Scott Wilson Mining

VICTORIA GOLD CORP.

TECHNICAL REPORT ON THE EAGLE GOLD PROJECT, YUKON TERRITORY, CANADA

NI 43-101 Report

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1 SUMMARY

EXECUTIVE SUMMARY

Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA) was retained by Victoria Gold Corp. (Victoria Gold) to prepare an independent Technical Report on the Eagle Gold Project (the Project), in central Yukon Territory, Canada. The purpose of this Technical Report is to summarize the results of a Pre-Feasibility Study (PFS) on the Project, including Mineral Resource and Mineral Reserve estimates. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects. Scott Wilson RPA visited the property on June 10 to 11, 2009.

In June 2009, Victoria Gold acquired its interest in the Dublin Gulch Property (which contains the Eagle Gold Project) as a result of its takeover of StrataGold Corporation. The Project comprises:

- An undeveloped gold deposit
- Access by highway and unpaved roads
- A 30-man exploration camp, with little other permanent infrastructure
- Mineral Reserves of 66 Mt, at a grade of 0.82 g/t Au, containing 1.8 Moz
- Potential development of a 9.1 Mtpa conventional heap leach open pit operation with an eight-year mine life
- Significant Mineral Resources and mineralization extending below the Mineral Reserve pit

The PFS was commenced immediately, led by Scott Wilson RPA, which covered geology, resource estimation, mine design, heap leach design, and cost estimation. Kappes Cassiday & Associates (KCA) carried out metallurgical testwork, process design, and process cost estimation. BGC Engineering Inc. (BGC) carried out geotechnical field investigation and analysis for open pit slopes and for infrastructure requirements. Stantec Inc. (Stantec) provided advice on environmental and permitting issues, in the course of preparing a Project Proposal (a permitting document) in parallel with the PFS.

CONCLUSIONS

In the opinion of Scott Wilson RPA and KCA, the PFS indicates that positive economic results can be obtained for the Eagle Gold Project, in a scenario that includes open pit mining, three-stage crushing to a P_{80} of 5 mm, and heap leaching followed by carbon ADR gold recovery.

The Life of Mine Plan (LOMP) for the Project indicates that Mineral Reserves of 66 million tonnes, at an average grade of 0.82 g/t Au, will be mined over eight years, starting in 2013. Gold production is projected to total 1,254,000 ounces. Capital costs for construction are estimated to total C$281 million, including a contingency of C$38 million. Cash costs are projected to average US$503 per ounce. At a gold price of US$900 per oz, the Project is estimated to generate a pre-tax NPV of C$78 million, at a discount rate of 7.5%, and has a pre-tax IRR of 15%.

Specific conclusions by area of the PFS are as follows.

GEOLOGY AND MINERAL RESOURCES

- Mineral Resources are estimated to be 154 Mt, at a grade of 0.65 g/t Au, containing 3.2 Moz. All resources are classified as Indicated, and were estimated at a cut-off grade of 0.21 g/t Au. Mineral Resources were constrained within a pit shell, generated at a gold price of US$1,050 per oz.

- In Scott Wilson RPA's opinion, the drill hole sampling, over the history of the project, appears to have been carried out to a reasonable standard. Data has been collected in several campaigns spanning many years, and for some years the documentation is not as detailed as for others. Specific concerns have been raised about certain sampling practices, which have resulted in some changes in protocols over the years. Most of the concerns raised by earlier reviewers were centred around assay repeatability and nugget effect. This could have an effect on the accuracy of local block grade estimates, however, in Scott Wilson RPA's opinion, there is no evidence of bias, and so the global grade estimates should be accurate.

- In Scott Wilson RPA's opinion, the sampling is representative of the mineralization.

- The core handling and security protocols, where documented, were consistent with common industry practice.

- Assaying was conducted using conventional methods, commonly used in the industry, and carried out by accredited commercial laboratories.

- Reasonable levels of assay QA/QC were applied throughout the history of the Project, although there are some programs for which documentation is not available. In Scott Wilson RPA's opinion the assays are of a reasonable standard and unbiased.

- The number of bulk density measurements taken to date is not sufficient and more should be taken.

- Metallurgical recoveries vary depending on the degree of alteration of the rock mass. Estimated recoveries range from 68% in unaltered granodiorite to 77% in weathered rock. Scott Wilson RPA has assigned ore types to the block model based on the logging of alteration in drill core. This ore type classification is presently quite rudimentary and requires more work. Relogging of the alteration should be conducted to improve the database for interpretation of ore types.

- There are exploration targets that remain to be explored in the vicinity of the Eagle deposit. Specific target areas that have potential for adding Mineral Resources are in the west end of the deposit, as well as along the trend of showings which encompass the Steiner, Olive, and Shamrock showings.

- There is potential for additions to the Mineral Resources with increased gold prices. A significant portion of the deposit extends below the US$1,050/oz Au pit shell used to constrain the Mineral Resource estimate. At higher gold prices, more of the deposit is captured by the pit optimization, which would result in an increase in Mineral Resources.

MINING

- Mineral Reserves are estimated to be 66 Mt, at a grade of 0.82 g/t Au, containing 1.8 Moz. All reserves are classified as Probable, and were estimated at a cut-off grade averaging 0.35 g/t Au. Mineral Reserves are based on a designed open pit, generated at a gold price of US$900 per oz. Scott Wilson RPA notes that mineralization extends considerably beyond the designed pit, as evidenced by the Mineral Resource estimate, and the mineralized wireframe.

- The reserve pit is slightly smaller than the optimum indicated by the reserve pit optimization, restricted by the size of the heap leach pad.

- Larger pits, and a longer mine life are certainly possible, given increased heap leach pad capacity, and higher gold prices or lower costs.

- Steeper bench face angles and overall steeper slopes in some design sectors of the pit are likely possible for a scenario including controlled blasting, however, some other design sectors have no potential increase in overall slope angle. The net effect is a limited benefit in the form of reduced strip ratio. In Scott Wilson RPA's opinion, the benefits are outweighed by the increased risks and costs attached to achieving steeper overall pit slopes.

- Further optimization of the waste haulage schedule may be possible, resulting in some small cost advantages.

- One metre depth of free-digging material over the footprint of the open pit was assumed, consisting of a portion of salvageable soil, and a portion of incompetent weathered rock. This assumption may be conservative. If greater quantities of free-digging material can be quantified, cost savings would be realized.

HEAP LEACHING

- Six potential heap leach pad sites were investigated, and the Ann Gulch pad location was assessed as the best, based on factors such as geotechnical conditions, favourable geometry for pad engineering, haulage distance from the pit, environmental considerations, and potential impacts on exploration targets.

- As currently designed, the pad capacity is limited to 67 Mt, based on a maximum thickness of 80 m, and maximum overall height of 180 m. It may be possible to increase this capacity up to approximately 80 Mt, by increasing the maximum heights, through introducing stabilizing measures to the design, or by carrying out testwork to confirm favourable material properties.

- Stacking operations are scheduled for 250 days per year, with a switch to winter stockpiling from early November to end of February, to avoid creation of permanent ice lenses within the heap. Natural weather variations and other uncertainties make it difficult to calculate the correct number of days with much precision. It is possible that stacking operations will average more days per year, with a corresponding improvement in Project economics.

- Preliminary agglomeration tests indicate that a minor amount of cement may be required in the lower lifts of the multi-lift heap leach operation. Additional testwork is required, but up to 2 kg/t of cement may be required during the first few years of operation. The pad location requires that the Dublin Gulch waterway be diverted from its current course, to accommodate the heap leach embankment that stabilizes the toe of the heap leach pad.

- A second pad location could be added in future studies, to accommodate a larger pit.

METALLURGICAL TESTWORK AND PROCESS DESIGN

- There is a distinct increase in gold recovery with decreasing crush size for all ore types. There is also an indication that High Pressure Grinding Roll (HPGR) crushing results in an increase in gold recovery, compared to conventional crushing to the same P_{80} crush size. The data set is not complete, however, there is an apparent two to six percentage point increase in gold recovery between conventional and HPGR crushing, based on comparison of data at the P_{80} crush sizes of 10 mm and 5 mm. Additional testing is required, but the potential for an increase in gold recovery by the use of HPGR crushers led to the decision to utilize these crushers for the PFS.

- Past testwork on gravity-flotation processes resulted in gold recoveries averaging 95%, at sizes that require grinding, however, capital and operating cost increases of a milling scenario were found to be in excess of the additional revenue from better recovery. Currently, testwork is underway on evaluating the addition of a gravity circuit to the PFS base case processing scenario, at a P_{80} of 5 mm. A gravity circuit would have the advantage of earlier gold production, however, KCA notes that cold winter ore would present operational difficulties, and gravity circuits may be best run seasonally.

- Cyanide neutralization and detoxification testing was completed on coarser-crushed column leach tailings and barren solutions in 1996 and 1997. No neutralization data are available at a 5 mm crush size. Additional testing will be required, and is currently underway.

INFRASTRUCTURE

- Power for the Project is assumed to be available from the Yukon Energy transmission grid by 2013. Yukon Energy is currently in the process of upgrading power generation capacity at Mayo (the Mayo Hydro Enhancement Project), and connecting the north and south Yukon transmission grids (the Carmacks-Stewart Transmission Project Stage 2). Funding agreements specify that both projects must be completed by March 31, 2012, however, Yukon Energy reports that both projects are on schedule to finish in 2011.

ENVIRONMENT

- In Scott Wilson RPA's opinion, environmental considerations are typical of open pit, heap leach operations, and are being addressed in a manner that is reasonable and appropriate for the stage of the Project.

RECOMMENDATIONS

Scott Wilson RPA and KCA recommend that Victoria Gold advance the Eagle Gold Project to the Feasibility Study stage. A Project Proposal (currently under preparation) should be submitted to the Yukon Environmental and Socio-Economic Assessment Board (YESAB) for Project permitting. Specific recommendations by area are as follows.

GEOLOGY AND MINERAL RESOURCES

Scott Wilson RPA makes the following recommendations:

- Additional bulk density measurements should be taken from intact core specimens. Where the rock is porous, the specimens should be sealed with wax or plastic wrap in order to prevent overestimation due to the porosity.

- Assay QA/QC results should be monitored and analyzed as soon as they are received, in order to allow for corrective actions, where required. Control sample failures should result in request for reassay of several samples ahead of and behind the failure. In extreme cases, entire assay batches should be rerun.

- A selection of drill holes should be relogged for alteration type and intensity. This relogging should be carried out with the intent of providing a consistent evaluation of the alteration across the deposit. The alteration data should then be used to create wireframes of the more intensely altered zones (if possible) and to update the ore-type classification in the block model.

- Additional drilling should be carried out to both expand the resource inventory and sterilize those areas that are proposed for key components of the mine operation.

MINING AND HEAP LEACHING

Scott Wilson RPA makes the following recommendations:

- Investigate scenarios with larger pits, either incrementally, by increasing the capacity of the Ann Gulch heap leach pad, or in a larger step, by adding a second heap leach pad. Incremental increases can likely be accommodated at the US$900 per ounce Mineral Reserve gold price, while larger increases are likely to require higher gold prices.

- Complete further optimization of the waste haulage schedule.

- Develop model surfaces for the soil/rock contact and the boundary at which drilling and blasting becomes necessary for excavation. Existing test pit and drill hole data should provide some information, which may need to be supplemented in the field.

- Carry out wet and dry slump tests on 5 mm crush size material.

- Complete agglomeration testwork.

- Further optimization of the Dublin Gulch waterway diversion to minimize environmental impact.

METALLURGICAL TESTWORK AND PROCESS DESIGN

KCA makes the following recommendations:

- Complete testwork for Feasibility-level design of HPGR crushing at a P_{80} of 5 mm.

- Investigate scenarios for addition of a gravity recovery circuit, based on testwork currently in progress.

- Complete testwork for cyanide neutralization and detoxification at a 5 mm crush size.

Victoria Gold provided the following budget for gathering information in the 2010 field season and completing a Feasibility Study (Table 1-1). In the opinion of Scott Wilson RPA and KCA, this budget is reasonable and appropriate for advancing the Project.

TABLE 1-1 FEASIBILITY STUDY BUDGET
Victoria Gold Corp. – Eagle Gold Project

Item	Cost (C$'000s)
Exploration	750
Condemnation & Geotechnical Drilling	1,000
Metallurgical Testwork	200
Environmental Fieldwork	500
Engineering	2,500
Camp Operations & Civil Testing	1,500
Total	**6,450**

ECONOMIC ANALYSIS

A pre-tax Cash Flow Projection has been generated from the Life of Mine production schedule and capital and operating cost estimates, and is summarized in Table 1-2. A summary of the key criteria is provided below.

ECONOMIC CRITERIA

Production

- Mineral Reserves of 66.1 Mt, at a grade of 0.82 g/t Au
- Waste mining of 68.5 Mt, for a total stripping ratio of 1.04:1, including 2.8 Mt of low-grade stockpile material
- Pre-production period of 20 months (January 2012 to August 2013)
- Mine life of eight years
- Open pit production of 9.1 Mtpa ore, 26,000 tpd
- Crushing to -5 mm year-round, heap leach pad loading 250 days per year, winter stockpiling

Revenue

- Leach Plant recovery by material type, as indicated by testwork, averaging 71.7%.
- Gold production schedule reflecting leach time vs. recovery in testwork.

- Silver credit, based on testwork showing doré bars contain 20% silver, 80% gold.
- Exchange rate C$1.00 = US$0.90
- Metal prices: US$900 per ounce gold.
  US$13.00 per ounce silver
- Net Smelter Return includes doré refining, and 1% NSR royalty.
- Revenue is recognized at the time of production.

Costs

- Pre-production capital cost of C$281 million in 2012 and 2013, including a contingency of C$38 million.
- Sustaining capital costs of C$55 million.
- Mine life capital totals C$337 million.
- Average operating costs over the mine life:

Mining	C$1.88 per tonne moved, or
	C$3.84 per tonne processed
Processing	C$5.05 per tonne processed
G&A	C$1.50 per tonne processed
Total	C$10.38 per tonne processed

TABLE 1-2 PRE-TAX CASH FLOW SUMMARY
Victoria Gold Corp. - Eagle Gold Project

	Units	Input	Year: Total/Avg.	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Physicals													
Mine Operating Days	days			200	308	365	365	365	365	365	365	308	0
Stacking Operating Days	days			0	116	263	263	263	263	263	263	263	0
Leach Operating Days	days			0	168	365	365	365	365	365	365	365	120
Annual Ore Tonnage for Leachi	'000 tonnes		66,141		2,000	9,100	9,100	9,100	9,100	9,100	9,100	9,541	0
Head Grade	g/t Au		0.823		1.019	0.863	0.983	0.855	0.861	0.753	0.731	0.682	0.000
Annual ROM Ore Tonnage	'000 tonnes		66,141		3,300	9,100	9,100	9,100	9,100	9,100	9,100	8,241	0
Head Grade	g/t Au		0.823		1.019	0.837	1.007	0.829	0.867	0.734	0.731	0.674	0.000
Low-Grade Stockpile Material	'000 tonnes		2,858		190	1,029	483	623	529	4	0	0	0
Head Grade	g/t Au		0.387		0.388	0.385	0.380	0.399	0.382	0.371	0.000	0.000	0.000
Waste Tonnes moved year	'000 tonnes		65,616	587	6,848	5,647	13,739	8,752	12,885	9,476	3,978	3,704	0
Total Material moved per year	'000 tonnes		134,615	587	10,338	15,777	23,322	18,475	22,514	18,579	13,078	11,944	0
	tpd			2,933	33,564	43,224	63,896	50,618	61,683	50,902	35,831	38,781	0
Strip Ratio			1.04		2.13	0.73	1.56	1.03	1.47	1.04	0.44	0.45	0.00
Contained Gold	ounces		1,750,957		65,523	252,459	287,482	250,088	251,998	220,363	213,887	209,156	0
Average Recovery	%		71.7%		74%	72%	73%	71%	72%	71%	70%	71%	0%
Recovered Gold	ounces		1,253,892		39,714	178,868	209,262	179,329	180,206	157,387	149,920	132,001	27,205
Revenue													
Gold Price	US$/oz Au	900	$900		900	900	900	900	900	900	900	900	900
Silver Price	US$/oz Ag	13.00	$13		13	13	13	13	13	13	13	13	13
Exchange Rate	US$=C$1.00	0.90	0.90		0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Gold Revenue	C$ '000		$1,253,892		39,714	178,868	209,262	179,329	180,206	157,387	149,920	132,001	27,205
Silver Revenue	C$ '000		$4,528		143	646	756	648	651	568	541	477	98
Gross Revenue	C$ '000		$1,258,420		39,857	179,514	210,018	179,976	180,857	157,955	150,461	132,478	27,304
Refining cost - per oz Gold	C$ '000	5.00	$6,269		199	894	1,046	897	901	787	750	660	136
Refining cost - per oz Silver	C$ '000	0.40	$125		4	18	21	18	18	16	15	13	3
NSR Royalty	C$ '000	1%	$12,522		397	1,786	2,090	1,791	1,800	1,572	1,497	1,318	272
Net Revenue	C$ '000		$1,239,504		39,258	176,816	206,861	177,271	178,138	155,581	148,200	130,487	26,893

TABLE 1-2 PRE-TAX CASH FLOW SUMMARY
Victoria Gold Corp. - Eagle Gold Project

	Units	Input	Year: Total/Avg.	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Operating Costs													
Mining	C$/t moved		1.88		1.90	2.07	1.68	1.95	1.69	1.87	2.11	2.13	
Mining	C$/t ore		3.84		9.81	3.59	4.31	3.97	4.19	3.81	3.04	2.67	
Processing	C$/t		5.05		7.63	4.92	4.92	4.92	4.92	4.92	4.92	4.59	
G&A	C$/t		1.50		3.21	1.41	1.43	1.43	1.43	1.42	1.39	1.33	
Total Operating Costs	**C$/t**		**10.38**		**20.66**	**9.91**	**10.65**	**10.31**	**10.54**	**10.15**	**9.35**	**8.60**	
Total Mining	C$ '000		$253,659		$19,627	$32,632	$39,186	$36,103	$38,159	$34,687	$27,659	$25,459	$146
Processing	C$ '000		$333,827		$15,264	$44,738	$44,738	$44,738	$44,738	$44,738	$44,738	$43,817	$6,317
G&A	C$ '000		$99,213		$6,425	$12,814	$13,015	$12,995	$13,025	$12,915	$12,644	$12,736	$2,644
Total Operating Costs	**C$ '000**		**$686,698**		**$41,316**	**$90,184**	**$96,940**	**$93,836**	**$95,922**	**$92,340**	**$85,041**	**$82,012**	**$9,106**
Unit Operating Cost	US$/oz Au		$493		936	454	417	471	479	528	511	559	301
Operating Cashflow	**C$ '000**		**$552,805**		**($2,058)**	**86,631**	**109,921**	**83,435**	**82,216**	**63,241**	**63,158**	**48,474**	**17,787**
Capital Costs													
Mining	C$ '000		$40,669	$6,790	$28,160	$657	$2,408	$304	$1,602	$412	$209	$126	$0
Crushing & Conveying	C$ '000		$73,134	$49,447	$18,562	$2,832	$131	$1,374	$263	$263	$263	$0	$0
Heap Leach Facility	C$ '000		$51,005	$14,760	$11,498	$2,511	$11,741	$0	$10,495	$0	$0	$0	$0
Process Plant	C$ '000		$17,252	$11,279	$4,797	$0	$0	$991	$0	$186	$0	$0	$0
Infrastructure	C$ '000		$45,509	$39,323	$3,247	$0	$2,939	$0	$0	$0	$0	$0	$0
Indirects	C$ '000		$55,306	$27,779	$27,527								
Contingency	C$ '000		$38,214	$25,762	$12,452								
Closure and Reclamation	C$ '000		$15,836	$0	$105	$105	$105	$105	$105	$105	$105	$105	$15,000
Total Capital Cost	**C$ '000**		**$336,925**	**$175,139**	**$106,347**	**$6,105**	**$17,325**	**$2,774**	**$12,464**	**$965**	**$576**	**$231**	**$15,000**
Cashflow													
Net Pre-Tax Cashflow	**US$ '000**		**$215,880**	**($175,139)**	**($108,405)**	**$80,527**	**$92,596**	**$80,661**	**$69,751**	**$62,276**	**$62,582**	**$48,244**	**$2,787**
Cumulative Pre-Tax Cashflow	US$ '000			($175,139)	($283,544)	($203,017)	($110,421)	($29,760)	$39,991	$102,267	$164,849	$213,093	$215,880
Payback	Years		3.4			1.0	1.0	1.0	0.4	0.0	0.0	0.0	0.0
Total Cash Cost	US$/oz		503		947	464	427	481	489	538	521	569	312
Capital Cost	US$/oz		242										
Total Production Cost	US$/oz		745										
Economics													
IRR	15.02%												
Pre-tax NPV at 5%	$115,291												
Pre-tax NPV at 7.5%	**$77,954**												
Pre-tax NPV at 10%	$47,081												
Pre-tax NPV at 15%	$156												

CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted pre-tax cash flow totals $216 million over the mine life, and simple payback occurs near the mid-point of 2017 (3.4 years).

The Total Cash Cost is US$503 per ounce of gold. The mine life capital unit cost is US$242 per ounce, for a Total Production Cost of US$745 per ounce of gold. Average annual gold production during operation is 170,000 ounces per year.

The pre-tax Internal Rate of Return (IRR) is 15%, and the pre-tax Net Present Values (NPV) at various discount rates are:

- 0% - $216 million
- 5% - $115 million
- 7.5% - $78 million
- 10% - $47 million

SENSITIVITY ANALYSIS

Key economic risks were examined by running cash flow sensitivities:

- Gold price
- Exchange rate
- Head grade
- Operating costs
- Pre-production capital costs
- Recovery

Sensitivity over the base case has been calculated for -20% to +20% variations to the key parameters, with the exception of recovery, which was tested over a range of 68% to 76%, in 2% increments. The sensitivities are shown in Figure 1-1 and Table 1-3.

The Project is most sensitive to gold price and head grade (which have the same impact), followed by exchange rate.

FIGURE 1-1 SENSITIVITY ANALYSIS

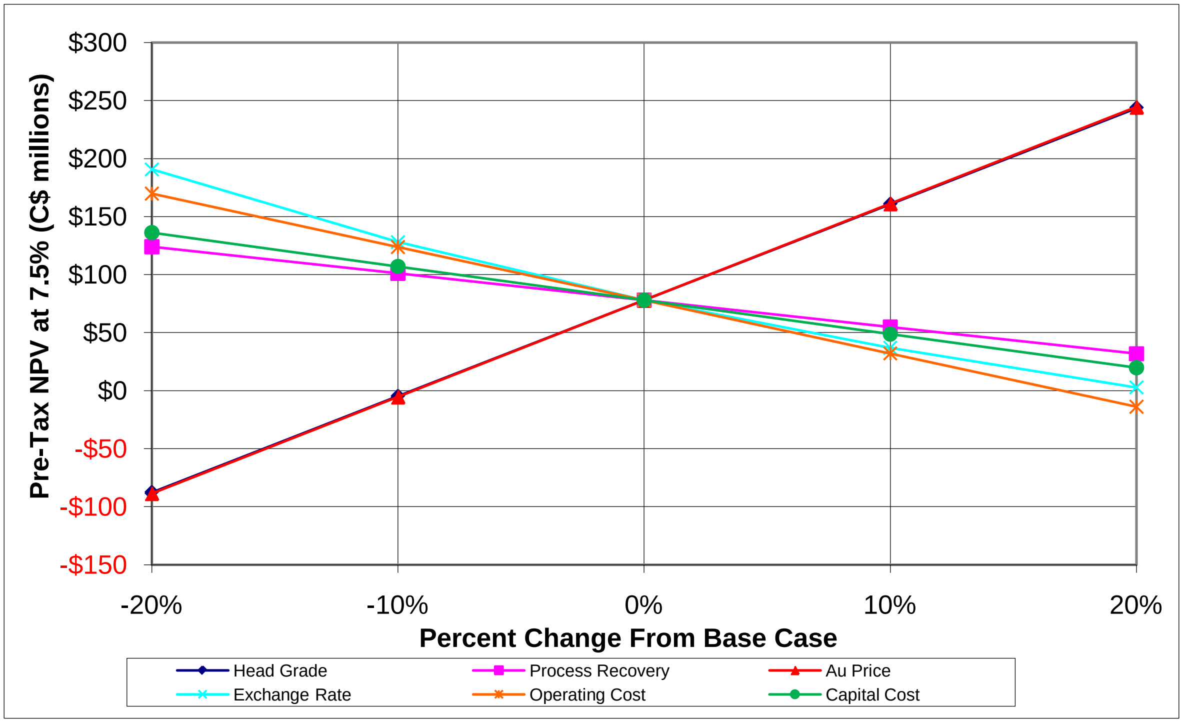


TABLE 1-3 SENSITIVITY ANALYSES
Victoria Gold Corp. – Eagle Gold Project

Parameter Variables	Units	-20%	-10%	Base	+10%	+20%
Gold Price	US$/oz	720	810	900	990	1080
Exchange Rate	US$/C$	0.72	0.81	0.90	0.99	1.08
Head Grade	g/t Au	0.66	0.74	0.82	0.91	0.99
Operating Cost	$/t	8.31	9.34	10.38	11.42	12.46
Capital Cost	$ millions	225	253	281	310	338
Recovery	%	68	70	72	74	76

NPV@7.5%	Units	-20%	-10%	Base	+10%	+20%
Gold Price	$ millions	(89)	(5)	78	161	244
Exchange Rate	$ millions	191	128	78	37	3
Head Grade	$ millions	(88)	(5)	78	161	244
Operating Cost	$ millions	170	124	78	32	(14)
Capital Cost	$ millions	136	107	78	49	20
Recovery	$ millions	32	55	78	101	124

TECHNICAL SUMMARY

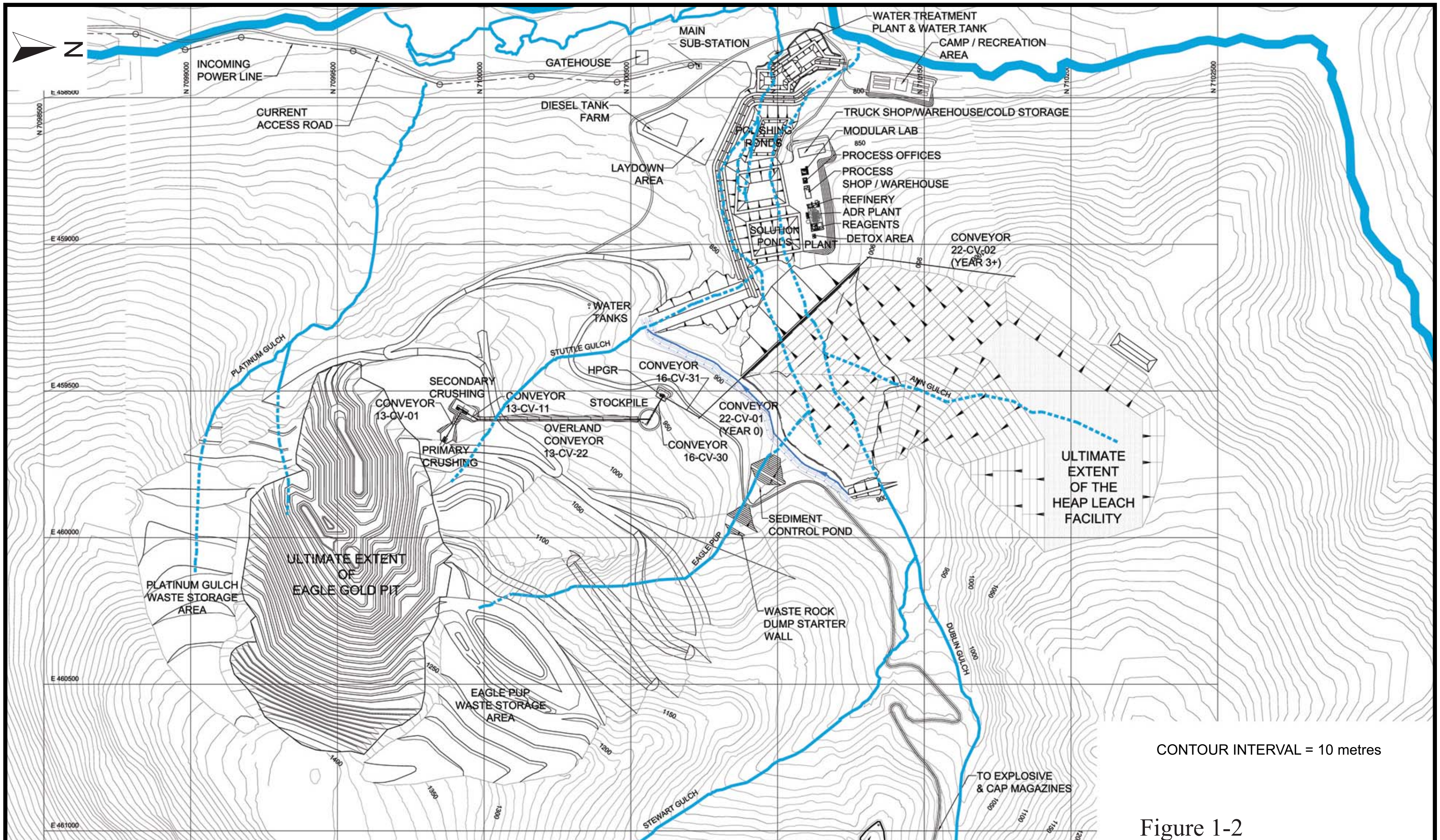
The base case PFS operating scenario includes open pit mining, three-stage crushing to a P_{80} of 5 mm, and heap leaching, followed by carbon Adsorption, Desorption, and Refining (ADR) gold recovery. A general arrangement of the site facilities, showing the ultimate extents of both pit and heap leach pad, is presented in Figure 1-2.

PROPERTY DESCRIPTION AND LOCATION

The Eagle Gold Project consists of 1,896 quartz claims, 10 quartz leases, and one Federal Crown Grant quartz claim totalling 34,576 ha located in the Mayo Mining District of central Yukon and centred approximately at 453,750mE, 7,100,950mN (NAD 83, Zone 8). The geographic coordinates are approximately 64°02'N and 135°45'W. The property is located within 1:50,000 scale NTS map sheets 105M/13, 14, 106D/3, 4, 115P/16, and 116A/1.

LAND TENURE

The Eagle Gold Project claims and leases are currently recorded in the name of StrataGold Corporation (StrataGold) and are valid until their respective renewal dates from 2010 to 2027. Mineral claims in Yukon can be maintained in good standing by performing approved exploration work, or making payments in lieu of work, of $105 per claim. By virtue of two underlying agreements covering portions of the property, Victoria Gold makes annual payments totalling $35,000. Exploration work is subject to the Mining Land Use Regulations of the Yukon Quartz Mining Act, which requires permits to be issued prior to the commencement of significant exploration programs.



0 250 500 750 1000

Metres

Figure 1-2

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Site Plan

April 2010

Source: Scott Wilson, KCA, 2010.

www.scottwilson.com

By virtue of various underlying agreements, portions of the Dublin Gulch Property are subject to the following royalties:

- with respect to a portion of the property historically known as the Mar Gold Zone, an annual royalty payment of $20,000 or payments of 2% of gross returns received from the sale of all metals produced from the claims, whichever is greater, to a maximum of $1,000,000 (of which $400,000 has been paid to date), after which the royalty reverts to 1% of the said gross returns;

- with respect to 36 claims on the Lynx Zone, a 1% net smelter return (NSR) royalty with annual advance royalty payments of $15,000; and

- with respect to the 63 claims and leases known as the Mar Tungsten Property and Mar Tungsten Leases, a 1% NSR royalty.

The Eagle Zone Mineral Resources are located on claims which are subject only to the Mar Gold royalty.

INFRASTRUCTURE

Victoria Gold maintains a 30-man exploration camp on the property. There is no other permanent infrastructure on the property.

HISTORY

Exploration and placer mining began on the Dublin Gulch property in 1895 with the first gold reported in 1898. In 1904, tungsten was identified in placer concentrates. The Geological Survey of Canada (GSC) discovered in situ scheelite in Dublin Gulch in 1916.

Exploration has been ongoing in the area since 1970, with gold and tungsten as the commodities of interest. Documentation of placer mining began in 1978 and since that time about 110,000 ounces of placer gold have been recovered. In 1971, Canex-Placer Limited drilled three holes and dug 20 trenches to test quartz-scheelite vein systems. Placer mining of gravels began in 1973.

In 1977, Queenstake Resources Ltd. (Queenstake) staked the Mar claims in the Ray Gulch area to cover tungsten-bearing skarn. This property was optioned to Canada Tungsten Mining Corp. (Canada Tungsten) that explored for gold and tungsten between 1977 and 1986. Canada Tungsten drilled 65 NQ diamond drill holes (DDH) for an aggregate length of 11,315 m in the tungsten skarn zone. In addition, 100 trenches were excavated, mapped and sampled and two new gold bearing shears were exposed by placer mining operations. These programs lead to the discovery of the Eagle zone three kilometres southeast of the original tungsten occurrences.

In 1991, the property was acquired by Ivanhoe Goldfields Ltd. (Ivanhoe) who utilized a “Fort Knox-type” intrusive-hosted gold exploration model. Ivanhoe entered into a joint venture with Amax Gold Inc. (Amax). In 1991-1992, Ivanhoe and Amax conducted trenching, soil sampling, diamond and reverse circulation (RC) drilling, and property evaluation. In 1992, Amax did not renew the option on the property and in 1993 Ivanhoe completed geophysical surveys, diamond and RC drilling, and baseline environmental monitoring.

In 2004, StrataGold acquired the Dublin Gulch and Clear Creek properties and completed regional magnetometer/electromagnetic airborne geophysical surveys to assist in exploration targeting and lithological interpretations.

In 2005, 34 HQ diamond drill holes were completed for a total of 8,105 m to test the western extent of the resource area.
The 2006 exploration program produced 10 HQ diamond drill holes for a total of 4,282 m.

Drilling in 2007 consisted of 20 holes for a total of 5,627 m designed to increase the confidence in the resource model and to test the western extent of the resource.

In 2008, 15 DDH were drilled totalling 4,249 m to test the extents of the mineralized zone.

In 2009, Wardrop produced a NI 43-101 compliant Mineral Resource estimate. The geological cut-off for interpretation was based on approximately 0.20 g/t Au. Wardrop estimated Indicated Resources of 98.5 Mt grading 0.849 g/t Au and Inferred Resources of 2.0 Mt grading 0.671 g/t Au at a 0.50 g/t Au cut-off.

GEOLOGY

The Eagle Gold Project is located in the northern portion of the Selwyn Basin, a fault controlled epicratonic basin underlain by four metasedimentary units. These units are, from oldest to youngest, the Proterozoic Hyland Group, the Paleozoic Upper Schist, and the Mesozoic Lower Schist and Keno Hill Quartzites.

The Hyland Group rocks have been juxtaposed by Cretaceous-aged laterally extensive, northward-directed thrust sheets. There are three principal thrust sheets, from east to west, the Dawson, Tombstone and Robert Service. The Robert Service thrust is proximal to the Project and superimposes older Hyland Group onto the Keno Hill Quartzite. The Tombstone and Robert Service thrusts are considered coeval and the hangingwall of the Tombstone thrust is a zone of regional scale, high deformed rocks known as the Tombstone High Strain Zone (THSZ). The THSZ was later subjected to folding by the west plunging McQuesten Antiform which lies south of the Project.

Four generations of deformation are documented, but prominent structures were limited to the initial two events. The first resulted in widespread development of foliation that was subsequently deformed by gentle, regional-scale folding. The second deformation event resulted in east trending, south plunging anticlines in the Project area, the most prominent of which being the Lynx Creek Anticline whose hinge is interpreted to pass through the Eagle Project.

Three granitoid intrusions were emplaced during the Cretaceous. These are the Selwyn Suite, dated between 104 and 98 Ma, the Tombstone Suite, between 94 and 92 Ma, and the McQuesten Suite, dated at 64 Ma. The intrusives were most commonly emplaced within the Hyland Group and Upper Schist rocks. The Tombstone Suite contains the five kilometre by two kilometre Dublin Gulch stock and is the primary host of vein, shear and skarn related mineralization.

The Tombstone Suite forms part of the Tombstone Gold Belt, which is in the eastern portion of the Tintina Gold Province. The western portion of the Tintina Gold Province has been dextrally displaced approximately 450 km by the Tintina fault.

At least four periods of faulting have occurred in the Project area including two low angle thrust and bedding plane faults, northwest trending faults, northeast trending faults, east trending faults, and north-south trending faults. It is the latter that may have the most significant impact on mineralization which appear to displace the Dublin Gulch stock.

The Project lies on the northern limb of the McQuestin Antiform and is underlain by Proterozoic to Lower Cambrian age Hyland Group metasedimentary rocks and the intrusive Dublin Gulch granodioritic stock.

The Eagle gold occurrence is located at the western limit of the Dublin Gulch stock at its narrowest extent. The intrusive-metasedimentary contact is both steep and cross-cutting the host rock foliation and shallow dipping to the southwest and paralleling the metasediment foliation. The host rock is comprised of two basic units: partially feldspathic, weak to moderately folded quartzites and well foliated phyllites that are compact aggregates of biotite and sericite. The granodiorite intrusive varies from relatively unaltered to strongly sericite ± hematite altered (weathered).

The zone is characterized by sub-parallel extensional quartz veins that are best developed within the intrusive proximal to both hangingwall and footwall contacts with the host metasediment. The veins strike from 060° to 085°, dip at approximately 60° to the south and range in width from 0.01 cm to 10 cm. Contacts are generally sharp and the veining density ranges between less than one vein per metre to more than 15 veins per metre. Maximum vein density coincides with the local apex of the intrusion and the tightest constriction between footwall and hangingwall contacts. Embayments and constrictions of the stock represent stress shadows that constitute favourable area for the formation of extensional quartz veins. Protrusions of the stock create favourable areas for the development of extensional shear veining in the metasedimentary host.

The veins are comprised of white to grey quartz with subordinate potassium feldspar. Sulphides occur throughout but account for less than 5% of vein material. Most common sulphides are pyrrhotite, pyrite, scheelite, arsenopyrite, sphalerite, bismuthinite, and galena. Gold occurs in veins as native gold liberated in gangue or associated with bismuth minerals and with an average grain size of 110-160 microns.

Alteration envelopes consist of secondary potassium feldspar. Sericite carbonate alteration also exists as narrow vein selvages and independently from veining.

MINERAL RESOURCES AND MINERAL RESERVES

The most recent Mineral Resource estimate was prepared by Wardrop in 2009. Scott Wilson RPA audited the 2009 Wardrop estimate in support of this study and, following the 2009 drilling program, updated the estimate. The current resource model is summarized in Table 1-4 at a range of cut-off grades. Scott Wilson RPA has selected 0.21 g/t Au as the most appropriate for reporting of Mineral Resources.

TABLE 1-4 INDICATED MINERAL RESOURCE ESTIMATE
Victoria Gold Corp. – Eagle Gold Project

Cut-Off (g/t Au)	Tonnage Mt	Grade (g/t Au)	Contained Gold (M oz Au)
0.50	89.2	0.86	2.45
0.30	138.0	0.69	3.08
0.25	147.7	0.67	3.16
0.24	149.3	0.66	3.18
0.23	151.0	0.66	3.19
0.22	152.2	0.65	3.20
0.21	**153.4**	**0.65**	**3.21**
0.20	154.3	0.65	3.21
0.19	155.2	0.65	3.22
0.18	155.9	0.64	3.22
0.15	157.4	0.64	3.23
0.10	158.0	0.64	3.23

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at an average pit discard cut-off grade of 0.21 g/t Au. This cut-off does not include mining costs, and is only valid within an optimized pit shell.
3. Mineral Resources are estimated using an average long-term gold price of US$1,050 per ounce, and a US$/C$ exchange rate of 0.90:1.00
4. Mineral Resources were constrained within an optimized pit shell.
5. A minimum mining width of three metres was used.
6. Indicated Mineral Resources are inclusive of Mineral Reserves.

The estimate was prepared using a block model constrained by 3D wireframes. Grade was interpolated into the blocks using Ordinary Kriging (OK). Samples were capped at

12.65 g/t Au prior to compositing. A pit shell was used to constrain the estimate, and demonstrate a potential for economic viability. Significant additional quantities of mineralization are defined at depth in the block model, below the pit shell constraint. All Mineral Resources reported in Table 1-4 are classified as Indicated. The model was constructed using GEMS (Gemcom) software, which is a commercial off-the-shelf package commonly used in the industry.

Mineral Reserves, as per the mine production schedule, are reported in Table 1-5. Mineral Reserves are based on Indicated Mineral Resources only, as there are no Measured Resources in the model, and Inferred Resources are too geologically speculative to be used as a basis for Mineral Reserves. There are no Inferred Resources within the final pit limits.

TABLE 1-5 PROBABLE MINERAL RESERVES
Victoria Gold Corp. – Eagle Gold Project

Ore Type	kTonnes	Au (g/t)	Au (Oz)	% Tonnes	% Au Oz
Type A	22,789	0.88	645,600	35	37
Type B	36,442	0.79	930,100	55	53
Type C	6,909	0.79	175,300	10	10
Total	**66,141**	**0.82**	**1,751,000**		

Notes:
1. CIM definitions were followed for Mineral Reserves.
2. Mineral Reserves are estimated at cut-off grades by pit phase and by material type, averaging 0.35 g/t Au.
3. Mineral Reserves are estimated using an average long-term gold price of US$900 per ounce and a US$/C$ exchange rate of 0.90.
4. A minimum mining width of 15 metres was used.
5. Bulk density is 2.66 t/m^3.
6. 98.0% mining recovery and 2.0% mining dilution at 0.2 g/t Au applied to Mineral Reserves.

MINING OPERATIONS

OPEN PIT OPTIMIZATION

The Mineral Reserve pit design is based on an open pit optimization, run using the following inputs:

- Pit slopes, Sediments: 35.5º to 44º
- Pit slopes, Intrusives: 32.5º to 43.5º
- Pit slopes, top 35 m of pit: 27º
- Mining recovery: 98.0%

- Mining dilution: 2.0% at 0.2 g/t Au
- Reference mining cost: US$1.55/tonne moved
- Ore mining cost adjustment factor: 1.35
- Process cost: US$4.70/tonne processed
- G&A cost: US$1.95/tonne processed
- Process recovery,
    - Type A – Oxidized: 77.0%
    - Type B – Unaltered: 68.0%
    - Type C – Sericitic: 72.0%
- Gold Price: US$900/oz Au
- Royalty: 1.0%
- Gold selling cost: US$5.00/oz Au

Note: small differences between these inputs and final cash flow inputs reflect later refinements.

Pit slope design criteria were developed by BGC Engineering Ltd. (BGC), based on historical information and four new oriented-core geotechnical drill holes completed in 2009. A global bench face angle of 65º was proposed, and overall slope angles for various design sectors by rock type domain were estimated.

A reference mining cost of US$1.55 per tonne moved is applied to waste rock, with a cost adjustment factor of 1.35 applied to ore; therefore, the cost to mine one tonne of ore is US$2.09.

Incremental mining costs are calculated for both ore and waste rock based on reference elevations, with additional costs for extra haulage incurred for material above or below the reference.

Two major rock types (Intrusive and Sediments) are identified in the model with one of three alteration types each: Type A, Type B, or Type C alteration. Process recovery is dependent on the alteration type.

A base case gold selling price of US$900 per ounce is used in the open pit optimization as revenue factor 1.0. A series of pit shells were generated at various gold revenue factors from 0.3 to 1.3 (gold prices ranging from US$270 per ounce to US$1,170 per ounce) for further design work and mine scheduling. A 1% royalty on the gold selling price is applied to all gold production from the Project.

Pit Shell 52, revenue factor 0.87 (gold price of US$783 per ounce), was selected as the optimum pit shell for use as a guide for the final design pit; larger pits were rejected because of material quantity constraints of the receiving heap leach pad, which is designed for a maximum of approximately 65 million tonnes of ore.

The Whittle reported results for Pit Shell 52 are 63.8 million tonnes of mineralized material above applied cut-off grades by alteration type.

OPEN PIT DESIGN

The final design pit limits are located on a mountainside with an average slope of 3 horizontal to 1 vertical, with the long axis of the pit striking east-west, roughly parallel to the fall-line of the slope. The footprint of the final pit is just under 70 ha. The west-facing final highwall of the pit has the greatest vertical presence at 410 metres to the intermediate pit bottom with a top elevation of 1,410 masl, followed by a 70-metre step down to the final pit bottom at 930 masl. The first bench containing ore is at the 1,260 masl elevation.

Bench geometry for pit design is based on the design criteria supplied by BGC, used in the pit optimization. Haul road width is based on 91-tonne payload class haul trucks. For the final pit, no haul roads are left in the highwalls. However, during pit phase construction, haul roads exist both internal and external of the pit. Road design gradients are typically between 8% and 10%, however, segments up to 200 m with gradients up to 12% may exist in various pit phases and on external roads. The bottom of the final pit is accessed via a sinking cut ramp with ramp retrieval in the last half bench to maximize mining recovery of ore.

OPEN PIT PHASES

A starter pit and three pushbacks to the final pit limits were selected to maximize the Project’s net present value at a mine production rate of 3.3 Mt ore in the first year (2013) followed by 9.1 Mt ore thereafter. The daily mining rate is 26,000 tonnes per day ore.

In selecting pit shell pushbacks, a minimum mining width of 50 m was considered. Due to the orientation and geometry of the model, a typical mining bench plan is shaped like the letter "U", with pinching and swelling, and intersects surface allowing access to all safety berms.

During the construction period in years 2012 and 2013, a waste rock only pre-strip is developed at the highest elevations of the Phase 1 starter pit and the Phase 2 pit. This waste rock is used as fill material for site road development, specifically from the starter pit down to the run-of-mine stockpile area and the road loop which provides access to the heap leach facility embankment rockfill. To increase the speed of fill road development, a second waste rock only borrow pit is developed lower down the mountainside at the base of the Phase 3 pit. Once the required site road development is complete, additional waste rock material from the pre-strip area is excavated for the embankment rockfill.

A breakdown of Probable Mineral Reserves by mining phase is shown in Table 1-6.

TABLE 1-6 OPEN PIT PHASES
Victoria Gold Corp. – Eagle Gold Project

Phase	Ore kTonnes	Ore Au g/t	Stockpile kTonnes	Stockpile Au g/t	Waste kTonnes	Strip Ratio Waste+Sp:Ore
Phase 1	2,895	1.06	142	0.38	2,507	0.92
Phase 2	18,180	0.92	1,511	0.38	14,055	0.86
Phase 3	19,281	0.85	1,205	0.39	23,098	1.26
Phase 4	25,785	0.71	-	na	25,957	1.01
Total	**66,141**	**0.82**	**2,858**	**0.39**	**65,616**	**1.04**

OVERBURDEN

Overburden stripping for the purpose of mine cost estimating is defined as any material at surface not requiring a drill and blast cycle to break the material prior to excavation. No overburden surface is defined in the resource model, however, an estimate of overburden thickness is seen in the numerous exploration roads that have been built, with a trend of thickening overburden at lower elevations. A thickness of one metre of overburden is assumed over the entire pit footprint of approximately 70 ha. This is likely a conservative estimate.

STOCKPILES

Two stockpile areas are defined in the mine plan for lower-grade mineralized material mined in Phases 1, 2, and 3. Potential for additional stockpile capacity is available if warranted for segregation of grade groups and or alteration types.

Stockpiles are designed with a 35% swell factor, 36º face angles (angle of repose minus one degree), and a minimum 5 m wide terrace every 20 m of vertical. The majority of terraces, however, are 18 m wide or greater, to allow for greater flexibility during reclamation.

Stockpiled material during the life-of-mine is currently not processed in the Project cash flow model, due to capacity restrictions of the final heap leach facility, thus is not included as ore in the Mineral Reserve statement. This material is economic, however, it is very close to break-even, and does not have a significant impact on the cash flow if included.

ORE

The primary crusher dump pocket is located at 1065 masl, just over 100 m north of the final pit rim. During regular operations, run-of-mine ore is to be direct-dumped into the dump pocket, or a small run-of-mine stockpile if the primary crusher is not available for direct dumping.

WASTE DUMPS

Using a 35% swell factor for waste, life-of-mine waste production is 31.5 million loose cubic metres including overburden, with an additional 1.5 million loose cubic metres going to stockpile. During the life-of-mine plan, waste rock is scheduled to go to one of five areas:

- Fill material for haul road development,
- Fill material for the heap leach facility embankment rockfill,
- Platinum Gulch runaway lanes and waste rock storage facility,
- Eagle Pup waste rock storage facility, and
- Pit backfilling.

Over the life-of-mine, typical haulage profiles for waste rock average approximately 1,500 m with a gradient of between plus or minus 5%.

Testwork on potential for acid generation or metal leaching from waste rock is currently in progress. Initial results and past work indicate that the material is fairly inert, with neither strong acid-generating qualities, nor strong buffering capacity. The current mine plan assumes that no segregation of any particular material will be required, although all industry-standard options for managing fairly inert waste rock are possibilities, if needed. Provision for collection of all drainage water from both Eagle Pup and Platinum Gulch has been included, allowing for testing before direct discharge, treatment, or use as process water, as appropriate.

MINE PRODUCTION SCHEDULE

The mine production schedule is generated based on the Indicated Resources within the designated pit phases and final pit limits. The life-of-mine production schedule by year is shown in Table 1-7.

TABLE 1-7 LIFE OF MINE PRODUCTION SCHEDULE
Victoria Gold Corp. – Eagle Gold Project

Year	Ore kTonnes	Grade g/t Au	Stockpile kTonnes	Waste kTonnes	Strip Ratio (Waste+Stockpile:Ore)	Contained Au Oz
2012	0	0.00	0	587	NA	0
2013	3,300	1.02	190	6,848	2.13	108,100
2014	9,100	0.84	1,029	5,647	0.73	244,800
2015	9,100	1.01	483	13,739	1.56	294,600
2016	9,100	0.83	623	8,752	1.03	242,700
2017	9,100	0.87	529	12,885	1.47	253,600
2018	9,100	0.73	4	9,476	1.04	214,800
2019	9,100	0.73	0	3,978	0.44	213,700
2020	8,241	0.67	0	3,704	0.45	178,600
Total	**66,141**	**0.82**	**2,858**	**65,616**	**1.04**	**1,751,000**

MINE EQUIPMENT

The Project is a conventional open pit mining operation with two diesel hydraulic excavators as the primary loading units, loading off-highway, rigid-frame, mechanical-drive haulage trucks. A fleet of self-propelled, diesel, rotary down-the-hole hammer drills are selected for production rock drilling. The mining fleet will be owned and operated by the owner, starting in Year 1 (2013). A mine contractor is used during the construction period for development work.

Production drilling in mineralized material and waste is performed by a fleet of two identical, track-mounted, diesel-powered, rotary down-the-hole hammer drills that can drill a 7.5 m bench height (including subdrill requirement) in a single pass. A smaller drill is selected for use in pioneering new benches at surface and for use in final wall control preshear drilling.

The supply of explosives has been costed as a contractor-provided service for delivery of explosives to the drill hole. All explosives contractor services for delivery of explosives to the hole have been priced into the unit cost of explosive product.

The selected primary shovel and truck fleet consists of two Hitachi EX1900 diesel hydraulic excavators with 15 m^3 buckets, loading a fleet of Cat 777 91-tonne payload mechanical-drive haul trucks.

A wheel loader has been included, with the primary function of loading run-of-mine stockpiled ore into the primary crusher dump pocket as required.

Track dozers are included for multi-purpose work, with a D10 dozer primarily for pioneering benches and road construction, and a D8 dozer for waste dump development.

Motor graders are included primarily to maintain haulage roads and pit floors for smooth haulage operations. Secondary functions include maintaining the shovel loading area floor, drill and blast support, and snow removal.

A water truck has been specified for haul road dust suppression and will have a water monitor for fire-fighting as required.

MINERAL PROCESSING

The proposed process consists of three-stage crushing to a P_{80} of 5 mm, followed by heap leach gold extraction, and recovery of gold from solution in a carbon adsorption, desorption, and recovery (ADR) plant. Mine life average gold recovery, across all material types, is estimated to be 72%.

Run-of-mine ore will be delivered by haul trucks from the open pit to the primary gyratory crusher, located north of the pit. The ore will be direct-dumped into the dump hopper situated above the 1,372 mm to 1,651 mm (50 inch to 65 inch) gyratory crusher, and will be discharged to the crusher with a discharge setting of 150 mm. The primary crushed ore will be collected in the discharge pocket below the primary crusher. A belt feeder will regulate the discharge rate of the primary crushed ore, at nominally 1,350 dry tph, onto a 1,524 mm wide primary crushing discharge conveyor.

The primary crushing discharge conveyor will deliver the primary crushed ore to two 50 t surge bins. Belt feeders will regulate the ore feed rate from the surge bins at nominally 675 dry tph each, to two 2,438 mm x 7,315 mm (8 ft x 24 ft) double-deck vibrating screens (secondary screens), with apertures of 89 mm and 38 mm. Screen oversize material (nominally 559 dry t/h each) will discharge to two Metso MP1000 standard head cone crushers (secondary crushers), with closed side settings of 19 mm.

The combined secondary crusher discharge product and screen undersize will be transported by a 1,219 mm secondary crushing discharge conveyor at a nominal rate of 1350 dry tph to an overland conveyor. The 1,067 mm wide overland conveyor will be approximately 530 m long, with a vertical drop of approximately 29 m. The secondary crushed ore will be delivered to a conical stockpile with 10,000 tonnes live capacity.

Secondary crushed ore will be withdrawn from the stockpile at a controlled rate (nominally 1,550 dry tph) by vibrating pan feeders (2 operating) onto a 1,372 mm wide reclaim conveyor, which will discharge into a 20 t bin, located above the High Pressure Grinding Roll (HPGR). The feed rate to the bin will be controlled so that the HPGR is choke-fed. Product with a nominal P_{80} of 5 mm will be discharged onto a 1,219 mm wide HPGR discharge conveyor, and then conveyed to the leach pad at a nominal rate of 1,550 dry tph.

The proposed heap leach facility will be located approximately 1.2 km north of the Eagle Zone. The heap will be constructed during non-winter months using a 1,219 mm wide conveyor stacking system. Crushed ore will be stockpiled on the heap leach pad throughout the winter, while the insulating layer of ore is covering the heap leach. The on-heap crushed ore stockpile will be fed to the conveyor stacking system by a bulldozer (via mobile feeder), when stacking operations resume in warmer weather.

The nominal stacking rate will be 2,170 dry tph, that will consist of 620 dry tph reclaimed from the crushed ore stockpile and 1,550 dry tph from the crushing circuit. Raw water will also be added to the HPGR discharge material to assist with the agglomeration of fines in the ore.

The ore heaps will be leached using a dilute solution of NaCN, applied by a system of drip emitters. Sprinklers can also be utilized in the late spring and summer months when increased evaporation may be required to maintain the system water balance.

The recovery plant for the Eagle Gold Project is a carbon adsorption, desorption, and recovery (ADR) facility located west of the heap and north of the events ponds.

The following major components will be included in the ADR facility:

- Three trains of five carbon adsorption columns, with nominal flow of 650 m^3/h each and design flow capacity of 715 m^3/h each;
- A pressure strip system, consisting of two 3.0 t elution columns, heat exchangers, solution heater, solution storage tanks, and electrolytic cells. The system is capable of processing a maximum of 6.0 t of carbon per 24 hour day;
- A 3.0 t capacity acid wash circuit, consisting of a single acid wash vessel, acid mix tank, and circulation pump;
- A tilting crucible-type diesel-fired doré furnace and baghouse;
- A carbon regeneration system, including a 2.4 tpd capacity horizontal rotary kiln;
- A carbon handling circuit, consisting of transfer pumps, vibrating screens, storage tanks, and a carbon fines filter;
- A mixing system, consisting of an agitated mix tank and storage tank for make-up and addition of NaCN to the process.

ENVIRONMENTAL CONSIDERATIONS

The Yukon is considered to have a generally favourable regulatory environment for mining activity, in part due to completion of First Nation land claim agreements, and in part to the devolution of regulatory responsibilities related to mining activity to the Yukon Government. Environmental and socio-economic assessment of development activities now occurs under the Yukon Environmental and Socio-economic Assessment Act (YESAA), which provides for a single assessment process.

The Eagle Gold Project currently has an approved, Class III, Operating Plan under the Yukon Quartz Mining Act for its advanced exploration activities at the site as well as its camp operations. The current Operating Plan is in effect until 2012.

The regulatory approval process for major hard rock mines in the Yukon occurs in two stages: 1) an assessment under the YESAA, and 2) the receipt of territorial and/or federal permits, authorizations, and licences. Consultation with First Nations is a formal requirement under the YESAA, with the underlying objective to ensure First Nation interests are considered and attempts are made to address and / or accommodate issues and concerns with respect to a proposed project. Consultation with the First Nation of Na-cho Nyak Dun and local communities is required under YESAA in developing the Eagle Gold Project Proposal.

A large amount of environmental baseline information has been collected throughout the Project area to compile the data set required to support the submission of a Project Proposal under the YESAA. Past studies combined with work recently completed has produced a comprehensive environmental baseline database that will be used to assess the existing environmental resources and conditions potentially affected by the Project.

