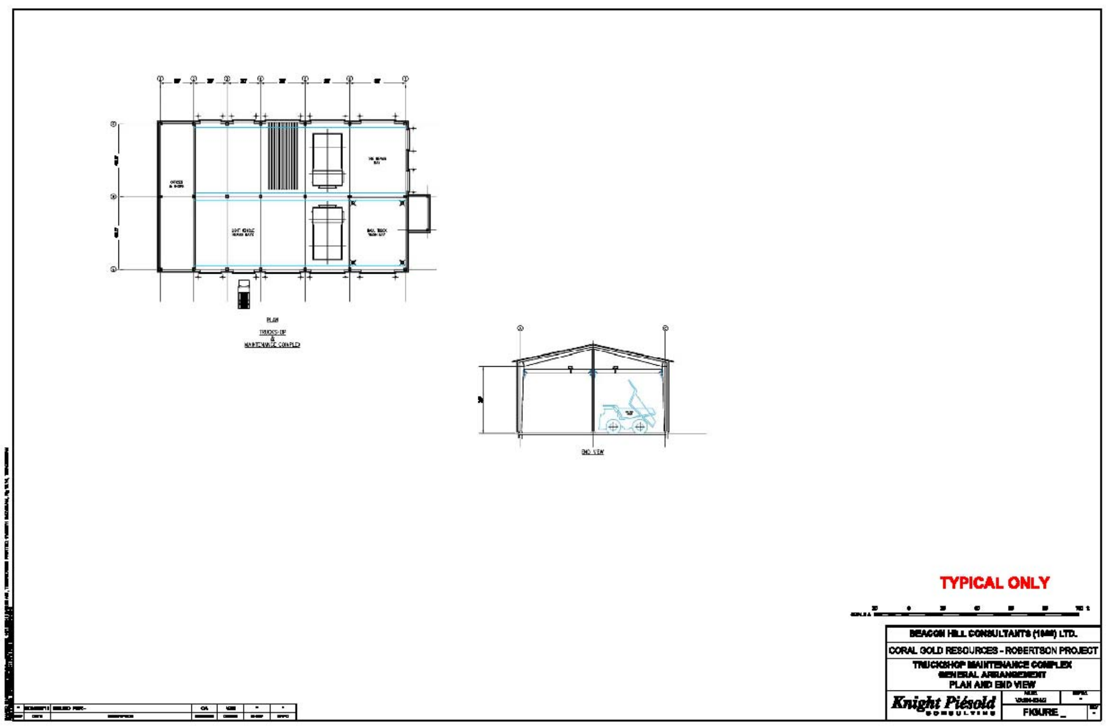


Figure 18-14: Truckshop Maintenance Complex – General Arrangement Plan and End View

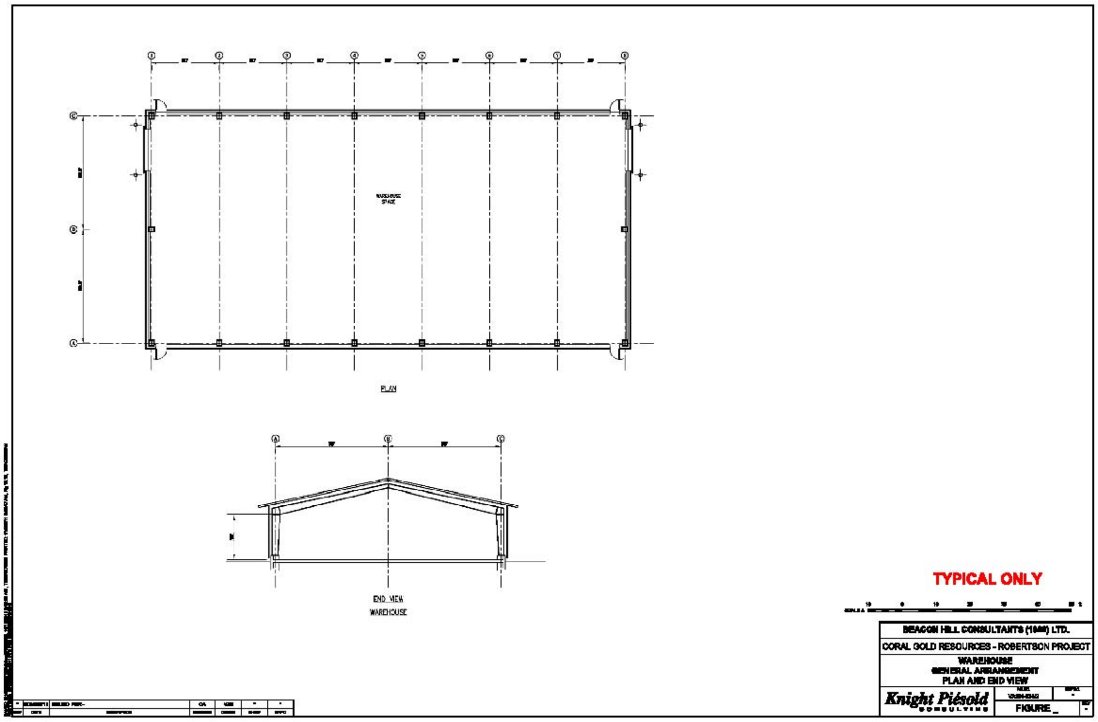


Figure 18-15: Warehouse – General Arrangement Plan and End View

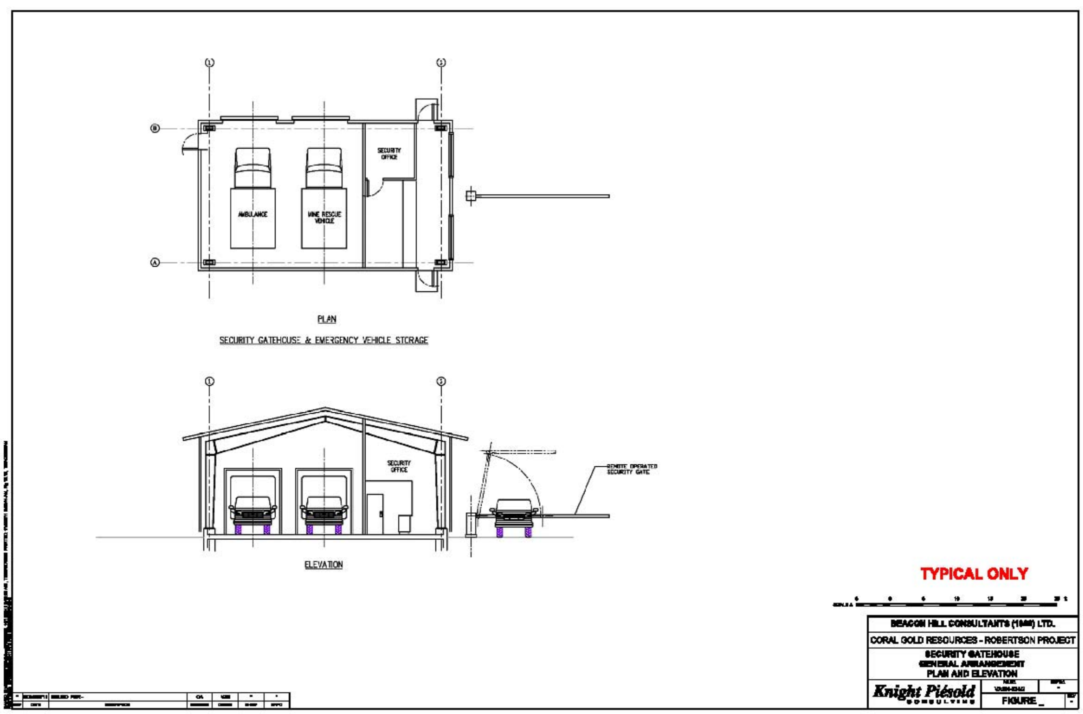


Figure 18-16: Security Gatehouse – General Arrangement Plan and Elevation

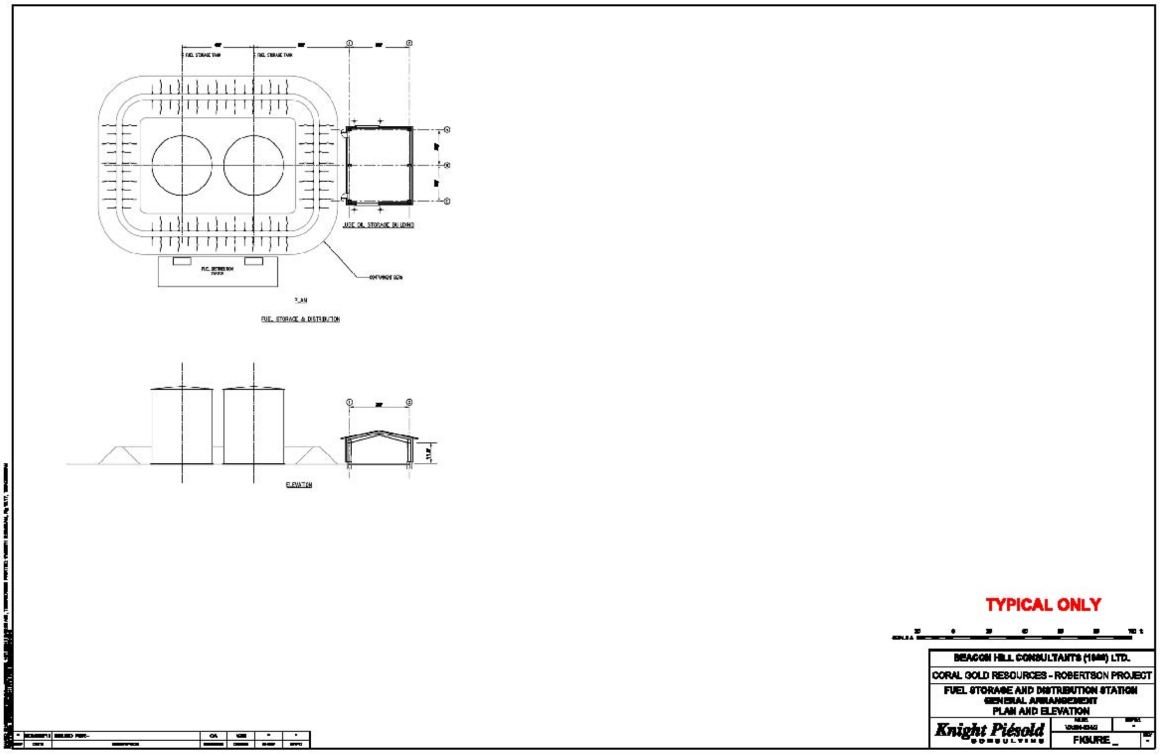


Figure 18-17: Fuel Storage and Distribution Station – General Arrangement Plan and Elevation

SECTION 19.0 MARKET STUDIES AND CONTRACTS

There have been no market studies completed and there are no contracts.

Gold Dore bar will be the product produced from this mining operation which has a ready market at the prevailing metal prices.

SECTION 20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1.1 Introduction

SRK Consulting (U.S.), Inc. (SRK) has prepared this environmental section for the Robertson Project Preliminary Economic Assessment. The work scope submitted to Beacon Hill Consultants (1988) Ltd. Identified the elements that are included in this document:

- Coordinate with the Beacon Hill team members regarding the mine plan and processing to assess the types of permits required for the anticipated operations;
- Review current permits and additional permits required to bring the project into full production;
- Create a permitting time line for future mining; and
- Review of current environmental liabilities, and assessment of requirements to bring to meet state and federal regulations.

List of Abbreviations

ABA	Acid Base Accounting
BLM	Bureau of Land Management
BMRR	Bureau of Mining Regulation and Reclamation
BSDW	Bureau of Safe Drinking Water
BWPC	Bureau of Water Pollution Control
BWM	Bureau of Waste Management
EA	Environmental assessment
EIS	Environmental impact statement
ET	Evapotranspiration
FPPC	Final Plan for Permanent Closure
HCT	Humidity Cell Test
MLFO	Mount Lewis Field Office
MWMP	Meteoric Water Mobility Procedure Test
NDEP	Nevada Division of Environmental Protection
NEPA	National Environmental Policy Act
NDOW	Nevada Department of Wildlife
NRHP	National Register of Historic Places
SWPPP	Stormwater pollution prevention plan

The Project area consists of approximately 7300 acres, of which 169 acres are private lands, held as patented mining claims either owned or controlled by Coral Gold Resources, Inc. (Coral). The remaining 7131 acres are public lands administered by the BLM Mount Lewis Field Office (MLFO) in Lander County; Coral controls approximately 601 unpatented lode and placer claims on these public lands.

This mixed estate makes the MLFO the primary agency for authorizing mining activities on public and private land; the MLFO works with the Nevada Division of Environmental Protection – Bureau of Mining Regulation and Reclamation (BMRR) under a memorandum of understanding to authorize mining on projects on both public and private lands.

The Robertson Project was developed in the Tenabo Sub-district of the Bullion mining district, which has been the focus of historic and modern mineral development operations. The historic operations, which began in 1869, are comprised primarily of underground and placer operations while the modern facilities have been focused primarily on open pit/heap leach facilities in the western portion of the district.

The Project area was first mined in 1905, and the town of Tenabo was founded. Placer gold was discovered in 1907 and again in 1916, but significant production did not begin until 1931. Since that time, mining has been limited, and production has been small. Coral operated the Robertson Mine in 1988 and 1989. The site is presently in post-closure monitoring; however, exploration activities have continued. Reclamation activities associated with previous mining disturbance as well as recent exploration disturbance have been ongoing. The site also has extensive mining-related disturbance by other parties.

20.1.2 Authorized Exploration Activities

Exploration operations on public lands administered by the MLFO have been authorized under the previous Plan for the Robertson Project (Plan) (NVN-067688 submitted in September 1986 and subsequently modified in June 1990, January 1991, July 1997, October 2003, September 2004, May 2006, and November 2007) and under a Notice of Operations (Notice) for the Try-View Project (NVN-087413 authorized June 8, 2009). The BMRR issued Reclamation Permit #0055 to cover this disturbance.

Coral is authorized to create 211.5 acres of surface disturbance under the Plan and reclamation permit for the Robertson Project and 2.45 acres of surface disturbance under the Notice for the Try-View Project, for a total of 214 acres. The site also has extensive mining-related disturbance by other parties of which Coral has elected to reclaim some portions.

To date, about 19 acres of waste rock dump and 13 acres of process area have complete surety release leaving about 214 acres that have not been entirely reclaimed and released. Of these 214 acres, about 39 acres are associated with roads and 11 acres are associated with exploration.

Table 20.1: Authorized and Existing Disturbance

Disturbance Type	Pre-2008 Public (acres)	Pre-2008 Private (acres)	2008 Authorized Public (acres)	2008 Authorized Private (acres)	Surety Release (acres)	Total Reclamation Obligation (acres)
Pits	19.9	0.0	0.0	0.0	0.0	*19.9*
Waste Rock Dumps[1]	53.5	0.0	0.0	0.0	18.7	*34.8*
Process Areas[2]	51.5	0.0	0.0	0.0	12.8	*38.7*
Roads	36.3	2.6	0.0	0.0	0.0	*38.9*
Exploration	0.8	0.1	9.7	0.7	0.0	*11.3*
General[3]	61.9	6.0	0.0	0.0	0.0	*67.9*
Other[4]	0.0	0.0	2.2	0.0	0.0	*2.2*
TOTAL	***223.9***	***8.7***	***11.9***	***0.7***	***31.5***	***213.7***

[1] Includes the Placer Gravel Dump.
[2] Includes the main heap leach pad, the former Gold Quartz pad, the plant yard and ponds, the freshwater reservoir, and the Triplet Gulch pad grubbed area.
[3] Includes general disturbance around the site including the “A” – “D” surface disturbances.

[4] Previously categorized as disturbance by others that Coral recontoured and now assumes reclamation responsibility.

