NI 43-101 Feasibility Study Technical Report For The Eagle Gold Project, Yukon Territory, Canada

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Victoria Gold Corp. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Victoria Gold Corp. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Executive Summary

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Victoria Gold Corporation (Victoria Gold) to lead and compile a Feasibility Study (FS) and Technical Report for the Eagle Gold project (Eagle Gold or project), an advanced exploration gold project owned by Victoria Gold. The project is located in the Mayo Mining District of Central Yukon Territory, approximately 45 km north of the community of Mayo. The FS was completed by the following independent authors:

- Merit Consultants International Inc. (Merit);
- Kappes, Cassiday & Associates (KCA);
- SRK Consulting (US) Inc. (SRK);
- DOWL Engineering (DOWL);
- Allan V. Moran Consulting LLC (Allan Moran);
- Allnorth Consultants Ltd. (Allnorth); and
- JDS Energy & Mining Inc. (JDS).

This report presents the results of the FS using the guidance of the Canadian Securities Administrators' National Instrument (NI) 43-101 and Form 43-101F1 and Canadian Institute of Mining (CIM) guidance on Resource and Reserve Estimation.

1.2 Project Description

The Eagle and Olive deposits, situated within Victoria Gold's Dublin Gulch property, are planned to be mined using open pit (OP) methods. The Eagle deposit will provide 116 million tonnes (Mt) of ore while the Olive deposit will provide 7 Mt for a total of 123 Mt. Waste mining will total 116 Mt for an overall 0.95:1 strip ratio. Production throughput will be an average of 12.5 million tonnes per annum (Mt/a) (33,700 tonnes per day (t/d)) of ore over a 10-year mine life, excluding the ramp-up period.

Gold will be extracted from ore into a solution by a heap leaching process using two heap leaching pads (HLPs) – a primary and a secondary. Heap leach feed will consist of crushed ore (108 Mt at 0.73 g/t Au) conveyed to the HLPs as well as run-of-mine (ROM), un-crushed (15 Mt @ 0.27 g/t Au) ore, which will be hauled directly to the primary HLP for leaching. Both crushed and ROM ore will be stacked on the primary HLP, while only crushed ore will be placed on the secondary HLP.

Two HLPs, the primary and secondary pads, will be used to extract gold from ore into solution. Both crushed and ROM ore will be stacked on the primary pad, while only crushed ore will be placed on the secondary pad.

Crushed ore will be fed through a three-stage crushing plant to produce an 80% passing (P_{80}) 6.5 mm product. ROM ore will bypass the crushing plant.

Gold will be leached with cyanide solution and recovered by an adsorption-desorption-regeneration (ADR) carbon plant.

A total of 1,884 koz of gold will be recovered over a 10-year mine life from 70.8% overall recovery.

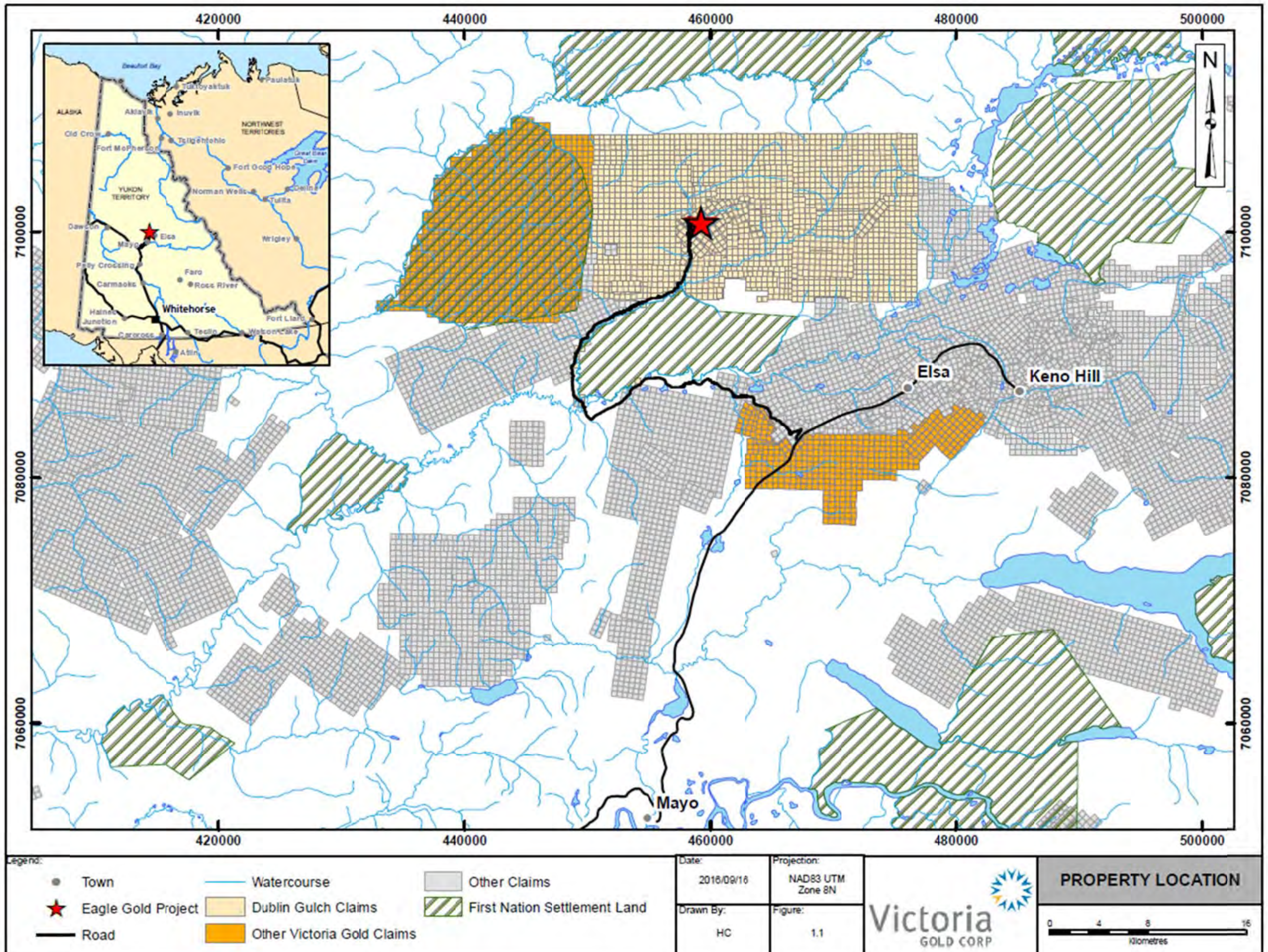
1.3 Property Description and Ownership

The Eagle Gold project is situated approximately 350 km north of the Yukon capital of Whitehorse (Figure 1.1). The centre of the project is at approximately 64°01'30"N latitude and 135°49'30"W longitude or Universal Transverse Mercator (UTM) Coordinates 7,100,060N / 459,680E, Zone 8, North American Datum (NAD) 83. Access to the project from Mayo is via the Silver Trail (Highway 11), then onto the South McQuesten and Haggart Creek Roads that terminate at the existing 100-person all-season camp, owned by Victoria Gold.

The project is situated within the Dublin Gulch property, which is a contiguous block of 1,914 quartz claims, 10 quartz leases, and one federal crown grant all of which are under the control of Victoria Gold's wholly owned, directly held subsidiary StrataGold Corporation (StrataGold). The Dublin Gulch property is rectangular in shape and extends approximately 26 km in an east-west direction and 13 km in a north-south direction covering an aggregate area of approximately 35,000 hectares (ha).

A property location map is provided in Figure 1.1.

Figure 1.1: Property Location Map



1.4 Geology & Mineralization

The geological setting of the Dublin Gulch property (Eagle Gold project) is underlain by upper Proterozoic to lower Paleozoic clastic sedimentary rocks that have undergone regional deformation including Cretaceous age thrust faulting and subsequent granitoid intrusions. Mineralization is associated with granitic intrusive bodies, here described as the Eagle Zone and Olive Zone gold deposits, which are hosted primarily in granodioritic rocks. The gold deposits occur within the Tombstone Gold Belt, located in the eastern portion of the Tintina Gold Province, which also hosts the Brewery Creek deposit and other gold occurrences in the Yukon.

The property is located on the northern limb of the McQuesten Antiform and is underlain by Proterozoic to Lower Cambrian-age Hyland Group metasediments and the Dublin Gulch intrusion, a granodioritic stock. The Dublin Gulch Stock is comprised of four intrusive rock phases, the most significant of which is Granodiorite. The stock has been dated at approximately 93 Mega annum (Ma).

The metasediments are the product of greenschist-grade regional metamorphism. Proximal to the Dublin Gulch Stock, these metasediments have undergone metasomatism and contact metamorphism. A hornfelsic thermal halo surrounds the stock and within the halo, the metasediments have been altered to schist, marble and skarn

The Eagle and Olive zones belong to the RIRGS class (Reduced Intrusion-Related Gold Systems) of mineral deposits.

The Eagle Zone gold occurrence is localized at the narrowest exposed portion of the stock. The Eagle Zone mineralization is comprised of sub-parallel extensional quartz veins that are best developed within the Granodiorite.

Sulphides account for less than 5% of vein material and occur in the centre, on the margin, and disseminated throughout the veins. The most common sulphide minerals are pyrrhotite, pyrite, arsenopyrite, chalcopyrite, sphalerite, bismuthinite, molybdenite and galena. Secondary potassium feldspar is the dominant mineral in alteration envelopes. Sericite-carbonate is generally restricted to narrow vein selvedges, although alteration zones of this type also occur with no obvious relation to veins. Gold mineralization also occurs within the metasedimentary rock package immediately adjacent to the Granodiorite.

The Eagle Zone is the principal concentration of mineralization within the property. The Eagle Zone is irregular in plan and is approximately 1,600 m long (east-west) and 600 m wide north-south. The Eagle Zone is near-vertical and has been traced for about 500 m below surface. Current drilling indicates that the mineralization is relatively continuous along this length and is open in several directions, including at depth. Mineralization occurs as elemental gold, both as isolated grains and most commonly in association with arsenopyrite, and less commonly with pyrite and chalcopyrite. The sulphide content in the veins is typically less than 5%, and is less than 0.5% within the deposit overall, with 1 to 4% carbonate (calcite) present.

The Olive Zone gold occurrence is localized at the contact zone on the northwest flank of the Granodiorite intrusive, and located 2.5 km northeast of the Eagle Zone. Olive measures approximately 20 to 80 m in width, 900 m in length, and has been drilled to approximately 175 to 250 m in depth. Over 97% of the gold mineralization in the Olive Zone is hosted in Granodiorite.

Compared to Eagle, the Olive mineralization is more associated with sulphides and quartz-sulphide veining in an interpreted shear-zone setting. An oxidation zone and a transition zone, from near total oxidation to only sulphides, have been defined. Veins can be only sulphides or sulphides with white quartz. Pyrite plus arsenopyrite (or arsenical pyrite) and quartz-pyrite veins are common, within the overall NE trending zone of mineralization.

1.5 History, Exploration and Drilling

Exploration drilling for intrusive-hosted gold mineralization began in the early 1990's, and continued sporadically by several owners through 2004, including through StrataGold. Victoria Gold acquired StrataGold in 2009, and continued exploration drilling on the property. Since the 2012 Wardrop FS, the majority of Victoria Gold's exploration work has been in-fill drilling at the Eagle Zone, and exploration efforts including trenching, geophysical surveys and drilling at the Olive Zone. Post the 2012 Wardrop FS resource estimate, Victoria Gold conducted a targeted in-fill drilling program in the winter of 2011-2012, consisting of core and RC drilling of an additional 130 drill holes in the Eagle Zone. The purpose of the targeted in-fill drilling program was to better define Measured and Indicated Mineral Resources.

The Olive Zone had been explored prior to Victoria Gold's ownership, with initial drilling in 1992, and sporadic follow-up drilling for a total of 19 holes by 2007. Victoria Gold conducted additional drilling of 58 holes in 2010-2012, in-fill drilling of 61 holes in 2014, and an additional 89 drill holes in 2016.

The additional drilling allows the Olive Zone to now be defined as a Mineral Resource. Additional exploration work conducted at the Olive Zone included 17 shallow trenches in 2014 and 29 trenches in 2016, to expose and sample oxidized sulphide mineralization and help define the surface trace and extensions to mineralization. As well, a program of Internet protocol (IP)-Resistivity geophysical surveys was conducted over the core area of the Olive Zone in 2015. The results of the program conclude there is a good correlation of IP chargeability highs with the modelled zone of anomalous gold mineralization in drilling, and a direct association of the gold with increased sulphide content.

A summary of exploration drilling and trenching, for which sample analyses have been used for Mineral Resource estimation, are presented below for the Eagle Zone in Table 1.1 and the Olive Zone in Table 1.2.

Table 1.1: Summary of Annual Exploration Programs – Eagle Zone

Company	Year	Number of Holes	Metres Drilled	Type
Canada Tungsten	1977	65	11,315	DDH
Queenstake Resources	1986	4	705	DDH
Can Pro	1989	4	653	DDH
Ivanhoe Goldfields	1991	16	2,410	DDH
Amax Gold Inc.	1992	13	1,943	DDH
Amax Gold Inc.	1993	56	7,729	RC
Amax Gold Inc.	1993	10	1,476	DDH
Ivanhoe Goldfields	1993	10	2,078	RD
First Dynasty Mines	1995	40	8,354	RC
First Dynasty Mines	1995	25	4,946	DDH
New Millennium Mining	1996	21	4.114	DDH
New Millennium Mining	1996	37	5,271	RC
New Millennium Mining	1996	19	189	Auger
New Millennium Mining	1996	33	797	Water
StrataGold	2005	34	8,105	DDH
StrataGold	2006	10	4,282	DDH
StrataGold	2007	20	5,627	DDH
StrataGold	2008	15	4,429	DDH
Victoria Gold	2009	10	5,122	DDH
Victoria Gold	2009	4	1,321	Geotech
Victoria Gold	2010	20	3,592	DDH
Victoria Gold	2010	5	1,341	Geotech
Victoria Gold	2011	3	616	Geotech
Victoria Gold	2011-2012	33	4,337	RC
Victoria Gold	2011-2012	58	17,538	DDH
TOTAL		**565**	**104,180**	

Source: Wardrop 2012 FS, updated in 2016 from drill hole database

Table 1.2: Summary of Annual Exploration Programs – Olive Zone

Company	Year	Number of Holes/Trenches	Metres Drilled/Trenched	Type
Prior owners	1991, 1992	7	959	RC and DDH
Prior owners	2007	5	868	DDH
Prior owners	1989,2009	10	707	Trenches
Victoria Gold	2010	19	4,144	DDH
Victoria Gold	2011	24	4,486	DDH
Victoria Gold	2011	4	300	RC
Victoria Gold	2012	11	2,997	DDH
Victoria Gold	2014	61	8,594	DDH
Victoria Gold	2014	10	1,027	Geotech
Victoria Gold	2014	17	885	Trenches
Victoria Gold	2016	89	12,546	DDH
Victoria Gold	2016	34	1,025	Trenches
TOTAL DRILLING		**230**	**35,921**	
TOTAL TRENCHES		**61**	**2,671**	

Source: Wardrop 2012 FS, updated in 2016 from drill hole database

1.6 Metallurgical and Mineral Processing Test Results

Extensive metallurgical testing programs, including column leach, bottle roll leach, gravity concentration and flotation tests, were conducted on various composites from the Eagle Gold project. Comminution, compacted permeability, cyanide neutralization and humidity cell studies were also performed. A limited number of bottle roll leach and column leach tests were conducted on composites from the Olive project. The results from these test programs indicate that heap leaching is a viable processing method.

Two metallurgical testing programs were conducted on the Eagle Gold project: one before and one after the 2012 Wardrop FS. The results from the post-2012 study program confirmed the metallurgical criteria presented in the 2012 Wardrop FS.

Leach data on the Eagle Gold project composites crushed with a high-pressure grinding roll and with conventional cone crushers were compiled at several crush sizes. The results from the column leach test programs indicate that gold recovery is sensitive to crush size and ore type, and, to a lesser extent, crush type. Overall gold recoveries ranged from 68 to 79% at a P_{80} crush size of 6.4 mm.

Leach data on Olive oxide, transition and sulphide composites, crushed with conventional cone crushers to approximately 6.4 mm, were also compiled. Gold recoveries ranged from 54 to 68%.

Column leach tests were conducted on Eagle composites at freezing conditions, and the test results were compared to ambient temperature column leach tests. These tests yielded similar results as those conduced at ambient temperatures, but with generally lower sodium cyanide consumptions.

The column leach test results show that crushing to a P_{80} size of 6.4 mm with conventional crushers will lead to the field recoveries by ore type for both Eagle and Olive, as summarized in Table 1.3. The calculated gold recoveries include 3% and 2% point deductions for estimated field recoveries from laboratory data, for Eagle and Olive, respectively. Overall gold recoveries are dependent on the distribution of ore types as shown in Table 1.3.

The results of the test programs indicate low reagent requirements and moderate leach times for both Eagle and Olive. Projected reagent requirements and leach times are presented in Table 1.4.

Table 1.3: Summary of Gold Recovery by Ore Type

Field Gold Recovery (%)			
Eagle		Olive	
Weathered Granodiorite	79	Oxide	66
Fresh to Weakly Altered Granodiorite	73		
Seretic, Chloritic, Carbonate Altered Granodiorite	68	Transition	55
Weathered Metasediments	73	Sulphide	53
Unaltered Metasediments	68		

Source: KCA 2016

Table 1.4: Summary of Reagent Consumption Estimate

Description	Reagent Requirements (kg/t)		
	NaCN	Lime	Cement
Eagle - Overall	0.35	1.0	2.0*
Olive - Oxide	0.46		4.0
Olive - Transition	0.24		4.0
Olive - Sulphide	0.25		4.0

*Only used in the first year of each HLP
Source: KCA 2016

Estimated gold recoveries for Eagle for ROM, and coarse crush sizes were estimated from the size fraction data from column leach test results on finer crushed composites. ROM size distribution was estimated from a combination of data from an operating mine with similar geological characteristics and crusher simulation software. The primary crushed ore size distribution was estimated based on results from crusher simulation software.

No tests have been conducted on coarse composites from Eagle, and additional testing is required. Due to the limited data, the estimated recoveries for the ROM and primary (coarse) crushed ore were presented as ranges. These ranges are 50 to 55% for ROM ore, and 55 to 60% for primary (coarse) crushed ore. Lime and sodium cyanide requirements were estimated to be 1.0 kg/t and 0.35 kg/t, respectively, for both the ROM and primary crushed ore options. Maximum leach time of 150 days was estimated to be the same for both.

1.7 Mineral Resource Estimates

This 2016 FS update includes an update to the Mineral Resource estimate (MRE) for the Eagle Gold deposit, here called the Eagle zone, as previously described in the Wardrop 2012 FS, and an initial MRE for the Olive Zone gold mineralization, a satellite deposit that is located approximately 2.5 km northeast of the Eagle Zone.

The MRE has been classified as “Measured”, "Indicated" and "Inferred" according to the Canadian Institute of Mining and Metallurgy (CIM) "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (May 2014).

A geological model was used for each deposit, consisting of lithology (Granodiorite and metasediments), a mineralized shape defined from drill hole gold assays, and oxidation surfaces. Classical statistical and geostatistical evaluations were done, and data for both Eagle and Olive drill hole and trench assays were composited to 2.5 m. A block size of 10 m x 10 m x 5 m was used for each deposit with standard block modelling methods in Datamine software. Kriging was used for grade estimation, and data validations were completed.

The current Eagle Mineral Resources are reported as in-pit resources at a cut-off grade of 0.15 g/t Au.

The current Olive in-pit Mineral Resources are reported at a cut-off grade of 0.40 g/t Au. The cut-off is supported by the same parameters as for Eagle, but with lower recoveries. The Olive cut-off grade was selected to provide higher grade material for the project. Olive has a complete assay database for silver, whereas Eagle does not; thus Olive has silver reported as an associated element.

Mineral Resources for the Eagle Zone and Olive Zone are stated in Table 1.5.

Table 1.5: Constrained Eagle Mineral Resource Estimate* (inclusive of Mineral Reserves)

Classification	Quantity (Mt)	In-Situ Gold Grade (g/t)	In-Situ Silver Grade (g/t)	Contained Gold (koz)	Contained Silver (koz)
Eagle (0.15 g/t Au cut-off)					
Measured	29.4	0.81	na	761	na
Indicated	151.3	0.59	na	2,870	na
Combined	**180.7**	**0.63**	**na**	**3,631**	**na**
Inferred	17.4	0.49	na	276	na
Olive (0.40 g/t Au cut-off)					
Measured	2.0	1.19	2.31	75	146
Indicated	7.5	1.05	2.06	254	498
Combined	**9.5**	**1.08**	**2.11**	**329**	**645**
Inferred	7.3	0.89	1.70	210	402

Source: Qualified Persons F. Daviess, R. Sharma, and A. Moran; 2016

*Notes:

1. CIM definitions were followed for Mineral Resources
2. Mineral Resources are estimated at a cut-off of 0.15 g/t Au for Eagle and 0.40 g/t for Olive
3. Gold price used for this estimate was US$1,700/oz
4. High-grade caps were applied as per the text of this report
5. Specific gravity was estimated for each block based on measurements taken from core specimens
6. Resources are In-pit resources as defined by pit parameters described in the text of this report
7. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues
8. The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category

1.8 Mineral Reserve Estimate

The mineral reserve for the property is based on the mineral resource estimate for Eagle and Olive.

The Mineral Reserves were developed by examining each deposit to determine the optimum and practical mining method. Cut-off grades (COGs) were then determined based on appropriate mine design criteria and the adopted mining method. A shovel and truck open pit mining method was selected for the various deposits.

The estimated Proven and Probable Mineral Reserves total 123 Mt at 0.67 g/t Au, containing 2,663 koz Au (Table 1.6 and Table 1.7).

Table 1.6: Mineral Reserve Estimate by Deposit

Area	Classification	Ore (Mt)	Diluted Gold Grade (g/t)	Contained Gold (koz)
Eagle	Proven	27	0.80	685
	Probable	90	0.62	1,778
	Total	**116**	**0.66**	**2,463**
Olive	Proven	2	1.02	58
	Probable	5	0.93	142
	Total	**7**	**0.95**	**200**
Eagle + Olive	**Total**	**123**	**0.67**	**2,663**

Note: Mineral Reserves are included within Mineral Resources

Source: JDS (2016)

Table 1.7: Mineral Reserve Estimate per Ore Type

Type	Area	Ore (Mt)	Diluted Gold Grade (g/t)	Contained Gold (koz)
Crushed Ore	Eagle	101	0.72	2,330
	Olive	7	0.95	200
	Total	**108**	**0.73**	**2,530**
ROM Ore	Eagle	15	0.27	133
	Olive			
	Total	**15**	**0.27**	**133**
Crushed + ROM	**Total**	**123**	**0.67**	**2,663**

Note: Mineral Reserves are included within Mineral Resources

Source: JDS (2016)

The COGs for Eagle and Olive by ore type are listed in Table 1.8.

Table 1.8: Eagle and Olive COGs by Ore Type

Area/Type	Material	Cut-off Grade (g/t Au)
Eagle Crush	Oxide Granodiorite	0.19
	Altered Granodiorite	0.21
	Unaltered Granodiorite	0.22
	Oxide Medasediments	0.21
	Unaltered Medasediments	0.22
Olive Crush	Oxide	0.24
	Mixed	0.29
	Sulphide	0.30
Eagle ROM	All	0.23

Source: JDS (2016)

The mineral reserve estimations take into consideration on-site operating costs (mining, processing, site services, freight, general and administration), geotechnical analysis for open pit wall angles, metallurgical recoveries, and selling costs. In addition, the Mineral Reserves incorporate allowances for mining recovery and dilution, and overall economic viability.

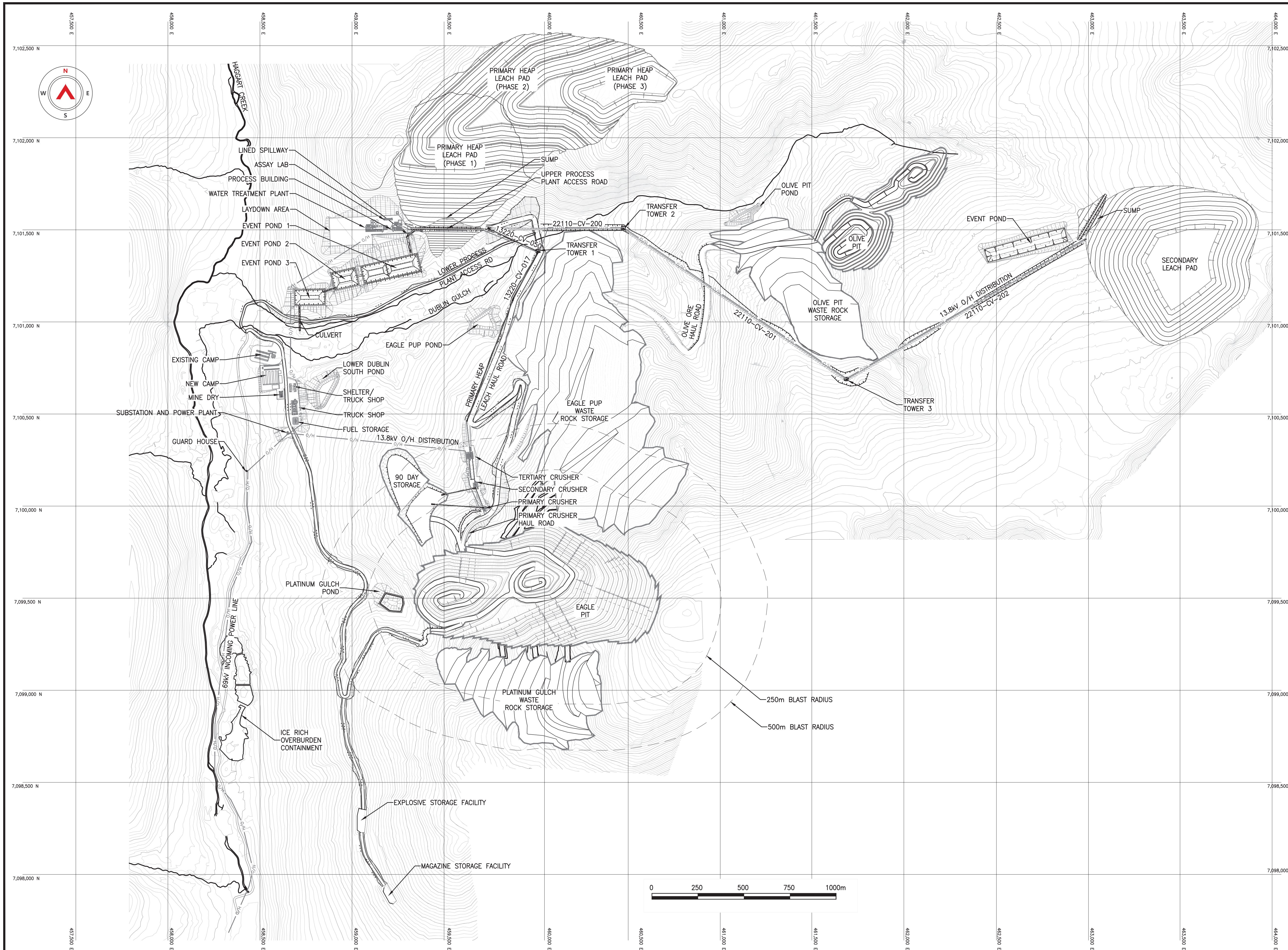
1.9 Mining

The Eagle and Olive deposits are planned to be mined using OP methods, and will operate as drill, blast, shovel and haul operations with a combined nominal rate of 33,700 t/d ore and a mine life of 10 years. Mining of the deposit is planned to produce a total of 123 Mt of the heap leach feed and 116 Mt of waste (at a 0.95:1 overall strip ratio). Ore to be crushed will be hauled to the primary crusher located towards the north-east side of the Eagle pit. ROM ore will be hauled directly to the primary HLP. ROM material will only come from the Eagle pit.

1.9.1 Open Pit Mine Plan and Phasing

The proposed overall site layout for the project, including the open pits at Eagle and Olive, various waste rock facilities, the heap leach facilities (both primary and secondary pads), and the plant site location is illustrated in Figure 1.2.

The mine design process for the deposits commenced with the development of open pit mine planning optimization input parameters. These parameters included estimates of metal price, mining dilution, process recovery, off-site refining costs, geotechnical constraints (pit slope angles) and royalties.



NOTES:

1. ALL ELEVATIONS ARE IN METERS.
2. CONTOUR DATA RECEIVED FROM UNDERHILL (FEBRUARY 2011) UTM NAD83.
3. 5m CONTOUR INTERVALS SHOWN.
4. 13.8kV O/H DISTRIBUTION TO FOLLOW ALL CONVEYORS EXCEPT 13220-CV-017

REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
B	16/10/19	ISSUED FOR FEASIBILITY UPDATE	JLC	NV
A	16/10/04	ISSUED FOR FEASIBILITY UPDATE	JLC	NV

CLIENT:

CLIENT NO:	#	DRWN:	JLC	DATE:	16/10/19
PROJECT NO:	16VA0008	DSGN:	SK/JLC	DATE:	16/10/19
DRAWING SIZE:	22 X 34	CHKD:	NV	DATE:	16/10/19
SCALE:	1:10,000	APVD:	SK	DATE:	16/10/19

PROJECT:

EAGLE GOLD FEASIBILITY STUDY

TITLE:

GENERAL ARRANGEMENT OPERATIONAL

DWG NO: **16VA0008-000-1011-002**

REV: **B**

Date: 2016/10/19 | User: Jessica Currie | File: C:\Users\jbaxter\Desktop\1000-Drawings - Eagle Gold\1011-Civil\01-Production\16VA0008-000-1011-Production | Layout: 002 | Paper Size: 863.6mm x 558.8mm

The current life of mine (LOM) plan focuses on achieving consistent heap leach production rates, mining of higher value material early in the production schedule, as well as balancing grade and strip ratios. Pit optimizations and analyses were conducted to determine the optimal mining shells. Detailed pit and phase designs were then generated for Eagle and Olive and mine planning and scheduling was conducted on these detailed pit designs.

1.9.2 Mine Schedule and Operations

The various pit designs for the project deposit were divided into phases (or pushbacks) for the mine plan development in order to provide flexibility in the schedule, maximize value, reduce pre-stripping requirements in the early years, and maintain the crush heap leach at full production capacity. The project deposits are most economical when open pit phases at Eagle are mined concurrently.

The open pits at Eagle and Olive are projected to provide the heap leach facilities at a combined nominal rate of 33,700 t/d ore over a of 10-year period (excluding the initial construction period). The open pit mining is envisioned to be undertaken by the Owner. Annual mine production of ore and waste is profiled to peak at 30 Mt/a, with a LOM waste to ore stripping ratio of 0.95:1. Given that the secondary and tertiary crushers and HLP will only be operated between April and December of each year, stockpiles will be used when necessary for stockpiling of ore from the open pit. The handling of the ore from the crusher to the HLPs is included in the open pit scheduling and operating cost estimation. Table 1.9 summarizes the LOM material movement by year for both the mine and the heap leach facilities.

Table 1.9: Summary of LOM Production Schedule by Year

Description	Unit	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Crushed Ore	Mt	**107.8**	0.0	8.8	11.0	10.9	10.9	10.9	11.0	11.0	10.9	10.9	11.0	0.5
Crushed Gold Grade	g/t	**0.73**	0.49	0.75	0.81	0.77	0.78	0.80	0.71	0.62	0.61	0.62	0.83	0.75
Crushed Contained Gold	koz	**2,530**	0	212	287	272	275	282	251	218	213	218	291	12
ROM Ore	Mt	**15.1**	0.0	1.1	1.6	1.5	0.4	1.4	1.7	2.2	1.8	2.5	0.8	0.0
ROM Gold Grade	g/t	**0.27**	0.29	0.27	0.28	0.27	0.28	0.27	0.27	0.28	0.27	0.27	0.28	0.00
ROM Contained Gold	koz	**133**	0	10	15	13	3	12	15	20	16	22	7	0
Waste	Mt	**116.3**	2.1	6.9	14.1	12.1	15.3	10.0	7.7	7.8	9.3	16.3	14.5	0.2
Total Ore	**Mt**	**122.9**	**0.0**	**9.9**	**12.6**	**12.5**	**11.3**	**12.3**	**12.7**	**13.2**	**12.8**	**13.5**	**11.7**	**0.5**
Total Gold Grade	**g/t**	**0.67**	**0.42**	**0.70**	**0.74**	**0.71**	**0.77**	**0.74**	**0.65**	**0.56**	**0.56**	**0.55**	**0.79**	**0.75**
Total Contained Gold	**koz**	**2,663**	**0**	**222**	**301**	**285**	**279**	**294**	**266**	**238**	**229**	**240**	**298**	**12**
Strip Ratio	w:o	**0.95**	83.49	0.70	1.12	0.97	1.35	0.81	0.61	0.59	0.73	1.21	1.23	0.47
Total Material	Mt	**239.2**	2.1	16.8	26.7	24.6	26.6	22.4	20.4	21.0	22.1	29.8	26.2	0.7
Total Mined	Kt/d		7.6	45.9	73.0	67.4	73.0	61.3	55.9	57.4	60.4	81.6	71.8	3.9
Heap Leach Schedule														
Total Recovered Gold	koz	**1,884**	0	142	208	213	213	210	192	166	160	162	184	35

Source: JDS (2016)

Mining will begin in Year -1 at Eagle pit to provide waste rock for construction and allow for access roads to be built. Leachate processing will commence in Q2 Year 1 and ramps up to full production in Year 2. Open pit mining will focus on the various Eagle pit phases with the smaller Olive pit coming into production in Year 9. Open pit mining and loading of the heap leach facilities will be completed in Q1 of Year 11.

Open pit mining operations will use a fleet comprising 22 m^3 front shovels, 12 m^3 front-end loaders and 144 t haul trucks. This fleet will be supplemented by drills, graders, and track and rubber tire dozers. A 10 m bench height was selected for mining in ore and waste with overall 20 m effective bench heights based on a double-bench final wall configuration.

1.9.3 Waste Management

In order to minimize haul distances, Eagle waste rock will be hauled to one of two waste rock storage areas immediately to the south and north of the open pit. Olive waste rock will be hauled to a waste rock storage area immediately south-west of the open pit. Total waste material produced is 116 Mt.

1.10 Recovery Methods

Two heap leach pads (HLP), the primary and secondary pads, will be developed to extract gold from ore into solution. The secondary pad will be developed in time to receive ore in Year 7 of operations. Both crushed and ROM ore will be stacked on the primary pad, while only crushed ore will be placed on the secondary pad.

Crushed ore will be fed through a three-stage crushing plant to produce an 80% passing (P_{80}) 6.5 mm product. ROM ore will bypass the crushing plant.

Gold will be leached with cyanide solution and recovered by an ADR carbon plant. The process flowsheet and design criteria are based on a heap leach processing rate of:

- 11 M dry tonnes per annum (Mt/a) of crushed ore with a LOM average gold feed grade of 0.73 g/t and a recovery of 70.9; and
- 1.5 Mt/a (on average) of ROM ore at a grade of 0.27 g/t and 55% gold recovery.

The process plant will be located near the primary HLP to minimize pumping and pipeline requirements for both pregnant and barren solutions during the first seven years of operation.

The HLPs will undergo year-round leaching with the stacking of ore occurring 275 days per annum (d/a). The stacking rate will be approximately 45,500 t/d (crushed plus ROM ore).

1.10.1 Ore Crushing, Handling and Stockpiling

1.10.1.1 Crushed Ore

Ore above 0.35 g/t from the Eagle pit will be sent to a three-stage crushing plant. The crushing circuit will consist of one 375 kW primary gyratory crusher, one 933 kW secondary cone crusher and three, parallel 933 kW tertiary cone crushers. Crushing plant feed material, with a maximum top size of 1,000 mm, will be trucked from the open pits and dumped directly into the primary gyratory crusher at a planned throughput of approximately 30,000 t/d. The primary crusher will operate 365 d/a, while the secondary and tertiary crushers will only operate 275 d/a when ore is stacked on the HLPs.

From Q2 through Q4 of each year, stockpiled crushed ore will be reclaimed via a loader/hopper/conveyor system to the secondary crusher. Crushed ore reclaiming will be done at 490 tonnes per hour (t/h), and combined with the primary crusher discharge, at a rate of 1,500 t/h, for a total feed rate of 1,990 t/h, or 39,800 t/d, to the secondary and tertiary crushing circuits. The tertiary product, screen undersize at P_{80} of 6.5 mm, will feed a series of conveyors and grasshopper conveyors to a radial stacker on the HLP. Lime and cement will be added to the tertiary screen discharge conveyor for pH control and agglomeration. Agglomeration takes place for all Olive ore and during the first year of stacking on the primary and secondary HLPs.

1.10.1.1.1 ROM Ore

ROM ore (less than 0.35 g/t but above the cut-off grade of 0.22 g/t) will be sent directly from the Eagle pit to the primary HLP during the stacking months, and to the ore stockpile during the stockpiling months (January to March). The ROM ore will be reclaimed from the stockpile, using a loader and trucks, and taken to the primary HLP. The ROM ore will be segregated from the crushed ore, but will be placed within the overall primary HLP.

1.10.1.1.2 Stockpiling

An ore stockpile area with a capacity of approximately 3.0 Mt will be established to allow the stockpiling of ore on a temporary basis during the coldest winter months (Q1 of each year). Crushed ore stockpiled during the winter months will be placed on the stockpile after passing through the primary crusher only. ROM ore will also be stacked on the stockpile during Q1.

1.10.2 Heap Leach Pad

The proposed primary HLP will accommodate approximately 77 Mt of ore and is planned to be located approximately 1.2 km north of the Eagle Zone orebody, in the Ann Gulch valley. The base of the primary HLP is planned to be located at an elevation of 880 metres above sea level (masl), and at full height, the primary HLP will extend up Ann Gulch to an elevation of approximately 1,225 masl at the top of the planned ore stack.

The proposed secondary HLP will commence in Year 7 and will accommodate the remaining estimated 46 Mt of ore (with expansion potential), and is planned to be located approximately 3 km east of the Eagle Zone orebody near the Olive Zone pit. The base of the secondary HLP is planned to be located in the upper portion of the basin at an elevation of 1,300 masl, and at full height, the secondary HLP will extend to an elevation of approximately 1,470 masl at the top of the planned ore stack.

The primary and secondary HLPs will each comprise a number of elements: a confining embankment to provide stability to the base of the HLP and a sump for operational in-situ storage of process solution, a lined storage area for the ore to be leached, pumping wells for the extraction of solution, a lined events pond to contain excess solution in extreme events, upstream surface water interceptor ditches, and leak detection, recovery and monitoring systems to ensure the containment of solution.

The HLP will be irrigated with a barren cyanide-caustic solution fed from the process plant through pipelines and drip emitters incorporated in the HLP. The barren solution will percolate through the HLP and dissolve gold producing a gold-bearing “pregnant” solution. The pregnant solution will be pumped from the HLP at a nominal rate of 2,070 m^3/h to the carbon adsorption circuit. The flowrate of barren solution is based on a 90-day leach cycle assuming an application rate of 10 $l/h/m^2$ and a lift height of 10 m.

1.10.3 Processing Plant

The pregnant solution will enter the ADR plant through the carbon adsorption circuit, which is planned to consist of two trains of five cascading-flow carbon columns. The barren solution discharged from the final carbon column will be pumped to the barren solution tank. Cyanide solution, liquid caustic and anti-scalant will be added to the barren solution to maintain the required pH and cyanide concentrations for leaching.

Loaded carbon will be extracted from the first carbon adsorption column at a rate of 8 t/d and will be acid washed prior to advancing to the desorption circuit for gold recovery in the strip vessel.

The pregnant solution from the strip vessel will flow to the electrowinning circuit. At the conclusion of the strip cycle, the stripped carbon will be thermally regenerated in the carbon reactivation kiln and then returned to the carbon columns.

Gold will be plated onto steel wool cathodes in the electrowinning cells. The gold-bearing sludge and steel wool will be dried, mixed with fluxes and then smelted to produce gold doré.

1.10.4 Gold Recovery

A summary of the throughput and gold recovery for each ore type are presented in Table 1.10.

Table 1.10: Throughput and Gold Recovery

	Units	Total
Eagle Crushed Ore		
Total Throughput	Mt	**101**
Gold Recovered From Heap Leach	koz	**1,697**
Gold Recovery	**%**	**72.9**
Eagle ROM Ore		
Total Throughput	Mt	**15**
Gold Recovered From HL	koz	**73**
Gold Recovery	%	**55.0**
Olive Ore		
Total Throughput	Mt	**7**
Gold Recovered From HL	koz	**113**
Gold Recovery	%	**56.8**
Total Recovery		
Total Throughput	Mt	**123**
Gold Recovered From HL	koz	**1,884**
Gold Recovery	%	**70.8**

Source: JDS (2016)

A gold production model was developed to predict the gold production from the heap leach operation and is based on a combination of metallurgical testing data, the mine production schedule, the HLP construction sequence (or stacking plan), and the leaching (irrigation) plan for the application of barren solution.

The gold production model uses cells that are approximately 300,000 t or a little over a week's worth of stacking. For each cell, a weighted average head grade and recovery based on the rock type was calculated to determine the gold recovered over time. Each year, over the 9-month stacking period, 36 cells are loaded. Barren solution is added to the active cells 365 d/a.

With an application rate of 10 l/h/m^2 the ultimate recovery per cell was calculated over time using the column leach test work. The gold recovery was calculated for each rock type to produce an average recovery of solution per tonne of ore versus days of leach. At 90 days, the gold recoverable from each cell is approximately 90% complete.

During the winter, when ore stacking onto the HLPs is stopped, the barren solution will still be applied to the HLP, and gold recovery from ore will continue.

During initial leaching of the primary and secondary pads, there will be an in-process inventory of recoverable gold built up. The inventory of recoverable gold will be from the recoverable gold in the ore in the heaps that has not been leached to completion, and is contained in solution inventories, carbon, and in the electrowinning/refining circuit that has not yet been processed into doré.

Gold inventory in solution, carbon and the electrowinning/refining circuit will ultimately be recovered in the final year of leaching.

Solution will continue to be added in Year 7 for the primary pad and in Year 11 for the secondary pad, to allow the last ore stacked to be leached to completion.

1.11 Infrastructure

The project is currently accessible year-round by road from the village of Mayo, YT. To support the additional construction and operation traffic, 23 km of the Haggart Creek site access road will receive minor upgrades.

New site ancillary facilities will include a camp expansion, truck shelter, fuel depot, explosive plant and storage and water management facilities. Existing structures including the 100-person camp, administration building and warehouse will be used for the permanent operation. Victoria Gold recently purchased a second 110-person camp close to site, and it was assumed this camp will be moved to site prior to construction.

Electric power will be sourced from the Yukon Energy Corp. grid via a new 44 km long, 69 kV transmission line as well as an on-site substation and distribution. Diesel-powered generators will be used for emergency power back-up.

Mining and processing facilities to be constructed are the crushing plant, the ADR plant, an assay lab, a mine water treatment plant (MWTP), HLPs, waste rock storage area (WRSAs), water management structures, and haul roads.

The existing placer tailings in the Dublin Creek Valley and waste rock from mine pre-stripping will provide all the aggregate needed for the project. A concrete batch plant will be mobilized to the site. A temporary crushing plant will be used for the production of concrete aggregates and HLP over-liner.

1.12 Project Execution Plan

The project has a 15-month construction period comprised of 12 months of Year -1 and three months of Year 1, when the open pit will be operational and ore stockpiled, while the secondary and tertiary crushing plants are completed and commissioned.

In Q1 of Year -1, engineering and procurement will begin. Long lead items may have to be ordered in Year -2, depending on availability at the time.

Camp expansion, early mobilization and earthworks will begin in Q2 of Year -1, with the primary focus of establishing the start of the primary HLP before Q4, when weather will become unfavourable for liner and over-liner placement. All construction work will be complete by the end of Q4 of Year -1, with the exception of the secondary and tertiary crushing plants, which will be complete by the end of Q1 of Year 1, when the first stacking of the primary HLP begins.

The permanent truck shop will be deferred until Year 1, and the secondary HLP and Olive pit will be built in Years 6 and 9 respectively.

1.13 Environment and Permitting

The Eagle Gold project has been assessed under the Yukon Environmental and Socio-economic Assessment Act (YESAA) and currently holds a Quartz Mining License (QML) and a Water Use License (WUL) to construct, operate and close the project.

As discussed in Section 1.5, the project area has an extensive exploration history involving a number of prior operators, some of whom had undertaken the collection of baseline environmental, socio-economic, land use, and heritage data. In 2007, StrataGold (a now wholly owned-directly held subsidiary of Victoria Gold) re-initiated the collection of environmental baseline data, which includes the disciplines of climate, water quality, hydrology, hydrogeology, aquatic biota, wildlife, and vegetation. Fieldwork to characterize climatic, hydrological, hydrogeological, and water quality conditions is ongoing.

Victoria Gold and prior operators have also characterized local and regional land use and socio-economic conditions, First Nations land use and activities, and archaeological and heritage resources.

Prior to construction or operational activities taking place, mining projects in the Yukon are required to undergo an assessment of potential project effects pursuant to YESAA. The YESAA process mandates that an applicant describe the scope of the project, the existing environmental and socio-economic setting, potential environmental and socio-economic effects of the project, and the measures that will be instituted by the applicant to mitigate those effects. The applicant also has a statutory obligation to consult any First Nation or resident of any community residing in the territory in which the project will be located or might have significant environmental or socio-economic effects.

This duty to consult the parties must be completed to the satisfaction of the Yukon Environmental and Socio-economic Assessment Board (YESAB), based upon their consideration of any submitted material and discussions with the parties, before the formal review of a project may commence.

The YESAA review process results in a recommendation by the YESAB to federal, territorial or First Nation governments or agencies that will regulate or permit the proposed activity for measures to reduce, control or eliminate project effects. These governments or agencies, referred to as Decision Bodies, will then decide whether to accept, reject, or vary the YESAB's recommendation in a final Decision Document. Upon receipt of positive final Decision Documents by the Decision Bodies, a project may then proceed to the licensing phase.

Mining projects in the Yukon require permits and approvals issued pursuant to various federal and territorial legislation. The major regulatory approvals that must be received for a mining project during the licensing phase are generally a QML, under Section 135 of the Yukon's *Quartz Mining Act*, and a WUL, under Sections 6 (1) and 7 (1) of the *Waters Act* (Yukon).

The Eagle Gold project has successfully completed the YESAA environmental assessment resulting in a positive final Decision Document in 2013. Victoria Gold subsequently applied for and received a QML and a Type A WUL for the construction, operation and closure of the project.

Collectively the QML and WUL currently allow for:

- The extraction of 92 Mt of ore from the Eagle open pit;
- The construction of the Ann Gulch HLP;
- The development of two WRSAs immediately adjacent to the pit for the permanent storage of 132 Mt of waste rock;
- The construction and operation of crushing and conveying infrastructure;
- The construction and operation of an ADR plant;
- The development of site haul roads; and,
- The construction and operation of all water management infrastructure required for mine and waste water treatment and for the extraction and/or conveyance of water required for processing.

Victoria Gold is able to begin the construction of the above facilities and undertake the associated activities immediately upon posting a bond, providing issued for construction drawings, and satisfying other minor requirements.

Project components not currently included in the QML or WUL include the Olive pit and secondary HLP. Both will need to undergo a review pursuant to YESAA and an amendment to each license. Victoria Gold has estimated permitting of these additional elements can be completed within three years.

The secondary HLP is not required until Year 7 of operations and the Olive pit has been scheduled for development in Year 8 of operations. This provides sufficient time to permit both the secondary pad and the Olive pit, well in advance of intended development, to ensure no interruption risk of operations.

1.14 First Nations' Considerations

The project is located entirely within the Traditional Territory of the First Nation of Na-Cho Nyäk Dun (FNNND). The statutory requirement to consult on the project and to satisfy previous, and any future, assessments of the project under the YESAA, essentially involves the FNNND. To ensure that the FNNND, and the community of Mayo, have an opportunity for input at all key stages of project development, Victoria Gold has made it a priority to conduct early and ongoing consultation with the FNNND, and the community of Mayo, to ensure opportunities for input from both parties at all key stages of project development.

On October 17, 2011, Victoria Gold and the FNNND signed a comprehensive Cooperation and Benefits Agreement (CBA). The CBA replaced an earlier Exploration Cooperation Agreement and applies to the Eagle Gold mine development and exploration activities conducted by Victoria Gold anywhere in the FNNND Traditional Territory south of the Wernecke Mountains.

The objectives of the CBA are to:

- Promote effective and efficient communication between Victoria Gold and the FNNND in order to foster the development of a cooperative and respectful relationship and FNNND support of Victoria Gold's exploration activities on the project;
- Provide business and employment opportunities, related to the project, to the FNNND and its citizens and businesses in order to promote their economic self-reliance;
- Establish a role for the FNNND in the environmental monitoring of the project and the promotion of environmental stewardship;
- Set out financial provisions to enable the FNNND to participate in the opportunities and benefits related to the project; and
- Establish a forum for Victoria Gold and the FNNND to discuss matters related to the project and resolve issues related to the implementation of the CBA.

1.15 Capital Cost Estimates

The capital costs for the mine, process plant, power line and infrastructure for the Eagle Gold project has been prepared in accordance with standard industry practices for this level of study, and to a level of definition and intended accuracy of ±15%.

There are four main parts of the cost estimate: direct costs, indirect costs, contingency and Owner's costs. Owner's costs were estimated separately by Victoria Gold with support from JDS.

The initial capital cost estimate is $370M and the sustaining cost estimate is $183M for a total of $553M expressed in Canadian dollars with no escalation (Q3-2016). Taxes are not included in the cost estimate. The capital cost summary and distribution is shown in Table 1.11.

Table 1.11: Summary of Capital Cost Estimate

Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
Mining & Pre-Production Development	35	46	80
Site General	18	10	28
Process	101	0	101
Ancillaries	22	30	53
Power Supply & Distribution	15	1	16
Water Management	6	15	21
Heap Leach	56	82	138
Owner's Costs	9	0	9
Indirect Costs	73	0*	73
Subtotal	**334**	**183**	**518**
Contingency (10.5%)	35	-	35
Total Capital Costs	**370**	**183**	**553**

*Sustaining capital indirect costs are included in direct costs for each area.
Source: Merit 2016

The closure cost of $35M net of salvage value was estimated and is not included in Table 1.11.

1.16 Operating Cost Estimates

The operating cost estimate (OPEX) is based on a combination of experience, reference projects, first principle calculations, budgetary quotes and factors as appropriate for a FS.

The total LOM costs are summarized in Table 1.12.

Table 1.12: Summary of Operating Cost Estimate (excluding Costs capitalized in Pre-Production)

Area	Unit Cost ($/t leached)	Unit Cost (US$/payable oz Au)	LOM Cost (M$)
Mining*	4.19	214	515
Processing	4.93	252	606
G&A	1.42	73	175
Total	**10.54**	**539**	**1,295**

*Average LOM open pit mining cost amounts to $2.17/t mined (excluding pre-production tonnes) at a 0.95 strip ratio.
Source: JDS (2016)

1.17 Economic Analysis

An economic model was developed to reflect projected annual cash flows and sensitivities of the project. All costs, metal prices and economic results are reported in Canadian dollars (C$ or $) unless stated otherwise.

Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

1.17.1 Results

The parameters used in the economic model and the results are shown in Table 1.13.

Table 1.13: Economic Results

Parameter	Unit	Value
Au Price	US$/oz	1,250
Exchange Rate	US$:C$	0.78
After-Tax Free Cash Flow	M$	714
	Avg M$/yr	71
Pre-Tax $NPV_{5\%}$	M$	778
Pre-Tax IRR	%	37.1
Pre-Tax Payback	Yrs	2.6
After-Tax $NPV_{5\%}$	M$	509
After-Tax IRR	%	29.5
After-Tax Payback	Yrs	2.8

Note: NPV = Net Present Value
IRR = Internal Rate of Return
Source: JDS (2016)

1.17.2 Timing of Revenues and Working Capital

Working capital has been considered in the economic analysis based on the assumption that the project will be cash flow positive in Q2 of Year 1. It accounts for the equivalent of expected operating costs for Q1 of Year 1, sustaining capital incurred in Q1 of Year 1 and parts and consumables inventory for Q2 of Year 1. Working capital amounts to $27M. The working capital is recuperated in two installments: $24M in Q2 of Year 1, and the remaining $3M during the last year of heap production in Year 11.

1.17.3 Sensitivities

Sensitivity analyses were performed using gold price, FX Rate, head grade, capital cost estimate (CAPEX), and operating cost estimate (OPEX) as variables. The value of each variable was changed plus and minus 15% independently, while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.14.

Table 1.14: Sensitivities Analyses

Variable	Pre-tax $NPV_{5\%}$ (M$)			After-tax $NPV_{5\%}$ (M$)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Metal Price	425	778	1131	287	508	728
FX Rate	1190	778	474	764	508	318
Head Grade	428	778	1128	289	508	726
OPEX	930	778	626	603	508	414
CAPEX	859	778	697	590	508	427

Source: JDS (2016)

1.18 Conclusions

The FS summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used.

The Eagle Gold project contains a substantial resource that can be mined by open pit methods and recovered with heap leach processing.

Based on the assumptions used for this preliminary evaluation, the project is economic and should proceed to the detailed engineering stage and ultimately construction. Engineering and construction costs are included in the CAPEX.

Initiatives which may further enhance project economics include:

- Year-round stacking;
- Continued near-mine exploration with a focus on the Potato Hills Trend, which hosts the Olive, Shamrock and other targets;
- Conversion of Inferred Mineral Resources to Indicated Mineral Resources, particularly at depth, to increase reserve potential and decrease waste;
- Further refinement of water management and water treatment to reduce costs; and
- Used mobile and stationary equipment.

The most significant potential risks associated with the project are, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, ability to raise financing, gold price and exchange rate. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

2 Introduction

2.1 Basis of Feasibility Study

This FS report was compiled by JDS for Victoria Gold. This technical report summarizes the results of the FS and was prepared to support the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

2.2 Scope of Work and Responsibilities

This report summarizes the work carried out by each company is listed below, and combined, makes up the total project scope.

JDS Energy & Mining Inc. (JDS) scope of work included:

- FS project management;
- Compilation of the report, including data and information provided by other consulting companies;
- Reserve estimation and mine planning;
- Development of a conceptual flowsheet, specifications and selection of leach process equipment;
- Ore crushing and handling;
- ADR process plant;
- Design and location of on-site infrastructure;
- Environmental permitting and community relations review;
- Water management review;
- Mining CAPEX and all OPEX estimation;
- Preparation of a financial model to enable economic evaluation; and
- Interpretation of results and recommendations to improve value and reduce risks.

Merit Consultants International

- CAPEX estimation; and
- Project Execution Plan.

SRK Consulting (U.S.) Inc. (SRK) scope of work included:

- Geotechnical assessment and design of open pits; and
- Geotechnical assessment of ground conditions for waste rock storage areas.

Allan V. Moran Consulting scope of work included:

- Project setting, history and geology description.

Ravindra K. Sharma scope of work included:

- Eagle mineral resource estimate.

Frank Daviess scope of work included:

- Olive mineral resource estimate.

Kappes, Cassiday & Associates (KCA) scope of work included:

- Implementation and supervision of the metallurgical testing program;
- Establishment of gold recovery values based on metallurgical testing results.

DOWL Engineering (DOWL) scope of work included:

- Design and construction methodology of the heap leach facilities and event ponds.

Allnorth Consultants Ltd. Ltd. (Allnorth) scope of work included:

- Site infrastructure engineering;
- Detailed engineering of processing facility; and
- Final alignment and design of site access road.

W.M. Brazier Associates Inc. (**Brazier**) scope of work included

- Power supply, including transmission line, main substation and back-up generation power engineering and capital cost estimation.

2.3 Qualified Person Responsibilities and Site Inspections

The Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, Mineral Resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in Victoria Gold and neither are they insiders, associates, or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Victoria Gold and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions. The QPs are responsible for specific sections as follows in Table 2.1.

Table 2.1: Qualified Person Responsibilities

Qualified Person	Company	Report Sections of Responsibility
Gord Doerksen, P.Eng.	JDS Energy & Mining Inc.	1, 2, 3, 4, 5, 6,18.5, 18.6, 18.7, 18.8, 19, 20, 22, 23, 24, 26, 27, 28 and 29
Jay Collins, P.Eng.	Merit Consultants	21, 25
Allan Moran, P. Geo.	Allan V. Moran Consulting LLC	7, 8, 9, 10, 11, 12
Dino Pilotto, P.Eng.	JDS Energy & Mining Inc.	15, 16 (except 16.3, 16.4)
Kelly McLeod, P. Eng.	JDS Energy & Mining Inc.	17 (except 17.2.4 and 17.2.6.3 to 17.2.6.6)
Troy Meyer, P.E.	DOWL Engineering	17.2.4 and 17.2.6.3 to 17.2.6.6
Michael Levy, P.E.	SRK Consulting (U.S.) Inc.	16.3., 16.4
Rui Adanjo, P.Eng.	Allnorth Consultants Ltd.	18.4.5, 18.4.6
Farhad Riahi, P.Eng.	Allnorth Consultants Ltd.	18.1, 18.2, 18.3
Carl Defilippi, RM SME	Kappes, Cassiday & Associates	13
Ravindra K. Sharma, MAusIMM, RM SME	Independent	14.1-14.11 and 14.23
Frank Daviess, RM SME	Independent	14.12-14.23
Neil Brazier, P.Eng.	W.N. Brazier Associates	18.4.1, 18.4.2, 18.4.3 & 18.4.4

QP visits to the Eagle Gold property were conducted as follows:

- Gordon Doerksen, Dino Pilotto, Allan Moran, Farhad Riahi, and Michael Levy visited the project site on May 26, 2016. Allan Moran also visited the site on September 22 and 23, 2011.
- Jay Collins visited the project site on August 22 and 23, 2011.
- Neil Brazier visited the project site from May 25 to 27, 2016.
- Troy Meyer visited the site from September 15 to 17, 2016.
- Kelly McLeod, Carl Defilippi, Ravindra Sharma, Frank Daviess, Rui Adanjo did not visit the project site and relied on the other QPs for their information.

2.4 Sources of Information

The sources of information include data and reports supplied by Victoria Gold personnel as well as documents cited throughout the report and referenced in Section 28. In particular, background project information was taken directly from the most recent technical report entitled "Eagle Gold Project Feasibility Study" prepared by Wardrop Engineering Inc. (Wardrop 2012), with an effective date of April 18, 2012.

2.5 Units, Currency and Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort has been made to clearly display the appropriate units being used throughout this technical report. Currency is in Canadian dollars (C$ or $) unless otherwise stated.

This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3 Reliance on Other Experts

The QPs opinions contained herein are based on information provided by Victoria Gold and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice are:

- Wentworth Taylor, CPA for taxation guidance;
- Victoria Gold for environment, permitting and water treatment guidance;
- Brad Thrall of Access Consulting provided oversight and review of metallurgical test work, crushing, HLP, the ADR process and gold recovery model.

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending.

4 Property Description and Location

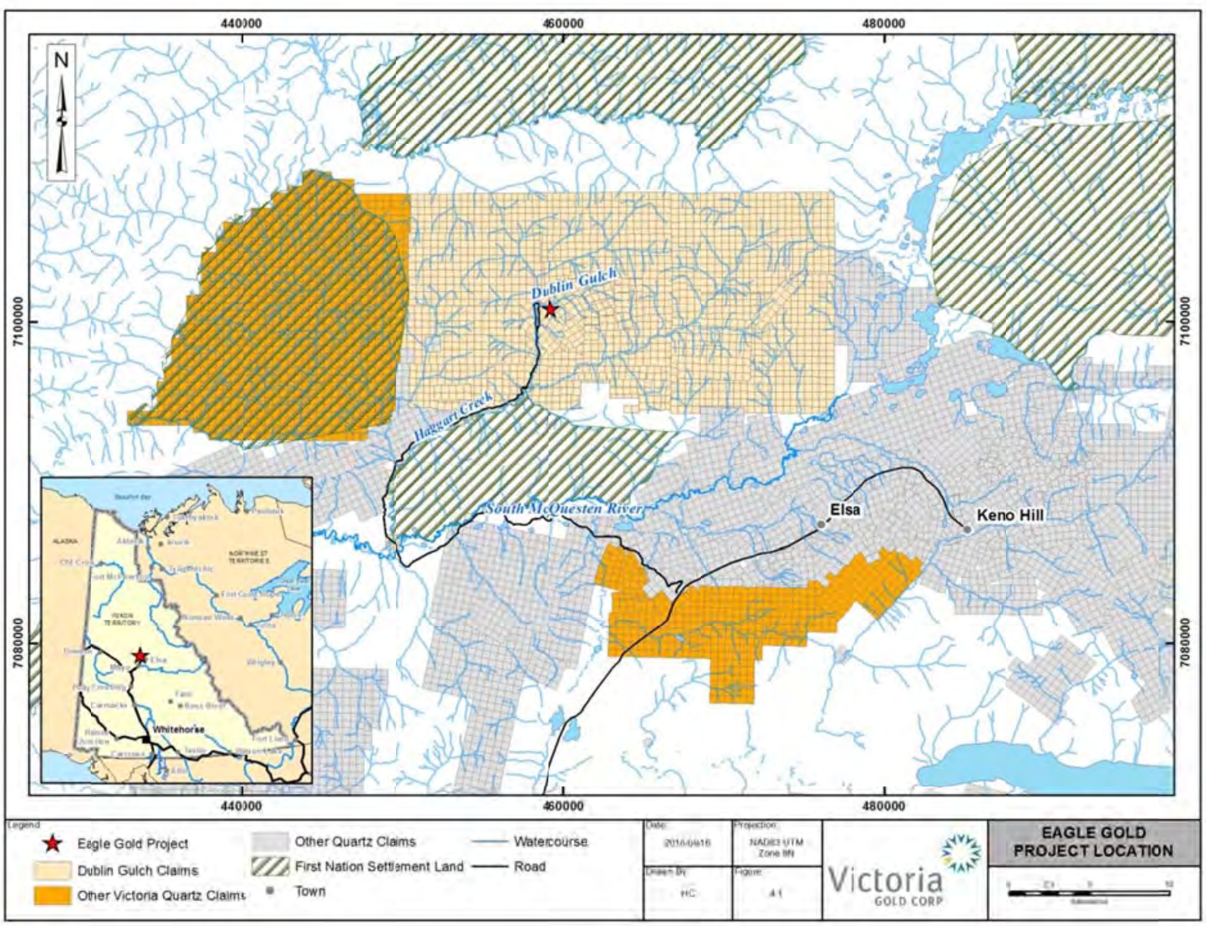
4.1 Location

The Eagle Gold project is located in central Yukon, in the Traditional Territory of the FNNND, and within the Stewart River sub-basin of the Yukon River Watershed. The majority of the project lies within the Dublin Gulch Watershed. Dublin Gulch is a second order stream that is a tributary to Haggart Creek, which flows to the South McQuesten River. Elevations in the vicinity of the project range from about 730 to 1,525 metres above sea level (masl).

The centre of the project is at approximately 64°01'30"N latitude and 135°49'30"W longitude or UTM Coordinates 7,100,060N / 459,680E, Zone 8, North American Datum (NAD) 83.

A project location map is provided in Figure 4.1.