Waste and ore geochemistry evaluations have indicated that the waste and ore associated with the Eagle Gold Project is likely to be non-acid generating (NAG). Kinetic testing indicates that concentrations of some metals from the waste rock and spent ore are likely to be somewhat elevated with respect to receiving water quality guidelines. Further work is required to develop water quality predictions for the waste rock, open pit, and heap leach pad. Some measures to control seepage and run-off water quality may be required. Further work is in progress to evaluate seasonal and annual variations during operations, drain down, and final closure.

Development, operation, and decommissioning of the mine will affect a range of environmental and socio-economic components. The project definition report will provide mitigation strategies to minimize or eliminate adverse environmental or socio-economic effects of the project. These strategies will form the basis for the development of environmental management plans for the construction, operation, and decommissioning phases of the project.

Security, reclamation, and closure for major mine projects in the Yukon is regulated under the Quartz Mining Act, the Yukon Mine Reclamation and Closure Policy (YMRCP), and the Waters Act for specific water-related issues. Financial security for major mines in the Yukon is held under a Quartz Mining Licence, or Water Licence, or a combination thereof. The amount and form of security is determined pursuant to the Quartz Mining Act or the Yukon Waters Act. A comprehensive Reclamation and Closure Plan must be prepared and submitted for approval as part of the Quartz Mine Licence application and be updated periodically throughout the operating mine life, and at a minimum, every five years. The YMRCP requires annual reporting and post closure monitoring. The mine owner must provide financial security for the full outstanding liability, based on the cost to reclaim and close the mine site in its current status, in accordance with the approved reclamation and closure plan. The outstanding liability is re-assessed periodically, or at minimum, every two years to reflect the impact of operations and progressive reclamation.

CAPITAL COST ESTIMATE

The estimated cost to design, construct, install and commission the Project operation and facilities described in the PFS is $281 million. This amount includes the direct field costs of executing the Project, plus the Owner's and indirect costs associated with design, construction and commissioning. Cost estimates are based on the PFS design, and are considered to have an accuracy of +/-25%. The capital cost estimate is summarized below in Table 1-8. All costs are expressed in fourth quarter 2009 Canadian dollars, with no allowance for interest or financing during construction.

TABLE 1-8 CAPITAL COST SUMMARY
Victoria Gold Corp. – Eagle Gold Project

Major Area	Construction Cost (C$'000s)	Ongoing Cost (C$'000s)	Total Cost (C$'000s)
Mining	34,950	5,719	40,669
Crushing & Conveying	68,009	5,126	73,134
Heap Leach Facility	26,258	24,747	51,005
Process Plant	16,075	1,177	17,252
Infrastructure	42,570	2,939	45,509
Subtotal Direct Capital Cost	**187,862**	**39,708**	**227,570**
Indirects	55,306	-	55,306
Contingency	38,214	-	38,214
Closure & Reclamation	105	15,732	15,836
Total Project Capital Cost	**281,486**	**55,439**	**336,925**

OPERATING COST ESTIMATE

Mine operating costs are estimated for year-round open pit mining at a rate of 26,000 tpd ore, (ranging from 35,000 tpd to 60,000 tpd, including waste mining). Process operating costs are estimated for three-stage crushing to a P_{80} of 5 mm. directly followed by stacking of ore, for 250 days per year. During the winter season, approximately 100 days of the year, ore will be crushed, then conveyed to the heap and stockpiled. During the summer season, this stockpiled material will be reclaimed and transferred to the pad stacking system. The ADR gold recovery plant will operate 350 days per year. General and Administrative (G&A) costs are estimated for a camp operation, with employees working a shift rotation, based out of Whitehorse.

Total operating costs average $91 million per year. Operating unit costs are summarized in Table 1-9:

TABLE 1-9 OPERATING COST SUMMARY
Victoria Gold Corp. – Eagle Gold Project

Area	Cost
Mine	$1.88 per t moved
	$3.84 per t processed
Process	$5.05 per t processed
G&A	$1.50 per t processed
Total	**$10.38 per t processed**

2 INTRODUCTION

Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA) was retained by Victoria Gold Corp. (Victoria Gold) to prepare an independent Technical Report on the Eagle Gold Project (the Project), in central Yukon Territory, Canada. The purpose of this Technical Report is to summarize the results of a Pre-Feasibility Study (PFS) on the Project, including Mineral Resource and Mineral Reserve estimates. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects.

Victoria Gold is a Canadian exploration company with projects in Canada and the United States. The company is primarily engaged in the acquisition, evaluation, exploration, and development of gold properties with a view to commercial production. Victoria Gold is incorporated in British Columbia and is a reporting issuer in British Columbia, Alberta, Ontario, and Quebec whose common shares trade on the TSX Venture Exchange.

In June 2009, Victoria Gold acquired its interest in the Dublin Gulch Property (which contains the Eagle Gold Project) as a result of its takeover of StrataGold Corporation. The Project comprises:

- An undeveloped gold deposit
- Access by highway and unpaved roads
- A 30-man exploration camp, with little other permanent infrastructure
- Mineral Reserves of 66 Mt, at a grade of 0.82 g/t Au, containing 1.8 Moz
- Potential development of a 9.1 Mtpa conventional heap leach open pit operation with an eight-year mine life
- Significant Mineral Resources and mineralization extending below the Mineral Reserve pit

The PFS was commenced immediately, led by Scott Wilson RPA which covered geology, resource estimation, mine design, heap leach design, and cost estimation. Kappes Cassiday & Associates (KCA) carried out metallurgical testwork, process design, and process cost estimation. BGC Engineering Inc. (BGC) carried out geotechnical field investigation and analysis for open pit slopes and for infrastructure requirements.

Stantec Inc. (Stantec) provided advice on environmental and permitting issues, in the course of preparing a Project Proposal (a permitting document) in parallel with the PFS.

Prior to Scott Wilson RPA involvement, the Eagle Gold Project was the subject of non-NI 43-101 compliant Technical Reports by Mineral Resource Development Incorporated (MRDI) in 1995 and James Sparling, P.Geo., in 1998. A Feasibility Study was completed by Rescan Engineering Ltd. (Rescan) in 1996. NI 43-101 compliant Technical Reports were produced on the Eagle Gold Project by Wardrop Engineering Inc. (Wardrop) in 2006 and 2009. Steffen, Robertson, Kirsten Consulting (SRK) produced a Preliminary Assessment in 2008 on the Mar-Tungsten Zone (now known as the Wolf Tungsten Zone), which comprises a small portion of the Dublin Gulch Property.

SOURCES OF INFORMATION

A site visit to the Eagle Gold Project was carried out by Jason J. Cox, P. Eng., Senior Mining Engineer, and David W. Rennie, P. Eng., Principal Geologist, both with Scott Wilson RPA, from June 10 to 11, 2009.

During the visit, discussions were held with:

- Michael Padula Project Manager, Victoria Gold Corp.
- Kevin Piepgrass Senior Project Geologist, Victoria Gold Corp.

Scott Wilson RPA, KCA, BGC, and Stantec personnel were on site, carrying out site visits or field programs at various times throughout the 2009 field season.

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 21, References.

LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the SI (metric) system. All currency in this report is Canadian dollars (C$) unless otherwise noted.

μ	micron
°C	degree Celsius
°F	degree Fahrenheit
μg	microgram
A	ampere
a	annum
bbl	barrels
Btu	British thermal units
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
d	day
dia.	diameter
dmt	dry metric tonne
dwt	dead-weight ton
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
g/t	gram per tonne
gpm	Imperial gallons per minute
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
hr	hour
ha	hectare
hp	horsepower
in	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
L/s	litres per second
m	metre
M	mega (million)
m^2	square metre
m^3	cubic metre
ma	million years ago
min	minute
masl	metres above sea level
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
m^3/h	cubic metres per hour
opt, oz/st	ounce per short ton
oz	Troy ounce (31.1035g)
oz/dmt	ounce per dry metric tonne
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA) and Kappes Cassiday & Associates (KCA) for Victoria Gold Corp. (Victoria Gold). The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Scott Wilson RPA and KCA at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by Victoria Gold and other third party sources.

For the purpose of Section 4 of this report, the authors have relied on ownership information provided by Victoria Gold, including an independent opinion by Davis LLP, of Vancouver, dated April 16, 2010. Scott Wilson RPA has not researched property title or mineral rights for the Eagle Gold Project and expresses no opinion as to the ownership status of the property. Scott Wilson RPA did review the status of some of the claims on the web site of the Yukon Department of Energy, Mines and Resources (http://yukonminingrecorder.ca).

Scott Wilson RPA has relied on Victoria Gold for guidance on royalties applicable to revenue or income from the Eagle Gold Project. Scott Wilson RPA has also relied upon Victoria Gold and Stantec for guidance on applicable permits and environmental obligations, and has not independently verified these requirements.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

The Eagle Gold Project is located within the Dublin Gulch Property, in central Yukon Territory, approximately 45 km north-northeast of the village of Mayo and 350 km north of the provincial capital of Whitehorse within portions of NTS sheets 105M/13, 105M/14, 106D/3, 106D/4, 115P/16 and 116A/1 (Figure 4-1). The property is rectangular in shape and extends approximately 26 km in an east-west direction and 13 km in a north-south direction. The Eagle Gold Project is accessible by a combination of paved and gravel roads from Mayo. The centre of the property is at approximately 7,100,950mN, 463,750E (NAD 83, Zone 8). The geographic co-ordinates are approximately 64°02'13" N latitude and 135°44'33" W longitude.

LAND TENURE

The Dublin Gulch Property consists of a contiguous block comprising 1,896 quartz claims, 10 quartz leases, and one Federal Crown Grant quartz claim totalling 34,576 ha within the Mayo Mining District (Figure 4-2). The claims and leases are currently recorded in the name of StrataGold. The claims are currently in good standing until their respective renewal dates from 2010 to 2027. Claims covering the Project area expire in 2016 at the earliest. Mineral claims in Yukon can be maintained in good standing by performing approved exploration work, or making payments in lieu of work, of $105 per claim per year. By virtue of two underlying agreements, Victoria Gold makes option payments totalling $35,000 per year.

Exploration work is subject to the Mining Land Use Regulations of the Yukon Quartz Mining Act and is carried out under a Class III Mining Land Use Permit. Victoria has been granted a permit (LQ00090) that is valid until July 25, 2012.

Victoria Gold reports that there are no known environmental liabilities on the property.

On June 4, 2009, Victoria Gold confirmed that the previously announced acquisition of StrataGold was approved by the Supreme Court of British Columbia and had closed as of that date, pursuant to the Plan of Arrangement.

By virtue of various underlying agreements, portions of the Dublin Gulch Property are subject to the following royalties:

- with respect to a portion of the property historically known as the Mar Gold Zone, an annual royalty payment of $20,000 or payments of 2% of gross returns received from the sale of all metals produced from the claims, whichever is greater, to a maximum of $1,000,000 (of which $400,000 has been paid to date), after which the royalty reverts to 1% of the said gross returns;

- with respect to 36 claims on the Lynx Zone, a 1% net smelter return (NSR) royalty with annual advance royalty payments of $15,000;

- with respect to the 63 claims and leases known as the Mar Tungsten Property and Mar Tungsten Leases, a 1% NSR royalty.

The Eagle Zone resources are located on claims which are subject only to the Mar Gold royalty (Figure 4-3).

Victoria Gold holds a 100% interest in all the claims and leases comprising the Eagle Gold Project with the exception of the Olive Federal Crown Grant which is 1/8 owned by G. William Vivon. Victoria Gold holds a 100% interest in the remaining 7/8 of the Crown Grant.

Victoria Gold, through its predecessor StrataGold, has established an Exploration Cooperation Agreement with the Na-Cho Nyak Dun (NND) First Nation to explore the Eagle Zone and other regional deposits. The agreement, dated May 21, 2008, is for a three year term with an option for an additional three years (SRK, 2008).

Victoria Gold provided an independent opinion by Davis LLP, dated April 16, 2010, which is in agreement with the above land tenure information.

Figure 4-1

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Location Map

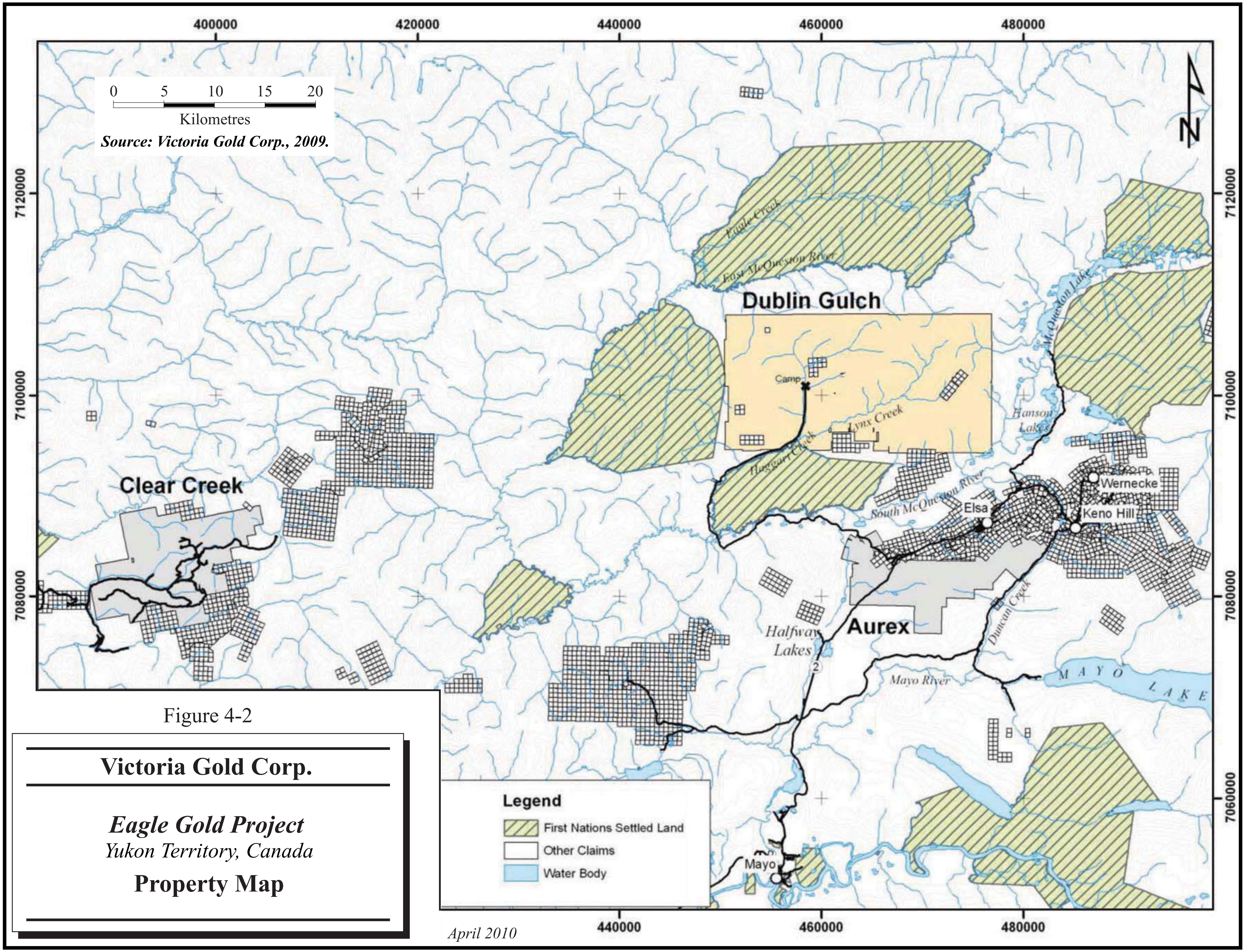


Figure 4-2

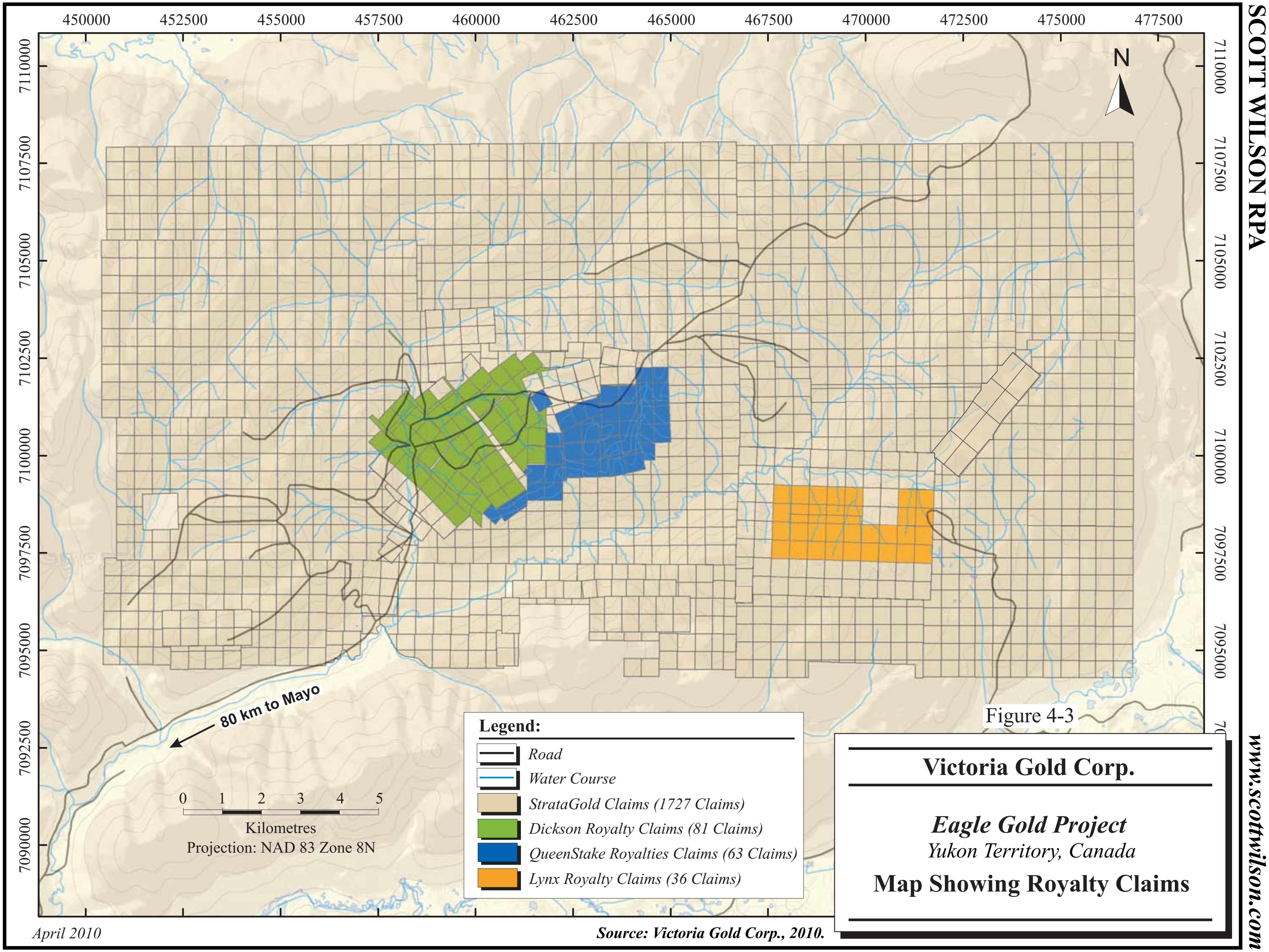
Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Property Map

April 2010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80 km to Mayo
Legend:
Road
Water Course
StrataGold Claims (1727 Claims)
Dickson Royalty Claims (81 Claims)
QueenStake Royalties Claims (63 Claims)
Lynx Royalty Claims (36 Claims)
0 1 2 3 4 5
Kilometres
Projection: NAD 83 Zone 8N
April 2010
Source: Victoria Gold Corp., 2010.
Figure 4-3
Victoria Gold Corp.
Eagle Gold Project
Yukon Territory, Canada
Map Showing Royalty Claims

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

The Eagle Gold Project is located approximately 45 km north-northeast of the village of Mayo, Yukon. The property can be accessed from Mayo by following Highway 2 for 35 km, then heading along the South McQuesten Road for 21 km. The last 25 km of the access road to the Project are not maintained by government maintenance crews and are unpaved, but are generally in good repair and allow passage for cars, trucks, and heavy haul highway truck and trailer units. Access within the greater property is by a network of four-wheel-drive roads.

CLIMATE

The property lies within the Mayo Lake-Ross River Eco-region in central Yukon. The area is characterized by a "continental" type climate, with moderate annual precipitation and a large temperature range. Summers are short and can be hot, while winters are long and cold with moderate snowfall. Rainstorm events can occur frequently during the summer, and may contribute 30% to 40% of the annual precipitation. Higher elevations are snow-free by mid-June. Frost action may occur at any time during the summer or fall.

Climate monitoring was conducted at Potato Hills (located in the upper Dublin Gulch watershed) within the project site from August 2007 to October 2009. Climate monitoring also took place near the project site camp to characterize the valley conditions. Regional climate data were collected from up to nine stations throughout the Yukon Territory.

The mean annual temperature for the area is approximately -3°C, with an annual range of 63.5°C for the period of record. The estimated mean annual precipitation in the study area ranges from 389 mm to 528 mm. This range in annual estimated precipitation reflects the elevation changes at the site.

This climate description has been summarized from Stantec, 2010.

LOCAL RESOURCES

Minimal resources are available at Mayo, a village with a population of 453 people (December 2009 Population Report, Yukon Bureau of Statistics), including motel accommodations, nursing station, fuel station (gasoline, diesel, and propane), grocery store, post office, earth moving contractors and restaurants. The government operates a 1,400 m gravel airstrip located three kilometres north of Mayo, which is suitable for charter flights. There is no scheduled air service to Mayo, though Alkan Air and Black Sheep Aviation, both of Whitehorse, provide charter air service.

Transmission lines from a hydroelectric facility near Mayo extend to the villages of Elsa and Keno City, located about 25 km southeast of the property. Yukon Energy Corporation is in the process of connecting this line to the main southern Yukon grid.

A broader range of services is available in Whitehorse, Yukon, located about six hours by road to the south of Mayo. Whitehorse has a population of 25,690 (Yukon Bureau of Statistics) and has daily air service to Vancouver, Edmonton, Yellowknife, and Fairbanks. The property is approximately 665 km by all-weather highway from the deep sea port of Skagway, Alaska.

INFRASTRUCTURE

Victoria Gold maintains a 30-man exploration camp on the property. There is no other permanent infrastructure on the property.

PHYSIOGRAPHY

The topography of the Eagle Gold Project is characterized by rolling hills and plateaus, with elevations ranging from 800 m at the confluence of Haggart and Lynx Creek to 1,537 m at the summit of Potato Hills. The hills and ridges are drained by gentle to steeply incised creeks and canyons. Glacial drift and colluvium constitute the main surface materials. Outcrop exposure averages about 2%.

Tree species include white and black spruce, alpine fir, lodgepole pine, trembling aspen, balsam poplar, and white birch. At higher elevations there are extensive areas of rolling alpine tundra characterized by sedge-dominated meadows and lichen colonized rock fields are common.

The region provides habitat for woodland caribou, moose, Dall's sheep, mountain goat, black and grizzly bear, marten, lynx, American pika, hoary marmot and arctic ground squirrel. Representative bird species include willow, rock and white-tailed ptarmigan and spruce grouse, along with a range of migratory songbirds and waterfowl.

6 HISTORY

PRIOR OWNERSHIP

In 1975, Dublin Gulch Mining staked the R and D claims and in 1977 Queenstake Resources Ltd. (Queenstake) staked the Mar 1-24 claims to cover tungsten-bearing skarns in the Ray Gulch area. Later that year, Queenstake staked the Mar 25-30 claims adjacent to the east of the original block. In 1978, Canada Tungsten Mining Corp. (CanTung) optioned the Mar, R, and D claims, and explored for both tungsten and gold until 1986. CanTung returned the gold and tungsten claims to Queenstake in 1986.

In 1987, Queenstake optioned the Mar gold property to Can Pro Development (Can Pro) and in 1988 leased the placer claims to Ron Holway.

In April 1991, an agreement among Queenstake, Can Pro, and Ivanhoe Goldfields Ltd. (Ivanhoe) was signed, giving Ivanhoe the option to acquire interests in three blocks of ground including:

- 311 claims comprising the “Mar Gold Property”
- 177 claims and 7/8th interest in the Olive Federal Crown Grant comprising the “Mar Tungsten Property”
- 32 mineral leases and claims comprising the “Mar Tungsten Leases”

Subsequent to signing the agreement, Ivanhoe staked 494 claims, of which 16 have since been allowed to lapse. Later in 1991, Amax Gold Inc. (Amax) entered into an option agreement with Ivanhoe giving them the right to earn a 50% interest in the property. Amax terminated the option in late 1992 without having earned an interest.

In 1994, First Dynasty Mines Ltd. (First Dynasty) acquired the property through its acquisition of Ivanhoe.

In 1996, First Dynasty formed a subsidiary, New Millennium Mining Ltd. (New Millennium), into which it transferred the property. In June 2002, First Dynasty changed its name to Sterlite Gold Limited (Sterlite). First Dynasty and its successor companies staked additional claims in 1995, 1996, 2001, and 2002.

In October 2004, StrataGold announced that it had entered into an agreement with Sterlite to buy the Dublin Gulch and Clear Creek gold properties, subject to underlying royalties, in consideration of US$6 million in cash and five million common shares of StrataGold. StrataGold staked additional claims in 2004 and 2005.

In June 2009, Victoria Gold confirmed its acquisition of StrataGold pursuant to a Plan of Arrangement.

EXPLORATION HISTORY

The following section was derived from MRDI, 1997; Sparling, 2008; and Wardrop, 2009.

Exploration and placer mining began on the Dublin Gulch property in 1895, with the first gold reported in 1898. In 1904, tungsten was identified in placer concentrates. The Geological Survey of Canada (GSC) discovered in situ scheelite in Dublin Gulch in 1916 (Wardrop, 2009).

Gold and tungsten exploration has been ongoing in the area since 1970. Placer mining of gravels began in 1973 and about 110,000 ounces of placer gold have been recovered since documentation of placer mining began in 1978. In 1971, Canex-Placer Limited drilled three holes and dug 20 trenches to test quartz-scheelite vein systems.

In 1977, Queenstake staked the Mar claims in the Ray Gulch area to cover a tungsten-bearing skarn. This property was optioned to CanTung, who explored for gold and tungsten between 1977 and 1986. CanTung drilled 65 NQ (47.6 mm dia.) diamond drill holes (DDH) for an aggregate length of 11,315 m in the tungsten skarn zone. In addition, 100 trenches were excavated, mapped, and sampled and two new gold bearing shears were exposed by placer mining operations. These programs led to the discovery of the Eagle Zone three kilometres southwest of the original tungsten occurrences. CanTung produced a resource for the tungsten skarn zone, however, as it predates NI 43-101 guidelines, it is not considered reliable and will not be stated herein.

In 1981 and 1982 CanTung's exploration activities consisted of regional geological mapping, heavy mineral sampling, and additional trenching. In 1986, CanTung returned the claims covering the gold vein system and the Mar Tungsten zone. Queenstake drilled four NQ holes, for a total of 705 m, on the Victoria and Catto veins. The Mar Gold property was optioned to Can Pro in 1987. Can Pro drilled four holes, for a total of 653 m, on the Victoria, Catto, and Cabin veins and excavated two trenches in the area of the Eagle and Scarp veins.

In 1991, the property was acquired by Ivanhoe, who carried out exploration work based on a "Fort Knox-type" intrusive-hosted gold exploration model. Ivanhoe entered into a joint venture with Amax. Five trenches were excavated for a total length of 2,000 m and 921 channel samples were collected. Sixteen HQ/NQ diamond drill holes were completed for a total depth of 2,410 m (Sparling, 2008), of which 13 holes comprising 1,943 m were drilled in the Eagle Zone (MRDI, 1997). The following year, Amax drilled 46 reverse circulation (RC) holes totalling 5,651 m and conducted sampling, mapping, and property evaluation. The Eagle Zone was tested by 34 of these RC holes for an aggregate depth of 500 m (MRDI, 1997). At the end of 1992, Amax did not renew the option on the joint venture.

On reacquiring the property in 1993, Ivanhoe drilled 10 RC holes for an aggregate depth of 2,078 m, testing the Eagle Zone with seven holes totalling 1,467 m (MRDI, 1997). In addition, Ivanhoe conducted trenching, soil sampling, geophysical surveys, baseline environmental monitoring, as well as mineralogical and metallurgical studies.

In 1995, Ivanhoe was acquired by First Dynasty. Eagle Zone drilling completed by First Dynasty was comprised of 40 RC holes (8,354 m), 19 HQ (63.5 mm dia.) core holes (3,706 m), and five PQ (85 mm dia.) metallurgical holes (1,091 m) for a total of 13,151 m drilled (MRDI, 1997).

In 1996, First Dynasty formed New Millennium and transferred Dublin Gulch to the new entity. On the Eagle Zone, New Millennium drilled 21 HQ core holes totalling 4,114.2 m and 37 RC totalling 5,271 m. In addition, New Millennium cut 33 water holes for 797 m and 19 auger holes for 189 m. The total drilling at Eagle for 1996 was 9,385 m (MRDI, 1997 and Sparling, 2008).

In 1997, Mineral Resource Development Inc. (MRDI) produced a resource estimate for the Eagle Zone. As this resource predates NI 43-101 guidelines it is considered unreliable and is quoted for historic purposes only. MRDI estimated Measured and Indicated Resources totalling 88.8 Mt at 0.698 g/t Au and Inferred Resources of 106 Mt at a grade of 0.345 g/t Au.

In 2004, StrataGold acquired Dublin Gulch along with the Clear Creek property for US$6 million and 5,000,000 common shares of StrataGold. Regional magnetometer/ electromagnetic (Mag/EM) airborne geophysical surveys were conducted to assist in exploration targeting and lithological interpretations. Snowden Mining Industry Consultants (Snowden) reviewed the 1997 MRDI resource estimate and concluded that it had been prepared in accordance with NI 43-101 guidelines with the exception of the resource classification. Snowden re-estimated the resource using MRDI data and parameters. The Snowden resource estimate was superseded by the Wardrop estimate, produced in 2006, incorporating StrataGold drilling from 2005 (Wardrop, 2009).

The 2005 drilling resulted in 34 HQ DDH for a total of 8,105 m. On occasion, due to poor ground or other issues, the core diameter was reduced to NQ. The purpose of the drilling was twofold; to test the western extent of the resource area as outlined by MRDI and to fill in data where the open pit had been designed (also by MRDI). In addition, a regional stream sediment and rock sampling program was undertaken. A total of 2,418 silt samples were collected at approximately 100 m intervals along current and seasonal drainages. An additional 534 rock samples were taken of quartz veins and mineralized outcrops/subcrops.

The 2006 exploration program included 10 HQ diamond drill holes with a total depth of 4,282 m. Soil sample and silt samples were collected to investigate interpreted geophysical anomalies, explore regional, underlying lithologies and structures, and to follow up on historical anomalous soil and grab samples. A total of 2,477 soils samples and 534 rock samples were collected with no mention of the number of silt samples. Trenching also took place, with 11 trenches excavated, from which a total of 347 samples were taken. In addition, selected historic drill core from the Mar Tungsten zone was relogged and resampled (Sparling, 2008).

Based on the available drilling, Wardrop produced a NI 43-101 compliant mineral resource estimate that outlined an Indicated Resource of 66.5 Mt grading 0.92 g/t Au and an Inferred Resource of 14.4 Mt grading 0.80 g/t Au using a 0.50 g/t Au cut-off. Wardrop recommended more drilling to increase the confidence of the resource estimate.

Drilling in 2007 comprised 20 HQ holes for a total depth of 5,627 m. The purpose of the 2006-2007 drill campaigns was to increase the confidence in the resource model and to continue to test the western extent of the resource model.

In 2008, 15 HQ DDH were drilled totalling 4,249 m to test the extents of the mineralized zone.

A summary of drilling on the Eagle Zone is shown in Table 6-1.

TABLE 6-1 SUMMARY OF DRILLING AT EAGLE GOLD*
Victoria Gold Corp. – Eagle Gold Project

Year	Operator	Core		RC		Total	
		Number	Depth (m)	Number	Depth (m)	Number	Depth (m)
1986	Queenstake	4 (NQ)	705			4	705
1989	Can Pro	4	653			4	653
1991	Amax	13 (HQ-NQ)	1,943			13	1,943
1992	Amax			34	4,200	34	4,200
1993	Amax			7	1,447	7	1,447
1995	First Dynasty	19 (HQ) 5 (PQ)	3,706 1,091	40	8,354	64	13,151
1996	New Millennium	21 (HQ)†	4,114	37	5,271	58	9,385‡
2005	StrataGold	34 (HQ/NQ)	8,105			34	8,105
2006	StrataGold	10 (HQ)	4,282			10	4,282
2007	StrataGold	20 (HQ)	5,627			20	5,627
2008	StrataGold	15 (HQ)	4,249			15	4,249
2009	Victoria Gold	14 (HQ3/PQ)§	5,122			14	5,122
Total		**159**	**39,597**	**118**	**19,270**	**277**	**58,867**

Notes:

*Information for this table was derived from MRDI, 1997; Sparling, 2008; and Wardrop, 2009; and has not been independently verified by Scott Wilson RPA

†Includes six geotechnical holes drilled in the pit walls and crusher area

‡Excludes 19 clay auger holes (189 m) and 33 water wells (797 m)

§Includes 4 HQ3 geotechnical holes (1320.6 m) and 3 metallurgical holes (504.8 m of PQ)

Totals may not equal the sum of the columns due to rounding.

In early 2009, Wardrop updated the mineral resource estimate, using the 39 additional holes (14,158 m total length) completed during the 2006, 2007, and 2008 campaigns. The cut-off grade for interpretation was nominally 0.20 g/t Au. The resource estimation was interpolated using Ordinary Kriging (OK), with validation checks run using both nearest neighbour (NN) and inverse distance (ID). Using a 0.5 g/t Au cut-off, the updated estimate totalled 98.5 Mt of Indicated Resources grading 0.849 g/t Au and 2.0 Mt of Inferred Resources grading 0.671 g/t Au. No constraint on resource depth was applied.

Wardrop recommended extending two existing drill holes, drilling three new holes to test mineralization at depth and three additional holes to define the potential of the northern hangingwall zone. Wardrop also recommended comprehensive metallurgical sampling and additional bulk density determinations of the mineralized material. Wardrop recommended that the next phase of the Project should be a pre-feasibility study.

A summary of historic Resource Estimates is given in Table 6-2.

TABLE 6-2 PREVIOUS RESOURCE ESTIMATES
Victoria Gold Corp. – Eagle Gold Project

Estimator	Year	Classification	Tonnage Mt	Grade g/t Au	Cut-Off Grade g/t Au
MRDI*	1997	Measured and Indicated	88.8	0.698	N/A
		Inferred	106.0	0.345	N/A
Wardrop	2006	Indicated	66.5	0.92	0.50
		Inferred	14.4	0.80	0.50
Wardrop	2009	Indicated	98.5	0.849	0.50
		Inferred	2.0	0.671	0.50

*The MRDI resource estimate predates NI 43-101 guidelines and is therefore considered unreliable. It is quoted for historic purposes only.

Victoria Gold's exploration efforts in 2009, and Scott Wilson RPA's subsequent resource estimate, are described later in this report.

7 GEOLOGICAL SETTING

REGIONAL GEOLOGY

The following section is adapted from Sparling, 2008 and Wardrop, 2009.

The Dublin Gulch Property is located in the northern portion of the Selwyn Basin, which is situated in the central Yukon. The Selwyn Basin is a fault controlled epicratonic basin that is underlain by four main metasedimentary units. Dominantly clastic, these units are, from oldest to youngest, the Proterozoic Hyland Group (also referred to as the Grit Unit), the Paleozoic Upper Schist, and the Mesozoic Lower Schist and Keno Hill Quartzites (Figure 7-1). The Hyland Group consists of interbedded gritty quartzite, phyllite, and minor limestone. The Upper Schist is comprised of interbedded graphitic phyllite, phyllitic quartzite, and quartz-sericite phyllite. The Lower Schist and Keno Hill Quartzites are homogenous, light grey, and massive. The Upper Schist is comprised of interbedded graphitic phyllite, phyllitic quartzite, limestone, and quartzite. Deposited offshore of the paleo-continental margin these rocks were later accreted onto the North American continent as allochthonous terranes (Sparling, 2008).

These units have been juxtaposed by Cretaceous-aged laterally extensive, northward-directed thrust sheets. There are three principal thrust sheets, from east to west, the Dawson, Tombstone and Robert Service. The Robert Service thrust is proximal to Dublin Gulch and superimposes the older Hyland Group on the Keno Hill Quartzite (Wardrop, 2009). The Tombstone and Robert Service thrusts are considered coeval. The hangingwall of the Tombstone thrust is a zone of regional-scale, highly-deformed rocks known as the Tombstone High Strain Zone (THSZ). The THSZ was later subjected to folding by the west plunging McQuesten Antiform which lies south of Dublin Gulch (Wardrop, 2009).

Four generations of deformation are documented, however, prominent structures were limited to the initial two events. The first resulted in widespread development of foliation that was subsequently deformed by gentle, regional-scale folding. The second deformation event resulted in east trending, south plunging anticlines in the Dublin Gulch area, the most prominent of which is the Lynx Creek Anticline, whose hinge is interpreted to pass through the Eagle Zone at Dublin Gulch (Sparling, 2008).

Three Cretaceous-aged granitoid intrusions were emplaced, including the Selwyn Suite, dated between 104 Ma and 98 Ma, the Tombstone Suite, between 94 Ma and 92 Ma, and the McQuesten Suite, dated at 64 Ma. The intrusives were most commonly emplaced within the Hyland Group and Upper Schist rocks. The Tombstone Suite contains the five kilometre by two kilometre Dublin Gulch stock, and is the primary host of vein-, shear- and skarn-related mineralization. Economic minerals of interest are gold, silver, lead, zinc, and tungsten. The Tombstone Suite is the primary source of intrusion hosted gold deposits (Wardrop, 2009).

The Tombstone Suite forms part of the Tombstone Gold Belt, which is in the eastern portion of the Tintina Gold Province. The western portion of the Tintina Gold Province has been dextrally displaced approximately 450 km by the Tintina fault. This western portion contains deposits such as Fort Knox, Pogo and Donlin Creek in Alaska. In the Yukon, Brewery Creek and Dublin Gulch, along with major gold zones associated with Gold Dome (previously known as Scheelite Dome) and Clear Creek, are part of the Tombstone Gold Belt (Sparling, 2008).

At least four periods of faulting have occurred in the Dublin Gulch area, including two low angle thrust and bedding plane faults, NW trending faults, NE trending faults, E trending faults, and N-S trending faults. It is the latter that may have the most significant impact on mineralization, which appears to displace the Dublin Gulch stock. The NW and N-S trending faults are believed to have predated the emplacement of the Dublin Gulch stock, however, they were reactivated during the intrusive event (Sparling, 2008).

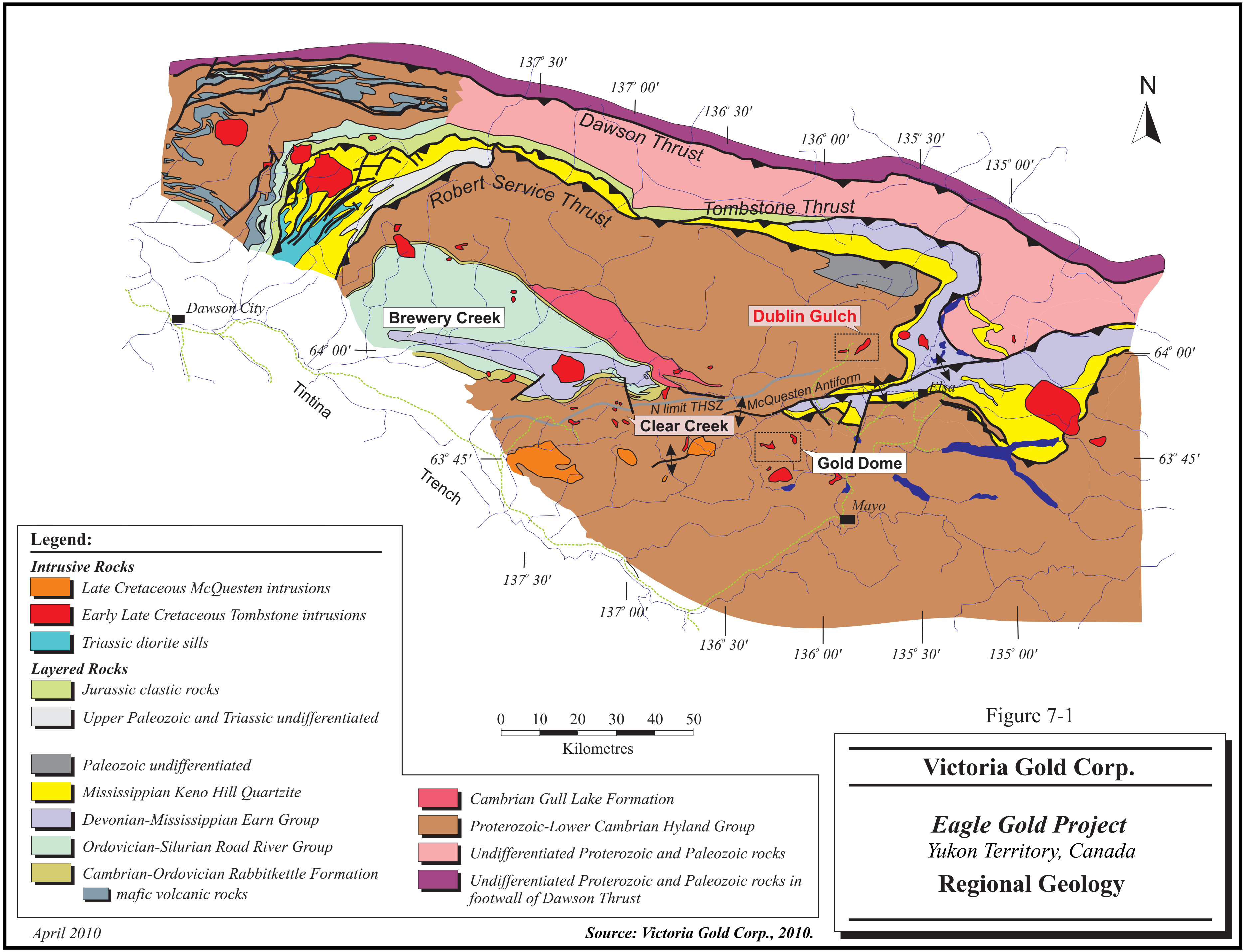

Figure 7-1

Victoria Gold Corp.

Eagle Gold Project
Yukon Territory, Canada
Regional Geology

PROPERTY GEOLOGY

Dublin Gulch lies on the northern limb of the McQuesten Antiform and is underlain by Proterozoic to Lower Cambrian age Hyland Group metasedimentary rocks and the intrusive Dublin Gulch granodioritic stock (Figure 7-2).

The Hyland Group is comprised of interbedded quartzite, phyllite and minor limestone. The quartzite is variably gritty, micaceous and massive. The phyllite is comprised of muscovite-sericite and chlorite. The metasedimentary rocks are the product of greenschist grade regional metamorphism. Proximal to the Dublin Gulch stock, metasomatism and contact metamorphism have occurred. Coarse clastic components of the host Hyland Group have been altered to quartz-biotite schist. The argillaceous component of the Hyland Group has been altered to sericite-biotite-chlorite schist and the limestone to marble, wollastonite-quartz skarn and pyroxenite skarn. The hornfels thermal aureole surrounding the Dublin Gulch stock extends 800 m to 2,000 m outward from the intrusive (Wardrop, 2009).

The Dublin Gulch stock, which intrudes the Hyland Group metasediments near the contact with the underlying Upper Schist, is comprised of four phases. The most significant of these is the granodiorite. Younger intrusive phases that also occur as dikes and sills are quartz diorite, quartz monzonite, leucogranite and aplite. These crosscut both the granodiorite and the host lithology.

The Dublin Gulch stock is elongate coincident with the axis of the Dublin Gulch anticline at about 070°. The intrusive sedimentary contact dips shallowly to the north on the northern side of the intrusive and steeply to the north or south at the southern margin (Wardrop, 2009). The stock is a medium grey coloured, medium-grained granodiorite and is comprised of phenocrysts of plagioclase, quartz, K-feldspar, biotite ± amphibole in a fine-grained groundmass dominated by K-feldspar, quartz and plagioclase. Other minor minerals occurring with the stock are muscovite, calcite, titanite, allanite, apatite and zircon (MRDI, 1997). The Dublin Gulch stock lacks deformational features such as foliation indicating emplacement after regional foliation (Sparling, 2008).

The Eagle Zone gold occurrence is located at the western limit of the Dublin Gulch stock at its narrowest extent. The intrusive-metasedimentary contact is both steep and cross-cutting the host rock foliation. It is shallow dipping to the southwest, paralleling the metasediment foliation. The host rock is comprised of two basic units; partially feldspathic, weak to moderately folded quartzites and well foliated phyllites that are compact aggregates of biotite and sericite. The granodiorite intrusive varies from relatively unaltered to strongly sericite ± hematite altered (weathered). Weathering is concentrated around faults and broken rock zones, and is variable to depths down to 180 m from surface. Weathering is less prevalent in metasediments or granodiorite that is capped with metasediment (MRDI, 1997).

The zone is characterized by sub-parallel extensional quartz veins that are best developed within the intrusive proximal to both hangingwall and footwall contacts with the host metasediment. Drilling has concentrated on the hangingwall contact where, ostensibly, more quartz veining is encountered. The footwall contact, having received less scrutiny, may possess more veining than currently recognized.

Veins strike from 060° to 085°, dipping approximately 60° south and possessing widths ranging from 0.01 cm to 10 cm. Contacts are generally sharp and veining densities range from less than one vein per metre to more than 15 veins per metre. Vein density coincides with the local apex of the intrusion and the tightest constriction between footwall and hangingwall contacts. Wardrop (2009) suggests vein formation is attributed to the contrasts in cohesion and tensile strength between the intrusive and country rocks. Embayments and constrictions of the stock represent stress shadows that constitute favourable area for the formation of extensional quartz veins. Protrusions of the stock create favourable areas for the development of extensional shear veining in the metasedimentary host.

The veins are comprised of white to grey quartz, with subordinate potassium feldspar. Sulphides occur in the centre, on the margin, and disseminated throughout. Mineralization accounts for less than 5% of vein material. Most common sulphides are pyrrhotite, pyrite, scheelite, arsenopyrite, sphalerite, bismuthinite, and galena (Wardrop, 2009). Gold occurs in veins as native gold, liberated in gangue or associated with bismuth minerals, and with an average grain size of 110 microns to 160 microns (MRDI, 1997).

Alteration envelopes consist of secondary potassium feldspar. Sericite carbonate alteration also exists as narrow vein selvages and independently from veining (Wardrop, 2009).

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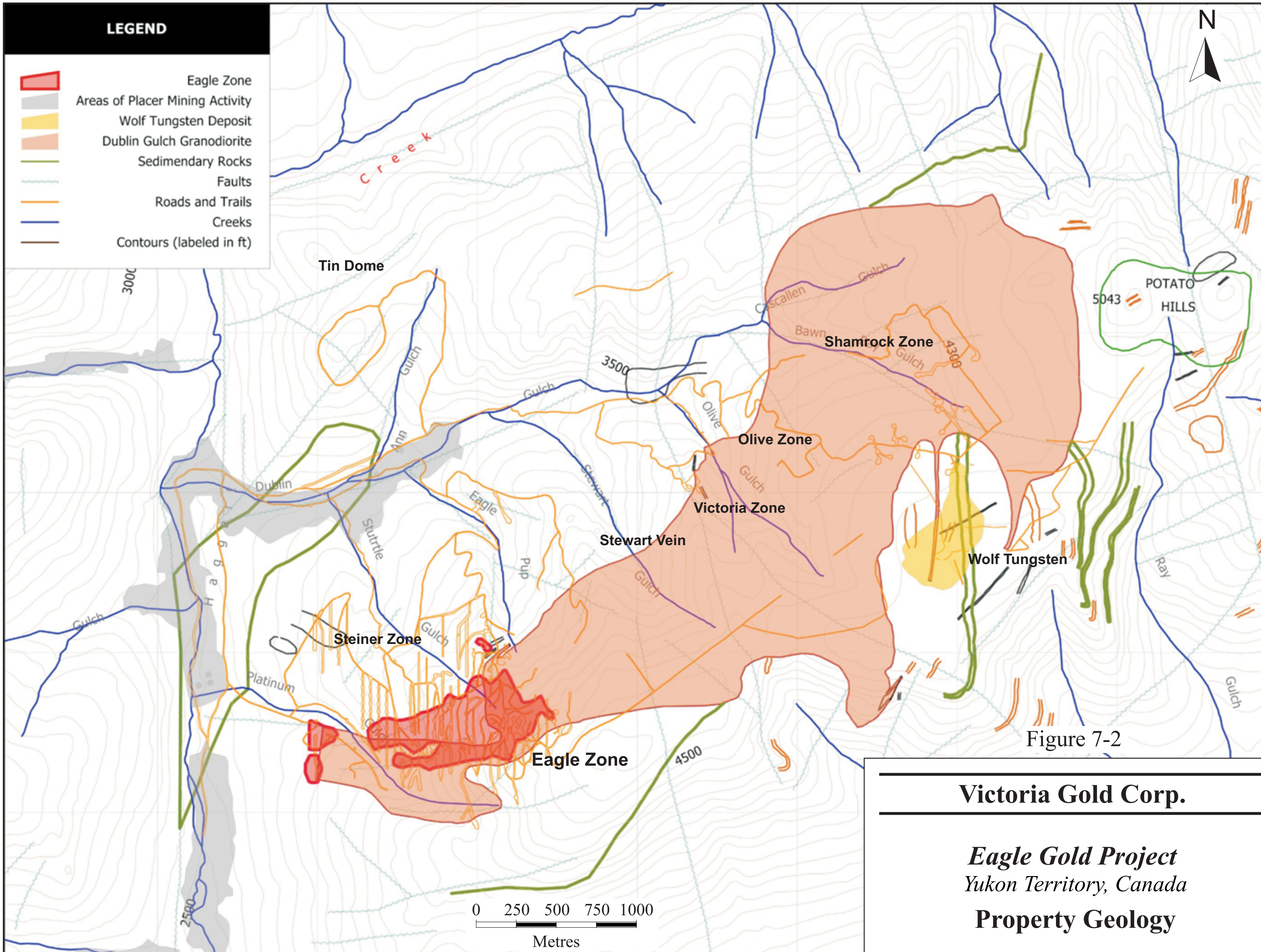

Figure 7-2

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Property Geology

April 2010

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8 DEPOSIT TYPES

The Eagle Zone gold deposit is associated with the Dublin Gulch intrusion, which forms part of the mid-Cretaceous Tombstone Suite. It falls within the 2,000 km long Tintina Gold Province, which stretches across central Yukon and the interior of Alaska. Gold deposits within the Tintina Gold Province have been extensively studied since the late-1990s and have been categorized as intrusion-related deposits by a variety of authors including Mair et al. (2006), Hart (2005), Brown et al. (2001), Maloof et al. (2001) and Baker et al. (2005).

Thompson and Newberry (2000) interpret the Eagle Zone as belonging to a class of deposits termed reduced intrusion-related gold systems (RIRGS). RIRGS are characterized by widespread arrays of sheeted auriferous quartz veins that preferentially form in the brittle carapace at the top of small plutons, where they form bulk-tonnage, low-grade Au deposits characterized by Au-Bi-Te-W metal assemblages (Hart, 2008).

The characteristics of RIRGS listed below, are taken from Hart (2005):

- Metaluminous, subalkalic intrusion of intermediate to felsic compositions that lie near the boundary between ilmenite and magnetite series;
- Carbonic hydrothermal fluids;
- A metal assemblage that variably combines gold with elevated Bi, W, As, Mo, Te and/or Sb and low concentrations of base metals;
- A low sulphide mineral content, mostly less than 5% by volume, with a reduced ore mineral assemblage that typically comprises arsenopyrite, pyrrhotite, and pyrite. Usually lacks magnetite or ilmenite;
- Commonly weak hydrothermal alteration, restricted in area;
- A tectonic setting well inboard of inferred or recognized convergent plate boundaries;
- A location in magmatic provinces best or formerly known for tungsten and/or tin deposits.

Other distinguishing characteristics are presented below. They are mostly designed to differentiate intrusion-related gold deposits – which are a product of local-scale fluids derived from a cooling pluton – from orogenic deposits that are considered to result from crustal-scale fluids derived through metamorphic dehydration. Some features exclusively differentiate RIRGS. No single characteristic is diagnostic, however, a suite of features is most effective to provide evidence for intrusion-related origin.

TECTONIC SETTING

RIRGS deposits are best developed in intrusions that were emplaced into ancient continental margins, behind accretionary or collisional orogens and subduction-related magmatic arcs. Preferred host strata include reducing basinal miogeoclinal sedimentary or metasedimentary rocks.

METAL ZONING

Thermal gradients surrounding cooling plutons are steep and result in temperature-dependent concentric metal zones that develop outward from pluton margins for distances up to a few kilometres, or just beyond the thermal aureole. Pluton proximal gold mineralization may be associated with Bi, Te, and W; aureole-hosted mineralization will have an As or Sb tenor; and distal mineralization may be related to Ag-Pb-Zn.

Skarns and replacements are generally pluton proximal, with an increase in structural control on more distal mineralization. There is also crustal-scale vertical zonation, with epizonal occurrences forming at shallower levels.

DIVERSE DEPOSITS

Fluids exsolving from cooling plutons are opportunistic and cool quickly, thus depositing metals in several available geological settings. Resulting mineralization is commonly of several different styles: variably intrusion and country-rock hosted, consisting of skarns, replacements, disseminations, stockworks, and veins. Gold mineralization is characterized by a wide range of gold grades, with bulk mineable volumes present up to the 1.5 g/t Au level.

SHEETED VEINS

The most distinctive style of gold mineralization in RIRGS are sheeted arrays of parallel, low-sulphide, single-stage quartz veins, which are found over 10s to 100s of metres and preferentially located in the pluton's cupola. These veins are unlike multidirectional interconnected stockworks characteristic of porphyry systems or antithetic tensional vein arrays typical of orogenic deposits.

PLUTON FEATURES

Mineralizing plutons have "smoking gun" characteristics that indicate the likelihood of generation of hydrothermal fluids. Physical features and geochemical support should exist for high volatile contents, fluid exsolution, evidence of rapid fractionation, zoned plutons, porphyry textures, presence of aplite and pegmatite dykes, quartz and tourmaline veins, greisen alteration, miarolitic cavities and/or unidirectional-solidification textures, preferably in the pluton's apices.

REDOX STATE

RIRGS are associated with felsic, ilmenite-series plutons that lack magnetite, have low magnetic susceptibilities and aeromagnetic response, and have low ferric:ferrous ratios of less than 0.3. These types of plutons are uncommon in arc and fore-arc settings, where orogenic gold deposits are most common.

TIMING

Intrusion related deposits are coeval (± 2 ma) with their associated, causative pluton.

9 MINERALIZATION

This section was derived from Sparling (2008) and Wardrop (2009).

The Eagle Zone displays gold mineralization similar to the Fort Knox deposit north of Fairbanks, Alaska. Both deposits occupy portions of the same belt, offset by the Tintina fault.

At Fort Knox, mineralization occurs as distinct grains, or within fractures in sulphides, in structurally controlled pegmatite dikes, grey quartz vein stockworks, and quartz-filled shears that cut the granite stock. These shears have been dated at 92 Ma to 90 Ma.

The stock is a multi-phase intrusive of granitic composition that was emplaced in the Fairbanks Schist, which is comprised of fine-grained muscovite-quartz schist and micaceous quartzite.

Veins are most abundant in the apical portion of the stock, and are thought to have formed as a result of doming and subsequent subsidence of the granite intrusive.

Gold mineralization at Fort Knox occurs with native, sulphide, and oxide bismuth minerals and tellurides. Associated sulphide minerals include marcasite, pyrite, pyrrhotite, arsenopyrite, and molybdenum, and occupy less than 1% of the rock by volume. Oxide minerals include scheelite and rutile.

Sparling (2008) suggests three types of gold/silver mineralization associated with the Dublin Gulch stock and related intrusions. These are:

- Sheeted E-W extensional low-sulphide quartz veins with minor alteration selvages within the intrusive that contain gold and silver (Fort Knox-type);

- Structurally controlled gold/quartz/arsenopyrite/scorodite fault/veins with the intrusive and surrounding metasediments, located primarily, but not exclusively proximal to the northern contact of the Dublin Gulch stock and metasediments;

- High silver/(±gold)/quartz/sulphide veins both distal to, but on trend with, the Eagle zone and near the NE extent of Eagle Zone.

The sheeted E-W extensional quartz veins occur within the intrusion on the hangingwall and footwall contacts at the narrowest portion of the stock. Overlying and adjacent metasediments also contain gold mineralization in quartz veins, however, vein development is less prominent. The overall strike of the veins parallels the Dublin Gulch stock. Multiple structural and mineralization events may have occurred.

In 1992, MRDI conducted mineralogical studies on 142 gold grains that led them to conclude that the gold at Dublin Gulch caused a “nugget effect” in resource estimation. The gold grains ranged in size from four microns to 1,400 µm with an average size of 155 micron. A subsequent mineralogical evaluation in 1993 examined 13 gold grains that averaged 118 µm and ranged from 30 µm to 322 µm in size. MRDI considered these to be sufficiently large as to be classified as “coarse-grained”.