The bond amount that is presently obligated to reclaim the 214 acres of disturbance is US$352,934.

Coral is presently constrained on disturbance associated with exploration activities as the allotted acreage and number of holes and pads authorized in the 2007 Plan amendment. The authorized disturbance included 10.4 acres for 50 drill sites and sumps as well as cross-country travel and new roads.

In addition to the water pollution control permit, the reclamation permit, and BLM authorization, Coral presently maintains a general stormwater permit with a stormwater pollution prevention plan.

20.1.3 Proposed Exploration Activities

An amended Plan and reclamation permit application was submitted to the MLFO and BMRR in April 2010 and was then withdrawn at the MLFO's suggestion to incorporate a different strategy. The revised Plan was re-submitted to the MLFO and BMRR in November 2010 and has undergone multiple MLFO reviews. This Plan proposes an additional 80 acres of disturbance associated with exploration and baseline data collection activities; the Try View notice will be rolled into this action, and the notice will be retired

An environmental assessment (EA) will have to be prepared to analyze potential environmental impacts associated with the proposed drilling activities. Baseline data collection for threatened and endangered plant and animal species has been completed, and Class III cultural resource surveys are underway. According to the cultural resources consultant, Kautz Environmental, Inc., the area contains numerous cultural resources that may be eligible for the National Register of Historic Places (NRHP) (personal comm. B. Malinky). Further, the Gylding Pit contains an eagle nest which has potential ramifications for curtailing exploration and mining activities within a buffer zone during the nesting season.

The May 2011 Plan responding to the MLFO comments has been re-submitted and has been deemed complete, so the EA can be formally initiated. This process usually takes six to eight months to complete; however, the MLFO presently has a staff shortage, so work on some projects has been delayed indefinitely. The best case scenario is that the EA can be initiated in July 2011 and be completed in seven months.

20.1.4 Future Mining

20.1.4.1 Conceptual Plan

A conceptual mining plan was provided that depicted the following facilities:

- three pits: Gold Pan, Porphyry, and Altenburg Hill;
- a waste rock dump;
- a low grade heap and a high grade heap; and
- a transmission line.

Other components that will be needed include:

- haul and secondary roads;
- processing facilities and solution ponds;
- water wells, pipelines, and water storage;
- shops, warehouse, and fuel and reagent storage;
- sanitary facilities and potable water;
- growth media stockpiles, borrow areas, fencing, exploration, power lines, landfill, and stormwater controls.

Because the future mine will be located on both public and private land, federal, state, and local permits will have to be acquired as shown in Table 20.2.

Table 20.2: Summary of Major Permits for Future Mining

Agency	**Permit Name**
Nevada Division of Environmental Protection	
Bureau of Mining Regulation and Reclamation	• Water Pollution Control Permit • Reclamation Permit
Bureau of Air Pollution Control	• Class II Permits to Construct and Operate • Mercury Permit
Bureau of Water Pollution Control	• General Stormwater Permit • Septic Permit
Bureau of Safe Drinking Water	• Potable Water Permit
Nevada Division of Water Resources	
State Engineer	• Permit to Appropriate Water
Division of Dam Safety	• Permit to Construct Dam
Nevada Department of Wildlife	
	• Industrial Artificial Pond Permit
Federal Permits	
Bureau of Land Management – Mount Lewis Field Office	• Decision Record/Finding of No Significant Impact
Bureau of Alcohol, Tobacco, Firearms, and Explosives	• Authorization to store and use explosives
Environmental Protection Agency	• Hazardous Waste ID No. (small quantity generator)
Lander County	
	• Special Use Permit

20.1.5 Federal Permitting

A mine plan of operations (Mine Plan) will have to be prepared to describe the construction, operation, reclamation, and closure of each facility along with a bond cost estimate that presents the reclamation and closure costs.

The Plan has to provide sufficient detail in order to identify and disclose potential environmental issues during the National Environmental Policy Act (NEPA) review, which will require the preparation of an environmental assessment (EA) or environmental impact statement (EIS).

The primary difference between the two types of documents is that an EA is prepared when no significant impacts are expected or the potential impacts are unknown. An EIS acknowledges the potential for significant impacts to occur, and analyzes and discloses what those potential impacts are. The BLM will look at several triggers to determine whether an EA or an EIS is the most appropriate document to disclose potential environmental impacts.

Because the Coral Project area is in such close proximity to the Cortez Gold Mines, the BLM may consider the development and operation of the mine to be significant. Based on recent experience the BLM will likely require an EIS due to the potential for cumulative impacts.

There are potential issues and risks of preparing and EA in lieu of an EIS. The greatest risk is that an EA is imminently more appealable than an EIS. It is far easier to appeal an EA than and EIS. Due to the size of the environmental footprint of the project and the general opinion that the EIS route is the likely requirement by governmental agencies this approach forms the basis for approval process for this project.

Recent experience in the Battle Mountain District indicates it is likely that an EIS for a mining/mineral processing project will take at least 36 months. The process is controlled by internal BLM situations that are beyond the control of the project proponent, and also a number of potential external events such as public or cooperating agency opposition that could, and often do, delay the EIS schedule. Thus, it is not uncommon for a mining EIS to take three to five years.

The MLFO is requiring that at least one year of baseline data be submitted with the Plan of Operations/Reclamation Permit Application. The ongoing cultural resources survey is expected to be completed in the fourth quarter of 2011, which will be a major milestone in baseline data acquisition.

Long-lead items that must be considered include groundwater sampling in the project area for depth and quality for use in the NEPA analysis and water pollution control permit application. Geochemical characterizations of waste rock, ore, and spent ore are another study that should be started as soon as possible and include acid base accounting (ABA), meteoric water mobility procedures (MWMP) testing, and humidity cell (HCT) testing. The geochemical characterization program must be approved in advance by the BLM and the NDEP.

Table 20.3 presents a listing of the types of studies that should be undertaken during the mine planning phase and in advance of the NEPA process; these studies will also be used to support the acquisition of other permits.

Table 20.3: Future Baseline Studies

	Permit/Authorization	Investigations/Studies	Estimated Range of Costs
Water	• NEPA Analysis • Water pollution control permit • Stormwater control	• Monitor surface waters in project vicinity on a seasonal basis for quality and quantity • Monitor groundwater for level and water quality especially in the pit, dump, and heap areas to collect baseline quality data	$10,000 – $30,000
Geology and Geochemistry	• NEPA Analysis • Water pollution control permit • Waste rock dump design • Dump and heap closure • Closure planning for dumps, heaps, and tailings	• Collect representative samples of waste rock, ore, and spent heap ore for geochemical characterization (acid base accounting, Meteroric Water Mobility Procedure testing, and humidity cell testing • Condemnation drilling in proposed locations of facilities	Costs will be dependent on the number of samples analyzed for ABA, MWMP, and HCT $20,000 - $50,000
Cultural Resources	• NEPA Analysis	• Conduct a Class III survey in previously unsurveyed or as directed by the MLFO[1] • Mitigate sites that cannot be avoided	Mitigation costs are unknown at this time but could be as high as $50,000/site
Biological baseline	• NEPA analysis	• Determine presence or absence of threatened, endangered, or special status plant and animal species including golden eagles in previously unsurveyed areas[1] • Determine presence or absence of game species	Will have to survey for raptors in a 10-mile radius, but may be able to use info from nearby mines.

[1]These studies will have been completed for the exploration EA.

Other federal permits that may have to be acquired include a hazardous waste identification number from the U.S. Environmental Protection Agency and an explosives use permit from the Bureau of Alcohol, Tobacco, Firearms, and Explosives.

20.2 STATE PERMITTING

The state of Nevada requires a number of permits for mining operations as shown in Table 20.4.

Table 20.4: State Permits for Future Mining

Permit	Agency	Comments
Air Quality Operating Permit	Bureau of Air Quality Control	Will need permits for surface disturbance, crushers, thermal processes and mercury if the mercury in the ore exceeds the de minimis level of 5 pounds per year
Water Pollution Control Permit Modification	BMRR	Covers pits, dumps, heaps, process ponds
Mining Reclamation Permit	BMRR	Addresses reclamation and disturbance
Permit to Appropriate Waters	Nevada Division of Water Resources	Will have to acquire permit to appropriate sufficient water to support heap leaching operation; may be able to acquire water from Barrick
Approval to Operate a Solid Waste System (Class III Waivered Landfill)	Bureau of Waste Management	An on-site landfill will be available to dispose of industrial solid waste
Stormwater NPDES General Permit (SWPPP)	Bureau of Water Pollution Control	Stormwater must be controlled and contained if contacted with process components
Permit to Construct Dam or Impoundment (Dam Safety Permit)	Nevada Division of Water Resources	Will need if a process pond impounds more than 20 acre-feet of water or is more than 20 feet deep
Industrial Artificial Pond Permit	Nevada Department of Wildlife	For any process ponds
Potable Water Permit	Bureau of Safe Drinking Water	A potable water system must be installed for employees; bottled water is not an acceptable alternative
Septic System Permit	Bureau of Water Pollution Control	
Liquefied Petroleum Gas License	NV Board of the Regulation of Liquefied Petroleum Gas	If LPG is stored and used on the site
Hazardous Materials Storage Permit (State Fire Marshal)	State Fire Marshal	

20.2.1 Local Permitting

A special use permit will have to be acquired from Lander County; usually a copy of the Plan of Operations is sufficient information for the County to review and issue a permit.

20.2.1.1 Schedule

The time to acquire permits is expected to be on the order of four years as the MLFO is now requiring a year of baseline data collection prior to submitting the Plan of Operations and Reclamation Permit Application. Much of the data has been collected or will be completed by the fourth quarter of 2011; however, the MLFO may request groundwater sampling to determine the depth and quality of water in the project area.

The EIS process is expected to take on the order of 36 to 48 months based on experience on Nevada over the last 10 years.

20.2.2 Liabilities

Historical Workings

The Robertson Project area has been mined since 1905 with many features left from that time. These artifacts not only include surface disturbance but also old features from processing. Liabilities associated with surface disturbance and feature related primarily costs and delays from having to mitigate site potentially eligible for the NRHP. The processing facilities constructed prior to environmental regulation may be associated with contaminated soils petroleum, mercury; however, signs of petroleum spills were not evident during the site walk on September 17, 2010.

Numerous underground openings can found within the Project area. No liability exists provided Coral does not use these as part of their exploration or mining. If underground openings are used, Coral will be responsible for fencing the openings to prevent inadvertent access.

A review of the Nevada Bureau of Corrective Actions web database did not identify active corrective actions as of April 28, 2011 or closed corrective action cases from 1990 and April 28, 2011.

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Fresh Water

The heap leaching facilities will require at a minimum several hundred gallons per minute of fresh make-up water. The Crescent Valley hydrographic sub-basin is presently over allocated with most of the water allotted for mine dewatering. Acquiring water rights under these circumstances may be difficult; however, Coral may be able to negotiate with Barrick Cortez Mines to purchase excess dewatering water.

Eagle Nest

A golden eagle nest is located in the Gylding Pit; this nest has been known to be active within the last several nesting seasons. Golden eagles have recently come under closer scrutiny by the U.S. Fish and Wildlife Service and subsequently the BLM.

Bald and golden eagles are protected under the *Bald and Golden Eagle Protection Act* (Act) (16 ISC 668-688d). The Act prohibits the taking or possession of and commerce in bald and golden eagles, parts, feathers, nests, or eggs with limited exceptions. The definition of “take” includes pursue, shoot, shoot at, poison, wound, kill, capture, trap, collect, molest, or disturb. “Disturb“ means to agitate or bother a bald or golden eagle to a degree that causes, or is likely to cause, based on the best scientific information available:

- Injury to an eagle;
- A decrease in its productivity by substantially interfering with normal breeding, feeding, or sheltering behavior; or
- Nest abandonment by substantially interfering with normal breeding, feeding, or sheltering behavior.

Further , *Instruction Memorandum 2010-156, Bald and Golden Eagle Protection Act – Golden Eagle National Environmental Policy Act and Avian Protection Plan Guidance for Renewable Energy* has been adopted by the BLM for mining activities on public land. This guidance requires the BLM to consider direct, indirect, and cumulative effects be considered. Best management practices and avian protection plans have to be developed to prevent or reduce impacts to golden eagles. Surveys will have to be conducted prior to ground disturbance in the breeding and nesting seasons to determine the presence or absence of eagles as well as other migratory avian species protected under the Migratory Bird Treaty Act. If nesting or brooding eagles are determined to be present, Coral would have to avoid the area using a buffer zone developed in coordination with the BLM and/or NDOW biologists. Some mines in the area are required to use a ten-mile buffer zone.

The presence of nesting and foraging habitat may limit future exploration and mining during the nesting season. Because the U. S. Fish and Wildlife Service and the BLM are presently determining the scope of the protection measures that will be required, the effect to mining operations cannot be ascertained.

Native American Concerns

The Western Shone Tribe and Great Basin Resource Watch continue to protest against the U.S. government (BLM) and the presence of the Barrick Cortez mines. This dispute is long-standing, but is unlikely to affect Coral's existing and proposed operations. Other nearby mining operations such as Klondex Mines Limited's Fire Creek Mine have not encountered concerns from the Native Americans during the permitting process.

Cultural Resources

The Project area contains many prehistoric and historic sites that are eligible for the NRHP. Ideally, Coral will avoid eligible sites and by doing that, save on time and expense. Sites cannot be mitigated until the NEPA process is complete, and construction will have to wait until the field work is completed and the BLM issues a *Notice to Proceed*. Based on correspondence with Margo Memmott (Kautz Environmental), at least two of these sites are likely to occur within the area of proposed disturbance.

Proximity of Cortez Gold Mines

The Robertson Project area is in close proximity to Barrick Cortez Inc.'s Cortez Gold Mines (Cortez). This operation is one of the largest open pit/underground mine complexes in Nevada. Numerous EIS's have been prepared over time to document baseline conditions and potential impacts from the operations. The proximity to Cortez offers some unique advantages:

- The baseline conditions are well-documented and can be used to supplement future NEPA analysis performed by Coral;
- Social and economic conditions are well known in Crescent Valley as a result of past studies;
- The hydrologic regime is well-studied;
- Cortez must dispose of dewatering water not utilized in their process. Coral may consider approaching Cortez to see about acquiring a portion of the dewatering water;

Claims by Others

Presently, two claim blocks held by others but without use agreements are located adjacent to and overlap into some of the proposed development areas. SRK understands Coral has attempted to negotiate agreements with the owners (Newmont and John Filipini) for use of the claims. Although the developments can be redesigned to avoid these claims, the presence of these claims without a use agreement is likely to cause problems for Coral in the future. The BLM and the BMRR will closely review the original mine plan and subsequent amendments with an eye to ensuring these claims are not affected by Coral's development. The claim owners will have to be notified of each proposed activity and subsequent amendments before the agencies will proceed with approvals

Mercury

The limited MWMP tests conducted did not identify mercury as occurring over the Nevada water quality standards. The Nevada Mercury Control Program (NMCP) is a state regulatory program that requires mercury emissions controls on thermal units located at precious metal mines. The program achieves mercury reduction via add-on control technologies. At this time, the NMCP regulations focus on the potential for mercury emissions from thermal processing units only.

At the core of the NMCP is "NvMACT." NvMACT is the Maximum Achievable Control Technology designated by the NDEP in accordance with NAC 445B.3677. Pursuant to NAC 445B.3625, owners or operators that operate, construct or modify a thermal unit that emits mercury must apply for, and obtain, a mercury operating permit to construct. Construction of a new or modified thermal unit that emits mercury requires a permit before construction begins.

A brief review of some available data indicated mercury appears to occur if fairly low concentrations. A mercury operating permit will not be necessary if Coral can demonstrate that less than five pounds per year of mercury will be processed.