Figure 4.1: Eagle Gold Project Location



Source: Victoria Gold (2016)

4.2 Mineral Tenure

The project is situated within the Dublin Gulch property which is a contiguous block of 1,914 quartz claims, 10 quartz leases, and one federal crown grant. All of the Dublin Gulch mineral titles are held by StrataGold Corporation, a wholly owned-directly held subsidiary of Victoria Gold. The Dublin Gulch property is rectangular in shape and is approximately 35,000 ha.

In 1996, the claims which host a portion of the Eagle deposit, as it had been defined at that time, were surveyed by a Canada Lands surveyor to ensure that full title was held over the deposit. The claims that were to host the main HLP for the 1996 mine plan were also surveyed. These surveys were completed and satisfactorily registered and the boundaries of those claims are considered definitive.

In 2013, the claims that host the Eagle deposit as it was then defined, the two proposed WRSAs immediately north and south of the Eagle deposit, and the proposed phase one area of the Ann Gulch HLP were surveyed by a Canada Lands Surveyor to define their boundaries and ensure that no gaps in the claims exist. As a result of this process, two additional claims were staked to cover minor errors in historic staking. These surveys have defined the boundaries of the additional claims and the surveyed claims cover the current Eagle open pit, the refined WRSAs to the north and south of the Eagle open pit, and phase one area of the Ann Gulch HLP.

The mineral rights held by Victoria Gold include all minerals and the right to enter on and use and occupy the surface of the claims for the operation the mine. Mineral claims in Yukon can be maintained in good standing by performing approved exploration work, or making payments in lieu of work, of $100 per claim per year.

A list of the claims, leases and grant that comprise the Dublin Gulch property are provided in Table 4.1 and are shown in Figure 4.2.

Table 4.1: Mineral Tenure Information

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	1-Jan-02	YB65585, YB65586	1-Mar-24	106D04
Claim	3-Jan	YB65587	16-Jan-24	106D04
Claim	4-Jan	YB65588	16-Jan-29	106D04
Claim	Bob 1 -7, 52, 86	YA17729 - YA17735, YA17780, YA43014	1-Mar-26	106D04
Claim	Dave 1 - 8, 17, 18	YA17802 - YA17809, YA17818, YA17819	1-Mar-26	106D04
Lease	Dave 13 - 16, 25, 27, 28	YA17814 - YA17817, YA42970, YA42972, YA42973	31-Jan-36	106D04
Claim	Dave 26	YA42971	1-Oct-24	106D04
Claim	Dave 29, 30, 31	YA42974, YA42975, YA43015	1-Mar-26	106D04
Claim	DG 43 - 55, 82, 83, 85, 100 - 103	YA14986 - YA14998, YA43044, YA43045, YA43046, YA43061 - YA43064	1-Mar-26	106D04
Claim	Dub 1 - 3	YC11075 - YC11077	1-Mar-26	106D04
Claim	Dub 1000 - 1017	YC38297 - YC38314	1-Mar-23	106D03, 106D04
Claim	Dub 1018 - 1026	YC38315 - YC38323	1-Mar-24	1064D04
Claim	Dub 1027 - 1029	YC38324 - YC38326	1-Mar-23	106D04
Claim	Dub 103, 104	YC11177, YC11178	1-Mar-28	106D04
Claim	Dub 1030	YC38327	1-Mar-24	106D04
Claim	Dub 1031 - 1033	YC38328 - YC38330	1-Mar-23	106D04
Claim	Dub 1034 - 1045	YC38331 - YC38342	1-Mar-24	106D04
Claim	Dub 1046 - 1063	YC38343 - YC38360	1-Mar-23	106D03, 106D04
Claim	Dub 105	YC11179	1-Mar-29	106D04
Claim	Dub 106	YC11180	1-Mar-24	106D04
Claim	Dub 1064 - 1103	YC38361 - YC38400	1-Mar-24	106D04
Claim	Dub 107 - 111	YC11181 - YC11185	1-Mar-29	106D04
Claim	Dub 11 - 16	YC11085 - YC11090	1-Mar-25	106D04
Claim	Dub 1104	YC38401	1-Mar-25	106D04
Claim	Dub 1105	YC38402	1-Mar-24	106D04
Claim	Dub 1106 - 1117	YC38403 - YC38414	1-Mar-25	106D04
Claim	Dub 1118 - 1127	YC38415 - YC38424	1-Mar-24	106D04
Claim	Dub 112	YC11186	1-Mar-25	106D04
Claim	Dub 1128 - 1146	YC38425 - YC38443	1-Mar-23	106D03, 106D04
Claim	Dub 113 - 129	YC11187 - YC11203	1-Mar-29	106D04
Claim	Dub 1147, 1148	YC38444, YC38445	1-Mar-24	106D04
Claim	Dub 1149, 1150	YC38446, YC38447	1-Mar-23	106D04
Claim	Dub 1151	YC38448	1-Mar-24	106D04
Claim	Dub 1152	YC38449	1-Mar-23	106D04
Claim	Dub 1153	YC38450	1-Mar-24	106D04
Claim	Dub 1154	YC38451	1-Mar-23	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 1155	YC38452	1-Mar-24	106D04
Claim	Dub 1156, 1157	YC38453, YC38454	1-Mar-23	106D04
Claim	Dub 1158	YC38455	1-Mar-24	106D04
Claim	Dub 1159	YC38456	1-Mar-23	106D04
Claim	Dub 1160	YC38457	1-Mar-24	106D04
Claim	Dub 1161	YC38458	1-Mar-23	106D04
Claim	Dub 1162 - 1190	YC38459 - YC38487	1-Mar-24	106D04
Claim	Dub 1191	YC38488	1-Mar-25	106D04
Claim	Dub 1192	YC38489	1-Mar-24	106D04
Claim	Dub 1193	YC38490	1-Mar-25	106D04
Claim	Dub 1194	YC38491	1-Mar-24	106D04
Claim	Dub 1195	YC38492	1-Mar-25	106D04
Claim	Dub 1196	YC38493	1-Mar-24	106D04
Claim	Dub 1197 - 1199	YC38494 - YC38496	1-Mar-25	106D04
Claim	Dub 1200 - 1209	YC38497 - YC38506	1-Mar-24	106D04
Claim	Dub 1210 - 1229	YC38507 - YC38526	1-Mar-23	106D03, 106D04
Claim	Dub 1230 - 1293	YC38527 - YC38590	1-Mar-24	106D04
Claim	Dub 1294	YC38591	1-Mar-23	106D04
Claim	Dub 1295	YC38592	1-Mar-24	106D04
Claim	Dub 1296	YC38593	1-Mar-23	106D04
Claim	Dub 1297	YC38594	1-Mar-24	106D04
Claim	Dub 1298	YC38595	1-Mar-23	106D04
Claim	Dub 1299	YC38596	1-Mar-24	106D04
Claim	Dub 130 - 135	YC11204 - YC11209	1-Mar-27	106d04
Claim	Dub 1300	YC38597	1-Mar-23	106D04
Claim	Dub 1301	YC38598	1-Mar-24	106D04
Claim	Dub 1302	YC38599	1-Mar-23	106D04
Claim	Dub 1303	YC38600	1-Mar-24	106D04
Claim	Dub 1304	YC38601	1-Mar-23	106D04
Claim	Dub 1305	YC38602	1-Mar-25	106D04
Claim	Dub 1306 - 1310	YC38603 - YC38607	1-Mar-24	106D04
Claim	Dub 1311	YC38608	1-Mar-25	106D04
Claim	Dub 1312	YC38609	1-Mar-24	106D04
Claim	Dub 1313	YC38610	1-Mar-26	106D04
Claim	Dub 1314, 1315	YC38611, YC38612	1-Mar-24	106D04
Claim	Dub 1316	YC38613	1-Mar-23	106D04
Claim	Dub 1317	YC38614	1-Mar-24	106D04
Claim	Dub 1318	YC38615	1-Mar-23	106D04
Claim	Dub 1319 - 1321	YC38616 - YC38618	1-Mar-24	106D04
Claim	Dub 1322	YC38619	1-Mar-23	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 1323	YC38620	1-Mar-24	106D04
Claim	Dub 1324	YC38621	1-Mar-23	106D04
Claim	Dub 1325, 1326	YC38622, YC38623	1-Mar-24	106D04, 116A01
Claim	Dub 1327	YC38624	1-Mar-23	116A01
Claim	Dub 1328 - 1344, 1345, 1346, 1347	YC38625 - YC38641, YC39876, YC38642, YC38643	1-Mar-24	106D04
Claim	Dub 1348, 1349	YC38644, YC38645	1-Mar-25	106D04
Claim	Dub 1350 - 1359	YC38646 - YC38655	1-Mar-24	106D04
Claim	Dub 136, 137	YC11210, YC11211	1-Mar-24	106D04
Claim	Dub 1360 - 1363	YC38656 - YC38659	1-Mar-23	106D04, 116A01
Claim	Dub 1364, 1365	YC38660, YC38661	1-Mar-24	106D04
Claim	Dub 1366, 1367	YC38662, YC38663	1-Mar-25	106D04
Claim	Dub 1368 - 1371	YC38664 - YC38667	1-Mar-26	106D04
Claim	Dub 1372 - 1395	YC38668 - YC38691	1-Mar-24	106D04
Claim	Dub 138 - 141	YC11212 - YC11215	1-Mar-26	106D04
Claim	Dub 1396 - 1399	YC38692 - YC38695	1-Mar-23	106D04, 116A01
Claim	Dub 1400 - 1403	YC38969 - YC38699	1-Mar-24	106D04
Claim	Dub 1404 - 1419	YC38700 - YC38715	1-Mar-26	106D04
Claim	Dub 142	YC11216	1-Mar-25	106D04
Claim	Dub 1420 - 1423	YC38716 - YC38719	1-Mar-24	106D04
Claim	Dub 1424 - 1437	YC38720 - YC38733	1-Mar-26	106D04
Claim	Dub 143 - 152	YC11217 - YC11226	1-Mar-29	106D04
Claim	Dub 1438	YC38734	1-Mar-24	106D04
Claim	Dub 1439	YC38735	1-Mar-26	106D04
Claim	Dub 1440 - 1443	YC38736 - YC38739	1-Mar-24	106D04
Claim	Dub 1444 - 1457	YC38740 - YC38753	1-Mar-26	106D04
Claim	Dub 1458 - 1463	YC38754 - YC38759	1-Mar-24	106D04
Claim	Dub 1464	YC38760	1-Mar-26	106D04
Claim	Dub 1465	YC38761	1-Mar-24	106D04
Claim	Dub 1466	YC38762	1-Mar-26	106D04
Claim	Dub 1467	YC38763	1-Mar-24	106D04
Claim	Dub 1468	YC38764	1-Mar-26	106D04
Claim	Dub 1469	YC38765	1-Mar-24	106D04
Claim	Dub 1470	YC38766	1-Mar-26	106D04
Claim	Dub 1471	YC38767	1-Mar-24	106D04
Claim	Dub 1472	YC38768	1-Mar-26	106D04
Claim	Dub 1473	YC38769	1-Mar-24	106D04
Claim	Dub 1474	YC38770	1-Mar-26	106D04
Claim	Dub 1475 - 1499	YC38771 - YC38795	1-Mar-24	106D04, 116A01
Claim	Dub 1500	YC38795	1-Mar-23	116A01
Claim	Dub 1501	YC38796	1-Mar-24	116A01

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 1502	YC38797	1-Mar-23	116A01
Claim	Dub 1503	YC38798	1-Mar-24	116A01
Claim	Dub 1504 - 1512	YC38799 - YC38808	1-Mar-26	106D04
Claim	Dub 1513 - 1529	YC38809 - YC38825	1-Mar-24	106D04, 116A01
Claim	Dub 153 - 159	YC11227 - YC11233	1-Mar-26	106D04
Claim	Dub 1530 - 1534	YC38826 - YC38830	1-Mar-26	106D04
Claim	Dub 1535	YC38831	1-Mar-24	106D04
Claim	Dub 1536 - 1538	YC38832 - YC38834	1-Mar-26	106D04
Claim	Dub 1539	YC38835	1-Mar-24	106D04
Claim	Dub 1540	YC38836	1-Mar-26	106D04
Claim	Dub 1541 - 1581	YC38837 - YC38877	1-Mar-24	106D04, 116A01
Claim	Dub 1582	YC38878	1-Mar-26	106D04
Claim	Dub 1583	YC38879	1-Mar-24	106D04
Claim	Dub 1584 - 1589	YC38880 - YC39856	1-Mar-23	106D04
Claim	Dub 1590 - 1602	YC39857 - YC39875	1-Mar-24	106D04, 105M13
Claim	Dub 160	YC11234	1-Mar-28	106D04
Claim	Dub 1603 - 1608	YC39860 - YC39865	1-Mar-24	105M13
Claim	Dub 1609 - 1619	YC42226 - YC42236	1-Mar-24	105M13
Claim	Dub 161 - 165	YC11235 - YC11239	1-Mar-29	106D04, 105M13
Claim	Dub 166 - 170	YC11240 - YC11244	1-Mar-27	105M13
Claim	Dub 17 - 20	YC11091 - YC11094	1-Mar-24	106D04
Claim	Dub 171 - 180	YC11245 - YC11254	1-Mar-26	105M13
Claim	Dub 181 - 189	YC11255 - YC11263	1-Mar-29	105M13
Claim	Dub 190	YC11264	1-Mar-26	105M13
Claim	Dub 191	YC11265	1-Mar-29	105M13
Claim	Dub 192	YC11266	1-Mar-24	105M13
Claim	Dub 193 - 197	YC11267 - YC11271	1-Mar-26	106D04, 105M13
Claim	Dub 198	YC11272	1-Mar-29	105M13
Claim	Dub 199 - 207	YC11273 - 11281	1-Mar-26	106D04, 105M13
Claim	Dub 208	YC11282	1-Mar-25	106D04
Claim	Dub 209 - 216	YC11283 - YC11290	1-Mar-29	106D04
Claim	Dub 21	YC11095	1-Mar-29	106D04
Claim	Dub 217	YC11291	1-Mar-26	106D04
Claim	Dub 218	YC11292	1-Mar-27	106D04
Claim	Dub 219	YC11293	1-Mar-26	106D04
Claim	Dub 22	YC11096	1-Mar-25	106D04
Claim	Dub 220 - 222	YC11297 - YC11296	1-Mar-27	106D04
Claim	Dub 223	YC11297	1-Mar-26	106D04
Claim	Dub 224	YC11298	1-Mar-27	106D04
Claim	Dub 225	YC11299	1-Mar-26	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 226	YC11300	1-Mar-27	106D04
Claim	Dub 227 - 229	YC11301 - YC11303	1-Mar-24	106D04
Claim	Dub 23	YC11097	1-Mar-29	106D04
Claim	Dub 230 - 232	YC11304 - YC11306	1-Mar-26	106D04
Claim	Dub 233 - 240	YC11307 - YC11314	1-Mar-29	106D04
Claim	Dub 24	YC11098	1-Mar-24	106D04
Claim	Dub 241 - 257	YC11315 - YC11331	1-Mar-27	106D04
Claim	Dub 25	YC11099	1-Mar-29	106D04
Claim	Dub 258	YC11332	1-Mar-26	106D04
Claim	Dub 259, 260	YC11333, YC11334	1-Mar-27	106D04
Claim	Dub 26	YC11100	1-Mar-24	106D04
Claim	Dub 261	YC11335	1-Mar-28	106D04
Claim	Dub 262 - 266	YC11336 - YC11340	1-Mar-25	106D04
Claim	Dub 267 - 272	YC11341 - YC11346	1-Mar-26	106D04
Claim	Dub 27	YC11101	1-Mar-26	106D04
Claim	Dub 273 - 279	YC11347 - YC11353	1-Mar-27	106D04
Claim	Dub 28	YC11102	1-Mar-24	106D04
Claim	Dub 280	YC11354	1-Mar-25	106D04
Claim	Dub 281 - 288	YC11355 - YC11362	1-Mar-26	106D04
Claim	Dub 289	YC11363	1-Mar-27	106D04
Claim	Dub 29	YC11103	1-Mar-29	106D04
Claim	Dub 290	YC11364	1-Mar-26	106D04
Claim	Dub 291	YC11365	1-Mar-27	106D04
Claim	Dub 292, 293	YC11366, YC11367	1-Mar-26	106D04
Claim	Dub 294, 295	YC11368, YC11369	1-Mar-25	106D04
Claim	Dub 296	YC11370	1-Mar-26	106D04
Claim	Dub 297 - 299	YC11371 - YC11373	1-Mar-27	106D04
Claim	Dub 30	YC11104	1-Mar-24	106D04
Claim	Dub 300 - 305	YC11374 - YC11379	1-Mar-26	106D04
Claim	Dub 306	YC11380	1-Mar-28	106D04
Claim	Dub 307 - 310	YC11381 - YC11384	1-Mar-26	106D04
Claim	Dub 31	YC11105	1-Mar-28	106D04
Claim	Dub 311	YC11385	1-Mar-25	106D04
Claim	Dub 312	YC11386	1-Mar-26	106D04
Claim	Dub 313 - 324	YC11387 - YC11398	1-Mar-25	106D04
Claim	Dub 32	YC11106	1-Mar-24	106D04
Claim	Dub 325 - 327	YC11399 - YC11401	1-Mar-26	106D04
Claim	Dub 328	YC11402	1-Mar-25	106D04
Claim	Dub 329	YC11403	1-Mar-26	106D04
Claim	Dub 33	YC11107	1-Mar-29	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 330 - 338	YC11404 - YC11412	1-Mar-27	106D04
Claim	Dub 339	YC11413	1-Mar-28	106D04
Claim	Dub 34	YC11108	1-Mar-24	106D04
Claim	Dub 340	YC11414	1-Mar-27	106D04
Claim	Dub 341	YC11415	1-Mar-29	106D04
Claim	Dub 342	YC11416	1-Mar-26	106D04
Claim	Dub 343	YC11417	1-Mar-28	106D04
Claim	Dub 344	YC11418	1-Mar-26	106D04
Claim	Dub 345	YC11419	1-Mar-28	106D04
Claim	Dub 346	YC11420	1-Mar-26	106D04
Claim	Dub 347	YC11421	1-Mar-28	106D04
Claim	Dub 348	YC11422	1-Mar-26	106D04
Claim	Dub 349	YC11423	1-Mar-28	106D04
Claim	Dub 35	YC11109	1-Mar-28	106D04
Claim	Dub 350	YC11424	1-Mar-25	106D04
Claim	Dub 351	YC11425	1-Mar-28	106D04
Claim	Dub 352	YC11426	1-Mar-26	106D04
Claim	Dub 353	YC11427	1-Mar-29	106D04
Claim	Dub 354	YC11428	1-Mar-26	106D04
Claim	Dub 355	YC11429	1-Mar-29	106D04
Claim	Dub 356	YC11430	1-Mar-25	106D04
Claim	Dub 357 - 359	YC11431 - YC11433	1-Mar-26	106D04
Claim	Dub 36	YC11110	1-Mar-26	106D04
Claim	Dub 360	YC11434	1-Mar-25	106D04
Claim	Dub 361 - 364	YC11435 - YC11438	1-Mar-26	106D04
Claim	Dub 365 - 368	YC11439 - YC11442	1-Mar-27	106D04
Claim	Dub 369 - 372	YC11443 - YC11446	1-Mar-25	106D04
Claim	Dub 37	YC11111	1-Mar-24	106D04
Claim	Dub 373, 374	YC11447, YC11448	1-Mar-26	106D04
Claim	Dub 375, 376	YC11449, YC11450	1-Mar-29	106D04
Claim	Dub 377 - 384	YC11451 - YC11458	1-Mar-28	106D04
Claim	Dub 38	YC11112	1-Mar-26	106D04
Claim	Dub 385 - 390	YC11459 - YC11464	1-Mar-29	106D04
Claim	Dub 39	YC11113	1-Mar-24	106D04
Claim	Dub 391 - 396	YC11465 - YC11470	1-Mar-26	106D04
Claim	Dub 397	YC11471	1-Mar-27	106D04
Claim	Dub 398	YC11472	1-Mar-26	106D04
Claim	Dub 399, 400	YC11473, YC11474	1-Mar-27	106D04
Claim	Dub 4	YC11078	1-Mar-25	106D04
Claim	Dub 40	YC11114	1-Mar-26	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 401	YC11475	1-Mar-26	106D04
Claim	Dub 402	YC11476	1-Mar-25	106D04
Claim	Dub 403	YC11477	1-Mar-26	106D04
Claim	Dub 404	YC11478	1-Mar-25	106D04
Claim	Dub 405	YC11479	1-Mar-26	106D04
Claim	Dub 406	YC11480	1-Mar-25	106D04
Claim	Dub 407	YC11481	1-Mar-26	106D04
Claim	Dub 408	YC11482	1-Mar-25	106D04
Claim	Dub 409	YC11483	1-Mar-28	106D04
Claim	Dub 41	YC11115	1-Mar-24	106D04
Claim	Dub 410	YC11484	1-Mar-29	106D04
Claim	Dub 411	YC11485	1-Mar-27	106D04
Claim	Dub 412	YC11486	1-Mar-29	106D04
Claim	Dub 413	YC11487	1-Mar-27	106D04
Claim	Dub 414	YC11488	1-Mar-28	106D04
Claim	Dub 415	YC11489	1-Mar-27	106D04
Claim	Dub 416	YC11490	1-Mar-28	106D04
Claim	Dub 417	YC11491	1-Mar-27	106D04
Claim	Dub 418	YC11492	1-Mar-28	106D04
Claim	Dub 419	YC11493	1-Mar-27	106D04
Claim	Dub 42	YC11116	1-Mar-27	106D04
Claim	Dub 420	YC11494	1-Mar-29	106D04
Claim	Dub 421	YC11495	1-Mar-27	106D04
Claim	Dub 422	YC11496	1-Mar-25	106D04
Claim	Dub 423	YC11497	1-Mar-28	106D04
Claim	Dub 424	YC11498	1-Mar-29	106D04
Claim	Dub 425	YC11499	1-Mar-24	106D04
Claim	Dub 426	YC11500	1-Mar-28	106D04
Claim	Dub 427	YC11501	1-Mar-24	106D04
Claim	Dub 428	YC11502	1-Mar-27	106D04
Claim	Dub 429	YC11503	1-Mar-24	106D04
Claim	Dub 43, 44	YC11117, YC11118	1-Mar-24	106D04
Claim	Dub 430	YC11504	1-Mar-27	106D04
Claim	Dub 431	YC11505	1-Mar-24	106D04
Claim	Dub 432 - 436	YC1150 - YC11510	1-Mar-27	106D04
Claim	Dub 437 - 440	YC11511 - YC11514	1-Mar-28	106D04
Claim	Dub 441 - 449	YC11515 - YC11523	1-Mar-24	106D04
Claim	Dub 45	YC11119	1-Mar-29	106D04
Claim	Dub 450	YC11524	1-Mar-27	106D04
Claim	Dub 451	YC11525	1-Mar-26	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 452, 453	YC11526, YC11527	1-Mar-27	106D04
Claim	Dub 454	YC11528	1-Mar-28	106D04
Claim	Dub 455	YC11529	1-Mar-27	106D04
Claim	Dub 456	YC11530	1-Mar-28	106D04
Claim	Dub 457 - 479	YC11531 - YC11553	1-Mar-24	106D04
Claim	Dub 46	YC11120	1-Mar-24	106D04
Claim	Dub 47	YC11121	1-Mar-29	106D04
Claim	Dub 48	YC11122	1-Mar-24	106D04
Claim	Dub 480 - 484	YC11554, YC32478 - YC32481	1-Mar-26	106D04
Claim	Dub 485 - 492	YC32482 - YC32489	1-Mar-24	105M13
Claim	Dub 49	YC11123	1-Mar-27	106D04
Claim	Dub 493	YC32490	1-Mar-25	105M13
Claim	Dub 494 - 496	YC32491 -YC32493	1-Mar-24	105M13
Claim	Dub 497 - 516	YC32494 - YC32513	1-Mar-26	105M13
Claim	Dub 5 - 8	YC11079 - YC11082	1-Mar-24	106D04
Claim	Dub 50	YC11124	1-Mar-24	106D04
Claim	Dub 51	YC11125	1-Mar-27	106D04
Claim	Dub 517	YC32514	1-Mar-25	105M13
Claim	Dub 518 - 544	YC32515 - YC32541	1-Mar-26	105M13
Claim	Dub 52	YC11126	1-Mar-24	106D04
Claim	Dub 53 - 56	YC11127 - YC11130	1-Mar-27	106D04
Claim	Dub 545 - 548	YC32542 - YC32545	1-Mar-24	105M13
Claim	Dub 567 - 581	YC32564 - YC32578	1-Mar-26	105M13
Claim	Dub 57 - 66	YC11131 - YC11140	1-Mar-29	106D04
Claim	Dub 582 - 587	YC32579 - YC32584	1-Mar-24	105M13
Claim	Dub 588	YC32585	1-Mar-23	105M13
Claim	Dub 589	YC32586	1-Mar-24	105M13
Claim	Dub 590	YC32587	1-Mar-23	105M13
Claim	Dub 591	YC32588	1-Mar-24	105M13
Claim	Dub 592 - 603	YC32589 - YC32600	1-Mar-23	106D04, 105M13
Claim	Dub 604 - 662	YC32601 - YC32659	1-Mar-24	106D04, 105M13
Claim	Dub 663 - 678	YC32660 - YC32675	1-Mar-23	105M13, 105M14
Claim	Dub 67, 68	YC11141, YC11142	1-Mar-28	106D04
Claim	Dub 679 - 682	YC32676 - YC32679	1-Mar-24	105M13
Claim	Dub 683 - 779	YC32680 - YC32700, YC38001 - YC38076	1-Mar-23	106D04, 105M13, 105M14
Claim	Dub 69	YC11143	1-Mar-29	106D04
Claim	Dub 70	YC11144	1-Mar-28	106D04
Claim	Dub 71	YC11145	1-Mar-29	106D04
Claim	Dub 72	YC11146	1-Mar-28	106D04
Claim	Dub 73 - 78	YC11147 - YC11152	1-Mar-24	106D04

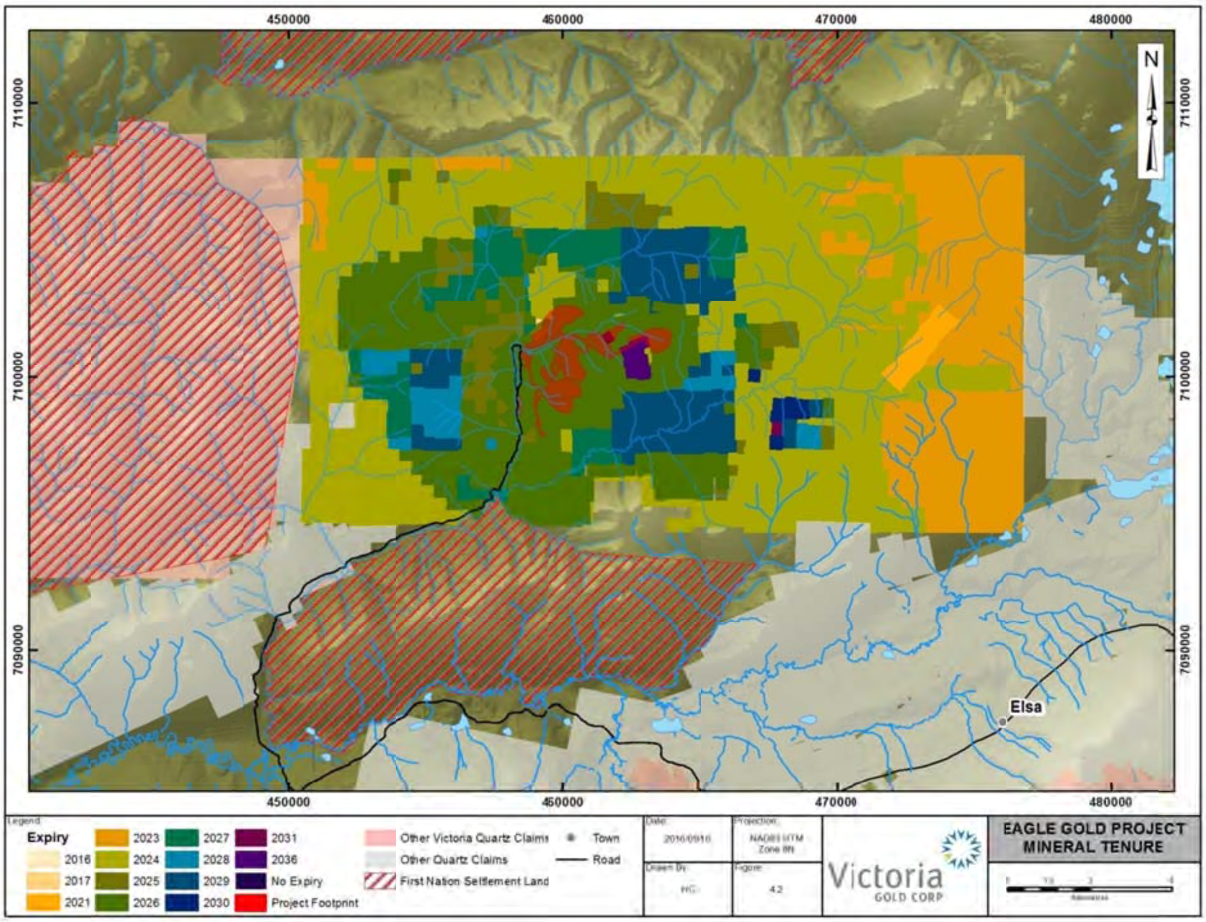
Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Dub 780	YC38077	1-Mar-24	106D04
Claim	Dub 781	YC38078	1-Mar-23	106D04
Claim	Dub 782	YC38079	1-Mar-24	106D04
Claim	Dub 783, 784	YC38080, YC38081	1-Mar-26	106D03
Claim	Dub 785 - 801	YC38082 - YC38098	1-Mar-24	106D03, 106D04
Claim	Dub 79	YC11153	1-Mar-29	106D04
Claim	Dub 80	YC11154	1-Mar-24	106D04
Claim	Dub 802 - 842	YC38099 - YC38139	1-Mar-23	106D03, 106D04
Claim	Dub 81 - 85	YC11155 - YC11159	1-Mar-29	106D04
Claim	Dub 843 - 879	YC38140 - YC38176	1-Mar-24	106D04
Claim	Dub 86	YC11160	1-Mar-27	106D04
Claim	Dub 87	YC11161	1-Mar-29	106D04
Claim	Dub 88	YC11162	1-Mar-27	106D04
Claim	Dub 880, 881	YC38177, YC38178	1-Mar-23	106D04
Claim	Dub 882	YC38179	1-Mar-24	106D04
Claim	Dub 883	YC38180	1-Mar-23	106D04
Claim	Dub 884, 885	YC38181, YC38182	1-Mar-24	106D04
Claim	Dub 886	YC38183	1-Mar-23	106D04
Claim	Dub 887 - 907	YC38184 - YC38204	1-Mar-24	106D04
Claim	Dub 89	YC11163	1-Mar-29	106D04
Claim	Dub 9, 10	YC11083, YC11084	1-Mar-26	106D04
Claim	Dub 90	YC11164	1-Mar-27	106D04
Claim	Dub 908 - 927	YC38205 - YC38224	1-Mar-23	106D03, 106D04
Claim	Dub 91	YC11165	1-Mar-29	106D04
Claim	Dub 92	YC11166	1-Mar-27	106D04
Claim	Dub 928 - 953	YC38225 - YC38250	1-Mar-24	106D03, 106D04
Claim	Dub 93 - 102	YC11167 - YC11176	1-Mar-29	106D04
Claim	Dub 954 - 969	YC38251 - YC38266	1-Mar-23	106D03, 106D04
Claim	Dub 970	YC38267	1-Mar-24	106D04
Claim	Dub 971	YC38268	1-Mar-23	106D04
Claim	Dub 972 - 975	YC38269 - YC38272	1-Mar-24	106D04
Claim	Dub 976 - 979	YC38273 - YC38276	1-Mar-23	106D04
Claim	Dub 980	YC38277	1-Mar-24	106D04
Claim	Dub 981	YC38278	1-Mar-23	106D04
Claim	Dub 982 - 999	YC38279 - YC38296	1-Mar-24	106D04
Claim	Dub Fr. 1620	YE55727	11-Feb-17	106D04
Claim	Fiji 1	YA63884	1-Mar-26	106D04
Claim	Fiji 2	YB03409	1-Mar-26	106D04
Claim	Fiji 3	YA63886	1-Mar-26	106D04
Claim	Fiji 5	YA63888	1-Mar-26	106D04

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Fiji 6	YA63889	1-Mar-26	106D04
Claim	Hla Hla 1 - 6, 7 - 14	YC10918 - YC10923, YC10828 - YC10835	1-Mar-21	106D04
Claim	Jeff 116	YC39877	1-Mar-24	106D04
Claim	Jeff 117, 118, 120	YB03408, YA42981, YA42983	1-Mar-26	106D04
Claim	Jeff 17, 18, 33, 34, 113 - 115	YA17842, YA17843, YA17858, YA17859, YA42976 - YA142978	1-Mar-26	106D04
Claim	Len 1, 2	YC02730, YC02731	15-May-30	106D04
Claim	Len 10	YA30530	15-May-26	106D04
Claim	Len 11	YC02736	15-May-31	106D04
Claim	Len 12	YC02737	15-May-30	106D04
Claim	Len 13, 14	YC02738, YC02739	15-May-29	106D04
Claim	Len 15 - 18	YC02740 - YC02743	15-May-28	106D04
Claim	Len 19, 20	YC02744, YC02745	15-May-25	106D04
Claim	Len 21 - 23	YC02746 - YC02748	1-Mar-24	106D04
Claim	Len 24	YA30544	15-May-26	106D04
Claim	Len 25	YC02749	1-Mar-24	106D04
Claim	Len 26	YA30546	15-May-29	106D04
Claim	Len 27	YC02750	1-Mar-24	106D04
Claim	Len 28	YA30548	15-May-30	106D04
Claim	Len 29	YC02751	1-Mar-24	106D04
Claim	Len 3	YC02732	1-Mar-24	106D04
Claim	Len 30	YA30550	15-May-30	106D04
Claim	Len 31	YC02752	1-Mar-24	106D04
Claim	Len 32	YC02753	1-Mar-24	106D04
Claim	Len 4	YA30524	15-May-30	106D04
Claim	Len 5	YC02733	1-Mar-24	106D04
Claim	Len 6	YA30526	15-May-30	106D04
Claim	Len 7	YC02734	1-Mar-24	106D04
Claim	Len 8	YA30528	15-May-29	106D04
Claim	Len 9	YC02735	1-Mar-24	106D04
Claim	Lynx 1 - 18	YC10463 - YC10480	1-Mar-24	105M13
Claim	Lynx 19	YC10481	16-Jan-30	105M13
Claim	Lynx 20 - 23	YC10482 - YC10485	1-Mar-24	105M13
Claim	Lynx 24	YC10486	16-Jan-24	105M13
Claim	Lynx 25	YC10487	1-Mar-24	105M13
Claim	Lynx 26	YC10488	16-Jan-24	105M13
Claim	Lynx 27	YC10489	1-Mar-24	105M13
Claim	Lynx 28	YC10490	16-Jan-24	105M13
Claim	Lynx 29 - 32	YC10491 - YC10494	1-Mar-24	105M13
Claim	Lynx 33	YC10495	16-Jan-24	105M13

Regulation Type	Claim Name	Grant Number	Expiry Date	NTS Map Sheet
Claim	Lynx 34 - 56	YC10496 - YC10518	1-Mar-24	105M13
Claim	Lynx 57	YC11555	1-Mar-24	106D04
Claim	Mar 1 - 12, 14 - 22, 24, 31, 33 - 40	YA14896 - YA14907, YA14909 - YA14917, YA14919, YA42984, YA43101 - YA43108	1-Mar-26	106D04
Claim	Mary 1 - 8	YA63876 - YA63883	1-Mar-26	106D04
Claim	Neera 1, 2	YC10822, YC10823	1-Mar-21	106D04
Grant	Olive Crown Grant	GR1054	N/A	106D04
Claim	R & D 1 - 8, 10, 12, 14 - 16	YA01393 - YA01400, YA01402, YA01404, YA01406 - YA01408	1-Mar-26	106D04
Lease	R & D No. 9, 11, 13	YA01401, YA01403, YA01405	31-Jan-36	106D04
Claim	Roni 1 - 14	YB64630 - YB64643	1-Mar-26	106D04
Claim	Smoky 1 - 10, 23, 25 - 30, 37 - 41, 44 - 47, 48, 49, 51 - 54, 56, 58, 62 - 65, 66 - 71, 74 - 77, 78, 80, 83 - 45, 91 - 100, 107 - 109	YA17930 - YA17939, YA17952, YA17954 - YA17959, YA17966 - YA17970, YA30072 - YA30075, YA17973, YA17974, YA30076 - YA30079, YA17977, YA17979, YA30080 - YA30083, YA17983 - YA17988, YA30084 - YA30087, YA17991, YA17993, YA43120 - YA43122, YA43128 - YA43137, YA43144 - YA43146	1-Mar-26	106D04
Claim	Smoky Fr. 55	YE55726	6-Dec-16	106D04
Claim	Tin Dome 1 - 4, 5 - 12	YC02842 - YC02845, YC02848 - YC02855	1-Mar-24	106D04
Claim	West 167 - 172, 174, 182, 184	YB18934 - YB18939, YB18941, YB18949, YB18951	1-Mar-26	106D04, 105M13

Source: Victoria Gold (2016)

Figure 4.2: Mineral Tenure Map



Source: Victoria Gold (2016)

4.3 Mining Rights

The primary legislation governing mining in Yukon is the Quartz Mining Act (QMA) and the Quartz Mining Land Use Regulations. The regulatory body charged with overseeing the QMA is the Department of Energy, Mines and Resources (EMR).

Ownership of quartz claims pursuant to the QMA carries the right to surface access and use for the exploitation of minerals contained within the claims. A claim holder must however make an application to the Minister of EMR to engage in development or production activities and may only conduct these activities in accordance with the terms and conditions of a license issued by the Minister. The license issued by the Minister is a Quartz Mining License which specifies the duration, activities, and claims, among other matters, that a licensee and claim holder must adhere to and operate within.

The permitting required for the project is discussed further in Section 20.

4.4 Project Agreements

The Dublin Gulch property is subject to three underlying agreements, two of which are material to the Eagle Gold project.

The Eagle deposit falls entirely within claims that are subject to a royalty historically known as the Mar Gold Zone Royalty. This royalty requires minimum annual royalty payments of $20,000 or a production royalty of 2% of the gross returns received from the sale of all metals produced from the claims to a maximum of $1,000,000 after which the royalty reverts to 1% with no end price.

A portion of the Olive deposit falls within a claim that is subject to the Queenstake Mar Tungsten Royalty. This royalty is a 1% net smelter return royalty payable only upon the commencement of production.

Other than the two royalties described above, the project is free and clear of any liens or third party interests.

4.5 Environmental Liabilities and Considerations

Exploration activities within the Dublin Gulch property are conducted under a Class IV Mining Land Use Approval (LQ00303) granted by EMR under the QMA and the Quartz Mining Land Use Regulations. LQ00303 authorizes Victoria Gold to conduct exploration activities and operate the existing 100 person camp facility.

The scope of exploration activities on the Dublin Gulch property has included the construction and use of a trail network, drilling pads, trenches, and the camp. All of these features will require reclamation to the satisfaction of EMR prior to the expiration of LQ00303 or any subsequent permit extension or replacement. EMR required Victoria Gold post a financial bond upon the issuance of LQ00303 in the amount of $149,000 to cover the reclamation of these features in the event that the company is unable for any reason to complete the work.

The scope of exploration activities undertaken on the Dublin Gulch property and the reclamation required for this work is considered industry standard for an advanced stage project and, based on the financial bond held by EMR and Yukon Government, do not present a significant environmental liability.

4.6 Property Risks

There are no known factors that may materially affect access, title or the right or ability to perform any of the activities contemplated herein.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

The project has year-round 90 km long access road connecting to the community of Mayo, Yukon. The property is accessed from Mayo by the Silver Trail (Highway 11) onto the South McQuesten Road (SMR) and then the Haggart Creek Road (HCR) which terminates at the project site. Together the SMR and the HCR comprise a 45 km road divided by the South McQuesten River. Both are public roads, regulated under the Yukon Highways Act; however, the SMR is only maintained during the summer by the Yukon Government Department of Highways and Public Works (HPW), whereas the HCR is considered a “public unmaintained” road.

Victoria Gold conducts snow clearing activities on both the SMR and HCR on an as needed basis and general maintenance on the HCR under the authority of permits granted by HPW.

5.2 Local Resources and Infrastructure

Mayo has a population of approximately 450 and offers accommodation, fuel, a nursing station, and earthmoving contractors. The Yukon Government maintains a 1,400 m gravel airstrip, suitable for charter flights, about 3 km north of Mayo. The project is about 45 km straight-line distance north-northeast of Mayo. There are no scheduled air services to Mayo. Most major services and supplies are available in Whitehorse.

Electrical transmission lines from a hydroelectric facility near Mayo extend to the villages of Elsa and Keno City, about 25 km southeast of the Dublin Gulch property. The existing 100-person camp on the property is currently served by on-site generators.

A broader range of services is available in Whitehorse, Yukon, located about six hours by road to the south of the project. Whitehorse has a population of 25,690 (Yukon Bureau of Statistics) and has regularly scheduled air service to Vancouver, Edmonton, Calgary, and Fairbanks.

The property is approximately 665 km by all-weather highway from the deep sea and barge port of Skagway, Alaska.

5.3 Climate

Central Yukon has a northern continental climate. The mean annual temperature for the area is approximately -3°C, with an annual range of 63.5°C. January is the coldest month, July the warmest. Annual precipitation ranges from 375 to 600 mm, about half of which falls as snow. The project will be in operation year round.

5.4 Physiography

The topography of the property area is characterized by rolling hills and plateaus ranging in elevation from approximately 800 masl to a local maximum of 1,650 masl at the summit of Potato Hills, and are drained by deeply-incised creeks and canyons. The ground surface is covered by residual soil and felsenmeer. Outcrops are rare, comprising generally less than two percent of the surface area, and are limited to ridge tops and creek walls.

Lower elevations are vegetated with black spruce, willow, alder and moss, and higher elevations by subalpine vegetation. Patchy permafrost occurs on north-facing slopes (Figure 5.1). There are sufficient surface rights held by Victoria Gold by virtue of the claims, leases and permits described herein for mining operations contemplated in the FS.

Figure 5.1: Typical Landscape in the Project Area

1 - From Eagle Zone looking North-West to camp and confluence of Dublin Gulch and Haggart Creek
2 - From above the Eagle Zone looking north to camp and Ann Gulch
3 - View of Haggart Creek road crossing approximately 3 km south of camp looking North-West

Source: Victoria Gold (2016)

6 History

6.1 Management and Ownership

In 1977, Queenstake Resources Ltd. staked the Mar claims in the Ray Gulch area to cover a tungsten bearing skarn. This property was optioned to CanTung, which explored for gold and tungsten during 1977 – 1986 which led to the discovery of the Eagle Zone 3 km southwest of the original tungsten occurrences. In 1991, the property was acquired by Ivanhoe Goldfields.

In 1994, First Dynasty Mines Ltd. acquired the property through its acquisition of Ivanhoe Goldfields, and subsequently formed New Millennium Mining Ltd., and transferred Dublin Gulch to the new entity. In June 2002, First Dynasty changed its name to Sterlite Gold Limited.

In October 2004, StrataGold Corporation purchased the Dublin Gulch and Clear Creek gold properties from Sterlite. In June 2009, through a Plan of Arrangement, StrataGold was acquired by Victoria Gold.

6.2 Exploration History

Queenstake focused their exploration activities on an area historically known as Mar Tungsten (now Wolf Tungsten) north-east of the Eagle Zone and completed a small geological mapping and sampling program. When CanTung assumed control of these claims, and additional claims located near the Eagle Zone from another prospector, they retained Bema to manage the program.

Bema conducted first phase geological mapping which included an outcrop sampling program delineating the stratigraphic controls of the tungsten mineralization. This was followed up with a trenching program to expose bedrock in areas of shallow to moderate overburden thickness. CanTung also conducted geophysical survey programs that were later supplemented with VLF-EM surveys focused on the tungsten skarns. Subsequently, Bema and CanTung completed an extensive diamond drilling program on the Mar Tungsten Zone and branched out to include trenching along the regional gold fault-vein system. After completing follow-up drilling programs on the tungsten target, CanTung returned the Mar Tungsten Zone and adjacent gold claims to Queenstake.

When Ivanhoe Goldfields acquired the property, they carried out exploration work based on a “Fort Knox-type” intrusive-hosted gold exploration model. Ivanhoe Goldfields continued exploratory work on the Eagle Zone via drilling, trenching, soil sampling, geophysical surveys, baseline environmental monitoring, as well as mineralogical and metallurgical studies.

First Dynasty Mines Ltd. subsequently undertook further exploration work on the Eagle Zone and through the newly formed New Millennium Mining Ltd. engaged Mineral Resource Development Inc. (MRDI) to produce a FS completed in 1997. Due to declining commodity prices, little further exploration work was undertaken on the Eagle Zone until the acquisition of the property by StrataGold Corporation.

In 2006, Wardrop Engineering Inc. (Wardrop) produced a NI 43-101 resource estimate for StrataGold consisting of an Indicated Resource totalling 66.5 Mt grading 0.92 g/t and an Inferred Resource totalling 14.4 Mt grading 0.80 g/t based on historic drilling and StrataGold's 2005 drill campaign. StrataGold conducted further drilling on Eagle from 2006 – 2008 and Wardrop completed an updated NI 43-101 Mineral Resource estimate on the Eagle Zone Deposit in January 2009 adding 37% to the Indicated Resource for a total of 2.69 Moz of gold averaging 0.849 g/t gold. This Mineral Resource estimate incorporated 13,057.65 m of drilling from 2006 – 2008 into the previously-stated resource estimate.

In 2008, StrataGold commissioned SRK to complete a Preliminary Assessment for tungsten on the Mar Tungsten deposit (now Wolf Tungsten). SRK estimated an Indicated Resource of 12.7 Mt grading 0.31% WO3 and an Inferred Resource of 1.3 Mt grading 0.30% WO3, an 11 year mine life, 15.5% IRR and NPV of $24M at an 8% discount rate.

In June 2009, after Victoria Gold acquired StrataGold, further exploration on the Eagle Zone was conducted and Victoria Gold commissioned a Pre-Feasibility Study by Scott Wilson Roscoe Postle Associates. Work in 2009 focused on gathering further information on the Eagle Zone by drilling deep exploration holes and it was found that mineralization extends to considerable depths beyond the pit bottom models at that time.

Further field work around Olive and Shamrock, two targets identified within the Dublin Gulch property by previous operators, identified a continuous, structurally controlled corridor of mineralization, collectively called the 'Potato Hills Trend'.

In 2010, Victoria Gold completed additional exploration and geotechnical drilling on Eagle to quantify alteration, to verify the absence of mineralization (condemnation holes), for exploration, and for geotechnical, engineering and environmental purposes. Data from the 2010 drill program was incorporated into a May 2011 NI 43-101 compliant update to previous resource and reserve estimates in advance of a FS.

In 2011, 78 holes were drilled on Eagle to quantify alteration, for exploration, and for geotechnical, engineering and environmental purposes. In February 2012, Victoria Gold announced the results of a NI 43-101 compliant FS for the project completed by Wardrop Engineering Inc., Tetra Tech, with an effective date of April 18, 2012.

During the 2012 and 2013 field seasons, additional holes were drilled for exploration, to verify the absence of mineralization (condemnation holes), and for geotechnical engineering investigation to support detailed engineering.

In 2014, exploration drilling and trenching focused on the Olive Zone with the completion of 68 drill holes for exploration, metallurgical testing, and geotechnical purposes. Material from this program was used to establish the heap leach recoveries and kinetic results in 2015. Victoria Gold continued drilling on the Olive and subsequently the Shamrock zones in 2016 to support the integration of satellite zones into future mine plans for the Eagle Gold project.

6.3 Production History

No material hard rock production has occurred on the project site however the Dublin Gulch area has a rich history of both placer and hard rock exploration and small scale/placer mining since the late 1800s. Dublin Gulch is a watercourse that discharges to Haggart Creek which is a major tributary of the South McQuesten River. Exploration and placer mining began on Haggart Creek in 1895. Haggart Creek and its tributaries near Dublin Gulch were prospected and mined by multiple claim owners using relatively small operations (pick and shovel and small placer workings) until the late 1930s when larger mechanized equipment was brought to the area (Mayo Historical Society 1999). Mining in the Dublin Gulch area was suspended in the early 1940s during World War II and restarted shortly after the war's end. Mining operations on Haggart Creek from 1953 – 1958 used heavy duty equipment including draglines. It was determined that much of the area was mined out in a few years for larger-scale placer operations, and smaller scale prospecting and mining resumed for the next several decades (Mayo Historical Society 1999). Since 1978 when documentation of placer mining production was initiated, approximately 110,000 ounces of placer gold has been recovered from the Dublin Gulch area until mining ceased in the mid-1990s.

Dublin Gulch, Eagle Creek, and Haggart Creek have been subject to extensive placer mining. There is little evidence of active reclamation being carried out on the placer mined areas and as a result the lower Dublin Gulch valley bottoms and the upper reaches of the Haggart Creek valley bottoms are comprised of exposed and eroding valley walls, large piles of unvegetated placer deposits and partially filled in sediment ponds.

Figure 6.1 depicts the existing conditions including the 100-person camp and historic placer mining areas.

Figure 6.1: Existing Site Conditions

Source: Victoria Gold (2016)

7 Geological Setting and Mineralization

7.1 Geological Setting

The geological setting of the Dublin Gulch property (Eagle Gold project) is one of upper Proterozoic to lower Paleozoic clastic sedimentary rocks that have undergone regional deformation including Cretaceous age thrust faulting and subsequent granitoid intrusions. Mineralization is associated with granitic intrusive bodies, here described as the Eagle Zone and Olive Zone gold deposits, which are hosted primarily in granodioritic rocks.

7.2 Regional Geology

The property is located in the north-central part of the Selwyn Basin, which is a fault- controlled epi-cratonic basin. The stratigraphy of this Basin is divisible into four predominantly clastic lithological units. From youngest to oldest they include; the Lower Schist, Keno Hill Quartzite, Upper Schist, and Hyland Group (formerly the Grit Unit). The Lower Schist is of probable Mesozoic age and the Upper Schist and Keno Hill Quartzite are of Paleozoic age (Devonian-Mississippian). The Hyland Group is of Proterozoic to Lower Cambrian age. These units have been juxtaposed by laterally-extensive, northward-directed thrusting that occurred in early Cretaceous time.

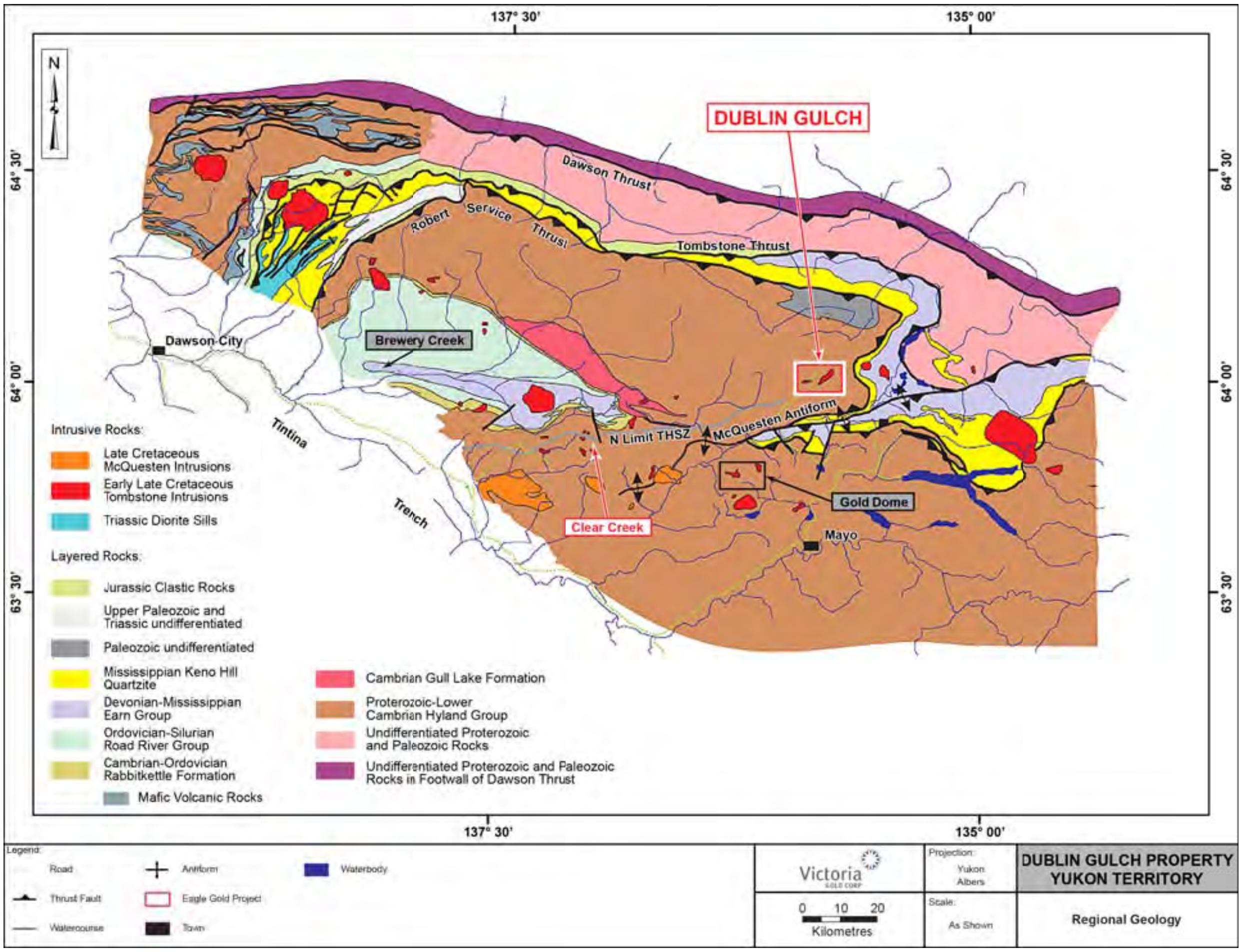
There are three principal thrust sheets in the region known as the Dawson Thrust, the Tombstone Thrust, and the Robert Service Thrust, respectively. The Robert Service Thrust is proximal to the property area and is inferred to have superimposed the Proterozoic–Cambrian age Hyland Group upon the Mississippian-age Keno Hill Quartzite (Figure 7.1).

Four phases of deformation have been documented. Only the first two resulted in the generation of penetrative structures. Thrusting during the first phase resulted in the widespread development of foliation that was subsequently deformed by gentle, regional-scale folding during the second phase of deformation. Several east-west trending, west-plunging anticlines in the Dublin Gulch area are attributed to this second deformational event.

During the mid-to-late Cretaceous period, there were three granitoid intrusion events: the Selwyn Suite (between 104 and 98 Ma), the Tombstone Suite (between 94 and 92 Ma), and the McQuesten Suite (64 Ma). The Selwyn and Tombstone intrusive events were probably synchronous with the second regional folding event. Intrusives are commonly emplaced within the Hyland Group, and less commonly within the Upper Schist.

Cretaceous-age deformation and intrusion are possibly related to north-northeast directed subduction and related arc-trench magmatism of the oceanic Farallon Plate beneath continental North America.

Figure 7.1: Regional Geology Setting



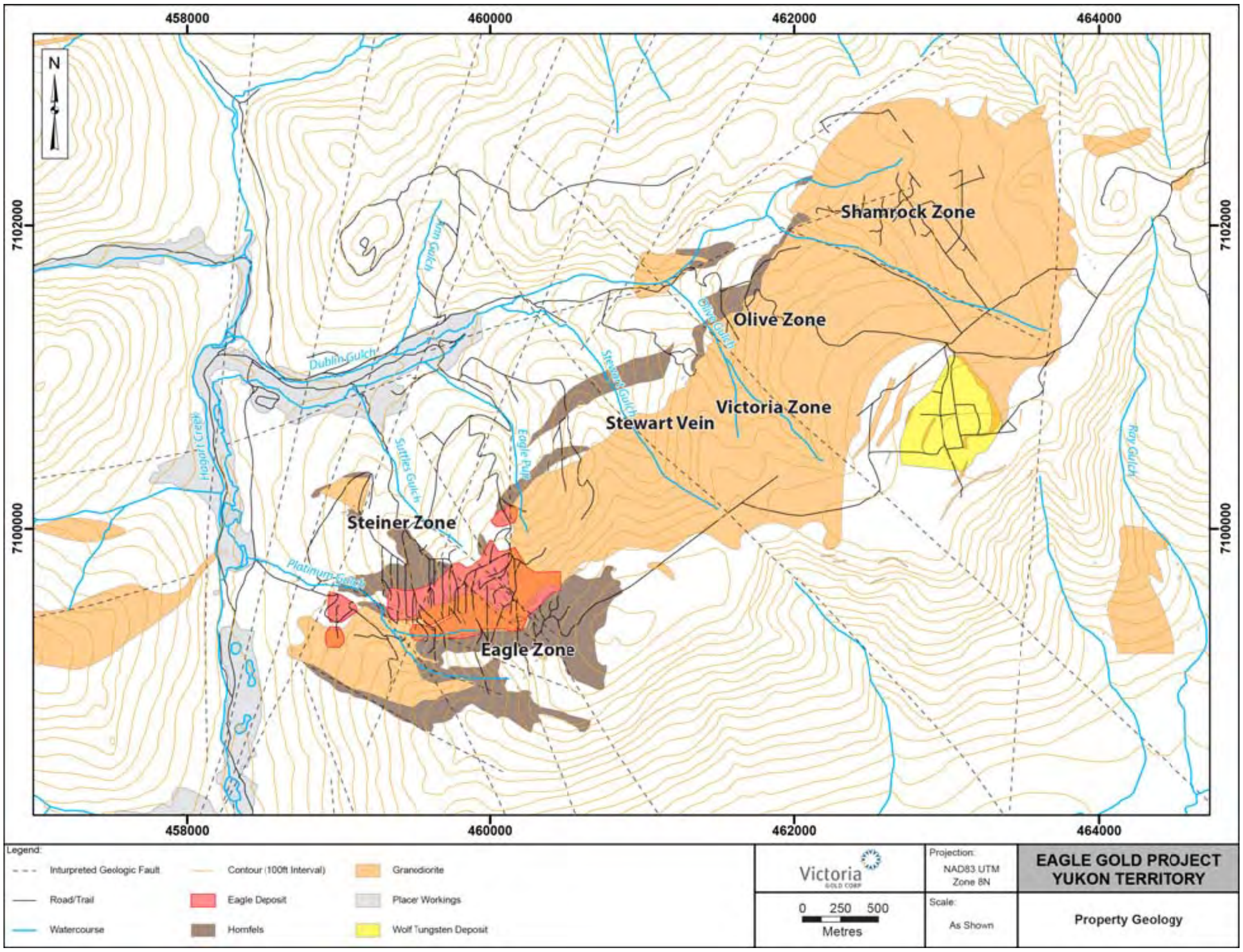
Source: Wardop (2012)

Numerous mineral deposits in the district are associated with the Cretaceous-aged intrusives and they are generally vein, shear, and skarn related. Gold, silver, lead, zinc and tungsten are the principal elements of economic interest. The Tombstone Suite forms part of the Tombstone Gold Belt, which is the eastern part of the Tintina Gold Province. The Tombstone Suite is the primary source of intrusion-hosted gold deposits in Yukon. The western portion of the Tintina Gold Province has been dextrally displaced approximately 450 km by the Tintina Fault and contains gold deposits that include Fort Knox, Pogo and Donlin Creek in Alaska. In Yukon, Brewery Creek and Dublin Gulch occur within the Tombstone Gold Belt.

7.3 Property Geology

The property is located on the northern limb of the McQuesten Antiform and is underlain by Proterozoic to Lower Cambrian-age Hyland Group metasediments and the Dublin Gulch intrusion, a granodioritic stock. The stock has been dated at approximately 93 Ma, and is assigned to the Tombstone Plutonic Suite (Figure 7.2).

Figure 7.2: Dublin Gulch Property Geology



Source: Wardrop (2012)

The Hyland Group is comprised of interbedded quartzite and phyllite. The quartzite is variably gritty, micaceous, and massive. The phyllite is composed of muscovite- sericite and chlorite. Limestone is a relatively minor constituent of this stratigraphic sequence.

The Dublin Gulch anticline, located midway between Dublin Gulch and Lynx Creek to the south, has folded the metasediments about an axis that trends at an azimuth of 070° and plunges gently to the west-southwest.

The metasediments are the product of greenschist-grade regional metamorphism. Proximal to the Dublin Gulch Stock, they have undergone metasomatism and contact metamorphism. A hornfelsic thermal halo surrounds the stock and within the halo, the coarse clastic components of the Hyland Group have been altered to quartz-biotite schist; the argillaceous components to sericite-biotite-chlorite schist and the carbonates to marble, wollastonite-quartz skarn and pyroxenite skarn. The halo extends from 80 to 200 m outward from the intrusive.

The Dublin Gulch Stock is comprised of four phases, the most significant of which is granodiorite. Quartz diorite, quartz monzonite, leucogranite and aplite comprise younger intrusive phases that occur predominantly as dikes and sills and cut both the granodiorite and surrounding country rocks. The stock has intruded the Hyland Group metasediments near their contact with the underlying Upper Schist.

The granodiorite stock is elongate, measuring approximately 5 km in length and trends 070°. It has a maximum width of approximately 2 km. The long axis of the stock is coincident with the axis of the interpreted Dublin Gulch anticline. Sheet-like sills of granodiorite extend from the stock and cut the metasedimentary strata at low angles.

The intrusive-metasediment contact dips shallowly to steeply to the north and northwest on the northern side of the intrusive, and steeply to the north or south along its southern margin. No chilled margin is apparent at the contact.

At least four periods of faulting have been documented in the Dublin Gulch area including low-angle thrusting and bedding-plane faults and normal faults with north, northeast, northwest, and easterly trends. North-trending faults are inferred to have displaced portions of the Dublin Gulch Stock and one of these is interpreted to form the eastern boundary of the Eagle Zone. No apparent fault offset to mineralization has been noted. The northeast and easterly trending structural directions are sub-parallel to mineralization trends and are likely in part pre-mineral structures.

7.3.1 Eagle Zone

The Eagle Zone gold occurrence is localized at the narrowest exposed portion of the stock, near its known western limit. The intrusive-metasediment contact is sharp but irregular and varies between steep attitudes that crosscut metasedimentary foliation, to shallow southwest dips parallel to foliation.

The Eagle Zone is comprised of sub-parallel extensional quartz veins that are best developed within the granodiorite proximal to both the hanging wall and footwall intrusive-metasediment contacts. Veining is apparently best developed on the hanging wall contact, but this may be more apparent than real as more drilling has taken place on the hanging wall side.

Veins are typically composed of white or grey quartz with subordinate potassium feldspar and strike at azimuths of 060° to 085°. They typically dip 60° south to vertical, and range in width from 1 mm to more than 10 cm. Contacts are typically sharp. Vein densities range from less than 1/m to more than 15/m, and average 3 to 5/m. The greatest concentration of veins appears to coincide with both the narrowest constriction as well as the local apex of the intrusion.

Sulphides account for less than five percent of vein material and occur in the centre, on the margin, and disseminated throughout the veins. The most common sulphide minerals are pyrrhotite, pyrite, arsenopyrite, chalcopyrite, sphalerite, bismuthinite, molybdenite and galena.