The principal sulphides present within the Eagle Zone and Dublin Gulch stock are, listed in descending abundance, arsenopyrite, pyrrhotite, pyrite, chalcopyrite, sphalerite, bismuthenite, galena, and molybdenite.

Elemental gold occurs as isolated grains within fractures, quartz veins and wall rock or associated with arsenopyrite or, less commonly, with pyrite, pyrrhotite, chalcopyrite and sphalerite. Most quartz veins that host gold are one centimetre to two centimetres wide. Wider arsenopyrite/scorodite fault/veins can range from two centimetres to 30 cm in width. Visible gold is occasionally found encapsulated within clots of arsenopyrite.

OTHER SHOWINGS AND DEPOSIT TYPES

Several other showings, encompassing a range of deposit types, occur in the area (see Figure 7-2). Most, if not all, are related to the Dublin Gulch granodiorite, and fall generally under the category of intrusion-related deposits. The Wolf Tungsten deposit (formerly known as the Mar Tungsten deposit) is located approximately 2.5 km east-northeast of the Eagle Zone. Scheelite occurs in calc-silicate skarn in meta-sedimentary rocks adjacent to the Dublin Gulch Granodiorite Stock. A Mineral Resource estimate for Wolf compiled by SRK Consulting Inc. (SRK) in 2008 totalled 12.7 Mt grading 0.31% WO_3 in the Indicated category, and an additional 1.3 Mt of 0.30% WO_3 in the Inferred category.

Gold-bearing quartz-sulphide veins occur in several localities, generally concentrated around the margins of the Dublin Gulch Stock. These are narrow veins, cm-scale in width, with steep dips and striking approximately 070°. Gold occurs along with arsenopyrite and scorodite, with grades that can be in the 10 g/t Au to 35 g/t Au range. Examples of these types of showings are the Shamrock, Stewart, Steiner, Olive and Victoria Zones. StrataGold, and more recently, Victoria Gold, have been conducting exploration work on these showings, and they remain active exploration targets.

The Peso and Rex showings, on the west side of Haggart Creek, opposite the Eagle Zone, are high-silver quartz-sulphide. North of Dublin Gulch, near Tin Dome, cassiterite occurs in a tourmalinized breccia zone.

Placer gold has been mined for many decades along Haggart Creek and Dublin Gulch, and this activity is ongoing.

10 EXPLORATION

Earlier exploration at Dublin Gulch has been described in Section 6, History.

2009 PROGRAM

Subsequent to the Victoria Gold acquisition of StrataGold, 5,122 m of diamond drilling was conducted at the Eagle Gold property. Exploration drilling for 2009 comprised seven HQ3 (61.1 mm dia.) core holes (3,296 m) and 2,018 samples, four HQ3 geotechnical holes (1,321 m), and three PQ (85 mm dia.) metallurgical holes (505 m). The material from the three PQ holes was neither logged nor assayed but was used for metallurgical testing of the granodiorite stock.

The eleven exploration and geotechnical holes utilized a “triple-tube” core barrel, which allows for the orientation of the drill core in three-dimensional space to accurately determine the azimuth and dip of crosscutting structures, veins, and faults. These data are used to define geotechnical parameters that support pit design and mine planning. The method also results in improved core recovery. Victoria Gold reports core recovery in excess of 90% for the 2009 campaign.

The exploration program was designed to delineate mineralization west and at depth relative to the existing resource. The program was successful in extending the zone of know mineralization with significant results summarized in Table 10-1.

TABLE 10-1 2009 EXPLORATION PROGRAM SIGNIFICANT RESULTS
Victoria Gold Corp. – Eagle Gold Project

Hole Number	From (m)	To (m)	Length (m)	Au Grade (g/t)
DG09-359C	428.03	439.05	11.02	1.22
DG09-360C	21.34	117.96	96.62	0.95
	297.79	346.26	48.47	1.28
	386.18	462.38	76.20	1.03
DG09-361C	372.44	409.80	37.36	1.68
DG09-362C	239.88	267.81	27.93	1.26
	320.65	363.32	42.67	0.92
DG09-363C	221.59	233.10	11.51	1.16
	332.84	354.18	21.34	0.75
	448.77	463.91	15.24	1.00
	534.01	550.77	16.76	1.26
DG09-364C	209.40	220.07	10.67	2.53
	287.12	336.90	49.78	1.21
	364.85	396.85	32.00	0.89
DG09-365C	276.45	354.18	77.73	1.03
	296.27	352.65	56.38	1.26

In addition to the drilling campaign, ten trenches were excavated during the 2009 program. Five trenches were dug at the Shamrock Zone and five at the Olive Zone. The trenching results were not incorporated into the Mineral Resource estimate, which covers only the Eagle Zone.

11 DRILLING

Drilling prior to the 2009 campaign is summarized in Section 6, History.

The 2009 drill program is summarized in Table 11-1:

TABLE 11-1 2009 DIAMOND DRILL CAMPAIGN
Victoria Gold Corp. – Eagle Gold Project

Type	Diameter	# of Holes	Depth (m)
Exploration	HQ3 (61.1 mm)	7	3,296.1
Geotechnical	HQ3 (61.1 mm)	4	1,320.6
Metallurgical	PQ (85.0 mm)	3	504.8
Total		**14**	**5,121.5**

Victoria Gold followed standard industry practices of logging and sampling of diamond drill core. Prior to logging, core was photographed using a colour digital camera. Photos for each hole were archived as electronic files together with other data for each hole. Core logging of each geological interval consists of records for the following:

- Start and end metreage of lithological intervals with sub-intervals by sample.
- Lithology.
- Presence and intensity of sericite, chlorite, clay and carbonate alteration, oxidation and silicification based on a scale of 0 to 5 (with 0=unaltered and 5=intensely altered).
- Number of major, minor, or other veins per metre, type of vein (number code from 1 to 6 representing different vein mineralogies), maximum width and total number of major, minor, and other veins.
- Primary and secondary (where applicable) vein angle to core axis.
- Maximum width, alteration type, and intensity (similar 0 to 5 scale as used for lithology) of vein selvages and the oxidation state (from 0 to 5) of any sulphides within the selvage. With the exception of selvage alteration type, all these data were numeric.
- Mineralization as a percentage of the vein material with further categorization by mineral (pyrite, pyrrhotite, chalcopyrite, arsenopyrite, bismuthite, molybdenite,

visible gold, or other). Visible gold, when encountered, is noted. This information was recorded directly into a spreadsheet.

- Oxidation as a percentage of the vein material, with further categorization by mineral (oxidized sulphide, scorodite, biotite or other).

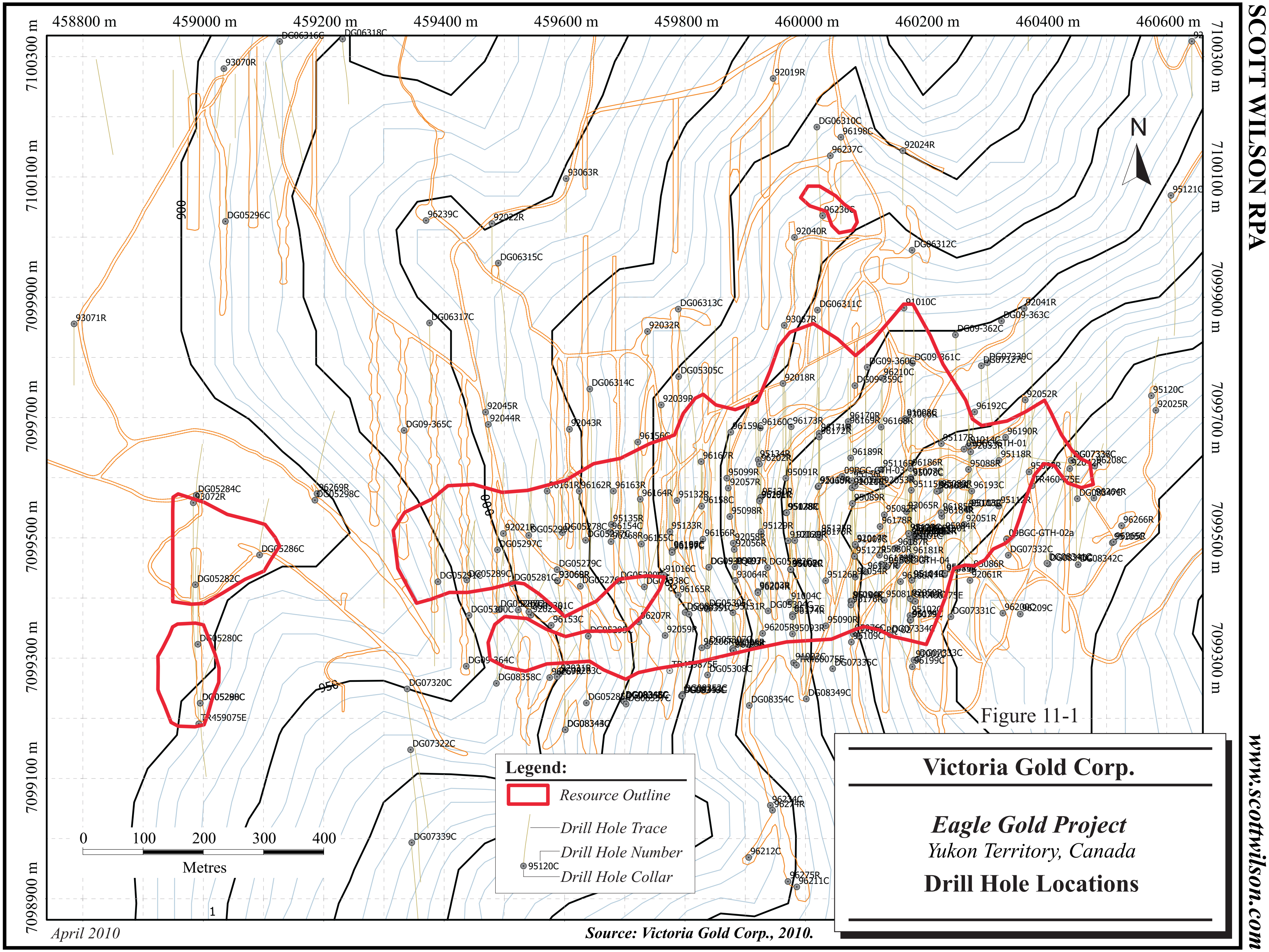
- Sulphide content (as a percentage) per sample.

Geotechnical logging was done on four HQ3 holes comprising 1,321 m. Each log recorded the following information:

- discontinuity type (joint, vein, etc.)
- measured depth downhole from the collar to the discontinuity
- Easting, Northing and Elevation of the discontinuity, as derived from core orientation
- dip of discontinuity
- dip direction of discontinuity
- aperture (in mm) of discontinuity
- minerals infilling the discontinuity
- joint roughness coefficient (JRC)
- joint wall compressive strength modifier (JWCSMod - to identify the relative strength of the joint compared to the host rock)

Logging and interpretation of geotechnical data was done by BGC, to support open pit mine design and mine planning.

Drill hole locations are displayed on Figure 11-1.


SCOTT WILSON RPA
www.scottwilson.com
458800 m
459000 m
459200 m
459400 m
459600 m
459800 m
460000 m
460200 m
460400 m
460600 m
7100300 m
7100100 m
7099900 m
7099700 m
7099500 m
7099300 m
7099100 m
7098900 m
N
Legend:
Resource Outline
Drill Hole Trace
Drill Hole Number
95120C
Drill Hole Collar
0
100
200
300
400
Metres
April 2010
Source: Victoria Gold Corp., 2010.
Figure 11-1
Victoria Gold Corp.
Eagle Gold Project
Yukon Territory, Canada
Drill Hole Locations

12 SAMPLING METHOD AND APPROACH

HISTORIC DRILLING

Information for this section was taken from Snowden, 2004 and Wardrop, 2009.

No detail is available regarding sampling methodologies for the 1991 and 1992 drill programs.

In 1993, sampling of RC holes consisted of quarter-splits of drill chips at the drill site, using a five-foot sample interval. Dry samples were split using a two-tier riffle splitter and wet samples utilized a rotary splitter. High water flows were sampled using three-tiered buckets with plastic baffles. Any overflow from the third bucket was not sampled.

Sampling protocols for the 1995 program were similar to 1993, with the exception that the dry samples were split in camp using a Jones splitter.

In 1996, the HQ core was whole-core assayed (for any holes included in the Eagle Zone Mineral Resource estimate) in nominal five-foot intervals. Fines were collected from the core box and included with the last sample. For the RC drilling, holes were cased to 20 ft. (10 ft. if collared in solid rock), and sampled on five-foot intervals thereafter. Dry holes were sampled by means of a “hopper gate” attached to the bottom of the cyclone. Sample material was directed via the hopper gate to one of two bags, which each could contain 2.5 ft. of sample. For wet holes, the hopper gate was replaced by a rotary splitter at the bottom of the cyclone. The rotary splitter had 16 troughs, of which four were used to collect the sample. The wet sample was collected using a cascading bucket system with a plastic garbage can used to collect the 25% fraction. The can had a notch that allowed any overflow to be discharged to a five-gallon pail. Any overflow from the five-gallon pail was not collected (Snowden, 2004).

From 2005 to 2007, the sample intervals were designated by the core logging geologist, generally in five foot intervals, and the core was taken to the core-cutting facility. Holes DG05-276C to DG05-297C were sampled in their entirety. Holes DG05-298 to DG05-309C were sampled in their entirety in the granodiorite, and in the sediments only where vein densities exceeded three per metre (Wardrop, 2009). The core was cut in half lengthwise with one half returned to the core box, where a sample ticket had been stapled for archival reference. The other half of the core sample was placed in a plastic sample bag along with a duplicate sample ticket for submission to the assay laboratory.

In 2008, the sampling protocol was changed to whole core (Wardrop, 2009).

In Scott Wilson RPA's opinion, the historical drill hole sampling appears to have been carried out to a reasonable standard. Data has been collected in several campaigns spanning many years, and for some years the documentation is not as detailed as for others. Specific concerns have been raised about certain sampling practices, which have resulted in some changes in protocols over the years. Most of the concerns raised by earlier reviewers were centred around assay repeatability and nugget effect. This could have an effect on the accuracy of local block grade estimates, however, in Scott Wilson RPA's opinion, there is no evidence of bias, and therefore the data is appropriate for use in Mineral Resource estimation. Global grade estimates should be robust.

2009 DRILLING

Drill core was delivered to the core processing facility by the drill crew at the end of each drill shift. The core was first processed for geotechnical characteristics. Drill-run lengths and depth measurements were checked for errors. Recovery and rock quality were then measured, together with fracture angle, abundance and type, weathering (hardness), abundance of fault gouge and breccia, and percentage of broken core. Geotechnical data was recorded on paper and archived with the other paper documentation for each hole.

The core was marked for sampling by stapling a portion of the ALS Chemex (Chemex) sample tag to the core box at the end of every interval and by a grease pencil mark where confusion may occur. The entire core interval was then placed in 6 mm, 24" x 36" poly ore bag and sealed with a zip tie (no split core is preserved). Each bag included a duplicate Chemex sample tag with bar code for tracking once in the lab, as well as the sample ID written on the outside of the bag with permanent ink. Two individual samples were then placed in rice bags (23" x 40") and sealed with steel wire loop ties. Each rice bag was clearly labeled with the address of the intended preparation lab and its order in the shipment. To ensure that sample batches were not mixed, each shipment was colour-coded using flagging tape. Samples were sent to Chemex facilities in either Terrace, BC, or North Vancouver, BC.

Where thick packages of metasedimentary rocks were encountered core was logged, but not sampled. The exception to this was instances where abundant quartz veining or alteration were observed.

Prior to 2008, the core samples were sawn with one half of the sample submitted to the laboratory and the other half remaining as a permanent record. To try to reduce any possible bias due to small sample size, the decision was made by StrataGold geologists to submit the whole core for assay during the 2008 drill program. This protocol was continued in 2009.

A total 2,749 samples were taken from 11 holes drilled during the 2009 exploration campaign. The average length of sample was 1.47 m with median length of 1.52 m. Victoria Gold estimates that the true width of mineralization is from 60% to 70% of the sample interval length (Victoria Gold, 2009). In general, core recoveries were good with Victoria reporting in excess of 90%.

Scott Wilson RPA has reviewed Victoria's written protocols for sampling at the Eagle Gold Project and is of the opinion that they are adequate and to a standard consistent with common industry practice. Sampling was conducted based on sound geological inference by qualified personnel with sample boundaries being delineated by changes in lithology, structure and/or mineralization.

Scott Wilson RPA inspected the logs, sampled core, as well as the logging and sampling facility. The samples appear to have been taken in an appropriate fashion, consistent with the documented protocols to industry-standard. In Scott Wilson RPA's opinion, the 2009 sampling has been conducted in a fashion that is appropriate for the deposit type and the mineralization. The logging has been done in a reasonable manner, by qualified personnel to a standard that is consistent with common industry practice.

13 SAMPLE PREPARATION, ANALYSES AND SECURITY

HISTORICAL DRILLING

The information from this section was derived from Snowden, 2004; Bloom, 2005; and Wardrop, 2008.

No detail is available on the sampling protocols or security for the 1991 drill program.

In 1992, sample preparation was conducted by Bondar-Clegg and consisted of the following:

- Approximately 10 kg to 20 kg of sample material from the RC drill were dried, weighed, and crushed to -10 mesh.
- This material was passed through a riffle splitter to obtain a 5 kg sub-sample that was reduced to -80 mesh using a ring-and-puck pulverizer.
- A 250 g split of that material was taken and further pulverized to 90% passing 150 mesh.
- Analysis by atomic absorption spectroscopy (AAS) using a 1 assay-ton charge with any results greater than 2 g/t Au being reanalyzed by fire assay (FA).

Assay analysis in 1993 was done by Chemex of Vancouver. Sample procedures remained essentially unchanged from 1992, with the exception of pulverizing of the -10 mesh crushed material. Chemex employed a disc-type grinder to reduce the material to -80 mesh in lieu of the ring-and-puck pulverizer used by Bondar-Clegg.

Chemex continued to analyze the samples in 1995 with a modification to the grinding procedure. Intermediate grinding to 60% passing -60 mesh screen using a modified Bico rotary pulverizer was introduced. In addition, a coarse reject duplicate assay was done by Min-En Laboratories (Min-En) in Vancouver.

In 1996 assay protocols were modified to account for lithology. Sample intervals within the metasedimentary lithological unit were analyzed using a one assay-ton charge with an AAS finish. Samples within the granodiorite were subject to the following (Snowden, 2004):

- Samples were crushed to 60% passing -10 mesh.
- Samples were split to 2 kg using a riffle splitter with the reject saved.
- 2 kg sample was split using a riffle splitter into two 1 kg sub-samples with one archived and the other assayed.
- Screening of 1 kg sub-sample, with the oversized assayed in entirety by Screen Fire Assay (SFA).
- The undersize was split to approximately 200 g to 250 g using a ¼ riffle splitter with the reject saved.
- Samples were placed in an envelope and sent to Chemex.
- At the assay laboratory, the samples were re-blended after screening by repeatedly passing them through a 1/8 microsplitter on a vibrating base and recombining.
- The recombined sample was then riffle split until a sub-sample between 25 g and 50 g was obtained.
- This final split was assayed in entirety allowing the weight to vary (variable fusion weight).

In 2005, sample preparation and assaying procedures consisted of the following:

- Split core samples, from 1 kg to 5 kg in weight, were shipped to Chemex.
- The entire sample was crushed to 70% passing 2 mm.
- A 250 g sub-sample was obtained from a riffle splitter and pulverized to better than 85% passing 75 microns.
- Pulps were analyzed for gold by FA on a 30 g aliquot with an Inductively Coupled Plasma (ICP) finish and analyzed for a multi-element suite by four-acid digestion and ICP-Optical Emission Spectroscopy (OES) finish.

Lynda Bloom of Analytical Solutions Ltd. was retained to review the sampling and quality control procedures. Recommendation based on that review were implemented in 2006 and carried through 2008. Procedures for sample preparation included:

- 1 kg to 5 kg split core samples were crushed to 85% passing 2 mm and passed through a riffle splitter to obtain a 1 kg sub-sample.
- From this 1 kg homogenized split, a 150 g sample was collected by taking scoops at right angles.
- The remaining 850 g reject was stored in Terrace, BC.

- The 150 g sample was placed in an envelope and shipped to Chemex in Vancouver.
- A 50 g sub-sample was taken from this 150 g sample by withdrawing two to three scoops of material from different places in the envelope.
- The 50 g sample was subjected to a gold FA with AAS and a 27 element ICP consisting of a four-acid “near total” digestion by HF-HNO_3-$HClO_4$ acid digestion, HCl leach and ICP-Atomic Emission Spectroscopy (AES).
- All results with gold greater than 10 ppm were subjected to a FA with a gravimetric finish (Wardrop, 2009).

No details are available in previous Technical Reports regarding security procedures implemented during the previous drilling campaigns. However, Scott Wilson RPA has no reason to believe that any unauthorized access to the samples occurred.

2009 DRILLING

Victoria Gold has written protocols for core handling and security that Scott Wilson RPA has reviewed and summarized below.

Drill core was delivered to the core processing facility by the drill crew at the end of each drill shift. The core logging facility was staffed continuously by Victoria Gold personnel and access was restricted to authorized individuals.

All samples were shipped from site to Whitehorse by Small’s Expediting Services, a third party expediting company, which ensured transport to the designated preparation laboratory via Byers Transport Services. A chain of custody form accompanied each shipment to its final destination, along with a sample submittal form that described the samples being sent and designating the required analytical procedures. Upon receipt of the samples, Chemex would notify Victoria Gold personnel via email. In Scott Wilson RPA’s opinion, the core was transported and handled in a safe and secure manner.

Chemex has received ISO 9001:2000 certification at all of its locations across six continents.

Upon receipt at the laboratory preparation facility, the following procedures were followed (with laboratory codes in parentheses):

- Samples were sorted, weighed, and entered into the Sample Receiving Log.
- Crushing by CRU36 method (reduction to 85% passing 2 mm).
- Splitting to 1 kg using a riffle splitter (SPL-21).
- 1 kg split was pulverized using ring mill pulverizer with a chrome steel ring set until 85% of sample passes – 200 mesh (i.e., 75 µm) (PUL-32).
- A 0.25 g charge was digested with perchloric, nitric, hydrofluoric, and hydrochloric acids.
- The residue from the digestion was topped up with dilute hydrochloric acid and the resulting solution was analyzed via ICP-AES, yielding results for 33 elements.
- Gold results were analyzed by FA with AAS (Au-AA24) on a 50 g charge.
- The prepared sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica, and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead.
- The bead was digested in 0.5 mL dilute nitric acid in a microwave oven followed by a second digestion in 0.5 mL hydrochloric acid in the microwave oven at a lower setting.
- The digested solution was cooled and diluted with demineralized water to a volume of 4 mL and analyzed by AA against matrix match standards.
- Upper detection limit for Au-AA24 was 10 ppb gold. The default overlimit method of analysis was Au-GRA22, which involves a gravimetric finish.
- After the cupellation stage, the bead was weighed and then treated with hot diluted nitric acid to dissolve the silver.
- The metallic gold was washed, annealed, cooled, and weighed. Silver concentrations were calculated from the initial weight of the bead, correcting for the inquart and reported along with the gold determination.

Where visible gold was noted by the geologist in the logging phase, the sample containing the gold, as well as those preceding and following, are subjected to SFA method of analysis (Au-SCR24). The procedure is as follows:

- The same procedure was followed to reduce the sample to 1 kg of pulverized material.

- The 1 kg split was passed through a Tyler 150 mesh (100 µm) stainless steel screen.
- Any material remaining in the screen (particle size greater than 100 µm) was analyzed in its entirety using FA with gravimetric finish and reported as the Au (+) fraction.
- The material passing through the screen was homogenized and split into two sub-samples (a nominal 50 g sample weight was used).
- The two sub-samples were analyzed by FA with AA finish. The average of the two sub-sample determinations was reported as the Au (-) fraction.
- All three determinations were used in calculating the combined total gold content.
- The weight and gold values of the Au (+) and the Au (-) fractions were reported along with the combined total gold content for the sample.

In the opinion of Scott Wilson RPA, the assay procedures employed by Victoria Gold are in adherence with common industry practice and are appropriate for the style of mineralization. Core and sample handling protocols provided a reasonable level of security, again, consistent with common practice.

14 DATA VERIFICATION

HISTORIC QA/QC PROGRAMS

The following summaries of past Quality Assurance/Quality Control (QA/QC) programs are derived from MRDI, 1997; Sparling, 2008; and Wardrop, 2009.

No information is available on QA/QC protocols, if any existed, prior to 1991. The bulk of the drilling between 1991 and 1996 was conducted during the 1995 and 1996 campaigns. Assay and QA/QC methodologies evolved primarily due to the gold grades (in the 1 g/t Au range) and the recognition of the coarse nature of the gold mineralization at the Eagle Zone.

1991

Limited sample studies were conducted, indicating that 300 g screen fire assay, 5 assay-ton fire assay on splits of -80 mesh material, and 2 assay-ton or 3 assay-ton determinations on splits of 150 mesh material all demonstrated similar levels of precision. Any of these methods were deemed superior to a 1 assay-ton result in terms of precision (reproducibility of result).

QA/QC protocols consisted of inserting an Amax “control pulp,” which was used in lieu of running check assays at another laboratory (MRDI, 1997). Scott Wilson RPA notes that there is no reference to these “control pulps” having any certification, so no conclusion can be drawn as to their adequacy in assessing accuracy.

1992

A mineralogical study provided information on gold particle size and distribution and confirmed the coarse nature of the particles. Drilling that year consisted of 46 RC holes, 34 of which were in the Eagle Zone. Sample preparation was performed by Bondar-Clegg.

QA/QC protocols consisted of the following:

- a “rig duplicate” every tenth sample
- the insertion of a crushed gravel blank “once per job” (presumably every twentieth sample)

- the insertion of Canmet, Nevada Bureau of Mines (NBM) and Amax Gold “control pulps” every twentieth sample
- an in-lab duplicate every tenth sample
- a coarse reject duplicate every fortieth sample

MRDI reviewed the data in 1993 and concluded that no significant overall bias was evident in the assays from the 1992 drilling (MRDI, 1997).

1993

Another mineralogical study was conducted that confirmed the classification of gold particle size at the Eagle Zone as coarse. Drilling that year was comprised of seven RC holes. Sample preparation was performed by Chemex.

QA/QC protocols were similar to those in 1992, except the gold-barren crushed gravel blank was inserted following suspected high-grade intervals and at random intervals. MRDI reports that the “control pulps” were within five percent of the sought values (MRDI, 1997).

1995

QA/QC procedures were modified to include 510 screen fire re-assay, using a -60 mesh coarse reject, on any sample result greater than 2 g/t Au on the initial 1 assay-ton fire assay and the submission of a 275 screen “coarse reject” duplicate assay (a second split of the -60 mesh reject) to an outside laboratory, Min-En . Additionally, 55 check assays were conducted on pulp duplicates by a third laboratory, Cone Geochemical (Cone) using a 1 assay-ton fire assay. Chemex also performed in-house same pulp duplicate analysis.

The screen fire assay on sample results greater than 2 g/t Au returned results about 15% below the originals. MRDI conducted further studies on this and concluded that the selection criteria for re-assay by screen fire assay were biased. Another bias was introduced due to change in procedure used for the re-assay.

MRDI also found poor reproducibility between the Chemex and Min-En coarse rejects using the 1 assay-ton charge, with the Min-En results an average of 12% lower when assay means were compared. MRDI did follow-up studies to determine the nature of the bias, and concluded there were two sources; analytical and sample preparation. Further investigation lead MRDI to conclude that the application of a 9.7 g/t Au grade cap appeared to adequately compensate for disparity between results.

MRDI concluded that no bias was found between the means of the original and Cone check assays, however, the gold distribution was not sufficiently homogenous to obtain good reproducibility using 1 assay-ton charges, based on the results from the Chemex in-house same pulp duplicates. The Cone analysis confirmed this fact. Cone also confirmed that the pulps were made to specification by performing a wet screen analysis on a subset of the submitted pulps.

The poor replicate assay results triggered modifications to the preparation procedure that commenced in 1996. The entire sample was reduced to -10 mesh and a 2 kg split was ground to 90% passing a 100 mesh screen. The resulting pulp was split with one-half subjected to a screen fire assay. The remaining undersized fraction was passed through a riffle splitter and the resulting split was fire assayed along with the oversized fraction (MRDI, 1997).

1996

The Eagle Zone drilling program consisted of 54 core holes for exploration/definition (of which 21 targeted the Eagle Zone) and 37 RC holes.

QA/QC measures for the program included:

- internal same-pulp replicate assays by Chemex
- “rig duplicates” (frequency not given, but presumed to be every tenth sample for the RC holes)
- the insertion of certified reference materials (CRM) every twentieth sample
- the insertion of blanks
- same pulp check assays by a second laboratory (Cone) using a 1 assay-ton fire assay and two duplicate screen fire assays, one at the -10 mesh stage (coarse) and one using the minus fraction passing the 100 mesh screen

MRDI concluded that all duplicates showed the same level of precision and that the use of the screen fire assay procedure had improved this precision. No bias was found in duplicate results. Blank results showed that there were no systematic contamination issues, and reference material returned values within an acceptable range (MRDI, 1997).

1997 TO 2004

No drilling was conducted on the property from 1997 to 2004.

2005 TO 2007

QA/QC protocols for these years consisted of the following:

- One field duplicate every twentieth sample (sawn quarter core sample).
- One preparation duplicate for every twentieth sample (the sample was crushed and divided into thirds; two samples were pulverized and assayed as the original and a duplicate, the other coarse third was sent to ACME to be pulverized and assayed).
- One check assay for every twentieth sample (the coarse split from Chemex that was pulverized and assayed by ACME).
- One CRM standard for every twentieth sample.
- One blank for every twentieth sample.

The same standards were used between 2005 and 2007 and the blank material was commercially purchased crushed dolomite.

Sparling reports that in 2005, no systematic contamination was observed, based on the results from the blanks. Standard reference material assays were with ±7% of the expected values, and preparation duplicate results showed no bias. The field duplicates showed poor reproducibility, as did the preparation duplicate/check assay determinations. While precision was poor, the relative percent differences (RPD) were generally split between positive and negative indicating a lack of analytical bias (Sparling, 2008).

QA/QC protocols for blanks and reference standards were unchanged for 2006. Sparling reports that no systematic contamination was evident based on the blanks submitted, and that the CRM assays returned values within ±7% of the expected values.

Sparling also concluded that the preparation duplicates results indicated a lack of analytical bias, since the RPD was generally split equally between positive and negative values. The field duplicates data indicated a bias toward higher assays from the original sample. This lack of precision is consistent with gold deposits having coarse gold mineralization.

QA/QC protocols remained unchanged for 2007. Results from the blanks inserted into the sample stream did not show any systematic bias. CRM standard results were within ±6% of the expected values. Sparling concluded that the preparation duplicates/check assay results show a small bias toward a higher original assay. This was also seen in the field duplicates data, where the original assays are higher than the duplicates.

2008

QA/QC protocols for 2008 consisted the following:

- One laboratory duplicate for every twentieth sample (the sample was crushed and pulverized then divided into thirds; two samples were assayed by Chemex as the original and a "check assay", the other third was sent to Acme Analytical Labs (ACME) to be assayed).
- One duplicate assay for every twentieth sample (the pulp from Chemex that was assayed by ACME).
- One CRM standard for every twentieth sample.
- One commercially-available crushed dolomite blank for every twentieth sample.
- No field duplicates were taken as whole core was sampled.

Wardrop did not comment on any results for the blank and CRM standard analysis but did report, graphically, the results of the duplicate sample submission to ACME. The graph shows a small bias toward a higher value derived from the original assay compared to the duplicate. These finding are consistent with past results and are thought to be due to the coarse nature of the gold mineralization at Eagle (Wardrop, 2009).

It is the opinion of Scott Wilson RPA that historic QA/QC protocols either met or exceeded industry-standard when they were executed and that the assay data is adequate to be included in a Mineral Resource estimate.

2009 QA/QC PROGRAM

The 2009 program included insertion of prepared CRM standards, blanks, duplicates and lab duplicates into the sample stream. For every 20 samples, a duplicate was inserted as the second sample, a lab duplicate as the ninth sample, a CRM standard as the tenth sample and a blank as the sixteenth sample. This routine was repeated after every twentieth sample for the entirety of the drill program. Additional blank samples were inserted following a sample with visible gold. Samples containing visible gold, as well the samples immediately preceding and following, were analyzed using SFA.

The duplicate was taken from the same pulp as the original sample and submitted to ACME, an ISO 9000 accredited laboratory for analysis. Lab duplicates consist of a second sample taken from the same pulp and reported as an independent determination by Chemex. Blank samples consisted of white dolomite chips or washed quartz gravel. Eighteen prepared CRM standards were used, representing typical low to high grade gold ore. These are summarized in Table 14-1.

TABLE 14-1 CERTIFIED REFERENCE MATERIAL STANDARDS
Victoria Gold Corp. – Eagle Gold Project

Internal Code	Producer	Certified Reference	95% Confidence Interval (=1.96 Standard Deviation) Recommended Au_ppm	Low Au_ppm	High Au_ppm	Number Submitted
Standard 1A	Canmet	CH-4	0.88	0.80	0.96	6
Standard 2B	Canmet	Ma-2C	3.02	2.96	3.08	3
Standard 3C	Oreas	42P	0.091	0.088	0.094	18*
Standard 4D	Oreas	51P	0.43	0.417	0.443	1
Standard 5E	Oreas	6Pb	1.422	1.396	1.448	8
Standard 6F	Oreas	7Pb	2.77	2.75	2.79	18
Standard 7G	Oreas	61Pa	4.46	4.39	4.54	13
Standard 8M	Oreas	15Pa	1.02	1.00	1.03	1
Standard 9I	Oreas	15Pc	1.61	1.58	1.63	4*
Standard 10J	Oreas	50Pb	0.841	0.825	0.857	1
Standard 11K	Oreas	53Pb	0.623	0.612	0.634	3
Standard 12L	Oreas	60b	2.57	2.52	2.61	10
Standard 13N	Oreas	2Pd	0.885	0.872	0.898	21**
Standard 14P	Oreas	6Pc	1.52	1.49	1.56	20
Standard 15Q	Oreas	4Pb	0.049	0.048	0.050	11
Standard 16R	CDN	CDN-GS-P8	0.78	0.72	0.84	11
Standard 17S	CDN	CDN-GS-1E	1.16	1.1	1.22	10
Standard 18T	CDN	CDN-GS-2E	1.52	1.38	1.66	12

* A blank was erroneously inserted as a CRM. This was identified and the results were not included in the tabulation.
** Two blanks were erroneously inserted as a CRM. These were identified and the results were not included in the tabulation.

A total of 685 QA/QC samples were inserted into the sample stream during the 2009 drilling campaign for a total of 19.95% of the samples submitted for analysis. This total was comprised of 165 blanks, 168 pulp duplicates sent to ACME, 181 pulp duplicates, and 171 CRMs. Victoria Gold personnel reviewed the QA/QC data to confirm if assay results were within acceptable parameters. Some failures were identified and investigated immediately. Scott Wilson RPA reviewed the QA/QC results to confirm that there are no systematic errors that would preclude their use for a Mineral Resource estimate.

CERTIFIED REFERENCE MATERIALS

A total of 171 submissions of CRMs were made during the 2009 campaign. Eighteen different CRMs from three different producers were included. The conventional approach to setting reference standard accuracy acceptance limits is to use the recommended mean assay value (RV) for the CRM (from round robin testing by a group of independent laboratories) ±3 standard deviations (SD) for an individual sample and a limit of RV ±2 SD for two or more adjacent samples. Victoria Gold's internal QA/QC protocols, however, called for acceptance limits defined by RV ± 10%. As a result, Victoria's acceptance limits were more tolerant, so that conventional "failures" were not identified.

Using Victoria's criterion, seven of the 171 CRM results failed (4.09% of the total), which is slightly higher than the 3%, which would be expected to fall outside the limits. Of these seven failures, four returned assayed values greater than 10% above the recommended mean. The remaining three were determined to be greater than 10% below the recommended mean. These failures were not investigated with the laboratory and the batches of samples were not rerun.

Scott Wilson RPA subjected the QA/QC results to the more conventional comparison described above and notes that there were several results outside of the "conventional" limits. Most failures were observed to be lower than the RV, which suggests that the 2009 samples may impart a conservative bias to the resource estimate. In Scott Wilson RPA's opinion, however, the degree of any possible bias is likely to be negligible.

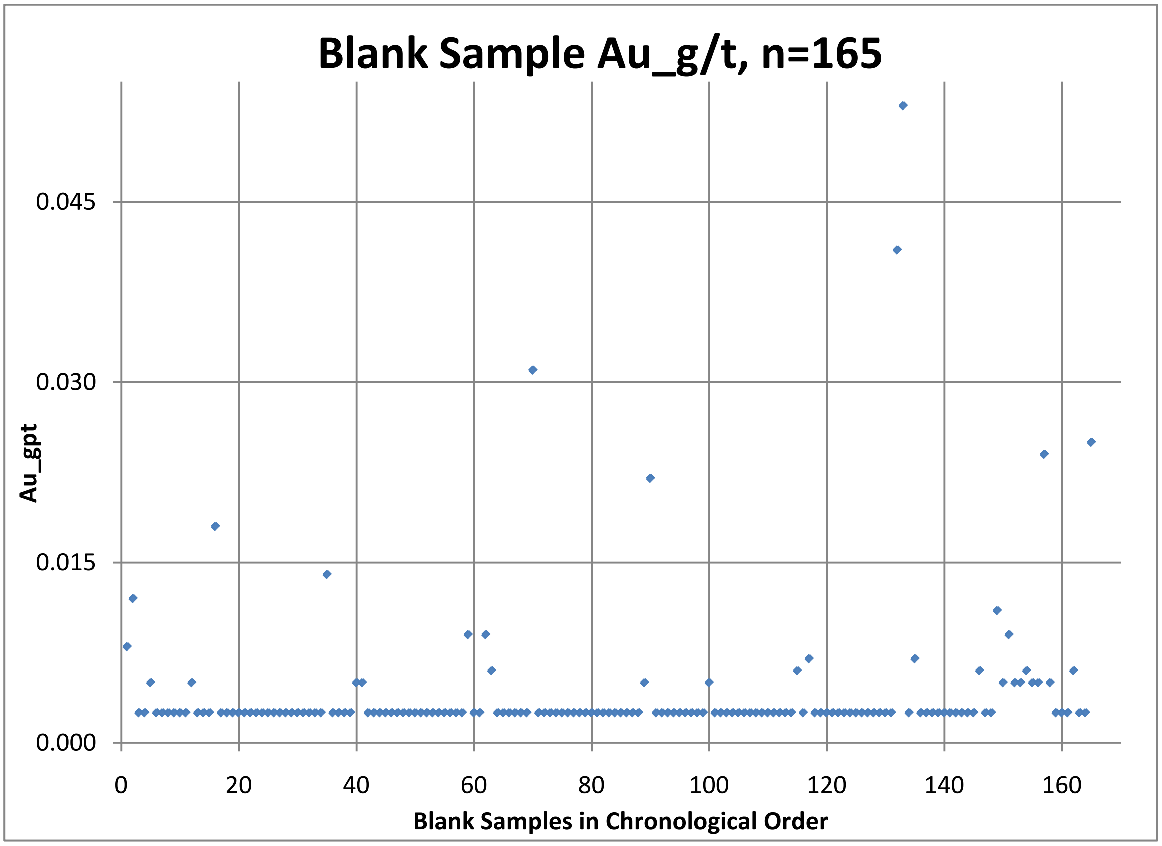
Scott Wilson RPA notes that CRMs produced by CDN Laboratories (CDN) and Canmet did not experience any failures using Victoria Gold's approach of setting acceptance limits. All the failures occurred with Oreas CRMs, with standard 4Pb experiencing the highest failure rate (three, with one low, two high). Scott Wilson RPA notes that this CRM has a very low RV with very tight acceptance limits. Victoria reports that on other projects where this CRM has been used, there have been a high number of failures. They suggest that the low grade of the CRM is more appropriate for use with soil samples. As the program progressed, fewer of the Oreas CRMs were employed.

Scott Wilson RPA recommends that the QA/QC samples be monitored upon receipt so failures can be identified and acted upon in a timely manner. Further, Scott Wilson RPA recommends that the use of the Oreas CRMs be suspended in favour of the CDN and Canmet CRMs. Scott Wilson RPA is also of the opinion that a more conventional approach to assessing assay accuracy and precision should be adopted.

BLANKS

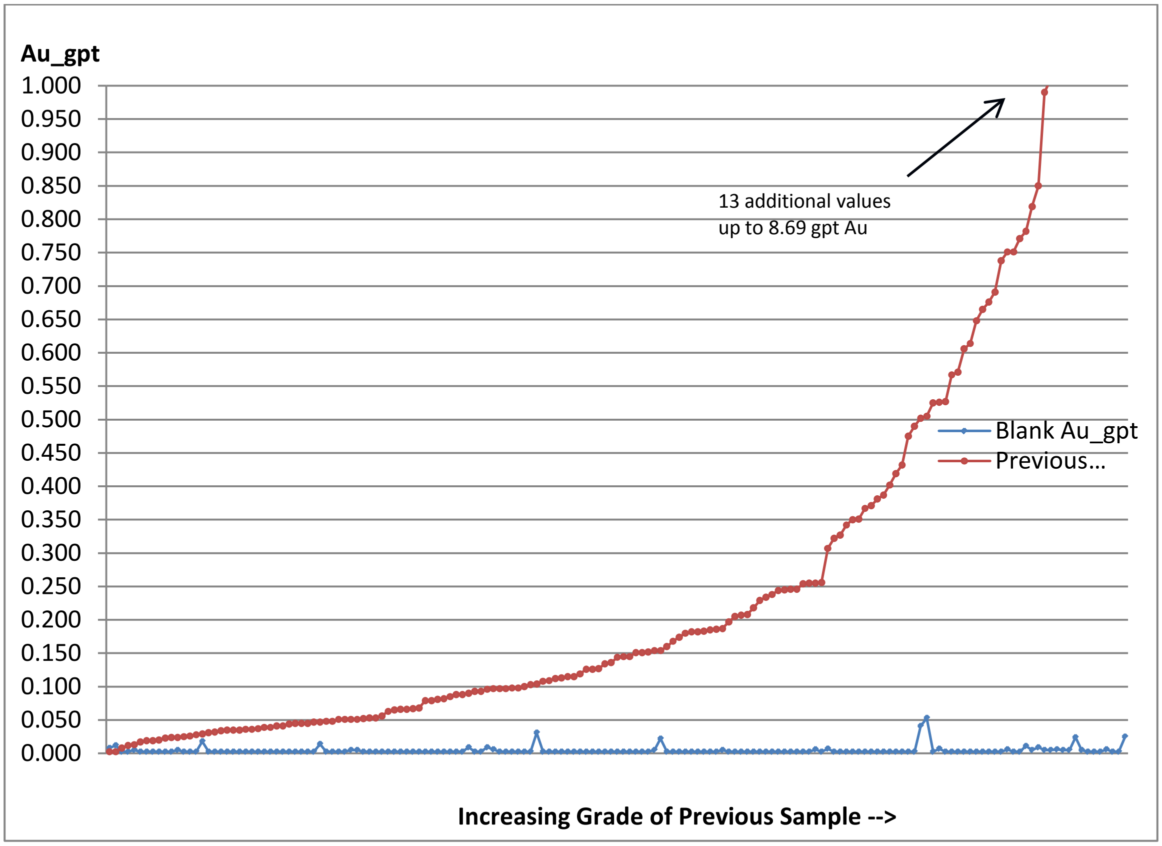
A total of 165 blanks were inserted into the sample stream. The results for these samples were plotted chronologically to determine if any trends had occurred over time. It was noted that some control samples that were submitted as blanks were later recognized to be CRMs and were re-assigned. In all, twenty-three returned values that were above the Chemex detection limit of 0.005 g/t Au and eight that were at least three times the detection limit. In Scott Wilson RPA's opinion, the higher values, even at three times the detection limit, are still very low and tend to occur at random intervals. As such, there is no evidence for systemic bias or contamination in the samples.

FIGURE 14-1 BLANK SAMPLE GRADES



To investigate if systematic contamination was present, the blank samples were paired with the values of the samples that preceded them in the sample sequence. These results were sorted in order of the grade of the preceding sample (see Figure 14-2). If the grade of the blanks assays also increases along with the grade of the preceding sample, this is evidence of systematic contamination. The resulting plot demonstrates no evidence of contamination.

FIGURE 14-2 BLANKS VERSUS PREVIOUS SAMPLE GRADE



In Scott Wilson RPA's opinion, the blanks assays are within a reasonable tolerance and do not indicate that there are any concerns.

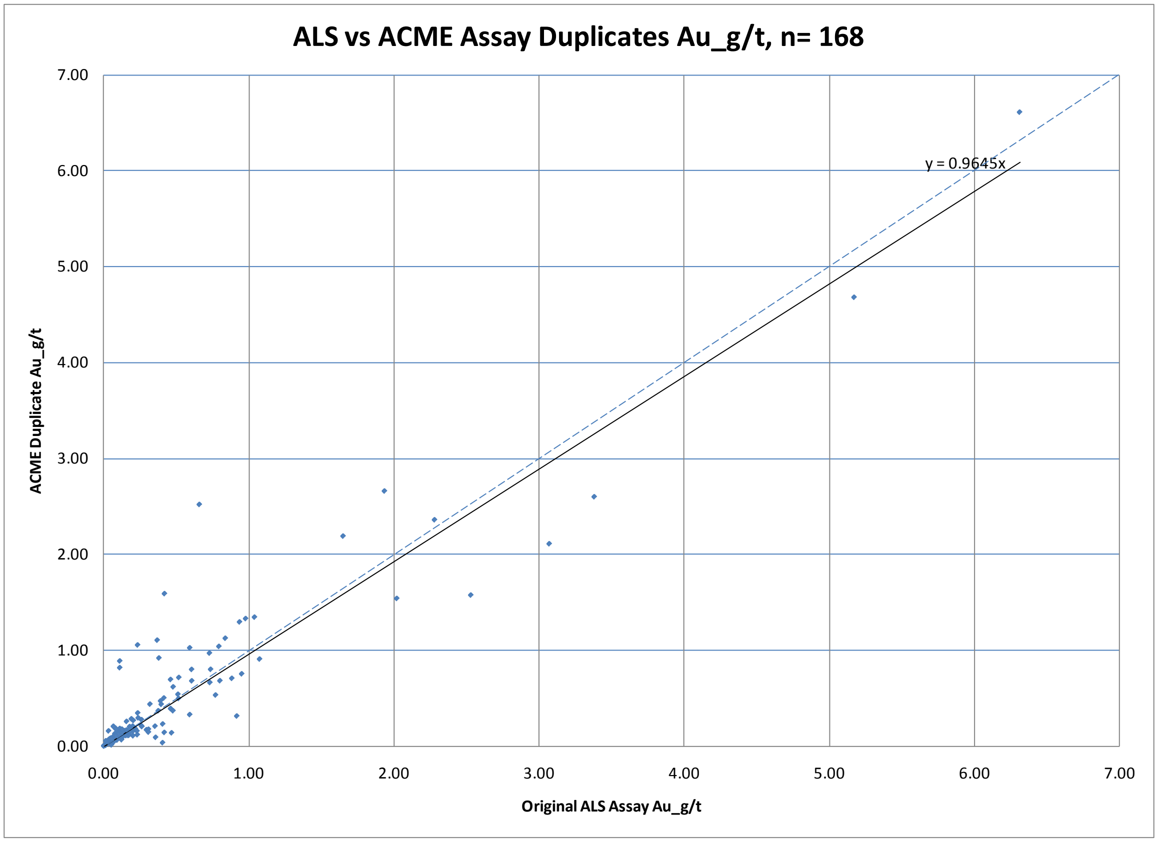
FIELD DUPLICATES

Due to the procedure of sampling whole-core, there was no opportunity for a core duplicate. Instead, duplicate sub-samples were taken from the master pulp and either sent to a second laboratory (Pulp Duplicate) or replicated at the original lab (In-Lab Duplicate). The results are discussed below.

PULP DUPLICATES

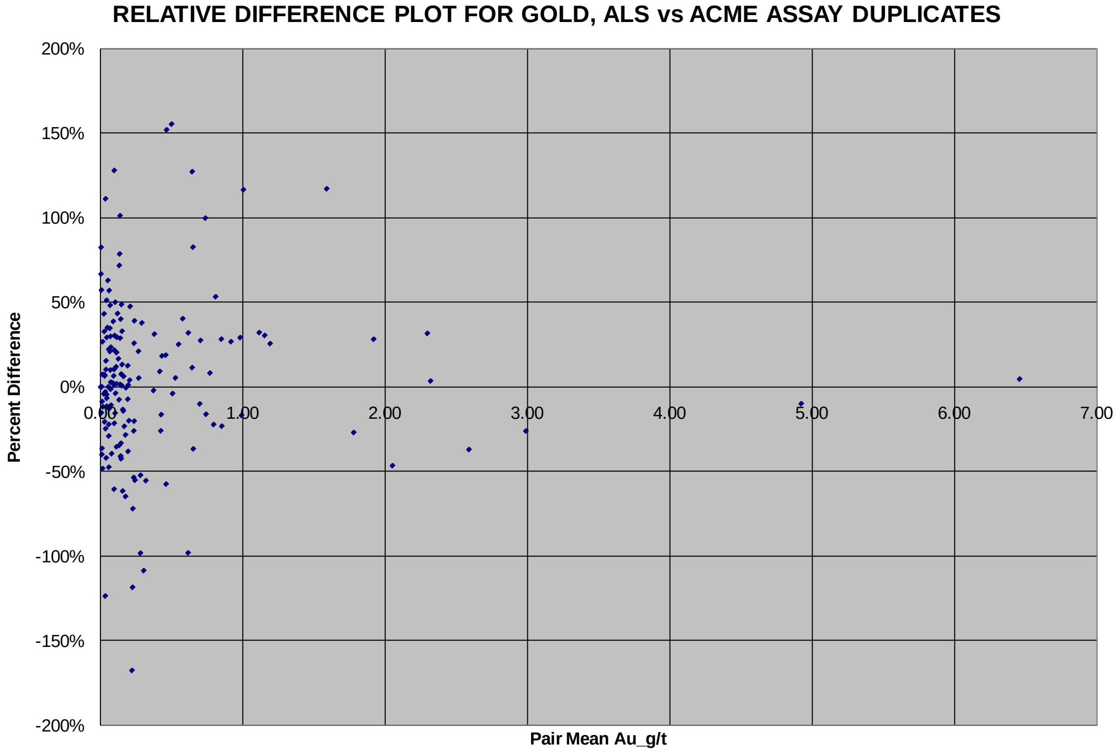
An additional 50 gram split from the master pulp was taken and forwarded to an independent laboratory (ACME) for analysis. The total number of re-assays was 168. The results from ACME were plotted against the original determinations from Chemex and are shown in Figure 14-3.

FIGURE 14-3 CHEMEX VERSUS ACME PULP DUPLICATE ASSAYS



In Scott Wilson RPA's opinion, the duplicate assays show reasonably unbiased correlation with the original assays, but with a fairly high degree of scatter. These duplicate assay results were plotted on a Relative Difference (Thompson-Howarth) diagram to investigate evidence of bias. The resulting plot shows a high degree of scatter particularly at low grades (Figure 14-4). The presence of coarse gold at Dublin Gulch could account for difference in the results at low grade. The relative pair difference, generally, remained between ± 50% for values greater than 1.0 g/t Au.

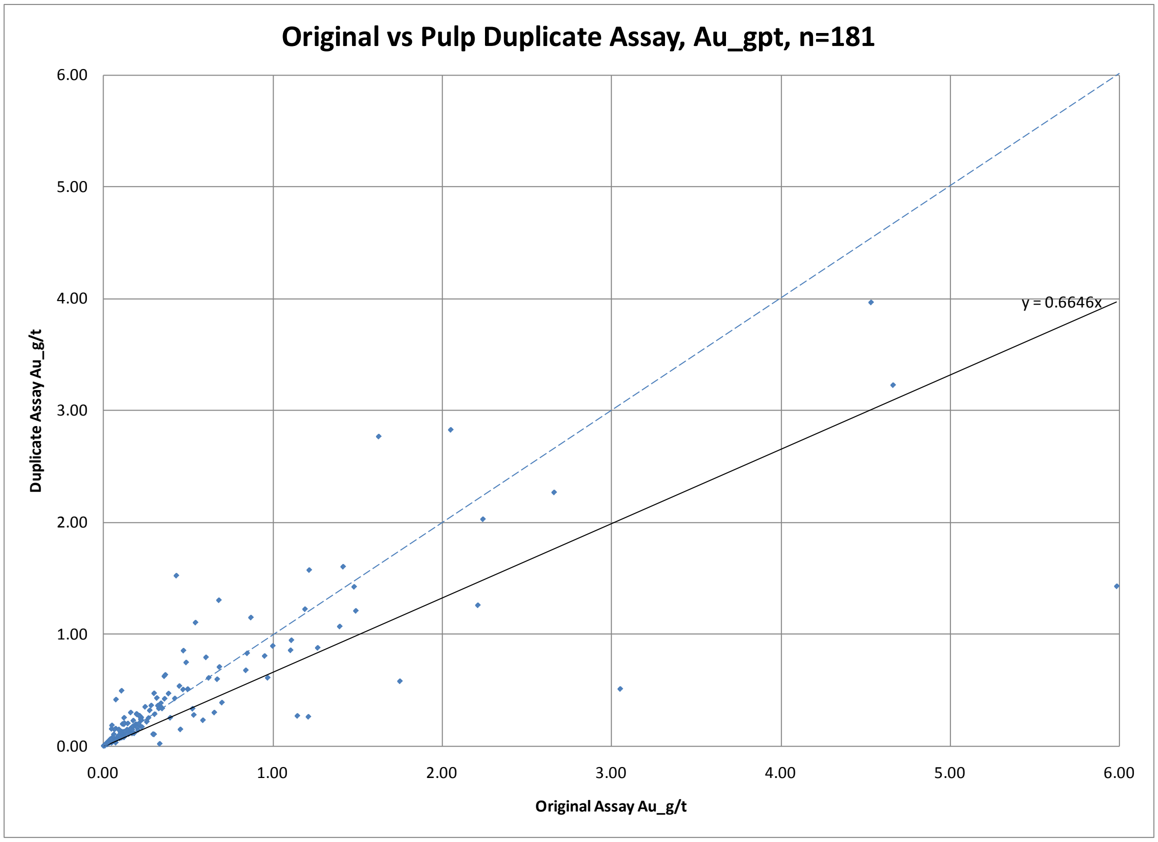
FIGURE 14-4 RELATIVE DIFFERENCE PLOT



IN-LAB DUPLICATES

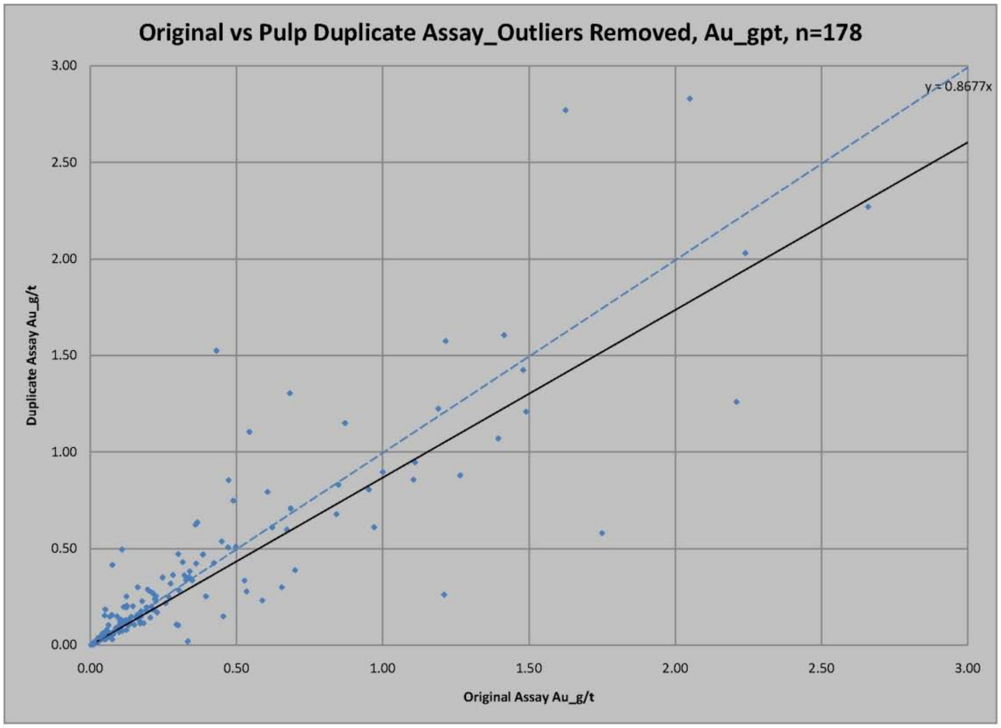
A total of 181 in-lab duplicates were taken during the 2009 drilling campaign. These were second 50 g sub-samples, taken from the master pulp and assayed by Chemex. The results of the original results plotted against the duplicate results are shown in Figure 14-5.