Long Term Trust Fund

The MLFO is presently asking about long term trust funds (LTTF) for mining and exploration projects and implying that an LTTF is necessary. In the past, an LTTF was used to cover uncertainties identified during an EIS process. The MLFO has been considering LTTFs in conjunction with the reclamation bond instead of as a contingency; the reclamation bond already contains a ten percent contingency in the indirect costs. Coral may be asked to provide additional funds in an LTTF for the Robertson Project if uncertainties are identified during the NEPA process.

SECTION 21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

21.1.1 Introduction

The capital costs for the Owner operated alternative have been estimated based upon budget quotes from suppliers, Costmine Mining Cost Data (Infomine USA Ltd.) and in house data. The contractor operated (Alternative 1) capital cost estimate mirrors the Owner operated cost and excludes that mining equipment as provided by the contractor.

21.1.2 Basis of Estimate

The cost of mobile equipment is based upon budget vendor pricing of new equipment for a recent North American project and from in-house files. All other equipment prices have been based on the cost of new equipment

The purchase of trucks has been timed to reflect the requirements throughout the mine life.

The equipment requirements were derived from the proposed mine plan and projected equipment performance criteria. Costs for replacement equipment were included as ongoing capital throughout the mine life.

All costs are expressed in fourth quarter 2011 US dollars, with no allowance for escalation, interest during construction, taxes, or duties. Rates of exchange are stated as CDN$1.00 to US$1.00. A 15% contingency has been added to the estimate to allow for unforeseen capital requirements and overruns.

21.1.3 Cost Summaries

A summary of initial capital expenditures and ongoing capital costs are shown below.

Table 21.1: Initial Capital Expenditures – Base Case (Owner Operator)

Description		2012	2013	2014	2015	2016	2017
		-6	-5	-4	-3	-2	-1
Project Development							
Royalty Payment and Fees		$176	$176	$176	$176	$176	$84
Environmental		$264	$341	$501	$330	$391	
Exploration		$2,817	$1,000	$1,000			
Metallurgical Test Work		$900					
Prefeasibility Study			$1,495				
Definition Drilling				$900			
Metallurgical Test Work				$500			
Full Feasibility					$2,019		
Detail Design						$1,050	
Purchase of Royalty							$1,250
Sub-total		$4,157	$3,012	$3,076	$2,525	$1,616	$1,334
Plant and Surface Facilities							
Crusher and Screening Plant							$11,243
Stacker, Conveyors & Silo							$5,102
Leach Pad and Ponds							$11,058
ARD Plant							$3,881
Yard Facilities							$3,503
Heavy Mobile Equipment							$1,170
Buildings & Structures							$4,151
Fuel supply							$500
Explosive Magazine							$150
Sub-Total Direct							$40,758
Mining							
Equipment							$23,498
Pre-production Development							$7,603
Sub-total							$31,102
Services							
Power Line							$1,000
Power Sub Stations							$1,250
Diesel Generation							$250
Power Distribution							$1,195
Water Distribution							$250
Sewage							$100
Sub-total							$4,045
Total Direct Capital Costs		$4,157	$3,012	$3,076	$2,525	$1,616	$77,238
Indirect Costs							
Engineering and Procurement	7.00%						$5,407
Construction Management	5.00%						$3,862
Construction Indirects	5.00%						$3,862
Freight	2.00%						$1,545
Start-up and Commisioning	1.00%						$772
First Fills and Capital spares	7.00%						$5,407
Sub-total							$22,399
Total without Contigency		$4,157	$3,012	$3,076	$2,525	$1,616	$99,637
Contingency	15.00%	$624	$452	$461	$379	$242	$14,946
Total Initial Capital		$4,780	$3,464	$3,538	$2,903	$1,859	$114,583

Table 21.2: Initial Capital Expenditures - Alternate 1 (Contractor Operator)

Description		2012	2013	2014	2015	2016	2017
		-6	-5	-4	-3	-2	-1
Project Development							
Royalty Payment and Fees		$176	$176	$176	$176	$176	$84
Environmental		$264	$341	$501	$330	$391	
Exploration		$2,817	$1,000	$1,000			
Metallurgical Test Work		$900					
Prefeasibility Study			$1,495				
Definition Drilling				$900			
Metallurgical Test Work				$500			
Full Feasibility					$2,019		
Detail Design						$1,050	
Purchase of Royalty							$1,250
Sub-total		$4,157	$3,012	$3,076	$2,525	$1,616	$1,334
Plant and Surface Facilities							
Crusher and Screening Plant							$11,243
Stacker, Conveyors & Silo							$5,102
Leach Pad and Ponds							$11,058
ARD Plant							$3,881
Yard Facilities							$3,503
Heavy Mobile Equipment							$1,170
Buildings & Structures							$4,151
Fuel supply							$500
Explosive Magazine							$150
Sub-Total Direct							$40,758
Mining							
Equipment							$415
Pre-production Development							$13,744
Sub-total							$14,159
Services							
Power Line							$1,000
Power Sub Stations							$1,250
Diesel Generation							$250
Power Distribution							$1,195
Water Distribution							$250
Sewage							$100
Sub-total							$4,045
Total Direct Capital Costs		$4,157	$3,012	$3,076	$2,525	$1,616	$60,296
Indirect Costs							
Engineering and Procurement	7.00%						$4,221
Construction Management	5.00%						$3,015
Construction Indirects	5.00%						$3,015
Freight	2.00%						$1,206
Start-up and Commisioning	1.00%						$603
First Fills and Capital spares	7.00%						$4,221
Sub-total							$17,486
Total without Contigency		$4,157	$3,012	$3,076	$2,525	$1,616	$77,782
Contingency	15.00%	$624	$452	$461	$379	$242	$11,667
Total Initial Capital		$4,780	$3,464	$3,538	$2,903	$1,859	$89,449

21.1.4 Environmental

The environmental costs are estimated as $1.83 million and covers all the environmental and associated costs to a construction decision. The summary of these costs are shown in Section 26, Recommendations.

21.1.5 Exploration

Exploration and definition costs for the period to prefeasibility study is estimated at $2.817 million and the cost summary is shown in Section 26, Recommendations. In addition an allowance of $2.9 million has been included to cover additional drilling and associated activities to provide information for a full feasibility study.

21.1.6 Metallurgical Test Work

The estimated cost for the metallurgical test work is $900,000. A description of the work together with a cost summary is shown in Section 26, Recommendations. An allowance of $500,000 has been included to cover additional metallurgical test work for full feasibility.

21.1.7 Prefeasibility and Full Feasibility Studies

It is estimated that a prefeasibility study will cost an additional $1.5 million and a full feasibility some $2.019 million. The full feasibility costs will be further defined after completion of the prefeasibility study.

21.1.8 Detail Design

Detail design upon completion of full feasibility has been included at $1.05 million.

21.1.9 Purchase of Royalty

The only applicable Royalty has a buyout of $1.25 million. It is expected that this would occur upon a construction decision being made and is included in the year that this decision would be taken.

21.1.10 Crusher and Screening Facilities

The estimated cost for these facilities is $11.243 million. The summary of the cost are shown on Table 21.3. It is expected that 7500 tpd will be crushed.

Table 21.3: Summary of Estimated Cost Crushing and Associated Facilities

Description	Estimated Cost
Jaw Crusher	$1,548,000
Cone Crusher	$1,816,000
Double & Single Deck Screens	$257,000
Conveyors/Samplers/Scales	$2,353,000
Installation	$2,413,000
Piping	$465,000
Instrumentation	$698,000
Auxiliaries	$931,000
Outside Piping	$762,000
Total	**$11,243,000**

21.1.11 Stacker, Conveyors and Silo

This has been estimated at $5.102 million. This cost was based providing two systems one for the HG leach pad and for the LG.

21.1.12 Leach Facilities

Preliminary capital cost items and estimated quantities have been developed for the proposed leach pad, events pond and solution recovery components of the facility for pre-production infrastructure and the 78 Mt mine configuration. A detailed summary of cost items and associated quantities have been summarized in Table 21.4.

Initial capital costs include the foundation works, waste dump construction and ancillary works required for the leach pads, solution ponds and solution recovery systems.

Table 21.4: Preliminary Capital Cost Estimate Total Construction Costs

Area	Type	Units	Quantity	Quantity (Rounded)	Unit Cost (US$)	Total Cost (US$)
Low Grade HLF						
101	Topsoil stripping - to stockpile	cy	302,407	302,400	4.50	1,361,000
102	Grade preparation for pads	cy	907,222	907,200	1.70	1,542,000
103	Secondary GCL	yd^2	816,500	816,500	4.80	3,919,000
104	Primary liner (80 mil textured HDPE)	yd^2	816,500	816,500	1.50	1,225,000
105	Overliner	cy	816,500	816,500	4.10	3,348,000
106	Solution piping (4" perforated corrugated plastic tubing)	ft	146,970	147,000	1.60	235,000
107	Solution piping (12" HDPE)	ft	14,697	14,700	11.20	165,000
108	Solution piping (20" HDPE)	ft	1,470	1,500	28.50	42,000
High Grade HLF						
201	Topsoil stripping - to stockpile	cy	285,519	285,500	4.50	1,285,000
202	Grade preparation for pads	cy	856,556	856,600	1.70	1,456,000
203	Secondary GCL	yd^2	770,900	770,900	4.80	3,700,000
204	Primary liner (80 mil textured HDPE)	yd^2	770,900	770,900	1.50	1,156,000
205	Overliner	cy	770,900	770,900	4.10	3,161,000
206	Solution piping (4" perforated corrugated plastic tubing)	ft	138,762	138,800	1.60	222,000
207	Solution piping (12" HDPE)	ft	13,876	13,900	11.20	155,000
208	Solution piping (20" HDPE)	ft	1,388	1,400	28.50	40,000
Waste Rock Dump - CAPEX						
301	Topsoil stripping - to stockpile	cy	149,704	149,700	4.50	674,000
302	Foundation preparation	cy	149,704	149,700	1.70	254,000
303	Seepage and drainage collection	LS	1	1	200,000	200,000
Crusher Site						
401	Topsoil stripping - to stockpile	cy	1,667	1,667	4.50	8,000
402	Cut	cy	58,333	58,300	6.50	379,000
403	Excess cut - load and haul	cy	52,500	52,500	1.20	63,000
404	Foundation preparation	ft^2	27,000	27,000	1.70	46,000
ADR Plant						
501	Topsoil stripping - to stockpile	cy	27,000	27,000	4.50	122,000
502	Cut	cy	4,500	4,500	6.50	29,000
503	Foundation preparation	cy	750	800	1.70	1,000
Pregant Solution Pond						
601	Topsoil stripping - to stockpile	cy	1,911	1,900	4.50	9,000
602	Cut to fill volume	cy	11,148	11,100	6.50	72,000
603	Interliner gravel	ft^2	38,700	38,700	1.40	54,000
604	Secondary liner (60 mil smooth HDPE)	ft^2	38,700	38,700	1.30	50,000
605	Geonet	ft^2	38,700	38,700	1.20	46,000
606	LRDS piping (2")	ft	774	800	1.60	1,000
607	Primary liner (60 mil textured HDPE)	ft^2	38,700	38,700	1.30	50,000
Barren Solution Pond						
701	Topsoil stripping - to stockpile	cy	1,911	1,900	4.50	9,000
702	Cut to fill volume	cy	11,148	11,100	6.50	72,000
703	Interliner gravel	ft^2	38,700	38,700	1.40	54,000
704	Secondary liner (60 mil smooth HDPE)	ft^2	38,700	38,700	1.30	50,000
705	Geonet	ft^2	38,700	38,700	1.20	46,000
706	LRDS piping (2")	ft	774	800	1.60	1,000
707	Primary liner (60 mil textured HDPE)	ft^2	38,700	38,700	1.30	50,000
Event Pond						
801	Topsoil stripping - to stockpile	cy	1,911	1,900	4.50	9,000
802	Cut to fill volume	cy	11,148	11,100	6.50	72,000
803	Interliner gravel	ft^2	38,700	38,700	1.40	54,000
804	Secondary liner (60 mil smooth HDPE)	ft^2	38,700	38,700	1.30	50,000
805	Geonet	ft^2	38,700	38,700	1.20	46,000
806	LRDS piping (2")	ft	774	800	1.60	1,000
807	Primary liner (60 mil textured HDPE)	ft^2	38,700	38,700	1.30	50,000
Solution Transfer Ditches						
901	Topsoil stripping - to stockpile	ft^2	16,600	16,600	4.50	75,000
902	Cut to fill volume	cy	15,370	15,400	6.50	100,000
903	Channel lining (80 mil textured HDPE)	ft^2	70,428	70,400	1.50	106,000
Haul Roads						
1001	Construction (roads, ditches, etc)	mile	2.3	2.3	805,000	1,868,000
	Total					**27,783,000**

21.1.13 ARD Plant and Associated Facilities

The summary of the estimated cost for ARD plant and associated facilities is shown on Table 21.5. These costs are based upon the Costmine data for this type of plant.

Table 21.5: Summary of Estimated Cost ARD Plant and Associated Facilities

Description	Estimated Cost
Carbon Colums	$407,000
Stripping Circuit	$87,000
Electro-Winning Cell	$47,000
Furnace	$73,000
Acid Wash (stainless Steel)	$139,000
Carbon Conditioner	$42,000
Carbon Regeneration Kiln	$308,000
Installation	$496,000
Piping	$399,000
Instrumentation	$240,000
Building	$420,000
Auxilairies	$240,000
Outside Piping	$319,000
Laboratory	$664,000
Total	**$3,881,000**

21.1.14 Yard Facilities and Mobile Equipment

Yard facilities were estimated at $3,503 million with heavy mobile equipment of $1.17 million. These estimates were based upon the costs shown in the Infomine cost data.

21.1.15 Buildings and Structures, Fuel Supply and Explosives Magazine

The cost for the offices warehouse, maintenance facilities, dry, security and miscellaneous buildings has been estimated at $4.141 million based upon the estimated size of the facilities and unit cost per sq ft. An allowance of $500,000 has been made for the fuel supply facilities and $150,000 for the magazine based upon in house documentation.

21.1.16 Mining Equipment

Mining equipment costs is estimated at $23.5 million and is based upon the pieces of equipment required to mine the mineral and waste and budget costs from the suppliers. The equipment list and cost is shown below.

Table 21.6: Mine Equipment

Description	# Req'd	Cost/unit	Total cost
Atlas Copco Viper Drill	2	$1,420,000	$2,840,000
Komatsu PC1800 Shovel	1	$2,453,706	$2,453,706
Komatsu WA90	1	$1,773,796	$1,773,796
Komatsu HD785-7	8	$1,269,147	$10,153,176
Komatsu HD155AX-6	2	$443,000	$886,000
Komatsu GD655	1	$253,908	$253,908
Komatsu 600-6	1	$635,841	$635,841
Air trac/compressor	1	$661,000	$661,000
Komatsu HD325-7	2	$583,197	$1,166,394
Service truck	2	$90,000	$180,000
Crane truck	1	$200,000	$200,000
Blaster's truck	2	$350,000	$700,000
Light plant	4	$20,000	$80,000
Pick ups	5	$35,000	$175,000
Pump truck	1	$30,000	$30,000
Misc. Shop Equipt (lot)	1	$200,000	$200,000
Spares @ 5% (lot)	1	$1,109,441	$1,109,441
Total			**$23,498,262**

21.1.17 Pre-production Development

The pre-production costs are based upon grubbing, overburden stripping and surface preparation for the Altenburg Hill and Porphyry and waste mining for the Porphyry. The Altenburg Hill deposit is a hill thus no waste mining is required prior to mining the mineral.

The estimated cost for this is $7.6 million.

21.1.18 Electrical Capital Cost

Table 21.7 is a summary of the electrical costs for NV Energy's capital charges, Robertson's site distribution and all site services.

Capital costs are listed below in Table 21.7.