Secondary potassium feldspar is the dominant mineral in alteration envelopes. Sericite-carbonate is generally restricted to narrow vein selveges, although alteration zones of this type also occur with no obvious relation to veins.

Vein formation can be attributed to contrasts in cohesion and tensile strength between the intrusion and the enclosing metasediments. Embayments and narrow portions of the stock represent stress shadows that constitute favourable areas for rheological failure leading to the formation of extensional quartz veins.

Protrusions in the stock created favourable areas for the development of extensional shear-veining in the adjacent country rocks. Gold mineralization also occurs hosted within the metasedimentary rock package immediately adjacent to the granodiorite. This mineralization represents a portion of the Mineral Resource.

7.3.2 Olive Zone

The Olive Zone gold occurrence is localized at the contact zone on the northwest flank of the granodiorite intrusive. The intrusive-metasediment contact is sharp and steep to nearly vertical, and has a general northeast trend.

Olive is defined by sulphide and quartz-sulphide+carbonate veining at various orientations (parallel to conjugate) to the general northeast mineralized trend, possibly indicative of vein formation within dilational zones or conjugate fractures between two or more shear planes.

Sericitic alteration and sulphide mineralization are more pronounced than at Eagle, and oxidation is less well developed. Moderate to strong sericitic alteration is present throughout the Olive Zone.

Oxidation varies as well from local zones of total oxidation at surface to un-oxidized sulphide-bearing granodiorite at depth. A transition zone from near total oxidation to only sulphides has been defined based on core-logged oxidation codes. Mixed oxides-sulphides are present at surface in shallow trenches. Veins can be comprised of exclusively sulphides or, more commonly, sulphides associated with white quartz.

Over 97% of the gold mineralization in the Olive Zone is hosted in granodiorite just south of the stock-metasedimentary rock contact, with very minor metasediment-hosted mineralization.

7.4 Mineralization

The Eagle Zone is the principal concentration of mineralization within the property. Within the Eagle Zone, gold occurs in extensional quartz veins that are most abundant on the hanging and footwall contacts of the narrowest portion of the Dublin Gulch granodiorite near its known western limits. Subordinate quantities of gold mineralization occur in quartz veins within the adjacent metasediments. Veins strike at azimuths of 060° to 85°, sub-parallel to the intrusive contact and are commonly fractured by repeated movement along the host fractures.

The Eagle Zone is irregular in plan and is approximately 1,600 m long (east-west) and 600 m wide north-south. The Eagle Zone is near-vertical and has been traced for about 500 m below surface. Current drilling indicates that the mineralization is relatively continuous along this length and is open in several directions, including to depth.

Mineralization occurs as elemental gold, both as isolated grains and most commonly in association with arsenopyrite, and less commonly with pyrite and chalcopyrite.

The sulphide content in the veins is typically less than 5%; and, is less than 0.5% within the deposit overall, with 1 to 4% carbonate (calcite) present as a buffer, acid generation from the ore and waste rock is not expected to be an issue (Stantec 2011).

In descending abundance, the principal sulphides present are pyrrhotite, pyrite, arsenopyrite and chalcopyrite. Minor sphalerite, galena and molybdenite are also present. Scorodite and limonite are common weathering products.

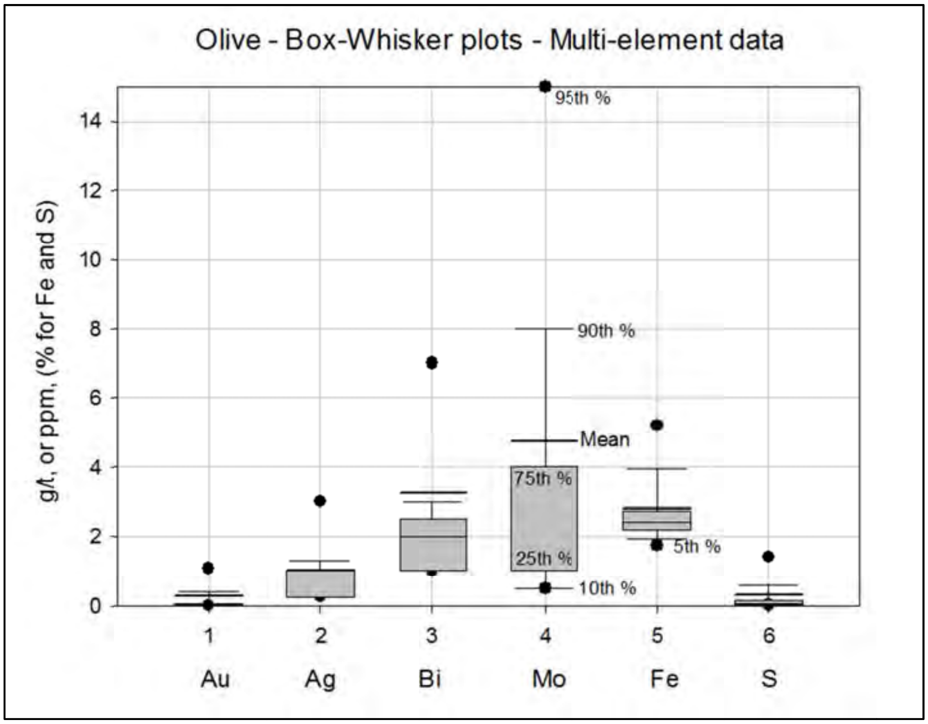
The Olive Zone is a narrow elongated zone sub-parallel to the intrusive-metasediment contact; located approximately 2.5 km northeast of the Eagle Zone. Olive measures approximately 20 to 80 m in width, 900 m in length, and has been drilled to approximately 175 to 250 m in depth. Compared to Eagle, the Olive mineralization is more associated with sulphides and quartz-sulphide veining in an interpreted shear-zone setting; with veining having an orientation at angles to the general northeast mineralized trend.

The Olive Zone differs from Eagle is some respects. Olive has more sulphide mineralization as both disseminated pyrite with moderate to strong sercitic alteration, and sulphide and quartz-sulphide veins, and is more tightly structurally controlled along the granodiorite-metasediment contact. Pyrite plus arsenopyrite (or arsenical pyrite) and quartz-pyrite veins to several centimetres in width have an average strike trend of azimuth 120°, and dips of 60° to 80° south, within the overall NE trending zone of mineralization. Vein densities vary significantly; however, trench exposures and assays indicate that good grade mineralization typically hosts multiple centimetre wide sulphide veins, on metre or less spacings, within areas of moderate to strong sericitic alteration with 3 to 5% disseminated sulphides. The most commons sulphides noted are pyrite, arsenopyrite, with minor to trace amounts of spahlerite, chalcopyrite, galena, bismuthinite, and molybdenite. Olive also has higher levels of silver than Eagle.

Multi-element geochemistry for Olive, based on over 17,300 analyses, shows the following:

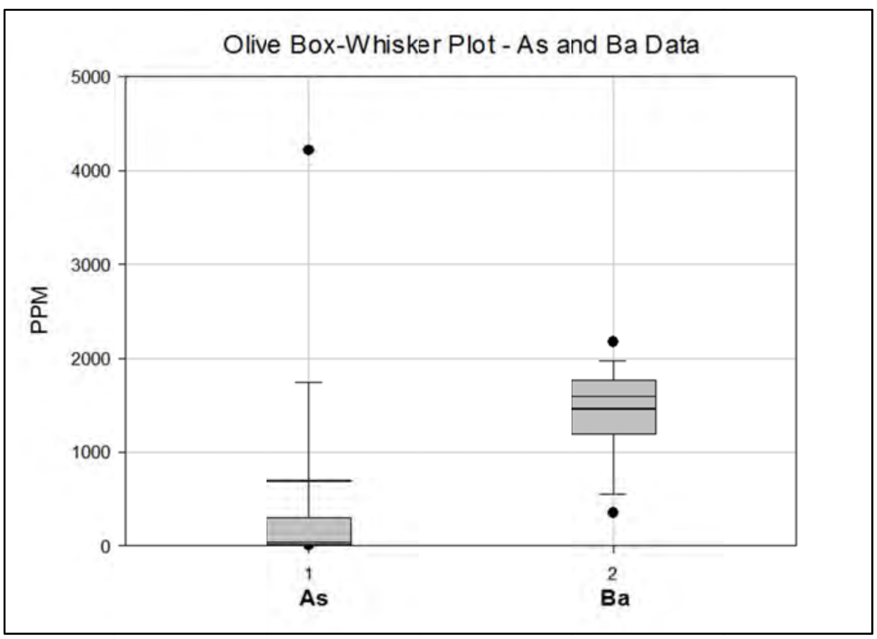
- A good Au-Ag-As correlation; with Au correlation coefficients of 0.50 with Ag, and 0.42 with As;
- A strong Au-Bi correlation coefficient of 0.74;
- A strong Ag-Bi-Cu-Fe correlation;
- Overall levels of associated elements at Olive are relatively low, as shown in the box-whisker plots of Figures 7.3 to 7.5. Similar multi-element associations at perhaps lower levels are indicated at Eagle, based on a less complete database.

Figure 7.3: Box-Whisker Plot for Olive - Au-Ag-Bi-Mo-Fe-S



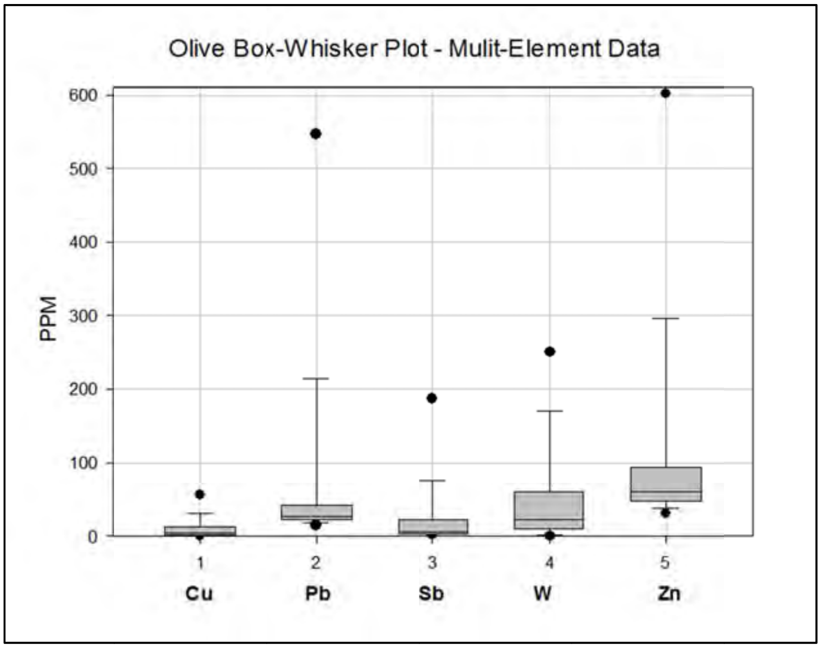
Source: AVMC (2016)

Figure 7.4: Box-Whisker Plot for Olive - As-Ba



Source: AVMC (2016)

Figure 7.5: Box-Whisker Plot for Olive - Cu-Pb-Sb-W-Zn



Source: AVMC (2016)

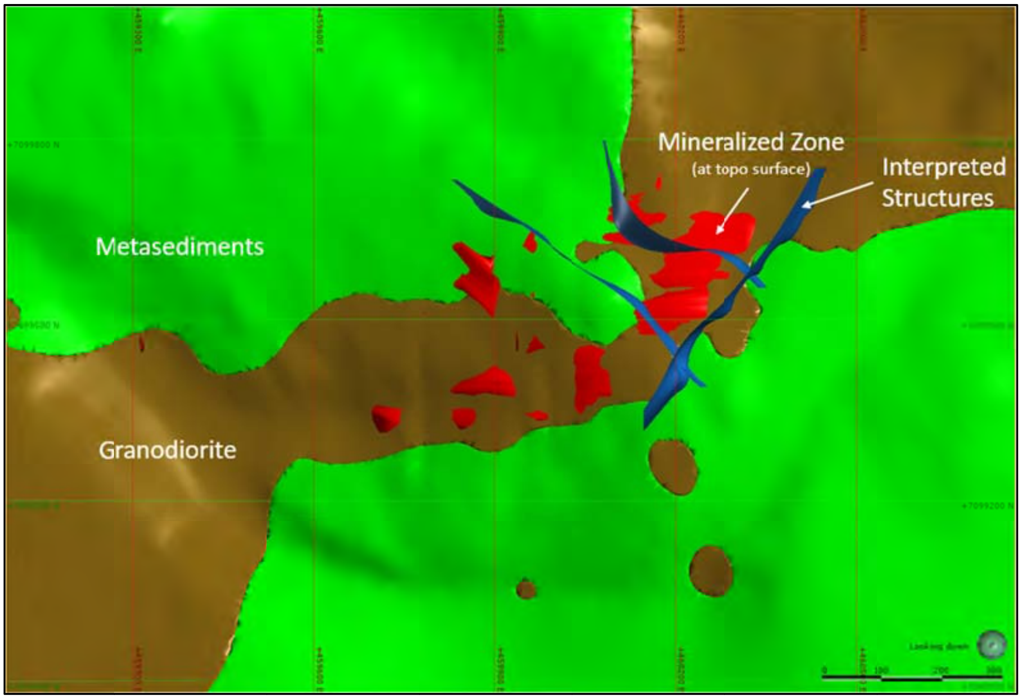
Several other mineralized showings occur within the property. Most of these are related to the Dublin Gulch granodiorite and are in part characteristic of RIRGS deposits (see Section 8) similar to the Eagle Gold deposit. Others are more characteristic of later structurally-hosted overprinting mineralization. The Wolf (formerly Mar) tungsten deposit is located approximately 3 km east-northeast of the Eagle Zone. Scheelite occurs in a calc-silicate skarn in metasedimentary rocks adjacent to the Dublin Gulch granodiorite.

A number of gold-bearing quartz-sulphide veins occur around the margins of the Dublin Gulch Stock. These veins are narrow (centimetre-scale), steeply dipping and generally strike at about 070°. Silver-quartz-sulphide veins also occur. These distal veins are infrequent relative to the sheeted vein system within the Dublin Gulch Stock and due to their small size, they are not a significant part of the Mineral Resource, with the exception of the Olive Zone.

Placer gold mining in the Dublin Gulch area began in 1895 and approximately 110,000 oz have been reportedly recovered to date. Placer gold is still being actively mined, particularly in the Haggart Creek area. Current placer gold production from these operations is unknown.

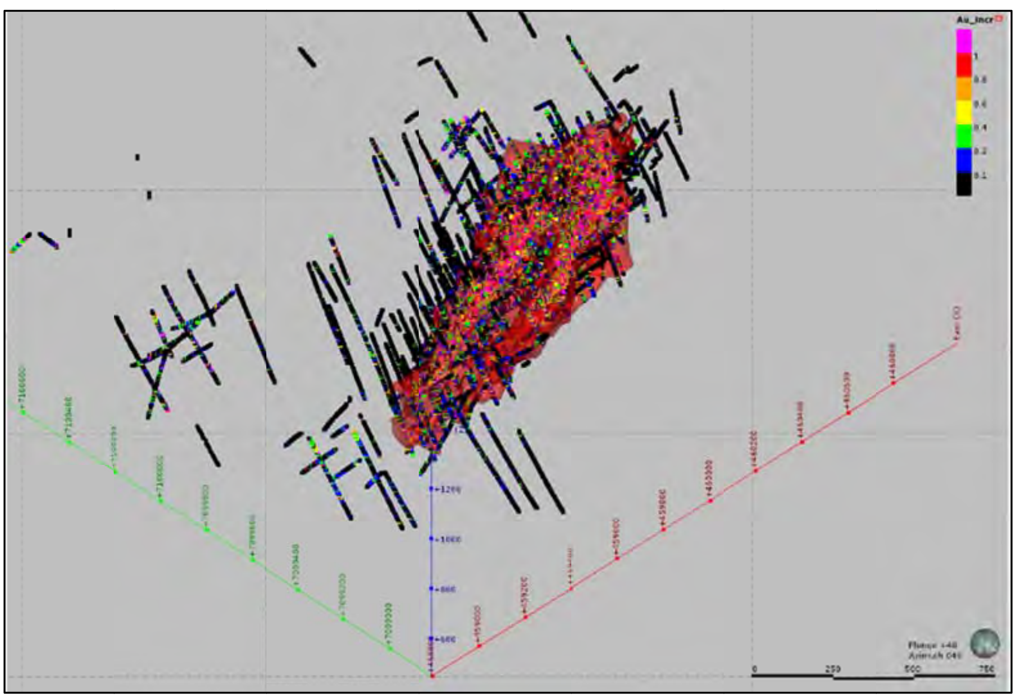
Figures 7.6 through 7.13 show representative images for Eagle and Olive.

Figure 7.6: Eagle Simplified Geology Plan Map



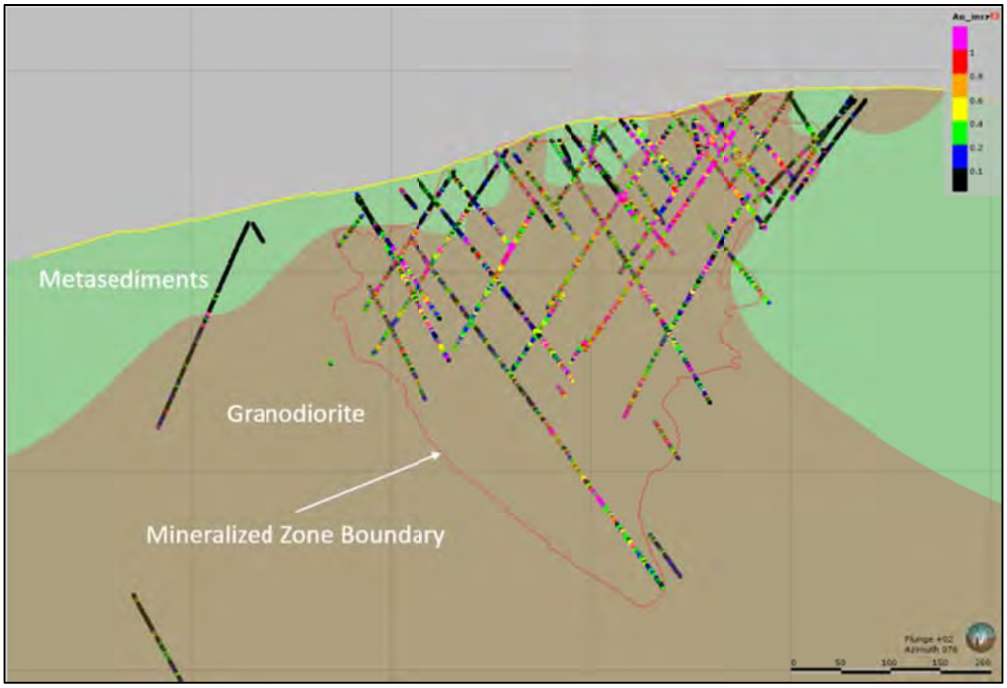
Source: AVMC (2016)

Figure 7.7: Eagle Drill Holes and Mineralized Shape - Perspective View



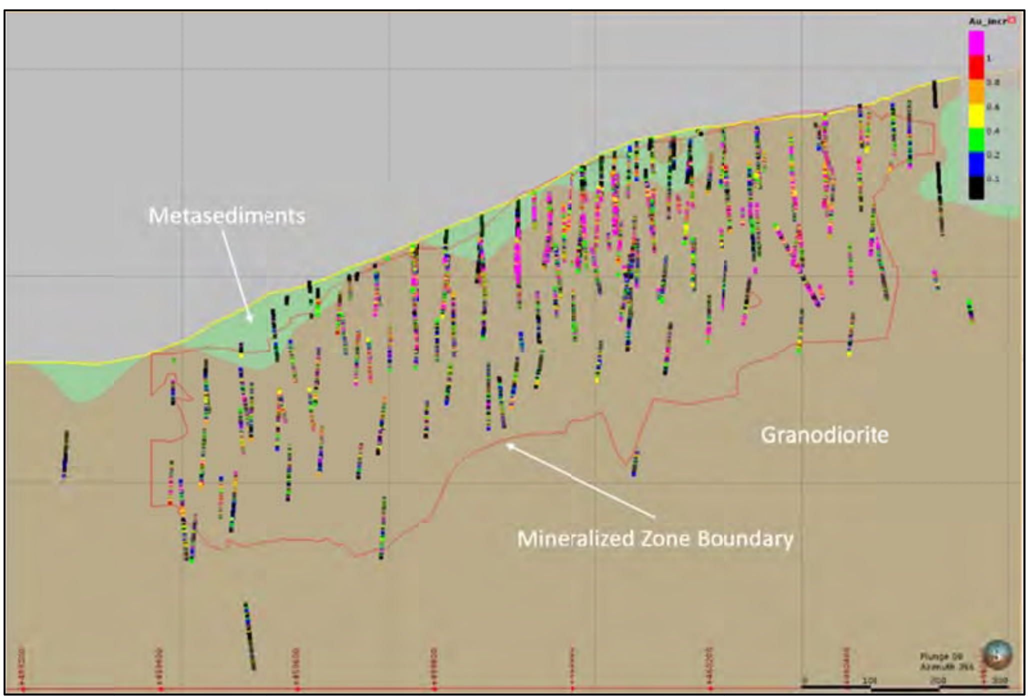
Source: AVMC (2016)

Figure 7.8: Eagle Geology and Drill Hole Assays, Representative Cross-Section - View to NE



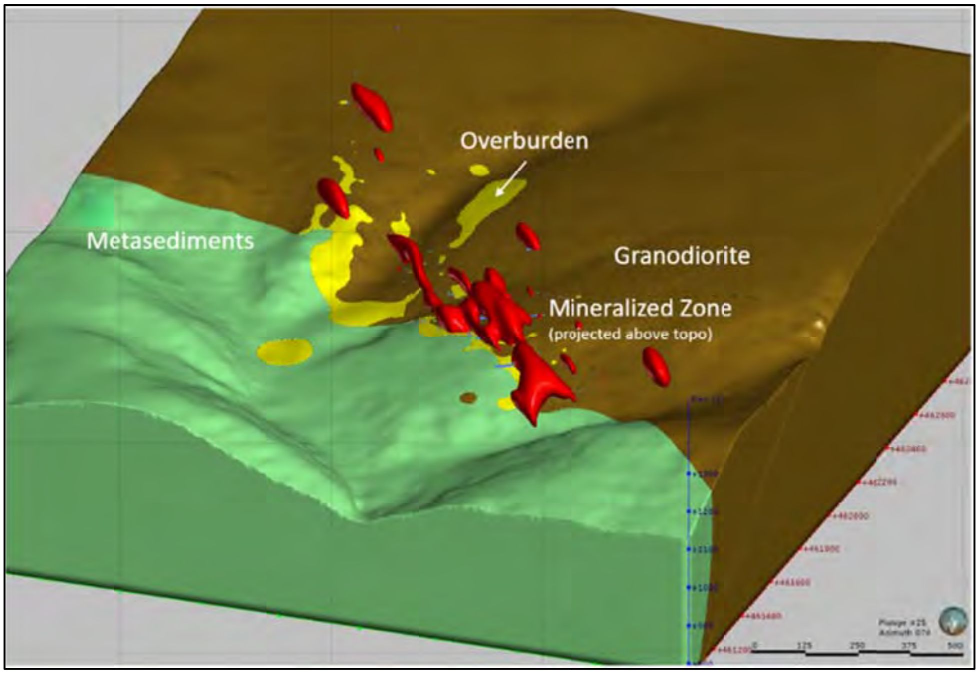
Source: AVMC (2016)

Figure 7.9: Eagle Geology and Drill Hole Assays, Long-Section - View to NW



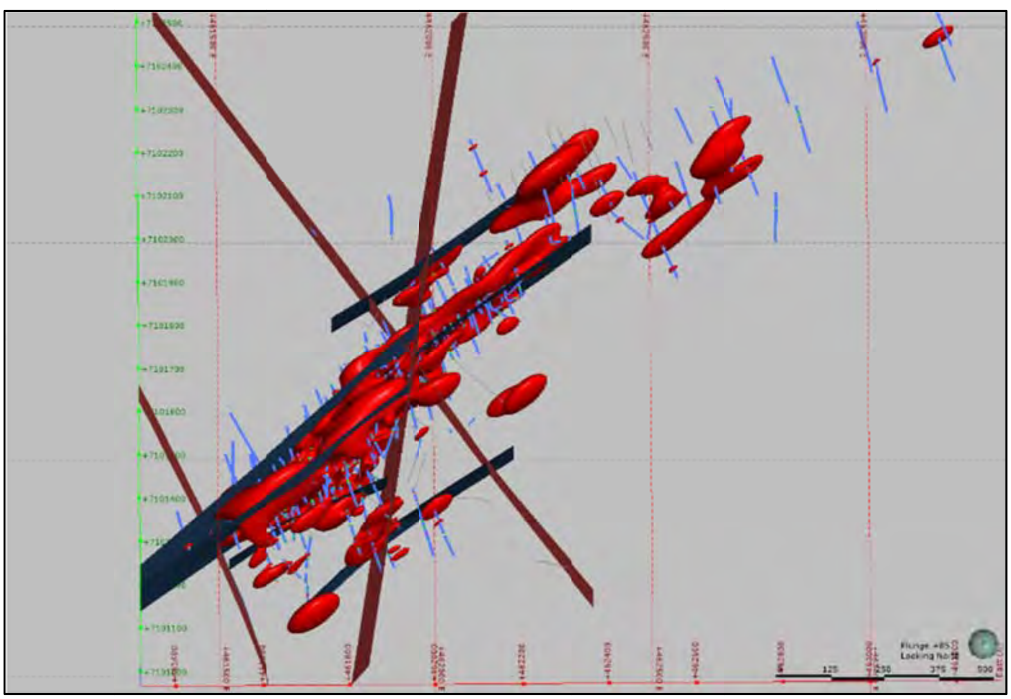
Source: AVMC (2016)

Figure 7.10: Olive Geology - Perspective View Looking NE



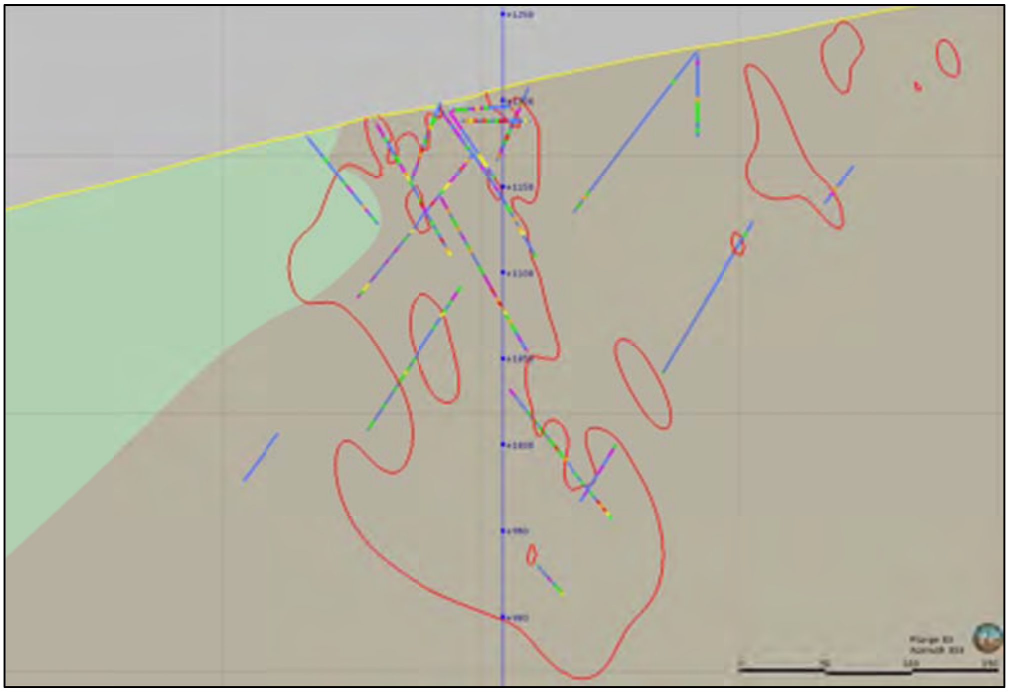
Source: AVMC (2016)

Figure 7.11: Olive Drill Holes, Mineralization Shape, and Interpreted Structures - Perspective View Looking down to the NE



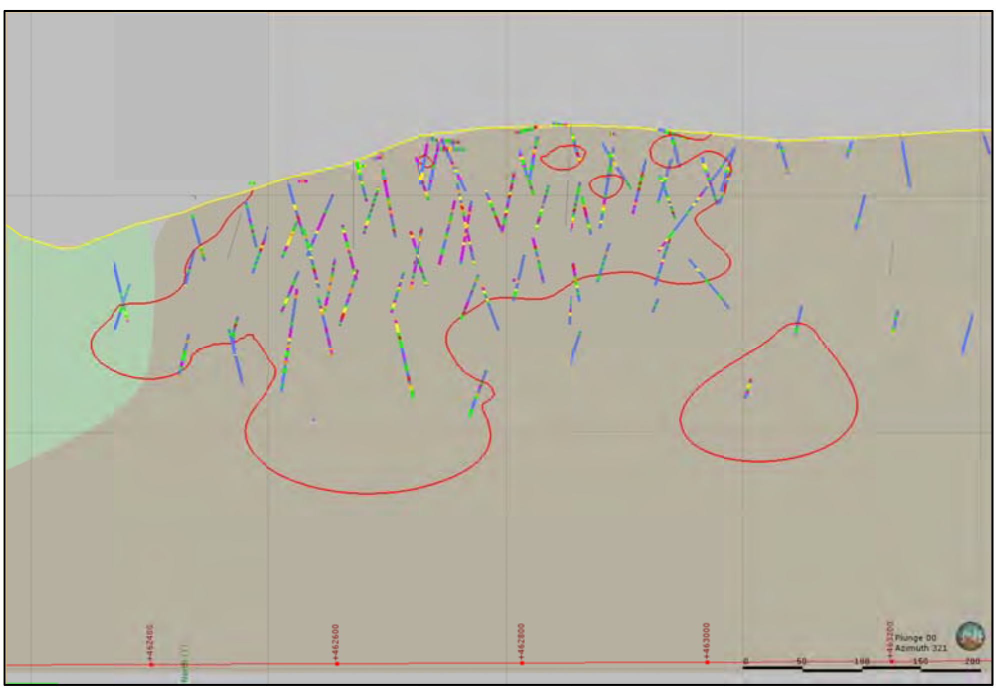
Source: AVMC (2016)

Figure 7.12: Olive Geology and Drill Holes Assays, Representative Cross-Section - Looking NE



Source: AVMC (2016)

Figure 7.13: Olive Geology and Drill holes Assays, Long-Section - Looking NW



Source: AVMC (2016)

8 Deposit Types

The Dublin Gulch intrusion is part of the mid-Cretaceous Tombstone Intrusive Suite of Alaska-Yukon granitoids, and the Eagle Zone belongs to the RIRGS class (Reduced Intrusion-Related Gold Systems) of mineral deposits. Gold mineralization in the Dublin Gulch intrusion shows strong similarities to the Fort Knox deposit in Alaska, including the presence of sheeted quartz veins and elevated levels of bismuth, arsenic, tellurium, and tungsten. The veins in the Eagle Zone consist of early quartz-scheelite with varied occurrences of pyrrhotite, pyrite and arsenopyrite, and are associated with K-feldspar and minor albite alteration envelopes. These are overprinted by sericite-carbonate and occasional chlorite alteration. The metasediments marginal to the intrusion are mineralized as well, but the bulk of the gold is hosted within the intrusive. The Dublin Gulch Stock is an elongate body trending 070°, with surface dimensions of approximately 6 x 2 km. Exploration for additional gold deposits is ongoing with excellent potential for further discoveries.

The Dublin Gulch intrusion is composed of mainly biotite hornblende granodiorite. Minor phases of diorite and granite occur within the intrusion. The overall low sulphide content of the rock, commonly less than 0.5%, and the presence of carbonate (Calcite 1 to 4%) make the rock non-acid generating. In a report prepared by SRK for Stantec in April 2011 (SRK, 2010: Geological Characterization and Water Quality Predictions Eagle Gold Project), SRK states that acid rock drainage (ARD) is not anticipated for the project.

RIRGS' class of mineral deposits are deposits that are:

- Metaluminous subalkalic intrusions of intermediate to felsic composition that lie near the boundary between ilmenite and magnetite series;
- Associated with carbonic hydrothermal fluids;
- A metal assemblage that variably combines gold with elevated bismuth, tungsten, arsenic, molybdenum, tellurium, and antimony as well as low concentrations of base metals;
- Associated with commonly weak hydrothermal alteration that is areally restricted;
- In a tectonic setting well inboard of inferred or recognized convergent plate boundaries; and
- Located in magmatic provinces best or formerly known for tungsten and/or tin deposits.

The RIRGS class of gold deposits was developed based on studies of gold and other mineral deposits hosted in granitoids in the Yukon and Alaska (Hart, C. R., 2007)

Additionally:

- 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;
- 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 halo;

- 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;
- The most distinctive style of gold mineralization in RIRGS deposits is sheeted arrays of parallel, low sulphide, single-stage quartz veins that are found over widths of tens to hundreds of metres and are preferentially located in the cupola of the pluton. These veins are unlike multidirectional, interconnected stockworks characteristic of porphyry systems or antithetic tensional vein arrays typical of orogenic deposits;
- Mineralized plutons have characteristics that indicate the likelihood of generation of hydrothermal fluids, high volatile contents, fluid exsolution, rapid fractionation and zonation, including the presence of porphyritic textures, aplite and pegmatite dikes, quartz and tourmaline veins, greisen alteration, miarolitic cavities and unidirectional solidification textures, preferably in pluton apices;
- RIRGS deposits are associated with felsic, ilmenite-series plutons that lack magnetite, have low magnetic susceptibilities and aeromagnetic response, and have 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; and
- Intrusion-related deposits are coeval with their associated, causative pluton.

8.1 Geological Model

The Eagle Zone geological model is simply described as a zone of mineralization containing sheeted quartz veinlets and post-veining fracturing hosting gold mineralization, located near the apex of a granodioritic stock, and mostly within the stock. As the gold mineralization is generally but not directly related to quartz veining, the geological modelling for resource estimation has been constructed based on the mineralization rather than the veining. A mineralized shape, based on the gold grades in drill holes, has been constructed to confine the resource estimation.

The Olive Zone geological model is that of a structural zone on the flank of the granodiorite stock, hosted essentially entirely in granodiorite, sub-parallel to the intrusive metasediments contact. Detailed structural controls that define the mineralization are interpreted, but not directly defined, and do not offset mineralization. Similar to Eagle, the mineralization for resource estimation purposes is confined by a mineralized shape based on gold grades in drill holes.

9 Exploration

9.1 Previous Exploration

Prior to Victoria Gold's involvement with the property, numerous drilling campaigns were conducted on the property as described in Section 6 - History. Exploration drilling for intrusive-hosted gold mineralization began in the early 1990's, and continued sporadically by several owners through 2004 with the work of StrataGold. Victoria Gold acquired StrataGold in 2009.

The majority of Victoria Gold's exploration work since the 2012 Wardrop FS has been in-fill drilling at the Eagle Zone, and exploration efforts including trenching, geophysical surveys and diamond drilling at the Olive Zone.

9.2 Victoria Gold Exploration

Victoria Gold completed a FS on the Eagle Zone in 2012 (Wardrop, 2012). Post the FS resource estimate of late 2011, Victoria Gold conducted a targeted in-fill drilling program of an additional 130 drill holes in the Eagle Zone, for the purpose of better definition of Measured and Indicated Mineral Resources. The drilling program was conducted in the winter of 2011-2012.

The Olive Zone had been explored prior to Victoria Gold's ownership, with initial drilling in 1992, and sporadic follow-up drilling for a total of 19 holes by 2007. Victoria Gold conducted additional drilling of 58 holes in 2010-2012, and in-fill drilling in 2014 with 61 holes and 2016 with 89 drill holes. The Olive Zone is defined as a Mineral Resource for the first time in this FS update report.

Additional exploration work conducted at the Olive Zone included 17 shallow trenches in 2014 and 29 trenches in 2016 to expose and sample oxidized sulphide mineralization and assist definition of the surface trace and extensions to mineralization. As well, a program of third IP-Resistivity geophysical surveys was conducted over the core area of the Olive Zone in 2014, which shows a good correlation of IP chargeability highs with the modelled zone of anomalous gold mineralization in drilling, a direct association of the gold with increased sulphide content. Trenching, sampling and IP-Resistivity surveys are a useful exploration tool to define gold mineralization at Olive and possible extensions to the northeast.

As the majority of work on the Eagle and Olive zones was drilling, the details of that exploration work are discussed in Section 10 of this report

10 Drilling

Previous project drilling has been accomplished by several different companies, from 1977 through 2009, when Victoria Gold became Owner of the property, as indicated in Table 10.1. Discussions of the previous drilling are included in prior NI 43-101 technical reports issued by Victoria Gold (StrataGold) as listed in Section 27.0. A substantial amount, 91 holes for 21,875 m, of additional drilling was completed by Victoria Gold in late 2011 and 2012, after completion of the Mineral Resource estimation in the 2012 FS. This drilling has been incorporated into an updated Mineral Resource estimate for this FS report, as described in Section 14 of this report.

Victoria Gold conducted a considerable amount of diamond drilling on the Olive Zone in 2014 and 2016, 160 holes totalling 22,167 m and 41 trenches totalling 1,910 m. The resulting data was sufficient for the initial Mineral Resource estimation for the Olive Zone, as described in Section 14 of this report.

Drilling was conducted for the in-fill and the definition or mineral deposit boundaries, metallurgical samples, and geotechnical information.

Since 2012, core drilling was done by Kluane Drilling, New Age Drilling Solutions, of Whitehorse, Yukon, and LynCorp Drilling Services Inc. of Smithers, BC. RC drilling was conducted by Midnight Sun Drilling Inc. of Whitehorse, Yukon. Holes were surveyed by a downhole instrument from REFLEX, as soon as the hole was stable, and with no interference from casing, at 75 m intervals thereafter, and at the bottom of the hole.

Core was drilled primarily as HQ core size. Core was transferred from the core tube into boxes by the drill crew who marked the end of each run with a wooden marker. Hole depth was measured in imperial units (feet) and subsequently converted into metric units (metres) on the depth markers. Core was transported by the drilling company from the drill site to the core logging facility that is located in the camp complex.

Core was laid out for logging inside the core shed and then measured for rock quality designation (RQD), recovery and permanent (aluminum) labels were then affixed to the core boxes. Core was then washed and marked for sampling. Most samples were 1.5 m in length but did not exceed 2 m in length and were shorter if lithological contacts or significant variations in sulphide content were present. In general, the entire length of the hole was sampled.

Core logging observations were recorded on paper and subsequently transferred to a computer database. Significant observations include rock type, weathering, alteration, foliation angle and intensity, fracture angle and intensity as well as descriptions of any veins present. Several types of alteration (oxidation, silicification, sericitization) were quantified from zero to five with zero equating to no alteration and five representing complete alteration. There is no unique convention with respect to fracture intensity although the attempt was made among those logging to apply the same criteria.

When logging was complete, sample tags were affixed to the core box at the start of each sample interval. Each sample tag was comprised of three pieces: one for the core box, one for the sample bag into which the sample was placed, and the third which remains in the sample book.

Given the variable orientation of mineralized quartz veins, the relationship between sample length and thickness of mineralization is also variable. However, given that the sampling was continuous and the mineralization is a bulk target, the variability of this relationship is not considered to be detrimental to the objectives of the sampling program.

In addition to the procedures described above, which pertain to all holes drilled since 2012, those holes drilled for geotechnical testing were logged by Mining Plus or BGC Engineering Inc., geotechnical specialists, for a range of parameters relating primarily to pit design.

Core drilling recoveries are generally +90%, and low recovery intervals are addressed in the resource estimation process of the exploratory data analysis (EDA). Several PQ size core holes were drilled for the purpose of metallurgical sample collections.

RC drilling was part of the in-fill program at Eagle, as detailed investigations of RC versus core drilling show no particular bias in the RC assays over core.

Drilling was done as angle holes across the primary strike orientation of the mineralization. The drilling methods, and sample handing procedures are in line with industry norms, and are acceptable methods for defining the gold mineralization at the Eagle and Olives Zones.

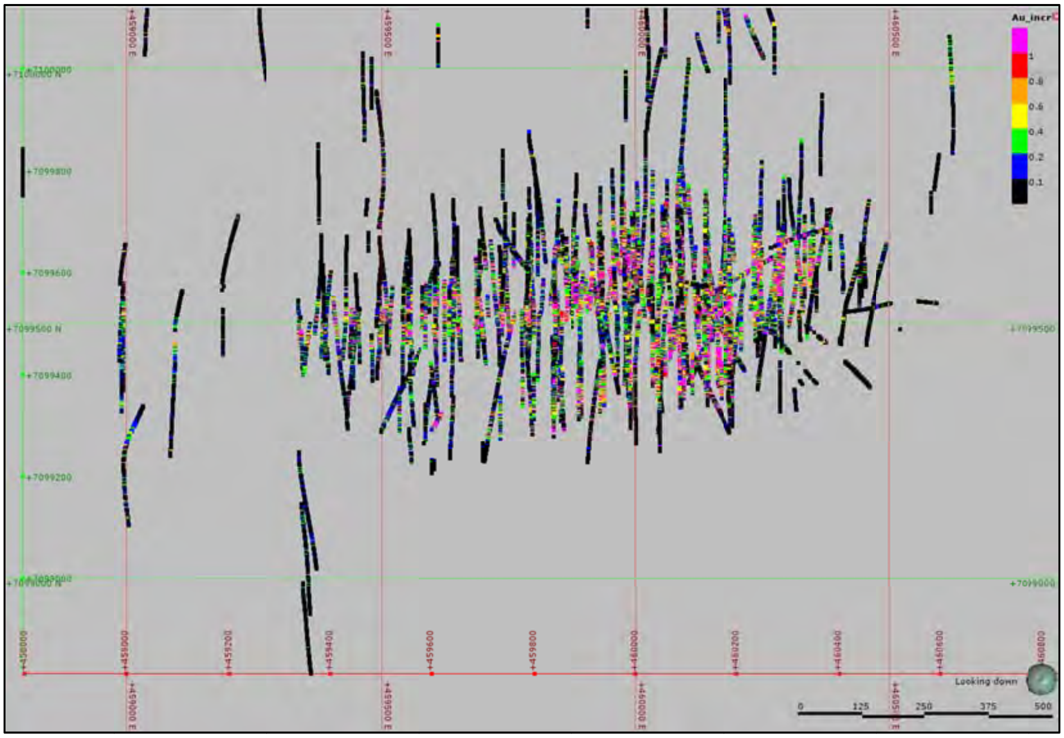
Core was sawn in half by diamond saw; one half was bagged for assaying and the other half was kept for reference. The sample to be analyzed was put in a plastic bag that contained another small zip-lock type bag with a sample tag and a piece of plastic flagging bearing the sample number. The sample number was also written on the outside of the bag. Each bag was then closed by cable ties and combined with others to fill woven plastic “rice bags” for shipping. Each rice bag was labelled with the numbers of the samples it contained. The rice bags were expedited by a contract shipper who picked up the samples in camp and delivered them to the assigned laboratory in Whitehorse and Vancouver. Standard chain-of-custody forms were used for the shipping process.

The boxes with the half-core are stored out of doors on covered racks or cross-piled on pallets in the central core storage facility.

Holes were generally sampled in their entirety, unless recovery was particularly poor in any single drilled interval, or the recovered material was considered highly unlikely to be significantly mineralized. Figure 10.1 shows the locations all drill holes for the Eagle Zone, and Figure 10.2 shows the location of all drill holes for the Olive Zone.

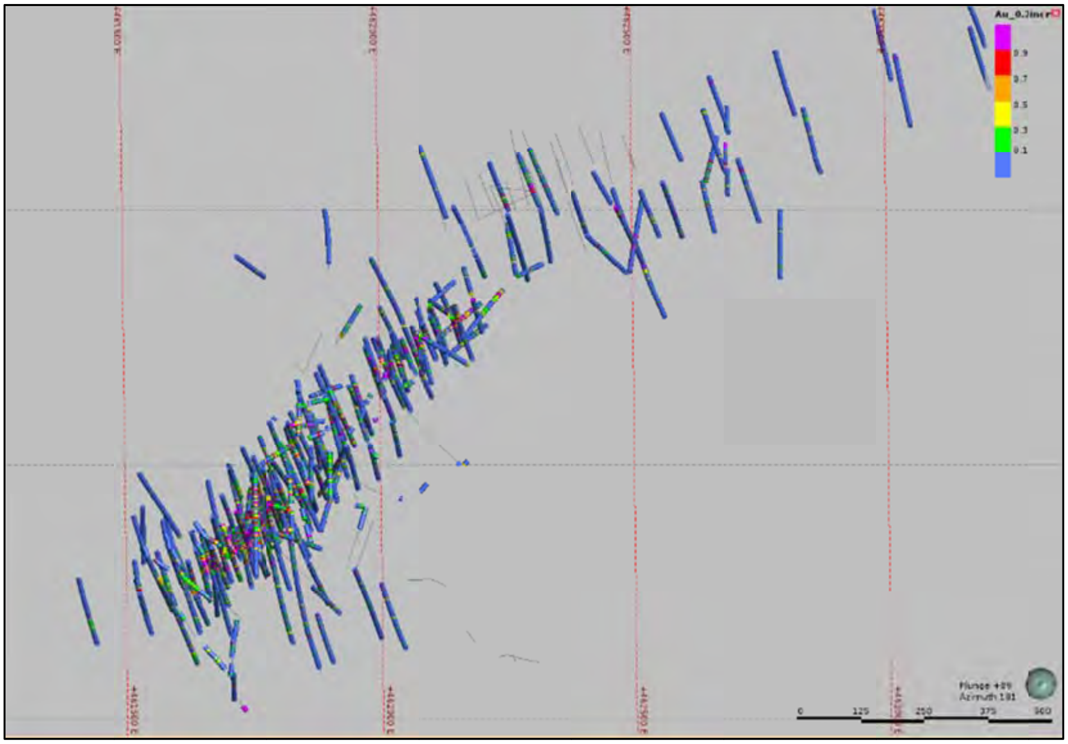
The author concludes the drilling methods used by Victoria Gold since 2012 are consistent with industry standard drilling procedures, consistent with previous drilling campaigns, and are sufficient to augment the previous drill hole database for the intended purposes of in-fill exploration, metallurgical samples and geotechnical information.

Figure 10.1: Plan Map Showing the Distribution of Drilling for the Eagle Zone



Source: AVMC (2016)

Figure 10.2: Plan Map Showing the Distribution of Drilling for the Olive Zone



Source: AVMC (2016)

10.1 Drilling Summary

Table 10.1: Project Drilling by Year - Eagle

Company	Year	Number of Holes	Metres Drilled	Type
Canada Tungsten	1977	65	11,315	DDH
Queenstake Resources	1986	4	705	DDH
Can Pro	1989	4	653	DDH
Ivanhoe Goldfields	1991	16	2,410	DDH
Amax Gold Inc.	1992	13	1,943	DDH
Amax Gold Inc.	1993	56	7,729	RC
Amax Gold Inc.	1993	10	1,476	DDH
Ivanhoe Goldfields	1993	10	2,078	RD
First Dynasty Mines	1995	40	8,354	RC
First Dynasty Mines	1995	25	4,946	DDH
New Millennium Mining	1996	21	4.114	DDH
New Millennium Mining	1996	37	5,271	RC
New Millennium Mining	1996	19	189	Auger
New Millennium Mining	1996	33	797	Water
StrataGold	2005	34	8,105	DDH
StrataGold	2006	10	4,282	DDH
StrataGold	2007	20	5,627	DDH
StrataGold	2008	15	4,429	DDH
Victoria Gold	2009	10	5,122	DDH
Victoria Gold	2009	4	1,321	Geotech
Victoria Gold	2010	20	3,592	DDH
Victoria Gold	2010	5	1,341	Geotech
Victoria Gold	2011	3	616	Geotech
Victoria Gold	2011-2012	33	4,337	RC
Victoria Gold	2011-2012	58	17,538	DDH
TOTAL		**565**	**104,180**	

Source: Wardrop (2012), modified by AVMC (2016)

Table 10.2: Project Drilling by Year - Olive Zone

Company	Year	Number of Holes	Metres Drilled	Type
Prior owners	1991, 1992	7	959	RC and DDH
Prior owners	2007	5	868	DDH
Prior owners	2007	10	707	Trenches
Victoria Gold	2010	19	4,144	DDH
Victoria Gold	2011	24	4,486	DDH
Victoria Gold	2011	4	300	RC
Victoria Gold	2012	11	2,997	DDH
Victoria Gold	2014	61	8,594	DDH
Victoria Gold	2014	10	1,027	Geotech
Victoria Gold	2016	89	12,546	DDH
Victoria Gold	2014	17	885	Trenches
Victoria Gold	2016	34	1,025	Trenches
TOTAL		**291**	**38,538**	

Source: AVMC (2016)

11 Sample Preparation, Analyses and Security

11.1 Sample Preparation and Security

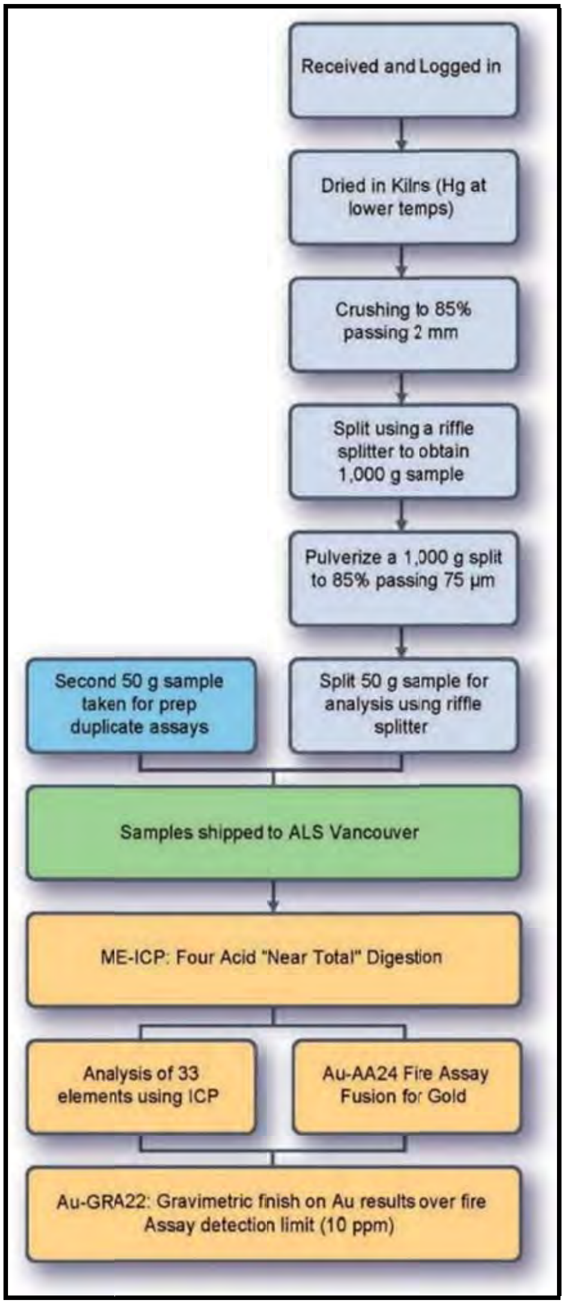
The following discussion in this section is derived from the Wardrop 2012 FS, as the general procedures have not changed since the sample preparation, analyses and security procedures in use by Victoria Gold for 2011, with the exception of the samples from the 2016 drilling on the Olive Zone, which instead underwent contract sample preparation on-site by SGS Canada Inc., and with selected sample pups shipped from site to the SGS analytical lab in Vancouver.

Prior to 2016, samples were shipped from camp to the ALS Chemex or Acme Analytical (Acme) labs prep laboratory in Whitehorse. The samples were dried, crushed, split and pulverized, and a 50 g split was sent to the ALS Chemex laboratory in North Vancouver or the Acme Analytical Labs laboratory in Vancouver for analysis.

The analytical procedure used by ALS Chemex and Acme is described in Section 11.2. The sample preparation and analytical procedure is summarized in Figure 11.1.

Chain-of-custody measures under Victoria Gold's control were followed with the shipping of samples from camp to Whitehorse and from Whitehorse to North Vancouver. Receipt of analytical results was restricted to key personnel.

Figure 11.1: Eagle Sample Preparation and Analytical Flowchart



Source: Wardrop (2012)

11.2 Analytical Procedures

The standard analytical procedure for the Eagle Zone was carried out at ALS Chemex in North Vancouver as follows:

- A 50 g sub-sample was taken from the 150 g pulp sample by withdrawing two to three scoops of material from different places in the envelope;
- The 50 g sample was subjected to a gold fire assay with atomic absorption spectroscopy and a 27 to 33 element inductively coupled plasma (ICP) analysis consisting of a four-acid “near total” digestion by hydrofluoric acid (HF)-nitric acid (HNO3)-perchloric acid (HClO4) digestion, hydrochloric acid (HCl) leach and ICP-atomic emission spectroscopy (AES); and
- All results with gold greater than 10 ppm were subjected to a fire assay with a gravimetric finish (Wardrop, 2009).

The ALS Chemex laboratory in Vancouver, the primary assay lab in use for the project work at Eagle Gold, is accredited to International Organization for Standardization (ISO)/International Electrotechnical Commission (IEC) 17025-2005 standards under the Standards Council of Canada, which provides specific assessments of the laboratory’s analytical capabilities. ALS Chemex laboratories in North America are also ISO 9001:2008 registered through SAI Global, ISO registration and accreditation provides independent verification that a quality management system (QMS) is in operation at the analytical laboratory. ALS Chemex is a worldwide based analytical company that has been providing analytical services to the mining and exploration industry of North America for over 30 years. The ALS Chemex analytical laboratory is located at 2103 Dollarton Hwy, North Vancouver, BC, Canada. ISO registration applies to the ALS Chemex preparation lab located at 78 Mt. Sima Road, Whitehorse, Yukon, Canada.

Approximately 5% of assays were re-assayed for gold alone by Inspectorate Exploration & Mining Services Ltd. (Inspectorate) laboratories at 11620 Horseshoe Way, Richmond, BC, Canada as the umpire (external check).

Inspectorate is used as a secondary lab for check assays. Inspectorate has ISO 9001:2008 certification and is an internationally known and reputable analytical laboratory that provides assay services to the exploration and mining industry.

The same above-described analytical procedure was used for drilling samples from the Olive Zone, most of which were collected in 2014 and 2016. However, the analytical labs involved were Acme Labs in Whitehorse (sample preparation) and Vancouver (analytical) in 2014, and an on-site SGS preparation lab in 2016, as well as the SGS labs in Vancouver during 2016.

Acme Labs has ISO/IEC 17025:2005 Accreditation and ISO 9001 Registration, and is a well-known and accepted mining and exploration industry utilized commercial lab. SGS labs is a worldwide leader in analytical services and is ISO 14001:2015 Certified.

All laboratories used for the analytical purposes are independent of Victoria Gold.

11.3 Quality Assurance and Quality Control Procedures

The Eagle Zone and Olive Zone drill programs employed blanks, duplicates and standards as part of the quality assurance/quality control (QA/QC) program. The following description of the materials, procedures and results has been adapted from a 2010 QA/QC document prepared by Victoria Gold.

- Crushed dolomite, purchased from a garden-supply centre, was used as blank material. Blanks were made by scooping roughly 200 g of crushed dolomite into a bag which was then added to the sample stream. Three blank controls were added for every 100 samples, usually where the sample numbers ended in 16, 56 and 96, although some were added in other locations according to local mineralizing conditions and at the discretion of the logging geologists;
- Drill core duplicates were obtained by submitting both halves of the core for analysis; with one half representing the original (normal) sample and the other half the duplicate. The gap left in the core box was marked by a piece of wood or polyvinyl chloride plastic pipe;
- Preparation duplicates were collected at the sample preparation stage by splitting a crushed portion of the sample, which was then pulverized. These samples were then issued to the assaying laboratory for analysis; and
- Standard Reference Material (standards) were obtained from Analytical Solutions Ltd., Toronto, who supplied six certified Ore Research & Exploration Assay Standards (OREAS). These are listed in Table 11.1, together with their mean values and lower and upper limits of two standard deviations.

Table 11.1: Standard Reference Material Statistics

OREAS Standard	Mean Value (Au ppm)	Low Threshold (Au ppm)	High Threshold (Au ppm)
152a	0.116	0.106	0.126
5Pb	0.098	0.092	0.105
52c	0.346	0.312	0.379
50c	0.836	0.78	0.891
15Pa	1.02	0.96	1.07
15Pb	1.06	1	1.12
6Pc	1.52	1.39	1.66
60b	2.57	2.35	2.78

Source: AVMC (2016)

11.4 2012-2016 QA/QC Results

The authors examined the resource database inclusive of 2011-2012 assay results for the Eagle Zone, and the 2011-2016 Olive Zone assay results. The drill programs employed blanks, duplicates and standards as part of the QA/QC program, in a similar fashion to the 2010 QA/QC procedures.

Summary QA/QC results are stated here for the post-2012 Wardrop FS data on Eagle and 2011-2016 data on Olive.

- A sufficient number or standards, 580 SRM's or Standard Reference Materials, representing 2.7% of the total number of samples for Eagle were inserted into the drilling sample batches sent for analysis. Results showed only one out-of-range analysis;
- Blanks representing 0.2% of the total number of samples for Eagle showed only five instances of assay values greater than 0.05 g/t Au, and none greater than 0.10 g/t Au;
- A total of 1,307 standard samples were inserted into the Olive sample stream; 22 failed to assay within +/- two standard deviations of the control value; six of which assayed high, and the remainder lower than the standard value;
- A total of 646 blanks were inserted into the Olive sample stream, one of which failed;
- In all cases for Eagle and Olive, if other standard or blank samples were not included in the batch for which a standard or blank failure occurred, then the batch was re-run.
- Field duplicates and sample preparation duplicates for Eagle and Olive show acceptable ranges of scatter relative to the original assays; and
- The QA/QC program for Eagle and Olive resulted in no significant identified issues.

The pre-2011 sampling, preparation, security and QA/QC procedures have been described in previous technical reports, have been reviewed by the authors, and are consistent with current procedures. The authors consider the 2011-2016 sampling, sample preparation, security, analytical procedures, and QA/QC procedures to be consistent with industry standards, and the results obtained verify the data as acceptable for use in resource estimation.

12 Data Verification

12.1 Verifications by Previous Workers

Previous work by others, as described below, has verified the Eagle Zone database as sufficient for use in the Mineral Resource estimation.

Wardrop conducted data verifications in 2006 and 2008 in relation to Mineral Resource estimation and reporting.

Data verification was also conducted by SRK in 2011, for the purpose of a resource model used in the 2012 FS, and is described in the Wardrop NI 43-101 Technical Report dated April 18, 2012. An extract from this report is summarized as follows below.

"A site visit verified the geology and select drill hole collar coordinates. It also confirmed the geology model of steeply dipping quartz veins and veinlets dominantly hosted in Granodiorite. A visual inspection of select drill core verified the presence and direct relationship of gold assays with quartz veins and veinlets in the Granodiorite and metasedimentary host rocks. Spot checking of the drill hole assay database against the assay certificates noted approximately a 1.7% error rate. A statistical evaluation and visual examination of the data in 3D verified the prior and current use of 13.0 g/t Au as a capping grade for high-grade gold assays, and visually demonstrated hole-to-hole continuity of mineralization. Database errors noted were deemed to have minimal effect on the Mineral Estimate, and SRK concluded that the "Eagle Gold Deposit database is sufficiently well defined, documented, and verified, to allow for use in resource estimation and for definition of reserves in a Feasibility Study."

12.2 Verifications by the Authors of this Technical Report

Victoria Gold's 2016 database included 130 additional drill holes (RC and core), completed since August 2011, for an increase of 39% of data, internal to the mineralized wireframe, as compared to the data used in the resource estimate of the 2012 FS.

The authors undertook a re-examination of the post-2012 FS database by completing the following steps:

- Verifying the database for 2011 and 2012 data against the assay certificates; 14,661 assays representing 27% of the total data were checked and verified with less than 0.5% error rate noted;
- Examining the QA/QC data for 2010 to 2012 that was deemed acceptable;
- Examining in-house versus ALS Chemex bulk density data for use in the resource model;
- Extensively examining the RC versus core assays data, for potential bias and identification of holes or assay intervals to exclude from the resource estimation; and
- Verifying the oxidation surfaces.

12.2.1 Database Verifications

In 2013, the following database verification checks were completed as part of an interim Mineral Resource evaluation. The authors verified the collar and down hole survey procedures. Several holes were identified hanging a few metres above the topo surface. Collar elevations not matching topo were observed in the field to be a result of side-hill road cut-and-fill of 1 to 3 m; therefore, no adjustments were made to collar elevations. Down hole survey data were checked for kinks in dip and azimuth. Verification was done visually and statistically in the form of dip change per metre and BRG (bearing azimuth of drill hole direction) change per metre histograms. It was concluded that there are minimal issues within acceptable limits, with kinks in down hole curves.

Assays and geology on sections were checked for any suspicious/outlier data. Two holes, 92-031R and 96-267R were set as not verified due to suspicions of grade relative to other nearby core holes. A total of 152 core and 168 RC holes were without core recovery data. Core recovery greater than 100% was also noticed in the database, mainly due to wrong measurements of core lengths while logging. The issue with core recovery existed mainly in the 1990's data. The database field column was created with fixed core recovery, setting core recovery greater than 100% to 100%. The authors are of the opinion that this will not have a material impact on resource estimation. A total of 48 intervals with "from-to" continuity errors were noticed and were fixed before importing data into Datamine software.

As an independent check, the authors compiled assay data from 443 lab assay certificates of the 2011 to 2012 drilling. These constituted of 14,661 assays out of 53,239 total assays (27% of total assays). The compiled assay file was compared with the assay database supplied by Victoria Gold; insignificant errors were identified in the database and were fixed before importing in Datamine software.

In the case where more than one gold assay method was used, the final assay rules that were applied were:

- Atomic Absorption (AA) finish if Au <=10ppm; and
- Gravimetric finish results if Au >10ppm.

Minor inconsistencies in the logging codes were identified and fixed by Victoria Gold geologists before using logging data in the resource estimation. Based on data verification carried out, the verification flag field was set as verified "Y" or "N" in the collar file. The following rules were applied for setting verification flag to "N":

- Holes with missing assays or not assayed;
- Holes with missing geology and logging data;
- Holes with missing survey data; and
- Data is suspect.

12.2.2 Quality Assurance/Quality Control Verification

Victoria Gold's QA/QC procedure included standards, duplicates and blanks to check the accuracy and precision of assay data. The authors evaluated the QA/QC data from 2009 to the 2016 drilling program. QC samples used in the 1990's historic drilling, and prior to 2009, have been summarized in earlier reports. Commercially supplied standard from Ore Research and Exploration Pty (ORE) were used for quality control, and frequencies of standards were approximately 3% of the total samples submitted. Field duplicates, prep duplicates and blanks were also inserted into the sample stream. Although the sample preparation was done by a certified laboratory, SRK still recommended increasing the frequency of blanks to 2%, considering high and low grade mix sample population, to identify any sample contamination. Table 12.1 summarizes the frequency of the control samples.