FIGURE 14-5 ORIGINAL VERSUS PULP DUPLICATE ASSAY



The regression line in Figure 14-5 suggests that there is a systematic bias in the pulp duplicates, however, Scott Wilson RPA notes that if three extreme results are removed, the apparent bias diminishes substantially (Figure 14-6). In Scott Wilson RPA's opinion, the pulp duplicates confirm observations by previous reviewers; that assay reproducibility may be low, but does not indicate the presence of a systematic bias.

FIGURE 14-6 ORIGINAL VERSUS PULP DUPLICATE ASSAYS WITH OUTLIERS REMOVED



SCOTT WILSON RPA CHECKS

Scott Wilson RPA selected a portion of the drill data and compared the assay database against electronic versions of assay certificates provided by Victoria Gold. The assay database used for the Mineral Resources estimate comprised 37,861 samples. A total of 5,260 samples, or 13.9% of the database, were verified.

Assay certificates were imported into a MS Access database that linked to the GEMCOM drill hole assay database. The assay certificate values were queried against the drill hole assays and any Au assay discrepancies were noted. This procedure was performed twice. The first iteration compared the drill hole assay database against the results generated from Fire Assay with AA finish or from Fire Assay with gravimetric finish, if the original result exceeded the overlimit value of 10 ppb Au. The second iteration compared the drill hole assay database against the results of SFA (Screen Fire

Assay) or, in the case of the 2009 drilling, re-assaying of chosen batches. SFA samples were designated where visible gold was noted by the geologist during the logging phase.

Discrepancies were tabulated and cross referenced against the other assay methods to reconcile any differences. From the 5,260 samples checked in this manner, 40 errors were found giving an error rate of 0.76%. These 40 errors were discovered within four drill holes. In 38 of the 40 instances, the database contained zero values for the respective sample intervals that contained values in the assay certificates. The average assayed grade of the 38 samples is 0.147 g/t Au with a median grade of 0.035 g/t Au. The two samples that exceeded the certificate grade did so by an insignificant amount. In the opinion of Scott Wilson RPA, the errors found will have negligible impact upon the Mineral Resource estimate.

The drill hole, assay, lithological and survey databases were validated using validation utilities within GEMS (resource modelling software) and found to be free of significant errors. Minor errors that were captured were corrected prior to proceeding with the Resource estimate.

15 ADJACENT PROPERTIES

There are no mineral properties adjacent to the Dublin Gulch Property.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

METALLURGICAL TESTWORK

Column leach, bottle roll leach, gravity concentration, and flotation tests were conducted on various samples from the Eagle Gold Project. Most of the cyanidation testing was conducted from 1995 to 1997 by KCA, while the gravity and flotation work was conducted in 2006 by Process Research Associates (PRA). In 2007, Analytical Solutions Ltd. (ASL) compared gold assays by fire and metallic screen assay procedures. The earlier test work was by Rescan Engineering in 1997.

The results from the column leach test program indicate that gold recovery is sensitive to crush size and, to a lesser extent, to ore type. Ore types tested included:

- Type A – weathered granodiorite, or oxidized
- Type B – fresh, or weakly altered granodiorite
- Type C – sericite altered granodiorite
- Type D – fine grained granodiorite, trace sulphides
- Type E – weathered sediments

Overall gold recoveries ranged from 40% to 45% at a P_{80} crush size of approximately 35 mm up to 80% to 85% at a P_{80} crush size of 2 mm. In KCA's opinion, crushing down to a P_{80} size of 2 mm is not a viable heap leach process option for Eagle Gold. Therefore, based on crush size and crusher type versus net revenue trade-off studies, a 5 mm P_{80} crush size was chosen for the pre-feasibility study.

The column leach test results show that crushing down to a P_{80} size of 5 mm with High Pressure Grinding Roll (HPGR) crushers will lead to an overall gold recovery of 72%. The results are preliminary and additional test work is required, however, the use of HPGR crushers appears to increase gold recovery by several percentage points as compared to conventional crushing to the same P_{80} crush size.

Silver grades are relatively low, and were not included in the resource block model or production schedule. Silver recoveries were followed in the testing program, however, these results are not presented in detail in this section. Based on the column leach test results, total silver recovered is generally on the order of 4 to 5 times less than total gold recovered. Revenue for silver has been included in the cash flow model, but does not make a significant economic contribution.

Column leach tests were conducted at freezing conditions and compared to ambient temperature column leach test results. These tests showed similar results.

NaCN requirements were estimated to average 0.34 kg/t at a 5 mm crush size. Lime requirements were estimated to average 1.0 kg/t. However, preliminary agglomeration tests indicate that a minor amount of cement may be required in the lower lifts of the multi-lift heap leach operation. Additional testwork is required, but up to 2 kg/t of cement may be required during the first couple of years of operation.

Gravity and flotation tests were conducted on various ore types at grind sizes ranging from approximately 63 microns to 147 microns. The results of these tests indicated high gold recoveries, with minor reductions in recovery with increasing grind size. Gravity concentration with flotation of the gravity tailings resulted in overall average gold recoveries of 95%. Gold recovery in the gravity stage varied somewhat between the various ore types, but averaged about 52%. Gravity concentration with cyanidation of the gravity tailings resulted in gold recoveries ranging from 85 to 95%.

The results of the KCA's leach test work are presented in Tables 16-1 and 16-2, and in Figures 16-1 through 16-3. PRA's gravity and flotation results are presented in Tables 16-3 and 16-4.

TABLE 16-1 KCA COLUMN LEACH TEST RESULTS
Victoria Gold Corp. – Eagle Gold Project

Test No.	Report Date	Ore Type	% of Each Type	Crush Type	p80 Crush Size, mm	Calc Hd g/t Au	% Au Recovery	% Au w project'ns	Leach Time Days	NaCN kg/t	Lime kg/t	Cement kg/t	p20 Crush Size mm
23030	Apr 97	A,C	60/40	Conventional	4.8	0.57	80.6%	**82.6%**	61	0.77	1.14	--	0.6
23018	Apr 97	A	100	Conventional	5.0	0.88	73.9%	73.9%	85	0.90	1.00	--	0.75
23021	Apr 97	A,B,C	37/40/23	Conventional	5.2	0.76	71.0%	**72.0%**	85	0.91	1.00	--	0.78
23027	Apr 97	B	100	Conventional	5.2	0.80	62.6%	**64.6%**	61	0.72	1.07	--	0.8
22660	Apr 96	A	100	Conventional	9.9	0.94	62.8%	**63.8%**	88	0.40	1.00	--	1.7
22666	Apr 96	A,B,C,D	Na	Conventional	9.9	1.21	66.1%	66.1%	115	0.53	1.03	--	1.8
22662	Apr 96	B	100	Conventional	10.2	0.81	58.0%	**59.0%**	115	0.48	1.03	--	1.6
22664	Apr 96	C	100	Conventional	10.3	0.63	61.9%	61.9%	115	0.55	1.03	--	1.6
22687	Apr 96	A	100	Conventional Cold*	10.3	0.90	60.0%	60.0%	84	0.25	1.00	--	1.7
22656	Apr 96	C	100	Conventional	29	0.66	42.5%	**43.5%**	62	0.16	1.05	--	4.8
22650	Apr 96	C, D	Na	Conventional	33	0.65	43.1%	43.1%	62	0.17	1.05	--	3
22685	Apr 96	A	100	Conventional Cold*	36.3	0.73	58.9%	58.9%	84	0.10	1.00	--	6.4
22652	Apr 96	A	100	Conventional	38	0.75	48.1%	48.1%	91	0.15	1.05	--	7.5
22654	Apr 96	B	100	Conventional	38	0.78	33.3%	**34.3%**	62	0.12	1.05	--	7.5
23036	Apr 97	A	100	Cone-HPGR	1.7	0.87	88.5%	**90.5%**	34	0.70	0.08	3.75	0.60
24640	Apr 97	A	100	Cone-HPGR	1.8	1.67	88.0%	88.0%	58	0.72	1.07	1.00	0.11
24610	Apr 97	A,C	84/16	Cone-HPGR	2.0	0.85	85.9%	85.9%	76	1.07	0.15	3.75	0.11
23075	Apr 97	A	100	Cone-HPGR	2.1	1.06	87.7%	**89.7%**	41	0.58	0.07	3.75	0.14
23078	Apr 97	A	100	Cone-HPGR	2.1	1.08	78.7%	**83.7%**	23	0.40	0.07	3.75	0.14

Test No.	Report Date	Ore Type	% of Each Type	Crush Type	p80 Crush Size, mm	Calc Hd g/t Au	% Au Recovery	% Au w project'ns	Leach Time Days	NaCN kg/t	Lime kg/t	Cement kg/t	p20 Crush Size mm
24604	Apr 97	A	100	Cone-HPGR	2.2	1.80	89.4%	89.4%	76	0.92	0.15	3.75	0.22
24607	Apr 97	B	100	Cone-HPGR	2.2	1.52	85.5%	85.5%	76	0.81	0.15	3.75	0.15
24637	Apr 97	A,B,C	25/50/25	Cone-HPGR	2.2	0.66	83.3%	83.3%	58	0.81	1.07	1.00	0.12
24601	Apr 97	E	100	Cone-HPGR	2.6	0.76	84.3%	84.3%	76	0.99	0.15	3.75	0.21
23057	Apr 97	A	100	Jaw-HPGR 1	9.5	1.03	76.7%	76.7%	27	0.34	1.08	--	0.12
24625	Apr 97	A,B,C	25/50/25	Jaw-HPGR 1	9.5	0.69	65.2%	**69.2%**	31	0.57	1.07	1.00	0.50
24628	Apr 97	A	100	Jaw-HPGR 1	9.5	1.69	58.6%	**62.6%**	31	0.61	1.07	1.00	0.48
24631	Apr 97	A,B,C	25/50/25	Jaw-HPGR 1	9.6	0.60	66.6%	66.6%	58	0.59	1.08	1.00	0.62
24634	Apr 97	A	100	Jaw-HPGR 1	9.5	1.73	64.1%	**65.1%**	72	0.79	1.08	1.00	0.40
23060	Apr 97	A	100	Jaw-HPGR 2	4.8	1.00	79.0%	**80.0%**	27	0.43	0.10	3.75	0.19
24619	Apr 97	A,C	84/16	Barmac	5.0	0.97	85.6%	85.6%	60	0.84	0.10	3.75	0.30

* Cold temperature leach, approx 0 C

1 indicates single pass, 2 double pass through a HPGR

TABLE 16-2 KCA BOTTLE ROLL LEACH TEST RESULTS
Victoria Gold Corp. – Eagle Gold Project

Test No.	Ore Type	Hole No.	Interval m	Crush Size mm	Calc Hd g/t Au	% Au Recovery	Leach Time hrs	NaCN kg/t	Lime kg/t
22648A	C, D	na	na	50	0.96	22.9	96	0.29	0.15
22648B	A	na	na	50	0.94	35.4	96	0.19	0.15
22668A	A	na	na	25	1.15	31.3	96	0.14	0.30
22669A	A	na	na	12.5	0.65	46.2	96	0.09	0.30
22670A	A	na	na	Pulv	0.68	92.6	24	0.08	1.60
22648C	B	na	na	50	0.74	29.7	96	0.14	0.25
22668B	B	na	na	25	0.61	26.2	96	0.14	0.20
22669B	B	na	na	12.5	0.73	32.9	96	0.14	0.20
22670B	B	na	na	Pulv	0.52	90.4	24	0.08	1.00
22649A	C	na	na	50	0.3	40.0	96	0.14	0.25
22668C	C	na	na	25	0.66	25.8	96	0.14	0.30
22669C	C	na	na	12.5	0.83	22.9	96	0.14	0.30
22670C	C	na	na	Pulv	0.44	90.4	24	0.18	1.00
22668D	A,B,C,D	na	na	25	1.06	43.4	96	0.14	0.25
22669D	A,B,C,D	na	na	12.5	1.32	42.4	96	0.19	0.25
22670D	A,B,C,D	na	na	Pulv	1.59	96.9	48	0.08	2.00
22626A	A	74C	9.1 - 47.3	Pulv	0.45	80.0	48	0.04	1.60
22626B	C	74C	47.3 - 99.1	Pulv	0.52	80.7	48	<0.01	1.40
22626C	B	74C	99.1 - 177.2	Pulv	0.38	71.1	48	0.15	1.20
22626D	C	74C	177.2 - 229.2 236.4 - 243.2	Pulv	0.33	78.8	48	0.04	1.60
22627A	B	74C	220.2 - 236.4	Pulv	0.33	66.7	48	0.15	1.40
22627B	E	75C	9.15 - 12.9	Pulv	<0.10	0.0	48	0.15	1.60
22627C	A	75C	12.9 - 13.9	Pulv	<0.10	0.0	48	0.05	1.60
22627D	E	75C	13.9 - 25.9	Pulv	0.29	75.9	48	0.15	1.00
22628A	A	75C	25.9 - 49.4	Pulv	0.76	84.2	48	0.04	1.60
22628B	E	75C	49.4 - 53.6	Pulv	0.6	88.3	48	0.15	1.60
22628C	A	75C	53.6 - 67.1	Pulv	0.85	85.9	48	0.05	1.60
22628D	E	75C	67.1 - 76.6	Pulv	0.98	87.8	48	0.15	1.60
22629A	A	75C	76.6 - 120.2	Pulv	0.65	86.2	48	0.05	2.00
22629B	D	75C	120.2 - 130.8	Pulv	0.68	76.5	48	0.05	1.60
22629C	B	75C	130.8 - 234.4	Pulv	1.1	89.1	48	0.15	1.60
22629D	A	76C	22.9 - 190.0	Pulv	0.49	89.8	48	0.05	1.60
22630A	C	76C	190.0 - 260.0	Pulv	0.6	85.0	48	0.05	1.00
22630B	A	77C	24.4 - 129.5	Pulv	2.01	95.0	48	0.05	1.60
22630C	B	77C	129.5 - 199.6	Pulv	0.59	84.7	48	0.15	1.60
22630D	B	78C	24.4 - 38.2	Pulv	0.28	75.0	48	0.05	1.60
22631A	C	78C	38.2 - 100.6	Pulv	0.48	89.6	48	0.05	1.60

FIGURE 16-1 COLUMN LEACH TEST RESULTS

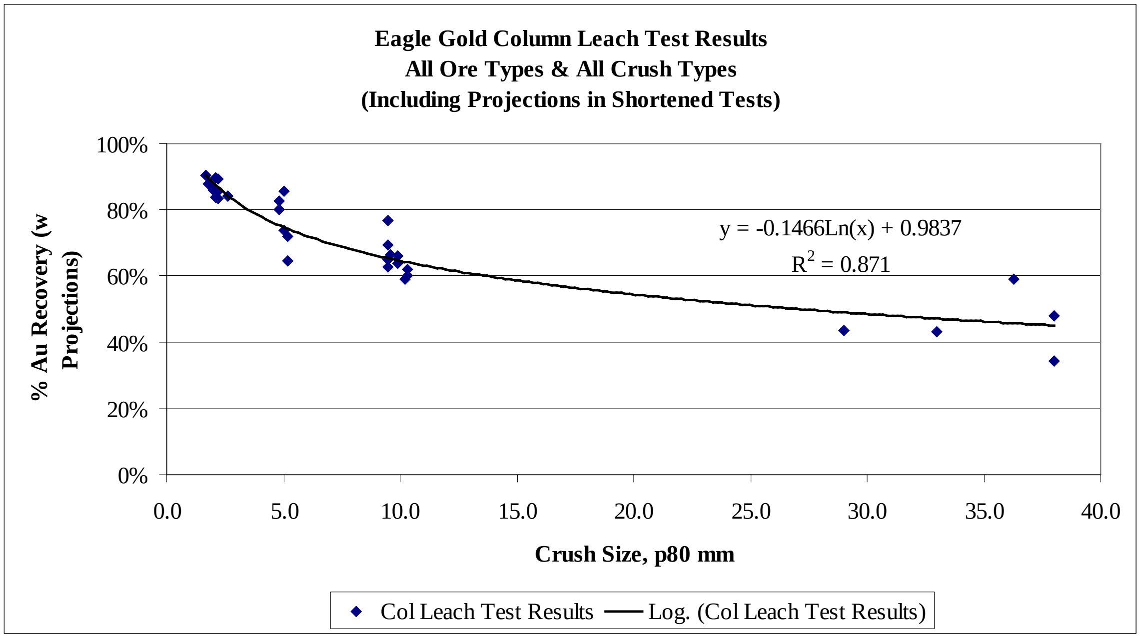


FIGURE 16-2 COLUMN LEACH TEST RESULTS – COMPOSITES

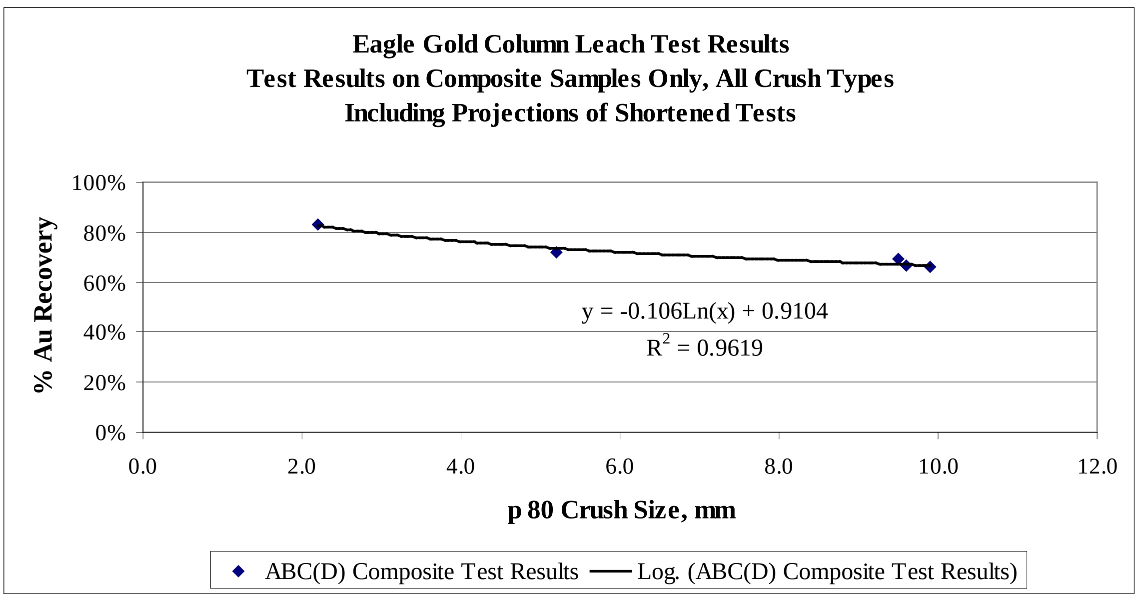


FIGURE 16-3 RECOVERIES AT P_{20} CRUSH SIZES

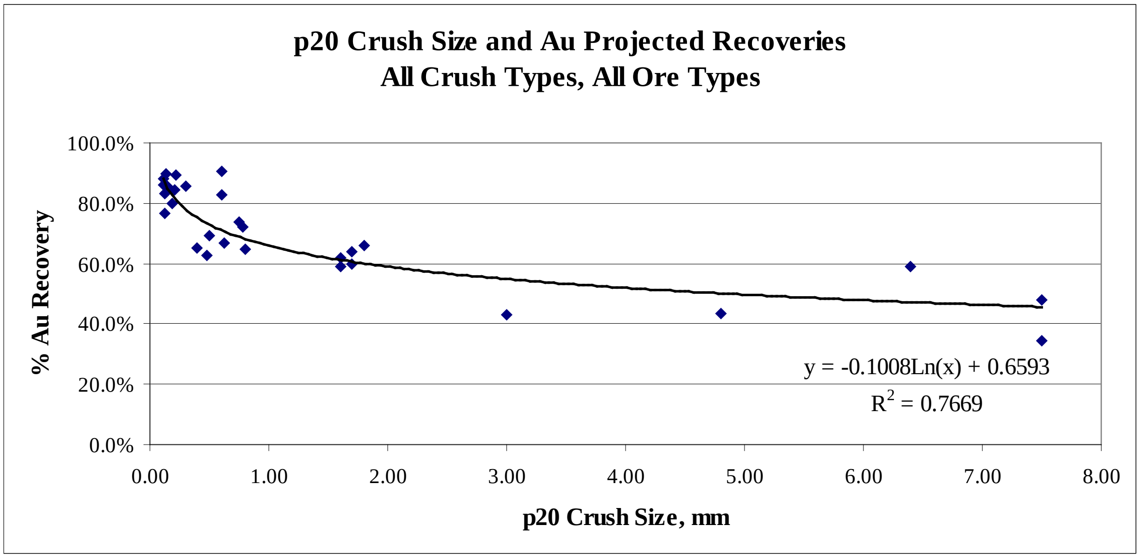


TABLE 16-3 SUMMARY OF PRA'S GRAVITY-FLOTATION TEST RESULTS

Victoria Gold Corp. – Eagle Gold Project

Ore Type	Grind Size p80, µm	Calc Head g/t Au	Gravity Conc g/t Au	Final R. Conc g/t Au	Mass Pull %	Gravity Conc Rec. %	Final R. Conc Rec. %	Overall Rec. %
Altered	79	0.89	188.0	4.40	10.2	44.6	50.3	95.0
Altered	107	0.94	130.8	5.42	6.7	56.7	38.4	95.1
Altered	147	0.94	101.9	8.63	6.1	32.7	55.4	88.1
Oxidized	63	1.12	817.4	2.62	11.3	70.3	26.5	96.8
Oxidized	101	0.66	434.0	2.45	9.8	57.1	36.1	93.2
Oxidized	140	1.01	1,171.1	4.68	6.9	62.5	32.0	94.5
Silicified	70	2.85	1,561.2	9.08	10.8	62.1	34.4	96.6
Silicified	96	3.00	963.3	8.32	15.1	54.6	42.0	96.6
Silicified	142	2.59	852.6	16.66	8.3	40.8	53.5	94.3
Unaltered	67	0.68	200.9	3.16	9.0	54.2	41.8	96.0
Unaltered	100	0.61	313.4	2.84	7.6	59.7	35.8	95.4
Unaltered	140	0.71	240.2	1.82	13.7	58.5	35.4	93.9

TABLE 16-4 SUMMARY OF PRA'S GRAVITY-CYANIDATION TEST RESULTS
Victoria Gold Corp. – Eagle Gold Project

Ore Type	Grind Size p80, µm	Calc Head g/t Au	Gravity Conc g/t Au	% Au Rec in Gravity Conc.	% Au Rec in Cyani-dation	Overall % Au Recover y
Altered	79	0.94	118.8	35.8	50.4	86.2
Altered	99	1.06	75.2	42.1	42.9	85.0
Altered	146	0.99	104.5	33.6	50.3	83.9
Oxidized	67	0.61	141.5	45.3	46.5	91.8
Oxidized	96	0.48	137.8	23.6	64.0	87.6
Oxidized	144	0.76	464.1	58.5	34.9	93.4
Silicified	70	3.15	672.9	51.4	42.9	94.3
Silicified	96	7.46	2,350.3	76.1	20.7	96.8
Silicified	142	6.21	1,603.9	71.5	25.0	96.5
Unaltered	71	1.38	303.7	41.6	49.0	90.6
Unaltered	104	1.06	391.5	62.3	29.2	91.6
Unaltered	147	0.99	321.7	50.3	39.6	89.9

Cyanide neutralization and solution detoxification tests using hydrogen peroxide were conducted on coarser crushed column leach tailings. Reagent consumptions were relatively low. The cyanide neutralized solutions were further treated to remove additional metals and other constituents. The final detoxified solutions passed fish toxicity testing. Standard toxicity test procedures were followed, based on A.P.H.A. Standard Methods, 18th Edition (1992) and Environment Canada's "Biological Test Method: Acute Lethality Test Using Rainbow Trout" EPS 1/RM/9 (1990).

ASL's work included a statistical data review to determine if there was any correlation between fire assay gold and other element concentrations, such as arsenic or bismuth, which could be used to request duplicate assaying of samples by metallic screen assay procedures.

LEACH DATA REVIEW AND FIELD GOLD RECOVERY ESTIMATES

KCA conducted a leach recovery test program on composites, and on the five ore types. Types A (oxidized), B (unaltered) and C (sericite-altered) account for nearly all of the ore in the reserve pit design, with weight percentages by ore type of 35% A, 55% B and 10% C. Type D (fine-grained, trace sulphides) is rare, and E (sediments) is relatively unmineralized, so only a few select tests were conducted on these two types.

All samples tested were from HQ core. The core tested mostly came from the central and southern sections of the ore body. Previous studies have questioned the representivity of the samples tested, due to potential loss of coarse gold during core drilling. The potential absence of this coarse gold and its effect on recoveries and leach times were not taken into account in this metallurgical review. New metallurgical samples were taken in the summer of 2009, both core and bulk samples. The 2009 core samples were taken with a triple-tube core drill, which should eliminate the problem of potentially washing out coarse gold. Additional testing will be conducted on these new samples and compared to past results.

As shown in Table 16-2, the results from the bottle roll leach tests generally indicate that finer crushing/grinding generally leads to higher gold recoveries. They also indicate that ore type A gives higher average gold recoveries than ore type C, which gives higher average gold recoveries than ore type B. There does not appear to be any correlation between interval depth and gold recoveries in the pulverized bottle roll test results.

Column leach tests were conducted on samples crushed to various sizes by several different types of crusher types. These crusher types tested were:

- Conventional laboratory scale cone crushed material
- Laboratory scale cone crushed, laboratory scale HPGR crushed material
- Laboratory scale jaw crushed, pilot-scale HPGR – single pass – crushed material
- Laboratory scale jaw crushed, pilot-scale HPGR – double pass – crushed material
- Laboratory scale jaw crushed, pilot-scale Barmac crushed material

Average gold recoveries by ore type, crusher type, and crush size, based on the column leach test results conducted by KCA from 1995 to 1997, are summarized in Table 16-5.

TABLE 16-5 AVERAGE COLUMN LEACH TEST RESULTS BY ORE TYPE, CRUSH TYPE AND CRUSH SIZE
Victoria Gold Corp. – Eagle Gold Project

P80 Crush Size mm	Crush Type	Ore Types w Available Test Data	Column Test Recoveries Including Projections, % Au			
			Ore Type A	Ore Type B	Ore Type C	Composite Test Results
33 to 38	1. Conventional	A, B, & C	54%	34%	44%	not available
9.9 to 10.3	1. Conventional	A, B, C & Composite	62%	59%	62%	66%
9.5 to 9.6	3. All HPGR	A & Composite	68%	not available	not available	68%
4.8 to 5.2	1. Conventional	A, B & Composite	74%	65%	not available	72%
4.8	4. All HPGR	A only	80%	not available	not available	not available
1.7 to 2.6	2. All HPGR	A, B & Composite	88%	86%	not available	83%

Several of the column leach tests were still leaching when they were ended, per instructions from the client at that time. The leach recovery curves were reviewed and additional time and gold recovery were estimated for the shortened tests (14 of the 30). An additional one to five percentage points were added to the 14 tests. Additional time to obtain the ultimate column leach recoveries was generally in the 15-day to 50-day time range.

As shown in Table 16-5, there is a distinct increase in gold recovery with decreasing crush size for all ore types. There is also an indication that HPGR crushing results in an increase in gold recovery, compared to conventional crushing to the same P_{80} crush size. The data set is not complete, however, there is an apparent two to six percentage point increase between conventional and HPGR crushing, based on comparison of data at the P_{80} crush sizes of 10 mm and 5 mm. Additional testing is required, but the potential for an increase in gold recovery by the use of HPGR crushers led to the decision to utilize these crushers for the PFS.

The results from the column leach test program, including the projected results, were used to estimate production heap leach recoveries at the Eagle Gold Project. These estimated recoveries are presented in Table 16-6. The calculated overall recoveries are based on an ore type mix of 37% A, 53% B and 10% C (based on gold content). These calculated overall recoveries compare reasonably well with the recoveries from the column leach tests on composite samples.

The recoveries shown in Table 16-6 that are underlined and italicized were calculated based on available data. Differences in available gold recoveries at different crush sizes and/or crush types were compared, then similar differences added to the missing ore type recovery data. Available recovery data were also plotted and interpolations based on these curves were made.

The recovery data in Table 16-6 includes deductions of two to three percentage points to take into account the imperfect field conditions as compared to the more controlled conditions in the laboratory and to account for variations in ore types.

TABLE 16-6 ESTIMATED FIELD RECOVERIES
Victoria Gold Corp. – Eagle Gold Project

			Estimated Field Recoveries, % Au*				
P_{80} Crush Size mm	Crush Type	Ore Types w Available Test Data	Ore Type A	Ore Type B	Ore Type C	Calculated Overall**	Discounted Composite Test Results
33 to 38	1. Conventional	A, B, & C	52%	32%	42%	40%	Not available
9.9 to 10.3	1. Conventional	A, B, C & Composite	60%	57%	60%	58%	63%
9.5 to 9.6	3. HPGR	A & Composite	66%	***60%***	***64%***	63%	65%
4.8 to 5.2	1. Conventional	A, B & Composite	72%	63%	***66%***	67%	69%
4.8	4. HPGR	A only	77%	***68%***	***72%***	72%	Not available
1.7 to 2.6	2. HPGR	A, B & Composite	85%	83%	***83%***	84%	80%

*Includes 2 to 3 percentage point deduction as deemed reasonable based on available leach test results
**Type A, B, and C mix of 37%, 53% & 10%, respectively
Italicized, underlined percentages are calculated based on other tests

Field cyanide consumption was based on the results from the conventional crushed sample at 5 mm, since this is the only test available on composite material at the study crush size. The stated cyanide consumption for this test was 0.91 kg/t. It was not run to completion, however, and additional cyanide would have been consumed if allowed to do so. It was estimated that an additional 0.05 kg/t of cyanide would have been consumed. Field cyanide consumptions are generally 25% to 50% of the laboratory column leach consumptions, depending on ore type and other metal constituents, especially copper. For the purposes of the PFS, KCA used a 35% factor to obtain a field consumption of 0.34 kg/t.

Field lime requirements are generally very close to the laboratory column leach test results. A 1.0 kg/t lime addition rate was used in the PFS.

A series of compacted percolation tests were conducted to determine heap permeability under various pressures. The tests simulated loads at various heap heights up to 100 meters. As shown in Table 16-1, cement instead of lime was added to several of the finer crush size tests for agglomeration purposes. The coarser crush size samples passed the compacted tests at all heap heights tested. However, there were several tests where slump percentage was high, which may lead to permeability issues in the field. Compacted test results on finer crush sizes are only available on an approximate 2 mm crushed sample. These tests all passed.

Due to limited compacted percolation testing results, for the purposes of the PFS, it was decided to have the ability to add cement during the first year or two of operation to ensure that there will not be any permeability issues. Additional tests are planned to better evaluate agglomeration requirements.

Production leach times were based on the 5 mm conventional test. Both tonnes of solution per tonne of ore and leach time data from the column leach test were used to estimate leach times. The tonnes of solution to tonnes of ore (T_s/T_o) ratio was estimated to be 0.67 from the applicable column leach test recovery curve. This ratio was estimated based on the quantity of solution applied to the column test just prior to when the recovery curve begins to flatten out. This ratio was then translated to leach times in the field to obtain a similar ratio (44 days). Additional leach time was then added to this calculated field time based on how much longer the column leach tests ran to obtain the final recoveries after reaching the selected T_s/T_o figure (approximately 80 days). For the purposes of the PFS, a leach time of 120 days was selected.

PRA TEST RESULTS

PRA conducted a testing program on four composites: oxidized, altered, unaltered and silicified. The silicified composite contained 2.47 g/t Au while the other three contained less than 0.7 g/t Au.

Tests utilizing gravity separation followed by panning on samples milled from 63 microns to 147 microns were conducted. These tests gave the following average gold recoveries for each composite:

- Oxidized: 53%
- Altered: 41%
- Unaltered: 54%
- Silicified: 59%

The gravity tailings were treated by cyanidation and flotation. The gravity-cyanidation tests gave gold recoveries greater than 90% for all composites except the altered, which achieved an 85% overall gold recovery. Cyanide consumptions were low and averaged approximately 0.27 kg/t. The grind size did not appear to affect gold extraction in the cyanidation tests. The gravity-flotation tests resulted in gold recoveries averaging 95%. Gold recoveries increased slightly with a finer grind size.

The potential to add a gravity circuit to the heap leach process to recover coarse gold will be evaluated in the near future. No additional flotation or milling testwork is planned at this time, as the capital and operating cost increases of a milling scenario were found to be in excess of the additional revenue from better recovery.

The fine crush size chosen for this study will facilitate gravity operation since a gravity circuit should be fed a fairly fine feed fraction (<3 mm). A mineralogical study was conducted by PMET in 1995, and the results are included in the 1996 KCA laboratory report. The PMET report indicates that gold particles are in the 40 micron to 250 micron size range, which should be recoverable by gravity. PMET conducted a gravity test where over 60% of the gold was recovered into a gravity concentrate at an approximate 300 micron grind size.

ASL STATISTICAL REVIEW

ASL's statistical review of data resulted in a number of conclusions about coarse gold:

- For half the samples, only 3% of the gold reports to the coarse fraction.
- For at least 80% of the samples no more than 10% of the gold is in the coarse fraction.
- Only 5% of the samples have more than 30% of the gold in the coarse fraction.

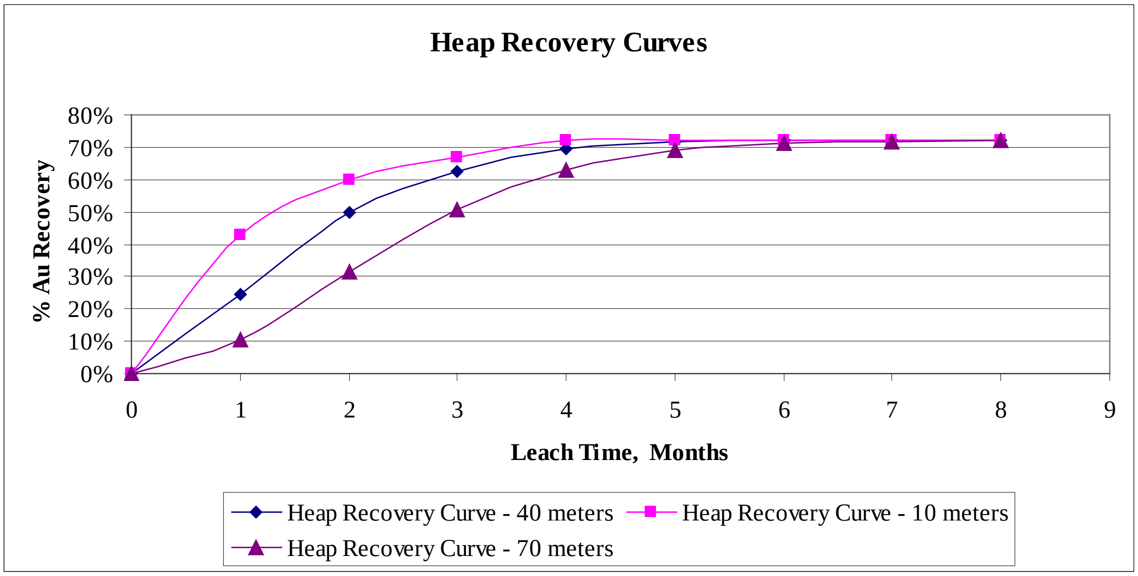
ASL concluded that for most of the samples, even if all of the coarse gold failed to fall within the assay aliquot, the overall assay would be expected to change by no more than 10%. ASL also indicated that "there is an important subset of the samples where the coarse fraction (greater than 100 mesh or 150 microns) contributes significantly to the overall grade of the material and this will likely cause sample representivity concerns."

In KCA's opinion, coarse gold did not appear to significantly impact metallurgical balances in the column leach tests, which form the basis of the metallurgical recovery for the PFS. The presence of coarse gold does have the potential to affect the economics of adding a gravity circuit, however, which is under consideration for future studies.

GOLD RECOVERY CURVE

Field gold recovery curves were generated by lift, based on the column leach test results on the composite sample crushed to 5 mm. The field curves were generated based on a combination of laboratory T_s/T_o and leach time data, a ten-meter lift height, and a 10 L/hr/m^2 leach solution application rate. Gold recovery will be delayed an incremental amount as the total heap height increases. Maximum heap height above the liner is approximately 80 m. For this study, field recovery curves modelling heap heights at 10 m, 40 m, and 70 m were made. These curves are shown in Figure 16-4.

FIGURE 16-4 HEAP RECOVERY-TIME CURVES



CYANIDE NEUTRALIZATION

Cyanide neutralization and detoxification testing was completed on coarser-crushed column leach tailings and barren solutions in 1996 and 1997. No neutralization data are available at a 5 mm crush size. Additional testing will be required.

Seven separate neutralization tests, on six column tests conducted on individual ore types, and on one column test conducted on a composite sample, were conducted using a copper-catalyzed hydrogen peroxide process. These results are presented in Table 16-7. Less than one tonne solution per tonne of ore was required to reach the cyanide levels shown in the table. In this series of tests, Weak Acid Dissociable (WAD) cyanide was utilized as the primary indicator upon which final detoxification was based.

TABLE 16-7 SUMMARY OF HYDROGEN PEROXIDE DETOXIFICATION TEST RESULTS
Victoria Gold Corp. – Eagle Gold Project

Ore Type	Crush Size mm	WAD CN ppm (1)	Total CN ppm (1)	Reagent Usages		
				30% H_2O_2, kg/t	Lime, kg/t	Copper, kg/t (2)
A	-50	<0.10	1.82	0.15	0.05	0.007
A	-12.5	<0.10	0.17	0.28	0.12	0.008
A	-50	0.03	0.08	0.15	0.09	0.009
A	-12.5	0.03	0.05	0.35	0.03	0.05
B	-12.5	<0.10	1.82	0.17	0.00	0.00
C	-12.5	<0.10	0.78	0.42	0.03	0.003
A,B,C,D	-12.5	<0.10	1.3	0.12	0.00	0.00

(1) Level in column effluent.
(2) Copper added as copper sulphate pentahydrate

Fish toxicity testing was also completed on the cyanide-neutralized barren solutions. Various other metals and other constituents had to be removed for the solutions to pass the fish toxicity tests. Additional treatment included cyanate hydrolysis with dilute sulfuric acid, metal removal with iron chloride, pH adjustment with caustic and air stripping to remove ammonia. A total of three different solutions were tested, with the third solution resulting in a 100% fish survival rate after 96 hours. These fish toxicity tests were conducted by EVS Environmental Consultants.

Full details on the testing are presented in the 1996 KCA laboratory report and in a letter report dated 25 April 1997.

MINERAL PROCESSING

The process consists of the following unit operations:

- Three-stage crushing to a P_{80} of 5 mm
- Stacking in 10 m lifts, 250 days per year (March to early November)
- Stockpiling of winter ore on the heap, to be reclaimed and stacked during the following stacking season
- Year-round heap leaching

- Recovery of gold from solution in a carbon adsorption, desorption, and recovery (ADR) plant

The process flowsheet is presented in Figure 16-5.

CRUSHING CIRCUIT

The crushing circuit flowsheet is based on three stages of crushing.

PRIMARY CRUSHING AND CONVEYING

Run-of-mine ore will be delivered by haul trucks from the open pit to the primary gyratory crusher, located north of the pit. The ore will be direct-dumped into the dump hopper situated above the 1,372 mm to 1,651 mm (50 in. to 65 in.) gyratory crusher, and will be discharged to the crusher with a discharge setting of 150 mm. The primary crushed ore will be collected in the discharge pocket below the primary crusher. A belt feeder will regulate the discharge rate of the primary crushed ore, at nominally 1,350 dry tph, onto a 1,524 mm wide primary crushing discharge conveyor.

SECONDARY CRUSHING AND CONVEYING

The primary crushing discharge conveyor will deliver the primary crushed ore to two 50 t surge bins. Belt feeders will regulate the ore feed rate from the surge bins at nominally 675 dry tph each, to two 2,438 mm x 7,315 mm (8 ft. x 24 ft.) double-deck vibrating screens (secondary screens), with apertures of 89 mm and 38 mm. Screen oversize material (nominally 559 dry t/h each) will discharge to two Metso MP1000 standard head cone crushers (secondary crushers), with closed side settings of 19 mm.

The combined secondary crusher discharge product and screen undersize will be transported by a 1,219 mm secondary crushing discharge conveyor at a nominal rate of 1350 dry t/h to an overland conveyor. The 1,067 mm wide overland conveyor will be approximately 530 m long, with a vertical drop of approximately 29 m. The secondary crushed ore will be delivered to a conical stockpile with 10,000 tonnes live capacity.

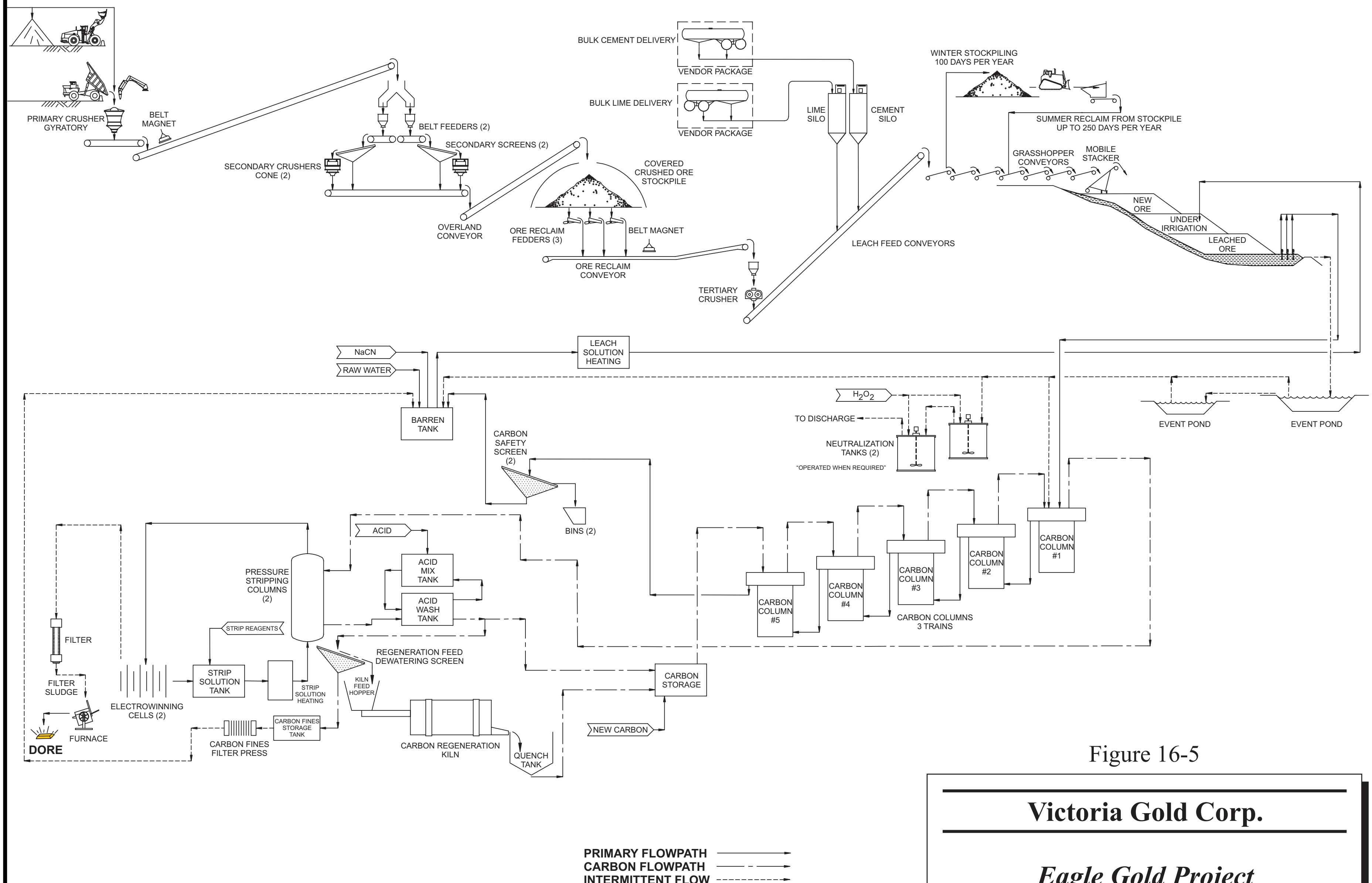


Figure 16-5

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Process Flowsheet

April 2010

Source: Kappes, Cassiday & Associates, 2010.

RECLAIM AND HPGR

Secondary crushed ore will be withdrawn from the stockpile at a controlled rate (nominally 1,550 dry tph) by vibrating pan feeders (2 operating) onto a 1,372 mm wide reclaim conveyor, which will discharge into a 20 t bin, located above the High Pressure Grinding Roll (HPGR). The feed rate to the bin will be controlled so that the HPGR is choke-fed. Product with a nominal P_{80} of 5 mm will be discharged onto a 1,219 mm wide HPGR discharge conveyor, and then conveyed to the leach pad at a nominal rate of 1,550 dry t/h.

HEAP STACKING

The heap will be constructed during non-winter months using a 1,219 mm wide conveyor stacking system. Crushed ore will be stockpiled on the heap leach pad throughout the winter, while the insulating layer of ore is covering the heap leach. The on-heap crushed ore stockpile will be fed to the conveyor stacking system by a bulldozer (via mobile feeder), when stacking operations resume in warmer weather.

The nominal stacking rate will be 2,170 dry t/h, that will consist of 620 dry tph reclaimed from the crushed ore stockpile and 1,550 dry t/h from the crushing circuit. Raw water will also be added to the HPGR discharge material to assist with the agglomeration of fines in the ore. The heap stacking system includes:

- Approximately 915 m of overland conveyors.
- A tripper conveyor to divert ore to the leach pad area (not required until 2015).
- Up to 20 "Grasshopper" portable transfer conveyors, each approximately 38 m in length.
- A 43 m long horizontal mobile bridge conveyor mounted on a dozer crawler carriage.
- A 37 m long radial stacking conveyor, capable of powered luffing, slewing and stacking to a height of 10 m and fitted with a 6.1 m stinger.

The heap feed conveyors will be installed adjacent to the leach pad, running north and east from the HPGR area. The grasshopper conveyors will transport the ore from the overland conveyors to the bridge conveyor. The ore will be placed in 10 m lifts using the radial stacker. The heap will be constructed from west to east, retreating up the slope of the pad. As the stacker retreats, grasshopper conveyors will be removed from the transfer train and relocated in an adjacent cell, so that the heap will be constructed from the toe upwards in a series of approximately 60 m wide x 500 m long ribbons.

The in-heap storage area will be the first area filled with ore. The maximum depth of ore in this area will be about 26 m. In subsequent operations, ore will be stacked on top of this heap base in 10 m lifts. Ramps will be established to allow conveyor access to the top of the heap for construction of further lifts.

For areas of the heap that will be under leach in the winter, the ore will be placed initially in seven-metre lifts. After installation of the drip emitter system, an additional three metres of ore for the insulation cover will be placed in stockpiles on the heap, and spread by low ground pressure dozers to facilitate rapid placement.

HEAP LEACHING

The ore heaps will be leached using a dilute solution of NaCN, applied by a system of drip emitters. Sprinklers can also be utilized in the late spring and summer months when increased evaporation may be required to maintain the system water balance.

The dilute cyanide leach solution will percolate through the ore and collect on the geomembrane liner at the base of the heap. Drainage pipes above the liner will deliver the solution to the in-heap pregnant solution storage area. Vertical turbine pumps located in sumps in the heap will pump pregnant solution directly to the ADR plant. All exposed pipelines will be insulated and heat traced.

In order to minimize the risk of freezing the drip irrigator lines and to maintain a “heat sink” within the heap, a low-pressure steam boiler and heat exchanger will be used, when required, to warm the barren solution. A bleed stream from the barren solution tank will be pumped through the heat exchanger and back to the barren solution tank. The boiler is sized for a maximum barren solution temperature rise of approximately 2.8°C (5°F).

RECOVERY PLANT

The recovery plant for the Eagle Gold Project is a carbon adsorption, desorption, and recovery (ADR) facility located west of the heap and north of the events ponds.

The following major components will be included in the ADR facility:

- Three trains of five carbon adsorption columns, with nominal flow of 650 m^3/h each and design flow capacity of 715 m^3/h each.
- A pressure strip system, consisting of two 3.0 t elution columns, heat exchangers, solution heater, solution storage tanks, and electrolytic cells. The system is capable of processing a maximum of 6.0 t of carbon per 24 hour day.
- A 3.0 t capacity acid wash circuit, consisting of a single acid wash vessel, acid mix tank, and circulation pump.
- A tilting crucible-type diesel-fired doré furnace and baghouse.
- A carbon regeneration system, including a 2.4 tpd capacity horizontal rotary kiln.
- A carbon handling circuit, consisting of transfer pumps, vibrating screens, storage tanks, and a carbon fines filter.
- A mixing system, consisting of an agitated mix tank and storage tank for make-up and addition of NaCN to the process.

ADSORPTION

The adsorption section of the ADR facility will consist of three parallel trains of five up-flow, open-top, carbon steel columns. The adsorption section will contain the following major components:

- Fifteen carbon columns, each column will be 4.0 m diameter and 3.0 m high and will contain 6 t of carbon.
- Three carbon safety vibrating screens, 1219 mm wide x 3048 mm long, 65 mesh stainless steel wire cloth screens.

Pregnant solution will be pumped to the adsorption columns at a nominal flow rate of 650 m^3/h per train. Magnetic flowmeters, with totalizers and wire samplers for continuous sampling of the pregnant solution will be installed.

Pregnant solution will continue to flow through the columns until the carbon contained in the lead column achieves the desired precious metal loading of approximately 4,000 g Au/t carbon. The carbon will then be pumped to either of two elution columns or to the acid wash vessel. Carbon will then be sequentially moved up the adsorption train counter-currently to the solution flow from column 5 to column 1. Stripped and regenerated carbon will then be pumped into column 5. The maximum time off-line per column for carbon transfer operations will be 30 minutes to 1 hour per day. Carbon transfer will be achieved using recessed impeller pumps.

Barren solutions from the last carbon columns will be continuously sampled by wire samplers for metallurgical accounting then discharged to the carbon safety screens to recover floating fugitive carbon. The discharge from the screens will flow by gravity to transfer tanks, from which it will be pumped to the barren solution tank. Any fugitive carbon will be collected and recovered into tote bins.

DESORPTION AND RECOVERY

The gold will be stripped (desorbed) from the loaded carbon and deposited onto stainless steel wool cathodes using a modified Zadra pressure strip procedure.

The desorption and recovery section of the plant will contain the following major components:

- Two elution columns, insulated carbon steel pressure vessels, each with a capacity of 3.0 t of carbon, 1.2 m diameter x 7.0 m high
- Two strip solution storage tanks, 18 m^3 capacity
- Two strip solution pumping systems rated at 15 m^3/h
- Plate and frame type heat exchangers
- A 12,000,000 Btu/h diesel-fired solution heater
- Four 2.12 m^3 (75 $ft.^3$) capacity electrowinning cells, each powered by a 0 - 1000 amp, 0 - 9 volt DC rectifier

A complete strip (desorption) cycle, including carbon transfers and strip solution preparation, will take 14 hours to 18 hours. After a batch of loaded carbon is transferred to an elution column from the adsorption circuit, caustic-cyanide strip solution will be pumped through the heat recovery heat exchangers and solution heater, then introduced to the elution column at a temperature of 135°C, and a pressure of approximately 450 kPa. As the strip solution rises through the bed of loaded carbon in the strip vessel, the precious metals will be desorbed from the carbon. The gold-laden strip solution will exit the column, flow through the cooling side of the heat recovery heat exchanger (to pre-heat the incoming solution), then through a cooling heat exchanger, where raw water is used to further cool the strip solution. Cooled to approximately 85°C, the solution will flow through an electrowinning cell, where the gold will be deposited onto stainless steel wool cathodes.

The barren strip solution will be pumped from the electrowinning cell discharge tank to the strip solution storage tank. Solution will be continually recycled until stripping of the carbon is completed.

Two or three times a week, the gold precipitates in the electrowinning cells will be washed in-situ using high-pressure water sprays, then filtered in a filter press. The filter cake will be dried and smelted periodically.

Periodically, a portion of the strip solution will be discarded to the barren solution tank and fresh solution will be made-up in the strip solution tank. Sodium hydroxide (caustic) and NaCN solution will be added as required.

Two 3.0 t capacity elution column circuits are incorporated to enable the desorption plant to process a maximum of approximately 6.0 t of carbon per 24 hour period. This maximum capacity will be required to handle peak gold production.

After stripping, the carbon will be transferred via a recessed impeller pumps to either the acid wash circuit or to the carbon regeneration kiln dewatering screen.

ACID WASH

The acid wash facility includes the following units:

- Acid wash vessel, mild steel, FRP lined, 3.0 t capacity, 1.4 m diameter x 4.9 high
- Acid mix tank, FRP or HDPE, 9.0 m^3 capacity
- Positive-displacement acid resistant metering pump
- Acid wash solution recirculation pump, 15 m^3/h capacity

Stripped carbon will be pumped to the acid wash vessel. Fresh water will be recirculated through the bed of carbon to remove any entrained cyanide. The rinse water will be drained to the strip solution storage tank. Concentrated hydrochloric acid will then be pumped from 200 L drums or one cubic metre totes into the acid mix tank to achieve and maintain a pH ranging from 1.0 to 2.0. The acid wash solution, at nominally 2% HCl by weight, will be circulated up-flow through the acid wash vessel. This process will remove scale and other inorganic contaminants that inhibit gold adsorption onto carbon. All gases generated in the acid wash process will be extracted by vent fan, scrubbed with caustic, and then vented out of the building.

After acid washing is complete, the spent acid wash solution will be pumped to one of the barren solution transfer tanks. The carbon will then be washed with process solution to remove any residual chlorides. The total time required for a 3.0 t batch of carbon to be acid washed is typically 4 hours to 6 hours. Washed carbon will then be pumped to either of two carbon column trains, or to the carbon regeneration circuit.

CARBON REGENERATION

The carbon regeneration portion of the recovery plant will contain the following major items of equipment:

- Vibrating dewatering screen, 914 mm wide x 2,438 mm long, with 24 mesh stainless steel wire cloth screens;
- Carbon reactivation kiln feed hopper, 3.0 t capacity;
- Rotary kiln reactivation furnace, diesel-fired, capacity of 100 kg of carbon (dry) per hour, complete with pre-drier;
- Quench tank, 3.0 t capacity.

Following acid washing (or carbon stripping, depending on operator preference), the carbon will be transferred via a recessed impeller pump to the dewatering screen, where it will be dewatered and discharged to the kiln feed hopper. If the carbon is not scheduled to be reactivated, it will be pumped to one of the "tail" adsorption columns, column No.5.

The carbon to be regenerated will be fed at a controlled rate by a screw feeder into the kiln and then thermally reactivated at approximately 750°C. The hot carbon will discharge into a "quench" tank, which will be partially filled with barren solution. The quench will increase the reactivity of the carbon, thus additionally enhancing the gold adsorption capabilities. Quenched carbon will be pumped to the dewatering screen to remove any fine carbon (less than 24 mesh) generated in the regeneration process.

On average approximately one in three carbon batches will be thermally reactivated. The carbon regeneration circuit will have the capability to process between one half and one third of the carbon stripped to satisfy the peak periods of gold production.

CARBON HANDLING

Carbon handling will include the components necessary to condition, store and add carbon to the system. Fine carbon collection is also incorporated. This section includes the following equipment:

- Agitated carbon attrition tank, 4.9 m^3 capacity
- Recessed impeller pumps, 20 m^3/h capacity
- Carbon dewatering vibrating screen, 914 mm wide x 2,438 mm long, 24 mesh stainless steel wire mesh screens
- Carbon fines tank, 6 m^3 capacity
- Filter press to remove carbon fines from process water

Carbon will be transferred between the various unit operations in the recovery plant by recessed impeller pumps. This type of pump minimizes attrition of carbon encountered during transfer.

The movement and regeneration of carbon will produce fines. Prior to transfer of regenerated carbon to the columns, the carbon will be screened on a vibrating carbon sizing screen. The undersize material will gravitate to the carbon fines tank, and will then be pumped through the carbon fines filter press. The resulting filtered solution will return to the adsorption system. The carbon fines will be placed in plastic-lined containers and eventually sent off-site to an appropriate facility for residual metal recovery from the carbon.

New carbon, supplied in 500 kg bulk bags, will be periodically added to the attritioning tank, and hence to the circuit, to make up for carbon lost through abrasion or during reactivation. Typical carbon losses are 3% of the carbon stripped.

REFINING

This section will include the following major equipment items:

- Tilting crucible-type diesel-fired smelting furnace, 430 kg red brass capacity;
- Furnace off-gas collection system, including a hood, baghouse and induced draft exhaust fan.

Smelting will take place up to three times per week. The dried metal deposits from the cathodes and fluxes will be added to the crucible, and the furnace brought up to temperature. The fluxes will be a combination of borax, fluorspar, soda ash and niter.

Slag will be poured off into slag molds. A jaw crusher will be provided to reduce the slag. The slag will be examined for gold prills and recycled to the next smelt if prills are present. Otherwise, slag will be transported to the heap where it will contribute any contained precious metal values to the leach system.