Table 21.7: Electrical Capital

Description		$
NV Energy Costs		
	120kV tap (allowance)	$200,000
	New 3.0 MVA Substation 120-25kV (allowance)	$500,000
	New 2 mile length 25kV powerline (allowance)	$300,000
	Subtotal	***$1,000,000***
Coral Gold Resources Ltd. Costs		
	Main Site Substation 25-4.16kV	$500,000
	Main switchgear line-up 4.16kV. 10 cells @ $50,000	$500,000
	2.0 MW Diesel Emergency Generator	$250,000
	4.16kV Distribution	
	Buried Cable 500ft @ $50/ft	$25,000
	Overhead powerline 3 miles @ $50,000/mile	$150,000
	Unit Substations (allowance)	
	Offices, warehouse, dry	$50,000
	Fresh water pumping	$50,000
	ADR Plant	$50,000
	Pregnant Solution Pond	$50,000
	Crushers	$50,000
	Truck Shop	$50,000
	Barren Solution Pond	$50,000
	4.16kV Cable (allowance)	$50,000
	4.16kV Motor Starters. 4 @ $10,000/each (allowance)	$40,000
	480V Feeders (allowance)	$75,000
	480V MCCs - 50 process, 20 bldg, and 20 feeders (allo	$100,000
	480V Cable (allowance)	$100,000
	Control cable (allowance)	$50,000
	SCADA Cable (allowance)	$50,000
	Building Services: lighting, heating, ventilation, small po	$750,000
	Control Hardware (devices, PLC, computers)	$100,000
	Control Software (devices, PLC, computers)	$20,000
	Fire Protection (allowance)	$50,000
	Communication (allowance)	$50,000
	Security (allowance)	$25,000
	Civil Works	$100,000
	Installation Labour	$100,000
	six men - 1000 hr each @ $60/hr	$360,000
Total Capital Cost		***$3,295,000***

21.1.19 Water Distribution and Sewage

An allowance for these items up and above those costs included in other areas is $350,000.

21.1.20 Additional Costs

The indirect costs have been estimated as a percentage of the capital cost and are shown on the Table 21.1. The percentages reflect industry standard.

21.1.21 Ongoing Costs

The ongoing costs for the two alternatives are shown below. The Owner operated approach requires replacement equipment throughout the life of the mine while these costs are included in the operating cost provided by the contactor. The leach pad and associated costs are the same for both cases and the sustaining capital items include the heap leach pads, the waste rock dump, solution transfer ditches and haul road construction.

Other costs associated with the heap leach facilities include operational costs and capital costs for leaching services. Table 21.10 shows these capital requirements by year throughout operations (11 years).

Table 21.8: Ongoing Capital Costs – Base Case (Owner Operator)

		Year												
Description	**%**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**Total**
Leach Pads			$3,122	$3,122	$3,122	$2,943	$2,943	$1,472						**$16,724**
Replacement vehicles	5.00		$1,175	$1,175	$1,175	$1,175	$16,175	$1,175	$1,175	$1,175	$1,175			**$25,579**
Reclamation													$7,000	**$7,000**
Contingency	10.00		$430	$430	$430	$412	$1,912	$265	$117	$117	$117		$700	**$4,930**
Total			**$4,727**	**$4,727**	**$4,727**	**$4,530**	**$21,030**	**$2,912**	**$1,292**	**$1,292**	**$1,292**		**$7,700**	**$54,233**

Table 21.9: Ongoing Capital Costs Alternate 1 (Contractor Operator)

		Year												
Description	**%**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**Total**
Reclamation													$7,000	**$7,000**
Contingency	10.00		$312	$312	$312	$294	$294	$147					$700	**$2,372**
Total			**$3,434**	**$3,434**	**$3,434**	**$3,237**	**$3,237**	**$1,619**					**$7,700**	**$26,096**

Table 21.10: Preliminary Sustaining Capital Cost - Surface Facilities

	Area	Year 2	Year 4	Year 6	Year 8	Year 10	Year 11	Total Cost (US$)
100	Low Grade HLF	1,434,788	1,434,788	1,434,788	1,434,788	1,434,788	717,394	11,837,000
200	High Grade HLF	1,354,545	1,354,545	1,354,545	1,354,545	1,354,545	677,273	11,175,000
300	Waste Rock Dump	153,818	153,818	153,818	153,818	153,818	76,909	1,128,000
400	Crusher Site							496,000
500	ADR Plant							152,000
600	Pregant Solution Pond							282,000
700	Barren Solution Pond							282,000
800	Event Pond							282,000
900	Solution Transfer Ditches	23,417	23,417	23,417				281,000
1,000	Haul Roads	155,667	155,667	155,667				1,868,000
	SUBTOTAL	3,122,000	3,122,000	3,122,000	2,943,000	2,943,000	1,472,000	16,724,000
	TOTAL							16,724,000

21.2 OPERATING COSTS

21.2.1 Basis of Estimate

21.2.1.1 Mining Base Case

Project operating costs were estimated based on an overall average production rate of 20,000 tpd mineral and an average stripping ratio of 0.6:1 (i.e., a total average daily production of approximately 32,000 tons of material) which includes the low grade (0.005 to 0.0147 ozAu/t) leached on a separate pad. An operating schedule of 2 x 10 hr shifts per day for 350 days per year was incorporated into the mine plan based upon a seven-day week. Statutory holidays were not worked (52 weeks @ 7 days per week is 365 days, less 15 for allowance for all statutory holidays, 350 days).

The operating cost estimate is based on local labour rates and productivity factors, as well as local material and supply costs where applicable for Nevada. The labour rates obtained covered a broad range of job categories for both hourly rated and staff personnel and included all allowances for shift premiums, statutory social benefit premiums, vacations, medical benefits, etc. The costs for some bulk supply and consumable items, such as fuel and explosives, were obtained from local sources, but items such as drilling supplies, tires, and equipment maintenance costs were estimated from typical North American cost criteria.

21.2.1.2 Mining Alternative 1

- The open it operating cost are based on the following;
- Production is based on 5 days/week, one and then 2 shifts per day and from 8 hours per shift up to 10 hrs per shift.

- All labor for open pit drilling, blasting, loading, and hauling to the crusher/stockpile/waste dump is included.
- All equipment for open pit drilling, blasting, loading, and hauling to the crusher/stockpile/waste dump is included. An estimate for the initial construction of haul roads is included.
- An estimate for topsoil removal and stockpiling in the pit and waste dump areas is included.
- All bulk explosives, explosives accessories, drilling, and drill accessories are included.
- All management and supervisory costs are included.
- Transportation of hourly employees to and from the worksite is included.
- The capital costs for a maintenance facility, an equipment wash facility, and a fuel storage facility are not included in the estimate per your instructions, however; the cost to operate and maintain the maintenance facilities are included in the estimated unit costs.
- The estimated unit costs include diesel fuel at a rate of $3.80 per gallon. The total of road diesel fuel cost is estimated to be 14,270,900 gallons at a total cost of $54,494,742. The estimated fuel consumption report is attached.
- No blast hole or pit dewatering is included in the estimate.
- No crushing, stockpiling, or stockpile re-handling costs are included.

The estimated cost summary by year is shown on Tables 21.11 and 21.12 below.

21.2.1.3 Cost Summaries

Table 21.11: Operating Cost Summary $(000)'s Base Case. (Owner Operator)

Description	Cost/t	1	2	3	4	5	6	7	8	9	10	11	12	Total
Processed Ore (000)'s Ton		3,880	6,977	7,151	8,886	8,933	9,156	8,993	9,144	6,368	6,824	1,896		**78,209**
Mining Cost	$2.52	$13,988	$20,291	$20,223	$21,426	$21,688	$21,685	$20,335	$20,873	$16,786	$15,402	$4,264		**$196,961**
Plant HG	$2.93	$1,287	$4,207	$7,873	$6,442	$8,428	$8,507	$10,338	$13,465	$7,899	$7,314	$1,910		**$77,671**
Plant LG	$1.73	$5,953	$9,585	$7,723	$11,570	$10,477	$10,817	$9,454	$7,869	$6,354	$7,487	$2,152		**$89,441**
Power	$0.17	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210		**$13,310**
G/A	$0.38	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707		**$29,777**
Owners Cost	$0.07	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500		**$5,500**
Total	$5.28	$25,644	$38,500	$40,236	$43,854	$45,011	$45,426	$44,543	$46,625	$35,455	$34,621	$12,744		**$412,660**
Yearly Cost/ton		**$6.61**	**$5.52**	**$5.63**	**$4.94**	**$5.04**	**$4.96**	**$4.95**	**$5.10**	**$5.57**	**$5.07**	**$6.72**		**$5.28**

Table 21.12: Operating Cost Summary $(000)'s Alternative 1 (Contractor Operator)

Description	Cost/t	1	2	3	4	5	6	7	8	9	10	11	12	Total
Processed Ore (000)'s Ton		3,880	6,977	7,151	8,886	8,933	9,156	8,993	9,144	6,368	6,824	1,896		**78,209**
Mining Cost	$3.70	$22,085	$27,061	$28,169	$28,961	$35,883	$35,724	$32,101	$30,039	$22,966	$21,065	$5,075		**$289,127**
Plant HG	$2.93	$1,287	$4,207	$7,873	$6,442	$8,428	$8,507	$10,338	$13,465	$7,899	$7,314	$1,910		**$77,671**
Plant LG	$1.73	$5,953	$9,585	$7,723	$11,570	$10,477	$10,817	$9,454	$7,869	$6,354	$7,487	$2,152		**$89,441**
Power	$0.17	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210	$1,210		**$13,310**
G/A	$0.38	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707	$2,707		**$29,777**
Owners Cost	$0.07	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500	$500		**$5,500**
Total	$6.45	$33,741	$45,271	$48,182	$51,389	$59,206	$59,465	$56,309	$55,790	$41,635	$40,283	$13,554		**$504,826**
Yearly Cost/ton		**$8.70**	**$6.49**	**$6.74**	**$5.78**	**$6.63**	**$6.49**	**$6.26**	**$6.10**	**$6.54**	**$5.90**	**$7.15**		**$6.45**

21.3 CRUSHING AND ARD PLANT

The process costs are based upon the crushing plant to crush at 7500tpd the HG ore and the ARD plant to process leachate for both the HG and LG leach pads.

The operating cost estimate for the crushing facilities and ARD plant have been based upon the costs established in Costmine documentation derived by Infomine.

These costs are indicative of the costs for projects similar to the conditions in northern Nevada, USA. The operating cost shown on Tables 21.13 reflects the cost to process 7500tpd which is the crushed HG mineralization and Table 21.14 the estimated cost to process 12500tpd of run-off-mine material.

Table 21.13: Summary Operating Cost 7500tpd

Description	Estimated Cost $/t Processed
Operating Labour	$1.00
Reagents	$1.01
Repair and Maintenance Supplies	$0.48
Wear Items	$0.09
Heavy Mobile Equipment Operation	$0.03
Staff / Supervision	$0.32
Total	**$2.93**

Table 21.14: Summary Operating Cost 12500tpd

Description	Estimated Cost $/t Processed
Operating Labour	$0.49
Reagents	$0.88
Repair and Maintenance Supplies	$0.15
Wear Items	$0.02
Heavy Mobile Equipment Operation	$0.03
Staff / Supervision	$0.16
Total	**$1.73**

21.3.1 Electrical

Project electrical operating costs are shown on Table 21.15 consist of the cost of power. Personnel, parts and consumables for ongoing repairs and maintenance to the electrical installation are included elsewhere. The estimated cost of power, based on NV Energy's current tariff schedule and the unit cost is 8 US cents/kWh.

Table 21.15: Annual Electrical Power Costs

Description		$
Annual NV Energy Tariff Charges		
	14GWhr per year @ $0.08 per kWhr	$1,120,000
Total Annual Electrical Tariff Cost		***$1,120,000***

21.3.1.1 Overheads (G&A) and Owner's Costs

The overhead operations costs were estimated based on the administrative labour expected for this size of operation, miscellaneous surface operations personnel, and expenses such as communication, insurance, and supplies that are general and not included in the mine or process plant costs.

Owner's costs were estimated to be at $500,000 per year for off-site costs.

21.3.1.2 Other Costs

Annual federal claim fees of US$84,140, payable to the BLM, and Notice of Intent to Hold Mining Claims of 6,310 have been filed for the 2011-2012 assessment year.

There were no other costs.

21.3.1.3 Royalties

Total annual payments for the various leases and minimum advance royalties or rental payments are US$91,700. A summary compilation of the terms of these agreements are presented in Table 4.1.

SECTION 22.0 ECONOMIC ANALYSIS

22.1 FINANCIAL ANALYSIS

22.1.1 Financial Analysis Method

The financial analysis has been prepared using standard discounted cash flow methods to determine the NPV and IRR of the project based upon 100% equity financing and metal price of US$1350/oz Au. This gold price is some 8% greater than the trailing three year average and reflects most analysts opinion of higher gold prices Than the trailing price would suggest. The analysis was performed in constant 2011 US dollars, excluding inflation. Various sensitivities have been completed to determine the effect of metal prices, capital costs, operating costs and leach recovery.

22.1.2 Results Base Case

Case 1 (Base Case) economic evaluation is are based on an initial construction capital expenditure of US$122.1 million operating cost of $5.28/ton mined and metal prices of US$1350/oz[6]. The project will generate an after-tax IRR of 15.44% and an NPV of US$180.6 million undiscounted and US$97 million discounted at 5%. Payback of initial capital can be achieved in 5.91 years. The inferred resources established in this study are 78.2 million tons grading 0.0138 oz/t of gold.

Case 1 summary is shown on Table 22.1.

Case 5 reflects a gold price of US$1500/oz. Under this scenario the project will generate an after-tax IRR of 20.13% and a NPV of US$247.2 million undiscounted and US$147.1 million discounted at 5%. Payback of initial capital can be achieved in 4.72 years.

Case 7 reflects a gold price of US$1750/oz. Under this scenario the project will generate an after-tax IRR of 27.24% and a NPV of US$358.3 million undiscounted and US$230.7 million discounted at 5%. Payback of initial capital can be achieved in 3.91 years.

Sensitivity studies have been prepared varying the price of gold, the operating and capital cost, grade and leach recovery. The project most sensitive to gold price, sensitive to operating cost, grade and recovery and least sensitive to capital cost. The sensitivity results are shown on Table 22.2.

Graphs have been prepared showing the sensitivities and are included in this section.

It should be noted that the financial analysis has been evaluated on the basis of the construction activities being set against proceeds and those costs for the period to a construction decision have been included as sunk costs although these development costs have not been incurred at the time of writing this report. This is considered as an acceptable approach since any decision to construct will be based upon the financial analysis at the time following completion of the development activities. Thus the expenditures for construction only are set against the proceeds of the project and not the development costs. To include the development costs in the capital costs will not provide a basis for a fair evaluation of the project.

[6] It should be noted that the authors of this report have not and do not forecast the future price of gold. The alternatives shown in this report are intended to indicate the variation of the project financials based upon sensitivity criteria shown.