Table 12.1: Frequency of Quality Control Samples

QC Samples	No. of Samples	% of Total Samples	Total Samples (Excluding QC Samples)
Standards (Au & Ag)	580	2.70	21,447
Blanks	34	0.16	21,447
Field Duplicates	407	1.90	21,447
Prep Duplicates	406	1.89	21,447

Source:

As a standard quality assurance protocol, if assay results were received of standards and/or blanks not within the QC limit, the laboratory was immediately asked to re-assay a particular batch including QC samples. If the re-assay passed the QA/QC criteria, the results of the second batch were used in the resource estimation.

Minimal issues with acceptable limits were identified in the QA and the authors concluded the assay data is acceptable to be used for the resource estimation. The authors have relied on earlier QP reports done on the QA/QC of historic drilling prior to 2010.

12.2.3 Bulk Density Verification

Victoria Gold compiled bulk density data from core in-house measurement, and a total of 1,227 bulk density determinations were reviewed. Of those, a total of 17 determinations were discarded as being either too high or too low for the respective rock type. The data were reviewed in detail in comparison with outside labs SGS and ALS Chemex, and to verify representative locations of data within the mineralized zones

Victoria Gold used a method of weighing the core pieces in air and in water, without the use of paraffin wax for coating the core in order to seal off porosity. In 2012, Victoria Gold sent the same 1,227 core samples to ALS Chemex for outside laboratory density determinations, and ALS used a paraffin wax coating process. As a QA/QC check, Victoria Gold also sent approximately 300 samples to SGS labs, who used the same process as ALS in their density determination method. The Eagle mineralized shape bulk density data was deemed to be sufficiently distributed throughout the deposit to be representative.

It was verified that all data, including in-house, SGS, and ALS, were in close agreement for the approximately 1,210 original data and 300 additional SGS determinations. The bulk density data comparison is shown in Table 12.2.

Table 12.2: Bulk Density Data Used for the Resource Estimation – by Rock Type

SRK Type	FS Type	Classification	In-House	SGS	ALS	Mean Value
		ALL DATA (No outliers)	2.66	2.65	2.65	2.65
1	A	Oxidized Granodiorite	2.62	2.62	2.61	2.62
3*	B	Fresh Granodiorite (unaltered)	2.66	2.65	2.65	2.65
2	C	Altered Granodiorite	2.65	2.62	2.63	2.63
4	E	Oxidized Metasedimentary Rock	2.62	2.59	2.61	2.61
6		Fresh Metasedimentary Rock	2.68	2.72	2.66	2.69

Note: * this is the correct type code - they were originally numbered from surface downward: Ox, Alt, Fresh, as 1,2,3
Source: AVMC (2016)

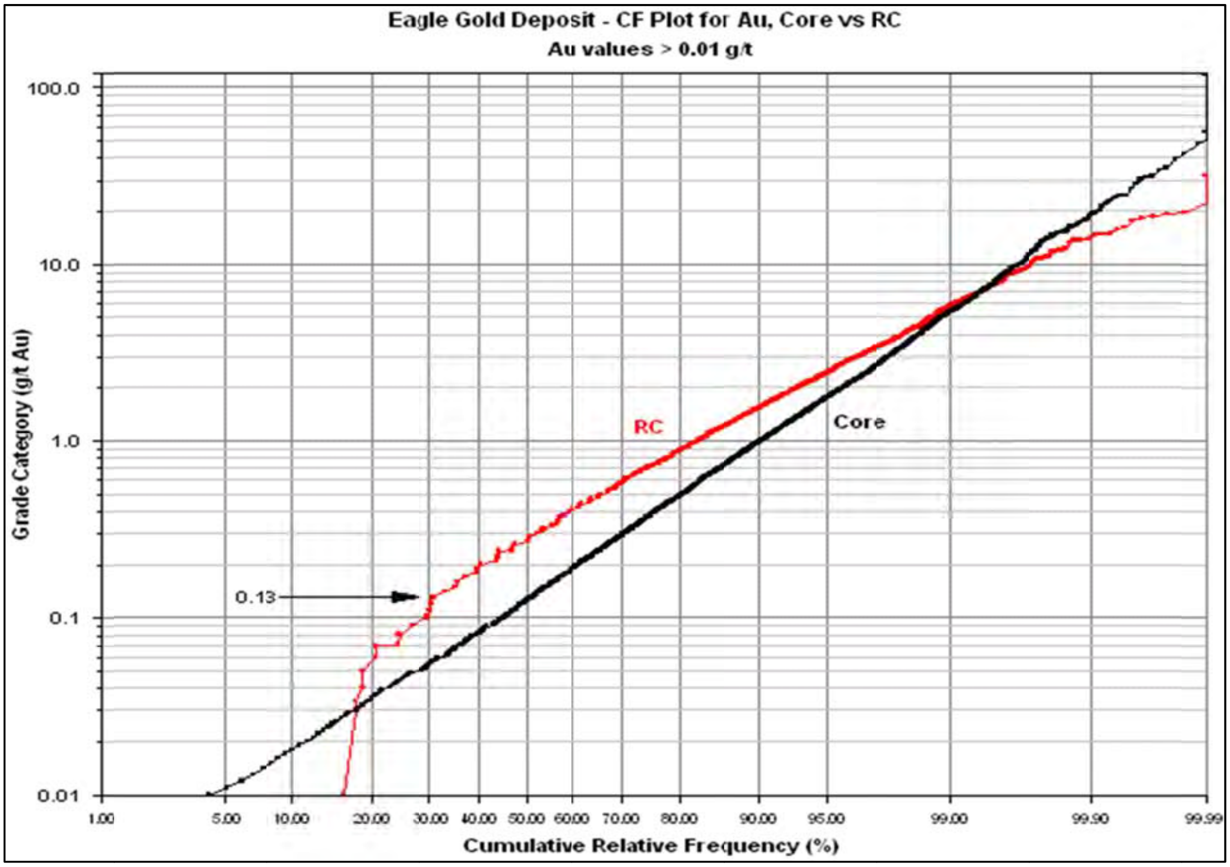
An assumed bulk density of 2.00 was used for overburden. Bulk density data by rock type was assigned to resource blocks by the nearest neighbour assignment, using the mean value for each rock type.

It was noted that the published average bulk density value for Granodiorite is 2.67 to 2.79 (Carmichael, 1980); and therefore the value for Eagle at 2.65 is reasonable.

12.2.4 RC Versus Core Assays Verification

An extensive examination was conducted on core versus RC drilling gold assays for possible bias in RC samples. A cumulative frequency (CF) plot of 2011 data for RC versus core is shown in Figure 12.2, which illustrates an apparent global difference, with RC assays biased high relative to core, for the entire range of assays.

Figure 12.1: CF plot of RC versus Core Au Assays – All Data (to August 2011)



Source: AVMC (2016)

To investigate further, a quartile-quartile (QQ Plot) comparison was done between each quintile of RC and core assays.

The majority of the differences between core and RC were observed below a 0.2 g/t cut-off. In other words lower grade assays. One reason for the differences was location bias for the RC holes. It was also noted that for lower grade assays close to the detection limit, the difference was more obvious in historical RC holes, which had an apparent higher analytical detection limit than that of the recent core assays. This apparent RC location bias is explained by the fact that core drilling was conducted in a more widespread manner rather than concentrated in the central higher grade portion of the deposit where the RC drilling was focused.

The authors also examined core and RC comparisons to check for bias due to orientation of drilling. All drilling was directed as angle holes either north or south to cross at various dip angels, intended to cross the primary orientation of mineralization at approximately 90°. Comparisons showed good correlation and therefore there was no bias of RC-versus-RC on north-versus-south directed drilling. A similar good comparison and no bias of core-versus-core on north-versus-south directed drilling was shown.

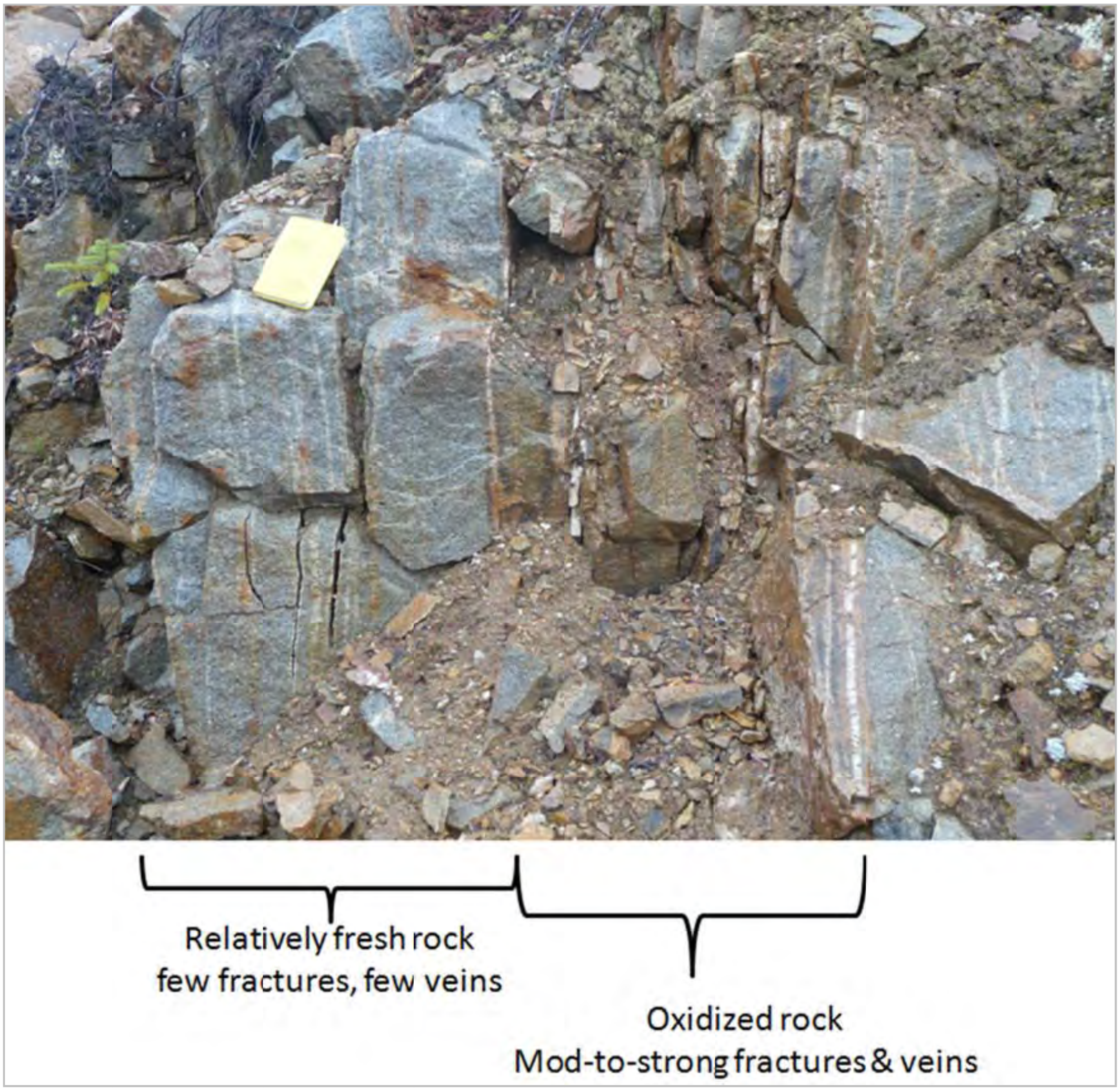
It was concluded that there was no material bias in the sampling method comparing core to RC, or orientation of drilling, above a cut-off of 0.15 to 0.20 g/t Au. The effect was minimal on the resource estimation, as the mineralized shell created for Eagle Gold was based on a modified 0.20 g/t Au grade shell.

12.2.5 Oxidation Surface Verification

For both the Eagle Zone and the Olive Zone, oxidation codes were present in the core logging and the drill hole database, and were used to determine metallurgical types with appropriately assigned recoveries.

A review of the oxide codes for Eagle Gold was done for the purpose of the 2012 FS and was updated for the current resource estimate. Observations at site of the core and the surface outcrops suggested that oxide codes in the drill hole logs for Eagle Gold may have been defined based on the relative amount of oxidation noted in the host rocks, not necessarily the amount of oxidation present in gold-bearing veins and fractures. This is of particular concern for the Eagle Zone, as RC drilling comprised a significant portion of the drilling, and oxidation in veins, as opposed to host rock may not be as discernable in RC cuttings. If the vein density is low, yet the rock is still a mineable grade, the core or RC cuttings may appear as a relatively low oxide code, when the veins are indeed well oxidized. Figure 12.2 illustrates the issue.

Figure 12.2: Photograph of Outcrop - Oxidized Sheeted Quartz Veins in Relatively Fresh Granodiorite



Source: SRK (2011)

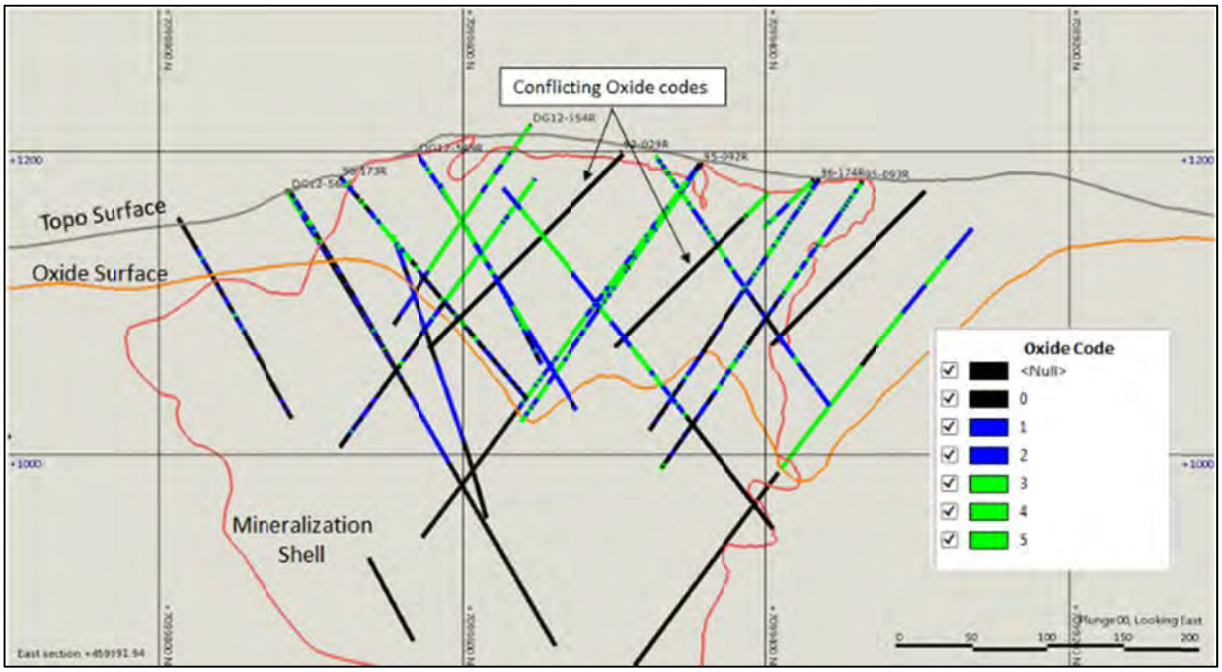
A horizontal drill hole through the outcrop shown in Figure 12.2 would be classified with a mix of oxide codes, yet all the veins and fractures containing quartz and sericite would be classified as oxidized, which is where the gold is located. At a proposed approximate 6 mm crush size, intended for the HLP, the oxidized veinlets containing gold will be exposed to leach fluids even if hosted in largely unaltered and un-oxidized Granodiorite.

As the oxidation state of veins and veinlets is important, and the oxidation state of Granodiorite (without veins and veinlets) is generally not, the authors took the approach of determining the base of oxidation as the base of the preponderance of oxide code 3 or greater.

A trace of the interpreted base of oxidation was digitized in a cross-section, linking section to section to create a surface, which was then modified in a long section, and to individual holes, to create the surface used in the model.

This process was updated in 2013 after the last drilling program at the Eagle Zone. Figure 12.3 indicates the drill hole codes for oxidation of three or greater (green) and the interpreted oxidation surface (orange) in cross-section for Eagle Gold.

Figure 12.3: Eagle Cross-Section 459992E (50m thick) Showing Oxidation Codes and Interpreted Oxidation Surface (Orange)



Source: SRK Leapfrog software, 2013

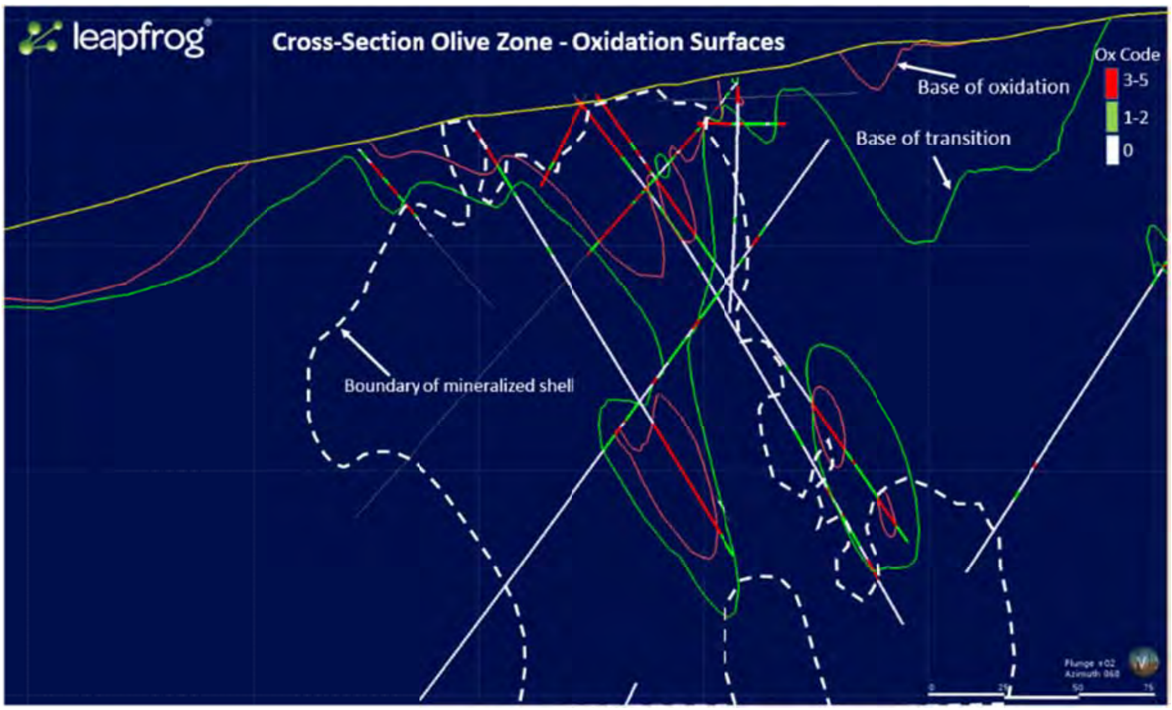
A general interpretative base of oxidation surface was generated for use in resource modelling. For the area above the interpreted oxide surface, some historical holes have no oxide code and/or low oxide codes (codes Null, 0, and 1), and are surrounded by holes with oxide code 3 or greater. These conflicting oxide codes are shown in Figure 12.4. Most of the conflicting low oxide codes are from older RC holes. These low oxide codes (and no oxide codes) are considered non-representative of the oxidation state of the veins.

The base of oxidation roughly mimicked topography, and generally dipped steeply downward to where the mineralization was of higher grade, and corresponding to the structural fracture/vein control of mineralization and oxidation.

An oxide surface was created for the 2012 FS resource model. For this 2016 updated resource estimate, the same process was used and incorporated into the 2012 in-fill drilling program data, to update the oxidation surface. The resulting modified oxide surface changed minimally from the previous 2012 surface.

For the Olive Zone, almost all of the drilling consisted of mostly recent and consistent core drilling. Leapfrog software was used to create a surface for the base of oxidation (Ox codes 3, 4, and 5), and a base of transition or mixed oxide-sulphide material (Ox codes 1 and 2). Below the transition zone, Ox code 0 defines un-oxidized fresh Granodiorite or sulphide-bearing Granodiorite (Figure 12.4). A transition or mixed oxide-sulphide zone was not defined for Eagle, but is more relevant for Olive where sulphide mineralization is more abundant, and where metallurgical recovery data is available for the transition or mixed material. At the Olive Zone, oxidation is relatively shallow and mixed oxides-sulphides are present at surface in trenches. Similar to Eagle, the base of oxidation and the base of transition or mixed material, tend to dip down vertically along some structural zones.

Figure 12.4: Olive Zone Cross-Section facing N68E - Oxide surfaces



Source: AVMC (2016)

12.2.6 Summary

Olive Zone data verifications were similar in process to the work done for Eagle Gold and included:

- Site verifications of rock types;
- Alteration, oxidation, mineralization in trenches and drill hole core;
- Spot check comparisons with assay data;
- Review of Victoria Gold's bulk density determinations; and
- QA/QC procedures and results for which the QA/QC procedures were in place.

The authors concluded that the databases for both the Eagle Zone and the Olive Zone were valid for use in the resource estimation, and were based on proper drilling, assaying, QA/QC procedures, and database construction. There were no identified data limitations or errors that would have bearing on the quality of the Mineral Resource estimations.

13 Mineral Processing and Metallurgical Testing

Metallurgical test work for the Eagle Zone has been conducted between 1995 and 2014. The test work to date is divided into two categories: Pre- and post-2012 work. The pre-2012 work was used as the basis of the 2012 Wardrop FS. Since the completion of the FS additional variability and confirmation test work has been performed by Kappes Cassidy & Associates (KCA) and McClelland Laboratories for the Eagle Gold project. The focus on this test work was to confirm feasibility test work results, provide additional test work results for Ore Type “E”, and to evaluate recoveries at ambient and freezing temperatures.

In addition to the post-FS work completed, a study on projected gold recoveries from run-of-mine (ROM) and primary crushed only ore was performed for Eagle Zone. Preliminary test work has also been performed on the nearby Olive Zone between 2014 and 2015.

13.1 Eagle Gold Project Metallurgical Test Programs Chronology

13.1.1 Pre-2012 Metallurgical Test Work

Extensive metallurgical test work has been conducted on material for the Eagle Gold project using sample sets deemed representative at the time with respect to the aim of the particular program embarked upon. The Wardrop FS on the Eagle Gold project included the evaluation of test results from test programs carried out by KCA between 1995 and 2011 which included:

- Bottle roll and column leach tests; High-pressure grinding roll (HPGR) and conventional crushing on master composites and on individual ore type composites conducted from 1995-1997; and
- Bottle roll and column leach tests; HPGR and conventional crushing on master composites conducted from 2009 and 2011;

In total, 36 column leach tests (including three for environmental testing) and 37 bottle roll tests were completed at varying crush sizes on material from 11 core holes and three surface pits and composites prepared from 17 t of core and 2.2 t of surface samples. The test work focused primarily on ore types “A” (Weathered Granodiorite, 39.1% of total contained gold), “B” (Fresh to Weakly Altered Granodiorite, 42.5% of total contained gold), and “C” (Seretic, Chloritic, Carbonate Altered Granodiorite, 11% of total contained gold). For the purposes of the study, ore types “D” (Fine-grained Granodiorite – assumed to be minimal) and “E” (Weathered Sediments, 7.4% of total contained gold) were considered as ore type “B” when calculating the overall recoveries. General results from these tests included:

- Gold recovery was dependent on crush size and ore type;
- Gold recoveries in the column leach tests varied from 43 to over 90% and recoveries in the bottle roll tests varied from 23 to 97%;
- Calculated gold recoveries for different material types at a conventional crush size of 80% passing 6.3 mm were:

  - o Type A: 79% Recovery Au;
  - o Type B: 68% Recovery Au;
  - o Type C: 73% Recovery Au; and
  - o Overall: 73% Recovery Au.
- Moderate leach time requirements and relatively low reagent requirements were:
  - o 150-day leach cycle;
  - o Lime addition at 1 kg/t; and
  - o Cyanide consumption at 0.39 kg/t.
- Compacted permeability tests with simulated loads between 0 and 150 m demonstrated good stability and low slump without cement agglomeration.
- Agglomeration with cement was recommended in the lower lifts:
  - o Cement addition at 2 kg/t.

Based on extensive KCA experience, the calculated gold recoveries presented above included a 3% deduction for estimated field recoveries from those achieved in the laboratory test program. This is common practice for scaling column tests to expected field results for this type of deposit.

13.1.2 KCA 2012 – Small Column Variability Testing

Material from KCA's 2009 and 2011 test work programs for the Eagle Gold project was used to conduct variability testing including bottle roll tests, compacted permeability tests, and small column leach tests. The samples included eight super sacks of material from the 2009 program 49 5-gallon (19 L) buckets and two super sacks of half split HQ and PQ core material from the 2011 program, which represented 26 separate samples. Composites were made for each ore type (A, B, C and E) which were then used to make one master composite.

Details on the master composite are presented in Tables 13.1.

Table 13.1: Master Composite of A, B, C, E Core

KCA Sample No.	Ore Type	% to Composite	Weight to Composite, (kg)
48126	A	17	27.2
48164	A	17	27.2
48165	B	46	73.28
48166	C	12	19.2
48167	E	8	12.8
		100	160

Source: KCA (2016)

Head analyses for gold and silver, carbon and sulphur, mercury and copper, multi-element analysis, and a whole rock analysis for were performed on the master composite sample. The sample had a calculated size of 80% passing 6.81 mm with an average weighted head assay of 1.235 g/t for gold and 0.39 g/t for silver. Mercury and copper levels were low and are not expected to be an issue.

13.1.2.1 Bottle Roll Leach Test Work

Bottle roll leach tests were performed on selected portions of individual samples as well as on the master composite. Bottle roll tests were conducted on head sample material pulverized to 80% passing 0.075 mm.

A summary of the results is presented in Table 13.2.

Table 13.2: Bottle Roll Leach Test Summary Results

KCA Sample No.	KCA Test No.	Ore Type	Domain	Calculated Head (g Au/t)	Extracted (g Au/t)	Avg. Tails (g Au/t)	Au Extracted (%)	Leach Time (hours)	Cons. NaCN (kg/t)	Addition Ca(OH)2 (kg/t)
48127	63507 A	A	north	0.333	0.32	0.014	96	48	0.32	1
48130	63507 B	A	north	1.181	1.126	0.055	95	48	0.21	1
48131	63507 C	A	north	0.639	0.593	0.046	93	48	0.13	1
48132	63507 D	A	south	2.232	2.166	0.067	97	48	0.1	0.5
48133	63507 E	B	north	0.719	0.668	0.051	93	48	0.32	1
48134	63508 A	B	north	0.776	0.74	0.036	95	48	0.16	0.5
48135	63508 B	B	north	0.89	0.795	0.094	89	48	0.33	0.5
48136	63508 C	B	north	0.998	0.904	0.094	91	48	0.39	0.5
48137	63508 D	B	north	1.288	1.106	0.182	86	48	0.48	1
48138	63508 E	B	south	0.384	0.364	0.021	95	48	0.29	1
48139	63509 A	B	north	0.908	0.852	0.057	94	48	0.4	0.5
48141	63509 B	B	south	0.633	0.599	0.034	95	48	0.1	0.5
48142	63509 C	B	south	1.08	1.014	0.066	94	48	0.23	1.5
48143	63509 D	B	north	0.269	0.237	0.033	88	48	0.21	1
48144	63509 E	C	north	2.279	2.192	0.087	96	48	0.33	1
48147	63510 A	C	south	0.492	0.46	0.033	93	48	0.17	1
48148	63510 B	C	south	0.373	0.316	0.057	85	48	0.14	1
48149	63510 C	C	south	0.795	0.696	0.099	88	48	0.23	1
48150	63510 D	E	south	1.18	1.143	0.038	97	48	0.23	1
48151	63510 E	E	north	3.197	2.879	0.319	90	48	0.27	1.5
48152	63511 A	E	north	0.78	0.655	0.125	84	48	0.29	1
48159	63511 C	C	--	0.643	0.609	0.034	95	48	0.32	1
48126	63511 D	A	--	1.101	1.054	0.047	96	48	0.09	1.5
48163	48171 A	A, B, C, E	--	1.018	0.973	0.045	96	96	0.34	1.5

Source: KCA (2016)

The bottle roll leach tests showed high recoveries for gold ranging between 85% and 97% with low reagent requirements. Based on these results, there are no significant correlations observed between sample head grades and overall gold recoveries.

A summary of the results by ore and domain type is presented in Table 13.3.

Table 13.3: Bottle Roll Leach Test Summary by Ore Type – Gold

Sample ID.	Ore Type	Domain	Calculated Head (g Au/t)	Au Extracted, (%)	Leach Time (hours)	Consumption NaCN (kg/t)	Addition Ca(OH)2 (kg/t)
Average	A	All	1.096	95	48	0.19	0.88
Average	A	North	0.718	95	48	0.22	1
Average	A	South	2.232	97	48	0.1	0.5
Average	B	All	0.795	92	48	0.29	0.89
Average	B	North	0.836	91	48	0.33	0.93
Average	B	South	0.699	94	48	0.21	0.78
Average	C	All	0.985	90	48	0.22	1
Average	C	North	2.279	96	48	0.33	1
Average	C	South	0.554	89	48	0.18	1
Average	E	All	1.719	90	48	0.26	1.17
Average	E	North	1.989	87	48	0.28	1.25
Average	E	South	1.18	97	48	0.23	1

Source: KCA (2016)

13.1.2.2 Compaction Test Work

Compacted permeability test work was conducted on portions of Type A composite and the master composite. Each composite was crushed to 100% passing 9.5 mm and assessed at a simulated 150 m overall heap height. The Type A composites were each agglomerated with 2.5 kg cement per tonne of ore, and the master composite was agglomerated with 3 kg cement per tonne of ore.

Results from the compaction testing are presented in Table 13.4.

Table 13.4: Summary of Compacted Permeability Tests

KCA Sample No.	KCA Test No.	Description	Cement Added, kg/t	Effective Height, metres	Flow Rate, $L/h/m^2$	Crush Size, Mm	% Pellet Breakdown	% Slump	Pass/Fail
48126	48153 A	Pit 1, 2, 3 Type A	2.5	150	128	9.5	<5	0	Pass
48126	48153 B	Pit 1, 2, 3 Type A	2.5	150	106	9.5	<5	0	Pass
48163	48172 A	A, B, C, E Composite	3	150	296	9.5	<5	1	Pass

Source: KCA (2016)

Results of the agglomeration tests were evaluated based on percent slump, out flow of solution, and solution color. In general, KCA considers the following test criteria as the basis for measuring the response of the various samples:

- Less than 10% slump;
- Measured flows of more than 10 times the heap design flow rate;
- Less than 15% pellet break down; and
- Excessive color and lack of clarity of solution.

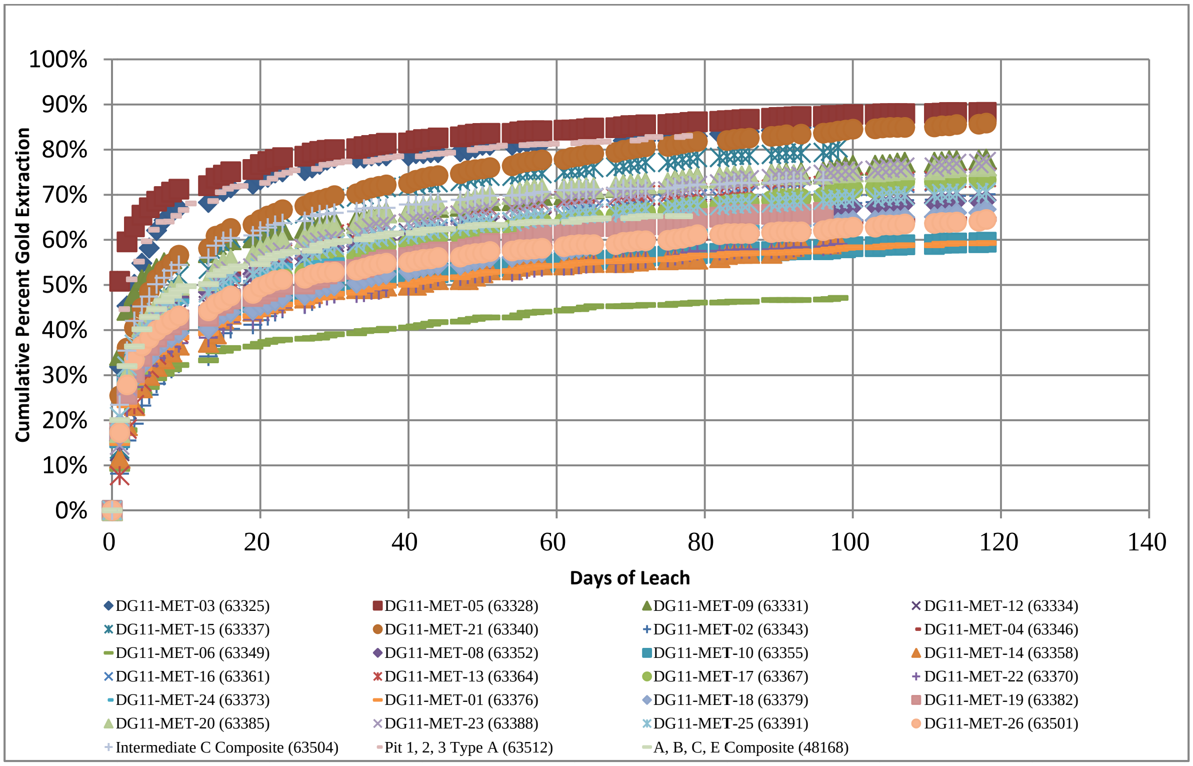
Based on KCA's criteria, all three tests passed with minimal breakdown of agglomerates and very little slump, however, it should be noted that one of the Type A composite tests marginally passed for flow rate based on the accepted criteria of 10 times the field application rate of 10 $l/h/m^2$. The master composite comfortably passed based on all criteria.

13.1.2.3 Column Leach Test Work

Small column leach tests were conducted utilizing each individual sample and composites at a crush size of 100% passing 9.5 mm (approximately 80% passing 6.3 mm). All of the columns were agglomerated with approximately 3 kg cement per tonne of ore and were leached for varying periods of time using a sodium cyanide solution with an initial strength of 1.0 grams sodium cyanide per litre of solution. At the conclusion of leaching, drain down tests were performed for the columns.

Gold leach curves for all of the columns are shown in Figure 13.1. The results of the column leach tests are presented in Table 13.5.

Figure 13.1: Small Column Leach Curves



Source: KCA (2016)

Table 13.5: Column Leach Test Results by Ore Type at 100% Passing 9.5 mm - Gold

KCA Sample No.	KCA Test No.	Ore Type	Domain	Calculated Tail p_{80} Size (mm)	Calculated Head (g Au/t)	Extracted, (g Au/t)	Avg. Tails (g Au/t)	Extracted (% Au)	Days of Leach	Consumption NaCN (kg/t)	Addition Cement (kg/t)
48127	63325	A	North	5.92	0.426	0.373	0.053	88	98	1.12	3
48128	63328	A	North	6.01	1.452	1.221	0.231	84	118	1.06	3.04
48129	63331	A	North	6.37	0.769	0.609	0.16	79	118	1.12	3.06
48130	63334	A	North	5.92	0.697	0.483	0.214	69	96	1.27	3.04
48131	63337	A	North	5.92	0.721	0.586	0.135	81	98	1.49	3.04
48132	63340	A	South	6.33	2.002	1.727	0.275	86	118	1.22	3.05
48133	63343	B	North	6	1.155	0.824	0.331	71	118	1.93	3.04
48134	63346	B	North	6.35	0.473	0.281	0.192	59	118	1.23	2.99
48135	63349	B	North	6.23	0.76	0.389	0.371	51	98	0.96	2.99
48136	63352	B	North	6.14	1.118	0.793	0.325	71	118	1.21	3
48137	63355	B	North	5.71	1.712	1.054	0.658	62	118	0.82	3
48138	63358	B	South	5.8	0.343	0.204	0.139	60	98	1.05	3.13
48139	63361	B	North	5.92	0.924	0.695	0.229	75	118	1.18	3.01
48140	63364	B	North	6.82	0.907	0.671	0.236	74	118	1.59	3.02
48141	63367	B	South	6.13	0.857	0.646	0.211	75	118	0.93	3.01
48142	63370	B	South	6.24	0.859	0.522	0.337	61	98	1.25	3.13
48143	63373	B	North	6.17	0.33	0.227	0.103	69	98	1.34	3.12
48144	63376	C	North	6.58	1.758	1.063	0.695	60	118	1.39	3.02
48147	63379	C	South	6.38	0.788	0.541	0.247	69	118	1.3	3.05
48148	63382	C	South	5.85	0.461	0.318	0.143	69	96	0.97	3.13
48149	63385	C	South	5.86	0.935	0.737	0.198	79	118	1.31	3.08
48150	63388	E	South	6.8	1.143	0.926	0.217	81	118	1.07	2.99
48151	63391	E	North	6.43	3.481	2.512	0.969	72	118	1.3	3
48152	63501	E	North	6.18	0.864	0.572	0.292	66	118	0.99	2.99
48159	63504	C	North	5.58	0.554	0.417	0.137	75	96	0.89	3.01
48126	63512	A	---	5.75	1.017	0.861	0.156	85	77	0.91	3
48163	48168	A, B, C, E	---	5.98	1.029	0.672	0.357	65	77	0.65	3.03

Source: KCA (2016)

The variability small column tests had the following results:

- Ore Type "A" gold extractions ranging from 69 to 88% with an average gold extraction of 81%;
- Ore Type "B" gold extractions ranging from 51 to 79% with an average gold extraction of 66%;
- Ore Type "C" gold extractions ranging from 60 to 79% with an average gold extraction of 70%;
- Ore Type "E" gold extractions ranging from 66 to 81% with an average gold extraction of 73%;
- Surface Type "A" composite gold extraction of 85%;
- Gold extraction for the master composite of 65%;
- Overall average cyanide consumption of 0.42 kg NaCN/t ore based on a 35% factor for calculating field consumptions from laboratory data; and
- Average 24-hour drain down of 41, 33.5 and 38.5 L/t dry ore for ore types A, B and C, respectively.

13.1.2.4 Variability Test Results

Overall, the results from the variability small column test work are in agreement with the results reported in the Wardrop FS. Results of the column leach tests by ore type and domain are presented in Table 13.6.

Table 13.6: Column Leach Test Results by Ore Type

Ore Type	Domain	Calculated Head, (g Au/t)	Extracted, (g Au/t)	Avg. Met. Screen Tails (g Au/t)	Extracted (% Au)	Days of Leach	Consumption NaC, (kg/t)	Addition Cement (kg/t)
A	All	1.012	0.833	0.179	81	108	1.21	3.04
A	North	0.814	0.654	0.16	80	106	1.21	3.04
A	South	2.002	1.727	0.275	86	118	1.22	3.05
B	All	0.859	0.573	0.286	66	111	1.23	3.04
B	North	0.925	0.617	0.308	67	113	1.28	3.02
B	South	0.686	0.457	0.229	65	105	1.08	3.09
C	All	0.901	0.615	0.286	70	109	1.17	3.06
C	North	1.156	0.74	0.416	68	107	1.14	3.02
C	South	0.731	0.532	0.199	72	111	1.19	3.09
E	All	1.827	1.337	0.49	73	118	1.12	2.99
E	North	2.312	1.719	0.593	77	118	1.19	3
E	South	0.857	0.572	0.285	67	118	0.99	2.99

Source: KCA (2016)

Based on the distribution of contained gold, the overall extractions are as follows:

- North Zone: 72.2%;
- South Zone: 76.1%; and
- All: 73.7%.

A lab field deduction was not applied to the final recoveries due to continued leaching in a majority of the tests when ended.

Cyanide consumptions for the variability column test work averaged 0.42 kg/t based on ore types tested and includes a 35% factor to the lab results for field consumptions. This is slightly higher than those tested in the Wardrop FS results (0.39 kg/t); however, the consumptions are in close agreement.

Compacted permeability tests show that the ore meets KCA's test criteria up to a heap height of 150 m at a cement addition of 3 kg/t.

13.1.3 KCA 2013 – Environmental Test Work

Reject material from each interval sample from samples from KCA's previous test work (three super sacks of bulk pit material, 26 core interval samples) was utilized to generate four ore type composites based on information provided by Victoria Gold. Each ore type composite was then combined and blended with a portion of reject material from the previously generated Type A composite and a portion of reject material from one interval sample (DG-MET-23) to generate the environmental composite, which was utilized for the environmental test program.

A summary of the environmental composite composition is presented in Table 13.7.

Table 13.7: Summary of Environmental Composite Generation

KCA Sample/Composite No.	Description	Ore Type	Type of Material	Weight to Composite (kg)	Weight to Composite (%)
48126	Pit 1, 2 and 3	A	Surface Pit	59.5	17
48154	DG11-MET-03, 05, 09, 12, 15, 21	A	Core	59.5	17
48155	DG11-MET-02, 04, 06, 08, 10, 14, 16, 13, 17, 22, 24	B	Core	159.5	45
48156	DG11-MET-01, 18, 19, 20	C	Core	44	13
48150	DG-MET-23	E	Core	5.25	1
48157	DG11-MET-25, 26	E	Core	23.45	7
Total:				**351.2**	**100**

Source: KCA (2016)

The environmental composite was prepared and utilized for head analyses, bottle roll leach test work and column leach test work. Detoxification test work was conducted on tailings from each column leach test. Tailings from select columns were further utilized for humidity cell test work and final tailings from select tests were utilized for compacted permeability test work.

13.1.3.1 Bottle Roll Leach Test Work

Bottle roll leach testing was conducted on a portion of the environmental composite. A portion of head material was pulverized to a target size of 80% passing 0.075 mm and leached for 48 hours. The test was maintained at a target concentration of 1.0 grams sodium cyanide per litre of solution.

The bottle roll leach test showed gold extractions of 92% after 48 hours with low sodium cyanide consumptions of 0.29 kg/t.

13.1.3.2 Column Test Work

Column leach tests including cyanide leaching, detoxification and humidity cell tests were conducted on individual portions of material split from the environmental composite. All of the tests were conducted in consecutive order within the original column leach test apparatus.

A total of eight column leach tests were completed with detoxification tests conducted on each column. Detoxified tailings from select columns were then further utilized for humidity cell test work.

Results from the column leach tests for gold extraction are summarized in Table 13.8.

Table 13.8: Summary of Column Leach Tests

KCA Sample No.	KCA Test No.	Column Temp., °C	Calculated Head, g Au/t	Extracted, g Au/t	Weighted Avg. Tail Screen, g Au/t	Extracted, % Au	Calculated Tail p80 Size, mm	Days of Leach	Consumption NaCN, kg/t	Addition Cement, kg/t
48158	63301	22	0.987	0.822	0.165	83	6.1	143	1.24	3.03
48158	63304	22	0.98	0.825	0.156	84	5.7	143	1.22	3.03
48158	63307	22	0.972	0.806	0.166	83	6.4	143	1.27	3.03
48158	63310	22	1.093	0.929	0.164	85	6	143	1.06	3.03
48158	63313	22	0.992	0.82	0.172	83	6.4	143	1.4	3.03
48158	63316	22	0.991	0.826	0.165	83	6.1	143	1.28	3.03
48158	63319	3	0.953	0.785	0.167	82	6	143	1.27	3.02
48158	63322	3	1.074	0.909	0.165	85	6.1	143	1.48	3.03

Source: KCA (2016)

All of the columns were leached for 143 days with two columns being leached at a target temperature of 3°C and the remaining six columns being leached at ambient temperature (~22°C). Gold extractions from the columns ranged between 82% and 85%. The overall average recovery of the columns was 83.5% with an average cyanide consumption of 1.28 kg/t ore. The recoveries for each column were very consistent and temperature had a negligible effect on overall recoveries.

Two types of detoxification test work were conducted on tailings from the column leach tests including bacterial and hydrogen peroxide detoxification. Bacterial detoxification was conducted in two phases including solution detoxification (Phase I) followed by continuous solution cycling (Phase II). In all but one of the tests, the bacterial detoxification reduced the Weak-Acid-Dissociable Cyanide (WAD CN) to less than 3 mg/L after 17 days and less than 1 mg/L after 45 days. After 180 days, the WAD CN for all tests ranged between 0.14 and 0.24 mg/L. The detailed bacterial detoxification test results can be found in the KCA report “Eagle Gold Metallurgical Test Work December 2013.”

Hydrogen peroxide detoxification work was conducted on the tailings material using a copper nitrate catalyst ($CuN_2O_6.5H_2O$). The copper catalyzed peroxide was added to detoxify the final barren solution and the columns were then restarted with the detoxification solution. Reagent additions to the detoxification solution were based upon the total cyanide and copper analyses conducted on the final barren solution at the end of the leach period. Daily hydrogen peroxide additions to the detoxification solution were calculated based upon the initial WAD cyanide analysis. After 20 days, the WAD CN was reduced to less than 1 mg/L in all but one of the columns. After 80 days the WAD CN for all columns ranged between 0.23 to 0.53 mg/L. The complete peroxide detoxification tests can be found in the KCA report “Eagle Gold Metallurgical Test Work December 2013.”

Humidity cell test work was conducted on the detoxified tailings material from select column leach tests and was performed in the original leach test column. The complete humidity cell results can be found in the KCA report “Eagle Gold Metallurgical Test Work December 2013.”

13.1.3.3 Compacted Permeability Test Work on Column Tailings

Compacted permeability tests were performed on the column tails material, which was agglomerated with approximately 3 kg cement per tonne of ore prior to leaching. Compacted permeability tests were conducted at equivalent heap height loadings of 25 m, 50 m, 75 m, 100 m and 125 m. Based on KCA’s test criteria, the material passed at all heights.

13.1.4 McClelland 2014 – Master Composites Test Work

McClelland Laboratories conducted test work in 2014 to evaluate the effects of heap leaching at near freezing temperatures. Tests were conducted at both ambient (~16°C) and near freezing (~1°C), on each of two samples, at a crush size of 80%, passing 6.25 mm (1/4”), including agglomeration testing, bottle roll leach tests and column leach tests. The samples were composed of material from 31 bags of broken drill core received in April 2013.

Head assays for gold are presented in Table 13.9.

Table 13.9: Gold Head Assay Results

Determination Method	Head Grade g Au/t ore	
	Sample 1	Sample 2
Direct Assay, Initial	0.343	0.617
Direct Assay, Duplicate	1.063	0.720
Direct Assay, Triplicate	0.171	0.240
Calculated, Bottle Roll Test	0.583	0.343
Calculated, Head Screen	0.377	0.617
Calculated, Column Leach Test Ambient	0.686	0.480
Calculated, Column Leach Test Cold	0.754	0.411
Average	0.514	0.514
Std. Deviation	0.343	0.206

Source: McClelland 2014

13.1.4.1 Direct Agitated Cyanidation Tests

Direct cyanidation bottle roll tests were conducted on each sample at 80% passing 6.25 mm to evaluate heap leach amenability. Gold recoveries from the bottle roll tests ranged between 60.0% and 70.6% after 96 hours with cyanide consumptions between 0.20 and 0.22 kg/t. The tests show that the material was still leaching at the conclusion of the bottle roll tests, indicating that higher recoveries may be achieved with longer leach time.

13.1.4.2 Agglomeration Tests

Agglomerate strength and stability tests were conducted on each sample at 80% passing 6.25 mm and at ambient temperatures followed by tests conducted on samples at cold temperatures (-10°C, 0°C, and 10°C). The results show that agglomeration with cement alone without lime resulted in poorer quality agglomerates. Optimal cement additions for agglomerate strength were found to be between 1 kg and 2 kg cement per tonne at ambient conditions (with 1.5 kg of lime per tonne). Agglomerates did not perform as well under freezing and near freezing temperatures and further testing may be required.

Load permeability tests were also conducted on the material to determine the permeability of the two ore types at different simulated heap stack heights with agglomerates produced using 2 kg per tonne cement and 1.5 kg per tonne lime at ambient and low temperatures. Permeability testing was conducted at ambient temperatures for all samples.

Results of the permeability test work are summarized in Tables 13.10 and 13.11. The results show that adequate percolation is achieved for all tests up to 97 m (318 ft) heap height both with and without cement agglomeration.

Table 13.10: Load Permeability Test Results – Sample 1

Non-Agglomerated		Agglomerated			
		10°C		-10°C	
Estimated Heap Height Metres	Application Rate g/m/ft^2	Estimated Heap Height Metres	Application Rate g/m/ft^2	Estimated Heap Height, Metres	Application Rate, g/m/ft^2
7.6	0.632	7.9	0.058	7.9	0.847
14.9	0.316	15.9	0.046	15.9	0.132
29.0	0.18	28.0	0.02	29.9	0.029
39.9	0.135	38.1	0.017	41.2	0.017
52.1	0.125	46.0	0.013	53.0	0.014
64.0	0.089	56.1	0.01	64.9	0.011
82.0	0.045	77.1	0.007	75.0	0.009
93.9	0.034	96.0	0.006	100.0	0.008

Source: McClelland 2014

Table 13.11: Load Permeability Test Results – Sample 2

Non-Agglomerated		Agglomerated			
		10°C		-10°C	
Estimated Heap Height Metres (Ft)	Application Rate g/m/ft^2	Estimated Heap Height Metres (Ft)	Application Rate g/m/ft^2	Estimated Heap Height, Metres	Application Rate, g/m/ft^2
7.6	0.632	7.9	0.058	7.9	0.847
14.9	0.316	15.9	0.046	15.9	0.132
29.0	0.18	28.0	0.02	29.9	0.029
39.9	0.135	38.1	0.017	41.2	0.017
52.1	0.125	46.0	0.013	53.0	0.014
64.0	0.089	56.1	0.01	64.9	0.011
82.0	0.045	77.1	0.007	75.0	0.009
93.9	0.034	96.0	0.006	100.0	0.008

Source: McClelland 2014

13.1.4.3 Column Leach Tests

Column leach tests were conducted on each sample at a crush size of 80% passing 6.25 mm, at both ambient and near freezing temperatures to determine the effects of heap leaching under cold weather conditions. Lime was added to each column at 1.5 kg per tonne ore based on the bottle roll lime requirements; column charges were not agglomerated.

Results from the column leach tests are presented in Table 13.12.

Table 13.12: Column Leach Test Results

Sample I.D. Metallurgical Results	Unit	Ambient (P-2)	Cold (P-1)	Ambient (P-4)	Cold (P-3)
Extraction	% of total Au	90	86.4	78.6	83.3
Extracted	oz Au/t ore	0.018	0.019	0.011	0.01
Tail Assay	oz Au/t ore	0.002	0.003	0.003	0.002
Calc'd Head	oz Au/t ore	0.02	0.022	0.014	0.012
Average Head	oz Au/t ore	0.015	0.015	0.015	0.015
NaCN Consumed	lb/t ore	5.36	2	5.27	1.79
Lime Added	lb/t ore	3	3	3	3
Final Solution pH		10.1	9.8	10.2	9.9
pH after Rinse		9.4	9.4	9.2	9.4
Leach/Rinse Cycle, Days		183	183	183	183

Source: KCA (2016)

The column tests show that there were no significant adverse effects to gold recovery at near freezing temperatures, nor did it affect the gold recovery rate. The average gold recovery for Sample 1 is approximately 88% and Sample 2 is 81%.

Cyanide consumption was significantly lower for the near freezing column tests compared to the tests on the same samples at ambient temperature.

13.1.4.4 McClelland Test Program Conclusions

Key conclusions from the McClelland test program are:

- No significant recovery differences between ambient and cold column leach tests;
- At a crush size of 80% passing 6.25 mm, the average gold recoveries for Sample 1 and Sample 2 were 88% and 81%, respectively;
- Higher recoveries were achieved compared to other test work with longer leach time (183 days);
- Load/permeability test results were variable and indicate that commercial heap leaching is possible without agglomeration pretreatment; additional testing should be conducted; and
- Cyanide consumption was significantly lower for cold column tests compared to ambient temperature tests.

13.1.5 Run-of-Mine and Primary Crushed Only Recovery

ROM recoveries have been estimated for Eagle Gold based on test work completed on the project to date. The recovery estimate has been based on the following information:

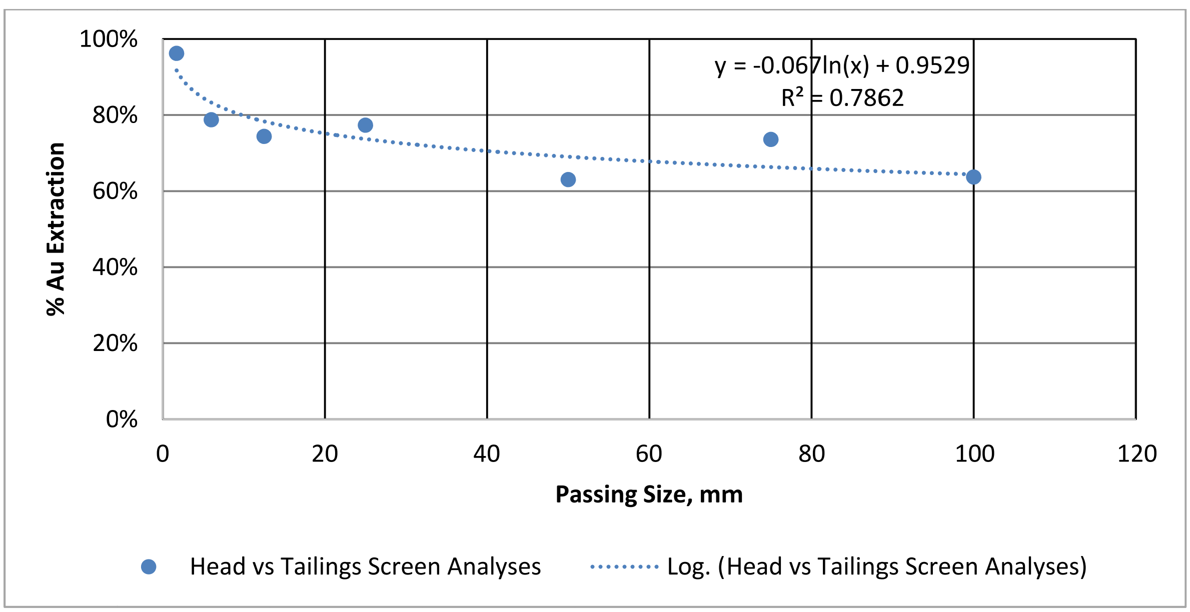
- Estimated ROM size distribution based on data from a nearby project with similar geology as well as simulated data;
- Primary only crushed size distribution based on data from a nearby project with similar geology as well as simulated data; and
- Estimated gold recoveries by size fraction.

13.1.5.1 Run-of-Mine Estimated Recovery

The ROM size distribution was determined using existing ROM data for a nearby project with similar geology. The existing data includes material size distribution at 100% +100 mm and -1,200 mm. Due to the correlation between crush size and recovery, Bruno Process Simulation software was used to determine the distribution of -100 mm material.

Recoveries by size fraction were estimated based on test work data conducted by KCA on conventionally crushed material at 100% passing 75 mm. This recovery data was plotted and a best fit line was used to estimate recoveries at coarser sizes. The plotted recovery by size fraction is presented in Figure 13.2. Overall estimated recoveries by size fraction are presented in Table 13.13.

Figure 13.2: Gold Extractions by Individual Size Fractions



Source: KCA (2016)

Table 13.13: Estimated ROM Recoveries by Size Fraction

KCA Test No. 44289		Calculated % Au by Fraction	ROM % Passing	
Size (mm)	% Au			
1.7	96	7.70	8.00	8.00
6	79	3.90	13.00	5.00
12.5	74	4.50	19.00	6.00
25	77	4.60	25.00	6.00
50	63	6.90	36.00	11.00
75	74	5.90	44.00	8.00
100	64	6.60	54.30	10.30
200	59	12.20	75.10	20.80
300	56	6.40	86.50	11.40
400	54	2.80	91.60	5.10
500	52	1.90	95.30	3.70
610	51	0.70	96.60	1.30
712	50	0.50	97.70	1.10
813	49	0.40	98.60	0.90
914	48	0.10	98.80	0.20
1016	47	0.20	99.30	0.50
1117	47	0.20	99.70	0.40
1200	46	0.10	100.00	0.30
Total		**66**		**100**

Source: KCA (2016)

Based on size fraction, the estimated recovery for the ROM material is calculated at 66%; however, comparing recoveries from this sample at -9.5 mm, the final recovery is approximately 12 percentage points higher than the 2012 FS estimate. The ROM recovery by size fraction was deducted by the 12 percentage points to obtain the final ROM field recovery range. Due to limited data, KCA estimates a ROM recovery in the range of 50 to 55%.

13.1.5.2 Primary Crushed Only Estimated Recovery

The material size distribution for the primary crushed only recovery was estimated based on the estimated ROM size distribution and data from the Bruno Process Simulation software. The same recoveries by size fraction determined for the ROM material were applied to the primary crushed only estimate and are presented in Table 13.14.

Table 13.14: Estimated Primary Crushed Only Recoveries by Size Fraction

Jaw Crushed Product				
KCA Test No. 44289		Calculated % Au by Fraction	Bruno Simulation, % Passing	
Size (mm)	% Au			
1.7	96	7.50	7.80	7.80
12.5	74	9.30	20.40	12.50
25	77	6.70	29.00	8.60
50	63	8.50	42.40	13.40
75	74	8.50	53.90	11.50
100	64	7.00	65.00	11.00
136	62	9.30	80.00	15.00
150	61	3.60	85.90	5.90
200	59	5.60	95.40	9.50
330	55	2.50	100.00	4.60
Total		**68**		**100**

Source: KCA (2016)

Based on size fraction, the estimated recovery of the primary crushed only material is calculated at 68%; however, comparing recoveries from this sample at -9.5 mm, the final recovery is approximately 12 percentage points higher than the 2012 FS estimate. The primary crushed only recovery by size fraction was reduced by the 12 percentage points to obtain the final primary crushed only field recovery range. Due to very limited results, KCA estimates a primary crushed only recovery in the range of 55 to 60%.

13.1.6 Summary of Eagle Gold Test Results

In general, the post-FS test work for the Eagle Gold project is in close agreement with the results from the previous test work. A summary of the key results from each group of test work is presented below.

Summary of 2012 FS Results:

- Master composite column test Au recovery – 74%;
- Individual ore type column leach test Au recovery – 67%;
- Average of master composite and individual ore types Au recovery – 70.5%;
- Overall estimated Au recovery for FS – 73%;
- Cyanide Consumption – 0.39 kg/t;
- Cement addition – 2 kg/t for first few years; and
- Lime addition – 1 kg/t.

Post 2012 FS Results:

- Variability average test Au recovery of individual ore types – 74%;
- Overall average Au recovery from master composites 78%, includes:
    - McClelland master composite average Au recovery – 84.5%;
    - Environmental composite recovery of 83.5%; and
    - Small column master composite recovery of 65%.
- Un-deducted cyanide consumption of 1.21 kg/t for variability testing and 0.95 kg/t to 2.65 kg/t for the McClelland master composite test work.

Based on the above information, KCA recommends the following parameters for the 2016 FS:

- Gold recoveries at:
    - Type “A” – 79%;
    - Type “B” – 68%;
    - Type “C” – 73%;
    - Type “D” (assumed minimal) – no test data available, assumed to be same recovery as Type “B” (68%);
    - Type “E” – 73%; and
    - Overall Au recovery of 73%, including field deduction (assumes no changes to ore type distribution as stated in the 2012 FS).
- Field cyanide consumption of 0.35 kg/t;
- Cement addition of 2 kg/t in lower lifts;
- Lime addition of 1 kg/t; and
- Ultimate leach cycle of 150 days.

Estimated recoveries for ROM material are estimated in the range of 50 to 55% and primary crushed only recoveries are estimated at 55 to 60%. Confirmatory test work is recommended if ROM or primary crushed only scenarios are further considered.

13.2 Olive Metallurgical Test Programs Chronology

Recent metallurgical test work has been carried out by KCA between 2014 and 2015 on material for the Olive Zone. Samples tested include bulk and core material that was deemed representative at the time they were composited with respect to the aim of the particular program embarked upon. Investigations have largely focused on heap leaching.

The following is a chronological list of test work completed to date that is being considered for this section:

- KCA, July 2014, Olive Project Report of Metallurgical Test Work;
- KCA, February 2015, Olive Project Report of Metallurgical Test Work;
- KCA, June 2015, Olive Project Report of Metallurgical Test Work Bottle Roll and Column Leach Test Work Oxide, Sulphide, Transitional and Shamrock Trench.

Test work and results from the programs to date are summarized chronologically below.

13.2.1 KCA – July 2014

The July 2014 test program included preliminary bottle roll leach tests on different sample composites from the Olive ore body to evaluate cyanide leaching of the ore. Samples were composited from buckets of half split and whole PQ and HQ core material, with three oxide sample composites and three sulphide sample composites. Four bottle roll tests were performed on portions of each composite at target sizes of 80% passing 6.3 mm, 80% passing 1.7 mm and 80% passing 0.075 mm. Two bottle roll leach tests were conducted at 80% passing 0.075 mm, one for direct bottle roll test and one for a carbon-in-leach bottle roll test. Each bottle roll test was run for a period of 96 hours.

A summary of the test results is shown in Tables 13.15 and 13.16 for oxide and sulphide, respectively.