The doré will be poured into ingot molds of 20 kg size. The doré produced will be sampled, cleaned, weighed and prepared for shipment.

A hood will collect the furnace fumes which will pass through a baghouse to remove particulates, then through an induced draft fan. The system will be designed to remove over 99.5% of the particulates present in the exhaust fumes.

REAGENT ADDITION

The reagent make-up and addition system will include the following major components:

- Agitated NaCN mix tank, mild steel, 15 m^3 capacity;
- Cyanide storage tank, mild steel, 22.5 m^3 capacity, and cyanide metering pumps;
- Caustic mix tank, mild steel, 3.2 m^3 capacity, and caustic metering pump.
- Acid metering pump to deliver concentrated acid to the mix tank;
- Acid mix tank, FRP or HDPE, 9.0 m^3 capacity, and acid circulation pump;
- Anti-scalant dosing pumps.

NaCN will be delivered to the site in sealed bulk bags (Supersacks, contained within plywood boxes) containing approximately 1,000 kg of NaCN in solid briquette form. Supersacks of NaCN will be unloaded into and dissolved in the agitated cyanide mix tank with barren solution, to make a 20% cyanide solution. Completed batches of cyanide solution will be transferred to the cyanide storage tank. Controlled quantities of cyanide solution will be added to the barren solution tank via metering pumps, to control the cyanide concentration in leach solutions to approximately 180 ppm to 250 ppm.

Cyanide solution will also be metered to the strip solution tank prior to a strip cycle to make-up the 0.1% cyanide strip solution. NaCN usage is estimated to be up to 8.8 tonnes per day.

Sodium hydroxide (caustic) solution will be pumped into the strip solution tank as needed. The concentration of sodium hydroxide typically contained in the strip solution will be 1% to 2%. Caustic consumption is estimated at 66 kg/day.

Anti-scalant solution will be delivered to site in 200 liter plastic barrels or 1 m^3 carboys, from which it will be metered into the barren solution pumps, pregnant solution pumps and strip solution pump via metering pumps.

RECOVERY PLANT BUILDING AND SITE

The process building will be constructed on a 93.1 m x 41.5 m concrete foundation and will contain

- All the ADR process facilities described above
- The main electrical distribution and MCC panels for the process equipment
- A secure refinery area
- A control room for the process operations
- Reagents receiving and storage area
- Boiler room containing the heating solution boiler

A 300 mm high curb wall will contain spills or fugitive solutions within the building. Except for the barren solution tank, all solution tanks and associated pumps will also be located in this building. The heated barren solution tank will be located just outside of the building, with pumps inside the building. Overflows and spills will report to a floor sump, which will gravity-drain to the events ponds through a 750 mm diameter HDPE pipe. The building will be of steel frame construction with insulated metal cladding.

CHEMICALS STORAGE AND HANDLING

Except for lime and cement, chemical storage will be indoors on concrete slabs located adjacent to the adsorption area in the ADR Building. NaCN, caustic, hydrochloric acid, and smaller quantities of other miscellaneous chemicals will be supplied and stored as tabulated below. Lime and cement will be delivered in bulk pneumatic trucks and stored in large silos adjacent to the reclaim conveyor. Table 16-8 lists the monthly usages and storage requirements for each chemical. The individual storage areas for major reagents are sized to contain a minimum of two weeks storage. Flux storage is sized for shrink-wrapped one-tonne pallet shipments.

Concrete curbing will separate each of the chemical storage areas to prevent any interaction between the chemicals and provide a minimum of 110% containment in case of spills.

TABLE 16-8 CHEMICAL PACKAGING AND STORAGE
Victoria Gold Corp. – Eagle Gold Project

Item	Packaging	Daily Consumption	Recommended Minimum Storage	Qty. Packages
NaCN	1000 kg Bulk Bags Packed in Plywood Boxes	8,784 kg	2 Weeks	122
Sodium Hydroxide	25 kg Bags	66 kg	2 Weeks	1 Pallet (49 Bags)
Hydrochloric Acid (32%)	200 L Drums, or 1 m^3 Totes	597 L	2 Weeks	42 drums or 9 Totes
Antiscalant, Leach	1 m^3 Totes	244 L	2 Weeks	4 Totes
Antiscalant, Strip	200 L Drums	1.20 L	2 Weeks	1 Drum
Lime (CaO)	Up to 24 t in Pneumatic Trucks	26 t	60 t	2+ Trucks
Cement	20 to 30 t in Pneumatic Trucks	26 t	60 t	2 Trucks
Fluxes	50 kg Bags	NA	1 Pallet	20 Bags
Hydrogen Peroxide (50%)	19 m^3 (5000 gal) Tanker Truck	NA	1 Truckload	1
Copper Sulphate (Pentahydrate)	50 kg Bags	NA	1 Pallet	20

CYANIDE DETOXIFICATION

The heap leach water balance indicates that there will not need to be any significant release of treated process water during the production period for average, dry, and wet meteorological conditions. Detoxification will be required after leaching operations have been completed. KCA recommends that a cyanide detoxification circuit is installed to accommodate any unforeseen event that would require release of solution into the environment.

Solution from the events pond(s) will be pumped at a rate of up to 160 m^3/h to the detoxification area.

Cyanide destruction by the controlled addition of hydrogen peroxide will take place in a series of two 75 m^3 agitated tanks. Copper sulfate solution will be also added if required as a catalyst.

Effluent from the detox tanks will flow to the two 15,000 m^3 double-lined polishing ponds to allow "aging" of the solution prior to pumping to the sedimentation ponds, and then to Dublin Gulch. The cyanide detoxification plant will be instrumented and controlled automatically. Specific control and measurement points will include:

- Flow measurement and control of event pond solution.
- Hydrogen peroxide addition.
- Copper sulfate addition; if required.

Additional manual analyzers that will be required for process control include total and WAD cyanide in influent and effluent streams.

Additional treatment, including pH adjustment, and additional metal removal with sulfuric acid and air stripping of ammonia, may be required before discharging solution to the environment.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

MINERAL RESOURCES

SUMMARY

Mineral Resource estimates for the Eagle Project were prepared by MRDI in 1997, and again by Wardrop in 2006 and 2009. Scott Wilson RPA audited the 2009 Wardrop estimate in support of this study and, following the 2009 drilling program, updated the estimate. The current resource model is summarized in Table 17-1 at a range of cut-off grades. Scott Wilson RPA has selected 0.21 g/t Au as the most appropriate for reporting of Mineral Resources.

TABLE 17-1 INDICATED MINERAL RESOURCE ESTIMATE
Victoria Gold Corp. – Eagle Gold Project

Cut-Off (g/t Au)	Tonnage Mt	Grade (g/t Au)	Contained Gold (M oz Au)
0.50	89.2	0.86	2.45
0.30	138.0	0.69	3.08
0.25	147.7	0.67	3.16
0.24	149.3	0.66	3.18
0.23	151.0	0.66	3.19
0.22	152.2	0.65	3.20
0.21	**153.4**	**0.65**	**3.21**
0.20	154.3	0.65	3.21
0.19	155.2	0.65	3.22
0.18	155.9	0.64	3.22
0.15	157.4	0.64	3.23
0.10	158.0	0.64	3.23

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at an average pit discard cut-off grade of 0.21 g/t Au. This cut-off does not include mining costs, and is only valid within an optimized pit shell.
3. Mineral Resources are estimated using an average long-term gold price of US$1050 per ounce, and a US$/C$ exchange rate of 0.90:1.00
4. Mineral Resources were constrained within an optimized pit shell.
5. A minimum mining width of three metres was used.
6. Indicated Mineral Resources are inclusive of Mineral Reserves.

The estimate was prepared using a block model constrained by 3D wireframes. Grade was interpolated into the blocks using Ordinary Kriging (OK). Samples were capped at 12.65 g/t Au prior to compositing. A pit shell was used to constrain the estimate, and demonstrate a potential for economic viability. Significant additional quantities of mineralization are defined at depth in the block model, below the pit shell constraint. All Mineral Resources reported in Table 17-1 are classified as Indicated. The model was constructed using GEMS (Gemcom) software, which is a commercial off-the-shelf package, commonly used in the industry.

MODEL GEOMETRY

The block model was aligned with the property grid and comprised blocks measuring 15 m x 15 m x 15 m. Model origin and extents are shown in Table 17-2. Note that the block model origin coordinates are quoted using the GEMS convention, which is the uppermost southwest corner (i.e., not the centroid) of the uppermost southwest block. Columns increment towards grid east, rows towards grid north, and levels downwards.

TABLE 17-2 BLOCK MODEL GEOMETRY
Victoria Gold Corp. – Eagle Gold Project

Item	Units	Value
Origin:	X (m)	458885
	Y (m)	7098800
	Z (m)	1440
Rotation:	º	0
Block Size:	X (m)	15
	Y (m)	15
	Z (m)	15
Columns:	#	130
Rows:	#	100
Levels:	#	60

DATABASE

The database used for grade interpolation comprised samples collected in diamond core and reverse circulation holes. The project database contains records for 479 holes and trenches. Many of these holes were drilled on other deposits in the area, and are not relevant to the Eagle deposit. The holes within the general area of the deposit number in the order of 277 (see Table 6-1), although many of these are quite distant and did not affect the estimate. Six records were actually “pseudo drill holes” used to depict trenches, and these were excluded from the estimate.

For the 2009 estimate, Wardrop had isolated several dozen sampled intervals, and some entire holes, that were deemed unsuitable for use in resource estimation due to issues such as poor recovery or unreliable assays. Scott Wilson RPA also excluded these intervals from the estimation process.

Contained within the records for the “nearby” drill holes were 33,179 sampled intervals, representing an aggregate hole length of 47,745 m. Of these samples, 21,566 were captured within the wireframe model of the zone, and actually used in the grade interpolation. Statistics for these samples are provided in Table 17-3, and histograms and probability plots are attached to this report in Appendix 1.

TABLE 17-3 SAMPLE STATISTICS
Victoria Gold Corp. – Eagle Gold Project

Item	Quantity
Number	21,566
Mean	0.7161
SD	1.451
CV	2.027
Median	0.320
Maximum	52.36
Minimum	0.00

The sample distribution is observed to be non-normal and positively skewed, which is common for Au deposits. This is evident in the histogram and in the Coefficient of Variation (CV), which is 2.027. Grade estimates derived from non-normal distributions are at risk of over-estimation due to the effect on the mean imparted by the higher-grade samples. It is standard industry practice to attempt to ameliorate this risk by applying a top-cut (or cap) to the samples. Wardrop used a cap of 12.65 g/t Au, which was applied to the sample database prior to compositing and grade interpolation. In Scott Wilson

RPA's opinion, this cap value is within a reasonable range, albeit at the upper end of that range. It has been applied to the current estimate.

WIREFRAME MODELS

The 2009 model by Wardrop was constrained by a grade shell wireframe drawn at a 0.2 g/t Au cut-off. Scott Wilson RPA inspected this wireframe and determined that it represented a reasonable interpretation of the mineralized body. The holes drilled in the summer of 2009 intersected extensions to the mineralization, mostly at depth. Figure 17-1 shows a 3D view of the original wireframe model from the Wardrop estimate along with the newer holes.

Other wireframes included in the database are the topography surface, the base of the overburden, and the sediment-intrusive contact. The intrusive body is the principal host rock type, and the mineralization does not typically extend very far into the surrounding sedimentary rocks. Consequently, the contact represents a significant control to the mineralization. Scott Wilson RPA re-constructed the 0.2 g/t Au grade shell to encompass the mineralized intercepts in the 2009 drilling, using the overburden and the intrusive contact as constraints. The wireframe expanded at depth, extending more than 450 m below surface in places.

A significant difference from the 2009 model to the present one is that the upper boundary of the mineralization is now constrained by a wireframe representing the base of the overburden. This wireframe was constructed using the bottom of the drill casings.

The modified grade shell is shown in Figure 17-2.

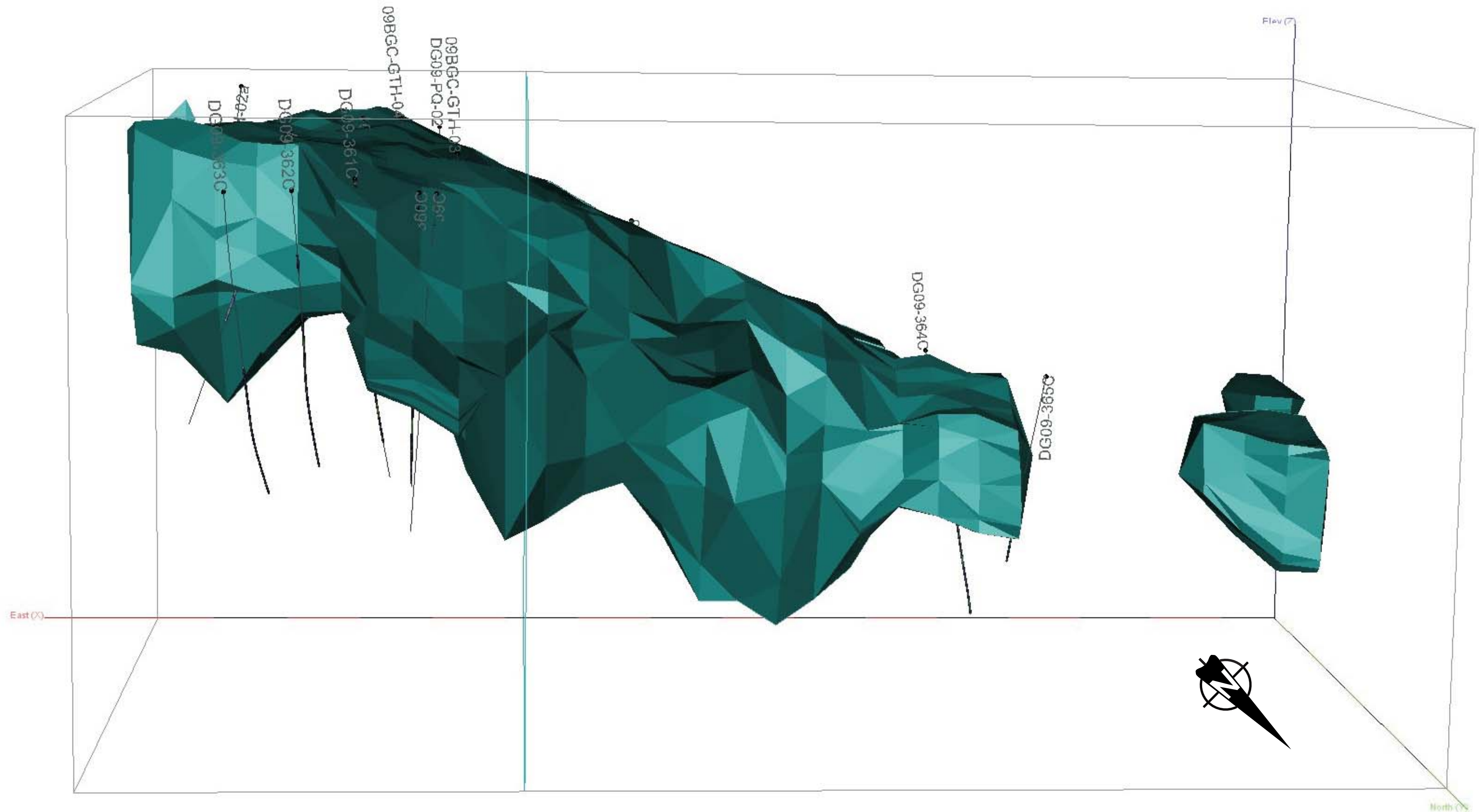


Figure 17-1

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

2009 3D Wireframe

(Looking South)

April 2010

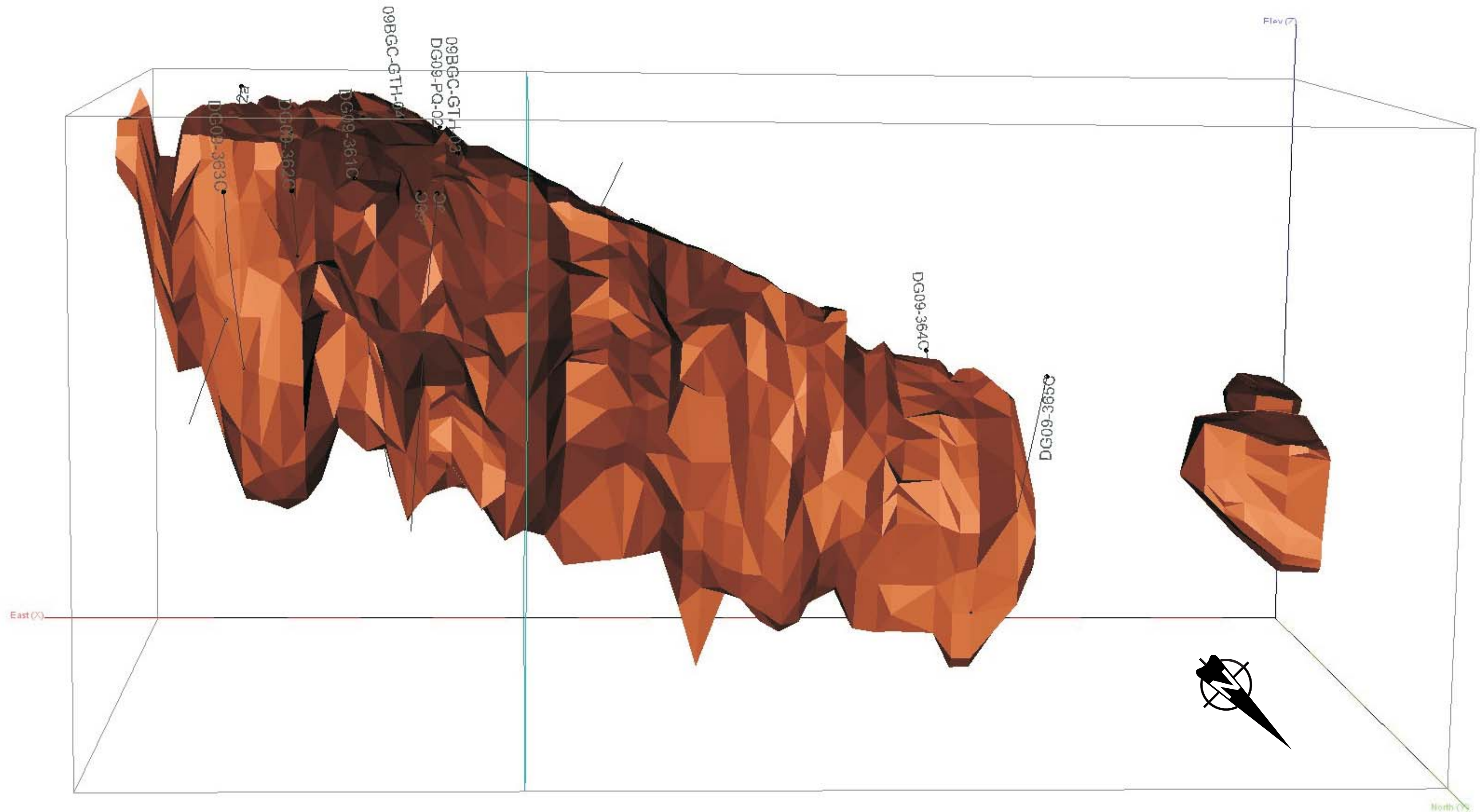


Figure 17-2

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

2010 3D Wireframe

(Looking South)

April 2010

COMPOSITES

Compositing of the samples is usually necessary to remove the possibility of bias due to correlation between sample length and grade. Ideally, the composites should be as short as possible, but not less than the original sample lengths. This is to ensure against “creating data” by splitting samples, which artificially reduces the population variance. Inspection of the probability plot indicates that the majority of the samples are 1.52 m in length, and all of the samples are less than 3.05 m (see Appendix 2). In Scott Wilson RPA’s opinion, a reasonable composite length is 3.05 m.

As stated above, the assays were capped at 12.65 g/t Au and then composited to nominal 3.05 m intervals. The compositing was configured to honour the wireframe boundary, beginning at the point of entry of the drill hole and progressing at regular intervals to the exit point. In order to eliminate small remnant composites at the exit point, composites for each individual hole were automatically adjusted to be of equal length. This creates slight differences in composite length between drill holes, but in Scott Wilson RPA’s opinion, the differences were small and did not introduce any bias in grade interpolation.

Composites with centroids contained within the grade shell were coded as “inside the zone”, and used in the grade interpolation. All other composites were ignored.

Statistics for the composited sample data are provided in Table 17-4. Histogram and probability plots for the composites are attached to this report in Appendix 2.

TABLE 17-4 COMPOSITE STATISTICS
Victoria Gold Corp. – Eagle Gold Project

Item	Quantity
Number	10,928
Mean	0.654
SD	0.889
CV	1.358
Median	0.372
Maximum	9.850
Minimum	0.000

GEOSTATISTICS

Scott Wilson RPA carried out a geostatistical analysis on the composited data to develop a kriging model and to help derive search parameters for the grade interpolation. The analysis was conducted using Sage software. In Scott Wilson RPA's opinion, the variography was quite coherent and relatively easy to interpret. The model comprised a nugget effect of 35.7% of the sill, and two exponential structures. Axis orientations agree reasonably well with observed geological and mineralogical trends. The variogram model is described in Table 17-5.

TABLE 17-5 VARIOGRAM MODEL
Victoria Gold Corp. – Eagle Gold Project

	Sill Components	Axes	Range (m)	Orientation (Az/Dip)
Nugget (C0)	0.257			
C1	0.473	Major	10.4	062/-29
		Semi-major	9.6	161/-15
		Minor	8.4	094/57
C2	0.270	Major	251.2	074/34
		Semi-major	118.7	218/51
		Minor	57.8	331/18

SEARCH PARAMETERS

The grade interpolation was configured to employ two passes. The first pass was conducted using an octant search measuring 2/3 the range of the 2nd structure of the variogram model (167.5 m x 79.1 m x 38.5 m). Block grade interpolations were constrained to a minimum of six composites and a maximum of 40, with a limit of five composites per octant and three composites from any one drill hole. A minimum of three octants with data were required in order to estimate a block.

The second pass was conducted at the full variogram range, using an ellipsoidal search, with the minimum composite limit lowered to one.

BULK DENSITY

A bulk density of 2.66 t/m^3 was used for all blocks in the model. The density was derived from the mean of 111 measurements made from drill core in 1995.

MRDI, in its 1997 model, applied a range of densities based on oxidation intensity. The density varied from a mean of 2.67 t/m^3 in unweathered rock to 2.27 t/m^3 in the most intensely oxidized zones. However, Wardrop (2009) was unable to confirm this relationship and opted to use the overall mean of the entire sample set (i.e., 2.66 t/m^3). In addition, Wardrop determined that there was no significant difference in density between the intrusive and sedimentary host rocks.

Scott Wilson RPA reviewed Wardrop's analysis and concurs that there does not appear to be a clear relationship between oxidation and bulk density. However, inspection of the oxidation logging suggests that there are significant inconsistencies with this data set to the extent that any trends that may exist could be obscured. In Scott Wilson RPA's opinion, it seems quite reasonable that there should be a reduction in density in oxidized zones, as this is a common trait in other deposits. A density range of the magnitude of that suggested by MRDI could have a significant negative effect on the Mineral Resource tonnage estimates. Scott Wilson RPA recommends the oxidation intensity be relogged, so that a consistent basis for comparison with the density measurements can be established. It is also recommended that additional density measurements be conducted, as the present number of 111 measurements is somewhat low.

VALIDATION

The block model results were validated using the following techniques:

- Visual inspection and comparison to the drill hole grades in section and plan views.
- Comparison of global composite and block mean grades.
- Comparison with an alternative method.

Following interpolation, the block grades were reviewed in cross-section, long-section and level plan views. In Scott Wilson RPA's opinion, the block grades were observed to agree reasonably well with the drill composites, and there are no obvious mismatches.

The global block mean for the first pass (i.e., the best-informed blocks) was 0.624 g/t Au, which compares reasonably well with the global composite mean of 0.693 g/t Au. The difference is 10%, which in Scott Wilson RPA's opinion, is within an acceptable margin.

Scott Wilson RPA estimated the gold grades using Inverse Distance Cubed (ID^3) weighting. A comparison of the two model grades at a range of cut-offs is provided in Table 17-6.

TABLE 17-6 OK VS. ID^3 COMPARISON
Victoria Gold Corp. – Eagle Gold Project

Cut-Off	OK Model			ID^3 Model			Percent Difference		
(g/t Au)	Kt	Au (g/t)	Au (ozs)	Kt	Au (g/t)	Au (ozs)	Tonnes	Au (g/t)	Au (ozs)
2.50	72	2.88	6,655	1,111	3.06	109,312	1446.9%	6.2%	1542.0%
1.50	4,504	1.76	255,566	8,828	1.98	562,477	96.0%	12.3%	120.1%
1.00	23,791	1.30	992,525	27,134	1.45	1,266,256	14.1%	11.9%	27.6%
0.90	33,294	1.20	1,282,460	35,031	1.34	1,506,711	5.2%	11.7%	17.5%
0.80	45,666	1.10	1,619,175	45,849	1.22	1,801,469	0.4%	10.8%	11.3%
0.70	62,660	1.01	2,027,135	59,764	1.11	2,135,194	-4.6%	10.4%	5.3%
0.60	84,837	0.91	2,488,530	78,699	1.00	2,528,686	-7.2%	9.5%	1.6%
0.50	114,345	0.82	3,008,632	103,429	0.89	2,964,618	-9.5%	8.9%	-1.5%
0.40	149,750	0.73	3,518,832	135,102	0.79	3,420,515	-9.8%	7.7%	-2.8%
0.30	189,325	0.65	3,964,216	172,156	0.69	3,835,999	-9.1%	6.4%	-3.2%
0.20	218,536	0.60	4,205,379	206,485	0.62	4,114,746	-5.5%	3.6%	-2.2%
0.10	229,342	0.58	4,261,450	226,578	0.58	4,216,599	-1.2%	0.2%	-1.1%
0.00	229,879	0.58	4,262,864	229,879	0.57	4,223,640	0.0%	-0.9%	-0.9%

At zero cut-off, the two models yield virtually identical results. The ID^3 estimate tended to yield fewer tonnes at a higher grade for most cut-offs up to 0.70 g/t Au. The difference in tonnes was offset by the increased grade, leaving the total metal content relatively constant. This suggests that the grade distribution was smoothed more in the OK model. Above the 0.70 g/t Au cut-off, the two models are observed to diverge markedly on a percentage basis, although the actual numbers are not really significant.

In Scott Wilson RPA's opinion, the ID^3 and OK models agree reasonably well on a global basis, particularly in the range of cut-off grades that are relevant to this project (0.10 to 0.50 g/t Au).

MATERIAL TYPE MODEL

The drill log data for oxidation and sericite alteration were used to apply a material type classification to the model. A database was compiled, which contained semi-quantitative estimates of intensity for a range of alteration suites. This database encompassed only a portion of the total number of drill holes. The most critical of these alteration types from the standpoint of recoveries are oxidation, sericitization, and silicification. On inspection of the data it was determined that the silicification data was very inconsistent, and appeared to differ significantly from hole to hole depending on the logger and on the drilling campaign. Consequently, it was not possible to apply silicification to the model, and only oxidation and sericitization ratings were used.

The alteration intensity comprised visually-estimated values from zero to five, with zero being unaltered and five being intensively altered. Statistical and geostatistical analyses of these values were carried out to determine the distribution of values for each alteration facies, and the overall characteristics of the logged values. This analysis was conducted in consultation with Victoria Gold geological personnel. It was determined that the oxidation appeared to reside closer to surface, in a distinct layer more or less parallel to the topographic surface (as expected). A wireframe model was constructed, which constrained most of the near-surface, more intensively oxidized material.

Oxide intensity was interpolated into the blocks across two domains: inside and outside of the wireframe for the oxidized zone. The wireframe tended to constrain the higher intensity values to a discrete zone and prevent dilution from the lower intensity intervals at depth.

Sericite appears to be somewhat more diffuse within the deposit and consequently was estimated without a constraining wireframe.

Blocks were assigned a category (and estimated metallurgical recovery) depending on the estimated level of oxidation or sericitization. The resulting material types (and recovery values for a 5 mm crush size) were as follows:

0 – Unaltered (corresponds to Type B in testwork, metallurgical recovery of 68%)
1 – Oxidized (corresponds to Type A in testwork, metallurgical recovery of 77%)
2 – Sericitized (corresponds to Type C in testwork, metallurgical recovery of 72%)

A value of 2.5 or higher (on the 0 to 5 scale) was used as the threshold for defining whether a block was sericitized or oxidized. Priority was given to oxidation such that if a block measured 2.5 or greater in both alteration styles, it was assigned as oxidized. It is noted that, while the oxidation tends to be highest near surface, the sericite tends to be more prevalent at depth, so incidences of dual assignment were relatively rare.

In Scott Wilson RPA's opinion, the quality of the alteration estimates is very poor, owing to the lack of data and the apparently subjective nature of the alteration intensity estimates in the logs. Compared to the rigor generally applied to resource estimate classification, the confidence level for the block estimates of alteration intensity is not high enough even for an Inferred categorization. The alteration model does, however, allow for differential assignment of metallurgical recoveries by material type. In Scott Wilson RPA's opinion, this allows a more accurate assessment of the economics of the orebody, in comparison with assigning a single global recovery value across the deposit.

Scott Wilson RPA recommends relogging holes (or perhaps a select number of holes) for alteration with a focus on establishing a consistent basis for estimation of alteration intensity. Ideally, the logging should be carried out by a single individual so that subjective judgments are consistent from hole to hole. If this is not practical, then care must be taken to ensure that all participants are as consistent as possible in their evaluations of alteration intensity. In addition, the logging should be carried out in consultation with the metallurgical engineers in order to confirm that the alteration logging is, in fact, relevant to the various ore types. As mentioned in the section of this report on bulk density, an improved alteration database should also allow for a more accurate comparison between oxidation and density to determine if there is a relationship.

CLASSIFICATION

The classification nomenclature and definitions used are those in the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines, and as such, are consistent with NI43-101.

All blocks within the wireframe model were initially assigned a minimum classification of Inferred. All blocks captured in the 1st interpolation pass (i.e., 2/3 the variogram range) were assigned to the Indicated category. The blocks estimated in the second pass (i.e., full variogram range) by at least two drill holes were also assigned to Indicated. It was noted that virtually all of the blocks contained with the resource wireframe achieved Indicated status, so the decision was taken to convert the entire Mineral Resource estimate to Indicated.

DEPTH CONSTRAINT

The CIM Standards definition of Mineral Resources includes the phrase “reasonable prospect of economic extraction,” which, in Scott Wilson RPA’s opinion, indicates that consideration for depth or strip ratio must be included when estimating open pit resources.

A pit optimization was carried out to determine how much of the block model may be amenable to open pit mining. The parameters used for this optimization were as follows:

- Gold price of US$1,050 per oz.
- Exchange rate of C$1.00 = US$0.90
- NSR Royalty of 1%.
- Unit costs factored from PFS estimates, assuming higher production rates (10-12 Mtpa):
    - Base Mining US$1.40 per t
    - Haulage incremental additions for pit depth / haul distance per PFS
    - Processing US$3.60 per t
    - G&A US$1.40 per t
- Pit slopes as recommended by BGC for use in the PFS:
    - Overburden 27º
    - Sediments 35.5º to 44º
    - Intrusives 33º to 43.5º

- Mining dilution of 2% at 0.2 g/t Au, and recovery of 98%.
- Metallurgical recovery by material type:
  - Type A Oxidized 77%
  - Type B Unaltered 68%
  - Type C Sericitized 72%

The Mineral Resource estimate includes only those blocks contained within the resulting pit shell. Scott Wilson RPA notes that gold prices in excess of US$2,000 per oz are required before the depth constraining pit reaches the bottom of the wireframe at any point.

CUT-OFF GRADE

Using the pit optimization inputs above, the pit discard (or incremental) cut-off grade for material that must be removed from the pit is 0.21 g/t Au. This is the gold grade that would cover processing and administration costs only, at a metal price of US$1050/oz. Scott Wilson RPA notes that pit discard cut-off grades are not valid outside of the associated pit shell, and therefore that Mineral Resources, quoted at a 0.21 g/t Au cut-off grade, include only material that is contained within the resource shell described above.

DIFFERENCE FROM PREVIOUS ESTIMATE

WIREFRAME CONTENTS

Table 17-7 illustrates the changes in the wireframe contents, as a result of the 2009 drill program. Wireframe contents are analogous to a “mineral inventory” (although that is not a term defined by CIM), or “known extent of mineralization”. Although Scott Wilson RPA does not consider that the entire wireframe contents qualify as a Mineral Resource, the 0.5 g/t Au cut-off grade results provide the most direct comparison to the previous estimate. The 0.2 g/t Au cut-off provides a point of comparison to current Mineral Resources, i.e., how much of the total wireframe, or known extent of mineralization, was captured within the pit shell depth constraint.

TABLE 17-7 WIREFRAME COMPARISON
Victoria Gold Corp. – Eagle Gold Project

Estimate	Cut-Off Grade (g/t Au)	Tonnes ('000s)	Grade (g/t Au)	Ounces ('000s)
Wardrop, 2009	0.5	100,607	0.845	2,735
Scott Wilson RPA, 2010	0.5	119,577	0.832	3,199
Change		+18,970	-0.013	+464
Wardrop, Feb. 2009	0.2	198,299	0.603	3,847
Scott Wilson RPA, 2010	0.2	217,228	0.620	4,331
Change		+18,989	+0.017	+484

Table 17-8 compares the present Mineral Resource estimate with the one reported by Wardrop in 2009.

TABLE 17-8 RESOURCE ESTIMATE COMPARISON
Victoria Gold Corp. – Eagle Gold Project

Wardrop 2009	Cut-Off Grade (g/t Au)	Tonnes ('000s)	Grade (g/t Au)	Ounces ('000s)
Indicated	0.5	98,600	0.85	2,690
Inferred	0.5	2,000	0.67	40
Scott Wilson RPA 2010	**Cut-Off Grade (g/t Au)**	**Tonnes ('000s)**	**Grade (g/t Au)**	**Ounces ('000s)**
Indicated	0.21	153,400	0.65	3,210
Inferred	n/a	n/a	n/a	n/a
Difference				
Indicated		+55.2%	-23.5%	+19.3%

The estimate has increased significantly in tonnage with a proportionately lesser decrease in grade, resulting in a net increase in contained ounces. All Inferred Mineral Resources have now been reclassified as Indicated. The principal reasons for the differences are as follows:

- Additional diamond drilling.
- Adjustments to the resource wireframe.
- Application of a pit shell constraint.
- Reduction of the cut-off grade.

- Modifications to the classification scheme.

In Scott Wilson RPA's opinion, the pit shell constraint and the reduction in cut-off grade are the most important influences in the changes to the estimate. Application of the pit shell decreased the tonnage fairly substantially, but had a relatively small effect on the grade. Lowering the cut-off grade resulted in a large increase in tonnage, but a reduction in overall grade. The new drilling, while successful in discovering new mineralization, did not affect the Mineral Resource estimate significantly, because the additional tonnage was largely at depth, outside of the pit shell. Similarly, the most significant changes to the wireframe model was also outside of the pit shell. The new drilling may, however, have been influential in conversion of Inferred Mineral Resources to Indicated.

MINERAL RESERVES

Mineral Reserves, as per the mine production schedule, are reported in Table 17-9. Mineral Reserves are based on Indicated Mineral Resources only, as there are no Measured Resources in the model, and Inferred Resources are too geologically speculative to be used as a basis for Mineral Reserves. There are no Inferred Resources within the final pit limits.

TABLE 17-9 PROBABLE MINERAL RESERVES
Victoria Gold Corp. – Eagle Gold Project

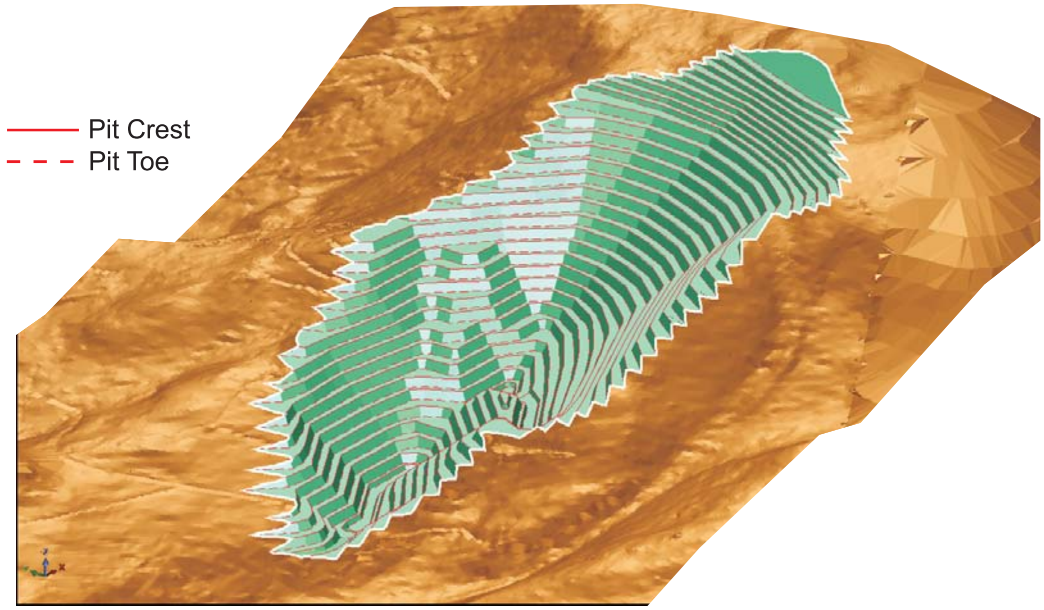
Ore Type	kTonnes	Au (g/t)	Au (Oz)	% Tonnes	% Au Oz
Type A	22,789	0.88	645,600	35	37
Type B	36,442	0.79	930,100	55	53
Type C	6,909	0.79	175,300	10	10
Total	**66,141**	**0.82**	**1,751,000**		

Notes:
1. CIM definitions were followed for Mineral Reserves.
2. Mineral Reserves are estimated at cut-off grades by pit phase and by material type, averaging 0.35 g/t Au.
3. Mineral Reserves are estimated using an average long-term gold price of US$900 per ounce and a US$/C$ exchange rate of 0.90.
4. A minimum mining width of 15 m was used.
5. Bulk density is 2.66 t/m^3.
6. 98.0% mining recovery and 2.0% mining dilution at 0.2 g/t Au applied to Mineral Reserves.

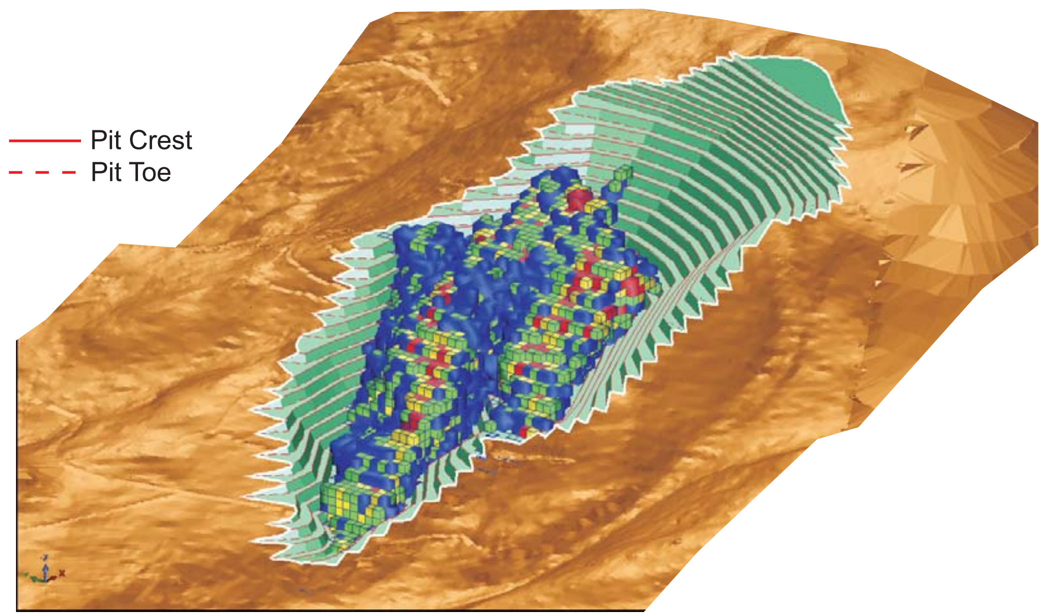
The pit limits are located on a mountainside, with an average slope of 3 horizontal to 1 vertical, with the long axis of the pit striking east-west, roughly parallel to the fall-line of the slope. The footprint of the final pit is just under 70 ha. The west facing final highwall of the pit has the greatest vertical presence at 410 metres to the intermediate pit bottom with a top elevation of 1,410 masl, followed by a 70 metre step down to the final pit bottom at 930 masl. The first bench containing ore is at the 1,260 masl elevation. Figure 17-3 shows the final pit design and Probable Mineral Reserve.

Further details on the generation of the Mineral Reserve pit design are contained in the following section of this Report.

ISOMETRIC VIEW OF FINAL PHASE 4 PIT LOOKING NORTHEAST
FINAL PIT LAYOUT WITH ORIGINAL TOPOGRAPHY



FINAL PIT LAYOUT WITH ORIGINAL TOPOGRAPHY AND RESERVE MODEL



Reserve Legend:

Block Grade g/t Au

- *Undefined*
- *0.00 - >0.50*
- *0.50 - >0.75*
- *0.75 - >1.00*
- *1.00 - >999.00*

Figure 17-3

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Mineral Reserve Final Pit Design

April 2010

18 OTHER RELEVANT DATA AND INFORMATION

The base case PFS operating scenario includes open pit mining, three-stage crushing to a P_{80} of 5 mm, and heap leaching, followed by carbon ADR gold recovery. A general arrangement of the site facilities, showing the ultimate extents of both pit and heap leach pad, is presented in Figure 18-1.

MINE DESIGN

Mine planning for the Eagle Gold Project is based on the current resource model, updated with the results of the 2009 drill campaign. The resource model was generated in Gemcom format as a block model with the following attributes of interest for the purpose of mine planning: *alt_code*, *au*, *percent*, and *rock_type*. Gold is the only element of economic interest in the model, as per the attribute *au*, and there are no penalty elements in the model.

The attribute *au* contains the estimated gold grade for a block for the reported percentage volume (i.e., the *percent* attribute) of that block within a 0.2 g/t Au cut-off wireframe. All Resources within the final pit limits are classified as Indicated; no Measured or Inferred Resources are reported within the final pit limits.

The block model encompasses a volume of 1.95 km x 1.50 km x 0.90 km vertical, with no rotation or dip, and has a defined individual block size of 15 m x 15 m x 15 m.

Mine planning was completed using Gemcom Surpac 6.1 and Gemcom Whittle 4.2 was used for open pit optimization using the Lerchs-Grossman algorithm. As open pit optimization is an iterative process, inputs used for the open pit optimization may not be identical to those in the final cash flow model.

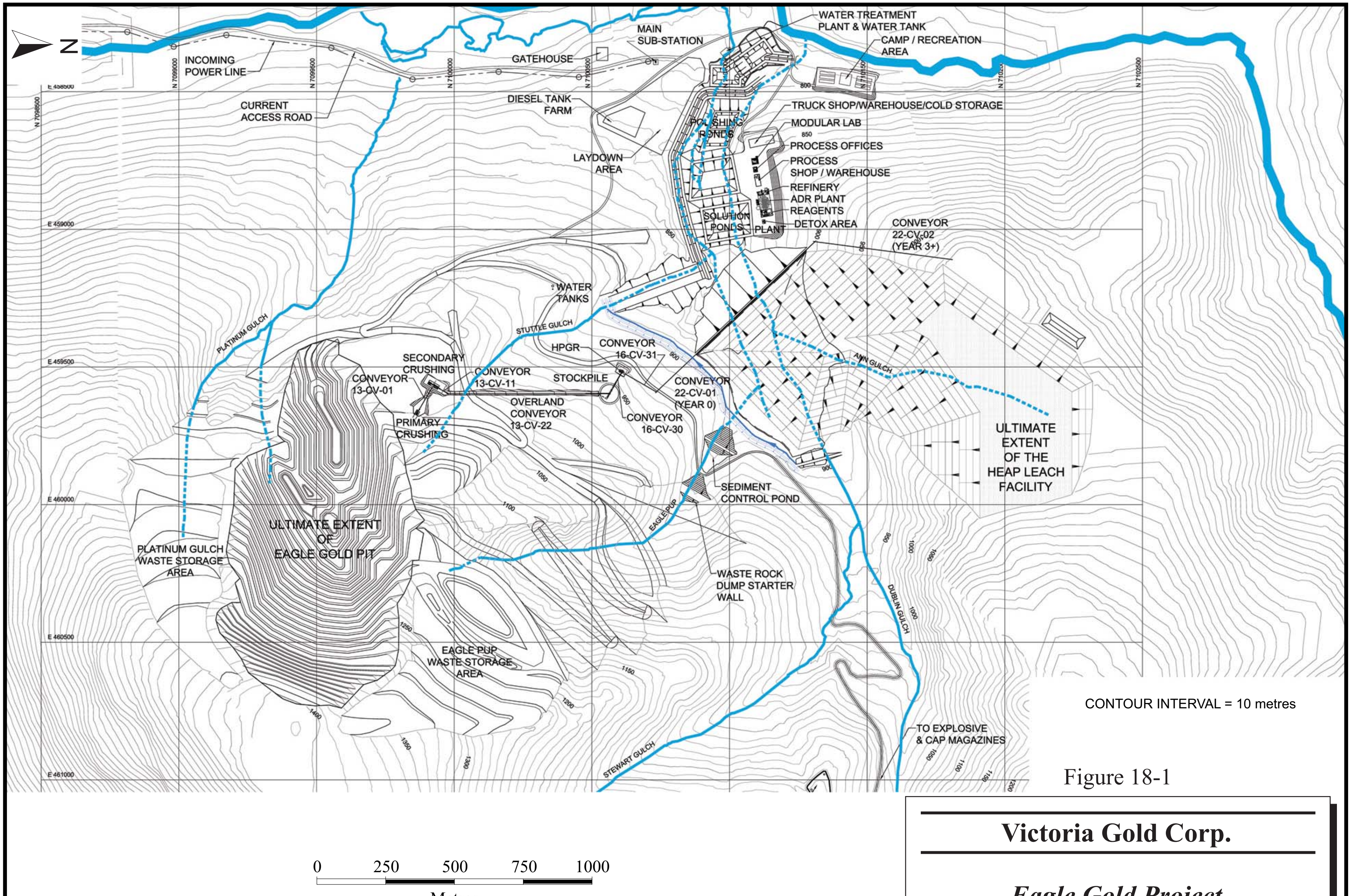


Figure 18-1

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Site Plan

April 2010

Source: Scott Wilson, KCA, 2010.

OPEN PIT OPTIMIZATION

For the open pit optimization, a whole block grade was calculated for export to Whittle. The *percent* of the block outside the constraining 0.2 g/t Au cut-off wireframe, if any, is assumed to carry zero grade.

All non-air blocks in the model are assumed to be competent rock requiring blasting in order to excavate. In Scott Wilson RPA's opinion, this should be a conservative assumption – overburden is present, as can be seen in historic road building within the final pit footprint, however, no surface for overburden has been defined. For the cash flow model, an overburden depth of one metre has been assumed.

A dry density of 2.66 tonnes per cubic metre is applied to all non-air blocks in the model. A moisture content of 5% is assumed for a wet density of 2.80 tonnes per cubic metre.

The final open pit optimization was run using the following inputs, explained in more detail below:

- Pit slopes, Sediments: 35.5º to 44º
- Pit slopes, Intrusives: 32.5º to 43.5º
- Pit slopes, top 35 m of pit: 27º
- Mining recovery: 98.0%
- Mining dilution: 2.0% at 0.2 g/t Au
- Reference mining cost: US$1.55/tonne moved
- Ore mining cost adjustment factor: 1.35
- Process cost: US$4.70/tonne processed
- G&A cost: US$1.95/tonne processed
- Process recovery,
    - Type A: 77.0%
    - Type B: 68.0%
    - Type C: 72.0%
- Gold Price: US$900/oz Au
- Royalty: 1.0%
- Gold selling cost: US$5.00/oz Au

Note: small differences between these inputs and final cash flow inputs reflect later refinements.

GEOTECHNICAL INPUTS

Open pit optimizations were run using pit slope design criteria developed by BGC, based on historical information and four new oriented-core geotechnical drill holes completed in 2009. A global bench face angle of 65º was proposed, and overall slope angles for various design sectors by rock type domain were estimated (see Table 18-1 and Figure 18-2). Based on BGC's design criteria, three slope profiles were developed for use in Whittle. The overall slope angles applied in each slope profile are the same as the corresponding BGC design sectors, or less, which was often the case where the applied slope was not able to change as quickly as the design slope.

The three slope profiles defined in Whittle are based on the *rock_type* attribute in the resource model. Profile 1 controls the sediment *rock_type*, Profile 2 controls the intrusive *rock_type*, and Profile 3 controls the overburden *rock_type*, where overburden is defined by a surface 35 m vertically below topography (this surface is not to be confused with the previous discussion on overburden, and does not represent the contact between free-digging material and material requiring breakage prior to excavation).

Overall slope angles used in Profile 1 range from 35.5º to 44º. Overall slope angles used in Profile 2 range from 32.5º to 43.5º. All slopes in Profile 3 are at 27º overall.

TABLE 18-1 BGC PIT SLOPE RECOMMENDATIONS
Victoria Gold Corp. – Eagle Gold Project

Domain	Design Sector	Slope Azimuth Start	Slope Azimuth End	Bh[1] (m)	Ba[2] (m)	Bw[3] (m)	Ia[4] (º)	Ia[5,6] Control	Oh[7] (m)	Oa[8] (º)	Oa Control[9]
M	M-103	073	133	15	65	14.5	34.5	LOCAL FAILURE	422.5	35.5	LOCAL FAILURE
	M-191	133	249	15	65	9.0	43.0	REGULATION	402.5	43.5	REGULATION
	M-279	249	309	15	65	9.0	43.0	REGULATION	196.5	44.0	REGULATION
	M-321	309	333	15	65	9.0	43.0	REGULATION	225.5	44.0	REGULATION
	M-003	333	033	15	65	9.0	43.0	REGULATION	308.5	43.5	REGULATION
	M-040	033	047	15	65	9.0	43.0	REGULATION	316.5	43.5	REGULATION
	M-060	047	073	15	65	14.0	35.5	LOCAL FAILURE	395.5	36.0	LOCAL FAILURE
I	I-163	120	206	15	65	17.0	32.0	FB2-FB1	410.5	32.5	FB2-FB1
	I-222	206	237	15	65	9.0	43.0	REGULATION	256.5	44.0	REGULATION
	I-242	237	247	15	65	10.0	41.0	FC1	148.5	43.0	FC1
	I-277	247	307	15	65	11.0	39.5	LOCAL FAILURE	190.5	41.0	LOCAL FAILURE
	I-318	307	329	15	65	9.0	43.0	REGULATION	216.5	44.0	REGULATION
	I-359	329	029	15	65	9.5	41.5	FD1	298.5	42.5	FD1
	I-060	029	090	15	65	9.0	43.0	REGULATION	418.5	43.5	REGULATION
	I-105	090	120	15	65	11.5	39.0	FB2	422.5	39.5	FB2

Notes:
1. The single bench height (bh) is 7.5 m; double benching is assumed for all domains and sectors.
2. The bench face angle (ba) has been determined based on industry experience with gold and copper porphyry deposits and based on regional foliation limitations.
3. The catch bench width (bw) is designed to provide rock fall catchment, local failure retention, and geometric compatibility between the ba and interramp angle (la).
4. The interramp angle (la) is determined from a combined review of achievable angle based on bench geometry and stability analysis of slopes.
5. The interramp scale geometry is controlled by kinematically possible failures or bench scale controls.
6. The interramp height may be limited in some sectors by rock mass quality to limit the potential for failure. Where no limit is provided, the interramp height may equal the overall slope height (oh).
7. The maximum overall slope height (oh) for each sector has been estimated from the preliminary pit shell provided by the client; no ramps have been assumed.
8. The overall angle (oa) has been determined from the interramp scale geometry and rock mass stability for the oh. Where rock mass controls the oa additional ramp width maybe required.
9. The oa maybe limited bench, interramp, or overall slope scale controls.

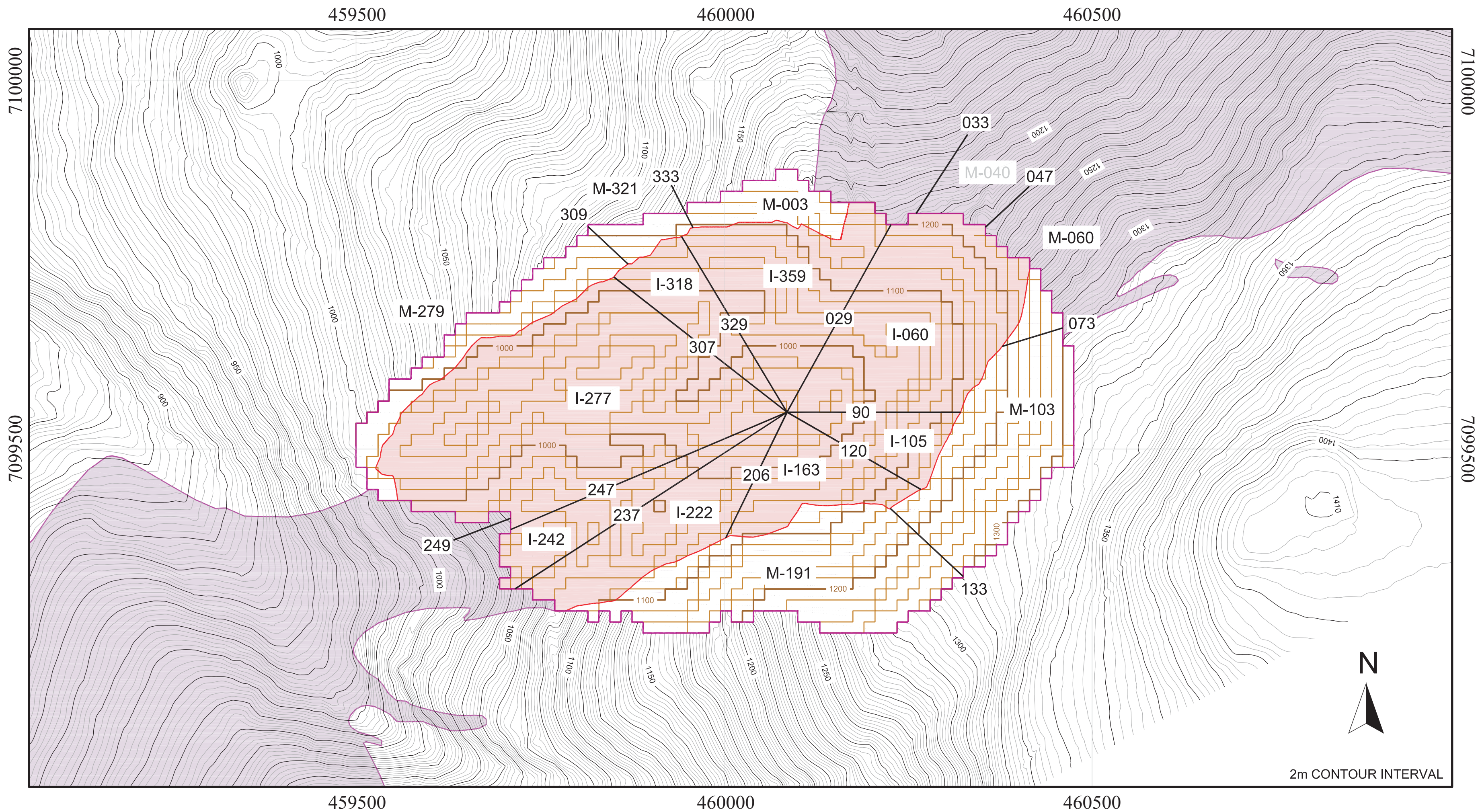


Figure 18-2

Legend:

I-277	*Design Sector*
133	*Design Sector Boundary*
	Whittle Pit Crest
	Intrusive Contact on Whittle Pit [1]
	Intrusive Contact on Surface [3]
	20m / 100m Contours on Whittle Pit

0 100 200 300 400
Metres
NAD83 UTM Zone 8

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Eagle Gold Project

Yukon Territory, Canada

Pit Slope Design Sectors

April 2010

Source: BGC Engineering Inc., 2010.

MINING INPUTS

Mining recovery and dilution are included in the open pit optimization, and are based on:

- mining method, described as a conventional hard-rock open pit mining operation employing medium-scale equipment,
- orebody continuity at the cut-over grade, and
- resource model block size.

Mining dilution of 2.0% at 0.2 g/t Au is applied to the whole-block grade to account for isolated blocks which are mined as ore and incorporate a heavy percentage of dilution, blast mixing, mining selectivity, truck box carryback, and stockpile reclaim activities. Mining recovery of 98.0% is applied to account for isolated ore blocks which are incidentally mined as waste, blast throw losses, loss of material at mining boundaries, misdirected loads, spillage during loading and/or hauling, carryback, and stockpile reclaim losses.

COST INPUTS

A reference mining cost of US$1.55 per tonne moved is applied to waste rock, with a cost adjustment factor of 1.35 applied to ore, therefore the cost to mine one tonne of ore is US$2.09.