Table 22.1: Financial Analysis (Owner Operator)

ROBERTSON PROJECT

ALTENBURG, PORPHYRY AND GOLD PAN DEPOSITS

Financial Analysis Base Case

$(000)s

Resource Ton (000)'s **78,209**

YEAR

Description	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	Total
Production Ton/year HG				439	1,436	2,687	2,198	2,877	2,903	3,528	4,596	2,696	2,496	652		**26,509**
Au Grade oz/t				0.017	0.019	0.021	0.019	0.019	0.019	0.020	0.023	0.022	0.024	0.021		**0.021**
Recovery Au %				67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%		**67.00%**
Production Ton/year LG				3441	5541	4464	6688	6056	6253	5465	4549	3673	4328	1244		**51,700**
Au Grade oz/t				0.010	0.010	0.010	0.010	0.011	0.010	0.011	0.010	0.010	0.010	0.010		**0.010**
Recovery Au %				45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%		**45.00%**
Au ounces payable				20,277	42,619	57,438	58,180	65,106	66,406	73,156	90,776	55,848	59,364	14,942		**604,111**
Gross Revenue				$ 27,350	$ 57,487	$ 77,476	$ 78,477	$ 87,819	$ 89,572	$ 98,678	$ 122,444	$ 75,332	$ 80,074	$ 20,155		**342,699**
Operating Costs				$(25,644)	$(38,500)	$(40,236)	$(43,854)	$(45,011)	$(45,426)	$(44,543)	$ (46,625)	$(35,455)	$(34,621)	$(12,744)		**($412,660)**
Income Tax				$ (29)	$ (315)	$ (1,115)	$ (1,822)	$ (3,409)	$ (4,622)	$ (5,857)	$ (20,477)	$ (9,084)	$(11,167)	$ (1,608)		**($59,506)**
Revenue Before Capital Exp.				$1,677	$18,672	$36,125	$32,800	$39,399	$39,525	$48,278	$55,342	$30,793	$34,286	$5,803		$342,699
Capital Expenditures																
- Development/Construction			($114,583)													**($114,583)**
- On-Going Capital					($4,727)	($4,727)	($4,727)	($4,530)	($21,030)	($2,912)	($1,292)	($1,292)	($1,292)			**($46,528)**
Working Capital Change				($3,039)	($1,697)									$4,735		
Reclamation Bond			($7,500)												$7,500	
Reclamation															($7,700)	**($7,700)**
Salvage															$6,750	**$6,750**
Total Capital			($122,083)	($3,039)	($6,423)	($4,727)	($4,727)	($4,530)	($21,030)	($2,912)	($1,292)	($1,292)	($1,292)	$4,735	$6,550	($162,061)
Net Cashflow			($122,083)	($1,362)	$12,249	$31,399	$28,074	$34,869	$18,495	$45,366	$54,050	$29,500	$32,993	$10,538	$6,550	**$180,638**
Discounted NCF 5%			**($122,083)**	**($1,297)**	**$11,110**	**$27,123**	**$23,096**	**$27,321**	**$13,801**	**$32,241**	**$36,583**	**$19,016**	**$20,255**	**$6,161**	**$3,647**	**$96,976**
Discounted NCF 8%			**($122,083)**	**($1,261)**	**$10,501**	**$24,925**	**$20,635**	**$23,732**	**$11,655**	**$26,471**	**$29,201**	**$14,758**	**$15,282**	**$4,520**	**$2,601**	**$60,937**
Discounted NCF 10%			**($122,083)**	**($1,238)**	**$10,123**	**$23,590**	**$19,175**	**$21,651**	**$10,440**	**$23,280**	**$25,215**	**$12,511**	**$12,720**	**$3,694**	**$2,087**	**$41,164**

Rate of Return **15.44%**

Notes:

1. Metal Prices US $ Au/oz 1350.00

2. Capital requirements based on 100% equity.

3. All funds are in US$ except where noted.

4. Taxes are approximate.

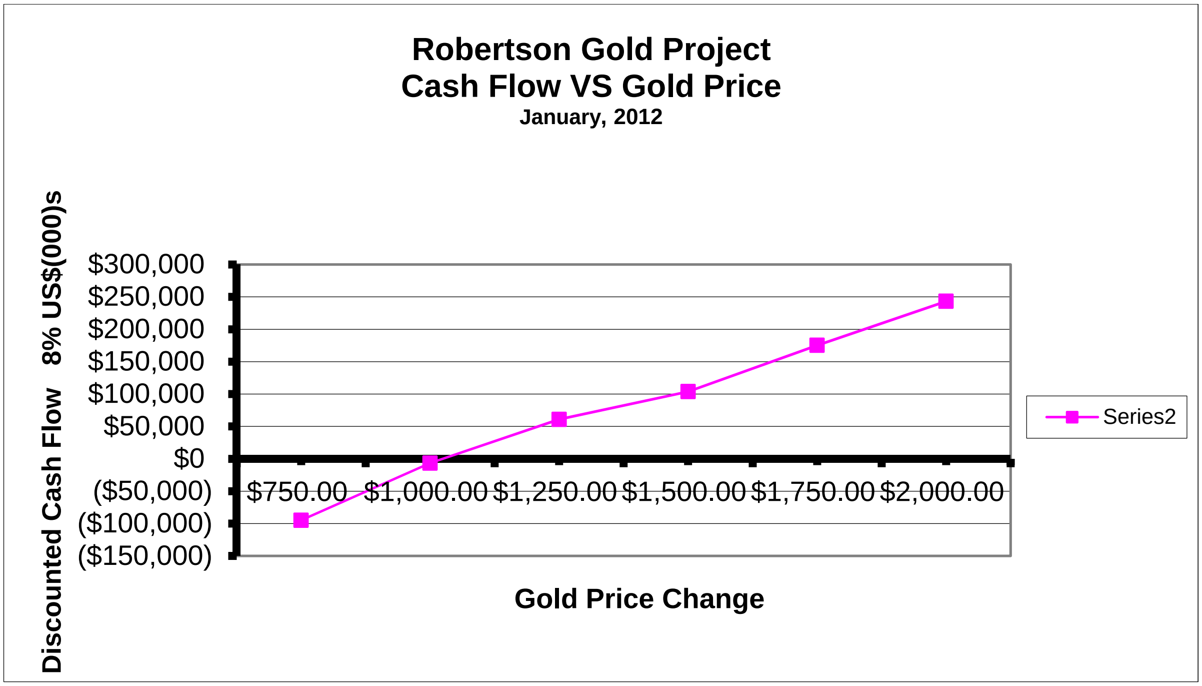
Payback **5.91 years**

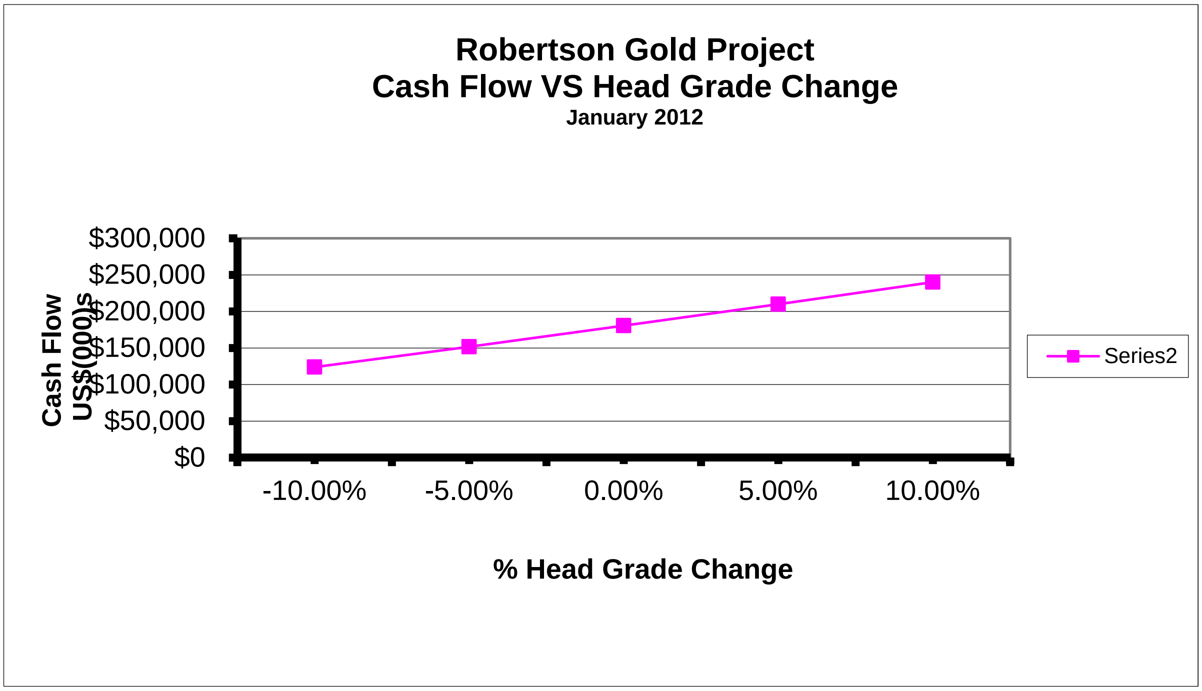
Au first full production year **20,277 ozs**

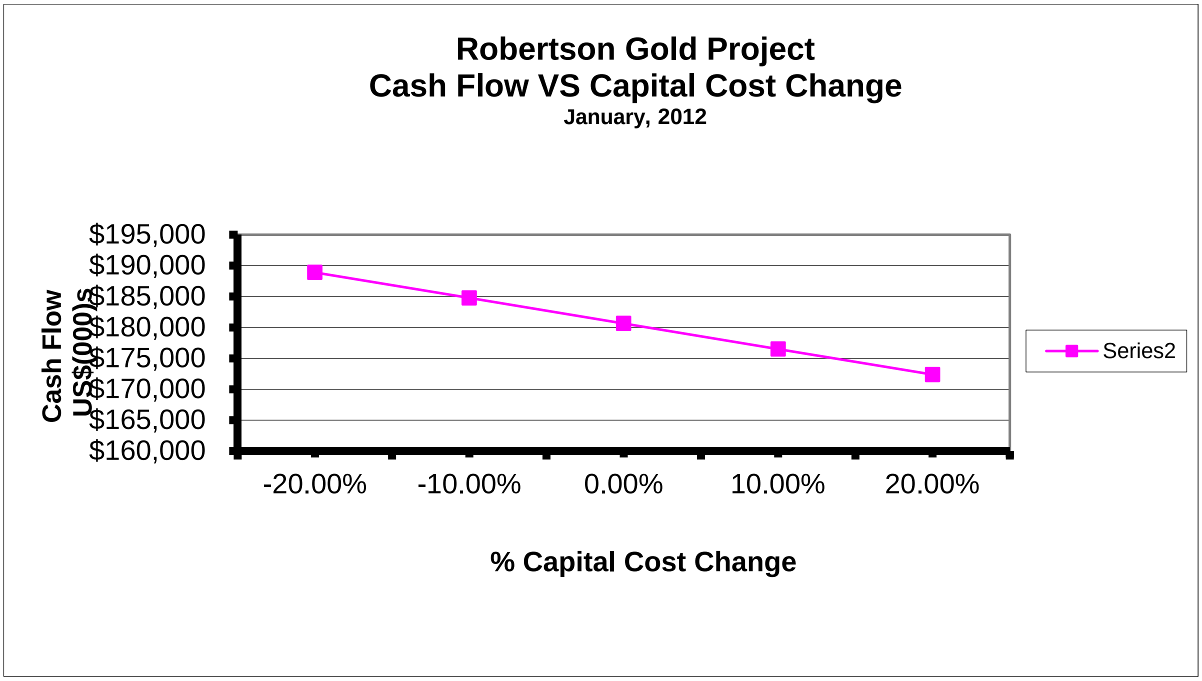
Average NSR/ton **$10.42**

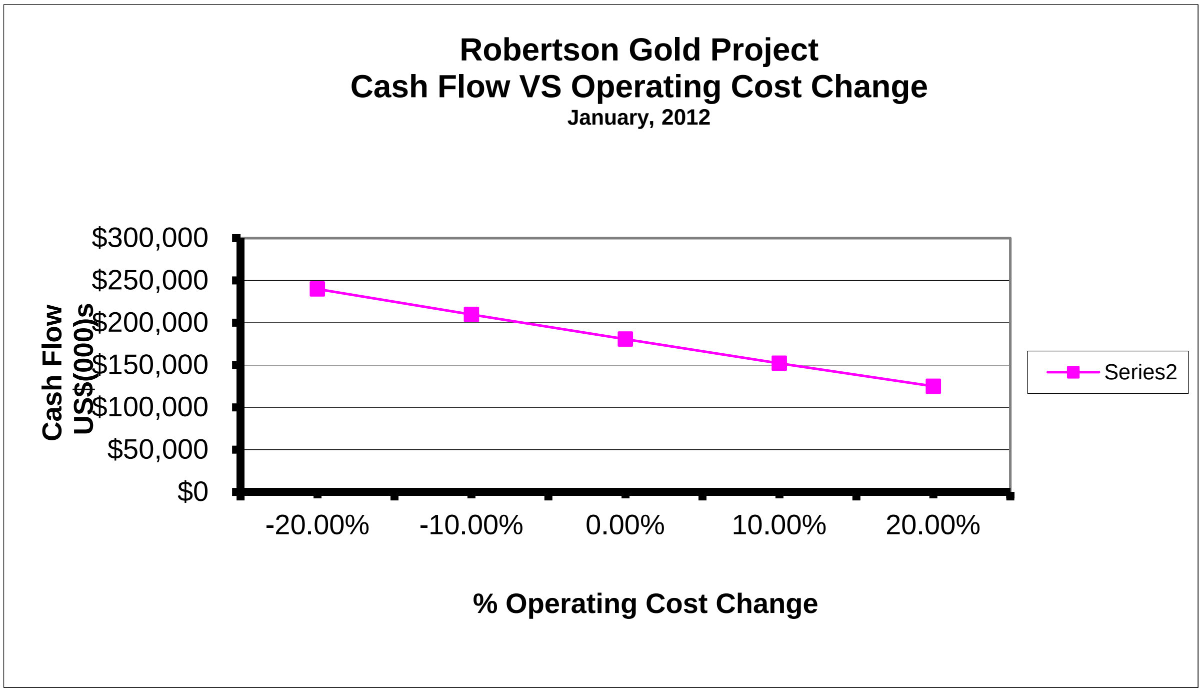
Table 22.2: Sensitivities (Owner Operator)