Table 13.15: Summary of Bottle Roll Leach Test Results on Oxide Composites after 96 hours

KCA Sample No.	KCA Test No.	Description	Target p80 Size (mm)	Test Type	Head Average (g Au/t)	Calculated Head (g Au/t)	Au Extracted (%)	Consumption NaCN (kg/t)	Addition Ca(OH)2 (kg/t)
71201 A	71254 A	Oxide Composite A	6.3	Direct	1.617	1.683	61	0.41	2.8
71201 B	71256 A	Oxide Composite A	1.7	Direct	1.617	1.514	74	0.55	2.4
71201 B	71258 A	Oxide Composite A	0.075	Direct	1.617	1.766	92	1.33	3.5
71201 B	71260 A	Oxide Composite A	0.075	CIL	1.617	2.299	95	2.16	3
71202 A	71254 B	Oxide Composite B	6.3	Direct	1.93	1.888	52	0.42	2.4
71202 B	71256 B	Oxide Composite B	1.7	Direct	1.93	2.036	75	0.59	2
71202 B	71258 B	Oxide Composite B	0.075	Direct	1.93	1.775	86	0.96	3
71202 B	71260 B	Oxide Composite B	0.075	CIL	1.93	2.094	89	1.93	4
71203 A	71254 C	Oxide Composite C	6.3	Direct	1.917	1.261	71	0.44	2.4
71203 B	71256 C	Oxide Composite C	1.7	Direct	1.917	1.264	81	0.28	2.4
71203 B	71258 C	Oxide Composite C	0.075	Direct	1.917	1.077	89	0.59	4
71203 B	71260 C	Oxide Composite C	0.075	CIL	1.917	1.931	95	1.54	3
Overall Average, g Au/t							**80**	**0.93**	**2.91**
Average Values - p_{80} 6.3 mm							**61**	**0.42**	**2.53**
Average Values - p_{80} 1.70 mm							**77**	**0.47**	**2.27**
Average Values - p_{80} 0.075 mm, Direct							**89**	**0.96**	**3.5**
Average Values - p_{80} 0.075 mm, CIL							**93**	**1.88**	**3.33**

Source: KCA (2016)

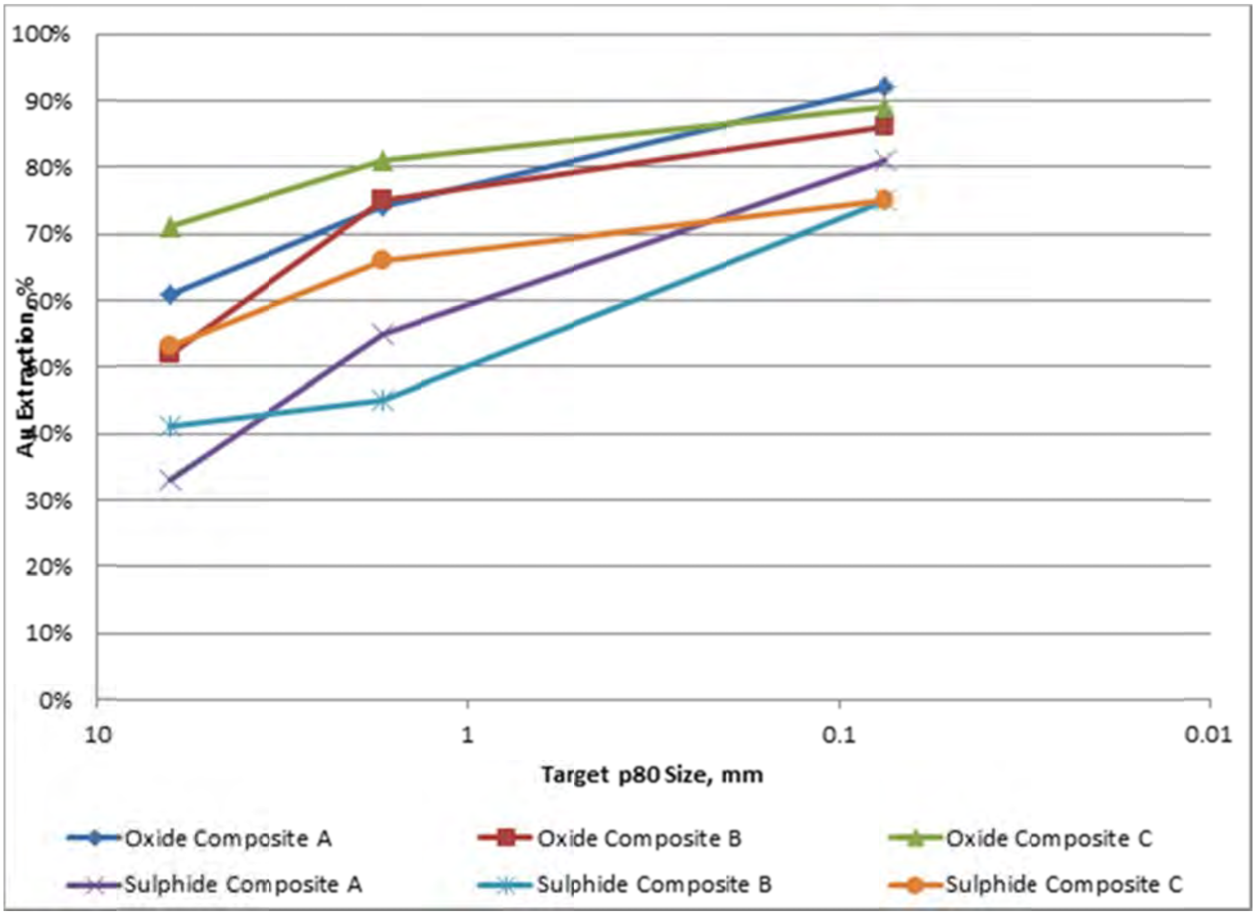
Table 13.16: Summary of Bottle Roll Leach Test Results on Sulphide Composites after 96 Hours

KCA Sample No.	KCA Test No.	Description	Target p_{80} Size (mm)	Test Type	Head Average (g Au/t)	Calculated Head (g Au/t)	Au Extracted (%)	Consumption NaCN (kg/t)	Addition Ca(OH)2 (kg/t)
71204 A	71254 D	Sulphide Composite A	6.3	Direct	7.661	7.358	33	0.31	2
71204 B	71256 D	Sulphide Composite A	1.7	Direct	7.661	7.827	55	0.34	2
71204 B	71258 D	Sulphide Composite A	0.075	Direct	7.661	8.313	81	0.8	2
71204 B	71260 D	Sulphide Composite A	0.075	CIL	7.661	8.434	83	1.6	2
71205 A	71255 A	Sulphide Composite B	6.3	Direct	0.792	1.17	41	0.22	2
71205 B	71257 A	Sulphide e Composite B	1.7	Direct	0.792	1.1	45	0.23	2
71205 B	71259 A	Sulphide Composite B	0.075	Direct	0.792	1.235	75	0.83	2
71205 B	71261 A	Sulphide Composite B	0.075	CIL	0.792	1.775	86	1.86	2
71206 A	71255 B	Sulphide Composite C	6.3	Direct	2.392	2.319	53	0.46	2
71206 B	71257 B	Sulphide Composite C	1.7	Direct	2.392	2.368	66	0.49	2
71206 B	71259 B	Sulphide Composite C	0.075	Direct	2.392	3.02	75	2.16	2
71206 B	71261 B	Sulphide Composite C	0.075	CIL	2.392	2.796	75	2.95	2
Overall Average, g Au/t							**64**	**1.02**	**2**
Average Values - p_{80} 6.3 mm							**42**	**0.33**	**2**
Average Values - p_{80} 1.70 mm							**55**	**0.35**	**2**
Average Values - p_{80} 0.075 mm, Direct							**77**	**1.26**	**2**
Average Values - p_{80} 0.075 mm, CIL							**81**	**2.14**	**2**

Source: KCA (2016)

Gold extraction and crush size showed a strong correlation as shown in Figure 13.3. Slightly higher recoveries were achieved for the CIL bottle roll tests compared to the direct bottle roll tests.

Figure 13.3: Gold Extraction vs. Crush Size



Source: KCA (2016)

13.2.2 KCA – February 2015

A preliminary evaluation of heap leaching for the Olive ore body was conducted in the February 2015 test program, including preliminary agglomeration test work and gold recovery through column leach tests. Tests were conducted on oxide composites from the 2014 KCA test program.

13.2.2.1 Column Leach Test Work

Column leach tests were conducted for each oxide composite, utilizing material crushed to 100% passing 9.5 mm. Each column was leached for 122 days using a sodium cyanide solution with an initial strength of 1.0 grams sodium cyanide per litre of solution with on-flow solution maintained at a target level of 0.6 grams sodium cyanide per litre of solution. Drain down tests were conducted on each column at the conclusion of leaching followed by a maximum percolation test.

A summary of the column test results is presented in Table 13.17.

Table 13.17: Column Leach Test Work Summary Metal Extractions and Chemical Consumptions

KCA Sample No.	KCA Test No.	Description	Crush Size (mm)	Calculated Head (g Au/t)	Extracted (g Au/t)	Weighted Avg. Tail Screen (g Au/t)	Extracted (% Au)	Calculated Tail p_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Cement (kg/t)
71201 A	71434	Oxide Composite A	9.5	1.769	1.206	0.563	68	6.45	122	1.53	6.02
71202 A	71437	Oxide Composite B	9.5	2.203	1.626	0.577	74	5.16	122	1.48	6.04
71203 A	71440	Oxide Composite C	9.5	1.32	0.862	0.458	65	6.31	122	1.37	6.08
Average			**--**	**1.764**	**1.231**	**0.533**	**69**	**5.97**	**--**	**1.46**	**6.05**

Source: KCA (2016)

Column test extraction results were based on carbon assays vs. the calculated head. Gold extractions ranged from 65 to 74% with an average recovery of 69%. Average cyanide consumption was 1.46 kg/t.

Column tests were also analyzed for copper and mercury. The tests showed less than 0.01 mg of mercury per kg of ore was extracted to the carbon. Copper extractions were low with the exception of KCA Test No. 71437 which showed high levels of leachable copper. Based on these results, copper and mercury are not expected to be a concern; however, should high levels of leachable copper be encountered in the field, this may result in higher cyanide consumptions.

The average 24-hour drain down for all three composites is 27.5 L/t ore.

Percentage slump and final apparent bulk density from the column tests are presented in Table 13.18. The composites had minimal percent slump with final apparent bulk densities ranging between 1.434 and 1.469 t/m^3.

Table 13.18: Percent Slump and Final Apparent Bulk Density

KCA Sample No.	KCA Test No.	Description	Crush Size (mm)	Initial Ht., (metres)	Final Ht. (metres)	Slump (%)	Final Apparent Bulk Density, (t dry/m^3)
71201 A	71434	Oxide Composite A	9.5	1.873	1.861	0.70	1.469
71202 A	71437	Oxide Composite B	9.5	1.905	1.899	0.30	1.434
71203 A	71440	Oxide Composite C	9.5	1.87	1.861	0.50	1.455

Source: KCA (2016)

13.2.2.2 February 2015 Program Conclusions

Based on this test program, 4 kg cement per tonne of ore is adequate for heap heights up to 8 m. Gold recoveries averaged 69%, while sodium cyanide consumptions averaged 1.46 kg/t. Mercury and copper are not expected to be a concern based on this test work.

13.2.3 KCA – June 2015

The KCA June 2015 test work focused on a comprehensive heap leach evaluation including bottle roll leach tests, agglomeration and compaction test work, and gold recovery through column leach tests. Samples for test work were composed of samples from four supersacks of bulk and core material, each containing a single sample. Samples included oxide material, sulphide material, transitional material and a bulk sample from the Shamrock trench. Results for material from the Shamrock trench were not considered in this evaluation.

Head analyses were completed on each sample. Portions of the head material were assayed for gold and silver content and a portion of each sample was also assayed semi-quantitatively for an additional series of elements and for whole rock constituents.

There are no deleterious elements noted in sufficient concentration to adversely affect processing by heap leaching methods.

13.2.3.1 Bottle Roll Leach Test Work

Bottle roll leach testing was conducted on portions of material from each sample. Direct bottle roll tests were performed at grind sizes of 100% passing 9.5 mm, nominal 1.7 mm material, and 80% passing 0.075 mm. CIL bottle roll tests were also conducted for each sample with a grind size of 80% passing 0.075 mm.

Results of the bottle roll leach test work are presented in Table 13.19. The bottle roll tests show a strong correlation between crush size and recovery. Recoveries for direct bottle roll leach tests and CIL bottle roll leach tests were very similar. There does not appear to be any correlation between head grade and final recoveries based on these tests.

Table 13.19: Summary of Bottle Roll Leach Test Results - Gold

KCA Sample No.	KCA Test No.	Description	Crushed/ Nominal/ Target p_{80} Size (mm)	Test Type	Head Average (g Au/t)	Calculated Head (g Au/t)	Extracted (g Au/t)	Avg. Tails, (g Au/t)	Au Extracted (%)	Leach Time (hours)	Consumption NaCN (kg/t)	Addition Ca(OH)2 (kg/t)
72278 B	72830 A	Oxide	9.5	Direct	1.094	1.445	0.989	0.455	68%	120	0.41	2.1
72278 B	72830 B	Oxide	1.7	Direct	1.094	1.286	1.013	0.273	79%	120	0.35	2.15
72278 B	72832 A	Oxide	0.075	Direct	1.094	1.32	1.222	0.098	93%	96	0.78	2.5
72278 B	72833 A	Oxide	0.075	CIL	1.094	1.184	1.099	0.086	93%	96	2.08	2.25
Average						**1.309**						
Standard Deviation						**0.107**						
Relative Standard Deviation						**8%**						
72279 B	72830 C	Sulphide	9.5	Direct	0.993	0.946	0.509	0.437	54%	120	0.47	1
72279 B	72830 D	Sulphide	1.7	Direct	0.993	1.084	0.756	0.328	70%	120	0.58	1
72279 B	72832 B	Sulphide	0.075	Direct	0.993	1.051	0.919	0.132	87%	96	0.53	1.5
72279 B	72833 B	Sulphide	0.075	CIL	0.993	1.068	0.943	0.125	88%	96	1.32	1.25
Average						**1.037**						
Standard Deviation						**0.062**						
Relative Standard Deviation						**6%**						
72280 B	72831 A	Transitional	9.5	Direct	0.426	0.486	0.295	0.19	61%	120	0.52	0.8
72280 B	72831 B	Transitional	1.7	Direct	0.426	0.517	0.329	0.189	64%	120	0.59	0.8
72280 B	72832 C	Transitional	0.075	Direct	0.426	0.549	0.475	0.074	87%	96	0.5	1.75
72280 B	72833 C	Transitional	0.075	CIL	0.426	0.751	0.682	0.069	91%	96	1.34	1.5
Average						**0.576**						
Standard Deviation						**0.12**						
Relative Standard Deviation						**21%**						

Note: The relative standard deviation for the calculated heads were good except for the Transitional sample. The cause of this is the CIL test.
Source: KCA (2016)

13.2.3.2 Agglomeration and Compaction Test Work

Compaction test work was conducted with 0 kg, 4 kg, 6 kg, 8 kg or 10 kg addition of cement per tonne of ore. Samples were placed into a column and subjected to calculated loads equivalent to 8 m, 16 m and 24 m of overall heap height.

Results of the agglomeration tests were evaluated based on percent slump, out flow of solution, and solution color. All tests for both preliminary agglomeration test work and compaction test work passed based on KCA's criteria. The scope of this test work did not include any evaluation of maximum heap height or at what height any specific cement addition rate fails. At this time, KCA recommends 6 kg cement per tonne of material, as this provides adequate pH control and is stable up to 24 m.

13.2.3.3 Column Leach Test Work

Each column was leached for 91 days with a sodium cyanide solution with an initial concentration of 1.0 grams sodium cyanide per litre of solution and on-flow solutions maintained at 1.0 grams sodium cyanide per litre of solution. Drain down tests were conducted on each column at the conclusion of leaching followed by a maximum percolation test. Results for the column leach tests are presented in Table 13.20.

Table 13.20: Column Leach Test Work Summary Metal Extractions and Chemical Consumptions

KCA Sample No.	KCA Test No.	Description	Crush Size (mm)	Calculated Head (g Au/t)	Extracted (g Au/t)	Weighted Avg. Tail Screen (g Au/t)	Extracted (% Au)	Calculated Tail p_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Cement (kg/t)
72278 B	72812	Oxide	9.5	1.62	1.1	0.52	68%	6.49	91	0.87	5.93
72278 B	72815	Oxide	9.5	1.684	1.133	0.551	67%	6.93	91	0.85	5.93
72279 B	72818	Sulphide	9.5	1.162	0.658	0.504	57%	6.26	91	0.59	5.93
72279 B	72821	Sulphide	9.5	1.177	0.639	0.538	54%	6.62	91	0.81	5.93
72280 B	72824	Transitional	9.5	0.538	0.318	0.22	59%	6.87	91	0.7	5.93
72280 B	72827	Transitional	9.5	0.522	0.295	0.227	57%	6.46	91	0.66	5.94

Source: KCA (2016)

Column gold extractions ranged from 54 to 68% based on calculated heads with cyanide consumptions ranging from 0.59 to 0.87 kg/t ore. Average non-deducted laboratory gold recoveries by material type are:

- Oxide – 68%;
- Sulphide – 56%; and
- Transition – 58%.

Calculated head for the column tests are generally within what KCA considers an acceptable deviation, which is 3% for gold.

Column tests were also analyzed for copper and mercury. The tests showed less than 0.02 mg of mercury per kg of ore was extracted to carbon. Leachable copper ranged from 18.2 mg/L to 79.6 mg/L for oxide material. Concentrations of copper for sulphide and transitional material were low, ranging between 2.39 to 17.8 mg/L for sulphide and 4.2 to 27.0 mg/L for transitional material. Based on these tests, copper and mercury are not expected to be an issue.

After completion of leaching, the percent slump and final apparent bulk density of the column tests were measured and are presented in Tables 13.21. Percent slump was minimal for all of the columns with a final apparent bulk density ranging between 1.640 and 1.686 t/m^3.

The average 24-hour drain down for all of the columns is 27.3 L/t ore.

Table 13.21: Percent Slump and Final Apparent Bulk Density

KCA Sample No.	KCA Test No.	Description	Crush Size, mm	Initial Height (metres)	Final Height (metres)	Slump (%)	Final Apparent Bulk Density, (t dry/m^3)
72278 B	72812	Oxide	9.5	1.778	1.775	0.20%	1.64
72278 B	72815	Oxide	9.5	1.772	1.772	0.00%	1.642
72279 B	72818	Sulphide	9.5	1.762	1.756	0.40%	1.665
72279 B	72821	Sulphide	9.5	1.74	1.734	0.40%	1.686
72280 B	72824	Transitional	9.5	1.734	1.73	0.20%	1.686
72280 B	72827	Transitional	9.5	1.746	1.746	0.00%	1.669

Source: KCA (2016)

13.2.3.4 Environmental Analyses

The final drain down solutions from the column leach tests were analyzed for several elements. The tests show that there are several constituents present in the un-treated solutions that do not meet water quality standards.

Environmental analyses consisting of meteoric water mobility tests (MWMT) were performed on the un-treated tailings material from each column leach test. Generally, these test results indicate a low probability of mobilizing additional contaminants.

13.2.3.5 June 2015 Program Conclusions

Based on this test program, 6 kg cement per tonne of ore is adequate for heap heights up to 24 m. Average test recoveries for gold are 68% for oxide, 56% for sulphide and 58% for transition ore types. Mercury and copper are not expected to be a concern based on this test work.

13.2.4 Summary of Olive Test Work Results to Date

Column leach tests for the KCA February 2015 and June 2015 test programs show similar gold recoveries for oxide material ranging between 67% and 74% at a crush size of 100% passing 9.5 mm (approximately 80% passing 6.3 mm). For heap leach FS purposes, KCA normally discounts laboratory gold extractions by two to three percentage points when estimating field extractions. Based on the column test work results from the two testing programs, the following recoveries are estimated for each ore type at Olive:

- Oxide – 66% gold;
- Sulphide – 53% gold; and
- Transition – 55% gold.

KCA calculated 110 days for the complete leach cycle based on solution application rates on the basis of tonnes of solution per tonne of ore with additional time required to reach the above estimated field recoveries. The column tests show an average 24-hour drain down of 27.3 L/t of dry ore and average retained moisture of 89.6 L/t.

With mostly clean non-reactive ores, cyanide consumption in production heaps is typically 25 to 35% of the laboratory column test consumptions. For ores containing high amounts of leachable copper and or silver, the higher factors should be utilized. Expected cyanide consumption is estimated at 0.37 kg/t ore.

Based on available test work, a cement addition of 6 kg/t ore should be used with a maximum stacking height of 24 m. Additional test work is required to determine if additional stacking height can be achieved at the recommended cement addition; however, based on agglomeration and compacted permeability tests completed, the ore does not appear to have any permeability issues.

Column tests predominantly show very little copper or mercury being extracted during leaching with only one column test showing high copper extraction. It is not expected that either of these elements will be deleterious to the heap leach operation and gold recovery.

14 Mineral Resource Estimates

This FS includes an update to the Mineral Resource estimate (MRE) for the Eagle Gold deposit (hereafter referred to as the Eagle Zone), which was previously described in the Wardrop 2012 FS, and an initial MRE for the Olive Zone gold mineralization, located approximately 2.5 km northeast of the Eagle Zone.

The MRE for the Eagle Zone has been prepared by Ravindra Sharma, MAusIMM (CP), SME (registered member). The MRE has been classified as “Measured”, "Indicated" and "Inferred" according to the Canadian Institute of Mining and Metallurgy (CIM) "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (May 2014).

The MRE for the Olive Zone has been prepared by Frank Daviess, MAusIMM, SME (registered member). The MRE has been classified as “Measured”, "Indicated" and "Inferred" according to the Canadian Institute of Mining and Metallurgy (CIM) "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (May 2014).

Geological data review and modelling, data verification, and QA/QC was carried out by Qualified Person Allan Moran (AIPD CPG) to support the data incorporated into resource estimations for both the Eagle and Olive mineralized zones.

Detailed data verification and QA/QC was carried out to support the data incorporated into the resource estimation. Datamine Studio3 software (Datamine), a commercially available geology and mining software package was used for examining geological domains, block modelling and grade estimation. The initial geological model was created in Leapfrog and was modified in sections using Datamine. Snowden’s Supervisor software was used for exploratory data assessment, analysis of grade variograms, and continuity analysis. The grade estimation was carried out in Datamine.

MRE methods and results are described here for both the Eagle and Olive Zones, with more detail provided for the Eagle Zone, as the primary deposit of interest.

14.1 Eagle Zone - Drill Hole Database

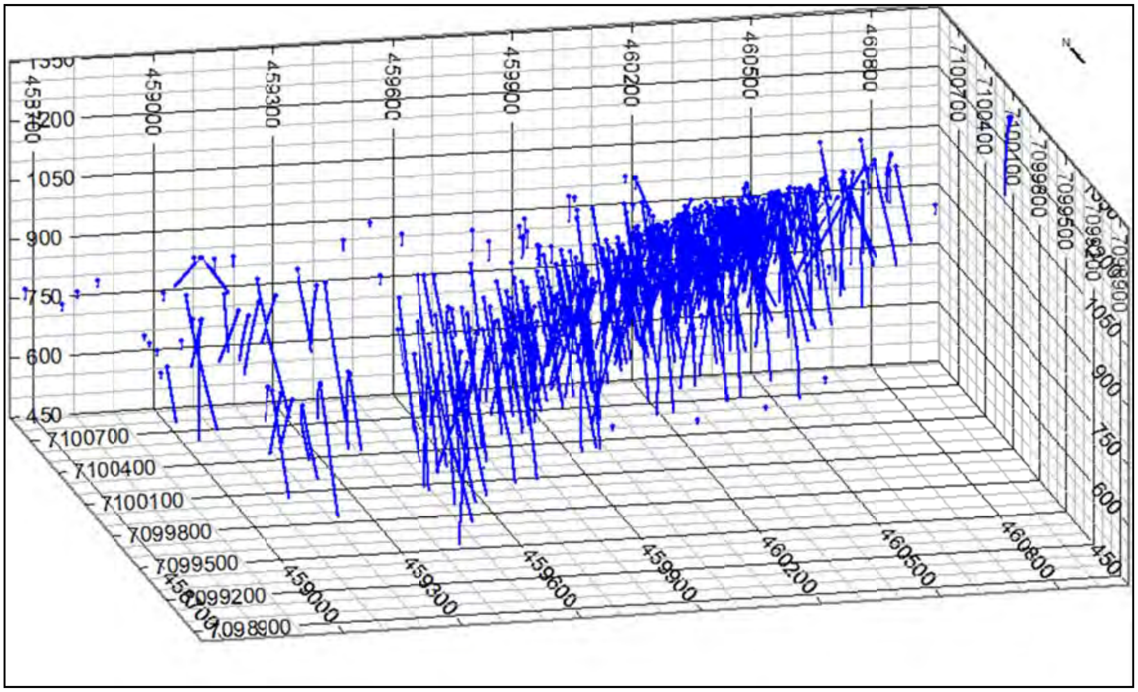
The drill hole database was received in MS Access Database and Excel format along with 470 CSV and PDF assay certificates from the analytical labs. The digital database included the following file information:

- Collar – drill hole name, easting, northing, elevation, total depth, type, and purpose;
- Survey – drill hole name, depth of survey, azimuth, and dip;
- Assays – drill hole name, from/to, sample ID, certificate number, Au, Ag and other elements from the ICP analysis;
- Geology – drill hole name, from/to, sample ID, degree of oxidation, sericitization, chloritization and silicification, vein type and thickness;
- Lithology – Drill hole name, from/to, rock type (such as overburden, Granodiorite, altered Granodiorite, hornfels, quartzite, and fault zone); and
- Recovery, RQD - Drill hole name, from/to, recovery percentage, RQD percentage.

Included as well in the database were digital copies of surface geological maps, scanned images of cross-sections and long sections, 1 m and 2 m digitized topo, bulk density file data (in Excel format), and standard operating procedures for collar, down hole survey, logging and in-house density determination procedures.

The Eagle database included 413 holes, consisting of 241 core and 172 RC drill holes. Out of 413 holes, 43 holes (25 core and 18 RC) were flagged as not verified, and are not used in the resource estimation. In addition, a total of 1,160 sampled and analyzed intervals out of 39,530 intervals within the grade shell were not used in the grade estimation, due to poor core recovery (less than 60%). The geology information from all the holes was used for geological interpretation and geological modelling. For grade estimation, only holes which could be verified and flagged as verified were used. Figure 14.1 illustrates the drill hole locations in 3D.

Figure 14.1: Eagle Zone Drill Hole Locations in 3D



Source: AVM (2016)

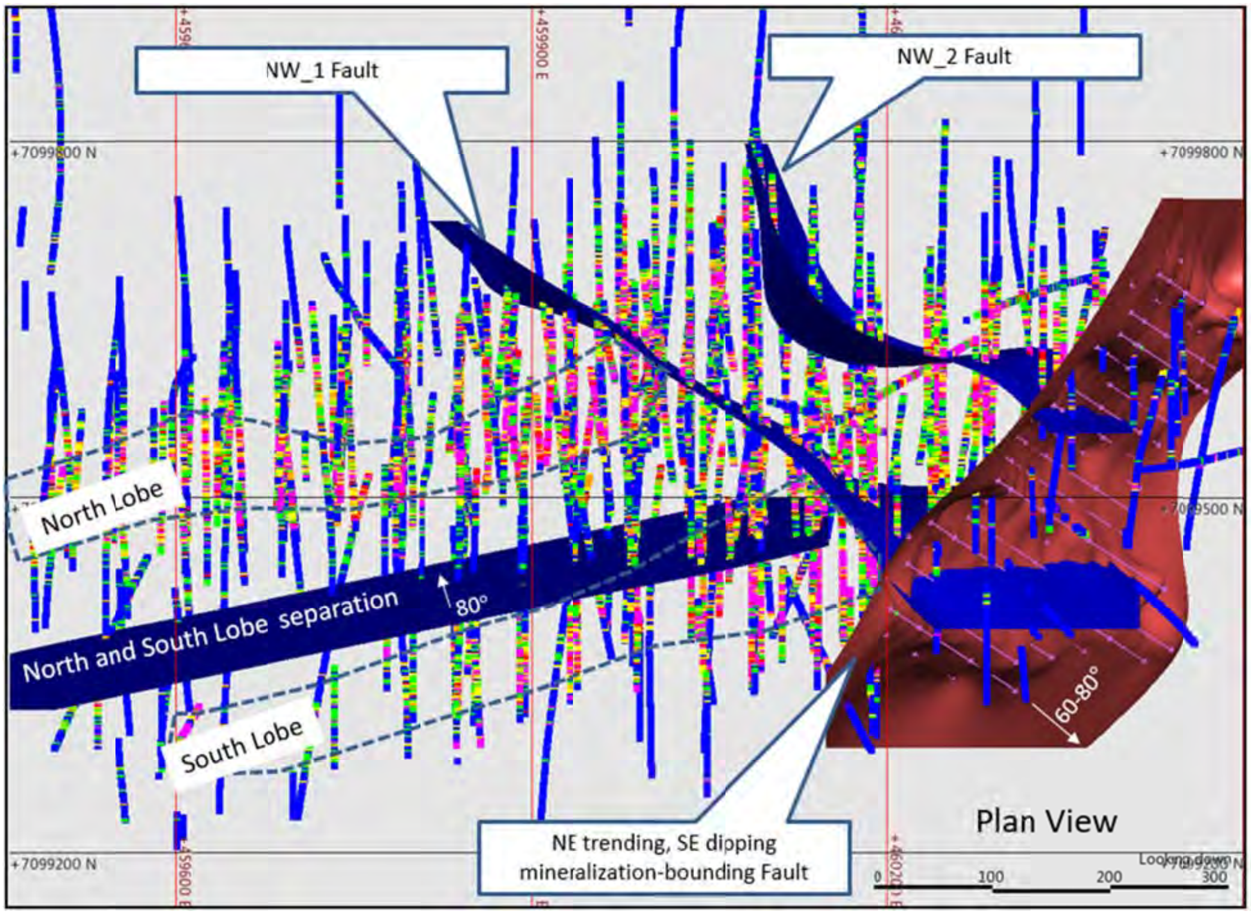
14.2 Eagle Zone - Exploratory Data Analysis

Exploratory data analysis (EDA) forms an essential part of Mineral Resource estimation and involves a thorough organization and understanding of data that is the basis for resource estimation. Exploratory data analysis was carried out with the following objectives:

- To understand the distribution of gold and to recognize any systematic spatial variation of grade with respect to major lithological units;
- To recognize and define distinctive geologic/structural domains that should be evaluated independently in resource estimation;
- To compare and understand bias/distribution of gold in RC and core drilling, as described in previous Section 12.2.4 (RC versus Core assays Verification);
- To understand the correlation between gold and silver;
- To understand relationship between gold and quartz vein density;
- To correlate Victoria Gold's gold in-house and independent-lab bulk density data, as described in previous Section 12.2.3 (Bulk Density Verification);
- To identify any analytical errors not picked up in the data verification process; and
- To improve the quality of estimation by understanding the data spread/distribution/behaviour.

Leapfrog was used to interpret fault/structural surfaces based on maps and sections, supplied by Victoria Gold geologists. Spatial distributions of gold grade in relation to these interpreted structures are illustrated in Figure 14.2.

Figure 14.2: Plan Map of Interpreted Geological/Structural Domains for EDA



Source: AVM (2016)

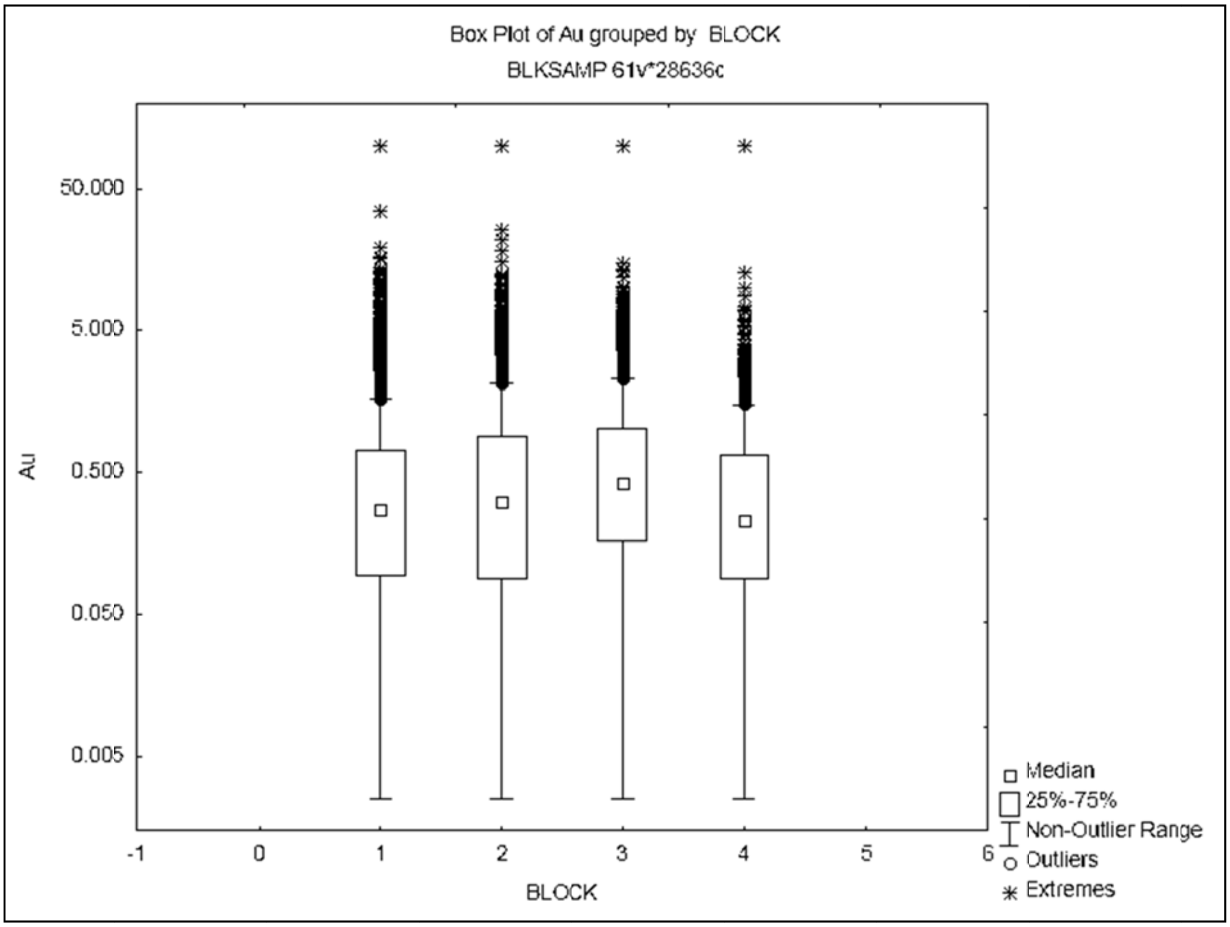
The following visual observations were made on the distribution of gold mineralization in relation to interpreted fault surfaces:

- Two distinct mineralization lobes, north and south lobes of gold mineralization are observed trending in a northeast–southwest direction;
- Minor displacements are noted in the mineralization on either side of the NW-1 fault surface;
- Minor displacements are noted in the mineralization on either side of the NW-2 fault surface; and
- The NE trending fault surface limits mineralization on the extreme SE side.

These surfaces were used to create four structural blocks. The South lobe was coded as block 1, the North lobe was coded as block 2, the areas between the two NW trending faults was coded as Block 3, and the far eastern mineralization was coded as Block 4. Box plots of the gold assays from these four structural blocks (Figure 14.3) did not indicate any significant variation in grade populations or distributions that should be considered separately for data partitioning and separate grade estimation of these blocks. Samples in these four blocks were thus considered as a single population.

Partitioning of the data into these four structural blocks may make sense geologically; however, the disadvantage in doing so will be the creation of artificial boundaries between these blocks. Those artificial boundaries will then result in a restricted number of samples used in the estimation process for each structural block, which in turn will negatively impact the quality of the resource estimate. Structural domains were thus deemed not necessary and were not used in the resource modelling.

Figure 14.3: Plan Map of Interpreted Geological/Structural Domains for EDA



Source: AVM (2016)

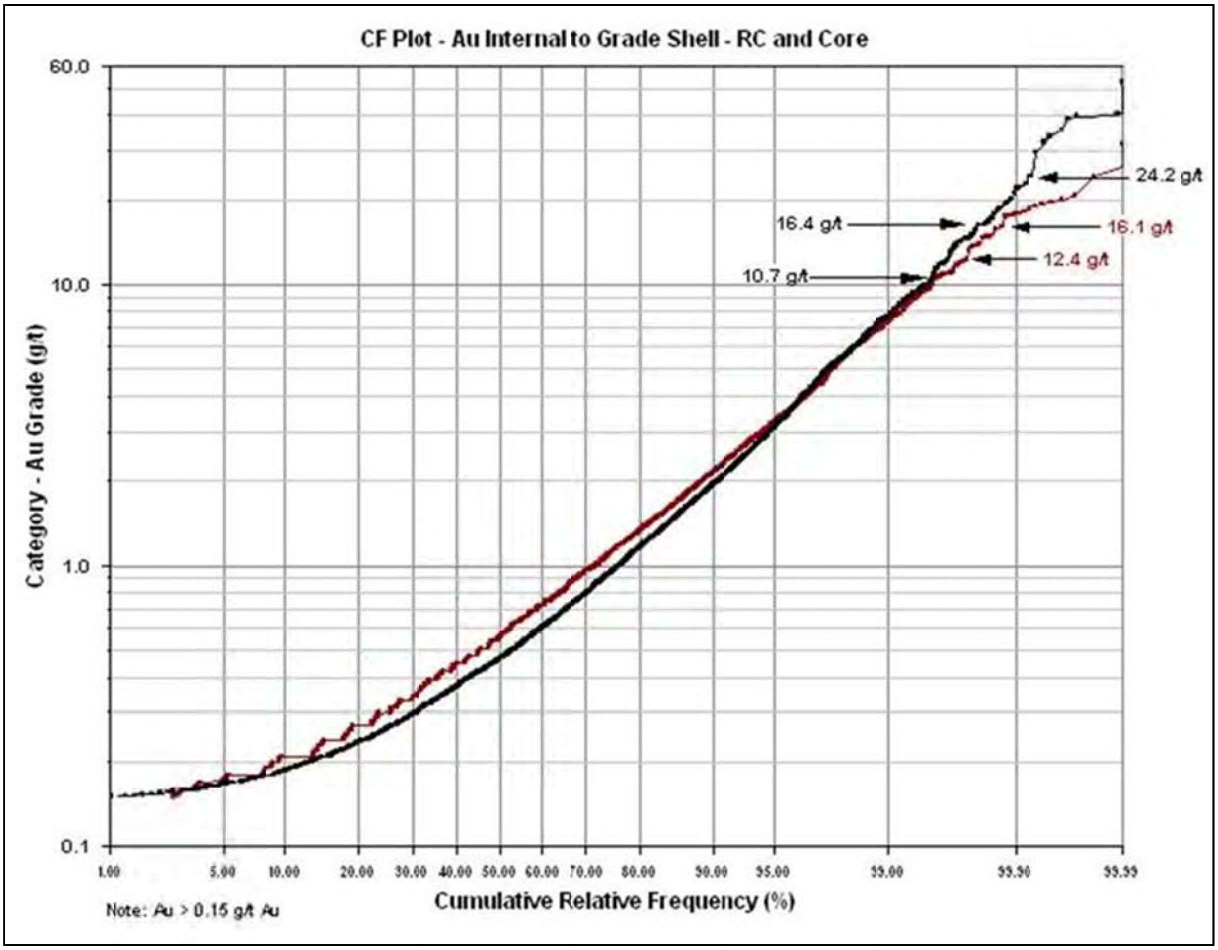
Section 12.2.4 describes the correlation between core and RC drill hole sample assays, which was undertaken to examine potential bias due to drilling technique. RC drilling was determined to have two characteristics that cause an apparent high bias of RC gold assays relative to core sample gold assays: a) a preferential location bias of RC drilling in the higher grade core of the mineralization, which results in fewer low grade assays, and b) an assay precision problem on the low-end detection limit of RC samples, assayed largely in the early 1990's, compared to the lower level detection limit for more recently assayed core sample gold assays.

A visual inspection of vein density and its correlation with gold assays concluded that, while there is a general direct relationship of quartz vein density to the general location of gold mineralization, the gold grades were geologically related to late fractures, and were not directly related to quartz vein intensity by a 1:1 ratio. The resource modelling was therefore based on the modelling of gold grades and not on lithology, structure or quartz veining. The authors examined RC versus core drilling assays, and determined the differences were negligible for the range of assays relevant to the resource estimation process. All RC holes were used, except for two shallow holes with assays that were significantly different from close-by (pseudo-twin) core holes.

Gold and silver did not have a direct correlation, and not all the drill hole samples were analyzed for silver. There were 23,219 silver assays, compared to 38,370 gold assays, for samples within the mineralized grade shell. However, the incomplete silver data precluded it from being used in the resource estimation.

Basic statistics showed a coefficient of variation (CV) of 2.01 for gold, which indicated there was not a significant impact of higher grade gold on the largely low grade gold population. The gold assay data was examined in a cumulative frequency (CF) plot, for the purpose of identifying the outlier grade for capping prior to grade estimation. Figure 14.4 illustrates the CF plot on gold for core and RC.

Figure 14.4: CF Plot of Au Grade for Capping Analysis



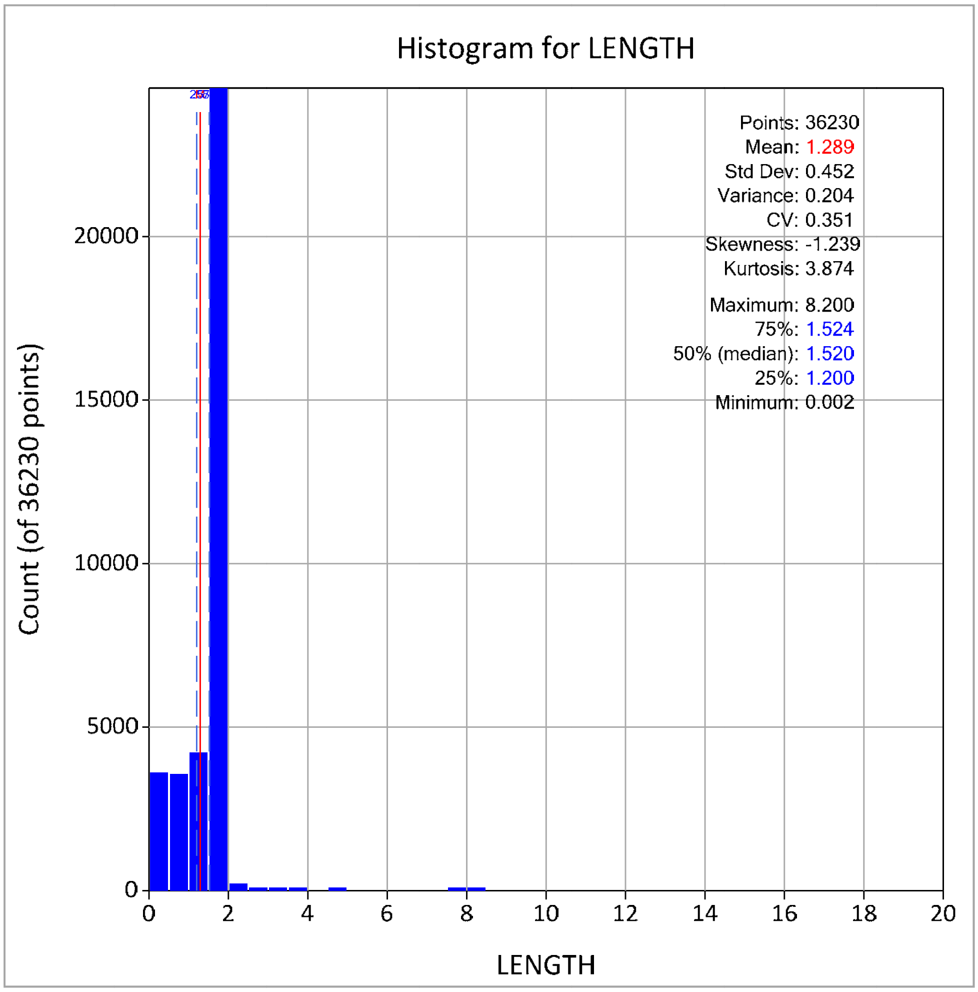
Note: Core (black), RC (red)
Source: AVM (2016)

A break in the CF curve was investigated for core and RC, with 16.0 g/t selected as a capping (top cut) on the gold grades. A total of 55 assays fell above 16.0 g/t Au and were capped there. Table 14.1 shows a comparison between the basic statistics before and after capping.

14.3 Eagle Zone - Compositing

Length-weighted compositing was carried out on capped samples from the collar to the end of the hole, using Datamine. Compositing was carried out after excluding non-verified drill holes and samples with a recovery of less than 60%. The average sample interval was 1.313 m, with more than 50% of the samples between 1.75 to 2 m in length. There were some extreme intervals with a length of 115.83 m. The statistics were evaluated for sampled intervals of the Datamine "holes" file with Au > 0.1 g/t. The average sample length in this case was 1.289 m, with a maximum sample length of up to 8.2 m (Figure 14.5). Composites were not restricted within the mineralized shell boundary, to allow for edge dilution at the mineralized shell boundary (shown in Figure 14.8).

Figure 14.5: Histogram of Sampled Holes File Intervals greater than 0.1 g/t Au



Source: AVM (2016)

A larger composite length of 2.5 m (approximately double the average length) was selected to reduce the overall variance of grade by effectively increasing the sample volume. The standard deviation after compositing dropped from 1.428 to 0.989 and the CV dropped from 2.034 to 1.426. The drop in mean gold grade from un-composited (0.702) to composited samples (0.693) was 1.3% (Table 14.1). Regular length compositing was carried out to provide a uniform sample support by length. Sample density weighting was not used during compositing, as not all the samples were analyzed for density and density variations were considered minimal.

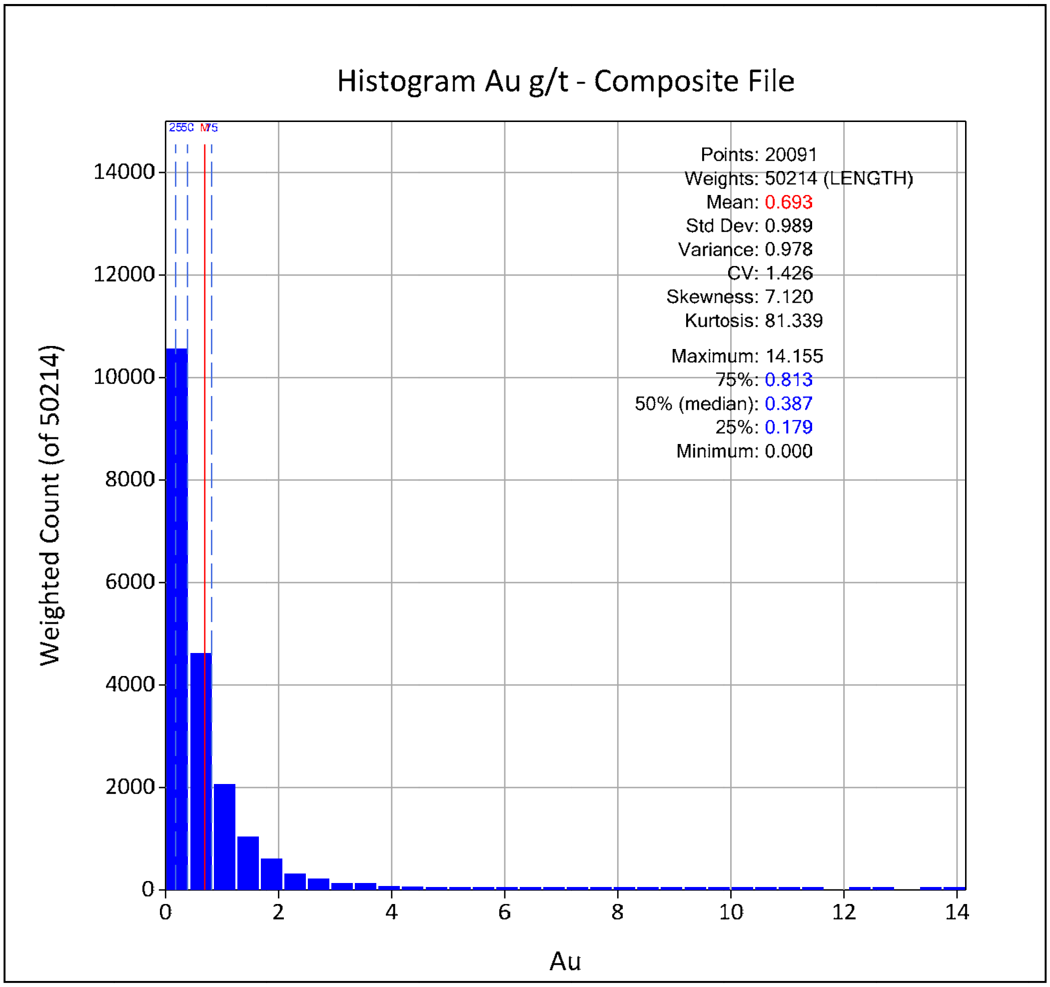
Table 14.1: Comparative Statistics of Gold between Capped and Uncapped 2.5 m Composites within the Grade Shell

	Number of Samples	Min.	Max.	Mean	Variance	Standard Deviation	Skewness	Kurtosis	CV
Not Capped, Not Composited Holes File	38,690	0	52.36	0.702	2.039	1.428	11.306	251.64	2.034
Capped, Not Composited Holes File	38,690	0	16	0.693	1.631	1.277	6.983	72.61	1.842
Capped and Composited Holes File	20,091	0	14.155	0.693	0.978	0.989	7.12	81.339	1.426

Source: AVM (2016)

The histogram of Au g/t grades from the composite file is shown in Figure 14.6, and shows a typical lognormal distribution.

Figure 14.6: Histogram of Au g/t Composite File

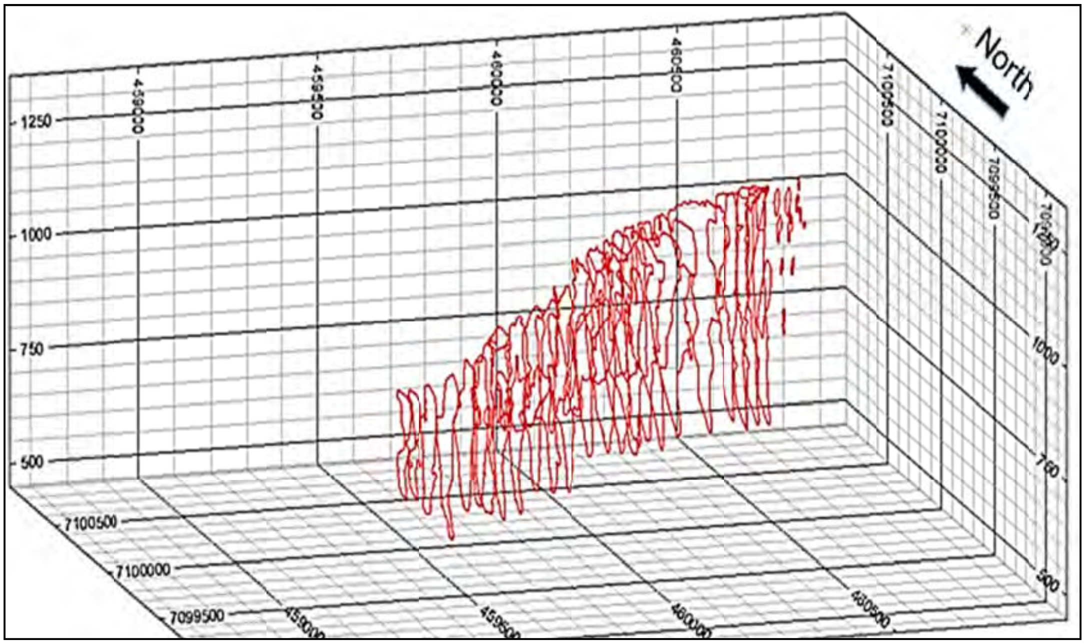


Source: AVM (2016)

14.4 Eagle Zone - Geological Model

As a first pass, Leapfrog software was used to define the mineralized shell based on drill hole gold assay data. The drill holes were used in 2011 to define a mineralized shell at a cut-off of 0.15 to 0.20 g/t Au. This mineralized shell was modified to include the most recent data of 2011 and 2012. The wireframe shape was exported from Leapfrog to Datamine, and used as a guideline for 2D cross-sectional interpretations and as an adjustment of the mineralized boundary in relation to drill hole data (digitized polygons or strings). This was carried out section by section, incorporating geological and assay information from drill holes. Sections were created at an approximate distance of 30 to 40 m, based on the number of holes passing through or near the section. Figure 14.7 illustrates the digitized sectional polygons (strings) that were subsequently used to create the mineralized shell.

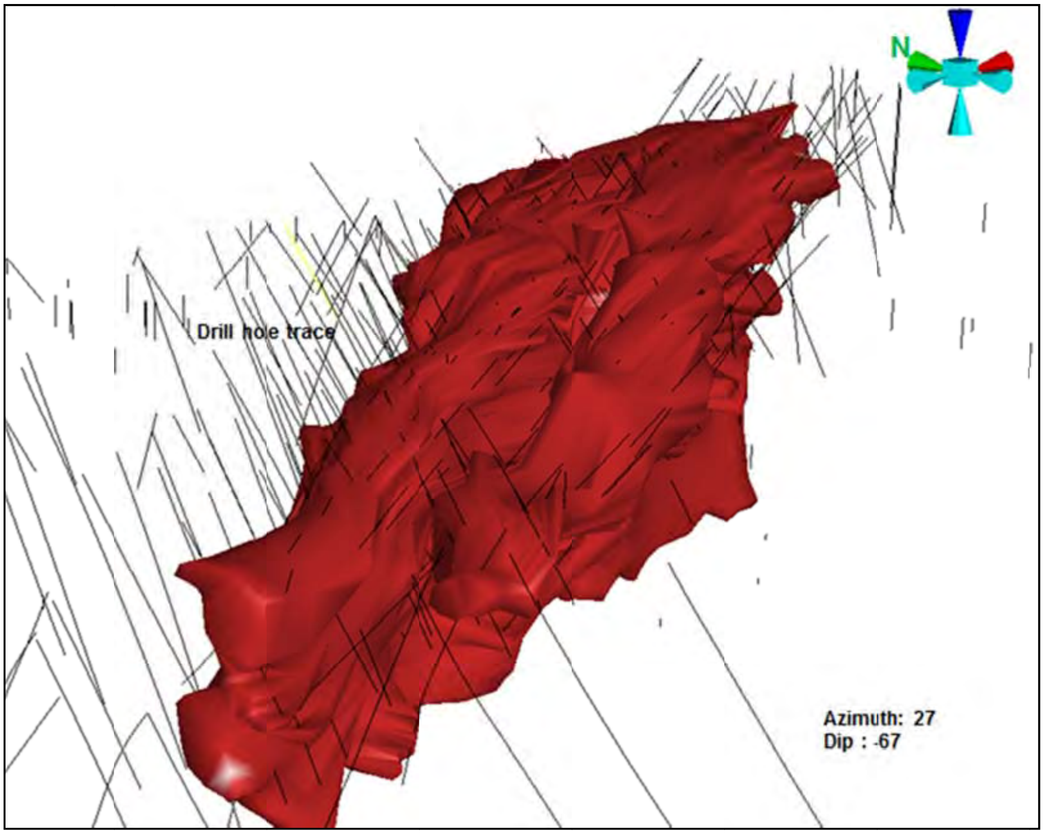
Figure 14.7: Digitized Sectional Polygons (Strings)



Source: AVM (2016)

All verified, as well as not verified, drill holes were used in geological wireframe interpretations, except hole 92-031 R and 96-267R (both VERIFIED=N), which showed suspect grades relative to nearby core holes. For grade estimation, only verified holes were used (VERIFIED=Y). The final wireframe limits are based on a gold grade ranging between 0.15 to 0.2 g/t, which generally represents a natural break or sharp change in grade representing mineralization. Figure 14.8 illustrates the 3D mineralized wireframe solid.

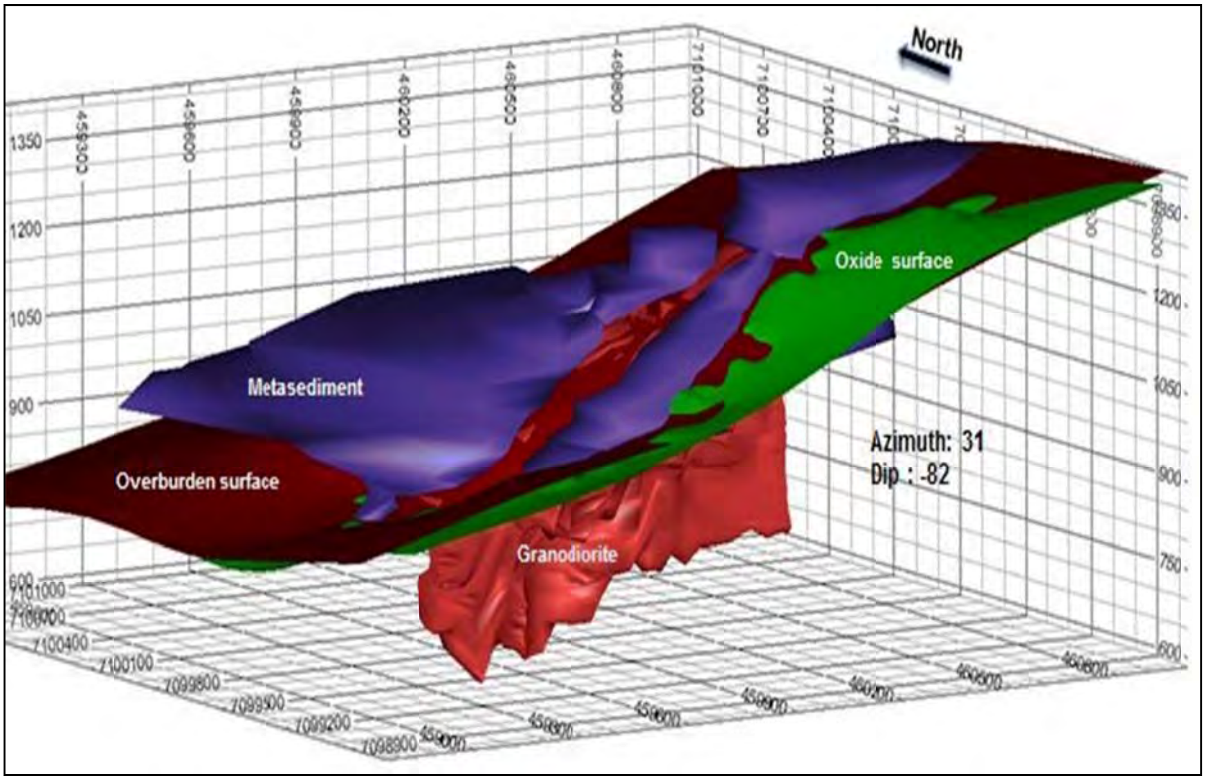
Figure 14.8: 3D Mineralized Wireframe Solid



Source: AVM (2016)

Leapfrog software was used to create oxidation and overburden surfaces, and metasediment-Granodiorite contacts, as illustrated in Figure 14.9.

Figure 14.9: Oxide, Overburden surface and Metasediment Wireframe



Source: AVM (2016)

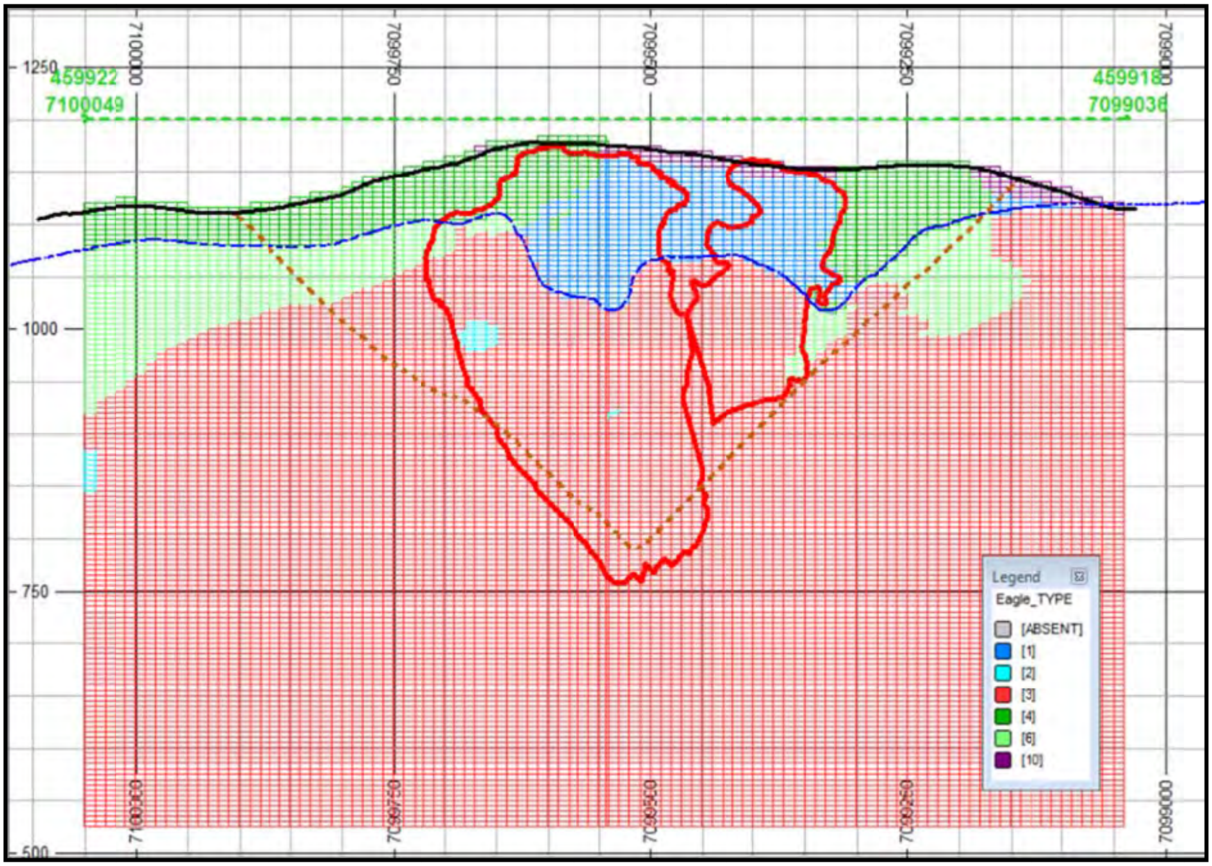
These surfaces and wireframes were imported into Datamine to code the block model with geo-metallurgical codes, as shown in Table 14.2. Figure 14.10 gives a north to south (N-S) section view for geo-metallurgical codes used in the block model.

Table 14.2: Lithologic/Oxidation/Alteration Model Types

Type	Description
1	Oxide Granodiorite
2	Altered Granodiorite
3	Unaltered Granodiorite
4	Oxide Metasediments
6	Unaltered Metasediments
10	Overburden

Source: AVM (2016)

Figure 14.10: Different Blocks based on Lithologic/Oxidation/Alteration Model Types



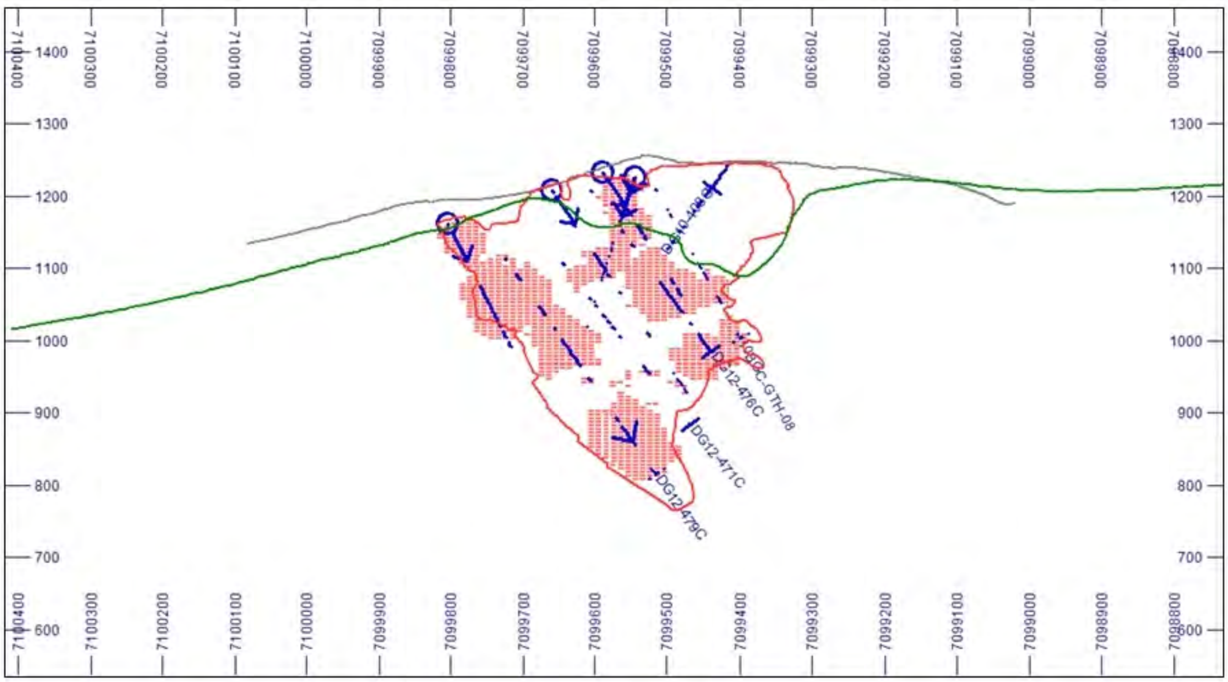
Note:
Topo Surface (black outline), oxidation surface (blue outline), mineralized shell outline (red) and pit outline (brown)
Source: AVM (2016)

The lithology code was assigned to the blocks, first by coding them as oxidized or fresh, according to their position relative to the oxidation surface (OXIDE=0) for blocks below oxidation surface and OXIDE=1 for blocks above oxidation surface. The type was then assigned according to the following conditions and order:

- Type 1 – Oxidized Granodiorite – Granodiorite above the oxidation surface;
- Type 3 – Unaltered Granodiorite – Granodiorite below the oxidation surface and then with a sericite indicator less than or equal to 0.52;
- Type 2 – Altered Granodiorite – Granodiorite below the oxidation surface and sericite indicator greater than 0.52;
- Type 10 – Overburden – Above the overburden surface;
- Type 4 – Oxidized Metasediments – Metasediments within metasediment wireframe and above the oxidation surface; and
- Type 6 – Unaltered Metasediments – Metasediments within metasediment wireframe and below the oxidation surface.