Incremental mining costs are calculated for both ore and waste rock based on reference elevations, with additional costs for extra haulage incurred for material above or below the reference. The reference elevation for ore is the primary crusher dump pocket at an elevation of 1,050 masl. An incremental factor of US$0.02 per tonne is applied to ore for each 15 m bench above the reference (hauling downhill loaded), and an incremental factor of US$0.04 per tonne is applied for benches below the reference (hauling up out of the pit loaded). For waste rock, a similar haulage profile was assumed for all elevations above 1,020 masl (waste hauled on-contour out the sides of the pit). For lower benches, an incremental mining cost of US$0.04 per tonne moved is added.

A processing cost of US$4.70 per tonne for crushing and heap leaching is applied.

General and administration cost is US$1.95 per tonne processed, which includes costs for employees to be housed at an onsite camp and a combination of ground and air transport to and from Whitehorse.

REVENUE INPUTS

Two major rock types (Intrusive and Sediments) are identified in the model with one of three alteration types each: Type A, Type B, or Type C alteration. Process recovery is dependent on the alteration type only. For the open pit optimization, gold recoveries of 77% for Type A, 68% for Type B, and 72% recovery for Type C alteration are used.

A base case gold selling price of US$900 per ounce is used in the open pit optimization as revenue factor 1.0. A series of pit shells were generated at various gold revenue factors from 0.3 to 1.3 (gold prices ranging from US$270 per ounce to US$1,170 per ounce) for further design work and mine scheduling. A 1% royalty on the gold selling price is applied to all gold production from the Project.

A gold selling cost of US$5.00 per ounce is applied to recover costs of selling including, but not limited to, security, transportation, refining, and marketing.

OPEN PIT OPTIMIZATION RESULTS

The optimum cash pit result for US$900 per ounce gold, revenue factor 1.0, is Pit Shell 65 as shown in Figure 18-3. Also highlighted are Pit Shells 56 and 52.

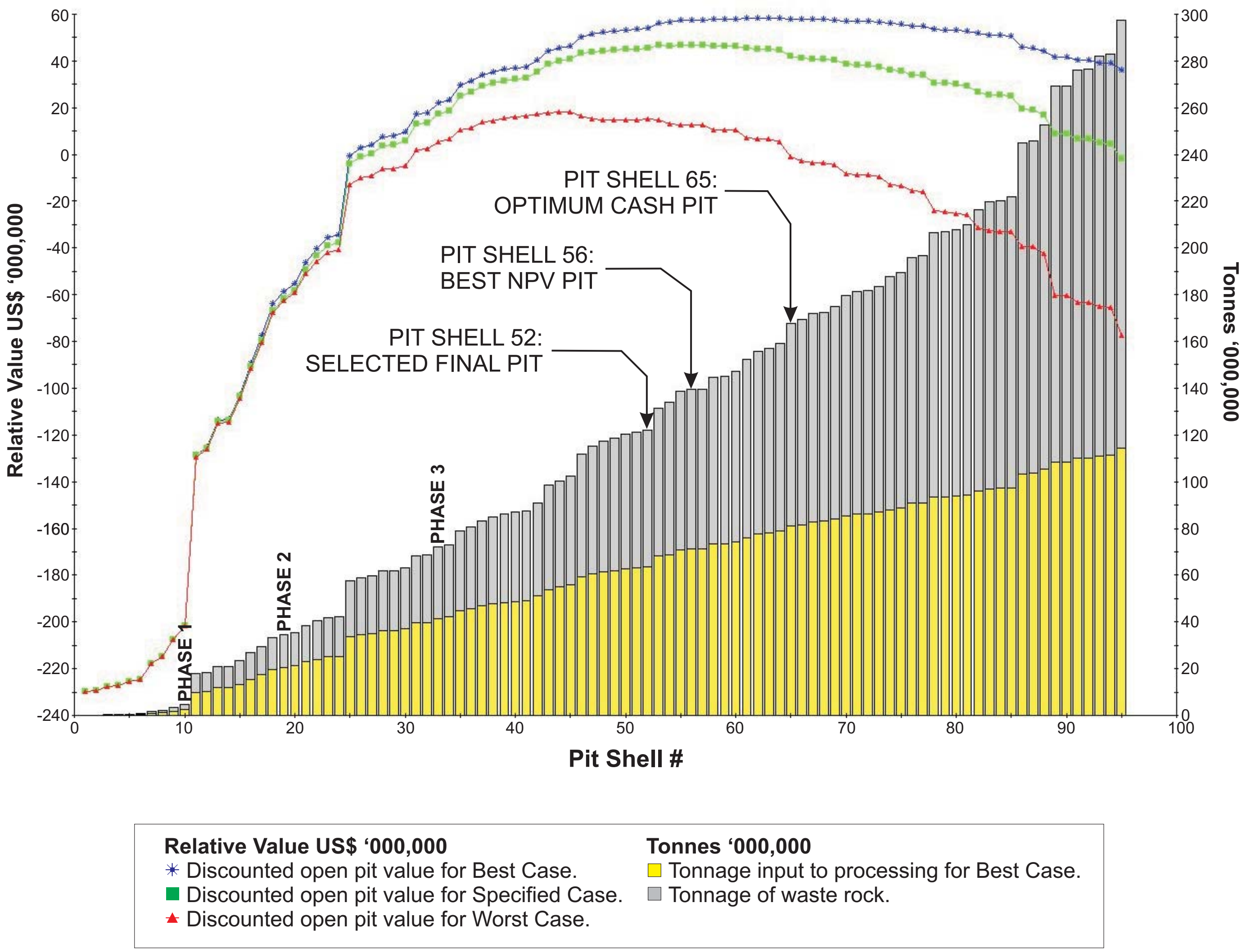


Figure 18-3

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Whittle Pit by Pit Graph of Gold Price Sensitivity

April 2010

Pit Shell 56, revenue factor 0.91 (gold price of US$819 per ounce), is the best net present value pit based on:

- 7.5% discount rate,
- Project capital cost of US$250 million and salvage value of US$20 million;
- Three select phase pits as shown in Figure 18-3; and
- Application of a raised process cut-off grade of 0.06 g/t Au on a recovered basis during the first three mining phases.

The selection of phase pits to run preliminary mine schedules was based on the yearly production requirements, consideration for minimum pushback distances, consideration for vertical rate of bench development, and the net present value. In addition, raised process cut-off grades between zero and 0.2 g/t Au on a recovered basis were examined over the life-of-mine, diverting lower grade material to stockpile.

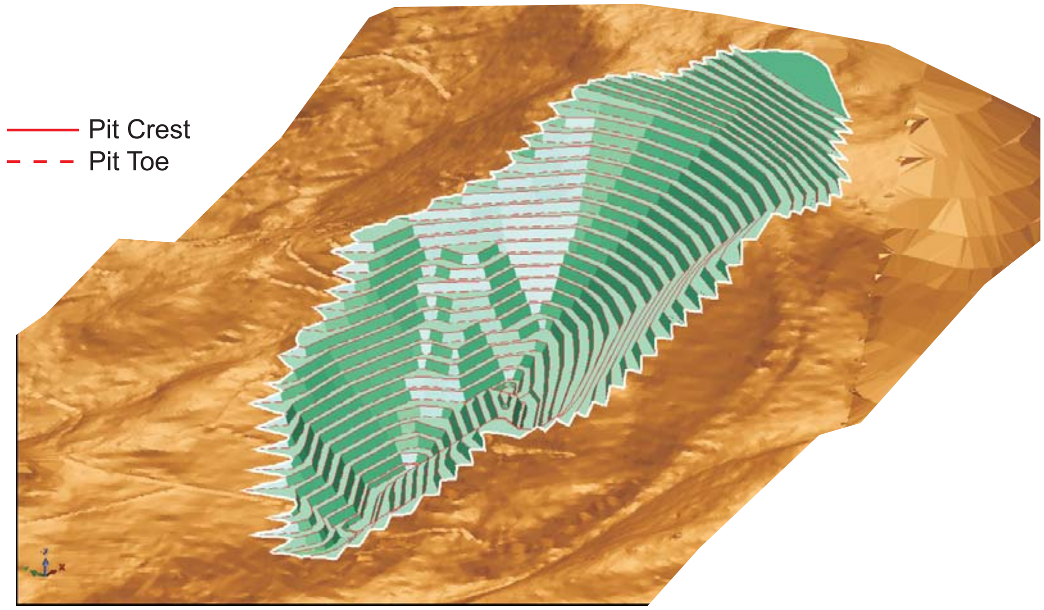
Pit Shell 52, revenue factor 0.87 (gold price of US$783 per ounce), was selected as the optimum pit shell for use as a guide for the final design pit, instead of Pit Shell 56, the best net present value pit, because of material quantity constraints of the receiving heap leach pad, which is designed for a maximum of approximately 65 million tonnes of ore.

The Whittle reported results for Pit Shell 52 are 63.8 million tonnes of mineralized material above applied cut-off grades, with the remaining heap leach pad capacity to be filled up from stockpiled material at the end of the pit life.

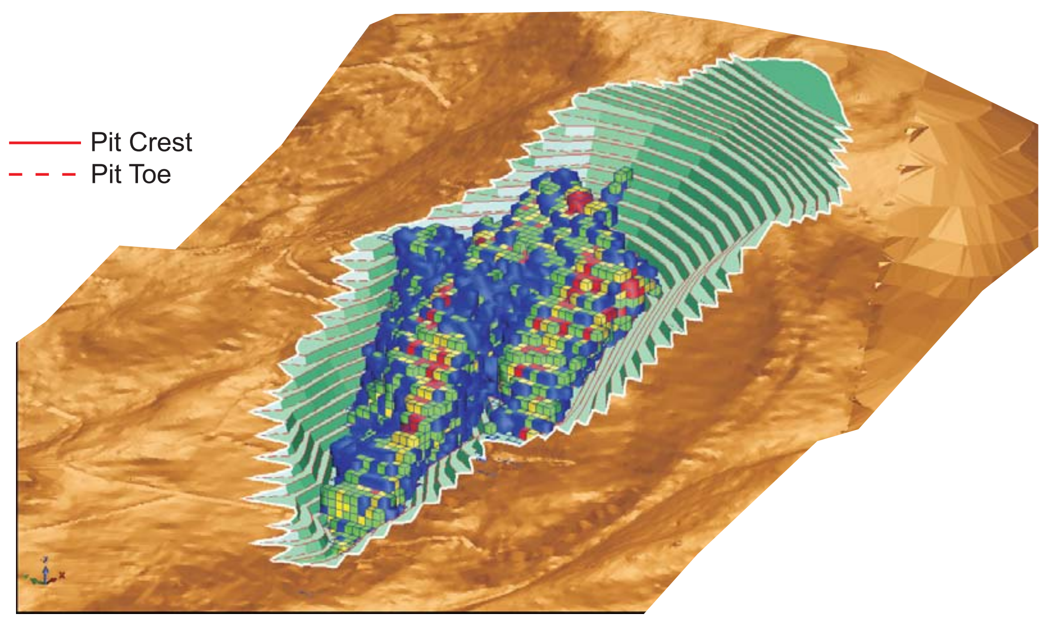
OPEN PIT DESIGN

The final design pit limits are located on a mountainside with an average slope of 3 horizontal to 1 vertical, with the long axis of the pit striking east-west, roughly parallel to the fall-line of the slope. The footprint of the final pit is just under 70 ha. The west-facing final highwall of the pit has the greatest vertical presence at 410 metres to the intermediate pit bottom with a top elevation of 1,410 masl, followed by a 70-metre step down to the final pit bottom at 930 masl. The first bench containing ore is at the 1,260 masl elevation. Figure 18-4 shows the final pit design (and Probable Mineral Reserve).

ISOMETRIC VIEW OF FINAL PHASE 4 PIT LOOKING NORTHEAST
FINAL PIT LAYOUT WITH ORIGINAL TOPOGRAPHY



FINAL PIT LAYOUT WITH ORIGINAL TOPOGRAPHY AND RESERVE MODEL



Reserve Legend:

Block Grade g/t Au

- *Undefined*
- *0.00 - >0.50*
- *0.50 - >0.75*
- *0.75 - >1.00*
- *1.00 - >999.00*

Figure 18-4

Victoria Gold Corp.

Eagle Gold Project
Yukon Territory, Canada

Final Pit Design

April 2010

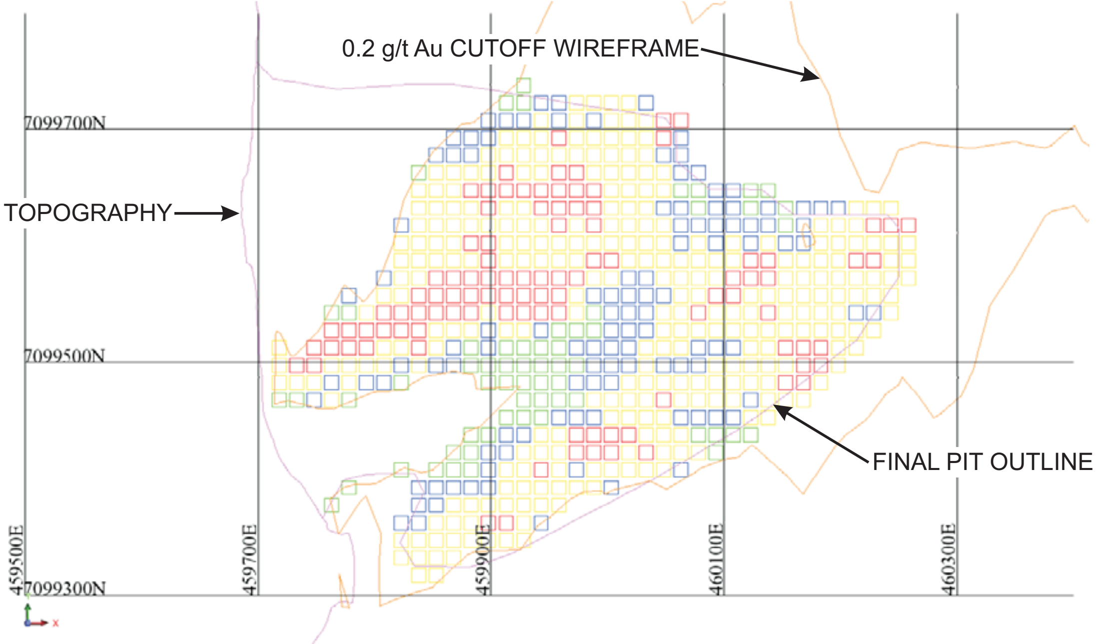
Bench geometry for pit design is based on the design criteria supplied by BGC, used in the pit optimization. A global bench face angle of 65 degrees is used, along with a 15 m bench height. Safety berm widths are variable by domain and design sector to achieve the minimum required overall and interramp slope angles. The minimum safety berm width is 9 metres, with a maximum of 17 metres. The application of controlled blasting to achieve steeper bench face angles and overall steeper slopes was investigated, however, due to the presence of numerous design sectors that had limited to no increase in overall slope angle controlling the overall slope model, limited benefit in the form of reduced strip ratio was observed for the increased risks and costs attached to achieving steeper overall pit slopes. The final pit design was reviewed by BGC for consistency with their recommended slope design criteria, and accepted.

Haul road width is based on 91-tonne payload class haul trucks. For two-way traffic and a single shoulder barrier, a minimum 24 m design width is allowed, based on three times the truck operating width and a shoulder barrier at three quarters height of the truck tire. A shoulder barrier is employed whenever there is a drop-off of greater than three metres. Similarly, for single-way traffic, a minimum of 18 m design width is used.

For the final pit, no haul roads are left in the highwalls. However, during pit phase construction, haul roads exist both internal and external of the pit. Road design gradients are typically between 8% and 10%, however, segments up to 200 metres with gradients up to 12% may exist in various pit phases and on external roads. The bottom of the final pit is accessed via a sinking cut ramp with ramp retrieval in the last half bench to maximize mining recovery of ore.

Upon completion of the final pit design, and with further study of the resource within the pit limits, it was determined that the mining recovery and dilution factors applied during the open pit optimization are appropriate for use in Mineral Reserve reporting. The blocks in the model above the cut-off grade exhibit good continuity on bench plans, with typical mining widths of over 50 m, limited internal dilution, and a limited number of isolated block combinations of less than 30 m in any one direction. Figure 18-5 shows two example bench plans. In addition, with the application of the raised process cut-off grade within the first three pit phases, dilution will often be from mineralized material above the economic cut-off grade.

EGP 1072.5masl BENCH PLAN



EGP 1162.5masl BENCH PLAN

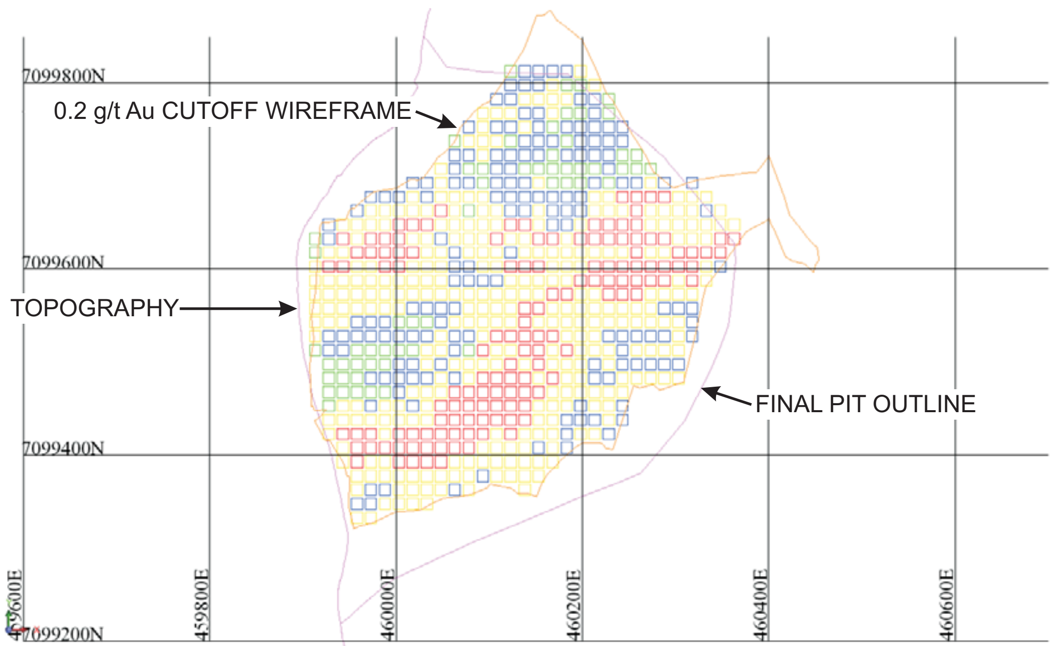


Figure 18-5

BLOCK LEGEND

- >=0.1, <0.3 g/t Au
- >=0.3, <0.5 g/t Au
- >=0.5, <1.0 g/t Au
- >=1.0 g/t Au

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Eagle Gold Project

Yukon Territory, Canada

Sample Bench Plan Diagram

April 2010

OPEN PIT PHASES

A starter pit and three pushbacks to the final pit limits were selected to maximize the Project's net present value at a mine production rate of 2 Mt ore in 2013, followed by 6 Mt per year ore thereafter, based on a 250 day per year operating schedule. Post completion of the phase selection and pit design, the mine schedule was adjusted to a year round operation, mining 3.3 Mt ore in the first year followed by 9.1 Mt ore thereafter, however, the daily mining rate remained the same at 26,000 tonnes per day ore.

In selecting pit shell pushbacks, a minimum mining width of 50 m was considered. Due to the orientation and geometry of the model, a typical mining bench plan is shaped like the letter "U", with pinching and swelling, and intersects surface allowing access to all safety berms. The last few benches are an exception, and are accessed via a sinking-cut ramp.

During the construction period in years 2012 and 2013, a waste rock only pre-strip is developed at the highest elevations of the Phase 1 starter pit and the Phase 2 pit. This waste rock is used as fill material for site road development, specifically from the starter pit down to the run-of-mine stockpile area and the road loop which provides access to the heap leach facility embankment rockfill. To increase the speed of fill road development, a second waste rock only borrow pit is developed lower down the mountainside at the base of the Phase 3 pit. Once the required site road development is complete, additional waste rock material from the pre-strip area is excavated for the embankment rockfill.

A raised process cut-off grade is applied during the first three mine Pit phases with mineralized material going to stockpile as follows:

- Type A – between 0.32 g/t Au and 0.38 g/t Au,
- Type B – between 0.37 g/t Au and 0.44 g/t Au, and
- Type C – between 0.35 g/t Au and 0.41 g/t Au.

Mineralized material grading higher than the raised process cut-off grade is either direct dumped to the primary crusher dump pocket, or stockpiled at the run-of-mine stockpile area adjacent to the primary crusher.

A breakdown of Probable Mineral Reserves by mining phase is shown in Table 18-2.

TABLE 18-2 PROBABLE MINERAL RESERVES BY PHASE
Victoria Gold Corp. – Eagle Gold Project

Phase	Ore kTonnes	Ore Au g/t	Stockpile kTonnes	Stockpile Au g/t	Waste kTonnes	Strip Ratio Waste+Sp:Ore
Phase 1	2,895	1.06	142	0.38	2,507	0.92
Phase 2	18,180	0.92	1,511	0.38	14,055	0.86
Phase 3	19,281	0.85	1,205	0.39	23,098	1.26
Phase 4	25,785	0.71	0	na	25,957	1.01
Total	**66,141**	**0.82**	**2,858**	**0.39**	**65,616**	**1.04**

OVERBURDEN

Overburden stripping for the purpose of mine cost estimating is defined as any material at surface not requiring a drill and blast cycle to break the material prior to excavation. No overburden surface is defined in the resource model, however, an estimate of overburden thickness is seen in the numerous exploration roads that have been built, with a trend of thickening overburden at lower elevations. A thickness of one metre of overburden is assumed over the entire pit footprint of approximately 70 ha. This is likely a conservative estimate, taken in the absence of any test pit or drill hole data.

During operations, salvageable soil for use in reclamation will be dozer pushed downslope into windrow piles for pickup and stockpiling in the designated area. This operation is preferred to be completed when the ground is dry and not frozen, typically July and August. Remaining overburden is windrowed by dozer for load and haul or used to build pioneering benches and drill platforms.

STOCKPILES

Two stockpile areas are defined in the mine plan for lower-grade mineralized material mined in Phases 1, 2, and 3. Potential for additional stockpile capacity is available if warranted for segregation of grade groups and or alteration types.

Stockpiles are designed with a 35% swell factor, 36º face angles (angle of repose minus one degree), and a minimum 5 m wide terrace every 20 m of vertical. The majority of terraces, however, are 18 m wide or greater, to allow for greater flexibility during reclamation.

The decision to send mineralized material to the stockpile is based on a raised process cut-off grade of 0.06 g/t Au on a recovered basis. The bottom cut-off for entry to the stockpile is based on the internal cut-off grade with an additional cost of material rehandle applied. Phase 1, 2, and 3 mineralized material allocated to stockpile is as follows:

- Type A – between 0.32 g/t Au and 0.38 g/t Au,
- Type B – between 0.37 g/t Au and 0.44 g/t Au,
- Type C – between 0.35 g/t Au and 0.41 g/t Au.

Stockpiled material during the life-of-mine is currently not processed in the Project cash flow model, due to capacity restrictions of the final heap leach facility, thus is not included as ore in the Mineral Reserve statement. This material is economic, however, it is very close to break-even, and does not have a significant impact on the cash flow if included.

A third short-term stockpile area is designed at the primary crusher dump pocket for run-of-mine ore with a few days production capacity. This stockpile is only used when the primary crusher is not available for direct dumping. The stockpile is accessed from both the 1,065 masl and 1,080 masl elevations, and it is within tramming distance for reclaim to the dump pocket by a front end loader.

ORE

The primary crusher dump pocket is located at 1065 masl, just over 100 m north of the final pit rim. During regular operations, run-of-mine ore is to be direct-dumped into the dump pocket, or the run-of-mine stockpile if the primary crusher is not available for direct dumping.

WASTE DUMPS

Using a 35% swell factor for waste, life-of-mine waste production is 31.5 million loose cubic metres including overburden, with an additional 1.5 million loose cubic metres going to stockpile. During the life-of-mine plan, waste rock is scheduled to go to one of five areas:

1. Fill material for haul road development
2. Fill material for the heap leach facility embankment rockfill
3. Platinum Gulch runaway lanes and waste rock storage facility
4. Eagle Pup waste rock storage facility
5. Pit backfilling

Fill road development is designed at an as-dumped 36º face angle (angle of repose minus one degree). Where fill development occurs above a road alignment, in the form of another road or a terrace, the as-dumped angle is re-sloped to a 1.5 horizontal to 1 vertical slope to improve stability and reduce potential road cleanup from minor sloughing.

Waste rock storage facilities are built by the top-down construction method, wrap around method, or in lifts from the bottom-up.

For top-down construction, dump crests are designed to a maximum vertical lift of 100 m at a 36º face angle (angle of repose minus one degree). Waste rock buttresses keyed in at the design toe along with intermediate buttresses are installed as required for increased stability and to reduce downslope risk. Terraces between lifts are incorporated with the minimum width designed to allow for re-sloping during closure to a 2 horizontal to 1 vertical overall slope or better.

For bottom-up construction, in order to maintain minimum operating widths of 18 m for single-way traffic and 24 m for two-way, single-shoulder traffic, dump faces are designed at angle of repose minus one degree, which can result in dump crest heights of up to 50 m vertical, as all dump toe foundations are on downhill sloped terrain. Dump toes are to be buttressed and keyed in as required for stability. Additional lifts are added as per the mine plan, with individual dump lifts stepped back to achieve the desired closures face angle, or terraces left in place of sufficient width to allow for resloping to the closure face angle.

The Platinum Gulch waste rock storage facility reaches its ultimate footprint of approximately 33 ha at the end of Year 3 (2015). The Eagle Pup Gulch waste rock storage facility reaches its ultimate footprint of approximately 80 ha at the end of Year 8 (2020), immediately prior to closure. Additional surface disturbance related to the mine from pit and rock storage facility site access roads is approximately 30 ha, plus an additional 10 ha impacted for the explosive site and magazine site access roads and site areas.

Over the life-of-mine, typical haulage profiles for waste rock average approximately 1,500 m with a gradient of between plus or minus 5%. This estimate was performed prior to the waste dump design work, on the basis that Eagle Pup would be the primary rock storage site. With waste dump design complete, further analysis of the waste rock haulage profile can be completed for a higher-level study, with cost advantages expected to be realized from shorter hauls to Platinum Gulch.

Testwork on potential for acid generation or metal leaching from waste rock is currently in progress. Initial results and past work indicate that the material is fairly inert, with neither strong acid-generating qualities, nor strong buffering capacity. The current mine plan assumes that no segregation of any particular material will be required, although all industry-standard options for managing fairly inert waste rock are possibilities, if needed. Provision for collection of all drainage water from both Eagle Pup and Platinum Gulch has been included, allowing for testing before direct discharge, treatment, or use as process water, as appropriate.

MINE PRODUCTION SCHEDULE

The mine production schedule is generated based on the Indicated Resources within the designated pit phases and final pit limits using the following guidelines:

- Target of a nominal 3.3 Mt of ore in Year 1 (2013), followed by 9.1 Mt per annum thereafter; 26,000 tpd based on the annual operating schedule.
- Processing cut-off grade of 0.38 g/t Au, 0.44 g/t Au, and 0.41 g/t Au for Type A, B, and C ores respectively during the first three mining phases.
- Processing cut-off grade of 0.31 g/t Au, 0.35 g/t Au, and 0.33 g/t Au for Type A, B, and C ores respectively during the final Phase 4 mining phase.

- Schedule waste rock for optimum equipment selection to maximize NPV of mine operations.

The life-of-mine production schedule by year is shown in Table 18-3.

TABLE 18-3 LIFE OF MINE PRODUCTION SCHEDULE
Victoria Gold Corp. – Eagle Gold Project

Year	Ore kTonnes	Grade g/t Au	Stockpile kTonnes	Waste kTonnes	Strip Ratio (Waste+Stockpile:Ore)	Contained Au Oz
2012	0	0.00	0	587	NA	0
2013	3,300	1.02	190	6,848	2.13	108,100
2014	9,100	0.84	1,029	5,647	0.73	244,800
2015	9,100	1.01	483	13,739	1.56	294,600
2016	9,100	0.83	623	8,752	1.03	242,700
2017	9,100	0.87	529	12,885	1.47	253,600
2018	9,100	0.73	4	9,476	1.04	214,800
2019	9,100	0.73	0	3,978	0.44	213,700
2020	8,241	0.67	0	3,704	0.45	178,600
Total	**66,141**	**0.82**	**2,858**	**65,616**	**1.04**	**1,751,000**

MINE EQUIPMENT

The Project is a conventional open pit mining operation with two diesel hydraulic excavators as the primary loading units, loading off-highway, rigid-frame, mechanical-drive haulage trucks. A fleet of self-propelled, diesel, rotary down-the-hole hammer drills are selected for production rock drilling. Table 18-4 lists the mine and support equipment list. The mining fleet will be owned and operated by the owner, starting in Year 1 (2013). A mine contractor is used during the construction period for development work.

TABLE 18-4 MINE EQUIPMENT
Victoria Gold Corp. – Eagle Gold Project

Equipment Description	2013	2014	2015	2016	2017	2018	2019	2020
Reichdrill C-700 Drill	2	2	2	2	2	2	2	2
Sandvik DX800 Drill	1	1	1	1	1	1	1	1
Hitachi EX1900 Excavator	2	2	2	2	2	2	2	2
Cat 777 Haul Truck	8	8	9	9	9	9	6	8
Cat 992 Wheel Loader	1	1	1	1	1	1	1	1
Cat D10 Track Dozer	1	1	1	1	1	1	1	1
Cat D8 Track Dozer	1	1	1	1	1	1	1	1
Cat 16H Motor Grader	2	2	2	2	2	2	2	2
Water Truck, 14,000 gal	1	1	1	1	1	1	1	1

DRILLING

Production drilling in mineralized material and waste is performed by a fleet of two identical, track-mounted, diesel-powered, rotary down-the-hole hammer drills, which can drill a 7.5 m bench height (including subdrill requirement) in a single pass. The selected drills are capable of hole diameters of up to 229 mm, while the design pattern is based on a 203 mm diameter hole. Production drilling assumptions are listed in Table 18-5. For Year 3 to Year 6 (2015 to 2018), it is expected that a contractor's drill with similar capabilities will be required at site on standby. Up to 50% utilization of the contractor's drill has been costed for Years 3 and 5 (2015 and 2017), to meet peak equipment requirements. For Years 4 and 6 (2016 and 2018), the owner's fleet has sufficient drill capacity to meet the production schedule (including mechanical availability factors), but with no spare capacity available, other than a smaller drill specified for pioneering work and preshear drilling.

TABLE 18-5 PRODUCTION DRILLING ASSUMPTIONS
Victoria Gold Corp. – Eagle Gold Project

Parameter	Mineralized Material	Waste
Bench Height	7.5 m	7.5 m
Subdrill	1.5 m	1.7 m
Burden	5.5 m	6.0 m
Spacing	6.3 m	6.9 m
Hole Diameter	203 mm	203 mm
Penetration Rate	35 m/hr	35 m/hr
Drill Capacity	8.5 Mtpa	10.0 Mtpa

A Sandvik DX800 down-the-hole hammer drill is selected for use in pioneering new benches at surface to ensure proper breakage and a good operating floor. It is also scheduled for use in final wall control preshear drilling. Based on the mine schedule, pioneering work is at its highest levels in the initial years of the mine life, declining over time, while conversely there is little to no final wall preshear drilling required in the first few years, with utilization increasing later in the mine life.

BLASTING

The supply of explosives has been costed as a contractor-provided service for delivery of explosives to the drill hole. The mine will supply a graded area for the explosives contractor, as per requirements of the explosives licence. The explosives contractor will supply all infrastructure and vehicles required to deliver the explosive product to the hole. Due to the remote nature of the operation and the volume of product scheduled for consumption, it is expected that a base factory licence will be a preferred option versus a satellite explosives storage licence; however, the final decision will be based on discussion with the potential explosives contractors. All explosives contractor services for delivery of explosives to the hole have been priced into the unit cost of explosive product.

For costing, 75% of the drill holes are assumed dry and will use Fortan15 emulsion, while 25% are assumed wet and will use Fortis wet hole product, with pricing based supplied by BXL Bulk Explosives Limited. A powder factor of 0.20 kg explosive per tonne has been factored for waste, and 0.23 kg explosive per tonne for mineralized material. The Fortan15 emulsion was selected over more traditional ANFO (Ammonium

Nitrate/Fuel Oil) as it presented a better overall cost per tonne for the complete drill and blast cycle.

The explosives contractor will also supply the magazine(s) for storage of initiation and detonation consumables and maintain supply for operations. The mine will supply a graded area for the explosives contractor to locate the magazine(s) as per requirements of the explosives licence. The mine will be responsible for the installation of the initiation system and detonating devices at the blast site, and firing of the blast patterns.

LOADING AND HAULING

The selected primary shovel and truck fleet consists of two Hitachi EX1900 diesel hydraulic excavators with 15 m^3 buckets, loading a fleet of Cat 777 91-tonne payload mechanical-drive haul trucks. Table 18-6 lists the design basis for determining excavator productivities.

TABLE 18-6 LOADING PRODUCTIVITIES
Victoria Gold Corp. – Eagle Gold Project

Parameter	Excavator
Bucket Size	15 m^3
Machine Power	1,087 Hp
Loaded Bucket Capacity	22.4 tonnes
Loading Passes	4
Load Time Per Truck	2.4 min
Productivity	1,800 t/hr
Loading Capacity	11.0 Mtpa

Figure 18-6 compares the run-of-mine production schedule to the haul truck fuel consumption, showing a fairly consistent fuel consumption profile, which corresponds with the yearly truck count in Table 18-4.

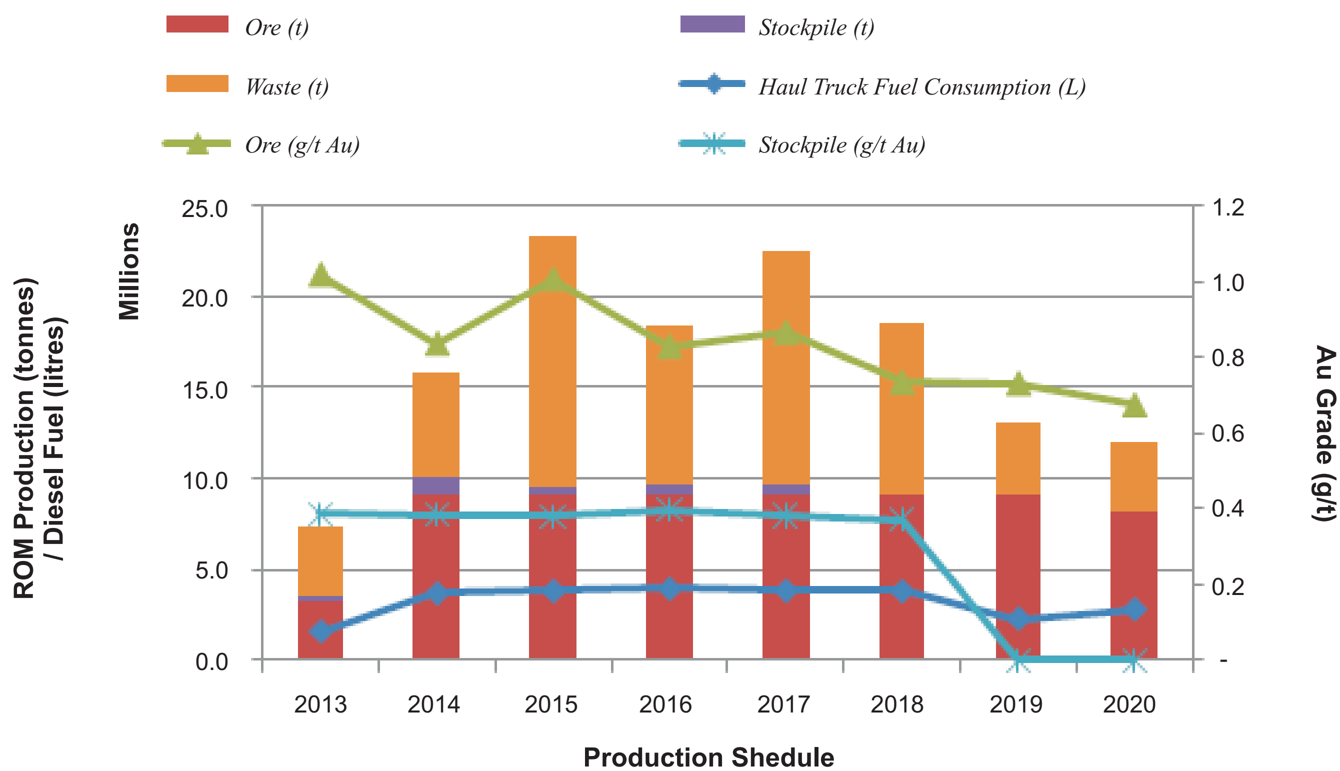

Figure 18-6

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Production Schedule

April 2010

MINE SUPPORT EQUIPMENT

A wheel loader has been included, with the primary function of loading run-of-mine stockpiled ore into the primary crusher dump pocket as required. In addition, it will be available to support the primary loading fleet during production shortfalls, and also has operating hours assigned for snow removal.

Track dozers are included for multi-purpose work, with the larger D10 dozer primarily for pioneering benches and road construction, and the smaller D8 dozer for waste dump development. A second D8 dozer will be available for use by the mine for up to half the year. This dozer's primary operation is as the second dozer on the heap leach pad, for dozing winter stockpiled crushed ore into the dozer trap, to be fed into the conveyor system for stacking crushed ore.

Motor graders are included primarily to maintain haulage roads and pit floors for smooth haulage operations. Secondary functions include maintaining the shovel loading area floor, drill and blast support, and snow removal.

A water truck has been specified for haul road dust suppression and will have a water monitor for fire-fighting as required.

MINE ANCILLARY EQUIPMENT

The mine ancillary equipment fleet is listed in Table 18-7.

TABLE 18-7 MINE ANCILLARY EQUIPMENT
Victoria Gold Corp. – Eagle Gold Project

Equipment Description	Quantity
Backhoe, 300 HP	1
Fuel/Lube Truck, 3,600 gal	1
Mobile Workshop/Service Truck	1
Tire Service Truck with Manipulator	1
Mobile Lighting Units	8
Light Vehicles and Personnel Carriers	15
Mine Rescue Truck	1

HEAP LEACH PAD DESIGN

The proposed heap leach facility (HLF) is located approximately 1.2 km north of the Eagle Zone orebody. The majority of the HLF is located in the Ann Gulch catchment, a tributary to Dublin Gulch. The base of the HLF is in the valley floor of Dublin Gulch, at an elevation of 840 m and at full height, the HLF extends up Ann Gulch to an elevation of 1080 m.

The HLF composes of a number of elements: a rock filled embankment to provide stability to the HLF, a lined storage area for the ore to be leached, an in-heap storage pond to contain the pregnant solution, pumping wells for the extraction of solution, ponds to contain excess solution in extreme events, diversions, sediment control ponds (SCPs) and leak detection, recovery and monitoring systems to ensure the containment of solution. An associated structure is the Dublin Gulch waterway diversion, to the south side of the valley.

SITE SELECTION

Site selection for the HLF site was based on a two stage assessment of the suitability of potential locations; an engineering assessment, and a project-wide assessment of impacts.

Following initial screening of a variety of potential heap leach sites in the wider Dublin Gulch area, six sites were considered for taking forward (Figure 18-7). The potential site options for the HLF include:

- Option 1 – Cross valley type HLF within Dublin Gulch (lower valley)
- Option 2 – Cross valley type HLF within Dublin Gulch (mid valley)
- Option 3 – Valley type HLF on Potato Hills within Bawn Boy headwaters (the selected site in the 1996 Rescan study)
- Option 4 – Side valley type HLF on slopes below the Eagle Zone ore deposit
- Option 5 – Valley type HLF on granodiorite ridge within Olive Gulch headwaters
- Option 6 – Side valley type HLF in Ann Gulch headwaters

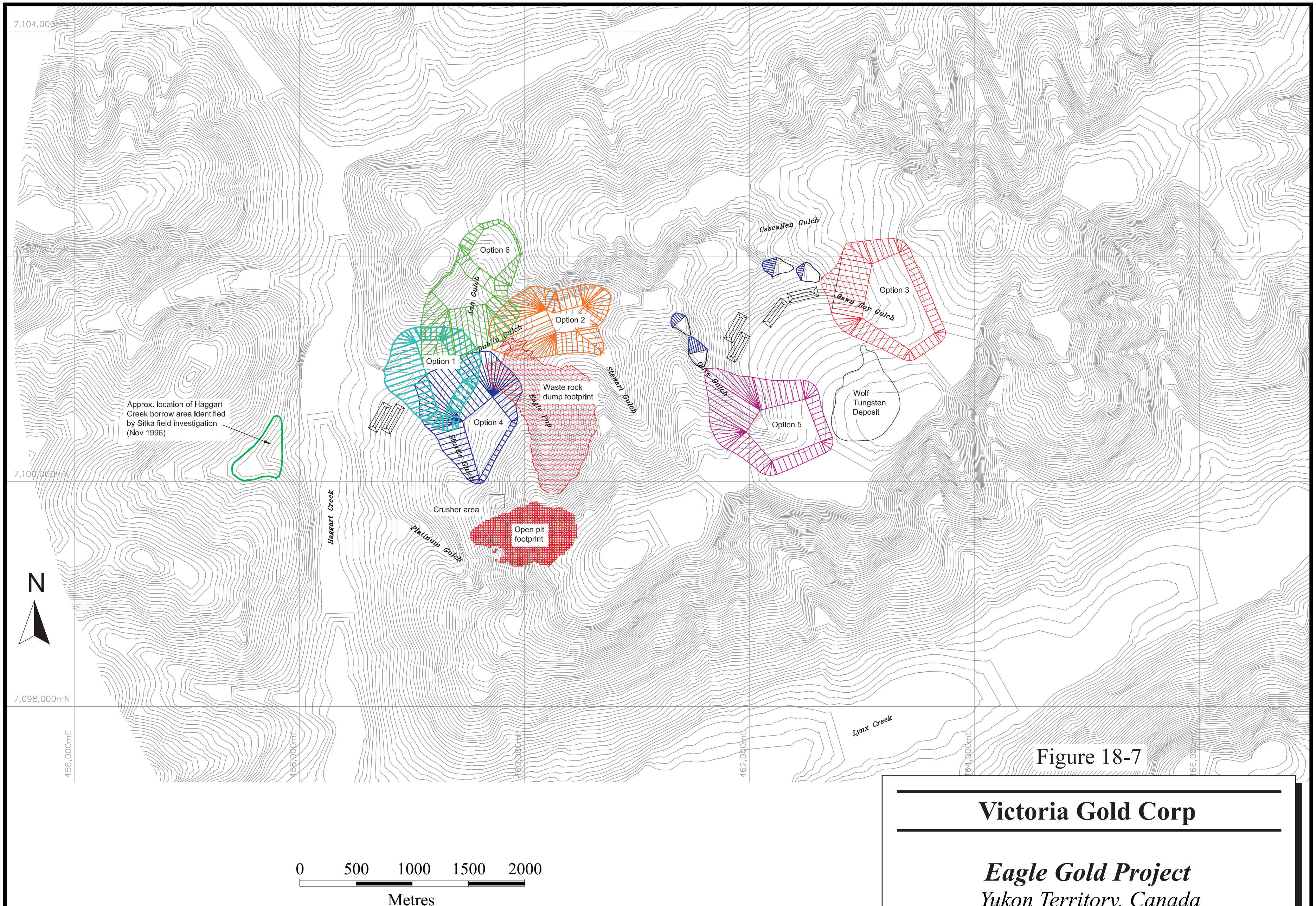

Figure 18-7

Victoria Gold Corp

Eagle Gold Project

Yukon Territory, Canada

Heap Leach Site Options

April 2010

Source: Scott Wilson, KCA, 2010.

ENGINEERING ASSESSMENT

The engineering assessment considered the factors that influence the suitability of the facility at each site, using a qualitative comparison of each site against a set of significant engineering (cost related) criteria. These criteria are drawn from Scott Wilson RPA's experience of the design, construction, and closure of heap leach facilities, and include:

- Land surface area – affecting pad capacity
- Topography – slope of area
- HLF shape – covering a compact area or spread out
- Material handling access – stacking operations
- Preparatory works – requiring embankment or not
- Geotechnical concerns – stability concerns due to permafrost, soil cover
- Closure – cost/difficulty

The engineering assessment of alternatives established Options 3, 5 and 6 as scoring significantly higher than options 1, 2 and 4. From an engineering and construction perspective of the heap leach pad, Option 3 - Potato Hills is the most favourable of the leading group.

PROJECT-WIDE ASSESSMENT

A Project-wide consideration of the leading options was undertaken in regard to impacts of the HLF site options on:

- Mining operations – particularly haulage and access
- Other infrastructure layouts
- Mineral resources - condemnation requirements
- Environment – notably on surface and ground water, fauna (fisheries), flora, and visual as well as consideration for archaeological, air quality, sociology

The results of the Project-wide review of the leading three sites established a clear site location preference in Option 6 - Ann Gulch, with similar neutral scores as compared to other sites, but much lower impacts on (costs to) mining and infrastructure. The main disadvantages of each option, in comparison to Option 6, are listed below:

- Option 1 – difficult foundation material, largest watershed
- Option 2 – slopes too steep for liner installation

- Option 3 – longest haul from open pit (and uphill)
- Option 4 – most permafrost, difficult foundation material
- Option 5 – longer uphill haul, some steep slopes in throat of valley

It was concluded that Option 6 was to be taken forward for PFS engineering.

HLF DESIGN BASIS

Heap leach design standards adopted for the project include:

- Regulatory requirements of Yukon and Canada.
- Permitting requirements of the State of Nevada. These are not regulatory requirements in the Yukon, but are considered as standards for best practice.
- Guidelines from the International Finance Corporation.

Taking in to account the requirements of the various stakeholders, the principal objectives of the Eagle Gold Project HLF are to:

- Ensure complete protection of the regional groundwater and surface water flows both during operations and in the long-term.
- Satisfy the environmental regulatory requirements of the Yukon Territory and the Federal Government.
- Provide permanent, secure storage and total confinement of the leach ore within a fully-engineered facility.
- Effectively collect and convey solutions for in-heap pregnant solution storage to ensure maximum recovery. In-heap storage of solution will be utilized to provide the necessary winter time storage of solution in an above freezing environment.
- Minimize the quantity of surface water runoff entering the facility and coming into contact with the process solutions.
- Provide additional external facilities (events ponds) to accommodate excess solution and rainfall/snowmelt when hydrological events exceed the storage capacity of the heap.
- Develop the facility in stages, where possible, to minimize the environmental disturbance at any one time, and to distribute capital expenditure over the life of the facility.
- Monitor all aspects of the facility to ensure that the design objectives are met and that there are no adverse environmental impacts.

- Rehabilitate the facility to a condition compatible with the original land use and is stable under extreme precipitation events and seismic events.

HEAP LEACH PAD DESCRIPTION

The heap leach pad is a combination valley and side valley heap leach. The pad is constructed from within Dublin Gulch and up Ann Gulch side valley. The HLF is constructed in three phases:

- Phase 1 - all facilities to provide
    - 2 years of operation including:
    - confining embankment
    - in-heap pond
    - lining system
    - events pond No.1; and
    - surface runoff diversions and SCPs.
- Phase 2
    - Extension to the HLF (additional lined area), and
    - construction of events pond 2
- Phase 3 - Extension to the HLF (additional lined area)

The Phase 1 elements, shown in Figure 18-8, are described in more detail below. The ultimate extent of the HLF is shown on Figure 18-1, above.

CONFINING EMBANKMENT

In order to provide a satisfactory initial operational area to confine the heap leach pad and in-heap storage pond, an embankment is constructed at the base of the facility in the Dublin Gulch valley. The embankment will be 50 m high, with a width at top of the embankment of 560 m and a total fill volume of 2.2 million m^3. It will be constructed from selected durable waste rock from the mining process, placed on a suitable foundation, with a filter zone on the upstream face to provide a transition to the sub-grade of the liner.

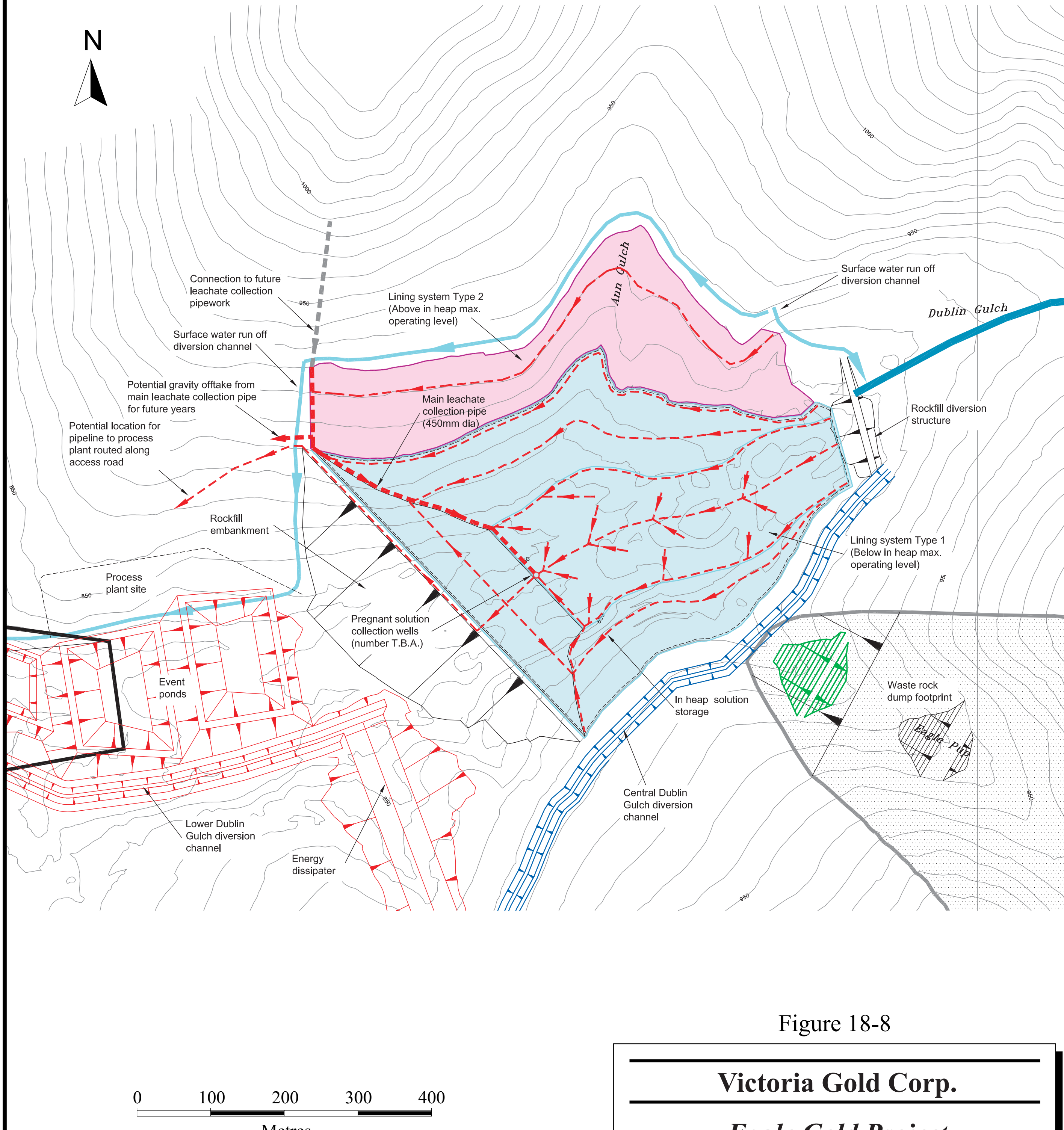

Figure 18-8

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Initial Pad and Dublin Gulch Diversion

April 2010

Source: Scott Wilson, KCA, 2010.

LINING SYSTEM

The heap leach is provided with an engineered lining system to prevent loss of solution and contamination of groundwater. The final lining system covers approximately 87 ha, and consists of a multiple composite PVC liner system, with dual leak detection, and a leachate recovery and collection systems to convey solution to the extraction well. The HLF liner system design provides a double composite liner in the upslope area of the pad (above the in-heap pond maximum operating level), and a triple liner in the in-heap storage pond area.

The liner system in the heap leach pad upslope area comprises the following elements from top to bottom:

- A cushion layer of 1 m thick ore, with leachate collection and removal system (LCRS) pipework

- Primary composite liner system comprising:
    - Primary 1.0 mm PVC geomembrane liner;
    - 300 mm thick compacted silt;
    - Geotextile separator;

- Primary Leak Detection and Recovery System (LDRS) comprising 300 mm thick fine gravel to coarse sand with pipes. On steep slopes, this is replaced with geonet;

- Secondary composite liner comprising:
    - Secondary 0.75 mm PVC geomembrane liner;
    - A 300 mm thick compacted silt layer.

Potential leakage through the primary liner into the LDRS in the upslope pad area is minimized by provision of a closely spaced network of leachate collection interceptors. These drains effectively reduce the hydraulic head over the liner.

In order to achieve compliance with the Nevada permitting guidelines (used here as a best practices reference point), with respect to liner leakage in the in-heap storage pond area, an additional liner element is required above the primary composite liner. This additional element comprises an upper 0.75 mm PVC geomembrane over an upper LDRS gravel layer. This upper liner serves to minimize the hydraulic head on the primary composite liner and therefore reduce the potential leakage rates into the primary LDRS.

By using a double composite liner in the upslope section and triple liner in the storage section of the pad, leakage into the LDRS will be below the limiting rates stipulated in the Nevada guidelines, and any subsequent leakage out of the system into the ground will be negligible.

IN-HEAP POND

Solution storage capacity for normal operations of 435,000 m^3 is provided with an in-heap pond, which consists of storing the solution within the pore space of the ore. This allows operation in the cold winter and spring climate conditions. As the heap is raised and the catchment area increases, additional storage is required for extreme rainfall events. Provision of external storage for this in an event pond is more economical.

EVENTS PONDS

Two events ponds are located downstream of the HLF and process plant to allow gravity drainage. The events ponds have a total storage volume of 200,000 m^3, and cater for excess solution in storm events from the HLF and plant drain-downs. As the in-heap capacity is significant, an event pond is not required in Years 1 and 2. However, a pond is provided at start-up to act as a temporary SCP, and to provide water storage for start-up.

The events ponds are double-lined and incorporate a geonet separation layer. The liner system to the events ponds comprises the following elements from top to bottom:

- Primary 2.0 mm thick HDPE geomembrane liner
- Primary LDRS geonet layer
- Secondary 1.0 mm thick HDPE geomembrane liner, and
- 300 mm thick compacted silt.

SURFACE RUNOFF DIVERSIONS AND SEDIMENT CONTROL PONDS

Control of surface water runoff and sediment is achieved with construction of runoff diversions around the HLF and sediment control features. A permanent SCP is located at the downstream extent of the HLF and events ponds infrastructure. The SCP has a volume of 36,000 m^3, and is sized to accommodate run-off events during construction and operations. Temporary use is also made of one of the events ponds, providing

100,000 m^3 of storage for sediment control during constructing of the Dublin Gulch Diversion.

LEAK DETECTION AND RECOVERY SYSTEMS

The performance of the lining system, as measured in terms of preventing loss of solution into the ground, is assessed by monitoring leak detection drains constructed below the liners. Separate leak detection and recovery systems (LDRS) are installed below each liner, and all collected solution is returned to the heap.

The LDRS consists of a series of 100 mm diameter pipes, within a 300 mm thick layer of 20 mm gravel, feeding to a 200 mm diameter collector pipe. Any leakage reporting to the drains flows to a sump below the in-heap pond, from where it is pumped back to the heap.

GROUNDWATER DRAINAGE

A groundwater drainage system is installed beneath the lowest liner of the HLF to prevent uplift pressures developing beneath the liner. The drainage system comprises a network of pipes placed in gravel-filled trenches and wrapped in geotextile. The pipe network comprises 100 mm diameter slotted corrugated polyethylene pipes (CPP) pipes, in a 300 mm x 300 mm gravel-filled trench, at a spacing of 25 m, feeding 200 mm diameter HDPE un-perforated collector pipes at 200 mm centres, in a 1200 mm x 1200 mm gravel-filled trench. In the base of the HLF, beneath the in-heap pond, the 200 mm pipes feed into a 300 mm diameter HDPE pipe. The 300 mm pipe requires a gravel-filled trench with cross-sectional area of 12 m^2, to convey the post-closure flow from the heap.

Monitoring of flow and quality will be undertaken on a regular basis. Water that meets the effluent standards will be released via a pipeline to the SCP. If the water does not meet the required standards, it will be pumped to the events pond for treatment or recycling. For this purpose, a sump is provided at the embankment toe, with valves to isolate flow.