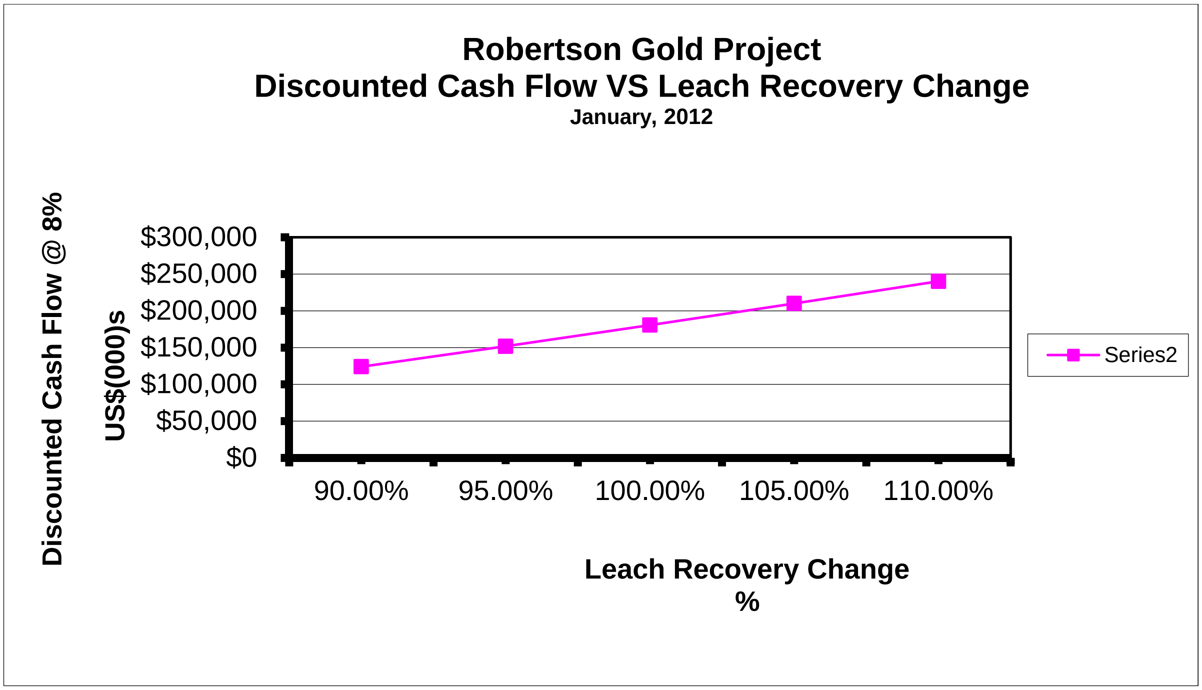
Case	Description of Sensitivity	NPV Dis.0%	NPV Dis.5%	NPV Dis.8%	IRR
		US$(000)s	US$(000)s	US$(000)s	%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
Case 2	Gold Price $850/oz	($57,729)	($84,166)	($94,825)	-6.12%
CASE3	Gold Price $1100/oz	$79,451	$19,046	($6,612)	7.15%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE5	Gold Price $1500/oz	$247,156	$147,053	$103,770	20.13%
CASE6	Gold Price $1750/oz	$358,295	$230,661	$175,241	27.24%
CASE7	Gold Price $2000/oz	$464,197	$310,181	$243,159	33.41%
CASE8	Grade -10%	$124,051	$54,202	$24,232	11.07%
CASE9	Grade -5%	$151,823	$75,315	$42,414	13.29%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE10	Grade +5%	$209,974	$119,021	$79,779	17.55%
CASE11	Grade +10%	$240,183	$141,688	$99,127	19.62%
CASE12	Capital Cost -20%	$188,904	$103,569	$66,733	16.13%
CASE13	Capital Cost -10%	$184,771	$100,272	$63,835	15.79%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE14	Capital Cost +10%	$176,505	$93,679	$58,039	15.09%
CASE15	Capital Cost +20%	$172,372	$90,382	$55,141	14.75%
CASE16	Operating Cost -20%	$239,844	$142,662	$100,598	20.04%
CASE17	Operating Cost -10%	$209,805	$119,521	$80,533	17.74%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE18	Operating Cost +10%	$152,031	$74,793	$41,610	13.12%
CASE19	Operating Cost +20%	$124,961	$53,382	$22,744	10.80%
CASE20	Leach Recovery +10%	$240,183	$141,688	$99,127	19.62%
CASE 21	Leach Recovery +5%	$209,974	$119,021	$79,779	17.55%
CASE 1	**Base Case**	**$180,638**	**$96,976**	**$60,937**	**15.44%**
CASE 22	Leach Recovery -5%	$151,823	$75,315	$42,414	13.29%
CASE23	Leach Recovery -10%	$124,051	$54,202	$24,232	11.07%


Robertson Gold Project
Cash Flow VS Gold Price
January, 2012
Discounted Cash Flow 8% US$(000)s
$300,000
$250,000
$200,000
$150,000
$100,000
$50,000
$0
($50,000)
($100,000)
($150,000)
$750.00
$1,000.00
$1,250.00
$1,500.00
$1,750.00
$2,000.00
Series2
Gold Price Change


Robertson Gold Project
Cash Flow VS Head Grade Change
January 2012
Cash Flow US$(000)s
$300,000
$250,000
$200,000
$150,000
$100,000
$50,000
$0
-10.00%
-5.00%
0.00%
5.00%
10.00%
Series2
% Head Grade Change


Robertson Gold Project
Cash Flow VS Capital Cost Change
January, 2012
Cash Flow US$(000)s
$195,000
$190,000
$185,000
$180,000
$175,000
$170,000
$165,000
$160,000
-20.00%
-10.00%
0.00%
10.00%
20.00%
% Capital Cost Change
Series2


Robertson Gold Project
Cash Flow VS Operating Cost Change
January, 2012
Cash Flow US$(000)s
$300,000
$250,000
$200,000
$150,000
$100,000
$50,000
$0
-20.00%
-10.00%
0.00%
10.00%
20.00%
% Operating Cost Change
Series2



22.1.3 Results Alternative Case A1

Case A1 economic evaluation is are based on an initial construction capital expenditure of US$97 million operating cost of $6.45/ton mined and metal prices of US$1350/oz. The project will generate an after-tax IRR of 15.43% and an NPV of US$159.4 million undiscounted and US$85.2 million discounted at 5%. Payback of initial capital can be achieved in 5.94 years.

Case A1 summary is shown on Table 22.3.

Case A5 reflects a gold price of US$1500/oz. Under this scenario the project will generate an after-tax IRR of 20.96% and an NPV of US$226.4 million undiscounted and US$135.9 million discounted at 5%. Payback of initial capital can be achieved in 4.86 years.

Case A7 reflects a gold price of US$1750/oz. Under this scenario the project will generate an after-tax IRR of 29.18% and an NPV of US$337.8 million undiscounted and US$219.7 million discounted at 5%. Payback of initial capital can be achieved in 3.82 years.

Sensitivity studies have been prepared varying the price of gold, the operating and capital cost, grade and leach recovery. The project most sensitive to gold price, sensitive to operating cost, grade and recovery and least sensitive to capital cost. The sensitivity results are shown on Table 22.4.

Table 22.3: Financial Analysis Alternative 1 (Contractor Operator)

ROBERTSON PROJECT
ALTENBURG, PORPHYRY AND GOLD PAN DEPOSITS
Financial Analysis Base Case
$(000)s

Resource Ton (000)'s **78,209** **YEAR**

Description	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	Total
Production Ton/year HG				439	1,436	2,687	2,198	2,877	2,903	3,528	4,596	2,696	2,496	652		26,509
Au Grade oz/t				0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02		0.02
Recovery Au %				67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%	67.00%		67.00%
Production Ton/year LG				3441	5541	4464	6688	6056	6253	5465	4549	3673	4328	1244		51,700
Au Grade oz/t				0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01		0.01
Recovery Au %				45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%	45.00%		45.00%
Au ounces payable				20,277	42,619	57,438	58,180	65,106	66,406	73,156	90,776	55,848	59,364	14,942		604,111
Gross Revenue				27,350	57,487	77,476	78,477	87,819	89,572	98,678	122,444	75,332	80,074	20,155		268,219
Operating Costs				(33,741)	(45,271)	(48,182)	(51,389)	(59,206)	(59,465)	(56,309)	(55,790)	(41,635)	(40,283)	(13,554)		($504,826)
Income Tax				($29)	($210)	($205)	($107)	($169)	($214)	($3,617)	($17,845)	($7,572)	($9,799)	($2,053)		($41,819)
Revenue Before Capital Exp.				($6,420)	$12,006	$29,090	$26,981	$28,445	$29,893	$38,752	$48,809	$26,125	$29,991	$4,548		$268,219
Capital Expenditures																
- Development/Construction			($89,449)													($89,449)
- On-Going Capital					($3,434)	($3,434)	($3,434)	($3,237)	($3,237)	($1,619)						($18,396)
Working Capital Change				($3,039)	($1,697)									$4,735		
Reclamation Bond			($7,500)												$7,500	
Reclamation															($7,700)	($7,700)
Salvage															$6,750	$6,750
Total Capital			($96,949)	($3,039)	($5,131)	($3,434)	($3,434)	($3,237)	($3,237)	($1,619)				$4,735	$6,550	($108,795)
Net Cashflow			($96,949)	($9,459)	$6,875	$25,656	$23,547	$25,207	$26,656	$37,132	$48,809	$26,125	$29,991	$9,283	$6,550	$159,424
Discounted NCF 5%			($96,949)	($9,008)	$6,236	$22,162	$19,372	$19,751	$19,891	$26,389	$33,036	$16,840	$18,412	$5,428	$3,647	$85,207
Discounted NCF 8%			($96,949)	($8,758)	$5,895	$20,366	$17,308	$17,156	$16,798	$21,666	$26,370	$13,069	$13,892	$3,981	$2,601	$53,394
Discounted NCF 10%			($96,949)	($8,599)	$5,682	$19,275	$16,083	$15,652	$15,047	$19,055	$22,770	$11,079	$11,563	$3,254	$2,087	$35,999

Rate of Return **15.43%**

Notes:

1. Metal Prices US $ Au/oz 1350.00

2. Capital requirements based on 100% equity.

3. All funds are in US$ except where noted.

4. Taxes are approximate.

Payback 5.94 years

Au first full production year 20,277 ozs

Average NSR/ton $10.42

Table 22.4: Sensitivities – Alternative 1 (Contractor Operator)

Case	Description of Sensitivity	NPV Dis.0%	NPV Dis.5%	NPV Dis.8%	IRR
		US$(000)s	US$(000)s	US$(000)s	%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
Case A2	Gold Price $750	($94,427)	($107,304)	($111,909)	-11.84%
CASE A3	Gold Price $1000	$50,380	$624	($20,123)	5.08%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A5	Gold Price $1500	$226,426	$135,895	$96,886	20.96%
CASE A6	Gold Price $1750	$337,781	$219,732	$168,591	29.18%
CASE A7	Gold Price $2000	$445,057	$300,377	$237,512	36.41%
CASE A8	Grade -10%	$106,068	$43,645	$17,154	10.42%
CASE A9	Grade -5%	$132,368	$64,225	$35,138	12.94%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A10	Grade +5%	$189,244	$107,787	$72,784	17.95%
CASE A11	Grade +10%	$219,453	$130,521	$92,229	20.37%
CASE A12	Capital Cost -20%	$162,796	$88,022	$55,930	15.80%
CASE A13	Capital Cost -10%	$161,071	$86,587	$54,640	15.62%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A14	Capital Cost +10%	$157,789	$83,836	$52,156	15.25%
CASE A15	Capital Cost +20%	$156,204	$82,502	$50,949	15.08%
CASE A16	Operating Cost -20%	$232,745	$142,161	$103,028	22.14%
CASE A17	Operating Cost -10%	$195,890	$113,650	$78,244	18.84%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A18	Operating Cost +10%	$127,084	$59,049	$30,099	12.14%
CASE A19	Operating Cost +20%	$95,153	$32,755	$6,515	8.88%
CASE A20	Leach Recovery +10%	$219,453	$130,521	$92,229	20.37%
CASE A21	Leach Recovery +5%	$189,244	$107,787	$72,784	17.95%
CASE A1	**Base Case**	**$159,424**	**$85,207**	**$53,394**	**15.43%**
CASE A22	Leach Recovery -5%	$132,368	$64,225	$35,138	12.94%
CASE A23	Leach Recovery -10%	$106,068	$43,645	$17,154	10.42%

SECTION 23.0ADJACENT PROPERTIES

The Robertson Property is completely surrounded by unpatented mining claims owned mostly by the Cortez JV and a few junior companies. Immediately west of the core Robertson claim block are the "Excluded" claims which are subject to a joint venture between Coral and Cortez Gold Mines in which Coral has a 39 percent carried interest. There are no known mineral resources located on the "Excluded" claims. North of the Robertson core claims are a series of scattered and mostly small groups of unpatented claims and private property (part of the Newmont "checkerboard"). Most of these properties are in the early stages of exploration and none have reported mineral resources.

Approximately 4 miles south of the center of the Robertson core claims is the Pipeline/South Pipeline mine complex owned and operated by Barrick Gold Corporation. As reported in Barrick's 2010 annual report, proven and probable reserves are estimated to be 317 million tons having an average grade of 0.046 ozAu/t. However, this estimate includes proven and probable reserves from Cortez Hills and the Pediment deposits. For the 9 months ending September 30, 2007, the last year it was reported separately, Pipeline/South Pipeline mine produced 240,000 ounces of gold from 15,948,000 tons of ore (Barrick, 2007 third quarter mine statistics). Mining is by open pit, while processing is by conventional milling using cyanide and CIL and standard heap leach technology.

The Pipeline/South Pipeline are Carlin-type, sediment hosted gold deposits within the Gold Acres structural window. The gold deposits are hosted by favorable carbonate rocks in the lower plate of the RMT. Besides favorable lithology, major ore controls include a series of northeast striking fault/fracture zones and a set of stacked, flat-lying shear zones. These deposits are in sharp contrast to the smaller and lower grade hornfels-hosted deposits that are developed at Robertson in the upper plate of the RMT.

SECTION 24.0 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information.

SECTION 25.0 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

Based on drilling results through 2008 the Robertson Property has an inferred mineral resource estimated to contain 178.9 million tons of mineralized material having an average grade of about 0.0189 ozAu/t at a 0.0106 ozAu/t cutoff grade (Beacon Hill, 2009). Of this total resource, about 64.5 million tons having an average grade of 0.017 ozAu/t and containing about 1.1 million ounces of gold are considered shallow oxide mineralization potentially amenable to open pit mining and possible heap leaching. In addition, approximately 85.7 million tons of the total resource, having an average grade of 0.0188 ozAu/t, are classified as sulfide and could potentially be mined by open pit. The amenability of the sulfide resource to heap leaching is unknown at this time and must be determined before the sulfide resources can be considered. It is anticipated, but not certain, that a large portion of the 178.9 million tons of inferred resources will be upgraded to a measured and indicated mineral resource as a result of the proposed 2012-13 drilling program. This will be accomplished by offset and in-fill drilling by diamond core drilling where needed and by conducting appropriate metallurgical and geotechnical studies.

With the exception of the Gold Pan Zone, the remaining resources were drilled out by either Amax Gold (1990-1996) or Coral Gold Resources (2004-2010). It is the opinion of the author that the quality of work by these companies justifies a high degree of confidence in the data that was generated. The use by these companies of highly skilled drilling companies, high quality assay labs and the routine employment of QA/QC programs provides additional confidence in the data generated during their exploration programs. One concern is the general lack of diamond drilling in most of the resources, except for the Porphyry Zone. It is strongly recommended that future resource definition drilling be conducted using mainly diamond drilling.

About one-half of the Gold Pan inferred resource was originally drilled out by E & B Exploration (1980-1981). While only a limited record of their work is available, it appears that the drilling and assaying met or exceeded industry standards of that time. During 1990-1996, Amax drilled approximately 25 RC holes in the Gold Pan Zone within the area initially drilled by E & B, as well as to the west where the zone overlies the 39A mineralization. In addition, Coral Gold also drilled the same areas during the period 2004-2010. The drilling by both Amax and Coral appear to confirm the results of the E & B drilling. However, one area where follow up drilling was unable to confirm the earlier drilling is the area now covered by an existing heap leach pad. The leach pad was constructed in 1989, well after the E & B drilling, but prior to the Amax and Coral drilling. Drilling on or in close proximity to the leach pad has been and continues to be prohibited by the BLM. The leach pad is approximately 850 ft by 800 ft and may cover as much as one-third of the resource. As a result, the portion of the resource that underlies the leach pad was estimated using only historic drilling (E & B and early Coral 1986-1988), some of which is considered of marginal quality.