An indicator estimation was done to assign sericite alteration to the Granodiorite blocks below the oxidation surface. Composites were coded as zero if the sericite alteration was greater than or equal to three, and one if the alteration was less than three. Variograms were calculated on this indicator variable and ordinary kriging was used to estimate the indicator. The resulting indicator variable in the model was then visually reviewed against the drill holes to determine the best value to use to assign sericite alteration to the blocks. In this case a value of 0.7 was deemed to have the best correlation with the drill holes. Figure 14.11 is a comparison of indicator values in composites filtered at 1, and the block model filtered at 0.7.

Figure 14.11: Comparison of Composites and Blocks for Indicator Estimation of Sericite Alteration



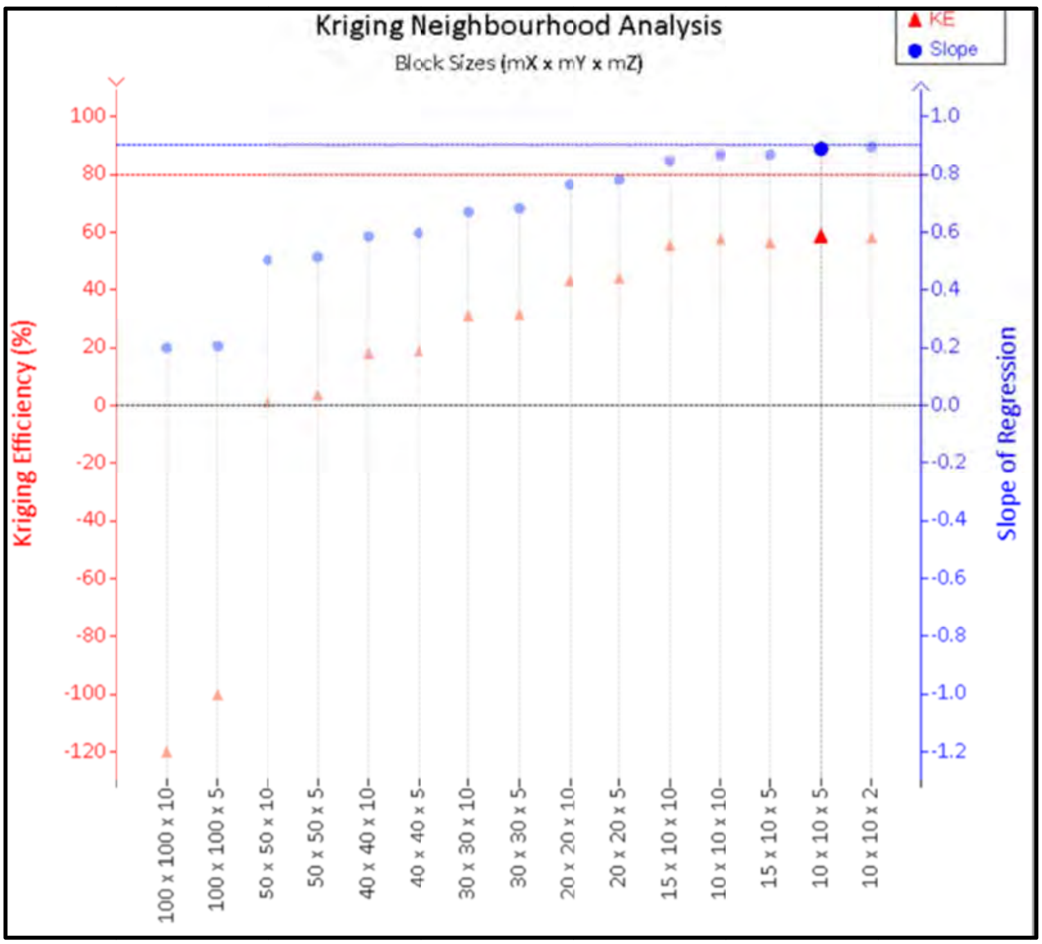
Source: AVM (2016)

14.5 Eagle Zone - Block Model

Optimum suggested block size was examined with Kriging Neighbourhood Analysis (KNA), where n, Kriging efficiency (KE) and slope regression were varied and evaluated. A block size of 10m x 10m x 5 m gave the best results using KNA. Figure 14.12 illustrates the KNA on various block sizes, from panel size to smaller size blocks, which were tried using variogram parameters. The block size, which gave the best combination of slope and KE, was investigated further for impact on the number of samples in the search, and the discretization of blocks.

A 10 X 10 X 5 block size resulted in a 90% slope and 60% KE; this justified using a small size block. A smaller block size is generally not preferred, as it is prone to conditional bias, if it is applied without understanding the kriging operation. The KE and Slope are functions of how well informed the block is given the sample density.

Figure 14.12: Kriging Neighbourhood Analysis on Different Block Sizes



Source: AVM (2016)

The regularization of the block model was done at the size of the parent block. FILLVOL and VOIDVOL parameters were created in an output model, refer to Table 14.4 for further explanation. The OREFRAC option was used in Datamine to report only the FILLVOL portion in the resource statement. The block model parameters are summarized in Table 14.3.

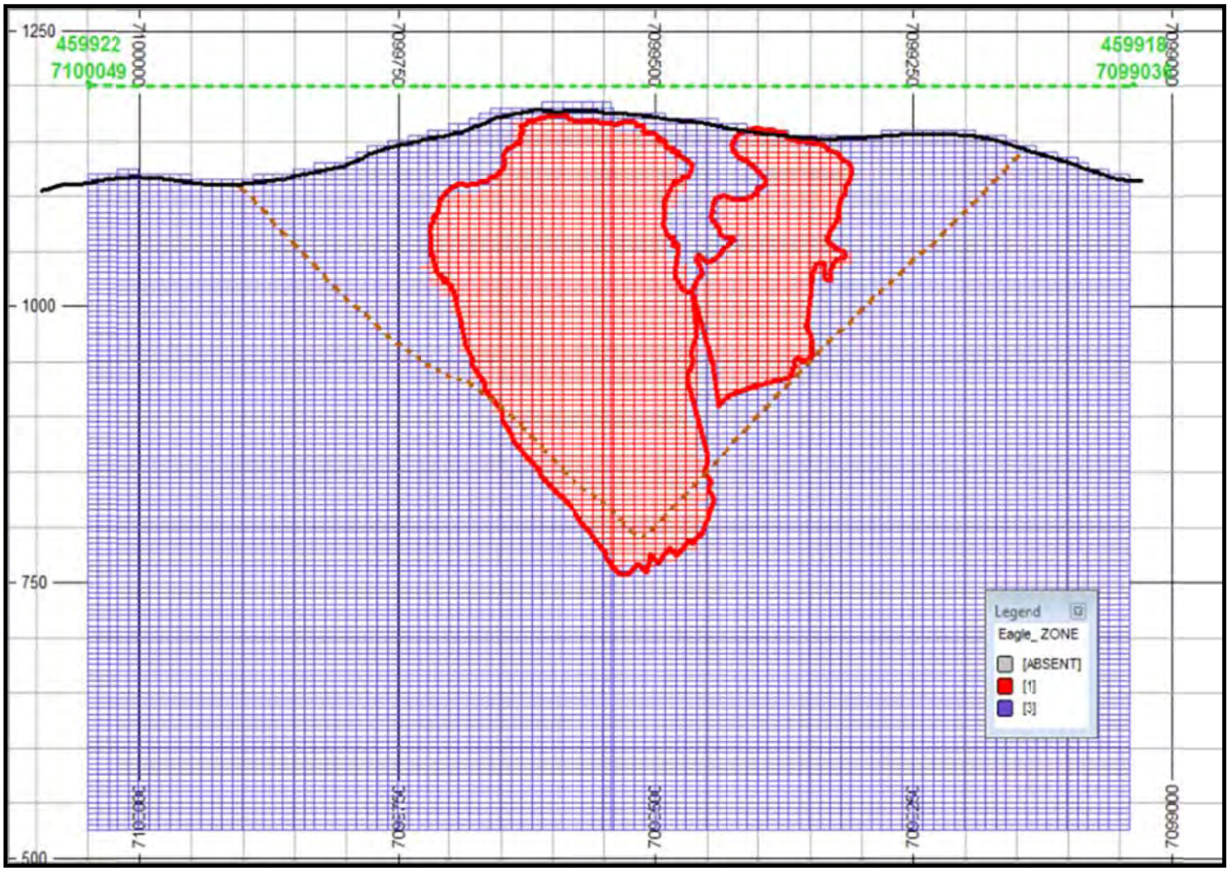
Table 14.3: Block Model Parameters

Parameter	Units	Value
Block Size in X	m	10
Block Size in Y	m	10
Block Size in Z	m	5
X Origin	m	458800
Y Origin	m	7098800
Z Origin	m	525
No. Of Blocks in X		240
No. Of Blocks in Y		141
No. Of Blocks in Z		190

Source: AVM (2016)

The block model was truncated with a topographic DTM and coded for Lithological/Oxidation/ Alteration Types. Based on the metallurgical recovery, assignment for Ore blocks was done as described in Table 14.3. The block model was also coded with field ZONE= 1 for the mineralized grade shell (refer to the red blocks on Figure 14.13) and ZONE=3 for waste blocks (refer to blue blocks on Figure 14.13). Table 14.4 provides details on the block model fields and their description.

Figure 14.13: Block Model North-South Section, Looking East, Illustrating ZONE Field, DTM, Mineralized Shell Outline and Pit Outline



Source: AVM (2016)

Table 14.4: Block Model Fields

Parameter	Explanation
XC	X Centre
YC	Y Centre
ZC	Z Centre
IJK	Used by Datamine to position parent cells within the model. Each Parent cell will have a unique IJK value
XINC	Block Size in X-Direction
YINC	Block Size in Y-Direction
ZINC	Block Size in Z-Direction
FILLVOL	Total volume in the output block filled by Lithological/Oxidation, Alteration TYPES
VOIDVOL	Total block volume minus volume defined in FILLVOL
TOPO	Used To Code Blocks below Topo DTM (TOPO=1)
OXIDE	Used to code blocks above Oxide surface (OXIDE=1) and to code blocks below oxide surface (OXIDE=0)
TYPE	TYPE, Lithological/Oxidation, Alteration Types (1=Oxide Granodiorite, 2=Altered Granodiorite, 3=Unaltered Granodiorite, 4=Oxide Metasediments, 6=Unaltered Metasediments, 10=Overburden)
ZONE	Used to Code Blocks within ore body wireframe (ZONE=1) and waste blocks (ZONE=3)
ACODE_OK	Indicator estimation on alteration code for coding Altered Granodiorite
DENSITY	Density in g/cm^3 applied as a constant value for each TYPE (TYPE 1=2.62, TYPE 2=2.63, TYPE 3=2.65, TYPE 4=2.61, TYPE 6=2.69, TYPE 10=2)
METREC	Metallurgical Recovery Assignment for Ore blocks (TYPE 1 = 79%, TYPE 2 = 73%, TYPE 3 = 68%, TYPE 4 = 68%, TYPE 6 = 68%, TYPE 10 = "-")
Au_OK	Ordinary Kriging for Au. This is final grade field
Au_ID	Inverse distance square for Au for comparing with Ordinary Kriging (Au_OK)
FVALUE	Geostatistical Fvalue for Regression Slope calculation
LAGR	Lagrange Multiplier for Regression Slope calculation
NSAMP	Number Of Samples Used To Estimate each Block
SVOL	Search Volume
KV	Kriging Variance
0	Dummy Field
BLKVAR	Block Variance used for calculating regression Slope and Kriging Efficiency
SLOPE	Regression Slope
EFFY	Kriging Efficiency
CLASS	Code for Measured, Indicated and Inferred Resource Category (1 - Measured, 2 - Indicated, 3 - Inferred)
OREPERC	Percentage of block filled (1 means 100%)
XMORIG	Block Model Origin in X
YMORIG	Block Model Origin in Y
ZMORIG	Block Model Origin in Z
NX	Number of blocks in X-direction.
NY	Number of blocks cell in Y-direction.
NZ	Number of blocks cell in Z direction

Source: AVM (2016)

14.6 Eagle Zone - Bulk Density

As detailed in Section 12.2.3, 1,210 samples analyzed by ALS Chemex and in-house by Victoria Gold for bulk density were checked with 300 determinations by SGS labs. The mean values by rock type, using all data determinations, were used for the resource estimation, by the nearest neighbour assignment method. Table 14.5 indicates the various mean value determinations by rock type.

Table 14.5: Bulk Density by Rock Type

Rock Type	MET Type		In House (t/m^3)	SGS (t/m^3)	ALS (t/m^3)	Mean (t/m^3)
		ALL DATA (No outliers)	2.66	2.65	2.65	2.65
1	A	Oxidized Granodiorite	2.62	2.62	2.61	2.62
3*	B	Fresh Granodiorite (unaltered)	2.66	2.65	2.65	2.65
2	C	Altered Granodiorite	2.65	2.62	2.63	2.63
4	E	Oxidized Metasedimentary Rock	2.62	2.59	2.61	2.61
6		Fresh Metasedimentary Rock	2.68	2.72	2.66	2.69

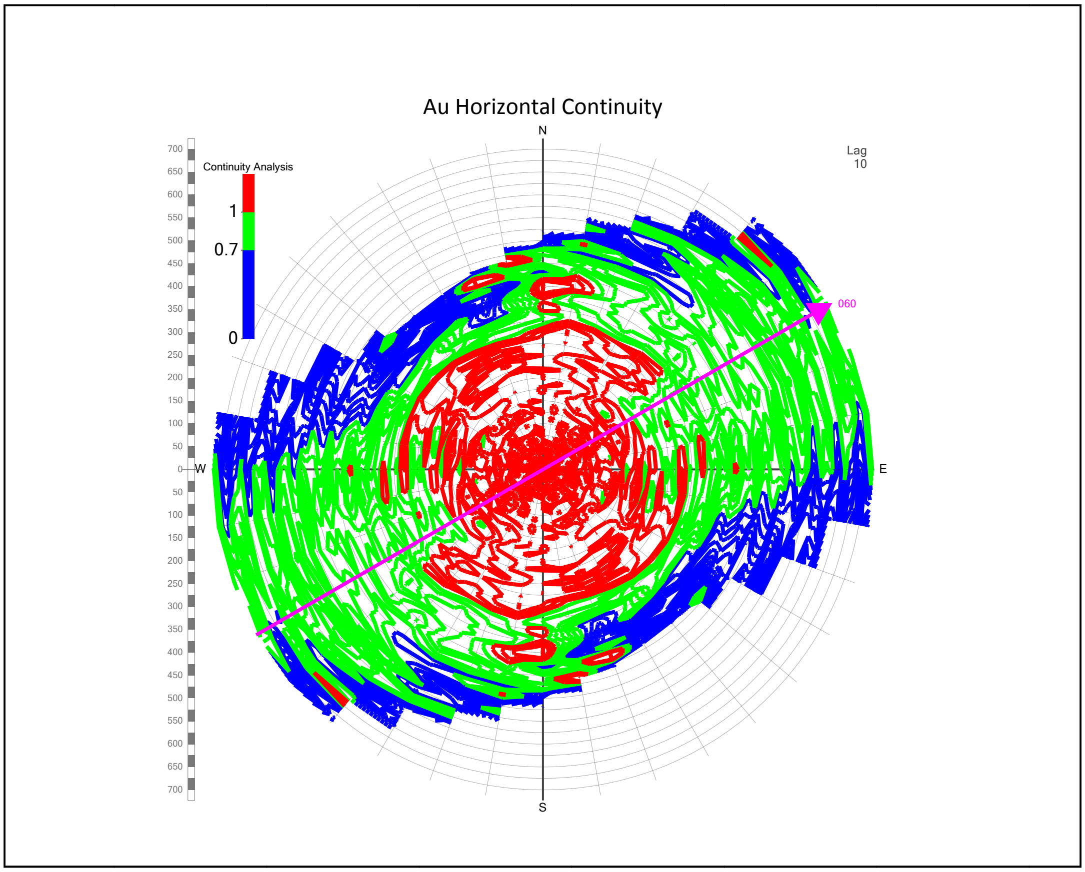
* Note: this is the correct type code - they were originally numbered by from surface downward: Ox, Alt, Fresh, as 1,2,3 for Granodiorite
Source: AVM (2016)

14.7 Eagle Zone - Variogram Analysis and Modelling

Snowden's Supervisor software was used to create directional pair-wise relative variograms on gold composites. Only composites within the mineralized shell were used for variogram analysis. The first step involved the generating of fan diagrams, and analysis of a series of fans in the horizontal-strike, across-strike, vertical, and dips planes. Each fan, which is made up of variogram contours, was analyzed to select the direction of maximum continuity in the fan.

Variance contours from the horizontal plane were used to identify the directions of continuity of mineralization, and the strike of the mineralization is N60E (Figure 14.14). The direction of maximum continuity across-strike, and in the direction of the dip of the mineralization, was similarly determined as 150°and 34°plunge, respectively.

Figure 14.14: The Variance Contours Fans in the Horizontal Plane to Identify the Strike of the Mineralization



Source: AVM (2016)

A nested spherical pair-wise relative variogram with three structures was modelled for composited gold. The downhole variogram indicated a short scale variability less than 15 m, and a high nugget effect of 50%, which is normal for gold mineralization. The resultant variogram generally demonstrated reasonable structure and orientation.

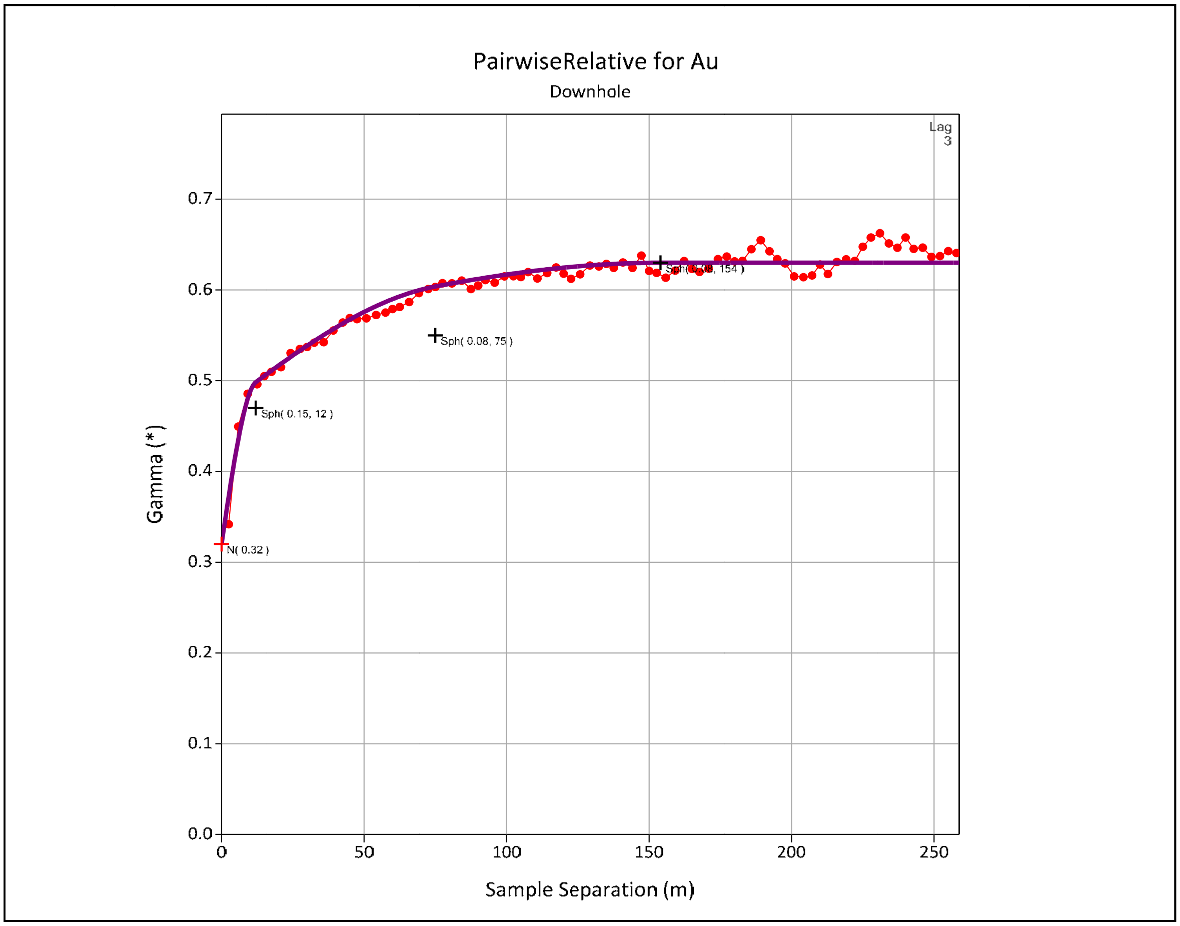
Variogram model parameters are tabulated in Table 14.6, and shown in Figures 14.15 through 14.18.

Table 14.6: Variogram Parameters

Assay		Au
VREFNUM		1
VANGLE1	degrees	60
VANGLE2	degrees	-140
VANGLE3	degrees	-150
VAXIS1		3
VAXIS2		2
VAXIS3		3
NUGGET		0.32
ST1		1
ST1PAR1	m	16
ST1PAR2	m	11
ST1PAR3	m	11
ST1PAR4		0.18
ST2		1
ST2PAR1	m	72
ST2PAR2	m	31
ST2PAR3	m	43
ST2PAR4		0.07
ST3		1
ST3PAR1	m	171
ST3PAR2	m	86
ST3PAR3	m	106
ST3PAR4		0.11

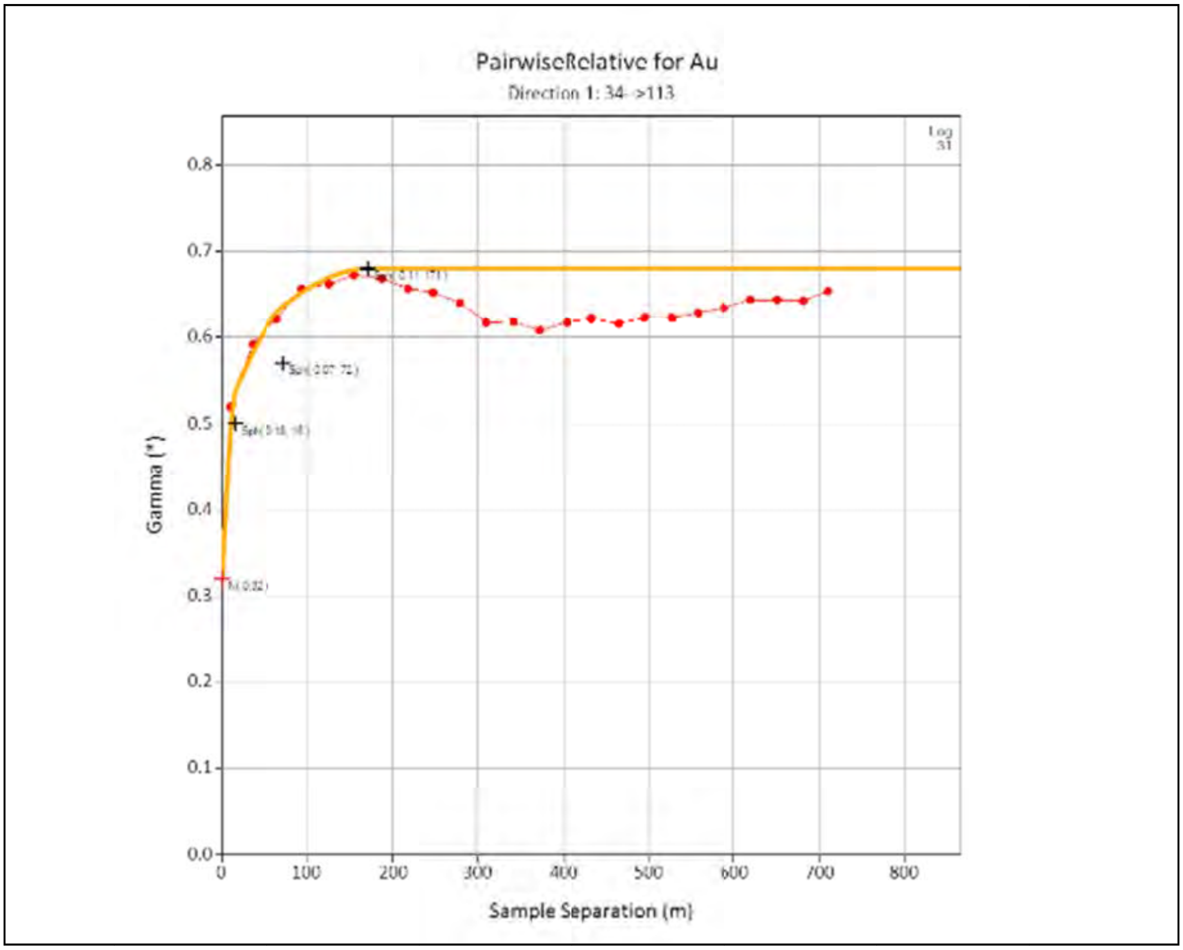
Source: AVM (2016)

Figure 14.15: Variogram – Downhole



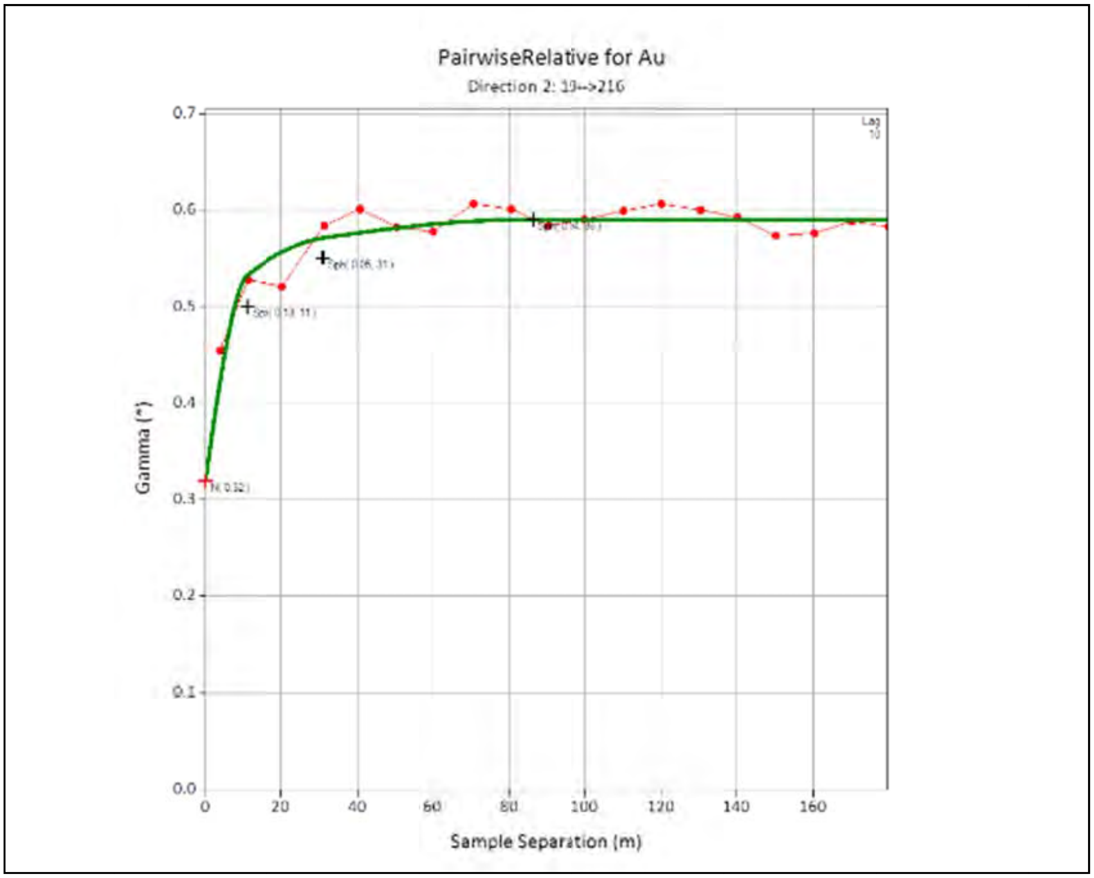
Source: AVM (2016)

Figure 14.16: Variogram – Direction 1



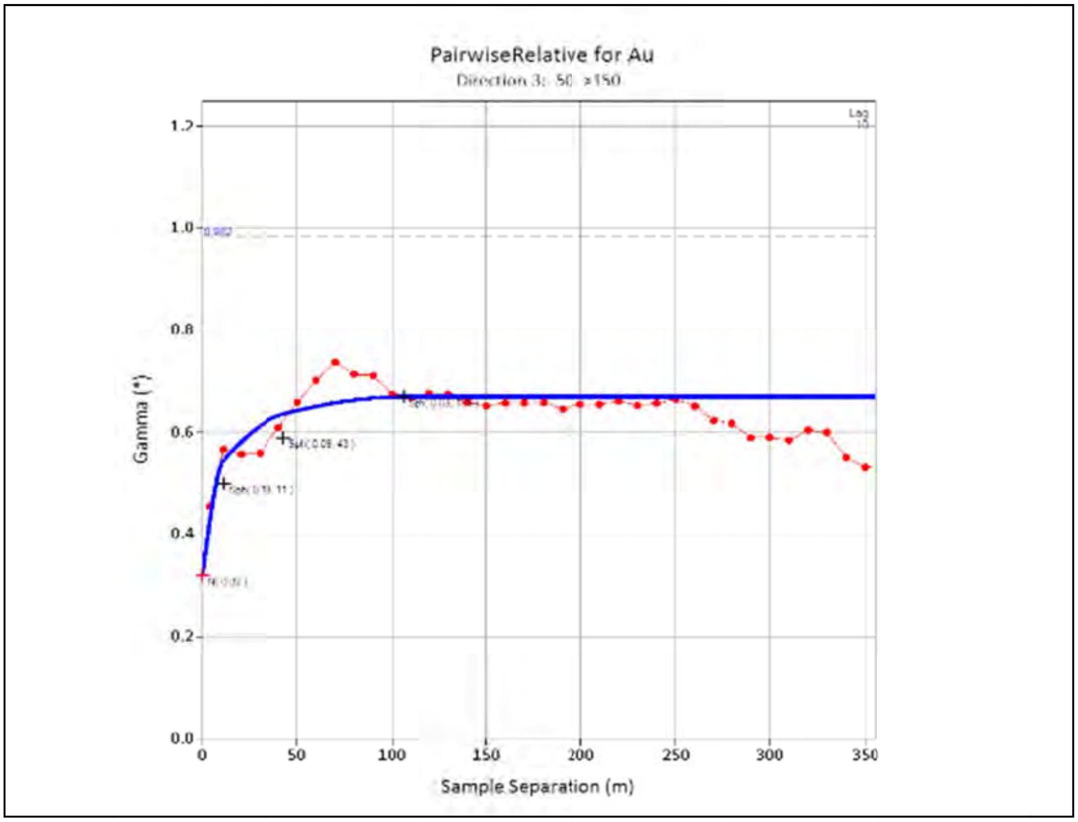
Source: AVM (2016)

Figure 14.17: Variogram – Direction 2



Source: AVM (2016)

Figure 14.18: Variogram – Direction 3



Source: AVM (2016)

14.8 Eagle Zone - Estimation Method and Mineral Resource Classification

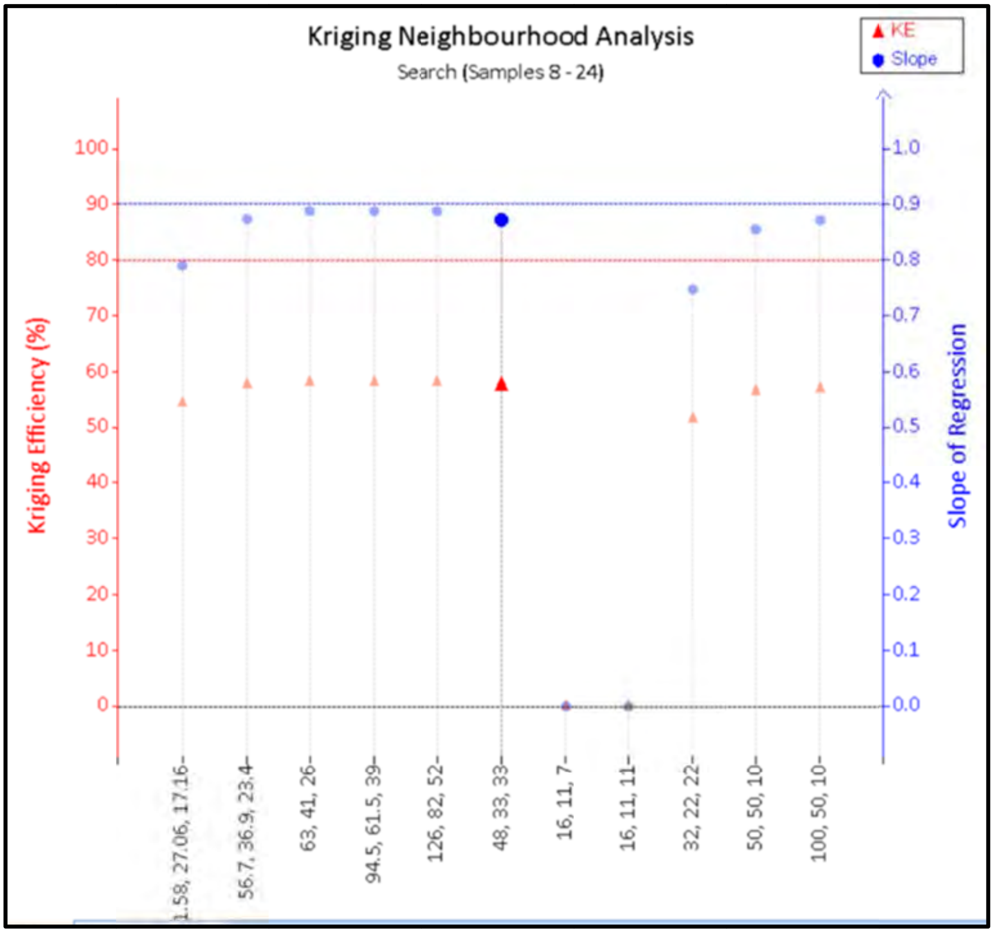
Drill hole statistics indicate that 75% of the data population is below 0.81 g/t and any higher grade population is not localized but scattered. The nugget effect reflects the short scale variability of mineralization. In this scenario, the reliability of the estimate is better with higher numbers of composites used in the estimation of blocks. The kriging parameters, block size and search strategy were optimized to get an un-biased estimate using QKNA (Qualitative Kriging Neighbourhood Analysis). The histogram of composite grades is highly positive skewed with a skewness of 7.12, and a Kurtosis of 81.339. The non-linear estimation methods, such as multiple indicator kriging, were not attempted due to a lesser number of high-grade samples for higher grade bins used in multiple indicator kriging (MIK). A total of 55 drill intervals with extreme high-grade values greater than 16 g/t were capped to avoid an undue influence of those scattered values. Table 14.7 displays the search parameters used for the resource estimation. The ellipsoidal search volume (SVOL) is reflecting the assumed preferential directions of continuity along strike and down dip noted in the variography. Only blocks within the mineralized wireframe were estimated and only the relevant domain composites were used in estimation.

To preserve local grade variations, a search neighbourhood strategy with three search ellipse (SVOL) volumes was used.

A 48 X 33 X 33 m search distance was used and resulted in optimal KE and slope (Figure 14.19), and an improvement over prior estimation parameters. An additional search criterion, referred to as the Octant search, was introduced, in which each search is divided into eight octants with minimum and maximum sample criteria. This additional measure takes into account clustering effects and improves the kriging performance.

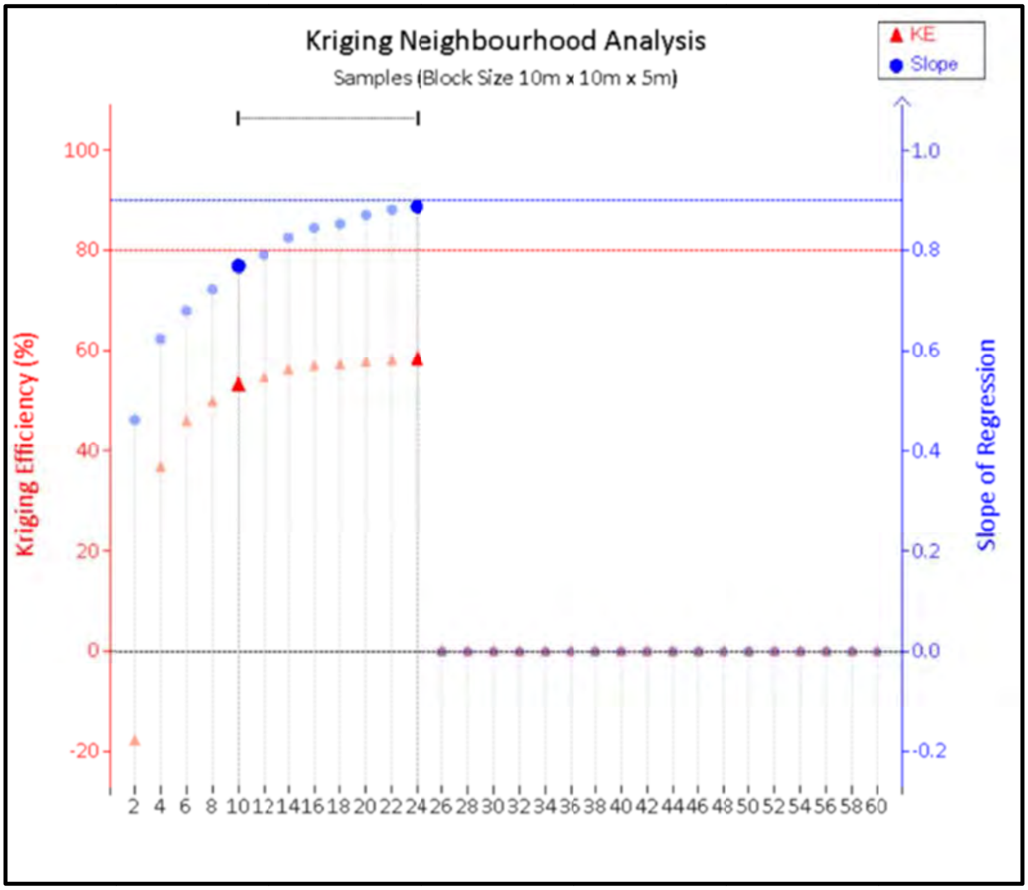
The minimum and maximum number of samples in the search ellipse is justified by the optimal KE and slope curves developed by KNA (Figure 14.19). Only blocks not estimated with the first set of parameters were estimated with a subsequent expanded search. For the first SVOL, a minimum of eight composites was required, with a maximum of three from any given hole. This forces the estimation to use a minimum of three drill holes to estimate any block in the first SVOL estimation pass. For the second search, a minimum of seven composites, from a minimum of three drill holes, were used. For the third search, a minimum of five composites, from a minimum of two drill holes, were used, with a maximum of three from one drill hole. The interpolation methodology and search neighbourhood strategy were selected subsequent to testing multiple scenarios, and were intended to preserve the variation of observed grades. Table 14.7 provides the search parameters used.

Figure 14.19: KNA on Search Distance



Source: AVM (2016)

Figure 14.20: KNA on Minimum and Maximum Number of Samples used in Search in Relation to the Selected 10 X10 X 5 m Block Size



Source: AVM (2016)

Table 14.7: Search Parameters

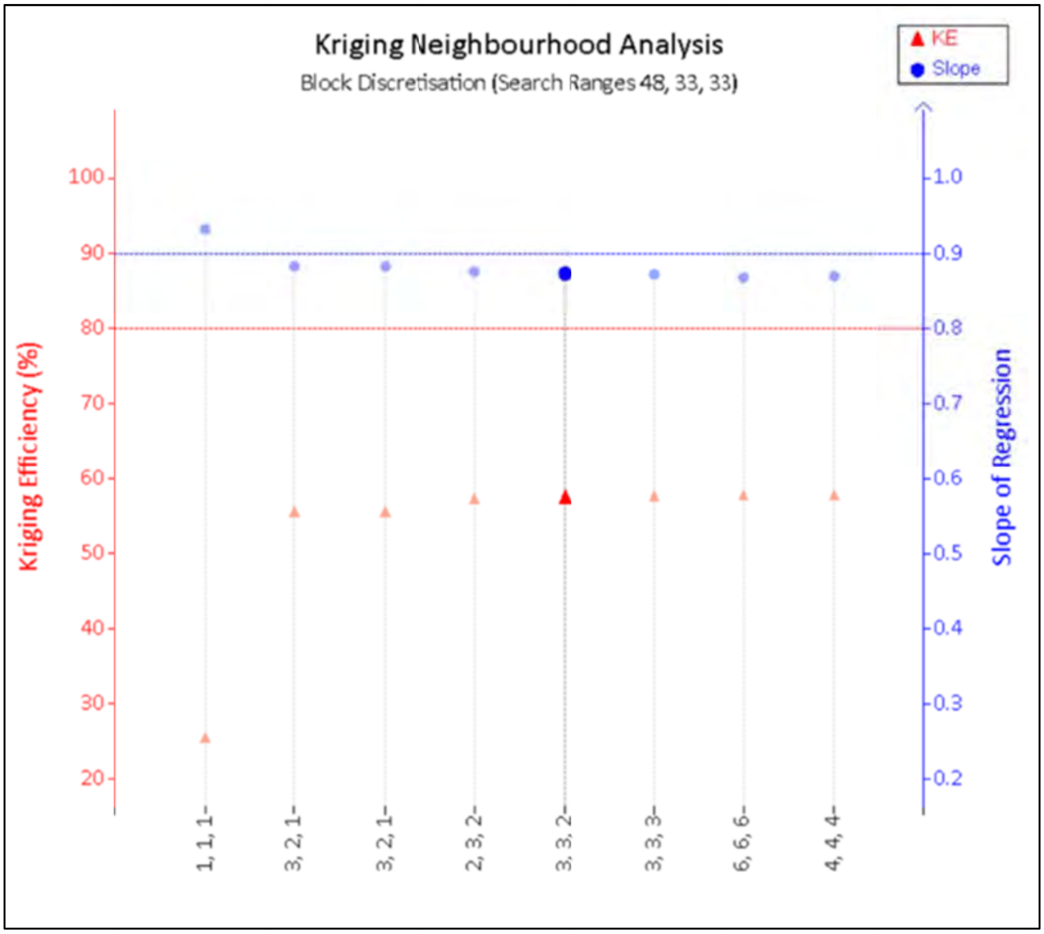
Parameters	Unit	Values
SREFNUM		1
SMETHOD		2
SDIST1	m	48
SDIST2	m	33
SDIST3	m	33
SANGLE1	degrees	60
SANGLE2	degrees	-140
SANGLE3	degrees	-150
SAXIS1		3
SAXIS2		2
SAXIS3		3
MINNUM1	No.	8
MAXNUM1	No.	50
SVOLFAC2	No.	2
MINNUM2	No.	7
MAXNUM2	No.	50
SVOLFAC3	No.	10
MINNUM3	No.	5
MAXNUM3	No.	50
OCTMETH	No.	1
MINOCT	No.	4
MINPEROC	No.	3
MAXPEROC	No.	20
MAXKEY	No.	3

Source: AVM (2016)

Block discretization was also used to simulate the block by a 3D array of points, distributed regularly within the block. It is used to avoid representing blocks just by a single central point of the block.

Figure 14.21 shows the justification for block discretization, with three points in the X-direction, three points in the Y-direction, and two points in the Z-direction.

Figure 14.21: KNA on Block Discretization



Source: AVM (2016)

Block model grades were estimated by ordinary kriging and as well by inverse distance squared (ID2) estimating.

Table 14.8: Datamine Estimation Parameter File

	Ordinary Kriging	Inverse Distance	F Value	Lag Value
VALUE_IN	Au	Au	Au	Au
VALUE_OU	Au_OK	Au_ID	FVALUE	LAGR
MINDIS	MINDIS			
NUMSAM_F	NSAMP			
SVOL_F	SVOL			
VAR_F	KV		0	0
SREFNUM	1	1	1	1
IMETHOD	3	2	101	102
POWER	-	2	0	0
ADDCON	0	0	0	0
VREFNUM	1	1	1	1
KRIGNEGW	1	1	1	1
KRIGVARS	1	1	1	1

Source: AVM (2016)

Multiple factors were used for the classification of resources based on mathematical rules. This included regression slope (adequacy of block size and search neighbourhood parameters), kriging variance (quality of the estimate), and number of composites used to estimate blocks. In the first run of the process, all blocks were coded as class=3 (Inferred), in the next run, blocks were coded for Indicated and Measured, if the combination of all the criteria given in Table 14.9 were satisfied. Figures 14.22 and 14.23 illustrate these factors for the distribution of Measured and Indicated blocks, respectively. Figure 14.24 illustrates the combined Measured and Indicated blocks, and Figure 14.25 shows the Inferred blocks.

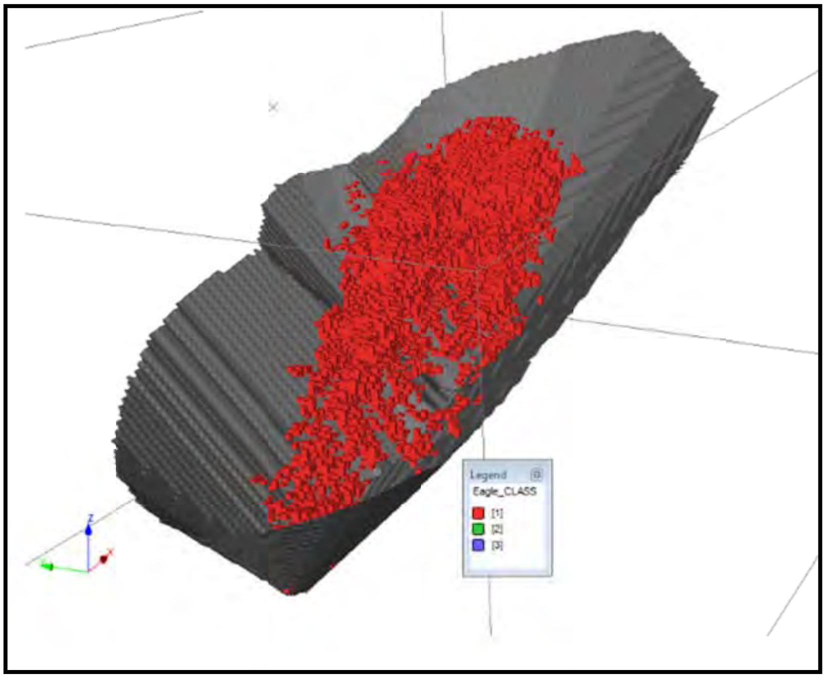
Table 14.9: Search Parameters – Measured and Indicated Classifications

Criteria	Measured	Indicated
	(Class=1)	(Class=2)
Regression Slope	>=0.88	>=0.5
No of Composites	>=25	>=10
Kriging variance	<= 0.15	<= 0.3

Source: AVM (2016)

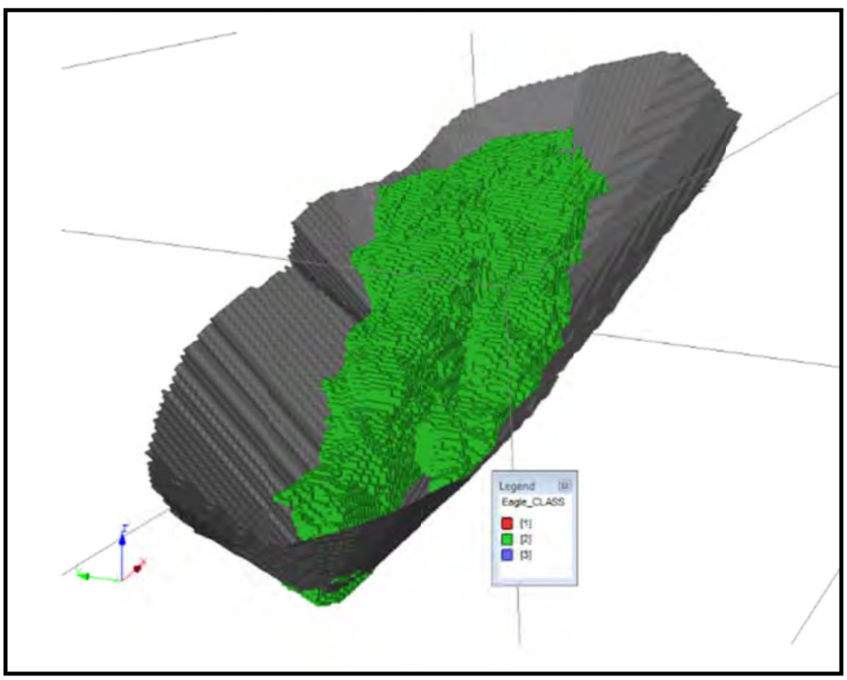
The classified block model was reviewed in sections in relation to the drill density before finalizing the classification.

Figure 14.22: Measured Blocks in 3D



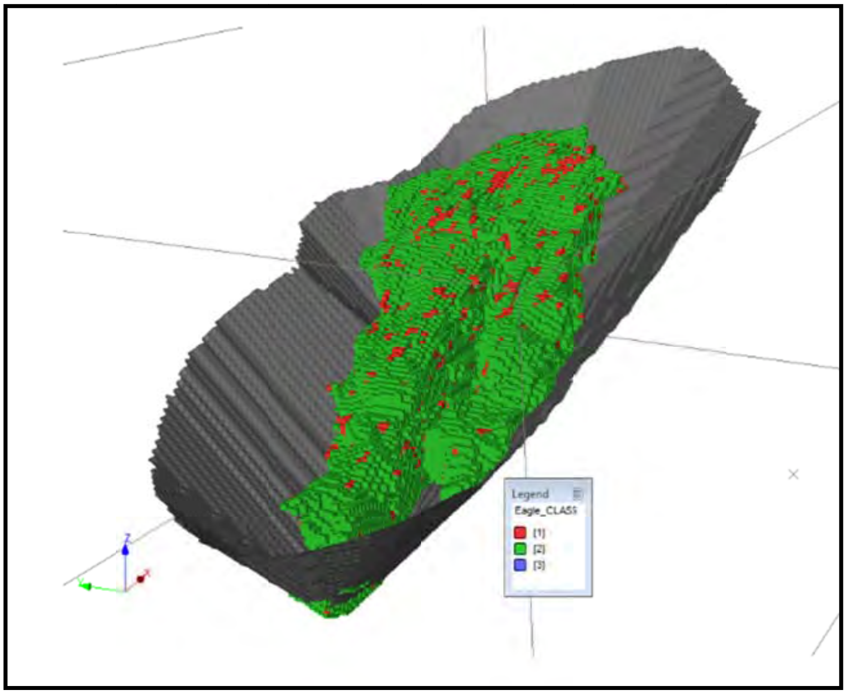
Source: AVM (2016)

Figure 14.23: Indicated blocks in 3D



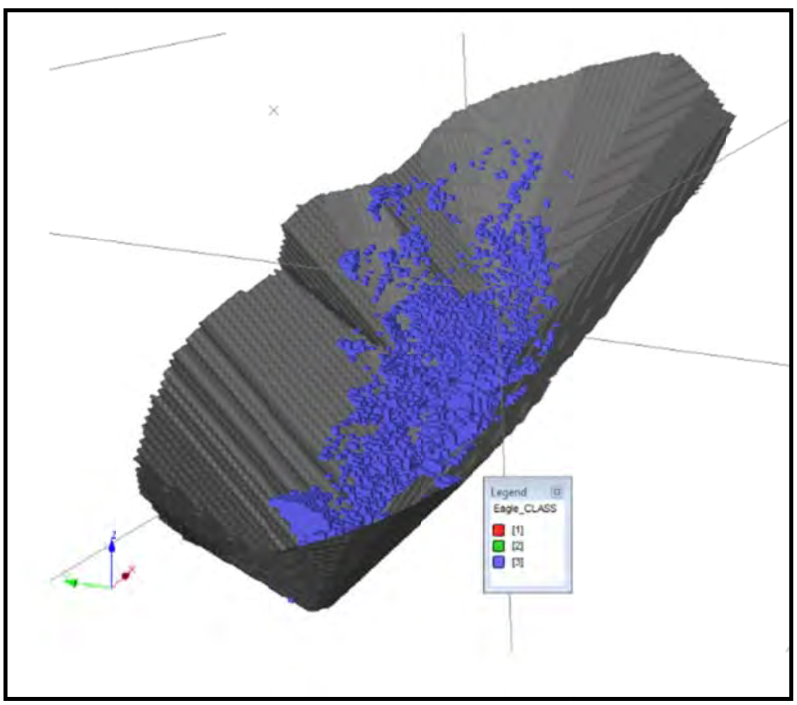
Source: AVM (2016)

Figure 14.24: Measured and Indicated Blocks in 3D



Source: AVM (2016)

Figure 14.25: Inferred blocks in 3D



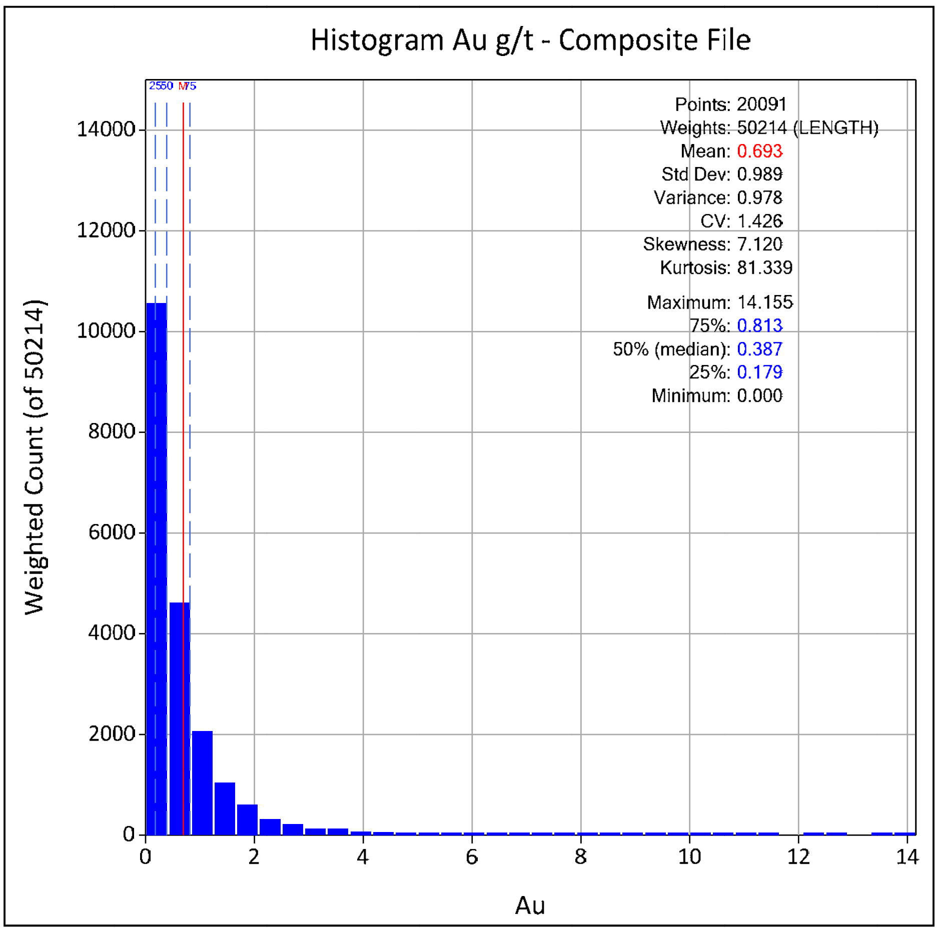
Source: AVM (2016)

14.9 Eagle Zone - Model Validation

The following validation checks were carried out to validate the model:

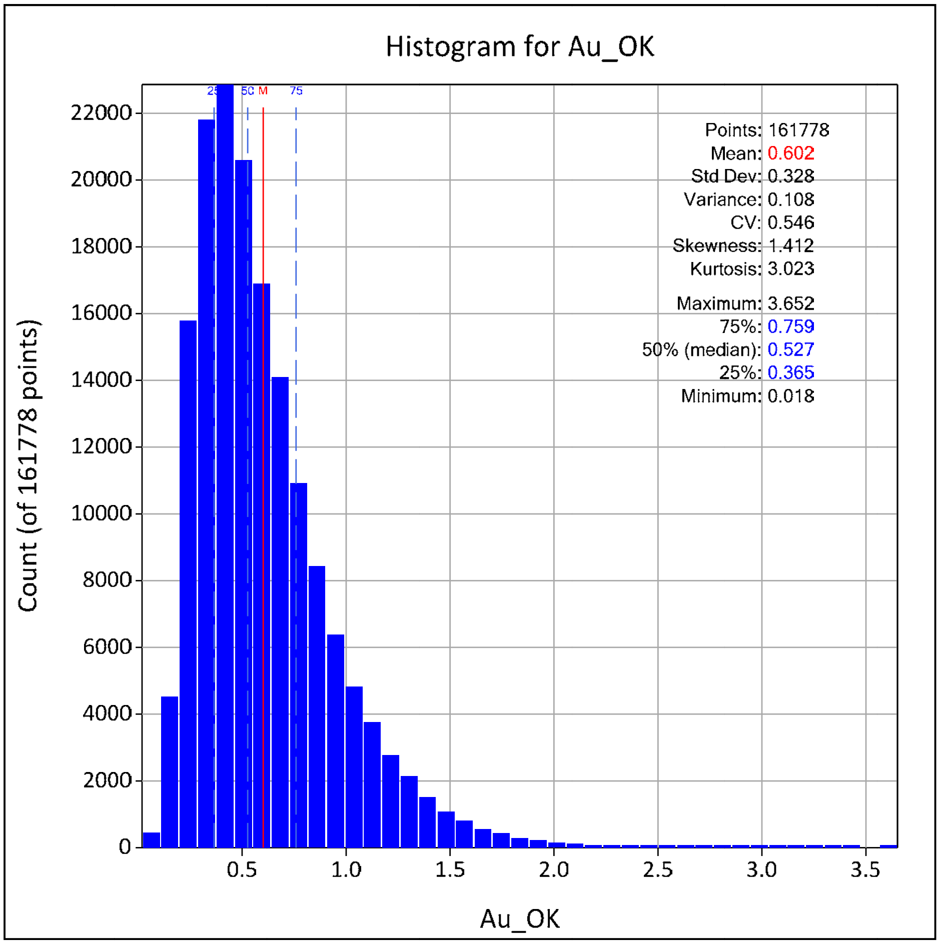
- Visual checks in section-view with drill hole composites. A fair correlation was observed in the model grade distribution versus the composites; and
- A classical statistical comparison was done between the block grade estimated by ordinary kriging versus the composite grade. The Au g/t average grade of composite samples was 0.693 Au g/t, and the Au g/t average grade of the block model was 0.602 Au g/t (Figures 14.26 and 14.27).

Figure 14.26: Histogram of the Composite File



Source: AVM (2016)

Figure 14.27:Histogram of Block Model estimated by Ordinary Kriging



Source: AVM (2016)

The quality of the estimate depends on various factors, including quality of data, block size and drill density. Krige (1996) presents a practical analysis of the effects of spatial continuity and the available data within the search ellipse as it affects measures of conditional bias. The two parameters which were investigated for validating grade estimation were regression slope (R) and kriging variance, which can also be used to calibrate the confidence in block estimates. The equations used for these estimation are as follows:

$KE = (BV-KV)/BV$

$R = BV-KV+ |\mu|$

$BV – KV + |2\mu|$

Where:

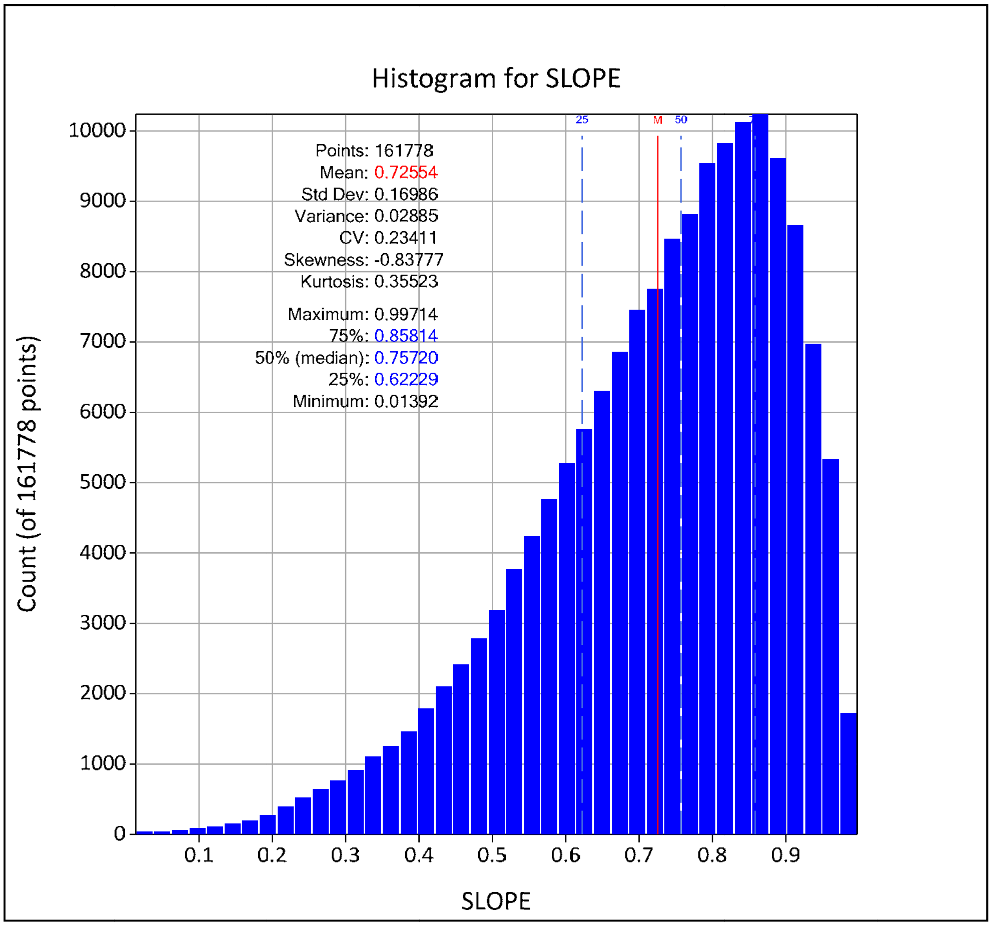
- BV = theoretical variance of blocks within domain;
- KV = variance between kriged grade and true (unknown) grade, i.e. kriging variance;
- R= Regression Slope; and
- μ = LaGrange multiplier.