DUBLIN GULCH DIVERSION

The diversion of the Dublin Gulch waterway is designed to convey stream flow safely past the HLF and return it back to the current course, approximately 1,500 m downstream of the inlet. The diversion comprises the following elements, as shown in Figure 18-8, above:

- An upstream structure that intercepts all Dublin Gulch stream flow and directs flow into a diversion channel
- A 900 m long diversion channel ("the upper diversion"), 3 m deep, with a slope of 1:100 leading to Stuttle Gulch
- Channelization of the Stuttle Gulch stream flow, with additional energy dissipation and erosion protection measures
- An enlarged and re-routed channel diversion ("the lower diversion") around the Event Ponds and Polishing Ponds
- A reconnection of the river flow into the current course of Dublin Gulch

Guidelines for diversions require design for a 1:200 year storm event, however, the diversion remains post-closure and therefore a design to the Probable Maximum Flood (PMF) is appropriate. Consequently the diversion is designed for a peak flow based on the PMF of 105 m^3/s.

The inlet consists of an embankment 12 m high designed to intercept all surface flows and the majority of sub-surface flows. The embankment consists of rock fill with a filter zone on the upstream face to provide a transition to the sub-grade of an HDPE liner. Placer tailings and alluvial material in the valley floor are removed, and an impervious zone barrier is created to direct sub-surface flows into the diversion. The HDPE liner is provided with damage protection measures grading from gravel back to rockfill.

From the upstream diversion structure, the 900 m long diversion runs nearly parallel to the contour at a slope of 1:100 to Stuttle Gulch. The construction of the upper diversion consists of earth-fill, HDPE liner and rock-fill erosion protection. The up-slope cut surfaces will be provided with erosion protection measures and flow from the disturbed surfaces will be channeled through an SCP, until runoff meets the suspended solids requirements.

The flow from the upper diversion is then directed into Stuttle Gulch, through energy dissipation and erosion protection measures to handle the higher flow. These measures comprise large size rock-fill placed on a gravel bed on a heavy duty geotextile. Stability of the slope, keying the structure into the slope, permafrost extents, and optimizing the design for environmental considerations, are issues to be reviewed further in the feasibility design.

The flow from the Stuttle Gulch energy dissipation channel re-enters the lower diversion of the Dublin Gulch valley floor at a channel inlet, which is an enlarged section of the lower diversion, provided with erosion protection measures. The stream at this point is then designed to be part of the Dublin Gulch fish habitat and detailed design will need to take this into account. The invert of the channel is presumed to be on competent bedrock and will intercept and drain the groundwater beneath the events ponds. Lining is not considered necessary, however, erosion protection to the banks is provided. Detailed investigations of the geotechnical and groundwater conditions along the route of the diversion will be undertaken as part of the detailed engineering.

MINERAL PROCESSING

The proposed process of three-stage crushing to a P_{80} of 5 mm, followed by heap leach gold extraction, and recovery of gold from solution in a carbon ADR plant, is described in more detail in Section 16 above. Mine life average gold recovery, across all material types, is estimated to be 72%.

INFRASTRUCTURE

ACCESS

Employees from Mayo and the surrounding communities will be transported to Mayo by a transfer van service. Employees from outside the local area will be flown in from Whitehorse to Mayo. All employees will be bussed the remaining distance to the mine site. Main access to the site from the airstrip at Mayo is by approximately 85 km of existing paved and gravel roads. All but the last 25 km is government maintained road. The last 25 km of road will be single lane, radio controlled gravel surface road with pullouts appropriately spaced. The access road requires minor alignment and water management upgrades, and will be maintained by the mine. Access to Mayo for freight and other deliveries is by government maintained public roads.

SITE ROADS

Numerous light vehicle roads and trails currently exist on the property from previous exploration activities, and will be upgraded as required for light vehicle access. A mine haul road for haul trucks will be constructed to provide access from the borrow pits within the final pit limits to the embankment rockfill site for construction purposes.

POWER

Power for the Project is assumed to be available from the Yukon Energy transmission grid by 2013. Yukon Energy is currently in the process of upgrading power generation capacity at Mayo (the Mayo Hydro Enhancement Project), and connecting the north and south Yukon transmission grids (the Carmacks-Stewart Transmission Project Stage 2). Funding agreements specify that both projects must be completed by March 31 2012. Yukon Energy reports that both projects are on schedule to finish in 2011.

Power to the site will be by a new 45 km transmission line connecting to the Yukon Energy grid, routed along the access road. The 69 kV transmission line will feed a main substation on site. Power will be distributed at 13.2 kV via two sets of overhead lines. The first set of overhead lines will provide power to primary crushing, secondary crushing, conveying and HPGR. The second set of overhead lines will provide power to the ADR facility and heap leach operation. Each area will require an electrical room housing high voltage (HV) and low voltage equipment including indoor 13.2 kV-4.16 kV and 13.2 kV-575 V step-down unit substations.

In the event of a power failure, three emergency diesel generation sets will be provided. These units will also be used to provide temporary power during construction activities in 2012. Each unit will be rated at 1,500 kW, 575 V with a 575 V/13.2 kV transformer and will supply power to the following items:

- Barren Solution Pumps
- Pregnant Solution Pumps
- Carbon Stripping Circuit
- Camp and Buildings

- Fire Water and Fresh Water Distribution Systems
- Fire and other alarm systems
- Security systems

The average seasonal forecast operating loads for the operation are estimated to be 11 MW.

FUEL STORAGE AND DISTRIBUTION

Diesel fuel will be stored in two main fuel storage tanks, of 750,000 L capacity each, within a bermed containment area.

A 100,000 L diesel fuel storage tank will be installed adjacent to the ADR building. This horizontal tank will be approximately 3.3 m diameter and 11.9 m long and will be located within a concrete bermed area. Fuelling equipment will include a receiving pump and strainer and delivery pumps and filters for the recovery area equipment and for the solution heating boiler.

A 10,000 L steel tank will also be available for storage of waste lubricating oil. It is anticipated that this oil, collected from the mine equipment, will be burned along with diesel fuel in the solution heating boiler. Oil filters and a blending system to facilitate in-line mixing of lube and diesel oil are included.

PROCESS WATER

Under average conditions, up to 300,000 m^3/year of raw water will be required as make-up for the heap leaching process. This amount will increase to a maximum of approximately 450,000 m^3/year in an extreme dry year.

Fresh make-up water for use in the process will be supplied from surface runoff collection, wells or from pit dewatering wells. The water will be pumped to the heated and insulated raw/fire water tank located near at the 940 m elevation near the crushing access road.

Fire water will flow by gravity through a main to the process facilities. Fire protection to the site and facilities will be provided by a standpipe outside, and two 100 mm diameter hose connections inside all of the heated buildings. The process offices, laboratory and shop / warehouse will also be fitted with sprinkler systems. Portable fire extinguishers will be provided in all buildings.

Raw water for the process will flow by gravity to the ADR processing area.

Small raw water pumps will pump process water to the crushing areas.

WATER TREATMENT

A water treatment plant will be located at the SCPs, near the end of the Dublin Gulch diversion. The plant is designed to treat Project water flows, when necessary, for arsenic, pH, sediment, and other trace metals of concern. Plant processes include chemical conditioning (coagulation/flocculation), sedimentation, filtration, and ion exchange.

- Coagulation/Flocculation – As will be removed through ferric based coagulation (e.g., $FeCl_3$) as ferric arsenate.

- pH Adjustment – The optimal pH values for the lowest Al, As, and Fe removal are different, including 6-6.5 for Al, 3-7 for As, and 6-7 for Fe. It is expected that the pH value of the wastewater would be reduced after coagulation (e.g., ferric based coagulant would drop the wastewater pH). Therefore, the wastewater pH will to be adjusted with alkali material (e.g., lime solution) to an optimal pH value that would maximize the removal rates of the three identified parameters of concerns.

- Sedimentation – the chemically conditioned wastewater will flow into a sedimentation tank for precipitation. Sludge will be removed regularly to maintain a good quality of effluent.

- Filtration – The clarified effluent might still exceed the projected effluent requirements, due to the fact that not all the precipitates could be settled out in the clarifier. Therefore, a filtration process is proposed to further remove metals associated with particulates.

- Ion Exchange – An ion exchange unit will be provided to remove soluble metals of concerns (e.g., As).

Water treatment requirements have yet to be confirmed by testwork (currently in progress). In Scott Wilson RPA's opinion, inclusion of a water treatment plant during initial construction and operation, when preliminary water balances indicate no expected requirement for discharge, is conservative.

WATER AND SEWAGE FACILITIES

Potable water will be supplied by an on-site facility that will treat ground water as required. Sewage will be treated on-site and solid waste will be removed to a local landfill. The fire water tank will be supplied by excess water from SCPs in addition to fresh water make-up if needed, and the water truck will be equipped with a fire monitor as required.

The SCP located in Dublin Gulch near the process facilities, is the primary sediment control and surface water management pond for the site.

BUILDINGS

The main site includes a 200 person camp and administration building, process plant, modular assay lab, mine truck shop, warehouse, fuel tank farm, and laydown area. The camp consists of six dorm units, which include washroom facilities; a kitchen and dining area; a recreation complex; and laundry facilities. Administration and mine offices are integrated into the camp complex, as are dry facilities.

SECURITY

A full-time security staff will be employed at site to monitor access through a main control gate, and maintain security on site. Measures for security of doré gold will include closed circuit monitoring, restricted access to the gold refining facilities, and safe transfer of refined doré bars to the designated security firm's trucks. Gold will be trucked off-site by a recognized security trucking firm.

FIRST AID

The site will have a first aid room and an ambulance, for transfer of personnel to Mayo as required. A helicopter landing area will be designated on site for emergency medical evacuation. Security staff will be trained as first aid attendants, and a full-time contract nurse will be on site at all times.

MARKETS

Gold, the principal commodity at the Eagle Gold Project, is freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. Scott Wilson RPA used a gold price of US$900 per ounce for the Base Case.

CONTRACTS

Scott Wilson RPA is not aware of any contracts to which Victoria Gold is currently a party that are relevant to the economic results presented in this Report.

The PFS mine life scenario includes provision of the following services by contractor:

- Construction
- Camp services, including meals and housekeeping
- Medical services on site
- Explosives supply and blasthole loading
- Gold transportation & handling
- Access road maintenance
- Personnel transportation to and from the site
- Reclamation and closure

ENVIRONMENTAL CONSIDERATIONS

PERMITTING

The Yukon is considered to have a generally favourable regulatory environment for mining activity, in part due to completion of First Nation land claim agreements, and in part to the devolution of regulatory responsibilities related to mining activity to the Yukon Government. Environmental and socio-economic assessment of development activities now occurs under the Yukon Environmental and Socio-economic Assessment Act (YESAA), which provides for a single assessment process. A number of mine development projects have recently been assessed and permitted in the Yukon, or are currently proceeding through the approval processes. They include the Carmacks Copper Project (heap leach), the Bellekeno Mine, and the Mactung Mine.

The Eagle Gold Project currently has an approved, Class III, Operating Plan under the Yukon Quartz Mining Act for its advanced exploration activities at the site as well as its camp operations. The current Operating Plan is in effect until 2012.

The regulatory approval process for major hard rock mines in the Yukon occurs in two stages: 1) an assessment under the YESAA, and 2) the receipt of territorial and/or federal permits, authorizations, and licences. The assessment and permit approval processes can occur concurrently, however, permits and licences cannot be issued until after the YESAA assessment is complete, and a positive assessment decision issued. YESAA includes regulated timelines and the overall project approval timeline can be estimated at 24-30 months. Table 18-8 provides a list of key environmental permits, licences, and authorizations required for the Project.

Consultation with First Nations is a formal requirement under the YESAA, with the underlying objective to ensure First Nation interests are considered and attempts are made to address and / or accommodate issues and concerns with respect to a proposed project. Consultation with the First Nation of Na-cho Nyak Dun and local communities is required under YESAA in developing the Eagle Gold Project Proposal.

TABLE 18-8 PERMITS, LICENCES AND AUTHORIZATIONS
Victoria Gold Corp. – Eagle Gold Project

Permit/Authorization	Enabling Legislation	Responsible Agency	Project Activity
Quartz Mining Licence (QML)	*Quartz Mining Act*	Energy Mines and Resources, Yukon Government	Mine construction, operation, decommissioning and closure.
Water Licence – Type A	*Waters Act*, Waters Regulation	Yukon Water Board	Water use and deposit of waste.
Section 35(2) Authorization	*Fisheries Act*	Department of Fisheries and Oceans Canada	Stream works and works related to water and fish habitat.
Section 5(2) Approval	*Navigable Water Protection Act*	Transport Canada	Working in relation to navigable water crossings.
Land Use Permit Quarry Permit or Quarry Lease	*Territorial Lands (Yukon) Act*, Land Use Regulations	Energy Mines and Resources, Yukon Government	Land use related works not covered by the QML (such as access road work or to utilize borrow pit resources).
Commercial Timber Permit	*Territorial Lands (Yukon) Act*, Timber Regulations	Energy Mines and Resources, Yukon Government	Cutting timber - clearing and grubbing.
Air Emissions Permit	*Environment Act*, Air Emission Regulations	Environment Yukon, Yukon Government	Potential emissions (diesel generator, fuel use, or quarry use over 4ha).
Burning Permit	*Forest Protection Act*, Forest Protection Regulation	Department of Community Services, Yukon Government	Burning (waste from clearing and grubbing; may be seasonal requirements).
Archaeological Sites Permit	*Yukon Historic Resources Act*	Department of Tourism and Culture, Yukon Government	Search for and research at archaeological / paleontological sites.
Storage Tank Systems Permit	*Environment Act*, Storage Tank Regulation	Department of Community Services, Yukon Government	Storage and handling of petroleum products (fuel); may be covered in the QML.
Solid Waste Disposal Permit	*Environment Act*, Solid Waste Regulation	Department of Environment, Yukon Government	Waste disposal for construction and for site facilities.
Special Waste Permit	*Environment Act*, Special Waste Regulations	Department of Environment, Environmental Programs, Yukon Government	Special waste handling, disposal, storage, generation.
Permit/certificate for transport of dangerous goods	*Dangerous Goods Transport Act*	Highways and Public Works, Yukon Government	For waste storage, may also need permit for Bill of Lading, driver training, placarding; federal transport legislation may also apply.
Temporary Magazine Licence, Factory Licence, ANFO permit	*Explosives Act*	Natural Resources Canada, Explosives Regulatory Division and Minerals and Metals Sector	Manufacture and storage of explosives.
Registered or Certified Aerodrome / Helidrome	*Aeronautics Act*, Canadian Aviation Regulations	Transport Canada, Civil Aviation, Regulatory Affairs	Airstrip use or helipad. May be needed for use of established airstrips.
Permit to install a sewage disposal system	*Public Health and Safety Act*, Sewage Disposal System Regulation	Department of Health and Social Services, Yukon Government	On-site sewage disposal system.
Compliance with Public Health Regulations	*Yukon Public Health and Safety Act*, Regulations Respecting Public Health	Department of Health and Social Services, Yukon Government	Public health, sanitation, facilities. Permit to operate a food premise is also needed.

BASELINE STUDIES

A large amount of environmental baseline information has been collected throughout the Project area to compile the data set required to support the submission of a Project Proposal under the YESAA. Past studies combined with work recently completed has produced a comprehensive environmental baseline database that will be used to assess the existing environmental resources and conditions potentially affected by the Project.

Waste and ore geochemistry evaluations have indicated that the waste and ore associated with the Eagle Gold Project is likely to be non-acid generating. Kinetic testing indicates that concentrations of some metals from the waste rock and spent ore are likely to be somewhat elevated with respect to receiving water quality guidelines. Further work is required to develop water quality predictions for the waste rock, open pit, and heap leach pad. Some measures to control seepage and run-off water quality may be required. Further work is in progress to evaluate seasonal and annual variations during operations, drain down, and final closure.

Development, operation, and decommissioning of the mine will affect a range of environmental and socio-economic components. The project definition report will provide mitigation strategies to minimize or eliminate adverse environmental or socio-economic effects of the project. These strategies will form the basis for the development of environmental management plans for the construction, operation, and decommissioning phases of the project. A component of the environmental management plans will be an environmental effects monitoring program, which will be developed to monitor the effectiveness of the mitigation and management strategies, and provide a basis for amending these strategies, as necessary, to achieve the desired outcome. Both the management plans and the monitoring program will be outlined in the project definition report, and will be refined during the permitting process.

In Scott Wilson RPA's opinion, environmental considerations are typical of open pit, heap leach operations, and are being addressed in a manner that is reasonable and appropriate for the stage of the Project.

RECLAMATION AND CLOSURE

Security, reclamation, and closure for major mine projects in the Yukon is regulated under the Quartz Mining Act, the Yukon Mine Reclamation and Closure Policy (YMRCP), and the Waters Act for specific water-related issues. Financial security for major mines in the Yukon is held under a Quartz Mining Licence, or Water Licence, or a combination thereof. The amount and form of security is determined pursuant to the Quartz Mining Act or the Yukon Waters Act. A comprehensive Reclamation and Closure Plan must be prepared and submitted for approval as part of the Quartz Mine Licence application and be updated periodically throughout the operating mine life, and at a minimum, every five years. The YMRCP requires annual reporting and post closure monitoring. The mine owner must provide financial security for the full outstanding liability, based on the cost to reclaim and close the mine site in its current status, in accordance with the approved reclamation and closure plan. The outstanding liability is re-assessed periodically, or at minimum, every two years to reflect the impact of operations and progressive reclamation.

TAXES

Scott Wilson RPA has relied on Victoria Gold for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Project.

The cash flow presented in this Report has been completed on a pre-tax basis. It includes a royalty of 1% of gross returns, as noted in Section 4 above.

CAPITAL COST ESTIMATE

The estimated cost to design, construct, install and commission the Project operation and facilities described in the PFS is $281 million. This amount includes the direct field costs of executing the Project, plus the Owner's and indirect costs associated with design, construction and commissioning. Cost estimates are based on the PFS design, and are considered to have an accuracy of +/-25%. The capital cost estimate is summarized below in Table 18-9. All costs are expressed in fourth quarter 2009 Canadian dollars, with no allowance for interest or financing during construction.

TABLE 18-9 CAPITAL COST SUMMARY

Victoria Gold Corp. – Eagle Gold Project

Major Area	Construction Cost ($'000s)	Ongoing Cost ($'000s)	Total Cost ($'000s)
Mining	34,950	5,719	40,669
Crushing & Conveying	68,009	5,126	73,134
Heap Leach Facility	26,258	24,747	51,005
Process Plant	16,075	1,177	17,252
Infrastructure	42,570	2,939	45,509
Subtotal Direct Capital Cost	**187,862**	**39,708**	**227,570**
Indirects	55,306	-	55,306
Contingency	38,214	-	38,214
Closure & Reclamation	105	15,732	15,836
Total Project Capital Cost	**281,486**	**55,439**	**336,925**

ESTIMATION METHODOLOGY

The capital cost estimate is based on the following project data:

- Design criteria
- Flowsheets
- General arrangement drawings
- Single line electrical drawings
- Equipment lists
- Supplemental sketches as required
- Budget quotations from vendors
- Regional climatic and hydrological data
- Local geotechnical investigations
- In-house database and operating experience
- Preliminary contractor quotations based on preliminary drawings and/or quantities
- Supplemental information provided by Victoria Gold and other consultants

The following is excluded from the capital cost estimate:

- Project financing and interest charges
- Owner's costs before 2012

- Land acquisition, leases rights of way and water rights
- Escalation during construction
- Permitting costs
- Environmental impact studies
- Any additional civil, concrete work due to the adverse soil condition and location
- Taxes
- Import duties and custom fees
- Cost of geotechnical investigation
- Working capital
- Sunk costs
- Exploration drilling
- Costs of fluctuations in currency exchanges
- Project application and approval expenses
- Future expansion
- Relocation of any facilities, if required
- Purchase of existing facilities and buildings

MINING

Scott Wilson RPA estimated capital costs for mining. Life of mine mining capital costs total $40.7 million, of which $35.0 million is spent during construction, plus $5.7 million spent on ongoing capital costs.

Construction capital consists largely of mobile equipment purchases of $30.6 million, followed by earthworks of $4.4 million, of which the majority is for road construction and the heap embankment rock fill. The remainder consists of various small items, including safety supplies, radios, office equipment, survey equipment, pit dewatering equipment, etc.

Mobile equipment costs are based on manufacturer's price lists for Q4 2009. Earthworks will be carried out initially by a contractor, and in 2013, by the Owner's mining crew; capital costs have been estimated on the same basis as for operating costs.

Ongoing capital is spent on incremental major equipment and light vehicle replacements.

CRUSHING & CONVEYING

KCA estimated capital costs for the crushing and conveying systems. Each area, such as primary crushing, leach, etc. in Table 18-10 below, is separated into the following categories where applicable: earthworks, civils, structural steel, platework, mechanical equipment, piping, electrical, and instrumentation. Costs for buildings housing each area are included.

TABLE 18-10 CRUSHING AND CONVEYING CAPITAL COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	Cost ($'000s)
Primary Crushing	8,922
Secondary Crushing	19,464
High Pressure Grinding Rolls	22,191
Stacking & Leaching	22,557
Total	**73,134**

Major earthwork quantities were estimated based on the preliminary site design. Unit rates for the major earthworks for the Project were based on contractor budget quotes.

For civils costs, concrete quantities were estimated from takeoffs based on previous quantities from similar equipment installations, on major equipment weights and on slab areas. Installed rates for concrete are based on a contractor budget quotation, and include the appropriate quantities of rebar. These costs include footing excavation, formwork, and all other tasks necessary to supply and place concrete.

Structural steel requirements for the various major equipment items and buildings were estimated from takeoffs based on previous quantities from similar equipment installations. Unit costs for steel, including installation labor and equipment requirements, are based on supplier budget quotations. Unit costs vary depending on the type and size of structural steel to be installed.

Platework includes the costs for tanks, bins and chutes. The quantity of platework was calculated based on appropriate plate thicknesses and sizes for each individual item. Platework weights for tanks and chutes were calculated from relevant drawings and sketches, with allowances made for any necessary stiffeners, weirs, launders, etc. The unit price includes plate purchasing, detailing, fabrication and installation. Installed costs for platework are based on supplier budget quotations.

Mechanical equipment was itemized and priced as per the project equipment list. Major process equipment was priced as new equipment FOB supplier. All motors were included with equipment costs where applicable. Supplier budget quotations were obtained for all major equipment items. Minor equipment items were estimated from relevant and recent KCA projects.

In order to estimate costs for the electrical, piping & valves, and instrumentation categories, KCA used percentages of mechanical equipment supply costs.

Ongoing capital costs in this area involve extensions of the stacking system as the heap increases in size.

HEAP LEACH FACILITY

Scott Wilson RPA estimated capital costs for the heap leach facility, based on quantities derived from the heap leach pad design, and contractor budget quotations combined with database unit rates from similar projects. The HLF capital costs by stage are shown in Table 18-11.

TABLE 18-11 HEAP LEACH FACILITY CAPITAL COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	Stage 1 – 2013 ($'000s)	Stage 2 – 2015 ($'000s)	Stage 3 – 2017 ($'000s)
Heap Leach Pad	18,622	11,741	10,494
Dublin Gulch Diversion	6,083		
Events Ponds	1,553	1,558	
Polishing Ponds		953	
Total	**26,258**	**14,252**	**10,494**

PROCESS PLANT

KCA estimated capital costs for the Process Plant, using the categories and methods described above, under Crushing and Conveying. A breakdown of Process Plant capital costs by area is presented in Table 18-12.

TABLE 18-12 PROCESS PLANT CAPITAL COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	Cost ($'000s)
Carbon Adsorption	2,853
Carbon Desorption & Regeneration	4,824
Refinery	1,738
Reagents	1,694
Cyanide Destruction	809
Facilities	2,110
Process Buildings	2,107
Mobile Equipment	923
Fuel	194
Total	**17,252**

Cyanide Destruction has been included in 2016, as it is not anticipated to be required before then. Facilities consist largely of a modular site assay lab, and associated storage. Fuel covers a small day tank for use by process personnel.

INFRASTRUCTURE

Infrastructure capital costs have been estimated by Scott Wilson RPA and KCA. A breakdown by area is shown in Table 18-13.

TABLE 18-13 INFRASTRUCTURE CAPITAL COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	Cost ($'000s)
Roads	12,755
Power	12,281
Water & Sewage	4,684
Buildings	14,973
Other	815
Total	**45,509**

Road costs include upgrades to the main access road, however, the majority is for site roads, such as the route for large haul trucks to transport mine waste to the heap leach embankment, for use as rockfill.

Power costs include a new 69 kV, 45 km transmission line to site, the main site substation, site electrical distribution, and emergency diesel power generation (back-up power).

Water and sewage costs include potable water wells, tankage, sewage treatment, fire water pumps, and a water treatment facility.

Building costs include the 200-man camp, truck shop, warehouse, cold storage facilities, main diesel storage tanks, and fuel distribution system.

Other costs include site communications hardware, lighting, and service vehicles (fire truck, ambulance, vacuum truck).

INDIRECTS

Indirect capital costs have been estimated by Scott Wilson RPA and KCA. A breakdown by area is shown in Table 18-14.

TABLE 18-14 INDIRECT CAPITAL COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	Cost ($’000s)
EPCM	21,219
Freight	6,174
Spares & First Fills	2,772
Construction Indirects	11,974
Mine/Maintenance Crew	6,231
Owner’s Costs	5,791
Miscellaneous	1,145
Total	**55,306**

Engineering, Procurement, and Construction Management (EPCM) was estimated at 12% of direct capital costs, based on experience on similar projects.

Freight costs have been estimated as separate items, and are factored from the equipment cost using an in-house experience factor. Included are all costs related to export packing, freight to port, marshalling, demurrage, and land freight to site. Shipping cost is based on an allowance of 10% of the mechanical equipment supply cost.

Process and infrastructure spare parts costs are based on an allowance of 4% of the process and mobile equipment purchase value. For piping and valves, electrical, and instrumentation, spare parts costs are estimated based on 4% of the estimated supply costs for these items. Spares for mine mobile equipment were included in the manufacturer quotes. Initial fills is an allowance for the acquisition of the initial supply of reagents and other operational consumables, based on a one-month supply.

Construction Indirects cover contractor's field distributable costs that are included in the capital cost estimate, but not in built-up labour rate costs, including the following:

- Contractor's mobilization and demobilization
- Use of construction equipment
- Temporary power supply
- Temporary water supply
- Warehouse and laydown costs
- Temporary toilets
- Temporary communications
- First aid personnel and supplies
- Ongoing and final cleanup
- Yard maintenance
- Janitorial services
- Site safety personnel and training
- Material testing
- Construction accommodation and catering
- Construction freight

For the process and infrastructure area, the indirect costs are estimated based on 25% of the install costs. For mining and heap leach construction activities, costs were estimated for power (diesel generation in 2012, hydroelectric in 2013), fuel, accommodation, and catering.

The mine/maintenance crew cost includes labour, camp costs, and transportation for the operations crew completing capital earthworks in 2013.

Owner's costs include salary, travel, and office expenses for the Project management team, and a support office in Whitehorse.

Miscellaneous costs include insurance and commissioning.

CONTINGENCY

Contingency accounts for unforeseen costs within the Project scope. Percentages were assigned by area and category, including indirect capital costs. Contingency ranged from 10% on well-defined areas (quoted equipment) to 30% on less-defined areas (earthworks, for example). Totalling $38.2 million, contingency is 16% of direct plus indirect capital costs.

CLOSURE & RECLAMATION

Closure and reclamation costs include environmental bonding costs starting in 2013, and a lump sum of $15 million after operations are completed.

OPERATING COST ESTIMATE

Mine operating costs are estimated for year-round open pit mining at a rate of 26,000 tpd ore, ranging from 35,000 tpd to 60,000 tpd, including waste mining. Process operating costs are estimated for three-stage crushing to a P_{80} of 5 mm, directly followed by stacking of ore, for 250 days per year. During the winter season, approximately 100 days of the year, ore will be crushed, then conveyed to the heap and stockpiled. During the summer season, this stockpiled material will be reclaimed and transferred to the pad stacking system. The ADR gold recovery plant will operate 350 days per year. General and Administrative (G&A) costs are estimated for a camp operation, with employees working a shift rotation, based out of Whitehorse.

Total operating costs average $91 million per year. Operating unit costs are summarized in Table 18-15.

TABLE 18-15 OPERATING COST SUMMARY
Victoria Gold Corp. – Eagle Gold Project

Area	Cost
Mine	$1.88 per t moved
	$3.84 per t processed
Process	$5.05 per t processed
G&A	$1.50 per t processed
Total	**$10.38 per t processed**

PERSONNEL

The Project will employ a peak of 384 people during operations. Annual employee counts by area are noted in Table 18-16, below.

Mine department personnel are divided into three areas: mine operations, mine services, and mine maintenance.

Mine operations accounts for the direct mining operation and performance of drilling, explosives handling by contractor, blasting, load and haul, mine support, and mine ancillary services. Staffing levels will vary over the life-of-mine and are dependent upon scheduled production rates. Mine operations will employ up to 115 salaried and hourly employees, plus three on-site contract workers.

Mine services includes management and supervision of mine operations and mine maintenance functions, and technical support for the operation by mine engineering, geology, and survey services; and will employ up to 26 salaried and hourly employees.

Mine maintenance includes supervision and implementation of mine maintenance performed under direct mining; and will employ up to 66 salaried and hourly employees.

Process department personnel are split into two areas: processing and laboratory. The process area includes supervision, crushing, heap leach, recovery plant, and maintenance areas; and will employ up to 98 salaried and hourly employees. The laboratory area has up to 15 salaried and hourly employees.

G&A department personnel include management, accounting, warehouse, environmental monitoring, health and safety, training, security, site services, and a small expediting office in Whitehorse. On-site contract personnel for nursing, camp catering and housekeeping are also included. G&A functions will employ 46 salaried and hourly employees on site, plus four in the Whitehorse office, and 14 on-site contract employees.

TABLE 18-16 EMPLOYMENT BY YEAR
Victoria Gold Corp. – Eagle Gold Project

Department	2013	2014	2015	2016	2017	2018	2019	2020
Mine Operations	97	102	114	112	115	108	89	97
Mines Services	19	26	26	26	26	26	26	26
Mine Maintenance	53	58	66	66	66	62	54	53
Subtotal Mine	169	186	206	204	207	196	169	176
Processing	91	98	98	98	98	98	98	98
Laboratory	15	15	15	15	15	15	15	15
Subtotal Process	106	113	113	113	113	113	113	113
General & Administrative	64	64	64	64	64	64	64	64
Total Manpower	**339**	**363**	**383**	**381**	**384**	**373**	**339**	**353**

ROSTER

The majority of Project personnel are scheduled to work 12-hour shifts on a two weeks on, two weeks off rotation. Four work crews are required for 24-hour per day year-round coverage. A small number of process department personnel working on the piping crew for the heap leach are scheduled to work 12-hour shifts on a two weeks on, one week off rotation for nine months of the year only; this operation requires only three work crews and are only required during stacking operations, when new ore is put under leach.

Mine services employees will work a weekly rotation of ten hour days, four days on, three days off, for a 40 hour work week.

General and administrative employees will work a weekly rotation of ten hour days, four days on, three days off, for a 40 hour work week.

Labour rates for salaried and hourly personnel are based on current expected labour rates in the area and include all burden costs appropriate to each position and work rotation.

MINE OPERATING COSTS

Mine operating costs have been estimated from first principles based on equipment productivity, operating hours, and operating cost per hour. Cost of diesel fuel is C$0.90 per litre delivered to site, based on current quotations from local suppliers. Productivities are based on haul truck cycle times, loader productivity, and support equipment needs. Mine operating costs average $1.88 per tonne of material moved over the life-of-mine, with unit operating costs listed in Table 18-17, on a per tonne moved and processed basis.

TABLE 18-17 MINE OPERATING COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	$/tonne moved	$/tonne processed
Clear & Strip	0.01	0.03
Drill	0.16	0.32
Blast	0.32	0.65
Load	0.21	0.42
Haul	0.48	0.97
Mine Support & Ancillary	0.28	0.57
Mine Services & Maintenance	0.42	0.86
Total	**1.88**	**3.84**

Notes:
1. Totals may not add up due to rounding.

PROCESS OPERATING COSTS

Process operating costs have been estimated based upon the design and operating criteria presented in this Report, and on the following sources:

- Project metallurgical test work and process engineering
- Budgetary quotations from potential suppliers of project operating and maintenance supplies and materials
- KCA operating experience from mines of a similar nature
- Recent KCA project file data
- Advice from suppliers

The average process operating costs per tonne of ore processed for the production of gold and silver at the Eagle Gold Project are presented in Table 18-18. These costs exclude doré shipping.

TABLE 18-18 PROCESS OPERATING COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	$/tonne processed
Labour	1.05
Crushing	0.53
HPGR	0.67
Stacking	0.37
Heap Leach	0.54
Plant & Services	0.54
Reagents	1.35
Total	**5.05**

Power costs are included in each area, and are based on Yukon Energy's current industrial user rates.

G&A OPERATING COSTS

General and Administrative (G&A) costs are estimated for a camp operation, with employees working a shift rotation, based out of Whitehorse. G&A is broken down into five major cost centres as detailed in Table 18-19: labour; administration, including the Whitehorse office; health, safety, and environment; transportation; and site services.

TABLE 18-19 G&A OPERATING COSTS
Victoria Gold Corp. – Eagle Gold Project

Area	$/tonne processed
Labour	0.54
Admin	0.12
Health, Safety & Environment	0.06
Transportation	0.12
Site Services	0.66
Total	**1.50**

ECONOMIC ANALYSIS

A pre-tax Cash Flow Projection has been generated from the Life of Mine production schedule and capital and operating cost estimates, and is summarized in Table 18-20. A summary of the key criteria is provided below.

ECONOMIC CRITERIA

PRODUCTION

- Mineral Reserves of 66.1 Mt, at a grade of 0.82 g/t Au
- Waste mining of 72.9 Mt, for a total stripping ratio of 1.1:1, including 2.8 Mt of low-grade stockpile material
- Pre-production period of 20 months (January 2012 to August 2013)
- Mine life of eight years
- Open pit production of 9.1 Mtpa ore, 26,000 tpd
- Crushing to -5 mm year-round, heap leach pad loading 8.5 months per year, winter stockpiling

REVENUE

- Leach Plant recovery by material type, as indicated by testwork, averaging 71.7%.
- Gold production schedule reflecting leach time vs. recovery in testwork.
- Silver credit based on testwork showing doré bars contain 20% silver, 80% gold.
- Exchange rate C$1.00 = US$0.90.
- Metal prices: US$900 per ounce gold.
  US$13.00 per ounce silver.
- Net Smelter Return includes doré refining, and 1% NSR royalty.
- Revenue is recognized at the time of production.

COSTS

- Pre-production capital cost of C$281 million in 2012 and 2013, including a contingency of $38 million.
- Sustaining capital costs of C$55 million.
- Mine life capital totals C$337 million.

- Average operating costs over the mine life:

Mining	C$1.88 per tonne moved, or
	C$3.84 per tonne processed
Processing	C$5.05 per tonne processed
G&A	C$1.50 per tonne processed
Total	C$10.38 per tonne processed

TABLE 18-20 PRE-TAX CASH FLOW SUMMARY
Victoria Gold Corp. - Eagle Gold Project

	Units	Input	Year: Total/Avg.	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Physicals													
Mine Operating Days	days			200	308	365	365	365	365	365	365	308	0
Stacking Operating Days	days			0	116	263	263	263	263	263	263	263	0
Leach Operating Days	days			0	168	365	365	365	365	365	365	365	120
Annual Ore Tonnage for Leachi	'000 tonnes		66,141		2,000	9,100	9,100	9,100	9,100	9,100	9,100	9,541	0
Head Grade	g/t Au		0.823		1.019	0.863	0.983	0.855	0.861	0.753	0.731	0.682	0.000
Annual ROM Ore Tonnage	'000 tonnes		66,141		3,300	9,100	9,100	9,100	9,100	9,100	9,100	8,241	0
Head Grade	g/t Au		0.823		1.019	0.837	1.007	0.829	0.867	0.734	0.731	0.674	0.000
Low-Grade Stockpile Material	'000 tonnes		2,858		190	1,029	483	623	529	4	0	0	0
Head Grade	g/t Au		0.387		0.388	0.385	0.380	0.399	0.382	0.371	0.000	0.000	0.000
Waste Tonnes moved year	'000 tonnes		65,616	587	6,848	5,647	13,739	8,752	12,885	9,476	3,978	3,704	0
Total Material moved per year	'000 tonnes		134,615	587	10,338	15,777	23,322	18,475	22,514	18,579	13,078	11,944	0
	tpd			2,933	33,564	43,224	63,896	50,618	61,683	50,902	35,831	38,781	0
Strip Ratio			1.04		2.13	0.73	1.56	1.03	1.47	1.04	0.44	0.45	0.00
Contained Gold	ounces		1,750,957		65,523	252,459	287,482	250,088	251,998	220,363	213,887	209,156	0
Average Recovery	%		71.7%		74%	72%	73%	71%	72%	71%	70%	71%	0%
Recovered Gold	ounces		1,253,892		39,714	178,868	209,262	179,329	180,206	157,387	149,920	132,001	27,205
Revenue													
Gold Price	US$/oz Au	900	$900		900	900	900	900	900	900	900	900	900
Silver Price	US$/oz Ag	13.00	$13		13	13	13	13	13	13	13	13	13
Exchange Rate	US$=C$1.00	0.90	0.90		0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Gold Revenue	C$ '000		$1,253,892		39,714	178,868	209,262	179,329	180,206	157,387	149,920	132,001	27,205
Silver Revenue	C$ '000		$4,528		143	646	756	648	651	568	541	477	98
Gross Revenue	C$ '000		$1,258,420		39,857	179,514	210,018	179,976	180,857	157,955	150,461	132,478	27,304
Refining cost - per oz Gold	C$ '000	5.00	$6,269		199	894	1,046	897	901	787	750	660	136
Refining cost - per oz Silver	C$ '000	0.40	$125		4	18	21	18	18	16	15	13	3
NSR Royalty	C$ '000	1%	$12,522		397	1,786	2,090	1,791	1,800	1,572	1,497	1,318	272
Net Revenue	C$ '000		$1,239,504		39,258	176,816	206,861	177,271	178,138	155,581	148,200	130,487	26,893

TABLE 18-20 PRE-TAX CASH FLOW SUMMARY
Victoria Gold Corp. - Eagle Gold Project

	Units	Input	Year: Total/Avg.	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Operating Costs													
Mining	C$/t moved		1.88		1.90	2.07	1.68	1.95	1.69	1.87	2.11	2.13	
Mining	C$/t ore		3.84		9.81	3.59	4.31	3.97	4.19	3.81	3.04	2.67	
Processing	C$/t		5.05		7.63	4.92	4.92	4.92	4.92	4.92	4.92	4.59	
G&A	C$/t		1.50		3.21	1.41	1.43	1.43	1.43	1.42	1.39	1.33	
Total Operating Costs	**C$/t**		**10.38**		**20.66**	**9.91**	**10.65**	**10.31**	**10.54**	**10.15**	**9.35**	**8.60**	
Total Mining	C$ '000		$253,659		$19,627	$32,632	$39,186	$36,103	$38,159	$34,687	$27,659	$25,459	$146
Processing	C$ '000		$333,827		$15,264	$44,738	$44,738	$44,738	$44,738	$44,738	$44,738	$43,817	$6,317
G&A	C$ '000		$99,213		$6,425	$12,814	$13,015	$12,995	$13,025	$12,915	$12,644	$12,736	$2,644
Total Operating Costs	**C$ '000**		**$686,698**		**$41,316**	**$90,184**	**$96,940**	**$93,836**	**$95,922**	**$92,340**	**$85,041**	**$82,012**	**$9,106**
Unit Operating Cost	US$/oz Au		$493		936	454	417	471	479	528	511	559	301
Operating Cashflow	**C$ '000**		**$552,805**		**($2,058)**	**86,631**	**109,921**	**83,435**	**82,216**	**63,241**	**63,158**	**48,474**	**17,787**
Capital Costs													
Mining	C$ '000		$40,669	$6,790	$28,160	$657	$2,408	$304	$1,602	$412	$209	$126	$0
Crushing & Conveying	C$ '000		$73,134	$49,447	$18,562	$2,832	$131	$1,374	$263	$263	$263	$0	$0
Heap Leach Facility	C$ '000		$51,005	$14,760	$11,498	$2,511	$11,741	$0	$10,495	$0	$0	$0	$0
Process Plant	C$ '000		$17,252	$11,279	$4,797	$0	$0	$991	$0	$186	$0	$0	$0
Infrastructure	C$ '000		$45,509	$39,323	$3,247	$0	$2,939	$0	$0	$0	$0	$0	$0
Indirects	C$ '000		$55,306	$27,779	$27,527								
Contingency	C$ '000		$38,214	$25,762	$12,452								
Closure and Reclamation	C$ '000		$15,836	$0	$105	$105	$105	$105	$105	$105	$105	$105	$15,000
Total Capital Cost	**C$ '000**		**$336,925**	**$175,139**	**$106,347**	**$6,105**	**$17,325**	**$2,774**	**$12,464**	**$965**	**$576**	**$231**	**$15,000**
Cashflow													
Net Pre-Tax Cashflow	**US$ '000**		**$215,880**	**($175,139)**	**($108,405)**	**$80,527**	**$92,596**	**$80,661**	**$69,751**	**$62,276**	**$62,582**	**$48,244**	**$2,787**
Cumulative Pre-Tax Cashflow	US$ '000			($175,139)	($283,544)	($203,017)	($110,421)	($29,760)	$39,991	$102,267	$164,849	$213,093	$215,880
Payback	Years		3.4			1.0	1.0	1.0	0.4	0.0	0.0	0.0	0.0
Total Cash Cost	US$/oz		503		947	464	427	481	489	538	521	569	312
Capital Cost	US$/oz		242										
Total Production Cost	US$/oz		745										
Economics													
IRR	15.02%												
Pre-tax NPV at 5%	$115,291												
Pre-tax NPV at 7.5%	**$77,954**												
Pre-tax NPV at 10%	$47,081												
Pre-tax NPV at 15%	$156												

CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted pre-tax cash flow totals $216 million over the mine life, and simple payback occurs near the mid-point of 2017 (3.4 years).

The Total Cash Cost is US$503 per ounce of gold. The mine life capital unit cost is US$242 per ounce, for a Total Production Cost of US$745 per ounce of gold. Average annual gold production during operation is 170,000 ounces per year.

The Internal Rate of Return (IRR) is 15%, and the Net Present Values (NPV) at various discount rates are:

- 0% - $216 million
- 5% - $115 million
- 7.5% - $78 million
- 10% - $47 million

SENSITIVITY ANALYSIS

Key economic risks were examined by running cash flow sensitivities:

- Gold price
- Exchange rate
- Head grade
- Operating costs
- Pre-production capital costs
- Recovery

Sensitivity over the base case has been calculated for -20% to +20% variations to the key parameters, with the exception of recovery, which was tested over a range of 68% to 76%, in 2% increments. The sensitivities are shown in Figure 18-9 and Table 18-21.

The Project is most sensitive to gold price and head grade (which have the same impact), followed by exchange rate.

FIGURE 18-9 SENSITIVITY ANALYSIS

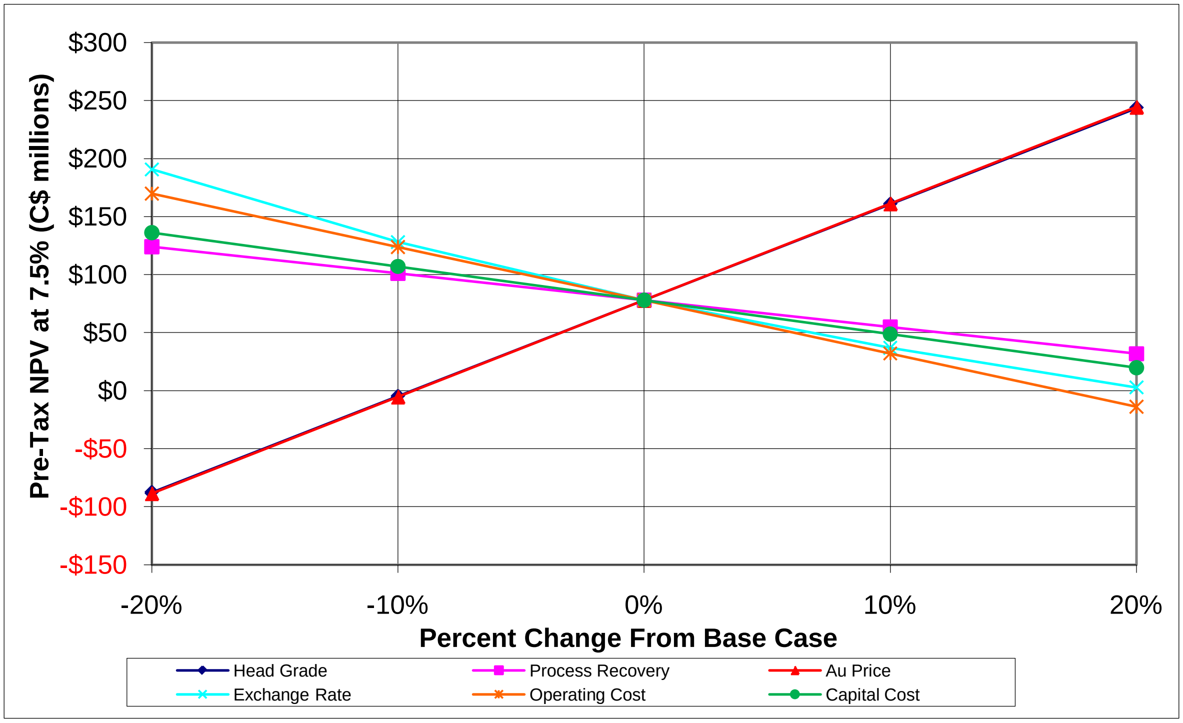

TABLE 18-21 SENSITIVITY ANALYSES
Victoria Gold Corp. – Eagle Gold Project

Parameter Variables	Units	-20%	-10%	Base	+10%	+20%
Gold Price	US$/oz	720	810	900	990	1080
Exchange Rate	US$/C$	0.72	0.81	0.90	0.99	1.08
Head Grade	g/t Au	0.66	0.74	0.82	0.91	0.99
Operating Cost	$/t	8.31	9.34	10.38	11.42	12.46
Capital Cost	$ millions	225	253	281	310	338
Recovery	%	68	70	72	74	76

NPV@7.5%	Units	-20%	-10%	Base	+10%	+20%
Gold Price	$ millions	(89)	(5)	78	161	244
Exchange Rate	$ millions	191	128	78	37	3
Head Grade	$ millions	(88)	(5)	78	161	244
Operating Cost	$ millions	170	124	78	32	(14)
Capital Cost	$ millions	136	107	78	49	20
Recovery	$ millions	32	55	78	101	124

19 INTERPRETATION AND CONCLUSIONS

In the opinion of Scott Wilson RPA and KCA, the PFS indicates positive economic results can be obtained for the Eagle Gold Project, in a scenario that includes open pit mining, three-stage crushing to a P_{80} of 5 mm, and heap leaching followed by carbon ADR gold recovery.

The Life of Mine Plan (LOMP) for the Project indicates that Mineral Reserves of 66 million tonnes, at an average grade of 0.82 g/t Au, will be mined over eight years, starting in 2013. Gold production is projected to total 1,254,000 ounces. Capital costs for construction are estimated to total C$281 million, including a contingency of C$38 million. Cash costs are projected to average US$503 per ounce. At a gold price of US$900 per oz, the Project is estimated to generate a pre-tax NPV of C$78 million, at a discount rate of 7.5%, and has an IRR of 15%.

Specific conclusions by area of the PFS are as follows.

GEOLOGY AND MINERAL RESOURCES

- Mineral Resources are estimated to be 154 Mt, at a grade of 0.65 g/t Au, containing 3.2 Moz. All resources are classified as Indicated, and were estimated at a cut-off grade of 0.21 g/t Au. Mineral Resources were constrained within a pit shell, generated at a gold price of US$1,050 per oz, and Scott Wilson RPA notes that mineralization extends considerably beyond the pit shell.

- In Scott Wilson RPA's opinion, the drill hole sampling, over the history of the project, appears to have been carried out to a reasonable standard. Data has been collected in several campaigns spanning many years, and for some years the documentation is not as detailed as for others. Specific concerns have been raised about certain sampling practices, which have resulted in some changes in protocols over the years. Most of the concerns raised by earlier reviewers were centred around assay repeatability and nugget effect. This could have an effect on the accuracy of local block grade estimates, however, in Scott Wilson RPA's opinion, there is no evidence of bias, and so the global grade estimates should be robust.

- The sampling should be representative of the mineralization.

- The core handling and security protocols, where documented, were consistent with common industry practice.

- Assaying was conducted using conventional methods, commonly used in the industry, and carried out by accredited commercial labs.

- It would appear as though reasonable levels of assay QA/QC were applied throughout the history of the Project, although there are some programs for which documentation is not available. Scott Wilson RPA has no reason to suspect that the assays are not of a reasonable standard and unbiased.

- The number of bulk density measurements taken to date is not sufficient and more should be taken. It is noted also, that early work conducted by MRDI led them to believe that there was a relationship between degree of weathering and density to the extent that tonnage estimates could be significantly affected. In Scott Wilson RPA's opinion, more testwork is necessary to improve the estimates of bulk density, and determine if there is a significant difference between unaltered and weathered (or altered) rock.

- Ore type classification is presently quite rudimentary and requires more work. The most important element of the ore-type categorization is the alteration logging, which at this time, is inconsistent from hole to hole. Different loggers appeared to have differing references as to the intensity and type of alteration in the core. Relogging of the alteration is required in order to provide a basis for consistent assignment of alteration type and intensity. Specific alteration facies that should be addressed are oxidation, silicification and sericitization. Once this logging has been completed, wireframe models should be constructed for any interpretable zones of alteration. The rock type interpolation into the block model should then be redone.

- There are exploration targets that remain to be explored in the vicinity of the Eagle deposit. Specific target areas that have potential for adding Mineral Resources are in the west end of the deposit, as well as along the trend of showings which encompass the Steiner, Olive, and Shamrock showings.

MINING

- Mineral Reserves are estimated to be 66 Mt, at a grade of 0.82 g/t Au, containing 1.8 Moz. All reserves are classified as Probable, and were estimated at a cut-off grade averaging 0.35 g/t Au. Mineral Reserves are based on a designed open pit, generated at a gold price of US$900 per oz. Scott Wilson RPA notes that mineralization extends considerably beyond the designed pit, as evidenced by the Mineral Resource estimate, and the mineralized wireframe.

- The reserve pit is slightly smaller than the optimum indicated by the reserve pit optimization, restricted by the size of the heap leach pad.

- Larger pits are certainly possible, given increased heap leach pad capacity, and higher gold prices or lower costs.

- Steeper bench face angles and overall steeper slopes in some design sectors of the pit are likely possible for a scenario including controlled blasting, however, due to the presence of other design sectors that had limited to no increase in overall slope angle controlling the overall slope model, limited benefit in the form of reduced strip ratio was observed, in comparison to the increased risks and costs attached to achieving steeper overall pit slopes.

- Further optimization of the waste haulage schedule may be possible, resulting in incremental cost advantages.

- One metre depth of free-digging material over the footprint of the open pit was assumed, consisting of a portion of salvageable soil, and a portion of incompetent weathered rock. This assumption may be conservative.

HEAP LEACHING

- Six potential heap leach pad sites were investigated, and the Ann Gulch pad location was assessed as the best, based on factors such as geotechnical conditions, favourable geometry for pad engineering, haulage distance from the pit, environmental considerations, and potential impacts on exploration targets.

- As currently designed, the pad capacity is limited to 67 Mt, based on a maximum thickness of 80 m, and maximum overall height of 180 m. It may be possible to increase this capacity up to approximately 80 Mt, by increasing the maximum heights, through introducing stabilizing measures to the design, or by carrying out testwork to confirm favourable material properties.

- Stacking operations are scheduled for 250 days per year, with a switch to winter stockpiling from early November to end of February, to avoid creation of permanent ice lenses within the heap. Natural weather variations and other uncertainties make it difficult to calculate the correct number of days with much precision. It is possible that stacking operations will average more days per year, with a corresponding improvement in Project economics.

- Preliminary agglomeration tests indicate that a minor amount of cement may be required in the lower lifts of the multi-lift heap leach operation. Additional testwork is required, but up to 2 kg/t of cement may be required during the first few years of operation. The pad location requires that the Dublin Gulch waterway be diverted from its current course, to accommodate the heap leach embankment that stabilizes the toe of the heap leach pad.

- A second pad location could be added in future studies, to accommodate a larger pit.

METALLURGICAL TESTWORK AND PROCESS DESIGN

- There is a distinct increase in gold recovery with decreasing crush size for all ore types. There is also an indication that HPGR crushing results in an increase in gold recovery, compared to conventional crushing to the same P_{80} crush size. The data set is not complete, however, there is an apparent two to six percentage point increase in gold recovery between conventional and HPGR crushing, based on comparison of data at the P_{80} crush sizes of 10 mm and

5 mm. Additional testing is required, but the potential for an increase in gold recovery by the use of HPGR crushers led to the decision to utilize these crushers for the PFS.

- Past testwork on gravity-flotation processes resulted in gold recoveries averaging 95%, at sizes that require grinding, however, capital and operating cost increases of a milling scenario were found to be in excess of the additional revenue from better recovery. Currently, testwork is underway on evaluating the addition of a gravity circuit to the PFS base case processing scenario, at a P_{80} of 5 mm. A gravity circuit would have the advantage of earlier gold production, however, KCA notes that cold winter ore would present operational difficulties, and gravity circuits may be best run seasonally.

- Cyanide neutralization and detoxification testing was completed on coarser-crushed column leach tailings and barren solutions in 1996 and 1997. No neutralization data are available at a 5 mm crush size. Additional testing will be required, and is currently underway.

INFRASTRUCTURE

- Power for the Project is assumed to be available from the Yukon Energy transmission grid by 2013. Yukon Energy is currently in the process of upgrading power generation capacity at Mayo (the Mayo Hydro Enhancement Project), and connecting the north and south Yukon transmission grids (the Carmacks-Stewart Transmission Project Stage 2). Funding agreements specify that both projects must be completed by March 31, 2012, however, Yukon Energy reports that both projects are on schedule to finish in 2011.

20 RECOMMENDATIONS

Scott Wilson RPA and KCA recommend that Victoria Gold advance the Eagle Gold Project to the Feasibility Study stage. A Project Proposal (currently under preparation) should be submitted to the Yukon Environmental and Socio-Economic Assessment Board (YESAB) for Project permitting. Specific recommendations by area are as follows.

GEOLOGY AND MINERAL RESOURCES

Scott Wilson RPA makes the following recommendations:

- Additional bulk density measurements should be taken from intact core specimens. Where the rock is porous, the specimens should be sealed with wax or plastic wrap in order to prevent overestimation due to the porosity.
- Assay QA/QC results should be monitored and analyzed as soon as they are received, in order to allow for corrective actions, where required. Control sample failures should result in request for reassay of several samples ahead of and behind the failure. In extreme cases, entire assay batches should be rerun.
- A selection of drill holes should be relogged for alteration type and intensity. This relogging should be carried out with the intent of providing a consistent evaluation of the alteration across the deposit. The alteration data should then be used to create wireframes of the more intensely altered zones (if possible) and to update the ore-type classification in the block model.
- Additional drilling should be carried out to both expand the resource inventory and sterilize those areas that are proposed for key components of the mine operation.

MINING AND HEAP LEACHING

Scott Wilson RPA makes the following recommendations:

- Investigate scenarios with larger pits, either incrementally, by increasing the capacity of the Ann Gulch heap leach pad, or in a larger step, by adding a second heap leach pad. Incremental increases can likely be accommodated at the US$900 per ounce Mineral Reserve gold price, while larger increases are likely to require higher gold prices.
- Complete further optimization of the waste haulage schedule.
- Develop model surfaces for the soil/rock contact and the boundary at which drilling and blasting becomes necessary for excavation. Existing test pit and drill hole data should provide some information, which may need to be supplemented in the field.

- Carry out wet and dry slump tests on 5 mm crush size material.
- Complete agglomeration testwork.
- Further optimization of the Dublin Gulch waterway diversion to minimize environmental impact.

METALLURGICAL TESTWORK AND PROCESS DESIGN

KCA makes the following recommendations:

- Complete testwork for Feasibility-level design of HPGR crushing at a P_{80} of 5 mm.
- Investigate scenarios for addition of a gravity recovery circuit, based on testwork currently in progress.
- Complete testwork for cyanide neutralization and detoxification at a 5 mm crush size.

Victoria Gold provided the following budget for gathering information in the 2010 field season, and completing a Feasibility Study. In the opinion of Scott Wilson RPA and KCA, this budget is reasonable and appropriate for advancing the Project.