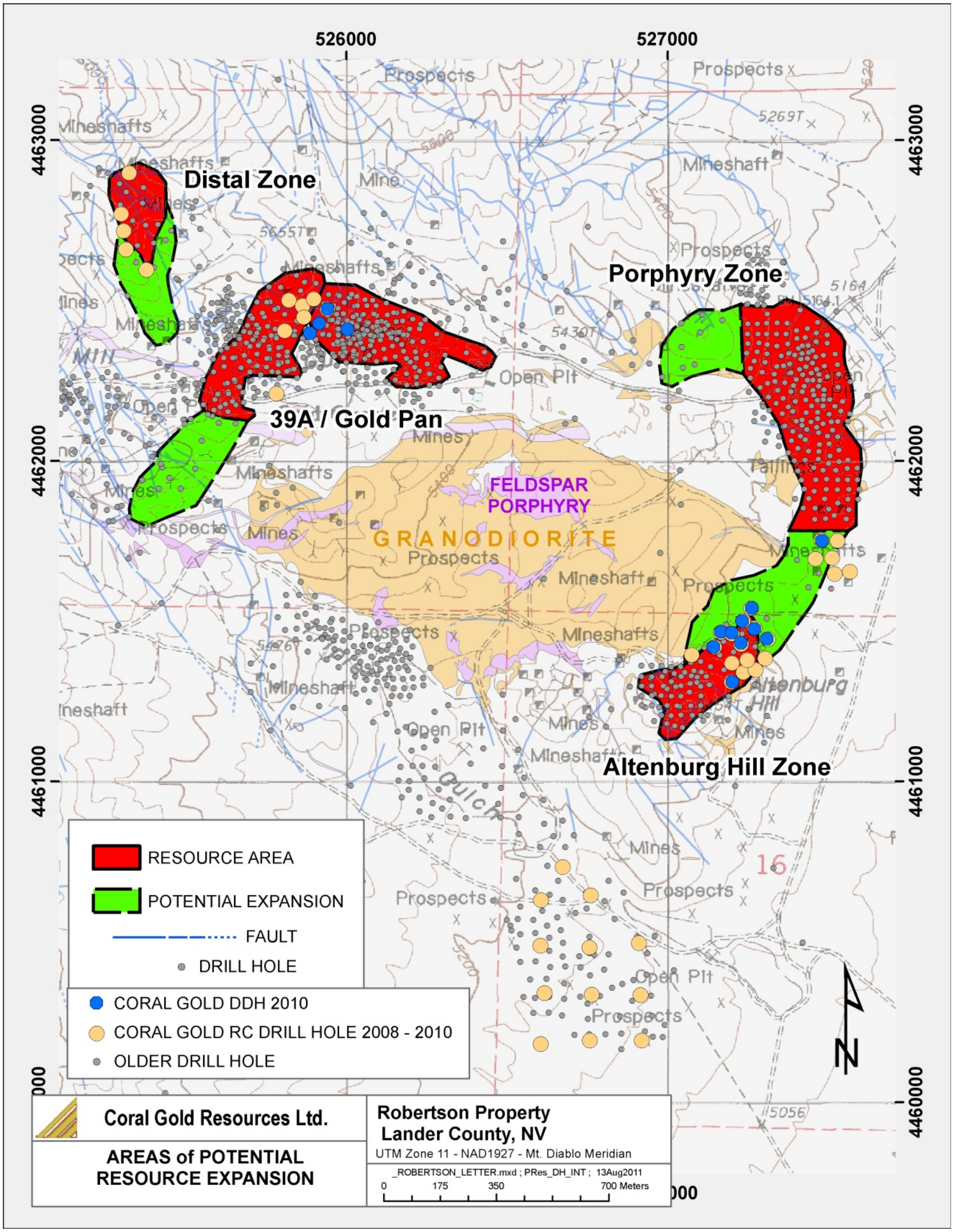


Figure 25-1: Areas of Potential Resource Expansion.

25.2 CONCLUSIONS

The following is a list of the conclusions;

- The Robertson property is one of merit upon which further work is warranted consisting of exploration and definition drilling, metallurgical test work, geotechnical test work and associated work that will allow the preparation of a project evaluation to a prefeasibility level.
- The relative low capital cost provides an opportunity for a junior company such as Coral to raise those funds and operate the project.
- The project is seen as one where a contractor operation would be beneficial due to the since the increased operating cost of some 49% above that estimated as an owner operation is off set by the decrease in capital cost, thus providing a lower capital cost6 for the Owner while maintaining similar economic benefits.
- The Nevada project approval system has elongated the development period for the project. Every effort should be made to attempt to reduce the time to achieve regulatory approval of the project and allow the project to be developed as quickly as is feasible.
- The financial analysis results indicate that the project breaks even at US$950/oz gold for the Owner operation and US$1010/oz for the Contractor.

SECTION 26.0 RECOMMENDATIONS

It is recommended that the project be developed in two stages. The first is that work required to take the project to complete a prefeasibility report and then that work required to complete a full "bankable" feasibility, i.e., one on which a construction decision could be made. Costs have been included for stage 2 in the capital cost section for guidance only and will be defined in the prefeasibility report.

The work in stage 1 consists of;

1. Exploratory and definition drilling.
2. Metallurgical test work program.
3. Environmental program.
4. Geotechnical and associated work.
5. Preliminary Feasibility Study.

A summary of stage 1, estimated to cost $7.9 million, is shown on Table 26.1.

Cost breakdowns are shown on Tables 26.2 and 26.3.

Table 26.1: Summary of Expenditures for Stage 1

Description	Estimated Cost $
Royalty and Regulatory Fees	$351,680
Exploratory and definition drilling	$2,817,000
Metallurgical test work program	$900,000
Environmental program	$1,826,138
Preliminary Feasibility Study	$1,495,000
Contingency	$510,182
Total	**$7,900,000**

26.1 EXPLORATION AND DEFINITION DRILLING

It is recommended that Coral conduct a two phase exploration program focused on expanding and up-grading the near-surface oxide and sulfide inferred mineral resources to measured and indicated. The details of the proposed work program and budget are presented in Table 26.2. The size of Phase II is contingent upon favorable results of Phase I.

The Phase I should consist of drilling 40 HQ diameter diamond core holes and 42 RC holes having an average depth of 400-500 ft and totaling about 40,000 ft.

- **Porphyry Zone:** Is currently defined by approximately 200 drill holes (150 RC holes and 50 core holes) totaling 80,000 ft drilled by Amax in 1992-94. Because these holes were completed prior to implementation of NI43-101 Standards of Disclosures for Mineral Projects, these drill holes are considered to be "historic" and, therefore, the Amax data must be verified before the Porphyry resource can be ungraded to measured and indicated. To comply with the NI43-101 rules, it is recommended that 10 percent (20 holes) of the historic drill holes be "twinned" by diamond core drilling. If the twin core holes compare favorably with the historic drilling, it can be assumed that the "historic" Amax drilling data can be used with confidence.

- It is also recommended that an additional 17 RC holes, totaling about 7,600 ft, be drilled along the west and south boundaries of the Porphyry Zones to test for possible extensions to mineralization.

- **Altenburg Hill/South Porphyry Area:** Twenty-five RC holes totaling 12,400 ft as offset drilling on the north flank of Altenburg Hill and in the gravel-covered area south of the Porphyry Zone. Of particular importance is testing the undrilled granodiorite-hornfels contact area extending at least 1,500 ft north of holes CR08-31/CC10-3. The intrusive/hornfels contact is the principal control of mineralization in the Porphyry and Altenburg Hill Zones.

- **Gold Pan Zone:** Twenty wide-spaced diamond core holes totaling 10,000 ft to verify continuity and grade retuned in historic drilling. These holes will also provide material for additional metallurgical test work.

- Phase II should consist of 20 HQ diameter diamond core holes totaling about 10,000 ft. The purpose of core drilling is to provide geological data on the controls of mineralization, acquire geotechnical data (RQD and specific gravity), confirm grade and continuity and provide material for metallurgical testing.

- **Altenburg Hill/South Porphyry:** Assuming the Phase I RC drilling identifies a substantial increase to the Altenburg Hill/South Porphyry it is recommended that follow up diamond core drilling (20 holes) be conducted in these areas to provide ore-grade oxide and sulfide mineralization for metallurgical studies and confirm the grade and continuity of mineralization.

Total cost of both phases of the proposed 2012-13 drilling program is expected to be about US$3.1 million. Phase I will begin as soon approval of the 2010 APO is granted by the BLM and NDEP, pending drill rig availability and weather conditions. As proposed and assuming two RC rigs and two core rigs are employed, it is estimated that Phase I RC drilling will take about 50 days to complete and core drilling will take 120 days to complete, including days off. The timely completion of Phase I will depend on the availability of diamond core drills which have been difficult to secure in Nevada during the last few years. Assuming that two core rigs are employed, the Phase II program should take about 60 days to complete, including days off.

Table 26.2: Recommended 2012-13 Drilling Program and Budget.

Target/Activity	Work Program	Total Footage (ft)	Cost in US$
PHASE I			
Porphyry Zone	20 vertical diamond core holes 500-ft-deep	10,000	650,000
Alt Hill/Porphyry Zones	42 vertical RC holes 400-500-ft-deep	20,000	250,000
Gold Pan Zone	20 vertical diamond core holes 500-ft-deep	10,000	650,000
Assaying/QA-QC	Sample prep + assays: 8,800 assays x $25/sample		220,000
Drill site construction	82 drill sites + sumps: 80 hrs x $140/hr		11,200
Access road constr.	4,800 ft + mob/demob		5,200
Reclamation	82 sites + 4,800 ft of road + mob/demob		12,600
Drilling supplies	Sample bags + chip trays: 8,200 bags + 200 chip trays		20,000
Surveying	Collar surveys		7,000
Database management	Consulting services		10,000
Geological Services	Consulting services: 8.7 man months		130,000
Expenses	Food,gas, vehicles		24,000
Hourly labor	Core prep, core sawing, etc.		20,000
Phase I Total		**40,000**	**2,010,000**

Target/Activity	Work Program	Total Footage (ft)	Cost in US$
PHASE II			
Alt Hill/Porphyry Zones	20 vertical diamond core holes 500-ft-deep	10,000	650,000
Assaying/QA-QC	Sample prep + assays: 2,200 assays x $25/sample		55,00
Drilling supplies	Sample bags: 2,100 bags		5,000
Surveying	collar surveys		4,000
Database management	Consulting services		5,000
Geological Services	Consulting services: 4.0 man months		60,000
Expenses	Food, gas, vehicles		18,000
Hourly labor	Core prep, core sawing, etc.		10,000
Phase II Total		**10,000**	**807,000**
Total Estimated Cost of Phase I + Phase II			**2,817,000**

26.2 METALLURGICAL TEST WORK PROGRAM

Testing of oxidized ore zones from Altenburg Hill, and Gold Pan deposits have been shown to respond to heap leach processing. Further evaluation of these two deposits and any other additional ore zones being considered for development are needed as the project proceeds into prefeasibility. Further testing will be required on the Porphyry deposit which has had prior studies performed by AMAX and others, but additional confirmation laboratory work will be needed.

Variability testing would be performed on samples obtained both spatially and at depth for the oxide and transition to sulfide ore zones. Prior to variability testing composited material representing larger zones of each deposit require more detailed work to better define the crush size and other process conditions. These master composites should be based on similar physical characteristics of the rock including SG, mineralogy, lithology, as well as chemical similarities based on the analyses. Once characterized, then representative samples need to be submitted for crushing work index and abrasion testing, as well as mineralogical evaluation of column feed and products. Some of this work requires receiving whole drill core. Extensive column work would be undertaken to determine optimum crush size and other process conditions. The recent 2011 laboratory program focused on 19 mm (3/4") product crush size. However, when comparing to some historical work a 12.5 mm (1/2") crush size and work will include this size as well as other sizes. Optimization and variability testing for oxide and partially oxidized material is expected to approach $350,000, not including the cost of generating the sample from drilling programs.

Similar testing as was performed on oxide materials would be required for sulfide and transition zone materials. The amount of column studies would be rationed depending on the resource tonnage identified in these deposits. In addition to gold grade it will be important that resource development and mine scheduling include classification of ore as oxide, partial oxide (transition) and sulphide material. Sulfide materials have shown a decreased leach response and additional processing parameters might be investigated including reagent use and concentrations. Dependent on grade shell characteristics the evaluation of alternate processing procedures can be investigated for higher grade sulfide feeds. This could include tank leaching and flotation studies. The estimated budget for this portion of a test program performed on sulfide material is $200,000.

A lower gold cut-off grade of 0.005 oz/ton (0.17 g/tonne) was included in this study. For the 2011 test program samples of this material were labeled as overburden (OB) and evaluated for cyanide amenability. The results show an encouraging response and the possibility of dump leaching of run of mine low grade materials. The next step of evaluation for these materials should involve large particle size rock. North American laboratories are capable of handling up to 10 tons of 100% minus 300 mm (~12") feed. Several such samples would need to be mined, transported, studied and disposed. The work should also include some conventional columns using coarse crush material. Estimated budget for such a program is $350,000.

Depending on the schedule of project advancement, on-site testing can be considered for both heap and dump leaching evaluation to supplement and /or replace portions of a laboratory program. This would involve bulk mining and building of cribs and/or test heaps (with pads and ponds) in order to better evaluate leaching of low grade run of mine rock as well as higher grade crushed material. Such a program would be costly as it involves contract bulk mining of several thousand tons of rock, mobile crushing and stacking, and long term monitoring. Costs would need to be developed with local contractors and suppliers.

The overall cost is estimated to be $900,000 as shown on Table 26.3.

Table 26.3: Estimated Cost - Metallurgical Test Work Program

Description	Est. Cost $
Oxide / Partially Oxidized High Grade Drill Core	
Sample Delivery / Preparation	$20,000
Ore Characterization / Geotechnical	$12,000
Crushing WI / Abrasion Testing	$25,000
Bottle Roll Testing	$18,000
Column Optimization Testing	$50,000
Column Variability Testing	$210,000
Solution Chemistry / Detox	$15,000
Sub-Total	**$350,000**
Sulfide and Transition High Grade Drill Core	
Sample Delivery / Preparation	$8,000
Ore Characterization / Geotechnical	$7,000
Crushing WI / Abrasion Testing	$12,000
Whole Ore Bottle Roll Testing	$10,000
Column Optimization Testing	$30,000
Column Variability Testing	$90,000
Flotation Studies	$25,000
Concentrate Cyanide Studies	$14,000
Solution Chemistry	$4,000
Sub-Total	**$200,000**
Low Grade Bulk Tonnage Samples	
Bulk Mine and Delivery	$200,000
Large ROM Column Studies	$125,000
Associated testing for characterization	$25,000
Sub-Total	**$350,000**
Total	**$900,000**

26.3 GEOTECHNICAL AND ASSOCIATED INVESTIGATIONS

The proposed geotechnical and associated investigation costs are included in the feasibility costs estimates. It is planned to geotechnically log the core holes drilled as part of the exploration program. This information together with the list below will form the basis for this work.

26.3.1 Heap Leach Pad Stability Assessment

The geotechnical stability of the heap leach pads has not been assessed. It is recommended that embankment stability analyses be carried out using a limit equilibrium model such as SLOPE/W. Such models utilize systematic algorithms to obtain the minimum factor of safety from a series of potential slip surfaces. The models assess the susceptibility of a given embankment to fail in circular, translational or composite slip modes.

The embankments should be analysed under both static and pseudo-static (seismic conditions) to ensure that appropriate design factors of safety are achieved by the design.

A minimum factor of safety of 1.3 is recommended for the static case. The consequences of failure of the heap leach pad during an earthquake event are likely to be minimal and restricted to some displacement of the heap leach pad slopes. There should be negligible impact on the integrity of the heap leach pad and little, if any, impact on other mine site facilities; however, for the design of the heap leach pads a conservative design earthquake such as the 1 in 475 year earthquake event is recommended.

The stability of the heap leach pads is likely to be controlled by the interface friction angles between the overliner, geomembrane liners and the GCL. Potential slip surfaces should be assessed and the design should have an appropriate factor of safety against this failure mode.

Pad stability during earthquake loading should be assessed by performing a pseudo-static analysis, whereby a horizontal force (seismic coefficient) is applied to the embankment to simulate earthquake loading to determine the critical acceleration required to reduce the factor of safety to 1.0. Deformation of the leach pad may occur if the critical acceleration is lower than the predicted average maximum ground acceleration along the potential slope surface from the design earthquake.

Potential deformations under earthquake loading should be estimated using the simplified methods of Newmark (1965) and Makdisi-Seed (1977). These two methods estimate displacement of the potential sliding mass based on the average maximum ground acceleration and the yield acceleration. The yield acceleration corresponds to the seismic coefficient required to initiate movement of the sliding mass. The yield acceleration may be determined by iterative stability analyses.

The stability of the heap leach facility is very sensitive to the interface shear strengths associated with the liner system. Direct shear testing will be required for future design studies to provide more definitive interface friction angles between the various proposed liner surfaces.