A perfect estimation would give values of KV = 0, KE = 100% and R=1.

Kriging Variance (KV) and regression slope (Slope) were calculated for each block in the block model. Figures 14.32 and 14.33 indicate histograms of these parameters. The following observation can be summarized from these histograms:

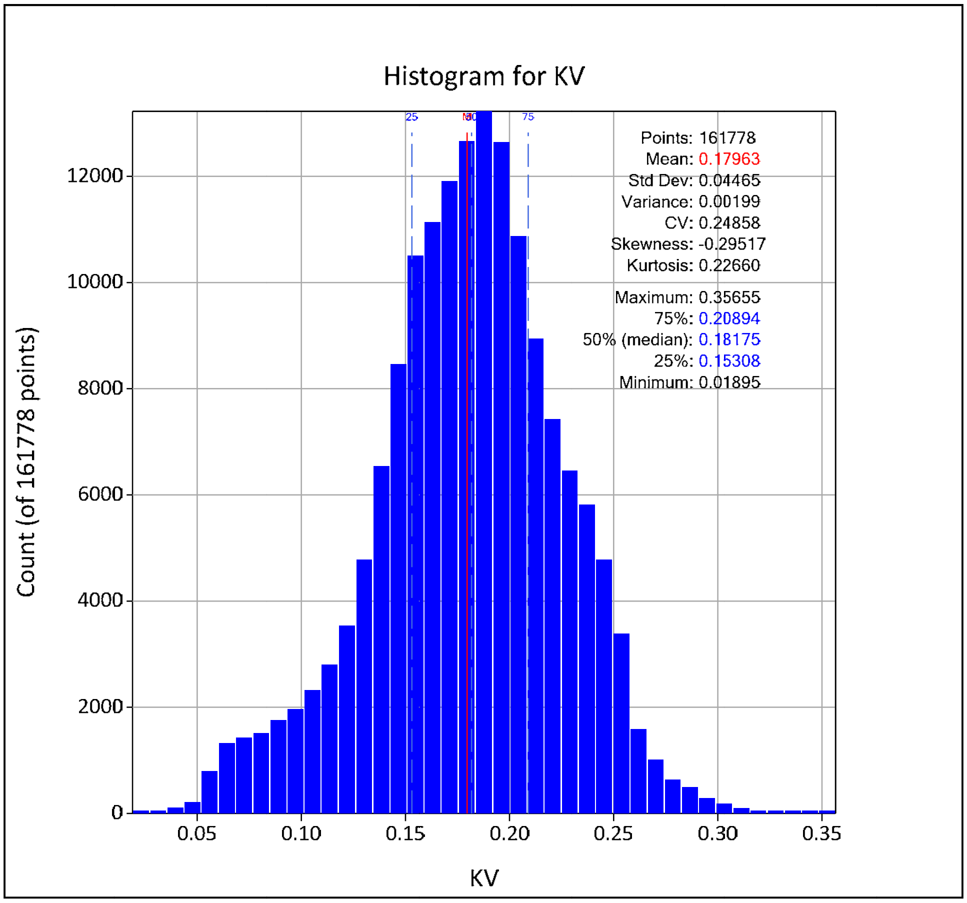
- The mean kriging variance was 0.18 and the majority of the blocks had a KV close to 0.2, this indicated a reasonable estimate;
- The mean regression slope for the mineralized zone was 0.726 with the majority of the block estimates above 0.85; indicating that the majority of blocks were well supported by the data.

Figure 14.28: Histogram on Regression Slope



Source: AVM (2016)

Figure 14.29: Histogram on Regression Kriging Variance



Source: AVM (2016)

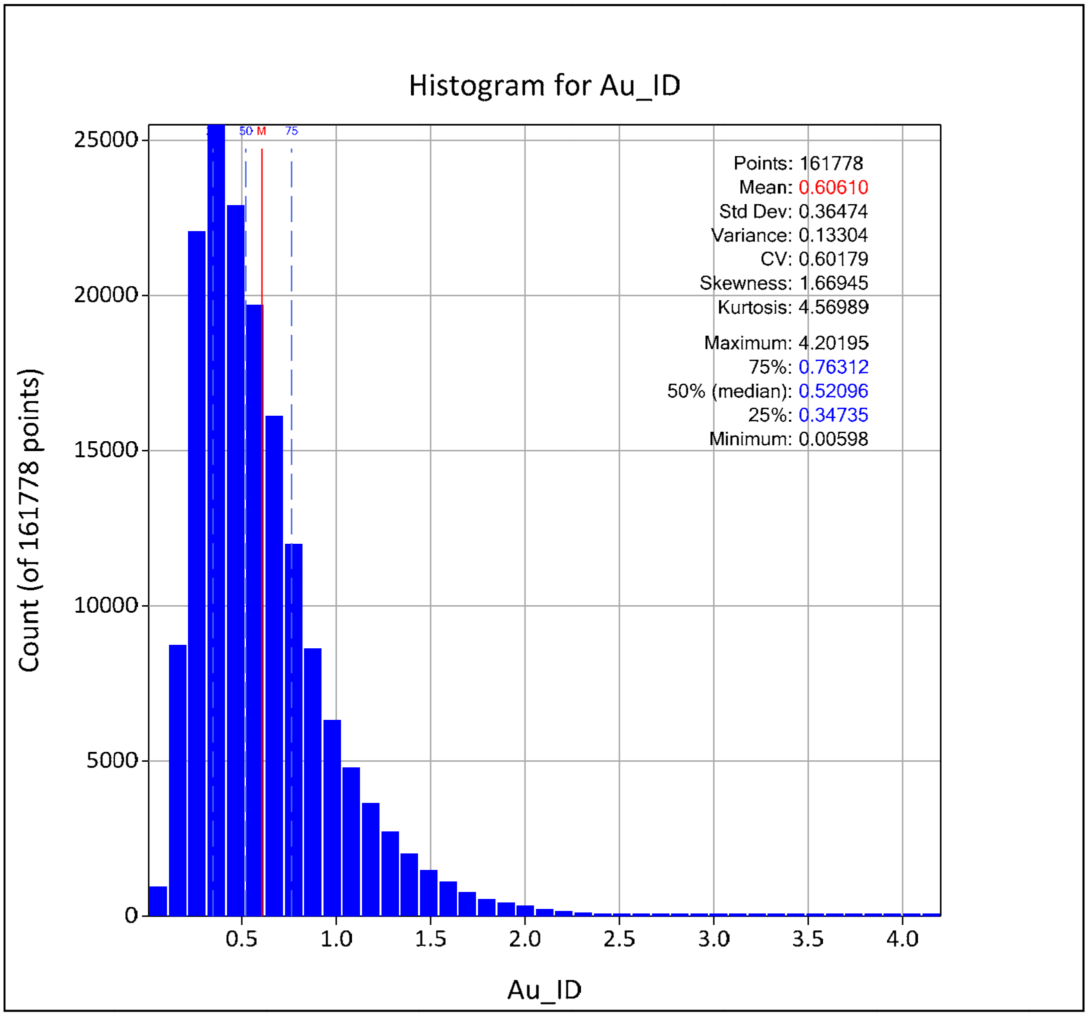
The block model was also estimated with ID2. Table 14.10 compares the global statistics between ID2 and ordinary kriging. Figure 14.34 illustrates the histogram of the block model with these two estimation methods.

Table 14.10: ID2 vs Kriging Comparison – Average Au grade

Au_ID2	Au_Krig	Difference (%)
0.606	0.602	0.66

Source: AVM (2016)

Figure 14.30: Histogram of Block Model Estimated by Inverse Distance



Source: AVM (2016)

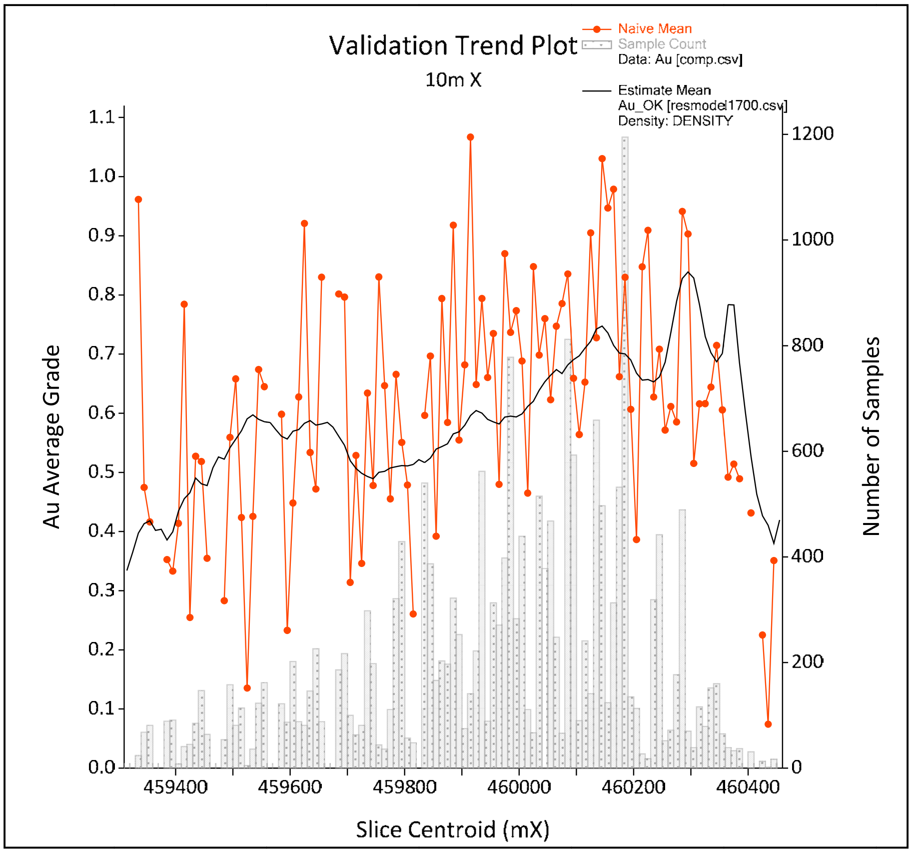
Kriging and ID2 resulted in similar histogram shapes and similar mean grades; 0.602 and 0.606, respectively.

The histogram shape of composites indicated that there were a higher number of lower grade samples (less than 1.0 g/t) present as compared to the higher grade sample (greater than 1.0 g/t). The drop in mean grade from 0.693 for composites to 0.602 for kriging was due to scattered higher grade samples, which were not influencing the OK or ID2 estimation mean grade compared to the un-estimated raw composite grades.

All the above observations indicated a reasonable quality of the current estimate, and provided confidence in the used estimation parameters.

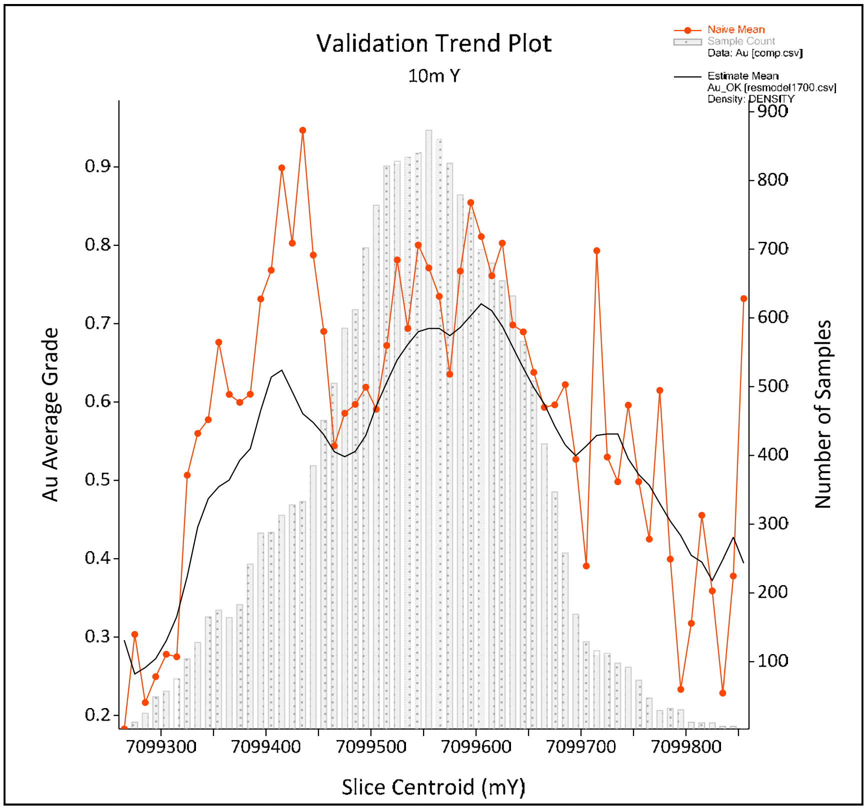
In addition to the stated validations, Trend Plots (swath plots) were generated in the X, Y and Z directions to compare block grades with composite grades (Figures 14.31 through 14.33), which show good correlation between estimated values (smooth curves and the sample data (orange colored line). All the validation exercises indicated the block model grade to be reasonably comparable and acceptable.

Figure 14.31: Trend Plot X on Au g/t



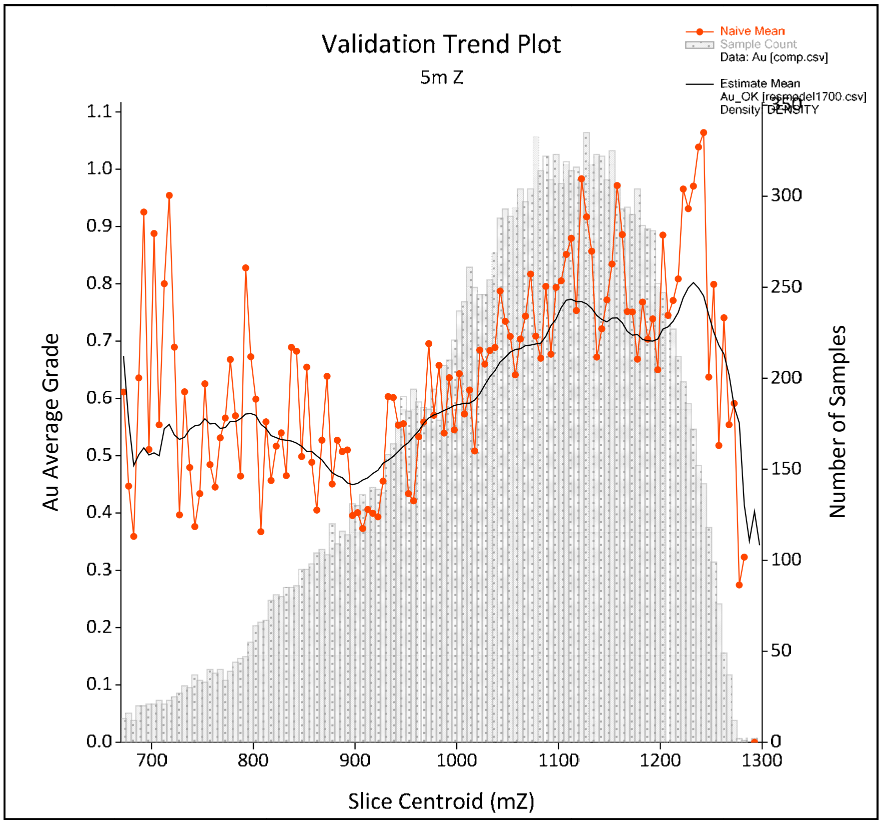
Source : AVM (2016)

Figure 14.32: Trend Plot Y on Au g/t



Source : AVM (2016)

Figure 14.33: Trend Plot Z on Au g/t



Source : AVM (2016)

14.10 Eagle Zone - Mineral Resource Statement

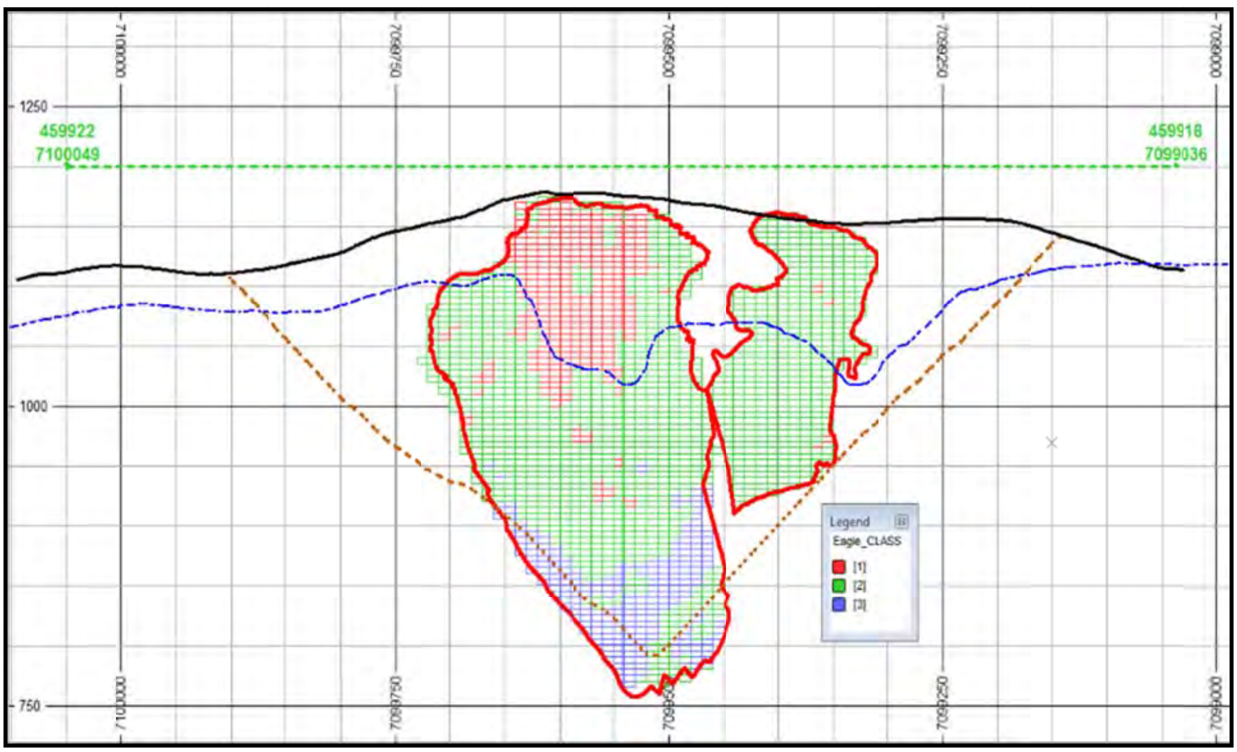
The current Eagle Mineral Resources are reported as in-pit resources at a cut-off grade of 0.15 g/t Au. The cut-off is supported by the following parameters for a pit shell.

- Mining cost of C$ 2.00/t;
- Processing cost of C$5.00/t;
- G&A cost of C$1.00/t;
- Au recoveries of 82%, 71% and 75% for oxidized rock, unaltered rock, and sericitically altered granodiorite, respectively(observed lab recoveries); and
- Au price of US$1,700, at an exchange rate of US$:C$ 0.75.

The resulting calculated cut-off grade for the highest-recovery material is 0.12 g/t Au. Mining is anticipated to be conducted to a 0.20 g/t Au cut-off grade. A gold price of US$1,700 is justified for use in defining a resource pit shell, based on the fact that gold prices during the period from mid-2011 to early 2013 varied from US$1,600 to US$1,800/oz, and it is reasonable to assume that gold prices can achieve US$1,700 in the future.

The Mineral Resources are confined within an optimistic pit shell based on the same parameters used for the cut-off grade and 45°slopes; the pit shell optimized on Measured + Indicated + Inferred Resources. The purpose of the pit shell, and the optimistic parameters used to create that shell, is to determine the material that has reasonable prospects for economic extraction. Figure 14.34 is a cross-section showing the resource pit shell with blocks coded by resource classification. Table 14.11 presents the current Mineral Resources for the Eagle Zone.

Figure 14.34: Cross-Section with a Gold Price of US$1,700 Resource Pit and Blocks Coded by Resource Classification (Measured in Red, Indicated in Green and Inferred in Blue).



Source : AVM (2016)

Table 14.11: Eagle Zone Current In-Pit Mineral Resources at 0.15 g/t Au cut-off grade

	Cut-off grade (g/t Au)	Tonnes (Mt)	Au (g/t)	Ounces Au (koz)
Measured	0.15	29.40	0.805	761
Indicated	0.15	151.31	0.590	2,870
Meas. + Ind.	0.15	180.72	0.625	3,631
Inferred	0.15	17.43	0.492	276

Notes:

1. Mineral Resources are reported in accordance with Canadian Securities Administrators (CSA) National Instrument 43-101 (NI 43-101) and have been estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. CIM definitions were followed for Mineral Resources;
2. In-Pit Mineral Resources are estimated at a cut-off of 0.15 g/t, with mining and processing costs and pit parameters as per the text of this report;
3. Metal prices used for this estimate were US$1,700/oz Au, at an 0.75 exchange rate;
4. High-grade caps were applied as per the text of this report;
5. Specific gravity (bulk density) was assigned for each block based on measurements taken from core specimens;
6. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding;
7. Resources are reported on a 100% basis for Victoria Gold controlled lands; and
8. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues, although the author is not aware of any such issues.

Source: AVM (2016)

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resource estimates do not account for mineability, selectivity, mining loss and dilution. These Mineral Resource estimates include Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is also no certainty that these Inferred Mineral Resources will be converted to Measured and Indicated categories through further drilling, or converted into Mineral Reserves once economic considerations are applied.

14.11 Eagle Zone - Mineral Resource Sensitivity

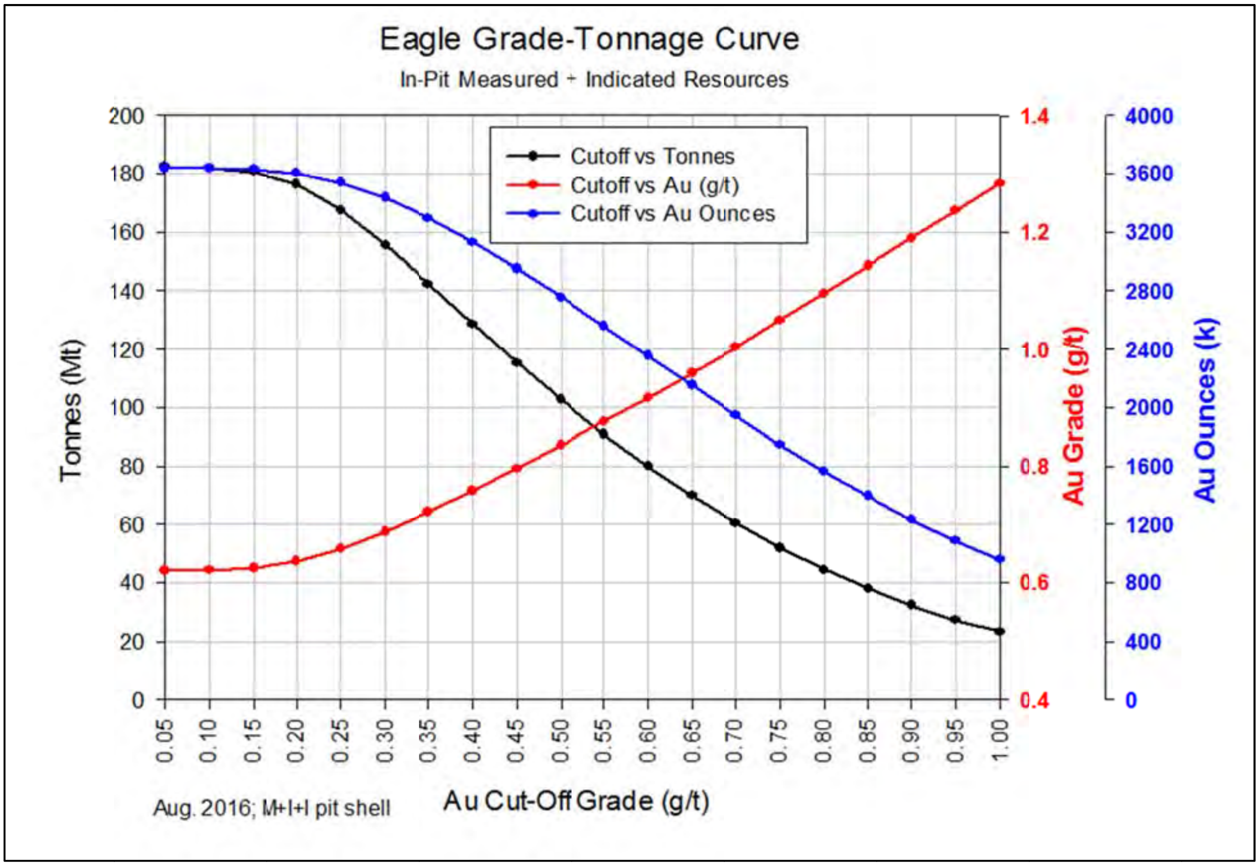
The Eagle Zone Measured + Indicated Mineral Resources at COGs between 0.05 to 1.0 g/t Au are shown in Table 14.12. Figure 14.35 shows graphically the Grade -Tonnage curves. Table 14.13 provides a break-down of the Mineral Resources by Metallurgical Type at the 0.15 g/t Au cutoff.

Table 14.12: Eagle Zone Grade Tonnage Sensitivity Data for Measured + Indicated Resources at various COGs

Measured + Indicated			
Cut-Off Au g/t	Tonnes (Million Tonnes)	Au (g/t)	Ounces (Million oz)
0.05	182.46	0.62	3.637
0.1	182.08	0.621	3.636
0.15	180.72	0.625	3.631
0.2	176.3	0.636	3.605
0.25	167.5	0.658	3.541
0.3	155.59	0.687	3.436
0.35	142.33	0.721	3.297
0.4	128.66	0.757	3.132
0.45	115.38	0.795	2.951
0.5	102.6	0.835	2.756
0.55	90.75	0.876	2.556
0.6	79.94	0.917	2.356
0.65	69.82	0.959	2.153
0.7	60.29	1.004	1.946
0.75	51.75	1.05	1.747
0.8	44.35	1.096	1.563
0.85	37.92	1.142	1.392
0.9	32.18	1.19	1.231
0.95	27.31	1.237	1.086
1	23.24	1.283	0.959

Source : AVM (2016)

Figure 14.35: Grade Tonnage Curves for the Eagle Deposit – Measured + Indicated Resources



Source: AVM (2016)

Table 14.13: Eagle In-Pit Mineral Resource Breakdown by Metallurgical Type and Classification at a 0.15 g/t Au Cut-Off

Cut-Off (Au g/t)	Type	Met Type	Measured			Indicated			Measured+Indicated			Inferred		
			Tonnes (M)	Au (g/t)	Ounces (k)	Tonnes (M)	Au (g/t)	Ounces (k)	Tonnes (M)	Au (g/t)	Ounces (k)	Tonnes (M)	Au (g/t)	Ounces (k)
0.15	1	A	13.1	0.926	390	27.08	0.662	576	40.18	0.748	967	0.67	0.432	9
0.15	2	C	1.41	0.676	31	13.21	0.554	235	14.62	0.566	266	0.82	0.514	14
0.15	3	B	11.42	0.701	257	103.18	0.579	1922	114.6	0.591	2179	15.73	0.494	250
0.15	4	E	2.96	0.77	73	4.96	0.576	92	7.91	0.649	165	0.03	0.49	0
0.15	6		0.32	0.499	5	1.68	0.442	24	2	0.451	29	0.07	0.509	1
0.15	10		0.19	0.696	4	1.21	0.54	21	1.4	0.561	25	0.11	0.37	1

Type	Met Type	Description
1	A	Oxidized Granodiorite
2	C	Altered Granodiorite
3	B	Unaltered Graodiorite
4	E	Oxide Metasediments
6		Unaltered Metasediments
10		Overburden/Weathered

14.12 Olive Zone - Drill Hole Database

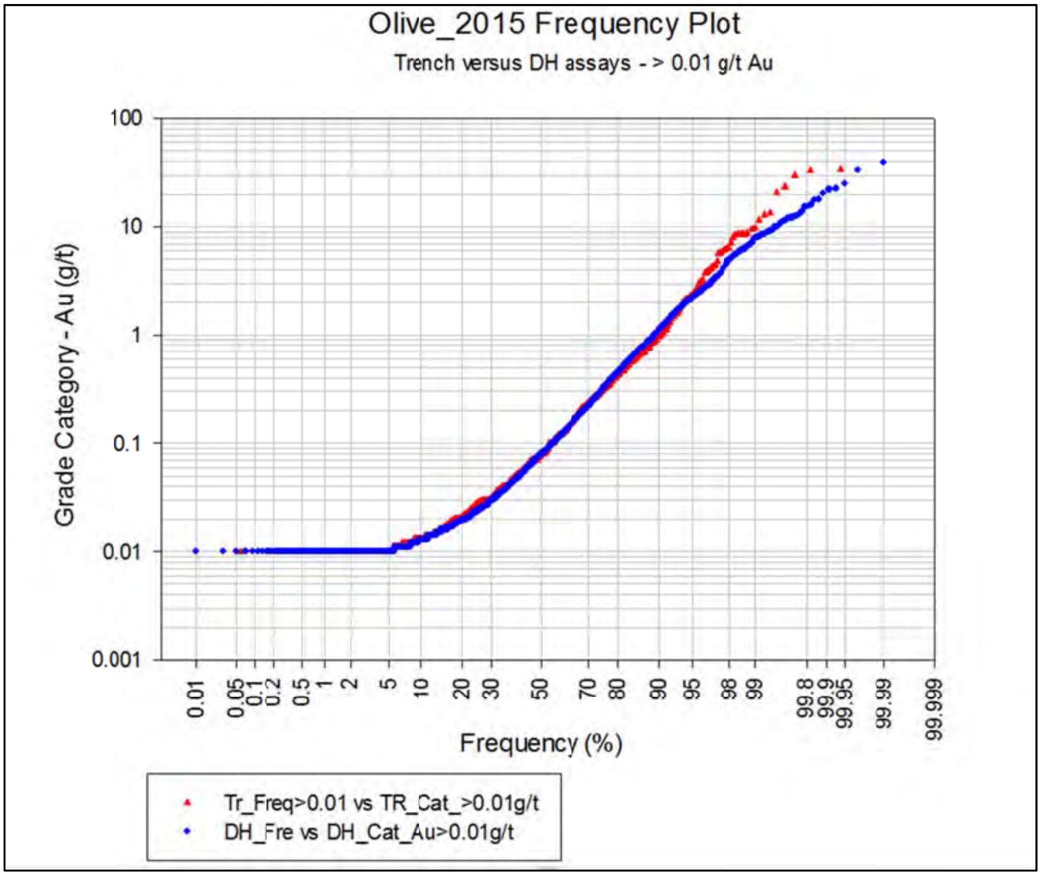
The drill hole database was received in Excel format. The digital database included the following file information:

- Collar – Drill hole name, easting, northing, elevation, total depth, type, purpose;
- Survey – Drill hole name, depth of survey, azimuth, dip;
- Assays – Drill hole name, from, to, sample id, certificate number, Au, Ag and other elements from the ICP analysis;
- Geology – Drill hole name, from, to, sample ID, degree of oxidation, sericitization, chloritization and silicification, vein type and thickness;
- Lithology – Drill hole name, from, to, rock type (such as overburden, Granodiorite, altered Granodiorite, hornfels, quartzite, and fault zone); and
- Recovery, RQD - Drill hole name, from, to, recovery %, RQD %.

In addition to this data, digital copies of surface geological maps, 2 m digitized topo, bulk density file data (in Excel format), and QA/QC files were supplied.

A database of 291 holes was provided including 221 core, nine RC drill holes and 61 trenches. Geology information from all the holes was used for geological interpretation and geological modelling. Trench data was considered to be acceptable for grade estimation; the data distribution curves were all nearly identical, and only diverged a bit above about 5 g/t Au, which constituted less than 5% of the trench data (see Figure 14.36). Trenches were used as sub-horizontal pseudo-drill holes. Table 10.2 in Section 10 – Drilling provides a breakdown of drilling and trenching details by type and year completed for Olive.

Figure 14.36: CF Plot of Au Grades, Trench and DH



Source : AVM (2016)

14.13 Olive Zone - Exploratory Data Analysis

EDA was carried out for Olive in the same manner as for Eagle (See Section 14.2).

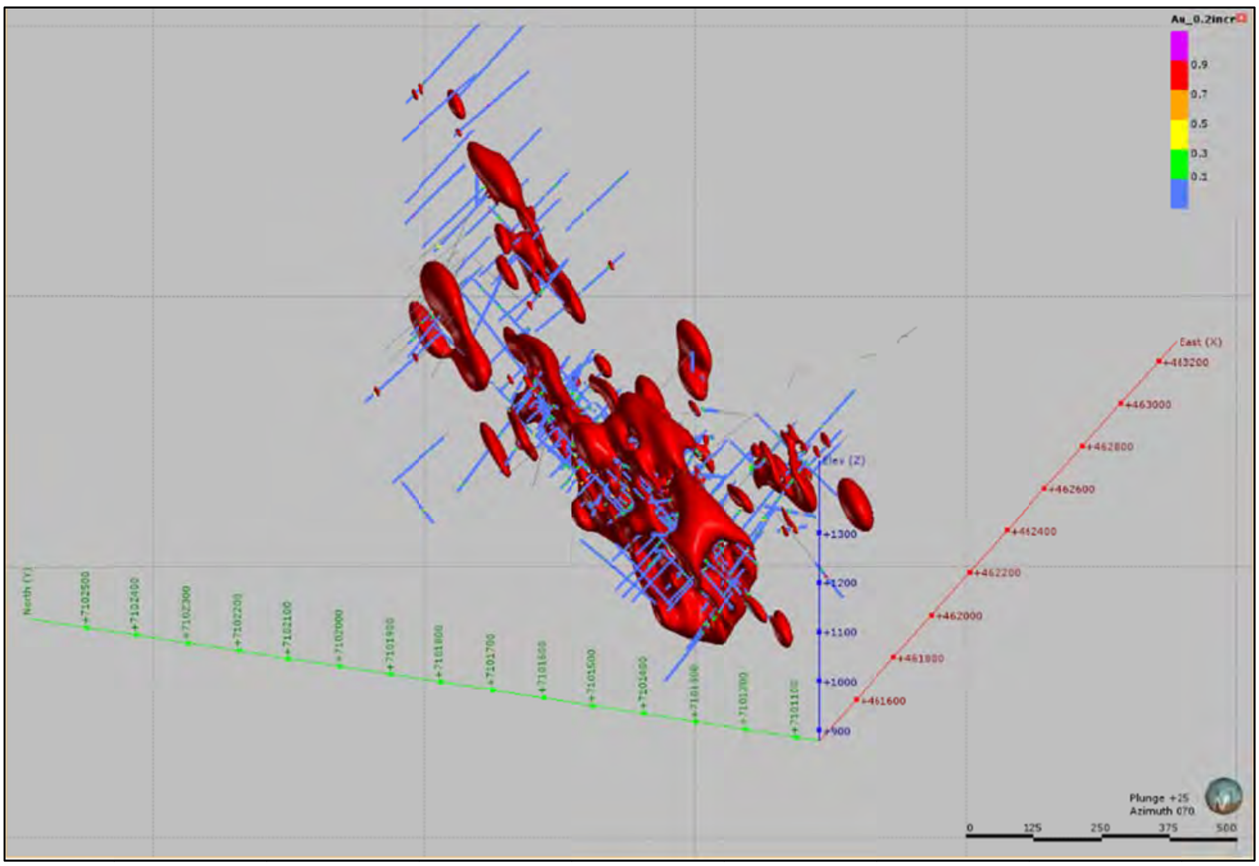
RC drilling accounted for eight holes and 792 m of total drilling, or 2% of the total meterage, and there was both insufficient data to run core-versus-RC analysis and an insignificant amount of data to materially affect the database. The assumption was that RC drilling for Olive was not biased with respect to core, based on the analysis of RC core versus for Eagle.

A visual inspection of vein density and its correlation with gold assays was carried out, in an attempt to determine if the density of the veining or the geological modelling of the vein zones could aid in the deposit modelling. It was concluded that, while there was a general direct relationship of sulphide and quartz vein density to the general location of gold mineralization, the gold grades were geologically not directly related by a 1:1 ratio to the vein intensity, which is logged as vein density (cm/m of sample). The resource modelling is therefore based on the modelling of gold grades and not on lithology, structure, or quartz veining. At Olive, approximately 98% of the mineralization is hosted in granodioritic intrusive rock types.

Leapfrog software was used to define the extent of the mineralization, as a mineralized shell, based on drill hole composite data (Figure 14.37). Unlike Eagle, the mineralization raw data, primarily at 1.5 m intervals, was a mix of higher grades and lower grades in a shotgun pattern, such that direct grade indicated hole-to-hole continuity was not apparent, and shells were not very meaningful.

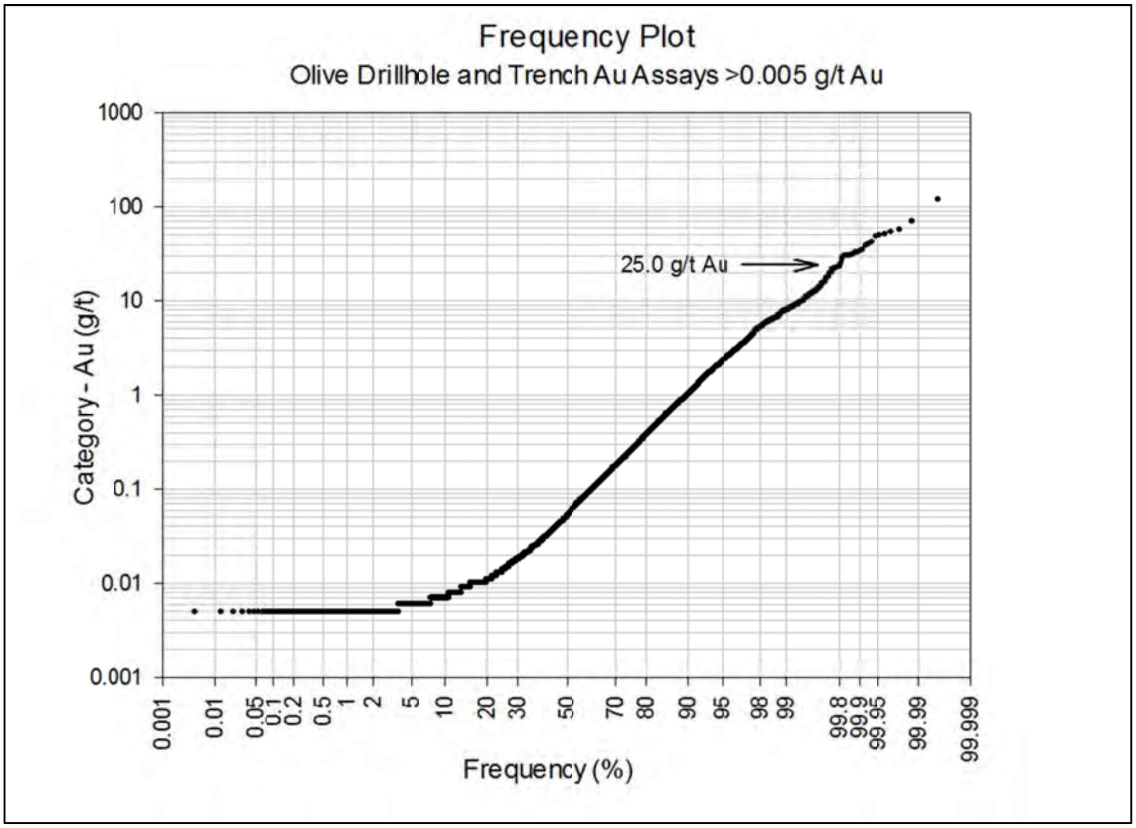
Compositing to 5 m, and snapping to composites greater than 0.10 g/t Au, defined a linear zone of mineralization with hole-to-hole continuity, and provided boundary limits to capture mineralization. The wireframe shape was exported from Leapfrog to Datamine, and used to select a relevant assay population for grade estimation; a total of 8,216 sampled and analyzed intervals were within the mineralized shell. All "missing" values for Au and Ag were set to zero prior to compositing. Frequency distribution plots were constructed to identify capping thresholds for both gold and silver of 25 g/t and 60 g/t, respectively. Figure 14.38 shows the frequency plot for gold. Values in excess of these were set back to these thresholds prior to compositing.

Figure 14.37: Olive 3D Mineralized Wireframe Solid - Perspective View



Source: AVM (2016)

Figure 14.38: Olive Frequency Plot for Au in Mineralized Shell



Source : AVM (2016)

14.14 Olive Zone - Compositing

Within the grade shell the average sample intervals were 1.5 m; the envisioned vertical mining selectivity was 5 m, and a 2.5 m interval was selected for compositing. Comparative statistics are displayed in Table 14.14.

Table 14.14: Au g/t Comparative Statistics

	Number	Min.	Max.	Mean	Variance	Standard Dev.	CV
All assays	21643	0	118	0.295	3.429	1.852	6.28
Capped assays within mineralized shell	8190	0	25	0.663	4.342	2.084	3.14
2.5 m Composites	4776	0	25	0.638	2.289	1.513	2.37

Source : AVM (2016)

The CV for all assays was high, reflecting the grade variation of the raw data. Capping and segregation within the shell resulted in a significant reduction, and with compositing to 2.5 m, the final CV of 2.37 suggested that linear estimation was acceptable for grade estimation with this population.

14.15 Olive Zone - Geological Model

Leapfrog software was used for implicit geological modelling of drill hole codes for lithology, resulting in a simple 3D model of intrusive (Granodiorite) metasediments with minor overburden cover. Structural controls to mineralization are likely present, and control the overall orientation of the mineralized shell and the veining internal to that shell; however, they were not definable from drilling codes. Post-mineral off-setting faults were not identified. Interpreted structures were therefore useful as exploration tools, but not as limiting features to the mineralization at Olive Zone.

Similar to the work at Eagle Zone, Leapfrog software was used to create oxidation surfaces using Ox codes from the core logging. A base of oxidation surface and base of mixed or transition material surfaces were created as further described in Section 12.2.5, Data Verification.

Rock/Lithologic type and oxidation type wireframes were created in Leapfrog and imported into Datamine to code the block model with geo-metallurgical code as shown in Tables 14.15 and 14.16. These codes were primarily used to assign bulk densities, as discussed in Section 14.17, and for the assignment of a recovery variable used during the pit optimization discussed in section 14.21. A display of the wireframes is shown in Figure 14.37.

Table 14.15: Rock/Lithologic Codes

Code	Type
1	Granodiorite
2	Metasediments
3	Overburden

Source: AVM (2016)

Table 14.16: Oxidation Codes

Code	Type	Recovery (Decimal %)
1	Oxide	0.66
2	Mixed/transition	0.55
3	Sulphide	0.52

Source: AVM (2016)

14.16 Olive Zone- Block Model

A block model was created using Datamine Studio3 with the spatial characteristics as described in Table 14.17.

Table 14.17: Block Model Spatial Characteristics

Direction	Minimum (m)	Maximum (m)	10 m x 10 m x 5 m Blocks: number of	
Easting	461,250	463,250	200	Columns
Northing	7,100,760	7,102,550	179	Rows
Elevation	875	1,400	105	Levels

Source: AVM (2016)

The relevant block model fields for resource estimation and reporting are tabulated in Table 14.18 along with the relevant descriptive report sections. For pit optimization purposes, all model blocks were 10 m X 10 m X 5 m, with no sub-cells or partial blocks. At the topographic interface “percent below topo” is represented by a fractional code varying from 0 (air) to 1 (in place). “Air” blocks were created such that each possible position in the model matrix has a record.

Table 14.18: Block Model Fields & Applicable Report sections

Relevant Block Model Fields		Section
XC	Easting Coordinate of Block Centroid	
YC	Northing Coordinate of Block Centroid	
ZC	Elevation Coordinate of Block Centroid	
MIN	1=inside mineralized shell, 0 otherwise	14.13
OX	1=Oxide,2=Mix/transition,3=Sulphide	14.15
ROCK	1=Granodiorite, 2=Metasediments,3=Overburden	14.15
DEN	Bulk Density	14.17
AUKRG	Au g/t, estimated with ordinary kriging	14.18
AGID2	Ag g/t, estimated with inverse distance squared	14.18
CLASS	1=Measured,2=Indicated,3=Inferred	14.19
REC	Recovery for pit optimization	14.15
TOPO	1=Inplace,0=Air	14.16
TOPO%	Percentage below topography from 0 to 1	14.16

Source: AVM (2016)

14.17 Olive Zone - Bulk Density

Bulk Density measurements for Olive Zone were conducted by Victoria Gold on core samples. Over 770 measurements were taken on core for Granodiorite, 70 for metasediments, 69 for oxidized material, 252 for mixed or transition material, and 582 for sulphide or fresh material.

Victoria Gold's method of measuring weights in water and air has proven at Eagle Zone to be accurate relative to outside analytical lab measurements.

Bulk density mean values by rock/oxidation type (codes listed in Tables 14.19) were assigned for resource tonnage calculation as tabulated on Table 14.19.

Table 14.19: Bulk Density by Rock & Oxidation Code

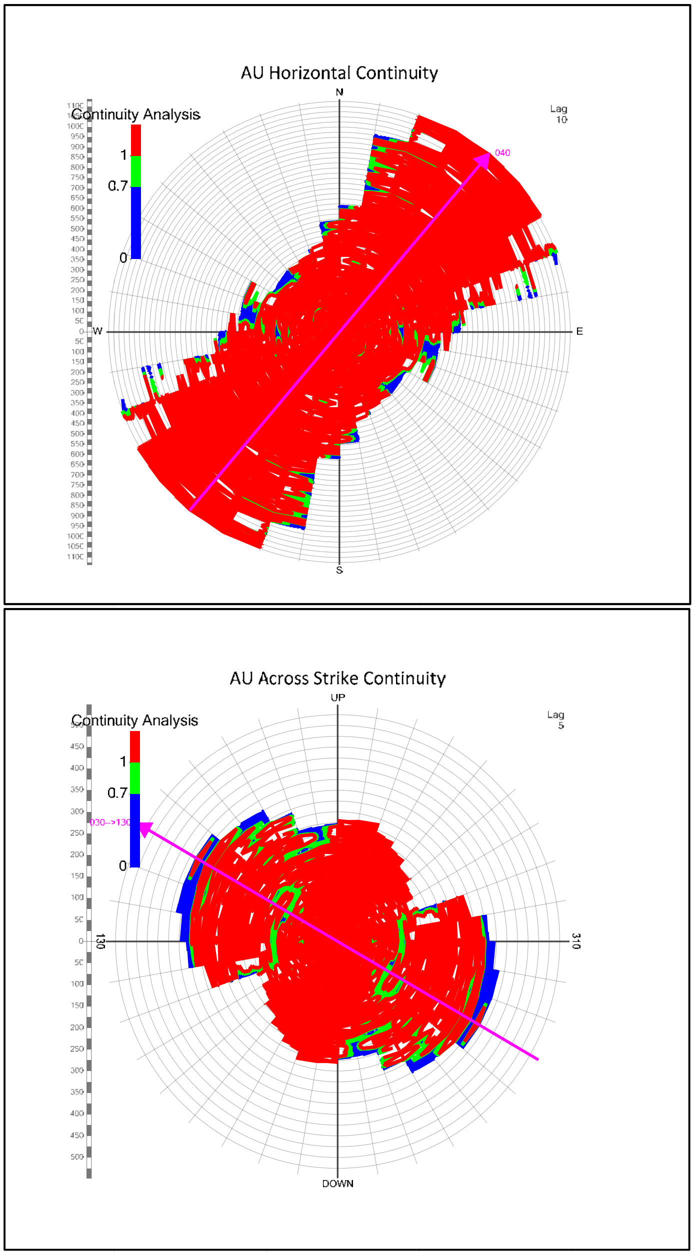
Density		
Rock	Oxidation	Density t/m^3)
3	All	2.00
1	1	2.61
1	2	2.67
1	3	2.68
2	1	2.61
2	2	2.69
2	3	2.70

Source: AVM (2016)

14.18 Zone - Variogram Analysis and Modelling

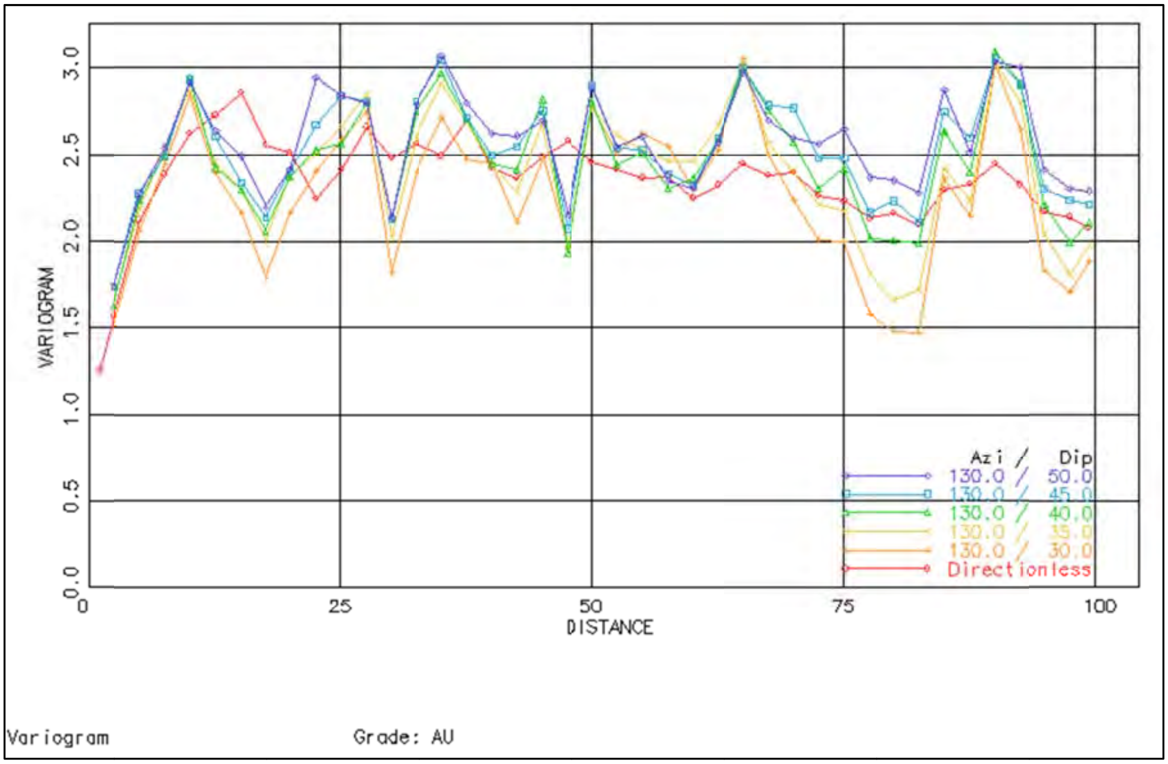
Snowden Supervisor and Datamine software was used to create ordinary and pair-wise relative directional and isotropic variograms on gold 2.5 m composites. Only composites within the mineralized shell were used for variogram analysis. Fan diagrams were created (Figure 14.39), and an analysis of a series of fans in the horizontal-strike, across-strike, vertical, and dips planes was made. Each fan, made up of variogram contours, was analyzed to select the direction of maximum continuity in the fan. While anisotropy can be interpreted from the results, well behaved variograms were difficult to achieve, given the variation of even the segregated capped and composited grades (Figure 14.40). For grade estimation with ordinary kriging (OK) the isotropic variogram (Figure 14.41) was fitted with a spherical model as tabulated on Table 14.18.

Figure 14.39: Variogram Fan diagrams



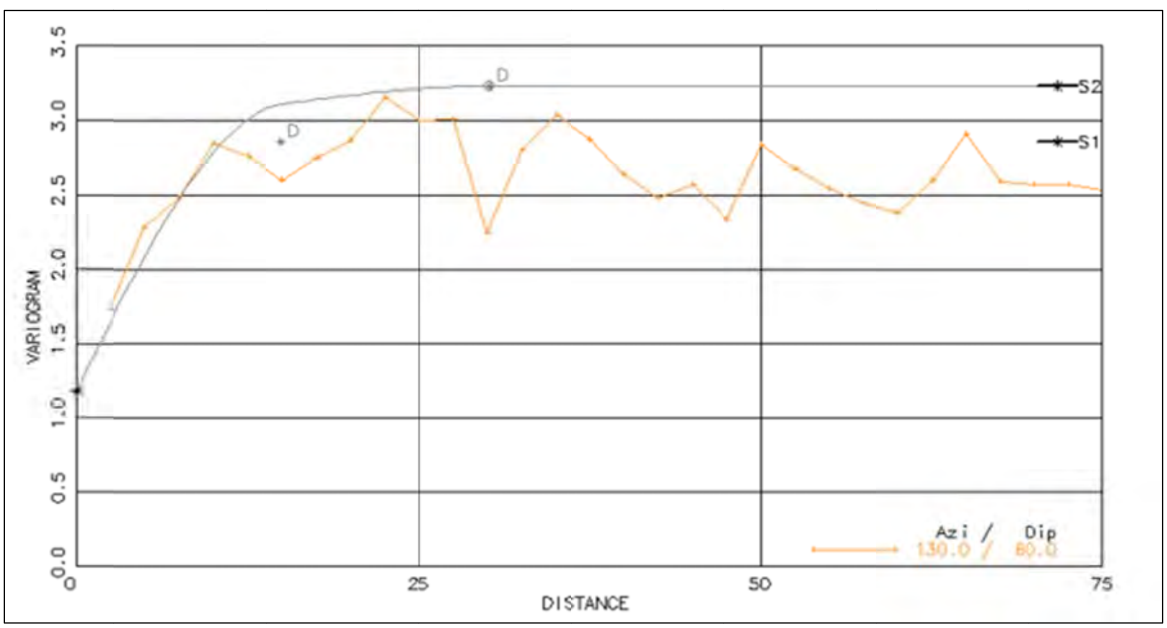
Source: AVM (2016)

Figure 14.40: Anisotropic Variography



Source: AVM (2016)

Figure 14.41: Isotropic Variogram



Source: AVM (2016)

Table 14.20: Kriging Parameters

Kriging Parameters (Au g/t)					
	Range (metres)				Nugget 1.2
Structure	Search Orientation	X	Y	Z	Spatial Variance C
1	Isotropic	15	15	15	1.7
2	Isotropic	30	30	30	0.4

Source: AVM (2016)

14.19 Olive Zone - Estimation Method and Mineral Resource Classification

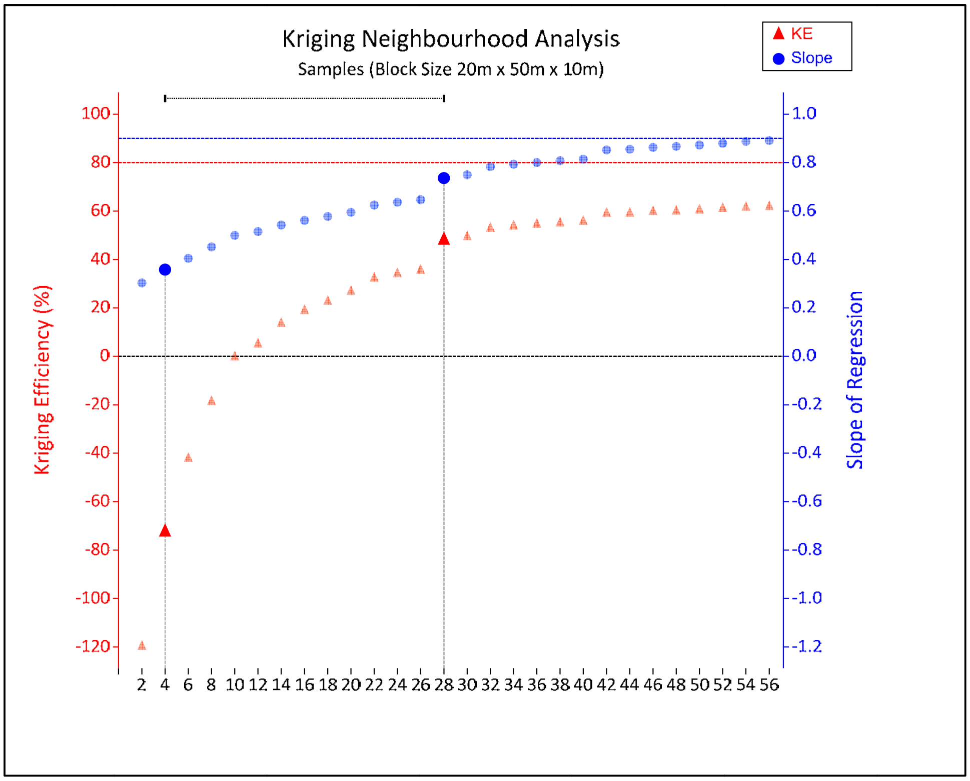
The search neighbourhood strategy used for the grade estimation of gold, with ordinary kriging, and silver, with inverse distance squared is displayed on Table 14.21. Only blocks not estimated with the first SVOL were estimated with a subsequent expanded search. For all volumes, a minimum of four composites were required, with a maximum of two from any given hole. This forces the estimation to use a minimum of two drill holes. The minimum and maximum number of the 2.5 m composites, used for estimation in the search ellipse, was justified with KNA undertaken using Snowden Supervisor software (Figure 14.42).

Table 14.21: Search Neighbourhood Strategy Au g/t

Search Neighbourhood Strategy Au g/t						
Search Volume	Orientation	X (m)	Y (m)	Z (m)	Minimum Number Of Composites	Maximum From One Drill hole
1	Isotropic	15	15	15	4	2
2	Isotropic	30	30	30	4	2
3	Isotropic	75	75	75	4	2

Source: AVM (2016)

Figure 14.42: Kriging Neighbourhood Analysis Minimum and Maximum Number of Samples



Source: AVM (2016)

The utilization of the minimum of four 2.5 m composites for estimation, along with the nested SVOLs, effectively preserved the grade variation of the data. Estimation with a minimum of two drill holes was justified by the drilling pattern and utilization of the mineralization shell. Sectional drilling, generally at 25 m spacing across the apparent strike of the mineralization, was bounded by the shell, preventing extrapolation beyond the limits of information. Table 14.22 summarizes the confidence classification; Measured and Indicated classes were effectively constrained to blocks positioned between the drilling sections.

Table 14.22: Confidence Classification

Confidence Classification			
Class	**Search Volume**	**Minimum Number Of Composites**	**Maximum From One Drill hole**
Measured	1	4	2
Indicated	2	4	2
Inferred	3	4	2

Source: AVM (2016)

14.20 Olive Zone - Model Validation

The following checks were carried out to validate model:

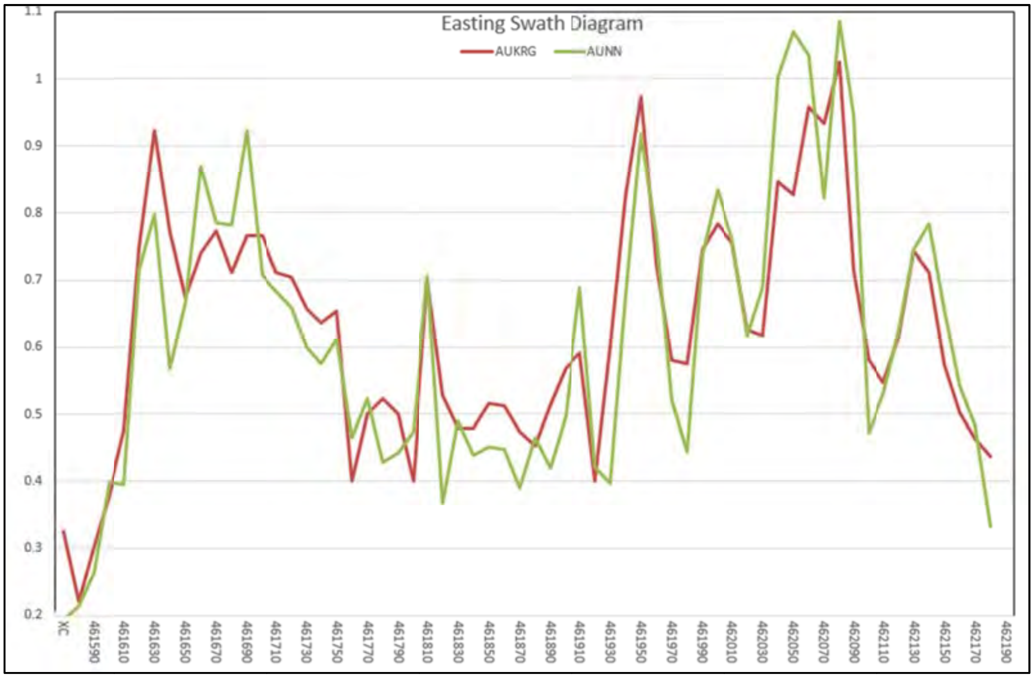
- A visual check in plan and section against drill hole composited values. Grade variation had been preserved and the confidence classification was acceptable;
- Comparison between block grades estimated by ordinary kriging and alternative estimators; Au g/t estimated by both methods was essentially the same (Table 14.23); and
- Swath diagrams, Figure 14.43, where nearest neighbour estimation (green) displays similar variations to ordinary kriging (red).

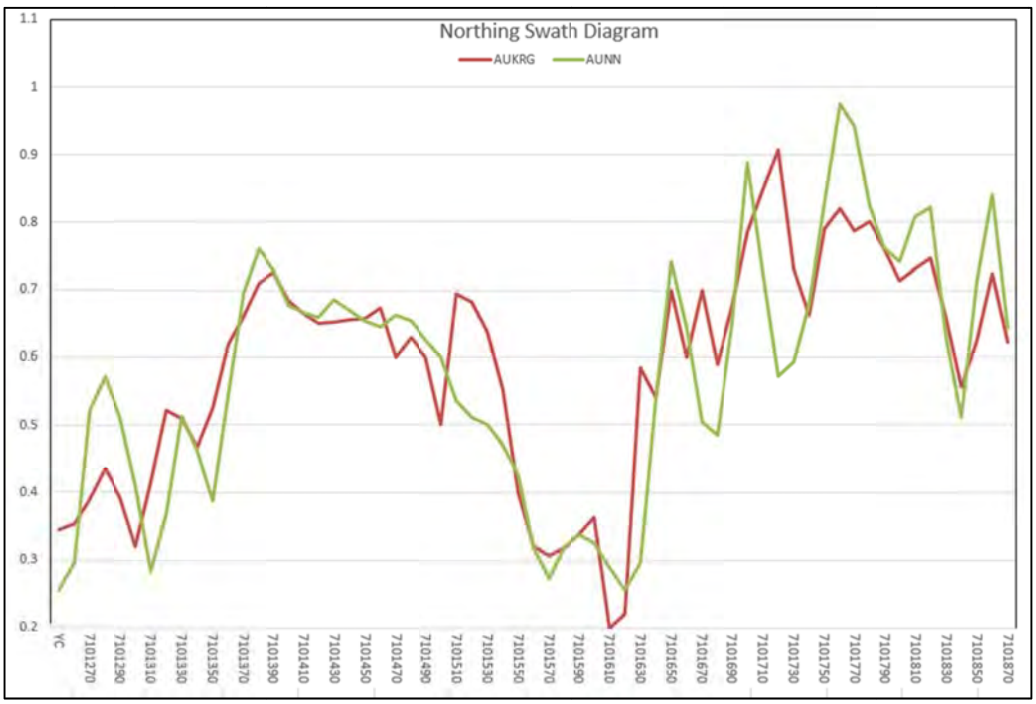
Table 14.23: Inverse to the Distance Squared (ID2) vs Ordinary Kriging (OK) Comparison

ID2 – Ave. Au Grade (g/t)	OK – Ave. Au Grade (g/t)	Difference (%)
0.7245	0.7215	0.41

Source: AVM (2016)

Figure 14.43: Easting and Northing Swath Diagrams





Source: AVM (2016)

14.21 Olive Zone - Mineral Resource Statement

The current Olive Mineral Resources are reported at a cut-off grade of 0.40 g/t Au. The cut-off is supported by the following parameters.