TABLE 20-1 FEASIBILITY STUDY BUDGET
Victoria Gold Corp. – Eagle Gold Project

Item	Cost (C$'000s)
Exploration	750
Condemnation & Geotechnical Drilling	1,000
Metallurgical Testwork	200
Environmental Fieldwork	500
Engineering	2,500
Camp Operations & Civil Testing	1,500
Total	**6,450**

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22 DATE AND SIGNATURE PAGE

This report titled "Technical Report on Eagle Gold Project, Yukon Territory, Canada" and dated April 23, 2010, was prepared and signed by the following authors:

	(Signed & Sealed)
Dated at Toronto, ON April 23, 2010	Jason J. Cox, P.Eng. Senior Mining Engineer Scott Wilson RPA
	(Signed & Sealed)
Dated at Reno, NV April 23, 2010	Dan Kappes, P.E. Principal Metallurgist Kappes Cassiday & Associates
	(Signed & Sealed)
Dated at Vancouver, BC April 23, 2010	David W. Rennie, P.Eng. Principal Geologist Scott Wilson RPA

23 CERTIFICATE OF QUALIFIED PERSON

JASON J. COX

I, Jason J. Cox, P.Eng., as an author of this report entitled "Technical Report on the Eagle Gold Project, Yukon Territory, Canada", prepared for Victoria Gold Corp., and dated April 23, 2010, do hereby certify that:

1. I am a Senior Mining Engineer with Scott Wilson Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of the Queen's University, Kingston, Ontario, Canada, in 1996 with a Bachelor of Science degree in Mining Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 90487158). I have worked as a Mining Engineer for a total of 14 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements.
    - Pre-Feasibility and Feasibility Study work on several projects.
    - Worked as a Mining Engineer at three North American mines, including experience in mine construction, operation, and closure.

4. 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.

5. I visited the Eagle Gold Property on June 10-11, 2009.

6. I am responsible for overall preparation of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43-101.

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

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd day of April, 2010

(Signed & Sealed)

Jason J. Cox, P.Eng.

DAVID W. RENNIE

I, David W. Rennie, P.Eng., as an author of this report entitled "Technical Report on the Eagle Gold Project, Yukon Territory, Canada", prepared for Victoria Gold Corp., and dated April 23, 2010, do hereby certify that:

1. I am a Principal Geologist with Scott Wilson Roscoe Postle Associates Inc. of Suite 388, 1130 West Pender St., Vancouver, BC, V6E 4A4 .

2. I am a graduate of the University of British Columbia, Vancouver, BC, Canada, in 1979 with a Bachelor of Applied Science degree in Geological Engineering.

3. I am registered as a Professional Engineer in the Province of British Columbia (Reg.# 13,572). I have worked as a Geological Engineer for a total of 31 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements.
    - Pre-Feasibility and Feasibility Study work on several projects.
    - Worked as a Geological Engineer at several mines and exploration projects in a number of countries.

4. 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.

5. I visited the Eagle Gold Property on June 10-11, 2009.

6. I am responsible for Sections 7 through 15, inclusive, 17, and parts of Sections 1, 20, 21, and 22 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43-101.

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

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 23rd day of April, 2010

(Signed & Sealed)

David W. Rennie, P.Eng.

DANIEL W. KAPPES

I, Daniel W. Kappes, P.E., as an author of this report entitled "Technical Report on the Eagle Gold Project, Yukon Territory, Canada", prepared for Victoria Gold Corp., and dated April 23, 2010, do hereby certify that:

1. I am President of the firm of Kappes, Cassiday & Associates at 7950 Security Circle, Reno, Nevada USA 89506.

2. I am a graduate of the Colorado School of Mines (1966) and the University of Nevada, Mackay School of Mines (1972), and hold B. Sc. and M. Sc. degrees in Mining Engineering.

11. I am a Professional Mining and Metallurgical Engineer (No. 3223) in the State of Nevada, USA, registered through the Nevada State Board of Professional Engineers and Land Surveyors. I have practiced my profession continuously since 1966. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements.
    - Pre-Feasibility and Feasibility Study work on many projects.

3. 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.

4. I have not visited the Eagle Gold Property.

5. I am responsible for preparation of Section 16 and portions of Section 18 of the Technical Report.

6. I am independent of the Issuer applying the test set out in Section 1.4 of NI 43-101.

7. I have had prior involvement with the property that is the subject of the Technical Report. Kappes, Cassiday & Associates conducted metallurgical test work on this property for another company in 1996 and 1997.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

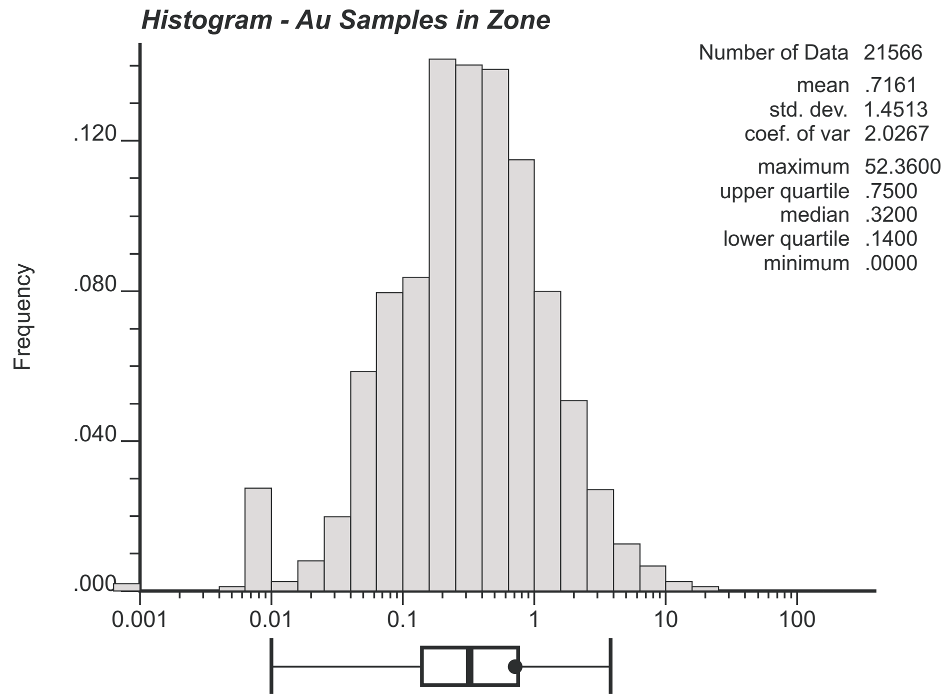
Dated this 23rd day of April, 2010

(Signed & Sealed)

Daniel W. Kappes, P.E.

24 APPENDIX 1

SAMPLE STATISTICS



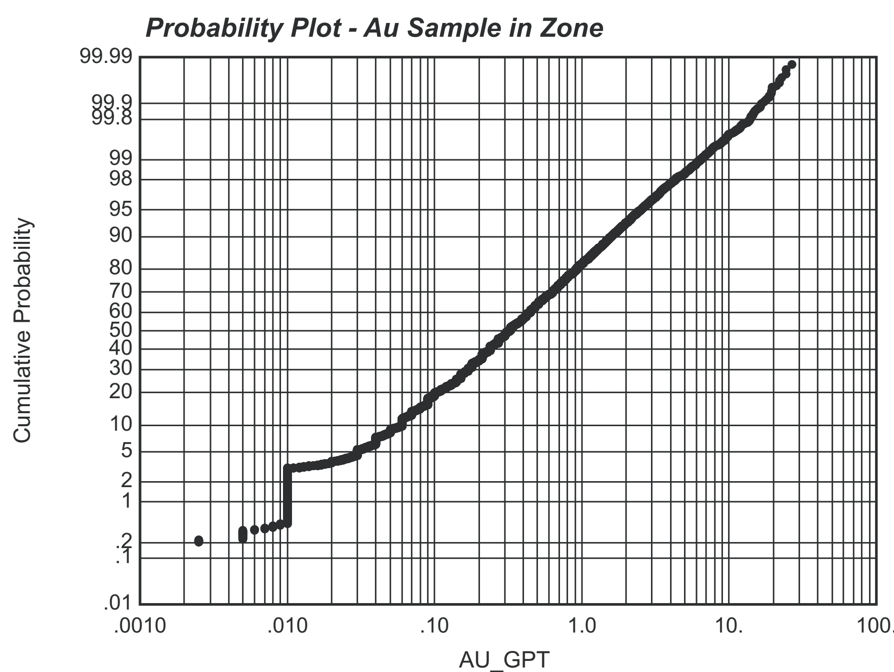


Figure A1-1

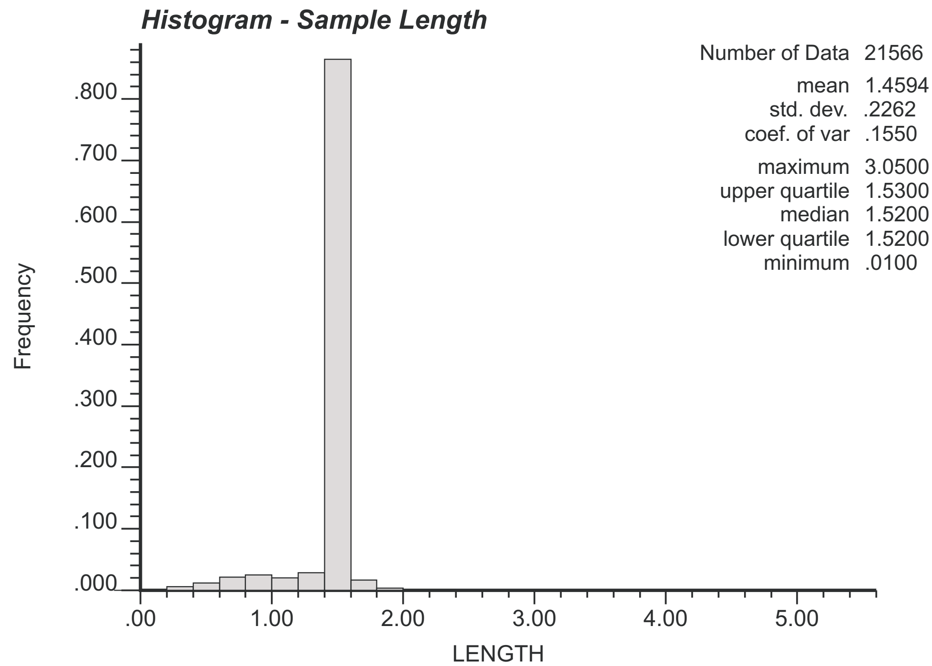
Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Sample Assay Statistics

April 2010



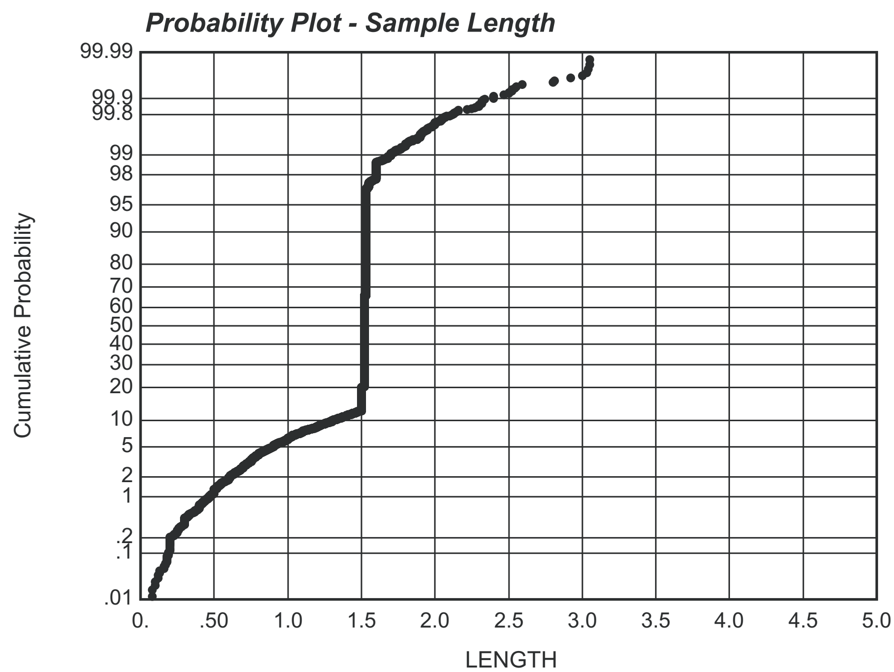


Figure A1-2

Victoria Gold Corp.

Eagle Gold Project

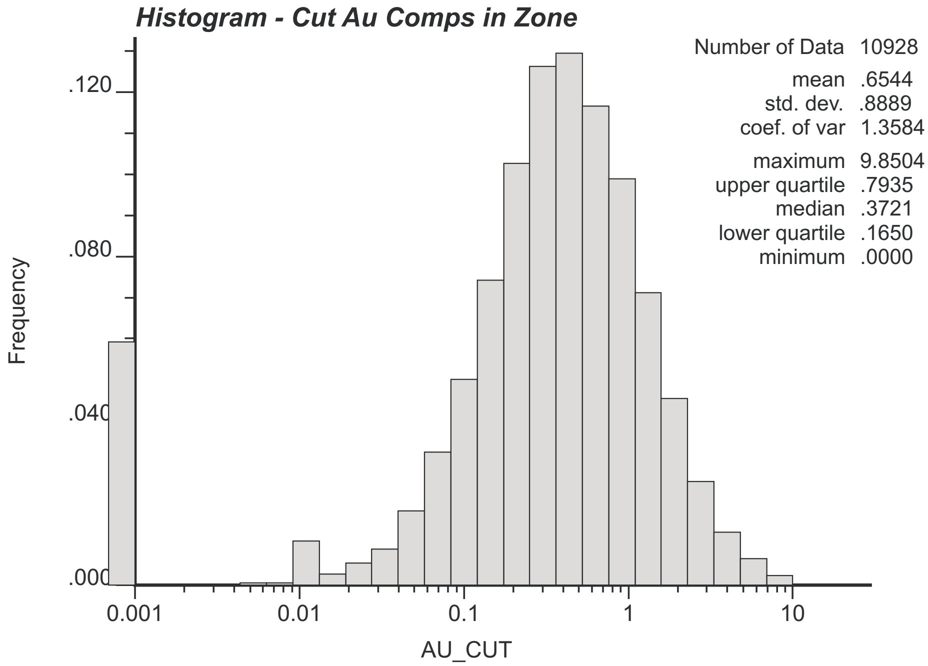
Yukon Territory, Canada

Sample Length Statistics

April 2010

25 APPENDIX 2

COMPOSITE STATISTICS



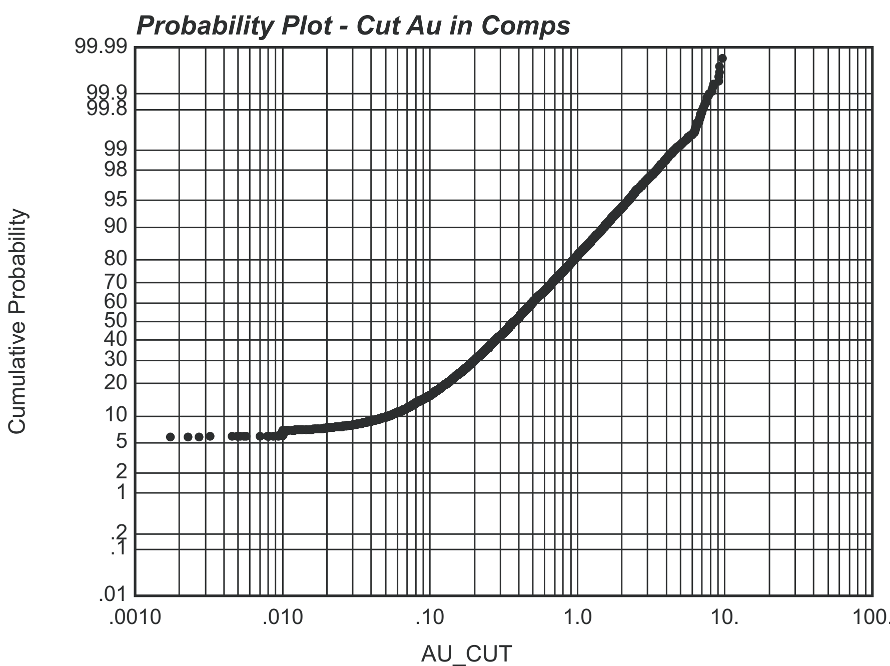


Figure A2-1

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

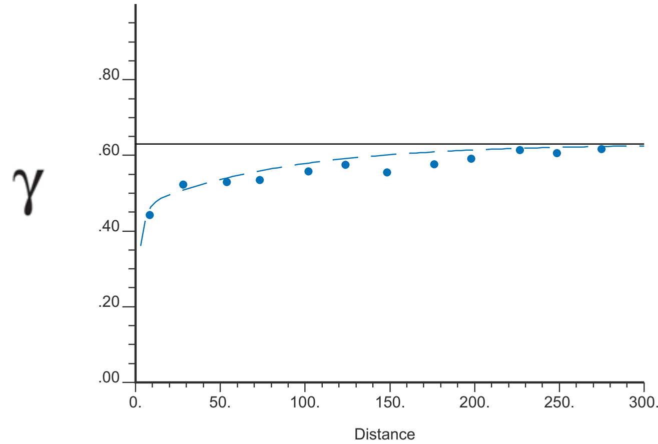
Composite Assay Statistics

April 2010

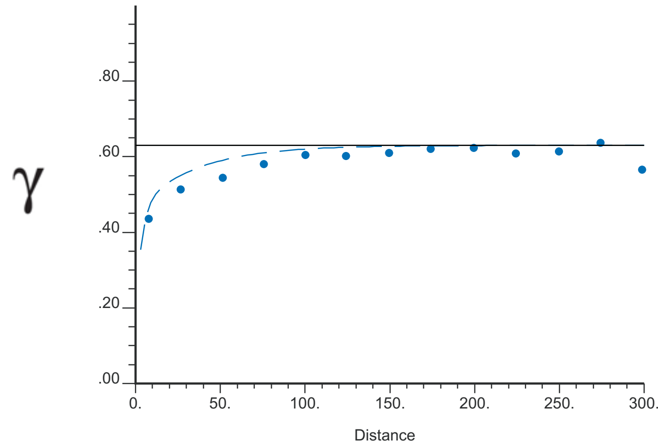
26 APPENDIX 3

GEOSTATISTICS

Variogram Plot - Major Axis



Variogram Plot - Semi-Major Axis



Variogram Plot - Minor Axis

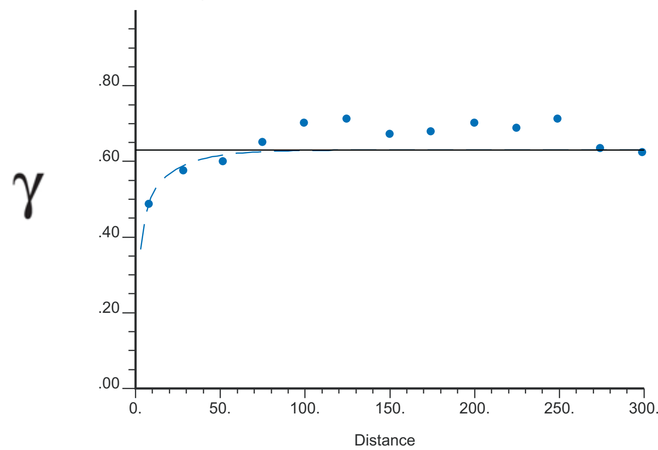


Figure A3-1

Victoria Gold Corp.

Eagle Gold Project

Yukon Territory, Canada

Variogram Model

April 2010

27 APPENDIX 4

CROSS SECTIONS

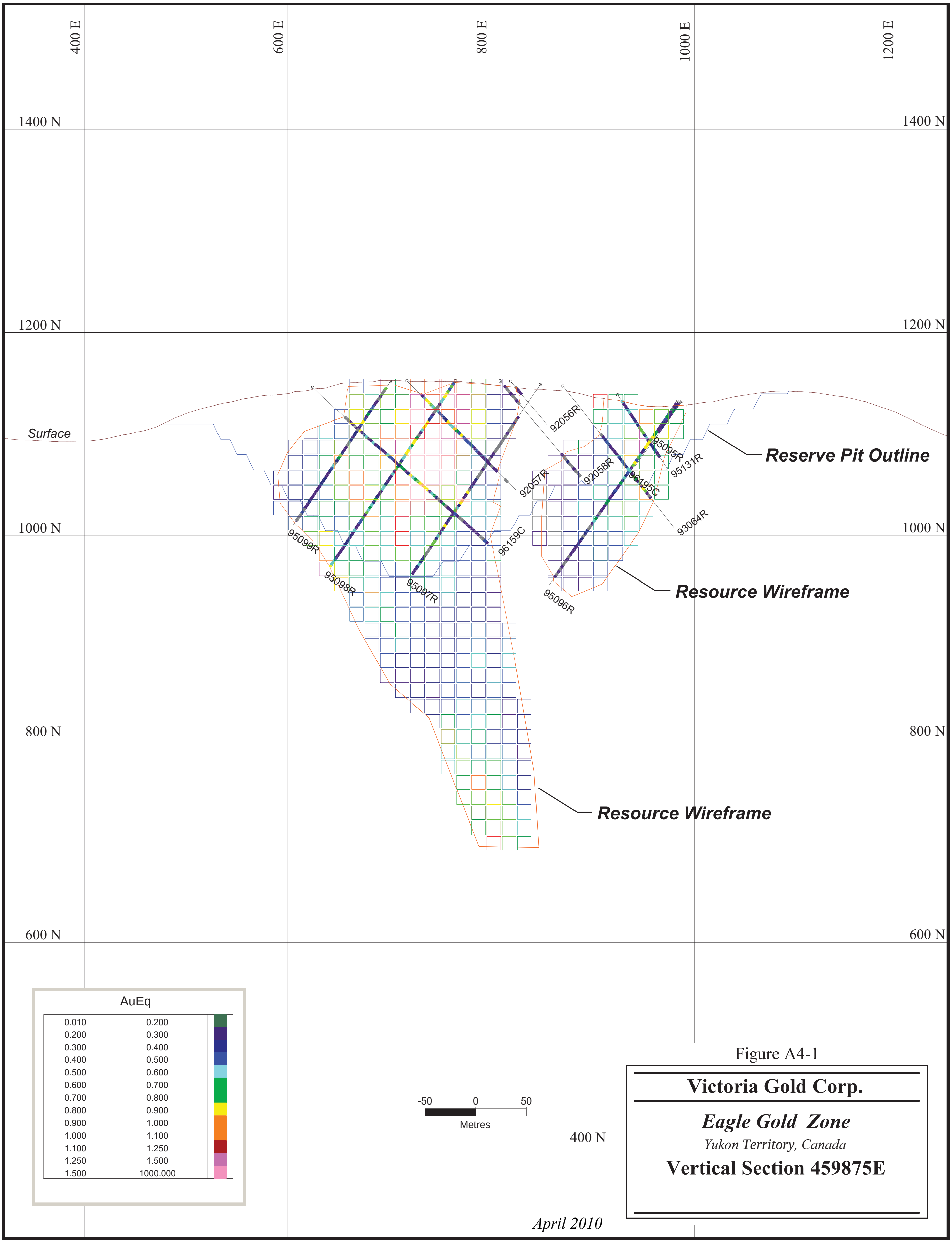


Figure A4-1

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Vertical Section 459875E

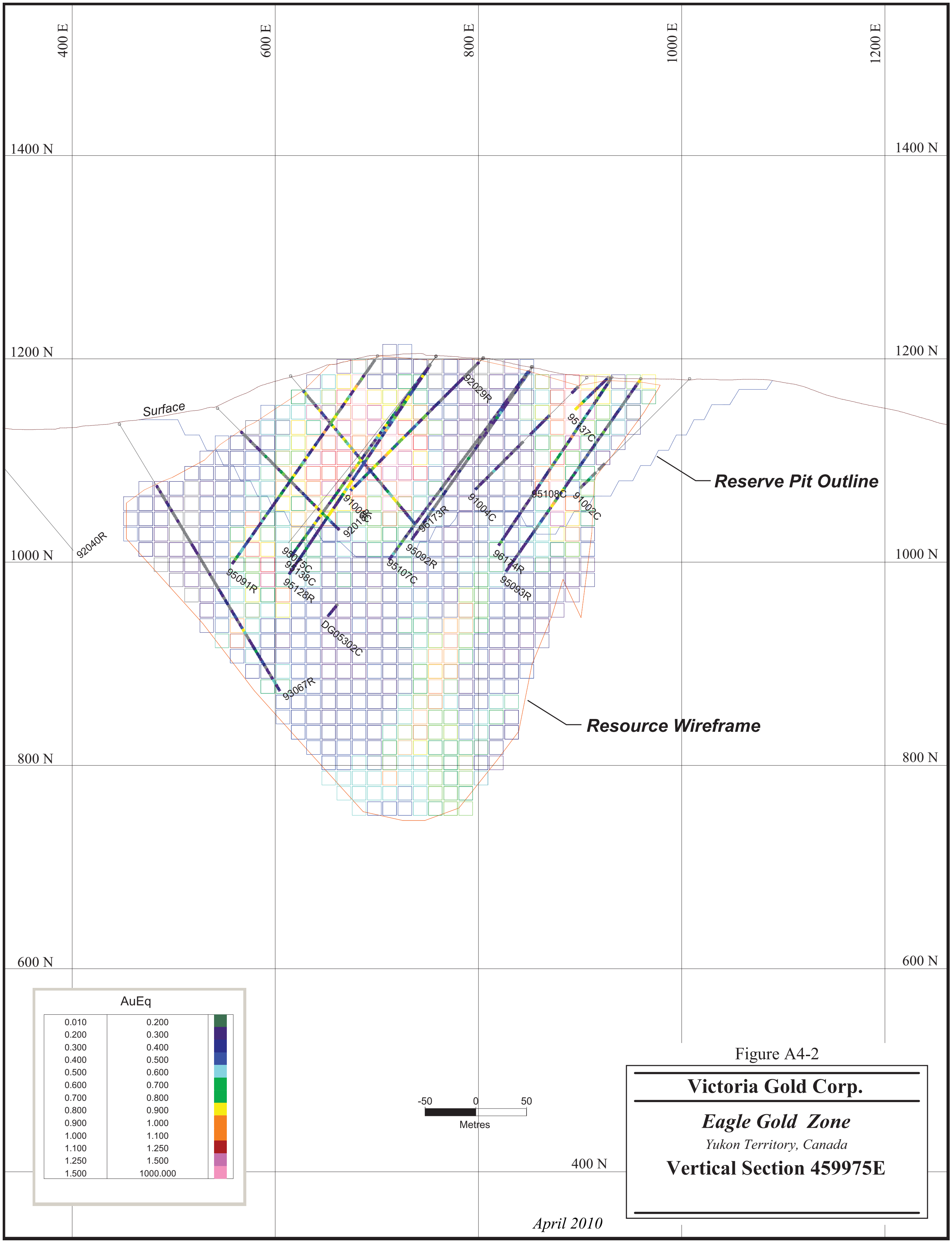


Figure A4-2

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Vertical Section 459975E

April 2010

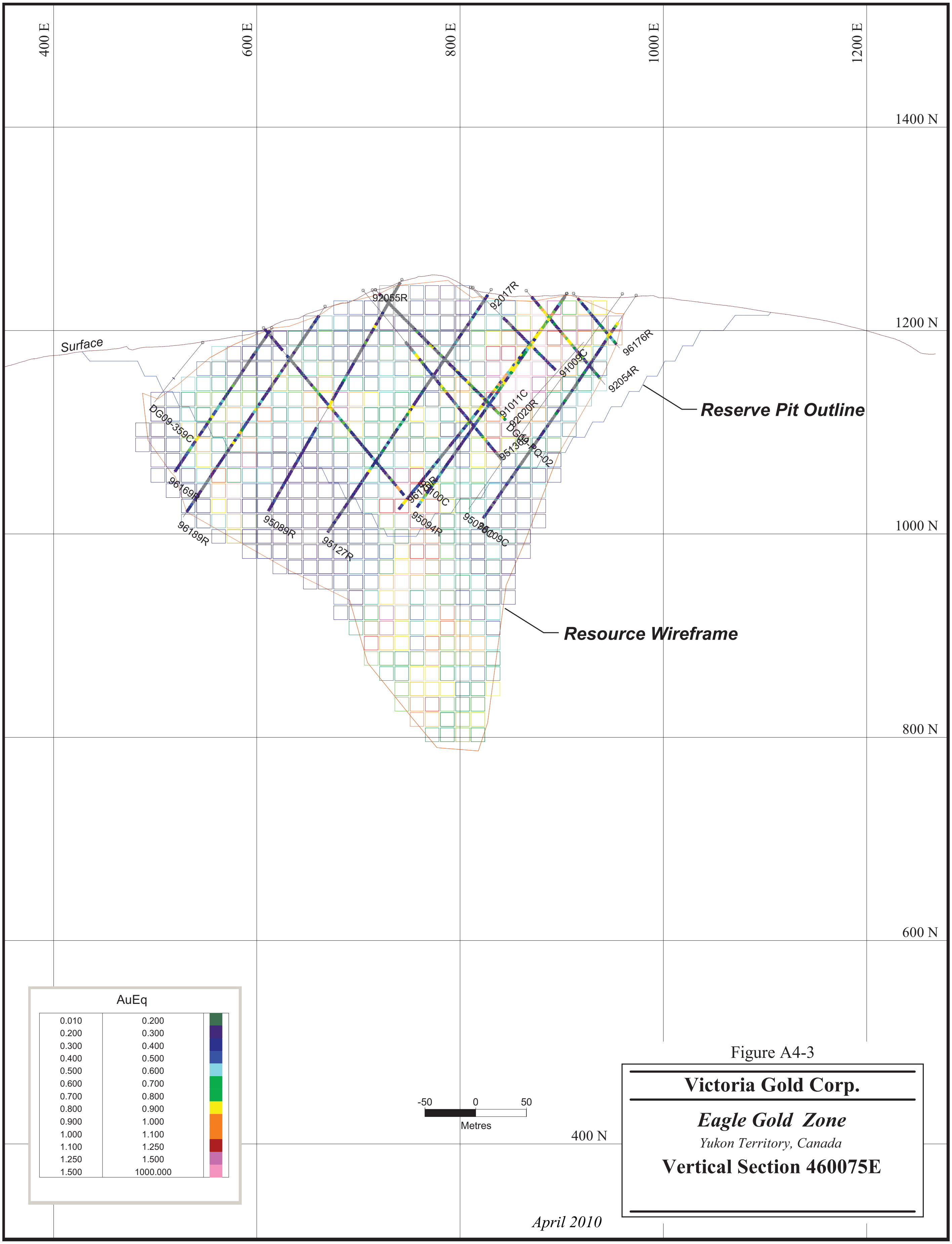


Figure A4-3

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Vertical Section 460075E

April 2010

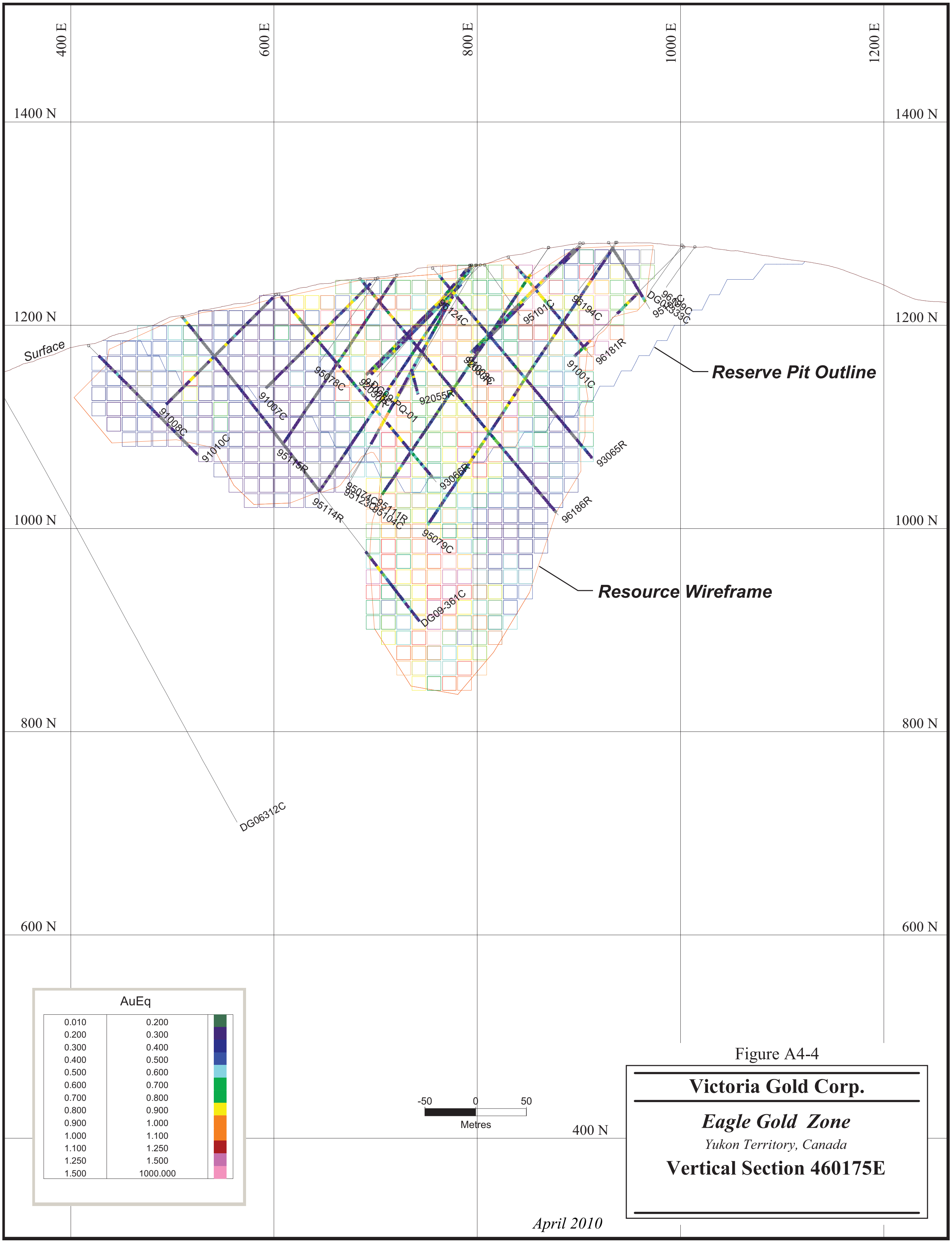


Figure A4-4

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Vertical Section 460175E

April 2010

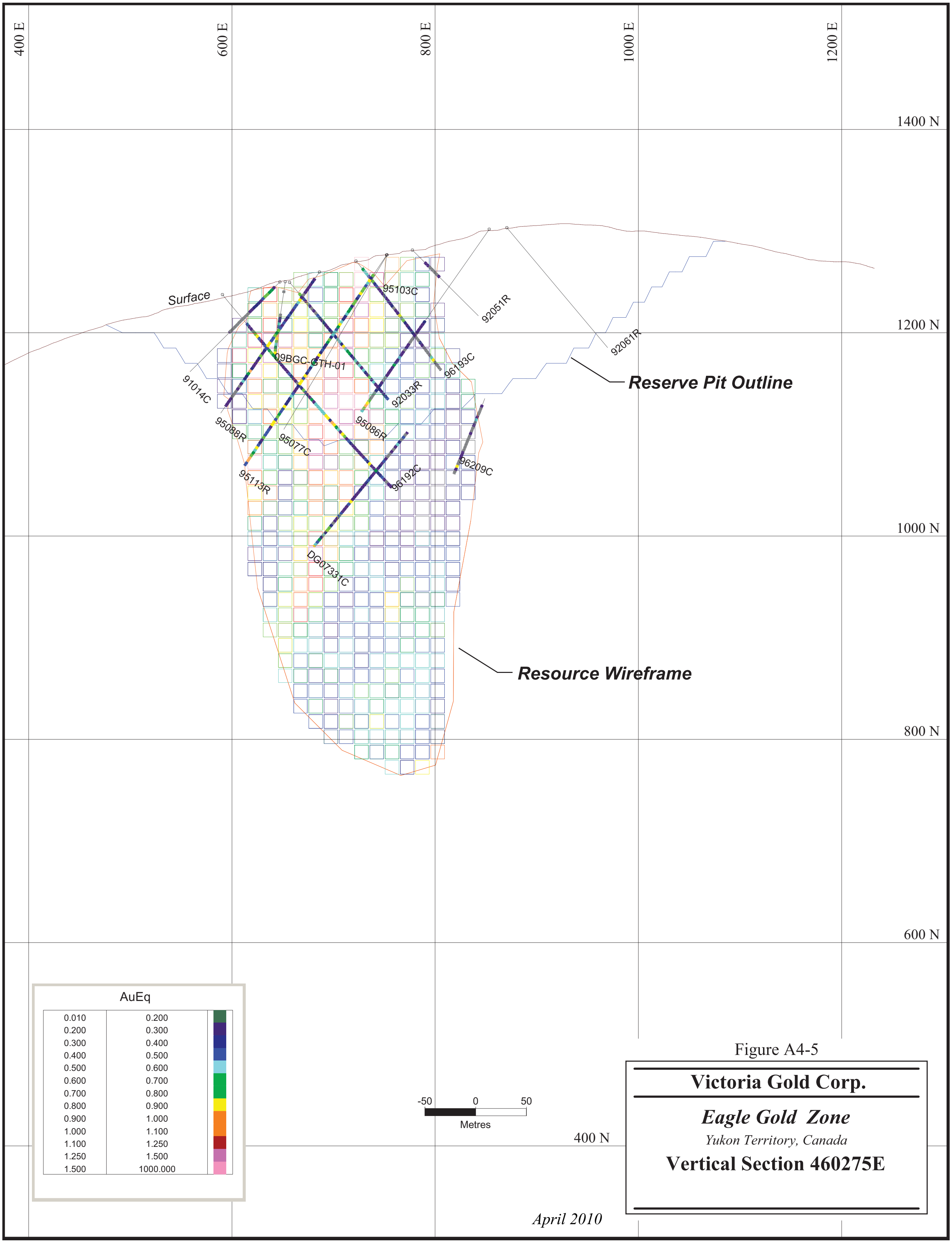


Figure A4-5

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Vertical Section 460275E

28 APPENDIX 5

PLANVIEWS

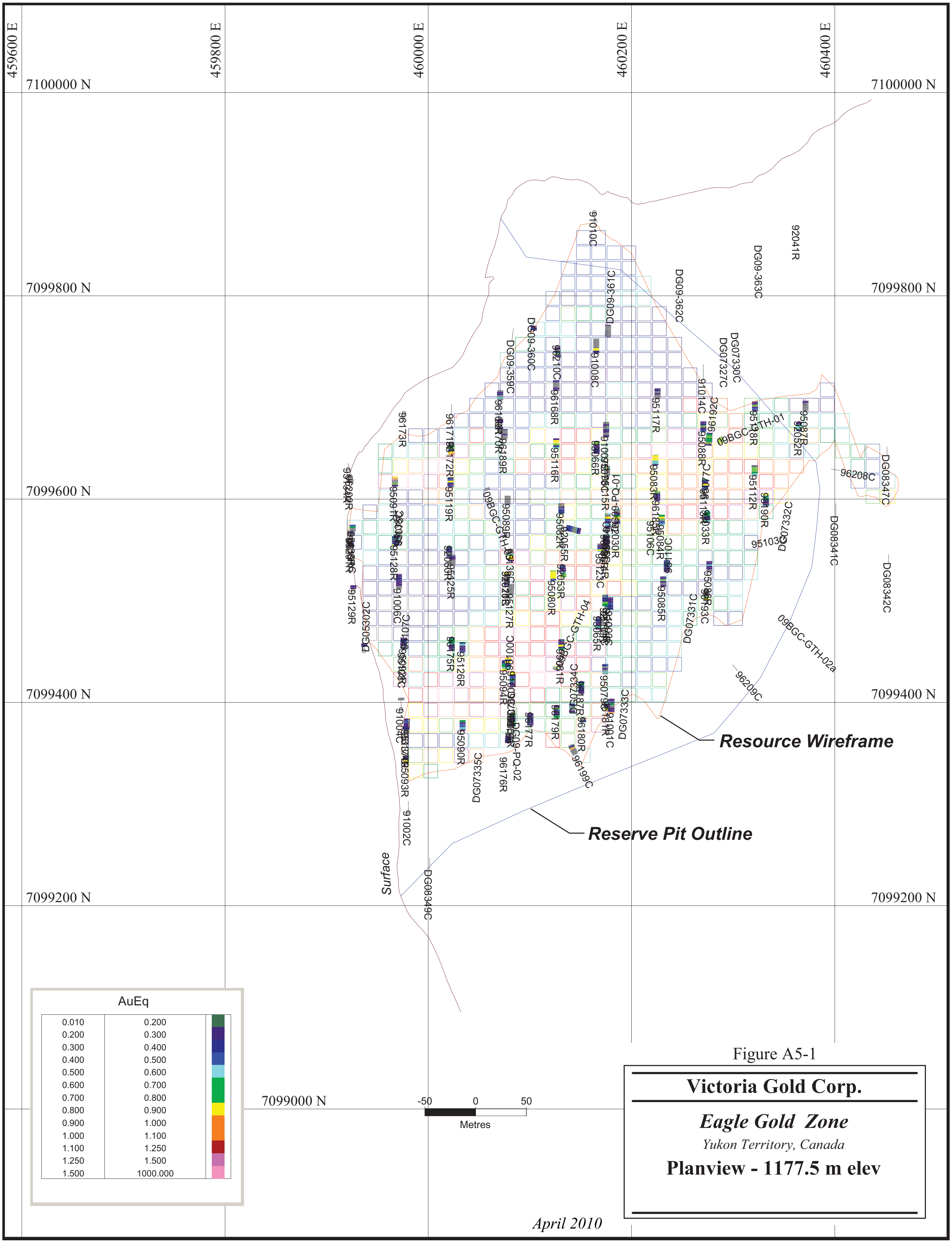


Figure A5-1

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Planview - 1177.5 m elev

April 2010

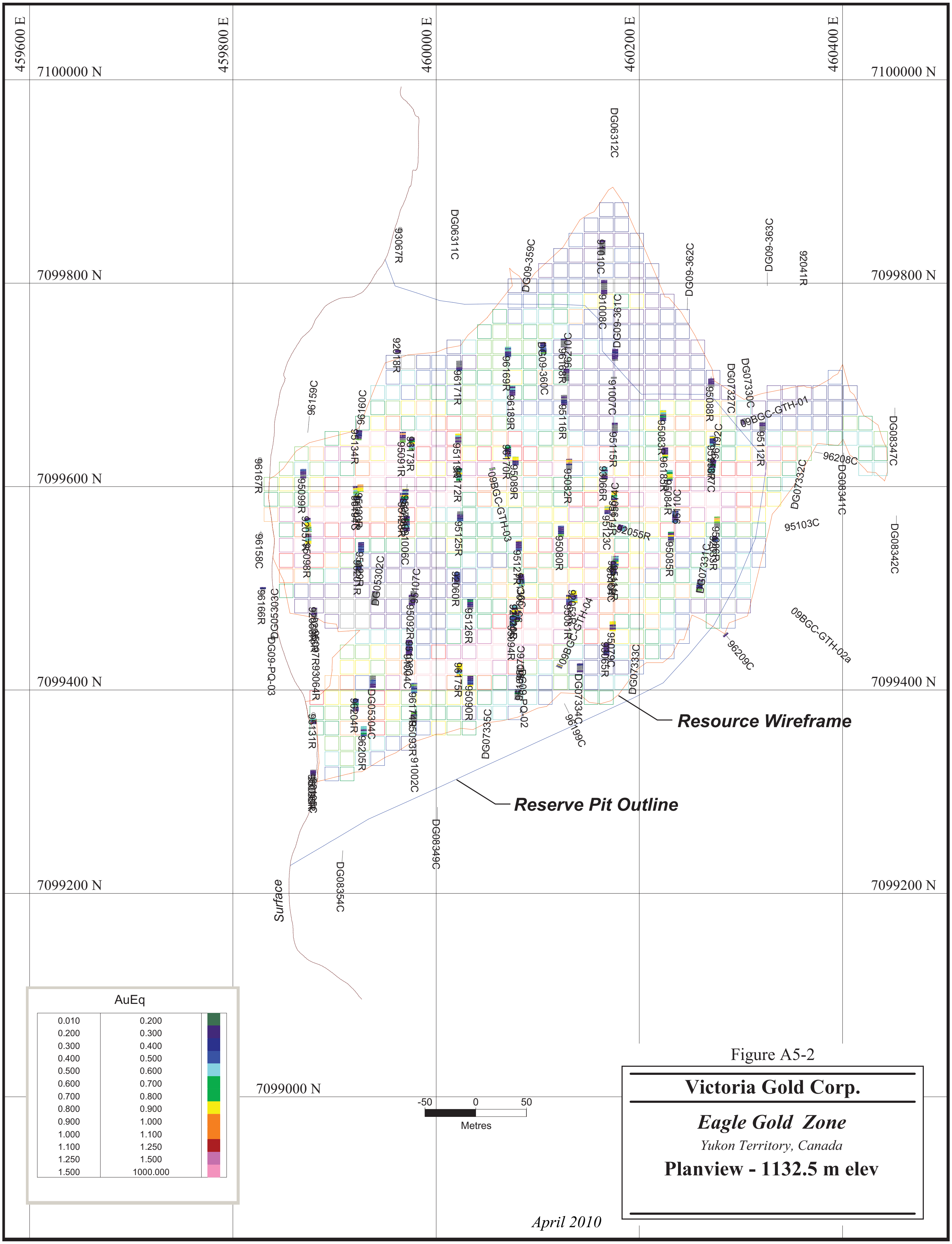


Figure A5-2

Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Planview - 1132.5 m elev

April 2010

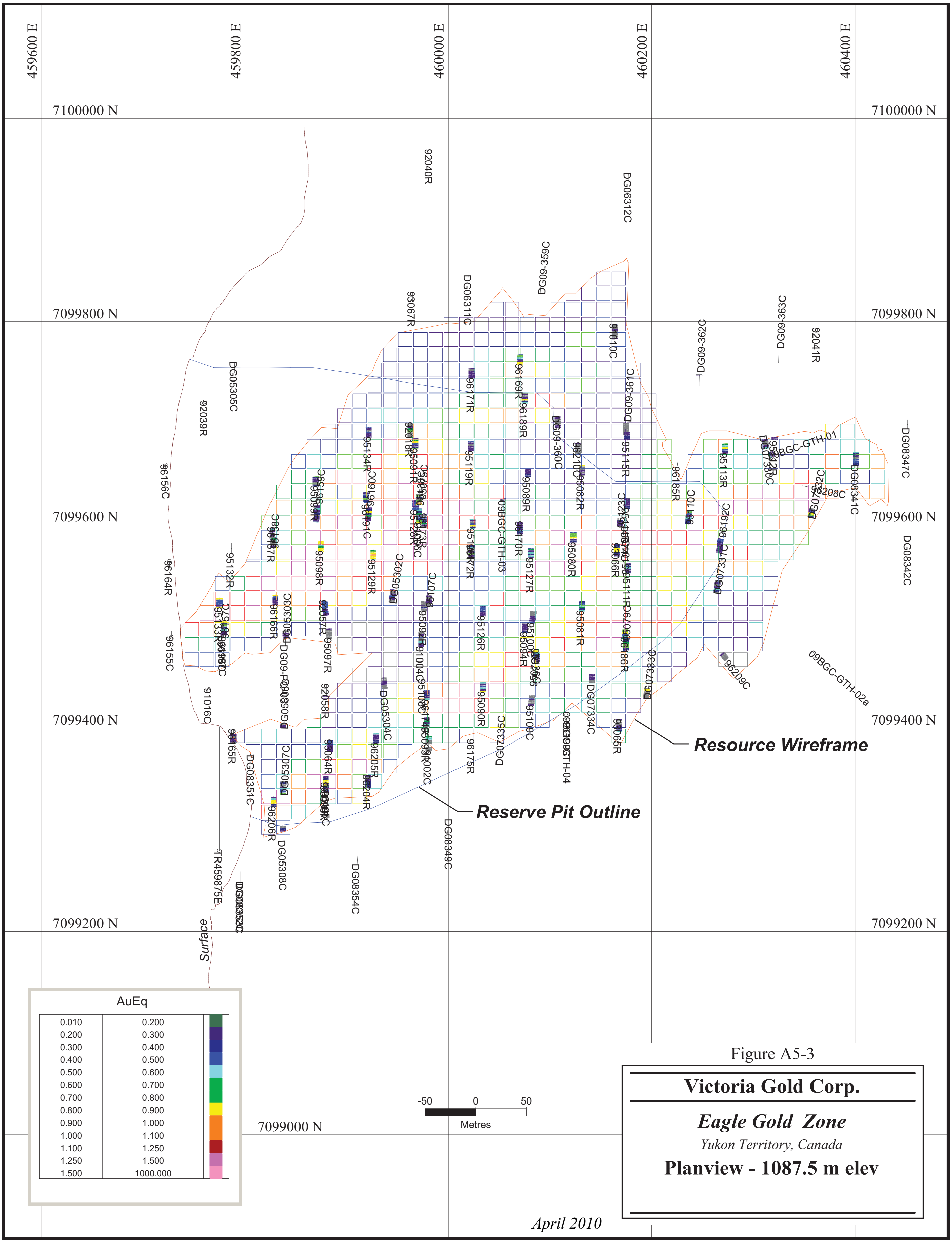


Figure A5-3

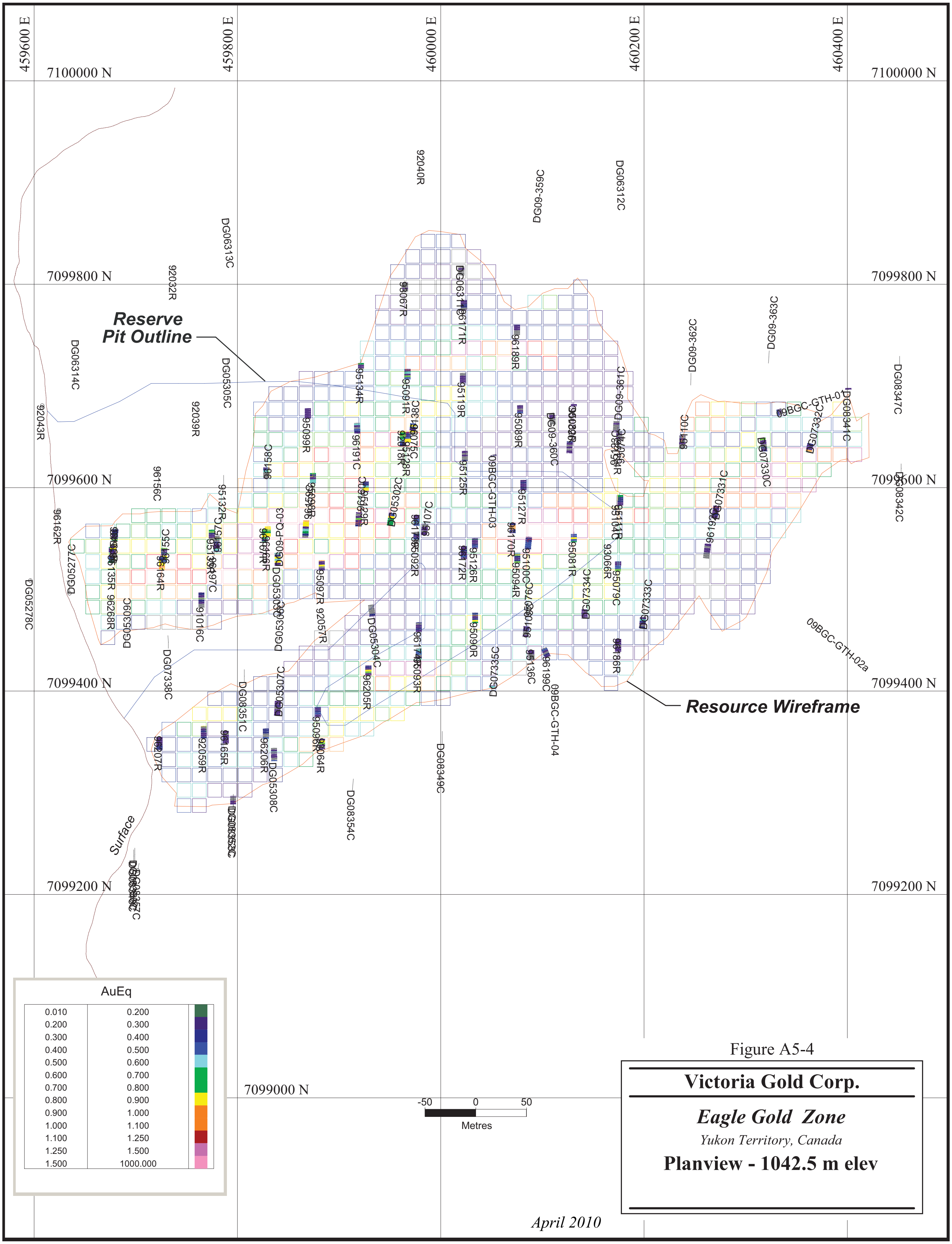
Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Planview - 1087.5 m elev

April 2010



AuEq	
0.010	0.200
0.200	0.300
0.300	0.400
0.400	0.500
0.500	0.600
0.600	0.700
0.700	0.800
0.800	0.900
0.900	1.000
1.000	1.100
1.100	1.250
1.250	1.500
1.500	1000.000

Figure A5-4

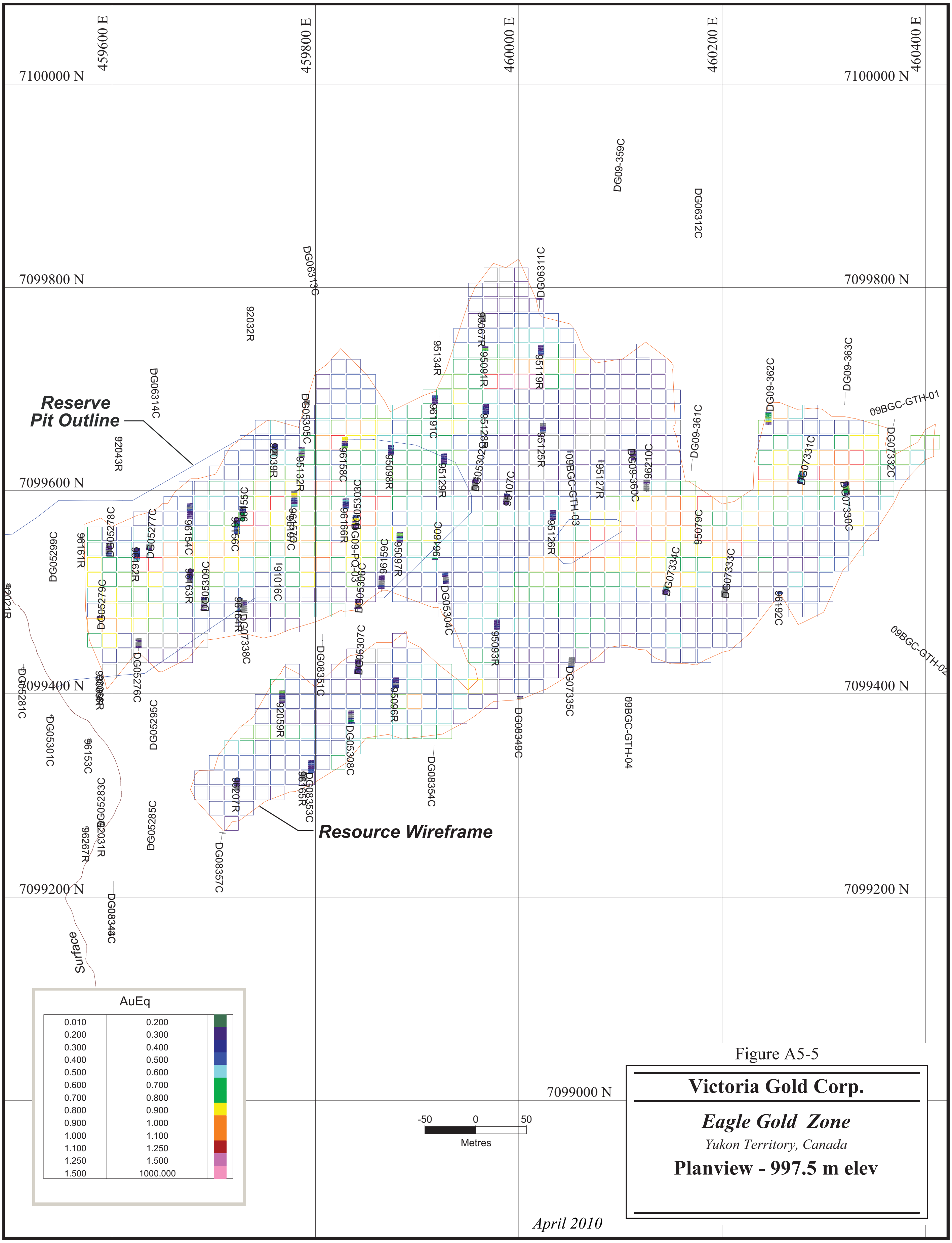
Victoria Gold Corp.

Eagle Gold Zone

Yukon Territory, Canada

Planview - 1042.5 m elev

April 2010


459600 E
459800 E
460000 E
460200 E
460400 E
7100000 N
7099800 N
7099600 N
7099400 N
7099200 N
7099000 N
Reserve Pit Outline
Resource Wireframe
Surface
AuEq
0.010 0.200
0.200 0.300
0.300 0.400
0.400 0.500
0.500 0.600
0.600 0.700
0.700 0.800
0.800 0.900
0.900 1.000
1.000 1.100
1.100 1.250
1.250 1.500
1.500 1000.000
-50 0 50
Metres
Figure A5-5
Victoria Gold Corp.
Eagle Gold Zone
Yukon Territory, Canada
Planview - 997.5 m elev
April 2010