26.3.2 Open Pit and Plant Site

Detailed foundation assessments have not been carried out for the pit and plant site areas; however, based upon a review of the site, it is unlikely that geotechnical conditions will preclude the development of mine site infrastructure.

It is recommended that the following points be assessed;

- Evaluation of the allowable soil bearing pressures induced by plant site facilities for a range of different footing shapes and sizes, and consideration of potential settlements
- Estimation of the allowable bearing pressures of the bedrock
- Seismic design parameters for the plant site, and
- Recommendations for the plant site foundation design.

26.3.2.1 Other Studies

The following is a list of recommendations for additional site investigations, test work and design studies required to carry the project through to final design and construction:

- Geotechnical Investigations and Test work:
- Test pits in the leach pad area to investigate foundation conditions, overburden materials, and depth to bedrock.
- Drill holes to investigate depth and quality of bedrock, for permeability testing, and for the installation of groundwater monitoring wells.

- Additional test pits to prove suitability, availability and quantity of borrow materials for earthworks construction.
- Additional laboratory index test works (including compaction tests) on potential borrow materials.
- Strength and permeability test work on potential borrow materials for pad foundation and embankment construction.
- Direct shear testing of the geosynthetic liner interfaces, to determine interface friction angles for stability assessment.
- Ore testing (including gradation, leaching moisture content, residual moisture content, Altenburg Hill limits, load-permeability, and load-density).

26.3.2.2 Design Studies and Analyses:

- Detailed hydrology studies to predict surface runoff and storm flows.
- Leach testing to determine optimal ore densities, leaching rates and resulting in-heap moisture contents.
- Detailed water balance analyses, based on results of hydrology and leach testing, to estimate solution storage area water volumes, peak storm flows for ditch design, and for the sizing of pump and pipework systems.
- Stability analyses based on results of laboratory strength test work.
- Seepage analyses to predict seepage flow patterns and solution losses for the design of the LDRS.

Detailed design of all civil and mechanical works, including sumps, intakes, outlets, pumps, pipe systems etc.

26.3.3 Foundation Design

The foundation recommendations provided are preliminary based on limited and preliminary information on site layout and design loads.

It is recommended that footings for critical plant site structures are constructed directly on bedrock or otherwise suitable foundation materials. Bedrock is anticipated at shallow depths of from 5 to 15 feet and should provide a solid foundation. Strip footings at least 3 feet in width are recommended for the plant site structures. They may be designed to impose a maximum pressure of 75 psi on the weathered bedrock, with negligible settlements.

Overburden materials should be removed at a 2H:1V slope. The site should be maintained dry and well drained during and after construction activities. Surface runoff from building roofs and access roads, etc., shall be directed away from foundations, and perimeter drains of coarse grained materials shall be installed to effectively carry water around the perimeter of the structures to a low sump.

26.4 ENVIRONMENTAL WORK

The proposed environmental work consists the following as shown in Table 26.4. It is expected based upon present legislation to be complete and the project approved in 2017. It may be possible to reduce this period and it is recommended that this be reviewed to see if a reduction in the 6 years can be reduced.

Table 26.4: Summary of Proposed Environmental work

Description	Year 2012	2013	2014	2015	2016	Total
Baseline Data Collection						
Water	$50,000	$60,000	$60,000	$30,000	$30,000	$230,000
Geochemistry	$30,000	$20,000				$50,000
Eagle/Raptor Survey	$25,000					$25,000
Plan of Operation/Reclamation Permit Application	$35,000	$20,000	$15,000	$10,000	$20,000	$100,000
EIS (draft and final)	$90,000	$200,000	$300,000	$200,000	$210,000	$1,000,000
Water Pollution Control Permit			$40,000	$40,000	$40,000	$120,000
Air Permits			$40,000	$10,000	$10,000	$60,000
Water Appropriations	$10,000	$10,000		$10,000		$30,000
Stormwater Pollution Prevention Plan					$10,000	$10,000
Industrial Artificial Pond Permit					$125	$125
Potable Water					$10,000	$10,000
Septic					$20,000	$20,000
Class III Landfill					$5,000	$5,000
Contingency 10%	$24,000	$31,000	$45,500	$30,000	$35,513	$166,013
Total	**$264,000**	**$341,000**	**$500,500**	**$330,000**	**$390,638**	**$1,826,138**

26.5 PREFEASIBILITY STUDY

The expected cost to complete a Preliminary Feasibility study, based upon the results of the PEA as described in this report, is $1.5 million. Table 26.5 shows the cost estimate breakdown for the Prefeasibility Study.

Table 26.5: Prefeasibility Cost Estimate

Task Description	Estimated Cost $
Project Management	$100,000
Geology	$150,000
Resource Estimate and Open Pit Optimization	$150,000
Mine Planning	$150,000
Metallurgy and Process Plant	$95,000
Geotechnical	$300,000
Surface Buildings	$50,000
Power supply/Electrical distribution/Communication	$70,000
Cost Estimates	$110,000
Financial Analysis	$75,000
Project Disbursements	$50,000
Contingency 15%	$195,000
Total Estimated Cost	**$1,495,000**

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SECTION 28.0 CERTIFICATE OF QUALIFIED PERSONS

CERTIFICATE OF AUTHOR

W. P. Stokes

I, Wilfred Peter Stokes, P.Eng., do hereby certify that:

1) I am a consulting mining engineer with an office at 113-6505 3rd Avenue, Delta, British Columbia, Canada.

2) This certificate applies to the entitled "Preliminary Economic Assessment of the Robertson Property, Lander County, Nevada, USA" January 15, 2012 ("Technical Report") prepared for Coral Gold Resources Ltd., Vancouver, British Columbia, Canada..

3) I am a graduate of the Stoke (England) College of Technology in 1967 with a HND in Mining Engineering.

4) I am a member in good standing of the Association of Professional of Professional Engineers and Geoscientists of BC.

5) I have continuously practiced my profession performing mining engineering since 1976, both as an employee of various mining companies and since 1986 Beacon Hill Consultants a mining consulting company, as principal and independent consultant.

6) I have visited the property on a number of occasions the last was February 9, 2011.

7) In the independent report titled "Preliminary Economic Assessment of the on the Robertson Property, Elko, Nevada, USA" January 15, 2012, I am responsible for Section 1, Sections 16.1 and 16.2, Sections 18.2.12 to 18.2.17, 19, 21, 22, 24, 25 and 26. I am also responsible for overall study management and compilation.

8) I have had prior involvement as one of the authors of the report titled "Mineral Resource Estimate for the Robertson Property, Lander County, Nevada, USA" dated January 27, 2008.

9) I am independent of Coral Gold Resources Ltd. as defined in Section 1.5 of National Instrument 43-101, other than providing consulting services

10) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

11) 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 and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

12) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

"W. P. Stokes"{Signed and Sealed}

W. P. Stokes, P.Eng.

Dated this 15th January, 2012 in Delta, British Columbia

CERTIFICATE OF QUALIFICATION

I, Robert T. McCusker, do hereby certify that:

1. I am a consulting geologist residing in Lakewood, Colorado, U.S.A..

2. This certificate applies to the "Preliminary Economic Assessment of the Robertson Property, Lander County, Nevada, USA" dated January 15, 2012 prepared for Coral Gold Resources Limited, Vancouver, B. C.

3. I am a graduate of San Jose State University with a B.Sc. in Geology in 1975 and M.Sc. in Geology in 1982. I have practiced my profession continuously since 1975. I am a Professional Geologist, registered with the Wyoming Board of Registration for Professional Geologists.

4. I supervised drilling on the Robertson Property of Coral Gold Resources Limited between March 28 - April 20, 2004, November 21 - December 18, 2004, June 7 - August 13, 2005, April 30 – June 10, 2006 and August 10 - December 3, 2006, July 20 - September 8, 2007, May 20 - September 22, 2008, June 1 - July 11, 2010 and September 16 - December 11, 2010. During the period 1990-94, I was Senior Project Geologist for Amax Gold Inc. at the Robertson Property, with responsibility for supervising the evaluation of the property and preparing part of the 1994 Amax Feasibility Study;

5. In the independent report titled "Preliminary Economic Assessment of the Robertson Property", I am responsible for preparing Sections 4-12.

6. I have read the definition of "qualified person" set out in the revised Nation Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in the revised National Policy 43-101.

7. 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.

8. I have read the revised National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101FI. This technical report has been prepared in compliance with that instrument and form.

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

"Robert T. McCusker" (signed and sealed)

Robert T. McCusker , P. Geol.
Dated January 15, 2012

CERTIFICATE OF AUTHOR

Garth Kirkham

I, Garth David Kirkham, P.Geo., do hereby certify that:

1) I am a consulting geoscientist with an office at 6331 Palace Place, Burnaby, British Columbia.

2) This certificate applies to the entitled “Preliminary Economic Assessment of the Robertson Property, Lander County, Nevada, USA” January 15, 2012 (“Technical Report”) prepared for Coral Gold Resources Ltd., Vancouver, British Columbia, Canada..

3) I am a graduate of the University of Alberta in 1983 with a B. Sc. in Geophysics.

4) I am a member in good standing of the Association of Professional Geoscientists of Ontario, Association of Professional Engineers and Geoscientists of the Province of Alberta, the Association of Professional Engineers and Geoscientists of BC, and the Northwest Territories and Nunavut Association of Engineers and Geoscientists.

5) I have continuously practiced my profession performing computer modeling since 1988, both as an employee of a geostatistical modelling and mine planning software and consulting company and as an independent consultant.

6) I have not visited the property.

7) In the independent report titled “Preliminary Economic Assessment of the on the Robertson Property, Elko, Nevada, USA” January 15, 2012, I am responsible for Section 14

8) I have had prior involvement as one of the authors of the report titled “Mineral Resource Estimate for the Robertson Property, Lander County, Nevada, USA” dated January 27, 2008.

9) I am independent of Coral Gold Resources Ltd. as defined in Section 1.5 of National Instrument 43-101, other than providing consulting services

10) I have read the definition of “qualified person” set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

11) 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 and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

12) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

“Garth Kirkham”{Signed and Sealed}

Garth Kirkham, P.Geo.

Dated this January 15, 2012 in Burnaby, British Columbia

CERTIFICATE OF AUTHOR

F. Wright

I, Frank R. Wright, P.Eng., do hereby certify that:

1) I am a consulting metallurgical engineer with an office at 427 Fairway Dr., North Vancouver, British Columbia, Canada.

2) This certificate applies to the entitled "Preliminary Economic Assessment of the Robertson Property, Lander County, Nevada, USA" January 15, 2012 ("Technical Report") prepared for Coral Gold Resources Ltd., Vancouver, British Columbia, Canada..

3) I am a graduate of BSc. Metallurgical Engineering in 1979, from University of Alberta, Edmonton, AB; and with a Bachelor of Business Administration in 1984, from Simon Fraser University, Burnaby, BC.

4) I am a member in good standing of the Association of Professional of Professional Engineers and Geoscientists of BC.

5) I have continuously practiced my profession for performing metallurgical and mineral process engineering for 28 years, both as an employee of various mining and consulting companies, and since 1997 as an independent consultant.

6) I have not visited the Robertson Property.

7) In the independent report titled "Preliminary Economic Assessment of the on the Robertson Property, January 15, 2012, I am responsible for Section 13 and Section 26.2 of the report.

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

9) I am independent of Coral Gold Resources Ltd. as defined in Section 1.5 of National Instrument 43-101, other than providing consulting services

10) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

11) 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 and that this technical report contains all scientific and technical information that is required to be disclosed to make the sections for which I am responsible in the technical report not misleading.

12) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

"Frank Wright"{Signed and Sealed}

Frank R. Wright, P.Eng.

Dated this January 15, 2012 in North Vancouver, British Columbia

CERTIFICATE OF AUTHOR

C. M. Aurala

I, Corey Michael Aurala, P.Eng., do hereby certify that:

1) I am a consulting civil engineer with an office at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, Canada.

2) This certificate applies to the entitled "Preliminary Economic Assessment of the Robertson Property, Lander County, Nevada, USA" January 15, 2012 ("Technical Report") prepared for Coral Gold Resources Ltd., Vancouver, British Columbia, Canada..

3) I am a graduate of the University of British Columbia in 2003 with a Bachelor of Applied Science degree in Civil and Environmental Engineering.

4) I am a member in good standing of the Association of Professional of Professional Engineers and Geoscientists of BC.

5) I have continuously practiced my profession performing civil engineering since 2003, as an employee of various consulting engineering companies, most currently Knight Piésold Ltd. located in Vancouver, British Columbia.

6) I have visited the property on February 9, 2011.

7) In the independent report titled "Preliminary Economic Assessment of the on the Robertson Property, Nr. Elko, Nevada, USA" January 15, 2012, I am responsible for Section 18.1-18.8.

8) I have had no prior involvement with this Project.

9) I am independent of Coral Gold Resources Ltd. as defined in Section 1.5 of National Instrument 43-101, other than providing consulting services

10) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

11) 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 and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

12) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

"C.A. Aurala"{Signed and Sealed}

C. A. Aurala, P.Eng.

Dated this January 15, 2012 in Vancouver, British Columbia

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CERTIFICATE OF AUTHOR

I, Jeffrey Vaughan Parshley, B.A., C.P.G do hereby certify that:

1. I am a Corporate Consultant of:

   SRK Consulting (U.S.), Inc.
   7175 W. Jefferson Ave, Suite 3000
   Denver, CO, USA, 80235

2. I graduated with a degree in Bachelor of Arts in Geology from Dartmouth College in 1980.
3. I am a Certified Professional Geologist registered through the American Institute of Professional Geologists.
4. I have worked as a Geologist/Environmental Specialist for a total of 31 years since my graduation from university. My relevant experience includes more than 23 years of mine permitting, closure and environmental studies in the state of Nevada, the Western U.S. and internationally.
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 Sections 1.7 and 20 of the technical report titled "Preliminary Economic Assessment", dated January 15, 2012 with an effective date of January 15, 2012 (the "Technical Report") relating to the Robertson property Lander County, Nevada, USA. I have visited the Robertson property no less than seven times for at least one day each visit from 1999 through 2004.
7. I have had prior involvement with the Robertson property that is the subject of the Technical Report. During the period 1999 through 2003 I worked as a consultant on the development, design and implementation of closure activities related to a previous operation at that site. During that time and through 2010 I have acted as a senior reviewer for ongoing permitting work related to the closed site and ongoing exploration activities.
8. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

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11. As of January 24, 2012, to the best of my knowledge, information and belief, Sections 1.7 and 20 contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24 Day of January, 2012.

Signed and sealed (AIPG CPG #8174)

Jeffrey Vaughan Parshley

DATE AND SIGNATURE PAGE

"W. P. Stokes" {signed and sealed}

____

W. P. Stokes, P.Eng.
Beacon Hill Consultants (1988) Ltd.
Effective date: January 15, 2012
Signing date: January 15, 2012

"Robert T. McCusker"

____

Robert T. McCusker, P. Geo.
Geological Consultant
Effective date: January 15, 2012
Signing date: January 15, 2012

"Garth Kirkham" {signed and sealed}

____

Garth Kirkham, P.Geo.
Kirkham Geosystems Ltd.
Effective date: January 15, 2012
Signing date: January 15, 2012

"Frank Wright" {signed and sealed}

____

Frank Wright, P.Eng.
F. Wright Consulting Inc.
Effective date: January 15, 2012
Signing date: January 15, 2012

"Corey Aurala" {signed and sealed}

____

Corey Aurala, P.Eng.
Knight Piésold Ltd.
Effective date: January 15, 2012
Signing date: January 15, 2012

"Jeff Vaughan Parshley" {signed and sealed}

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Jeff Vaughan Parshley, B.A. C.P.G.
SRK Consulting (U.S.) Inc.
Effective date: January 15, 2012
Signing date: January 15, 2012