- Mining cost of C$ 2.00/t;
- Processing cost of C$5.00/t;
- G&A cost of C$1.00/t;
- Au recoveries of 69%, 58% and 52% for oxidized rock, transition or mixed oxide-sulphide, and sulphide-bearing Granodiorite, respectively (3% above anticipated actual recoveries); and
- Au price of US$1,700, exchange rate of 0.75.

The resulting calculated cut-off grade for the highest-recovery material is 0.12 g/t Au. Mining is anticipated to be conducted to a 0.20 g/t Au cut-off grade, and the reporting at a 0.40 g/t Au cut-off grade was done to offset the lower recoveries at Olive and to provide higher grade material to the project.

The Mineral Resources are confined within a conceptual pit shell based on the same parameters used for the cut-off grade and 45° slopes; the pit shell optimized on Measured + Indicated + Inferred Resources. Table 14.24 presents the Olive Zone Measured, Indicated and Inferred Mineral Resources.

Table 14.24: Measured, Indicated and Inferred Mineral Resources for Olive Zone

Class	Cut-off grade (g/t Au)	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Ounces Au (koz)	Ounces Ag (koz)
Measured	0.4	1.97	1.19	2.31	75	146
Indicated	0.4	7.55	1.05	2.05	254	498
Meas.+ Ind.	**0.4**	**9.51**	**1.07**	**2.11**	**329**	**645**
Inferred	0.4	7.33	0.89	1.70	210	402

Notes:

1. Mineral Resources are reported in accordance with Canadian Securities Administrators (CSA) National Instrument 43-101 (NI 43-101) and have been estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. CIM definitions were followed for Mineral Resources;
2. In-Pit Mineral Resources are estimated at a cut-off of 0.40 g/t, with mining and processing costs and pit parameters as per the text of this report;
3. Metal prices used for this estimate were US$1,700/oz Au, at an 0.75 exchange rate;
4. High-grade caps were applied as per the text of this report;
5. Specific gravity (bulk density) was assigned for each block based on measurements taken from core specimens;
6. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding;
7. Resources are reported on a 100% basis for Victoria Gold controlled lands; and
8. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues, although the authors are not aware of any such issues.

Source: AVM (2016)

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves.

There is also no certainty that these Inferred Mineral Resources will be converted to Measured or Indicated categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

14.22 Olive Zone - Mineral Resource Sensitivity

The Measured +Indicated Mineral Resources at COGs between 0.0 to 1.0 g/t Au are shown in Table 14.25. Figure 14.44 displays grade-tonnage sensitivity data graphically.

Table 14.25: Olive Zone Grade Tonnage Sensitivity Data for Measured + Indicated Resources at various COGs

Cut-Off (Au g/t)	Tonnes (Million Tonnes)	Au (g/t)	Ag (g/t)	Au oz (Million ounces)	Ag oz (Million ounces)
0.05	17.21	0.69	1.67	0.384	0.927
0.1	16.33	0.73	1.72	0.382	0.902
0.15	15.21	0.77	1.78	0.378	0.870
0.2	13.97	0.82	1.84	0.371	0.829
0.25	12.66	0.89	1.92	0.361	0.781
0.3	11.47	0.95	1.98	0.351	0.730
0.35	10.45	1.01	2.04	0.340	0.686
0.4	9.51	1.07	2.11	0.329	0.645
0.45	8.65	1.14	2.18	0.317	0.606
0.5	7.84	1.21	2.25	0.305	0.567
0.55	7.19	1.27	2.32	0.294	0.536
0.6	6.58	1.33	2.36	0.282	0.500
0.65	6.02	1.4	2.42	0.271	0.468
0.7	5.54	1.46	2.41	0.261	0.430
0.75	5.07	1.53	2.44	0.250	0.398
0.8	4.68	1.6	2.48	0.240	0.373
0.85	4.34	1.66	2.51	0.231	0.351
0.9	3.99	1.72	2.55	0.221	0.327
0.95	3.68	1.79	2.61	0.212	0.308
1	3.37	1.87	2.65	0.202	0.286

Source: AVM (2016)

Figure 14.44: Grade Tonnage Curves for the Olive Zone – Measured + Indicated Resources

Source: AVM (2016)

Table 14.26: Olive In-Pit Mineral Resource Breakdown by Rock and Oxidation Type and Classification at a 0.40 g/t Cut-Off

Class	Tonnes (M)	Au (g/t)	Ag (g/t)	Au oz (k)	Ag oz (k)
Granodiorite					
Measured	1.96	1.19	2.31	75	145
Indicated	7.41	1.04	2.07	249	493
Inferred	7.13	0.89	1.72	205	395
Metasediments					
Measured	0.01	1.11	0.92	0	0
Indicated	0.13	1.19	1.11	5	5
Inferred	0.18	0.78	0.75	4	4
Overburden/Weathered Rock					
Measured	0.01	1.42	4.04	0	1
Indicated	0.01	0.66	4.02	0	1
Inferred	0.02	1.37	3.59	1	3
Oxide					
Measured	0.39	1.38	2.81	17	35
Indicated	0.68	1.32	3.07	29	67
Inferred	0.21	1.55	4.42	10	30
Mix					
Measured	0.9	1.24	2.58	36	75
Indicated	3.52	1.04	2.24	118	253
Inferred	2.81	0.97	2.57	88	232
Sulphide					
Measured	0.68	1.01	1.68	22	37
Indicated	3.34	0.99	1.66	106	178
Inferred	4.31	0.81	1.01	112	140

Source: AVM (2016)

14.23 Summary Conclusions

14.23.1 Eagle Zone

The MRE for the Eagle Zone follows industry best practices. The database is sufficiently verified to allow for resource estimation at a Feasibility level study, and adequately represents the mineralization.

Drilling added since the resource model of the 2012 FS (drilling from late 2011 through 2012) has added significantly to the classification confidence of the updated resource estimate for the Eagle Zone. The added in-fill drilling resulted in a slightly lower average grade (-0.03 g/t) than the prior drilling campaigns (current 0.69 g/t Au average grade versus 0.72 g/t Au average grade prior to 2012, and internal to the mineralized envelope). However, the confidence in the deposit average grade has increased substantially, and the density of drilling has allowed for 21% of the resource to be classified as Measured. On an equivalent cut-off grade basis, the current updated resource model results in approximately the same grade and total contained ounces as in the 2012 FS, but at an increased confidence classification – a demonstration of robust deposit data and modelling in light of a 39% increase in assay data from the 2012 in-fill drilling program.

14.23.2 Olive Zone

The resource estimation methodology employed for the Olive Zone is consistent with industry best practices. The database is sufficiently verified to allow for resource estimation at a FS level, and adequately represents the mineralization.

The Mineral Resource estimation for the Olive Zone is robust, such that further improvements are not indicated as necessary. The Olive Zone gold mineralization has a very short range variography and a higher CV than Eagle Zone, representing a scattered mix of higher and lower grades. This grade distribution pattern could possibly benefit from non-linear estimation methods, such as indicator kriging or uniform conditioning, to provide a better local block estimate. The Olive Zone mineralization should be further evaluated upon completion of drilling at the Shamrock area.

15 Mineral Reserve Estimates

The mineral reserve documented in this section was estimated based on Canadian Institute of Mining (CIM) guidelines that defines Mineral Reserves as “the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A mineral reserve includes diluting materials and allowances for losses that may occur when the material is mined.”

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term ‘mineral reserve’ need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

To convert Mineral Resources to Mineral Reserves estimates of gold price, mining dilution, process recovery, refining/transport costs, royalties, mining costs, processing, and general and administration costs were used to estimate cut-off grades (COG) for each deposit. Along with geotechnical parameters, the COG formed the basis for the selection of economic mining blocks.

The QPs have not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves, except for the risk of not being able to secure the necessary permits from the government for development and operation of the project. The QPs are not aware of any unique characteristics of the project that would prevent permitting.

A summary of the Mineral Reserves for the project are shown in Table 15.1. The effective date of the mineral reserve contained in this report is September 12, 2016.

Table 15.1: Summary of Mineral Reserves

Area	Classification	Ore (Mt)	Diluted Grade (g/t)	Contained Gold (koz)
Eagle	Proven	27	0.80	685
	Probable	90	0.62	1,778
	Total	**116**	**0.66**	**2,463**
Olive	Proven	2	1.02	58
	Probable	5	0.93	142
	Total	**7**	**0.67**	**200**
Eagle + Olive	**Total**	**123**	**0.67**	**2,663**

Note: Mineral Reserves are included within Mineral Resources
Source: JDS (2016)

15.1 Open Pit Mineral Reserve

15.1.1 Open Pit Mineral Reserve Basis of Estimate

The mineral reserve for the property is based on the mineral resource estimate completed by was carried out by QPs Allan Moran, Ravindra Sharma and Frank Daviess, with an effective date of September 12, 2016 for the Eagle and Olive Zones.

The Mineral Reserves were developed by examining each deposit to determine the optimum and practical mining method. COGs were then determined based on appropriate mine design criteria and the adopted mining method. A shovel and truck open pit mining method was selected for the various deposits. Only Measured and Indicated Mineral Resources were included in the optimization process. Inferred resources were considered as waste.

A thorough analysis of the optimized shells was then conducted in order to select the shells to be used as guides to the subsequent detailed pit designs.

15.1.2 Mining Method and Mining Costs

The deposits at the Eagle Gold site are amenable to extraction by open pit methods. For the purposes of the preliminary optimization, mining costs of $2.30/t mined were assumed. The open pit cost estimate was generated from first principles and by benchmarking comparable Canadian operations in similar northern locations. Optimized shells were developed for the deposits.

The open pit optimizations resulted in open pits at each of the two deposits at the site, Eagle and Olive, and provide the basis of estimation for the open pit Mineral Reserves.

15.1.3 Dilution

As input to the initial pit limit optimization and subsequent mine scheduling, and in order to reflect the selectivity of the mining method chosen when compared to the block model parameters, an external mining dilution was calculated and applied to the various deposits.

This external mining dilution was based on calculating the number of waste blocks adjacent to an ore block in the mineral inventory block model (utilizing Hexagon Mining MineSight™ "four side contact routine"). Only Measured and Indicated blocks which were contained within a given zone and above a given gold cut-off grade were considered as ore blocks.

The number of waste block face contacts, with ore block faces for each block, was calculated on each horizontal plane in the model. The number of waste faces (or edges) may vary from zero (i.e. block is surrounded by ore blocks) to four (i.e. block is totally surrounded by waste blocks). Dilution was estimated using the number of waste edges for each block, an assumed grade of zero for all waste and a width of dilution of 0.3 to 0.5 m for each edge.

The analysis resulted in external dilutions of 5% and 9% being applied to the Eagle and Olive deposits, respectively.

15.1.4 Geotechnical Considerations

SRK Consulting (US) Inc. (SRK) carried out slope stability analyses based on previous geotechnical characterization programs conducted by BGC (2012) to develop feasibility level open pit mine design parameters for the FS update. The various pit slope design parameters, including geotechnical considerations, are discussed in detail in the Section 16 - Mining Methods.

Based on the location and characteristics of the geomechanical domains and the pit shells, design sectors were identified for each of the proposed pits. Slope stability analyses were undertaken on each sector to define achievable slope configurations. The results from these analyses provided guidance on achievable bench face, interramp and overall slope angles for each design sector.

The results of the SRK analyses and a review of precedent practice suggest that the recommended geometries are reasonable and appropriate. To achieve these angles, the design assumes that controlled blasting and pro-active geotechnical monitoring would be undertaken. In addition, "drained" slopes were assumed based on the 2012 FS BGC hydrogeology work and horizontal drain recommendations, along with an ongoing commitment to geomechanical data collection and analyses over the LOM operation.

15.1.5 Lerchs-Grossman Optimization

The sizes and shapes of the ultimate open pits were determined using the Lerchs- Grossman (LG) pit optimization algorithm as implemented in DataMine NPV Scheduler (NPVS) software. Key inputs used for the LG runs are presented in Table 15.2.

Criteria for crushed ore versus ROM ore were specified. Ore to be crushed will be hauled to the primary crusher located towards the north-east side of the Eagle pit. ROM ore will be hauled directly to the primary HLP at a combined nominal production rate of 33,700 t/d ore.

Table 15.2: Mine Planning Optimization Input Parameters*

Parameter	Unit	FS - Eagle Deposit	FS - Olive Deposit
Revenue, Smelting & Refining			
Gold price	US$/oz Au	$1,275	
Exchange Rate	C$:US$	0.75	
Payable metal	%	99.5	
TC/RC/Transport	C$/oz Au	10.00	
Royalty @ 1% NSR	C$/oz Au	17.00	
Net gold value per ounce	C$/oz	1,665	
Net gold value per gram	C$/g	53.51	
OPEX Estimates			
OP Waste Mining Cost	C$/t waste mined	2.30	
OP Ore Mining Cost (Crush)	C$/t ore mined	2.30	
OP Ore Mining Cost (ROM)	C$/t ore mined	2.60	-
Strip Ratio (estimated)	W:O	1.0	2.5
OP Mining Cost	C$/t leached	4.60	8.05
Crush HL Pad			
Heap Leach Processing, Water Treatment, Rinsing	C$/t leached	6.00	
G&A	C$/t leached	1.70	
Total OPEX Cost (excluding mining)	C$/t leached	7.70	7.70
Total OPEX Cost (including mining)	C$/t leached	12.30	15.75
ROM HL Pad			
Heap Leach Processing, Water Treatment, Rinsing	C$/t leached	4.50	-
G&A	C$/t leached	1.70	-
Total OPEX Cost (excluding mining)	C$/t leached	6.50	-
Total OPEX Cost (including mining)	C$/t leached	10.80	-
Recovery and Dilution			
External Mining Dilution	%	5	9
Mining Recovery	%	95	
Crush Gold Recovery			
Type1 - Oxide Granodiorite	%	79	-
Type 2 - Altered Granodiorite	%	73	-
Type 3 - Unaltered Granodiorite	%	68	-
Type 4 - Oxide Medasediments	%	73	-
Type 6 - Unaltered Medasediments	%	68	-
Oxide	%	-	66
Mixed	%	-	55
Sulphide	%	-	52
ROM Gold Recovery			
All	%	55	-

Source: JDS (2016)

*These parameters differ slightly from those used in the economic model due to subsequent, more detailed estimation work but the differences are not considered material

A separate series of pit optimization runs was completed for each deposit to determine the final open pit shapes.

Based on the analysis of the shells and preliminary mine schedule, the base case ultimate shell was selected for each deposit. In all cases, ultimate shells were selected on the basis of maximizing NPV but also minimizing additional lower grade and higher strip ratio material (i.e. higher incremental strip ratios with minimal increases in value) that have minimal benefit to the overall NPV. In addition, pit phases were also selected for Eagle based on the optimization results and used as the basis for the detailed ultimate pit and phase designs.

15.1.6 Cut-Off Grade and Resource Classification Criteria

Once pit shapes were established, marginal COG were used to determine the total amount and grade of ore in each pit. The marginal, or incremental, COG is specific to the mining method and is defined as the minimum grade at which mineralized material, already located at the pit rim (i.e. contained within the pit and already mined), pays for all additional costs incurred if it is sent for processing. According to this definition, the marginal COG for each deposit and oxidation type is summarized in Table 15.3 and this corresponds to a break-even grade that excludes mining costs. The open pit Mineral Reserves comprise all mineralized material with grades equal to or above this marginal COG.

Table 15.3: Marginal COGs by Deposit and Material Type

Area/Type	Material	Cut-off Grade (g/t Au)
Eagle Crush	Oxide Granodiorite	0.19
	Altered Granodiorite	0.21
	Unaltered Granodiorite	0.22
	Oxide Medasediments	0.21
	Unaltered Medasediments	0.22
Olive Crush	Oxide	0.24
	Mixed	0.29
	Sulphide	0.30
Eagle ROM	All	0.23

Source: JDS (2016)

15.1.7 Mine Design

Detailed pit design involves the conversion of the optimized pit shells into an operational open pit mine design, which is discussed further in Section 16. Table 15.4 gives the main parameters used in determining the pit designs.

Table 15.4: Pit Design Parameters

Description	Value
Ultimate Pit Design Parameters – All Pits	
Bench Height	10 m (single, working)
	20 m (double; final pit configuration)
Face Angle	60° to 70° (double-bench, final pit)
Berm Width	10 m to 12 m
Interramp Angle (IRA)	38° to 49°
Ramp Width – Double lane	27 m (total excavation)
Ramp Width - Single lane -lower benches	20 m
Ramp Gradient – Double lane	10
Ramp Gradient – Single lane – lower benches	12
Overall Angle (OSA)	36° to 45°

Source: JDS (2016)

15.1.8 Open Pit Mineral Reserves Estimate Statement

The Eagle Gold open pit mineral reserve is presented in Table 15.5 and Table 15.6.

Table 15.5: Eagle Gold Open Pit Mineral Reserve Estimate

Area	Classification	Ore (Mt)	Diluted Grade (g/t)	Contained Gold (koz)
Eagle	Proven	27	0.80	685
	Probable	90	0.62	1,778
	Total	**116**	**0.66**	**2,463**
Olive	Proven	2	1.02	58
	Probable	5	0.93	142
	Total	**7**	**0.95**	**200**
Eagle + Olive	**Total**	**123**	**0.67**	**2,663**

Note: Mineral Reserves are included within Mineral Resources

Source: JDS (2016)

Table 15.6: Eagle Gold Mineral Reserves by Type

Type	Area	Ore (Mt)	Diluted Grade (g/t)	Contained Gold (koz)
Crushed Ore	Eagle	101	0.72	2,330
	Olive	7	0.95	200
	Total	**108**	**0.73**	**2,530**
ROM Ore	Eagle	15	0.27	133
	Olive	-	-	-
	Total	**15**	**0.27**	**133**
Crushed + ROM	**Total**	**123**	**0.67**	**2,663**

Note: Mineral Reserves are included within Mineral Resources
Source: JDS (2016)

16 Mining Methods

16.1 Introduction

The Eagle Gold project comprises the Eagle and Olive deposits which are planned to be extracted by open pit shovel and truck mining methods.

Pit optimizations were conducted to determine the optimal open pit mine plan. This analysis and subsequent detailed mine design estimated 123 Mt of ore at an average gold head grade of 0.67 g/t. The contained gold is estimated to be 2.7 Moz.

Industry standard mining methods, equipment, dilution calculations and production rates were applied in the planning process.

16.2 Open Pit Mining

16.2.1 Mine Design Methodology and Design Criteria

16.2.1.1 Design and Planning Methodology

Industry standard methodologies for pit limit analysis, mining sequence, cut-off grade optimization, and detailed design were adopted.

The main steps in the planning process were:

- Assignment of economic criteria to the geological resource models;
- Definition of optimization parameters such as gold price, preliminary operating cost estimates, pit wall angles, preliminary dilution and metallurgical recovery estimates for each mine area and material type;
- Calculation of economic ultimate pit limits for the various deposits using the NPVS software. This software applies the Lerchs Grossmann algorithm to define optimal mining shells;
- Establishment of an economic scheduling sequence using the NPVS series of optimum nested pits as guides;
- Development of detailed pit designs (incorporating pit accesses and appropriate bench heights and pit geometry) for the ultimate pits using Hexagon Mining MineSight™ (MineSight) software;
- Determination of optimal pit phasing using the same tools as those applied for the ultimate pit designs;
- Determination of incremental (or heap leach feed) cut-off grade based on economic parameters;
- Determination of external mining dilution based on Mineral Resource block model;
- Development of the life of mine (LOM) production schedule to maximize economic return, while satisfying heap leach feed and mine production constraints;
- Development of waste rock storage area (WRSA) designs and volume estimations;

- Calculation of hauling distances per bench and per pit or phase, according to the LOM plan for each of the deposits, and design of the haulage network;
- Determination of equipment requirements based on the LOM production schedule, haul distances, and performance and operational characteristics of the proposed equipment using the Runge Pincock Minarco Talpac software. A spreadsheet model was created for estimation of operating hours and number of units required. This model was also used to calculate equipment procurement schedules, workforce requirements, capital expenditures and operating costs; and
- Industry equipment- operating parameters were applied, with due consideration to the size and location of the operation, to select the equipment. Equipment and workforce productivities were estimated according to industry standards for a northern environment, for the size of the equipment and the mine production rate.

16.2.1.2 Site Topography and Climatic Conditions

Mine planning for the project considered the northern climate environment, permafrost, and a remote camp situation, as well as the physical characteristics of the ore and waste rock.

16.2.1.3 Topographic and Resource Model Description

16.2.1.3.1 Topography

Mine topography, including the WRSA and HLP areas, was provided digitally by Victoria Gold in UTM NAD83, Zone 8 coordinates. Topography was supplied as a digital file with contour intervals of 1 m. This was used for all pit design calculations and engineering estimates. Volumetric estimates were derived from design surfaces intersecting the topographic surface.

16.2.1.3.2 Resource Model

The 3D resource block models for the Eagle and Olive deposits used in this FS were prepared by Qualified Person Allan Moran (AIPD CPG) and explained in detail in the Mineral Resource estimate. The models comprise parameters that describe lithology, in-situ density, ore and waste types, resource classification, ore and waste percentage, oxidation type, and gold grade.

16.2.2 Open Pit Optimization and Sensitivity Analysis

16.2.2.1 Objective and Scope

The optimization process generates a series of nested pit shell surfaces for the purpose of designing open pits across the various deposits. The Lerchs Grossmann algorithm in the NPVS software package was used for the optimization and associated analysis. The resulting nested pit shells were generated by varying the revenue factor (gold price factor) applied to the base case values.

Measure and Indicated Mineral Resources were included in the pit optimization process. Inferred resources were treated as waste.

Table 15.2 above summarizes the parameters used for each deposit, while Table 15.3 illustrates the calculated incremental COGs for the various ore types. The internal cut-off grade takes into account all operating costs except mining costs. This internal cut-off grade is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the NPVS optimization. This internal cut-off was applied to all of the mineral reserve estimates that follow.

Pit shell generation was not constrained by any existing infrastructure as the only existing features are exploration access roads. All of the major infrastructure facilities planned for the project (WRSAs, HLPs, offices, maintenance shops, fuel storage, processing facilities, permanent camp, and water storage ponds) will be external to the ultimate pit designs and their area of influence.

16.2.2.2 Open Pit Optimization Results

A series of optimized shells were generated for the two deposits at Eagle Gold based on varying revenue factors. The results were analyzed with shells chosen as the basis for ultimate limits and preliminary phase selection.

NPVS produces both a best case (i.e., mine out shell 1, the smallest shell, and then mine out each subsequent shell from the top down, before starting the next shell) and a worst case (mine each bench completely to final limits before starting next bench) scenarios. These two scenarios provide a bracket for the range of possible outcomes.

The shells were produced based on varying revenue factors (0.3 through to 1.3 of base case) to produce the series of nested shells and their respective NPV results.

To better determine the optimum shell on which to base the phasing and scheduling, and to gain a better understanding of the mineability of each deposit, the various pit shells were analyzed in a preliminary schedule. The schedule assumed a combined (crush plus ROM ore) nominal production rate of 33,700 t/d ore. No stockpiles were used in the analysis and no capital cost estimate (CAPEX) was considered.

Based on the analysis of the shells and preliminary schedule, shells were chosen as the base case ultimate shell for each deposit. In all cases, ultimate shells were selected not only on the basis of maximizing NPV, but also minimizing the addition of increasingly lower grade and higher strip ratio ore (i.e. higher incremental strip ratios) that generate only a minimal improvement on the overall NPV. In addition, pit phases were also selected for Eagle based on the optimization results and used as the basis for the detailed ultimate pit and phase designs.

The results of the pit optimizations, based on the mineral inventory block models and subsequent analyses and shell selection of the various deposits at Eagle Gold, are summarized in Table 16.1.

Table 16.1: Pit Optimization Results – Eagle and Olive

Description	Unit	Eagle (Pit #30)	Olive (Pit #47)	Total
Total Heap Leach Feed	Mt	116	7	122
Diluted Gold Grade	Au (g/t)	0.66	0.98	0.68
Contained Gold	Au (koz)	2,470	209	2,679
Waste	Mt	102	13	114
Total Material	Mt	217	19	236
Strip Ratio	t:t	0.88	1.88	0.93

Source: JDS (2016)

16.2.3 Open Pit Design Parameters

16.2.3.1 General Design Parameters

The general design parameters used in the various detailed pit and phase designs are summarized in Table 15.4 above.

16.2.3.2 Haul Road and Ramp Design Parameters

The Eagle Gold project site roads fall into two categories as summarized in Table 16.2. All site roads are considered private roads and access will be controlled by Victoria Gold.

Table 16.2: Eagle Gold Project Road Design Criteria

Type	Design Vehicle	Overall Width	Maximum Gradient
In-pit haul Road	Largest mine truck (144 t)	27 m	10% standard
			12% for pit bottom access
Site Road (light vehicle traffic)	Standard operating vehicles (light trucks, crew transport, supply and delivery vehicles, service vehicles and occasional use by heavy equipment)	8 to 10.4 m	8%

Source: JDS (2016)

The primary haulage roads are required between the various open pit deposits and the primary ore crusher, WRSAs, primary HLP, construction areas and maintenance facilities. Roads are planned to be, as far as practical, constructed using all-fill techniques, utilizing waste rock sourced from the open pits, to achieve design alignment and grade.

Roads within the ultimate WRSAs are designed to be all-fill construction. Dust control on the roads will be done using water trucks, with the addition of chemical suppressants if needed.

The main haul roads and ramps are designed to have an overall road allowance width of 27 m. The selected road allowance is adequate for accommodating three times the width of the largest haul truck (144 t), with additional room for drainage ditches and safety berms as summarized in Table 16.3.

Table 16.3: In-Pit Haulage Road Design Parameters

Item	Metres
Truck (144 t) operating width	6.9
Running surface - 3x truck width	20.7
Berm height (Three quarters tire height)	2.2
Berm width at 40° slopes	4.4
Ditch width	2.0
Total Road Allowance	27.0

Source: JDS (2016)

Ramps are designed with a maximum grade of 10% (steepened to 12% for final access to lower portions of the open pits). Ex-pit roads are designed to allow access to roads connecting the various pits to the crusher and waste dumps and are planned to be a maximum of 30 m wide (i.e. an all-fill road).

16.2.4 Open Pit Designs

Detailed mine designs were undertaken for the Eagle and Olive Zones and the approximate dimensions are shown in Table 16.4, with plan views of each open pit design shown in Figure 16.1 to Figure 16.2.

For the Eagle pit, in order to maintain access to the primary crusher (elevation 1,050 masl), and access to the primary HLP for ROM material, a haul road spirals down to the bottom of the western side of the pit. This haul road also connects to the external access road which leads to the truck shop. No final haul roads are designed along the final highwall above the crusher elevation in order to minimize waste stripping requirements.

At Olive, haul roads are designed to spiral down to the pit bottoms. These ramps tie in to the access road to the WRSA located to the southwest of the pit and further to the primary crusher adjacent to the Eagle pit.

Table 16.4: Open Pit Dimensions

Open Pit	Length (m)	Width (m)	Maximum Depth (m)
Eagle	1,300	550	475 (east highwall)
Olive	850	200	180 (southeast highwall)

Source: JDS (2016)

Figure 16.1: Eagle Pit Design

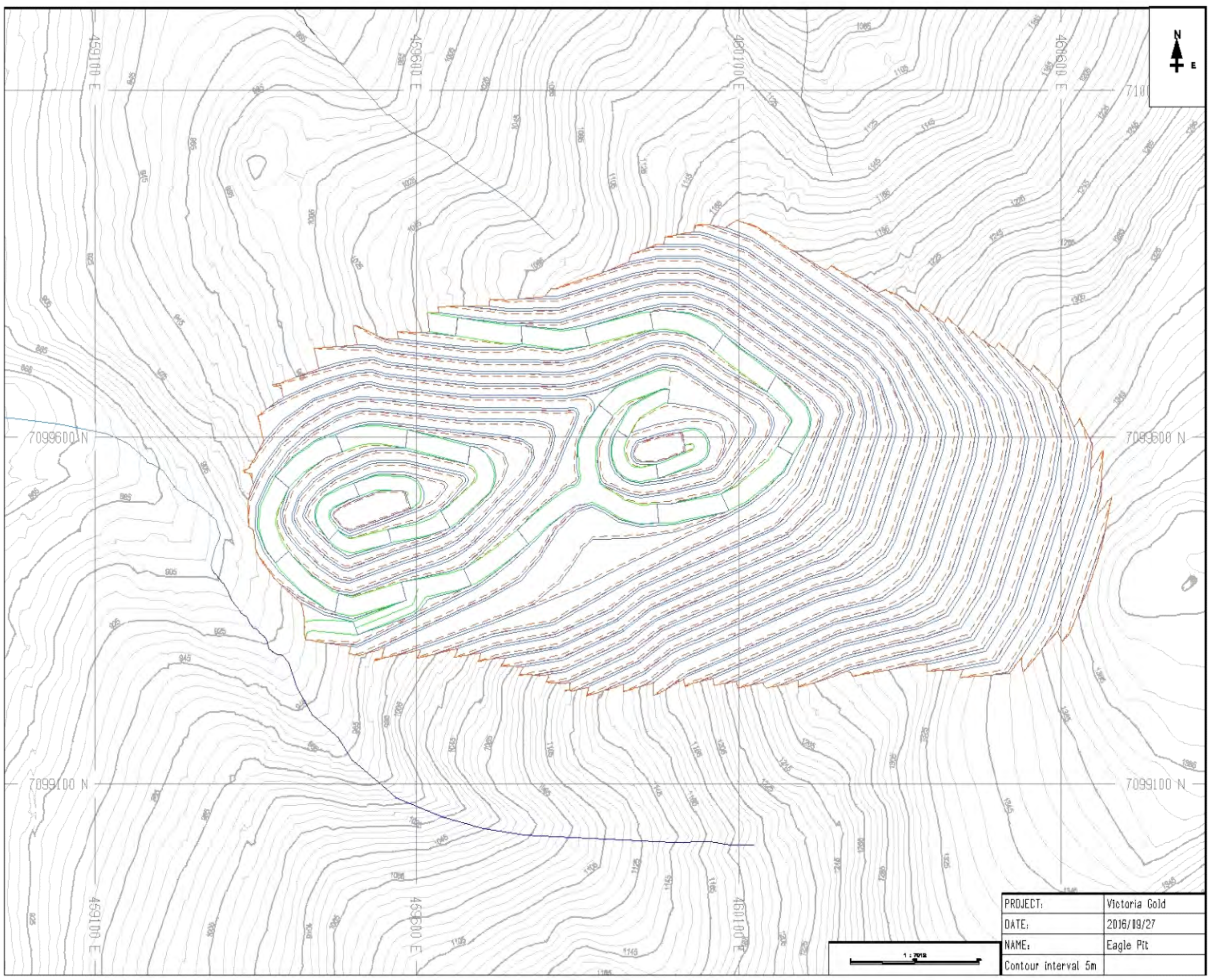
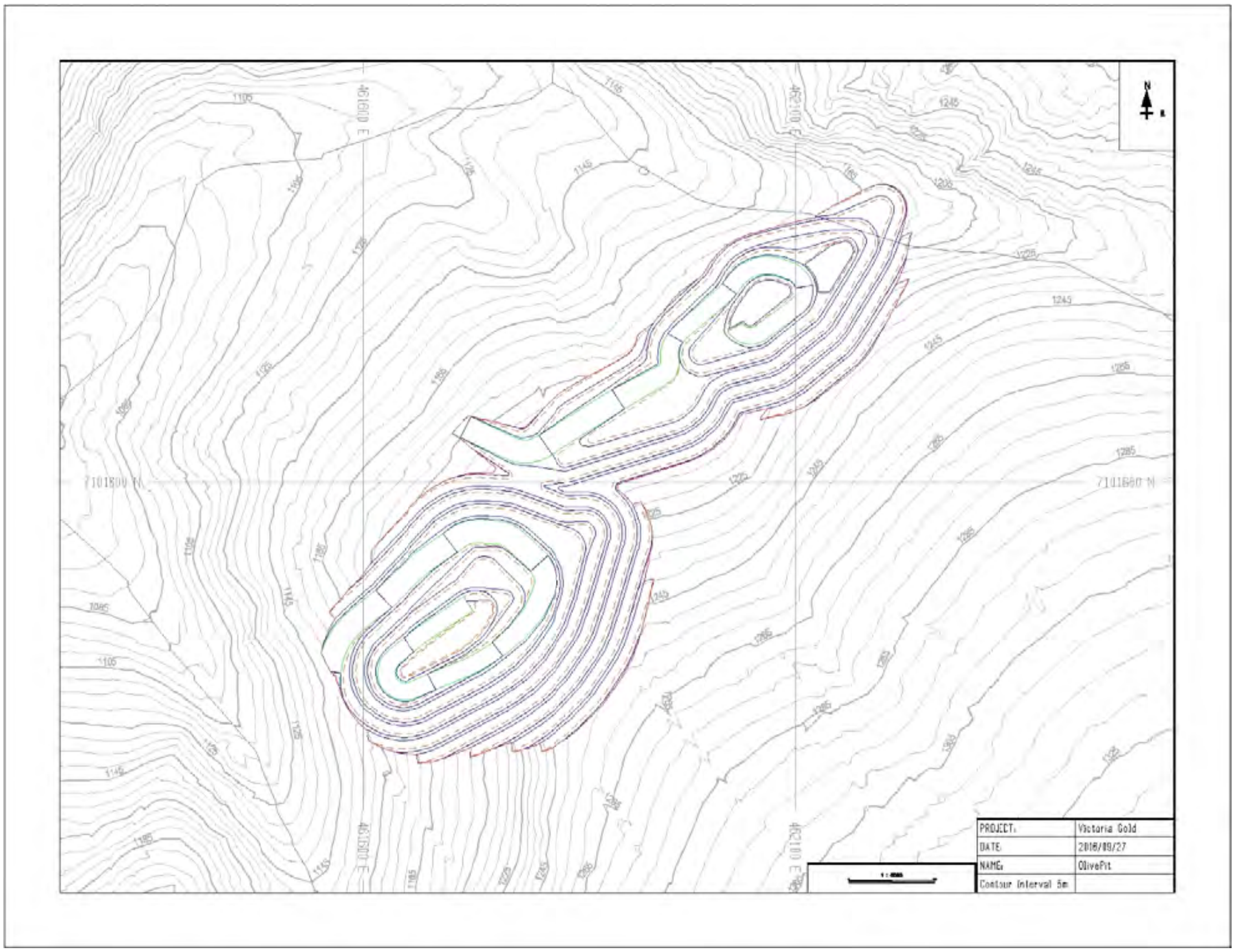


Figure 16.2: **Olive Pit Design**



16.2.5 Comparison of Final Pit Design and Optimized Pit Shells

The optimized pit shells and final pit design ore and waste tonnages along with diluted grades are compared in Table 16.5. Total ore tonnages in the final open pit designs match the optimized shells, with the corresponding waste material 2.0% higher. These slight differences are due to maximizing the ore extraction, pit shell smoothing (to achieve a practical and realistic pit design) and maintaining minimum mining widths in the final designs and are considered to be well within acceptable limits.

Table 16.5: Material in Optimized Shell versus Final Pit Designs

Description	Total Heap Leach Feed			Waste Quantity (Mt)	Total Quantity (Mt)	Strip Ratio (t:t)
	Quantity (Mt)	Grade Au (g/t)	Metal Au (koz)			
Pit Optimization Results						
Eagle	116	0.66	2,470	102	217	0.88
Olive	7	0.98	209	13	19	1.88
Total	122	0.68	2,679	114	236	0.93
Mineral Reserves (Final Pit Design)						
Eagle	116	0.66	2,463	99	216	0.85
Olive	7	0.95	200	17	23	2.60
Total	**123**	**0.67**	**2,663**	**116**	**239**	**0.95**
Difference Reserve vs. Optimization	0%	-1%	-1%	2%	1%	2%

Source: JDS (2016)

16.2.6 Waste Materials

Geochemical characterization studies to identify and quantify the potential for metal leaching and acid rock drainage (ML/ARD) for waste and ore associated with the project were included in the feasibility studies conducted in 1995/1996 by New Millennium Mining Ltd.; in 2007 baseline studies by StrataGold; and in a more comprehensive program completed in 2010 by Stantec. These evaluations have indicated that the waste and ore associated with the project are likely to be non-acid generating. Minor proportions may have some propensity, albeit likely low, to generate localized acidity and therefore not necessarily to sort the small proportion of waste that may have a low potential to generate acid from the vast majority that is anticipated to be non-acid generating. Therefore, waste rock will be placed in the WRSAs without regard to chemical composition.

Waste rock material produced from the Eagle and Olive pits was divided into three categories, as outlined in Table 16.6. Note that no significant amounts of overburden are expected within the various open pits.

Table 16.6: Open Pit Waste Rock Summary

Type	Definition
Metasedimentary	Rock which is highly weathered and foliated and generally shows poor mechanical properties
Intrusive	Rock exhibiting a similar weathering pattern as the metasedimentary but has a noticeably higher inherent strength and a higher structural integrity
Miscellaneous	Includes topsoil (thickness from 0.2 to 0.5 m) and colluvium (thickness from 2 to 7 m)

Source: JDS (2016)

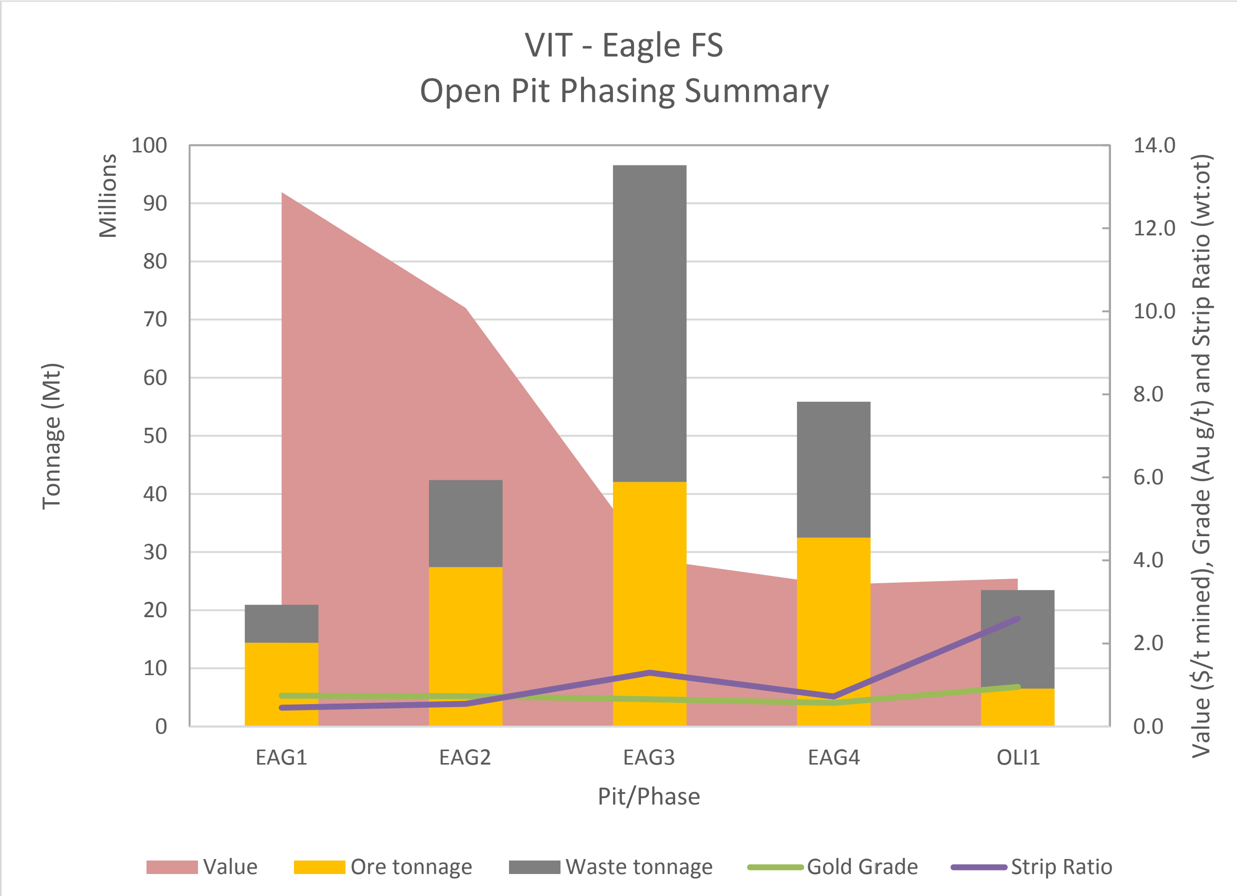
16.2.7 Open Pit Phase Design

For the Olive pit no additional pushbacks or phases were allowed for in the mine plan development due to its relatively small footprint. For the much larger Eagle pit, a total of four phases or pushbacks are designed in order to optimize the mine schedule and maximize the project value.

The mining schedule maximizes economic returns and achieves the target crush heap leach throughput target of 10.95 Mt/a through concurrent mining of the various phases and deposits. The open pit mining sequence, which is based on mining higher value material early on in the mine life, begins with the various phases of the Eagle pit, with Olive pit being mined during the latter part of the mine plan.

Figure 16.3 further summarizes the pit and phase designs for each of the deposits, illustrating ore and waste mined tonnages, gold grade, strip ratio and contained value. The contained value (which drives the optimized mining sequence) is based on the mine design criteria taking into account net metal price, operating costs and heap leach gold recoveries.

Figure 16.3: Open Pit Summary



Source: JDS (2016)

16.2.8 Mine Production Schedule

16.2.8.1 Summary

The basic criteria used for the development of the LOM production schedule are to:

- Maximize NPV of the project;
- Maximize the value in the early years of the operation through the use of stockpiles (when no stacking of heap leach) and concurrent open pit mining of the various phases at Eagle;
- Ensure crush heap leach ore loading of 10.95 Mt/a;
- Minimize pre-production mining while ensuring adequate waste material suitable for construction is produced from the Eagle pit in the pre-production period;
- Capitalize pre-stripping tonnage (Year -1) of 2.1 Mt total material using Owner-operated equipment and resources;
- Maximize pit production rate per period according to the geometry of the phases and the number of shovels that can work within that geometry. Resultant maximum total yearly mine open pit production is 29.8 Mt (LOM average 24.0 Mt/year);
- Establish both a crush and ROM stockpile to accommodate mining throughout the year including the winter period (January through March) when there is no ore stacking of the HLP;
- Crush ore to be conveyed to the primary and secondary HLP, while ROM ore to be hauled to the primary HLP;
- Convey all crush ore to the secondary HLP (near Olive deposit) once the primary HLP crush capacity is met (Year 7);
- Send ROM ore to the Primary HLP only; and
- Plan on operating the open pit mine 365 days per year (allowing for 10 non-operating days per year due to weather delays).

16.2.8.2 Heap Leach Feed Schedule and Constraints

The heap leach loading rate is a function of the mining production schedule, capital cost constraint and operating cost optimization. Stacking of the HLPs is planned to occur 275 days per annum. Primary crushing is planned to occur throughout the year but no stacking is planned to occur during the coldest months of the year, January through March. An average annual throughput of 10.95 Mt has been assumed for the crushed ore while the ROM ore is to be stacked as it is produced from the mine.

16.2.8.3 Mine Plan and Open Pit Production Schedule

Table 16.7 is a summary of ore (for crush and ROM) and waste rock movement by year and by pit for the LOM production schedule along with the heap leach feed schedule.

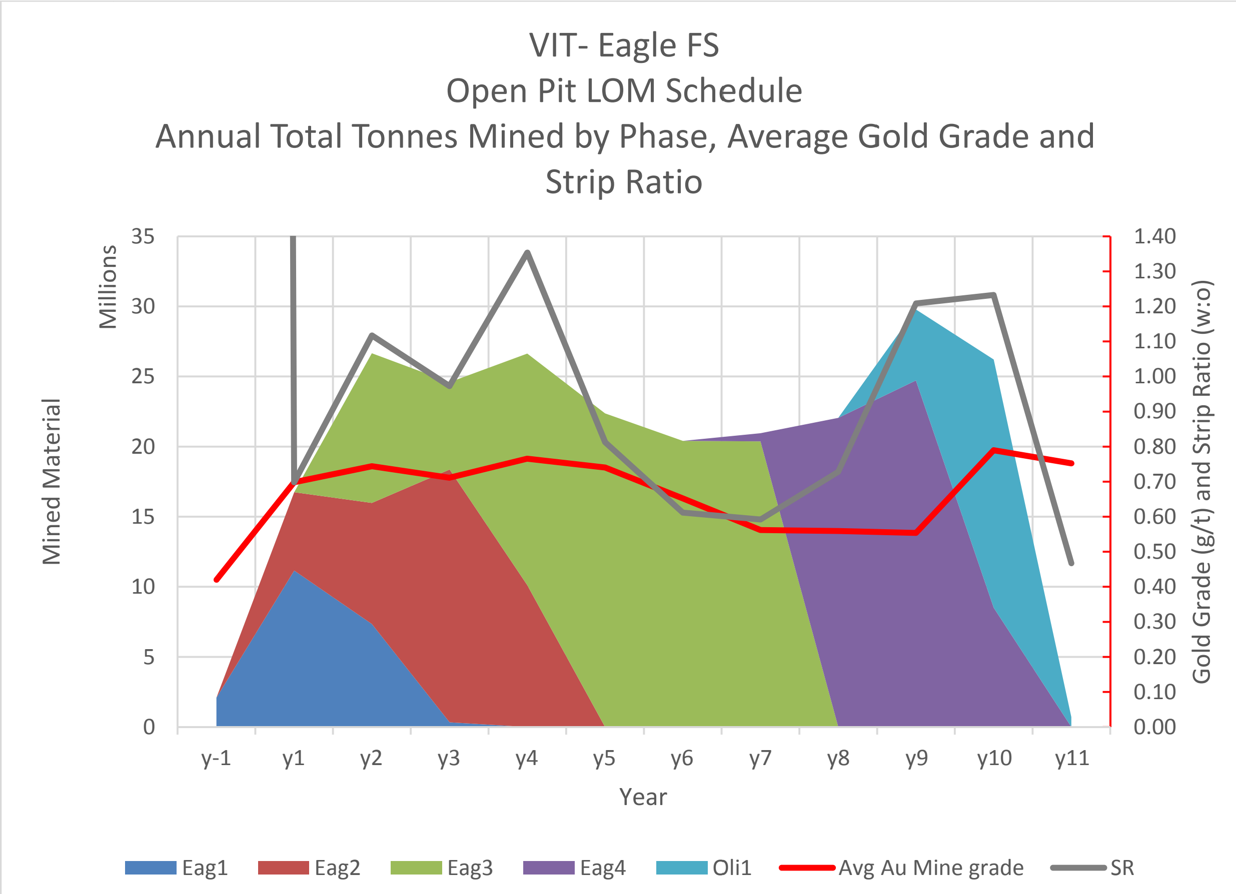
Figure 16.4 and Figure 16.5 summarize ore/waste tonnages, grade, recovered gold and strip ratio by year.

Table 16.7: LOM Production Schedule – Eagle Gold Deposits

Description	Unit	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
EAGLE														
Crush Ore	Mt	**101.3**	0.0	8.8	11.0	10.9	10.9	10.9	11.0	11.0	10.9	10.3	5.6	-
Crush Gold Grade	g/t	**0.72**	0.49	0.75	0.81	0.77	0.78	0.80	0.71	0.62	0.61	0.58	0.71	-
Crush Contained Gold	k oz	**2,330**	0	212	287	272	275	282	251	218	213	192	128	-
ROM Ore	Mt	**15.1**	0.0	1.1	1.6	1.5	0.4	1.4	1.7	2.2	1.8	2.5	0.8	-
ROM Gold Grade	g/t	**0.27**	0.29	0.27	0.28	0.27	0.28	0.27	0.27	0.28	0.27	0.27	0.28	-
ROM Contained Gold	k oz	**133**	0	10	15	13	3	12	15	20	16	22	7	-
Total Ore	Mt	**116.4**	0.0	9.9	12.6	12.5	11.3	12.3	12.7	13.2	12.8	12.8	6.4	-
Total Gold Grade	g/t	**0.66**	0.42	0.70	0.74	0.71	0.77	0.74	0.65	0.56	0.56	0.52	0.66	-
Total Contained Gold	k oz	**2,463**	0	222	301	285	279	294	266	238	229	215	135	-
Waste	Mt	**99.4**	2.1	6.9	14.1	12.1	15.3	10.0	7.7	7.8	9.3	11.9	2.1	-
Strip Ratio	wt:ot	**0.85**	83.49	0.70	1.12	0.97	1.35	0.81	0.61	0.59	0.73	0.93	0.33	-
OLIVE														
Crush Ore	Mt	**6.5**	-	-	-	-	-	-	-	-	-	0.7	5.4	0.5
Crush Gold Grade	g/t	**0.95**	-	-	-	-	-	-	-	-	-	1.15	0.94	0.75
Crush Contained Gold	k oz	**200**	-	-	-	-	-	-	-	-	-	25	163	12
Waste	Mt	**16.9**	-	-	-	-	-	-	-	-	-	4.4	12.3	0.2
Strip Ratio	wt:ot	**2.60**	-	-	-	-	-	-	-	-	-	6.33	2.31	0.47
TOTAL MINE														
Crush Ore	Mt	**107.8**	0.0	8.8	11.0	10.9	10.9	10.9	11.0	11.0	10.9	10.9	11.0	0.5
Crush Gold Grade	g/t	**0.73**	0.49	0.75	0.81	0.77	0.78	0.80	0.71	0.62	0.61	0.62	0.83	0.75
Crush Contained Gold	k oz	**2,530**	0	212	287	272	275	282	251	218	213	218	291	12
ROM Ore	Mt	**15.1**	0.0	1.1	1.6	1.5	0.4	1.4	1.7	2.2	1.8	2.5	0.8	
ROM Gold Grade	g/t	**0.27**	0.29	0.27	0.28	0.27	0.28	0.27	0.27	0.28	0.27	0.27	0.28	
ROM Contained Gold	k oz	**133**	0	10	15	13	3	12	15	20	16	22	7	
Total Ore	Mt	**122.9**	0.0	9.9	12.6	12.5	11.3	12.3	12.7	13.2	12.8	13.5	11.7	0.5
Total Gold Grade	g/t	**0.67**	0.42	0.70	0.74	0.71	0.77	0.74	0.65	0.56	0.56	0.55	0.79	0.75
Total Contained Gold	k oz	**2,663**	0	222	301	285	279	294	266	238	229	240	298	12
Waste	Mt	**116.3**	2.1	6.9	14.1	12.1	15.3	10.0	7.7	7.8	9.3	16.3	14.5	0.2
Strip Ratio	wt:ot	**0.95**	83.49	0.70	1.12	0.97	1.35	0.81	0.61	0.59	0.73	1.21	1.23	0.47
Total Material	Mt	**239.2**	2.1	16.8	26.7	24.6	26.6	22.4	20.4	21.0	22.1	29.8	26.2	0.7
Total Mined	t/day		5,726	45,892	73,048	67,365	72,976	61,280	55,912	57,409	60,425	81,590	71,810	1,931
Heap Leach Schedule														
Total Crush	Mt	**107.8**	-	8.8	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	0.5
Crush Gold Head Grade	g/t	**0.73**	-	0.75	0.81	0.77	0.78	0.80	0.71	0.62	0.61	0.62	0.83	0.75
Crush Contained Gold	k oz	**2,530**	-	212	287	272	275	282	251	218	213	218	291	12
Total ROM	Mt	**15.1**	-	1.1	1.6	1.5	0.4	1.4	1.7	2.2	1.8	2.5	0.8	-
ROM Gold Head Grade	g/t	**0.27**	-	0.27	0.28	0.27	0.28	0.27	0.27	0.28	0.27	0.27	0.28	-
ROM Contained Gold	k oz	**133**	-	10	15	13	3	12	15	20	16	22	7	-
Total Crush/ROM	Mt	**122.9**	-	9.9	12.6	12.5	11.3	12.3	12.7	13.2	12.8	13.5	11.7	0.5
Total Gold Head Grade	g/t	**0.67**	-	0.70	0.74	0.71	0.77	0.74	0.65	0.56	0.56	0.55	0.79	0.75
Total Contained Gold	k oz	**2,663**	-	221	301	285	278	294	266	238	229	240	298	12

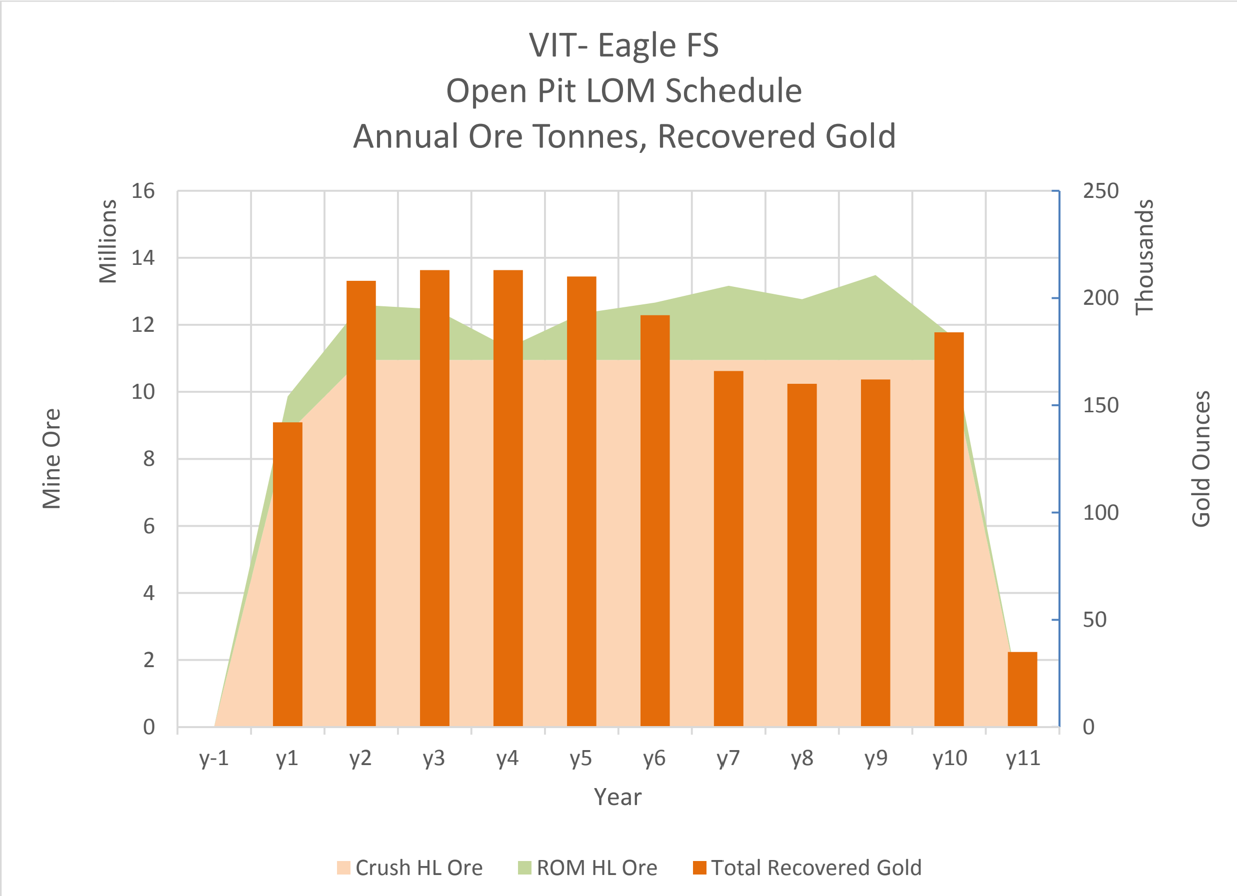
Source: JDS (2016)

Figure 16.4: Total Phase Ore and Waste Tonnages, Gold Grade, Strip Ratio



Source: JDS (2016)

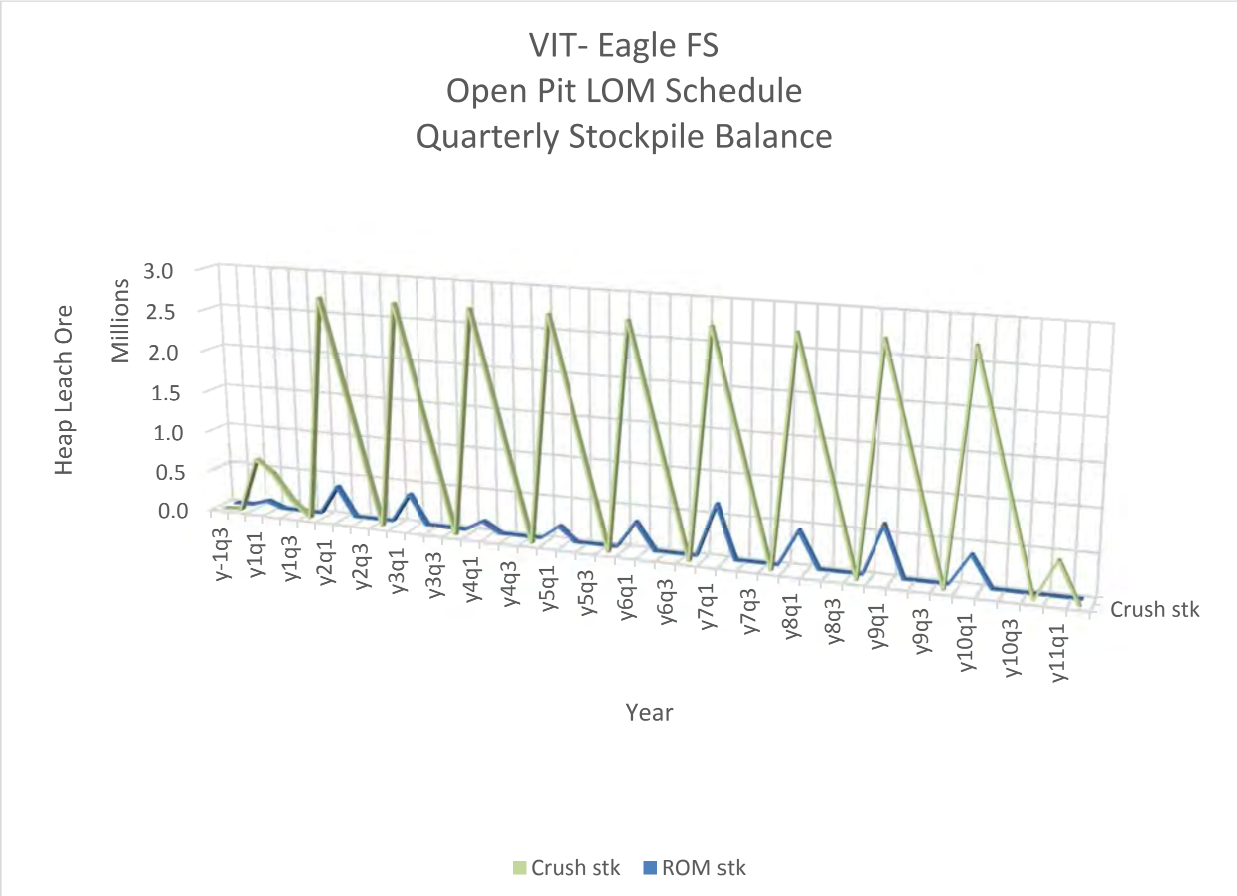
Figure 16.5: Mine Ore Tonnages and Recovered Gold



Source: JDS (2016)

Figure 16.6 illustrates the quarterly stockpile (both crush and ROM) closing balances. The seasonal nature of the heap leach stacking requirements versus the continuous mining operation results in variable closing stockpile balances. The stockpile closing balances peak in the first quarter of every year since no heap leach stacking occurs in this period. The mine schedule is such that the stockpiles are depleted during the remainder of each year.

Figure 16.6: Stockpile Balance



Source: JDS (2016)

Figure 16.7 to Figure 16.16 provide layout drawings with the status of the open pit configuration, WRSAs, as well as HLP advance, at the end of each year.

16.2.8.4 Open Pit Development

Year -1: This period covers the pre-production period. Open pit mining commences with development of the Eagle pit. Suitable waste rock is planned to be used for construction (roads, laydown areas and the HLP). A total of 2.1 Mt of waste mined in this period with trace amounts of low grade ore used as HLP over-liner.

Year 1: First year of heap leach stacking and processing (assumed at 80% of maximum process rate). During the active stacking period, crush ore is to be conveyed to the primary HLP while ROM ore is hauled. Open pit mining at Eagle continues in Phase 1 and 2. A total of 9.9 Mt of ore is scheduled to be mined in the year (crush + ROM). Mined gold grade for the year will average 0.70 g/t. Waste rock totalling 6.9 Mt will be produced for a strip ratio of 0.7:1.

Year 2: Mining in Eagle Phase 1 and 2 will continue with waste stripping of Phase 3 commencing. Average gold head grade over the period is expected to be 0.74 g/t at the target crush HL production rate of 10.9 Mt/a (additional 1.6 Mt of ROM ore mined). 14.1 Mt of waste rock is planned to be mined for a strip ratio of 1.1:1.

Years 3 to 5: Mining at the Eagle Phase 1 will be completed in Year 3, with Phase 2 ending in Year 4, while Phase 3 continues over the entire period. Crush gold head grade is expected to average 0.78 g/t. The waste produced over the three year period is planned to total 37.5 Mt with a total of 36.1 Mt of total heap leach ore feed for an average strip ratio of 1.0:1.

Years 6 to 8: Phase 3 at Eagle will be completed in Year 7 with the final pushback (Phase 4) commencing in the same year. The primary HLP is expected to reach the crush ore capacity in Year 7 and remainder of crush ore is conveyed to the secondary HLP near the Olive deposit. The ROM ore will continue to be hauled to the primary HLP throughput the mine life. A total of 38.6 Mt of heap leach feed will be mined (crush + ROM ore). Gold head grades are estimated to average 0.59 g/t and total waste produced from the Eagle pit is estimated to be 24.8 Mt.

Years 9 to 11: Mining at Eagle Phase 4 is completed in Year 10. The Olive open pit commences in Year 9 and will be completed at the beginning of Year 11. Overall gold head grades average 0.67 g/t with strip ratios increasing to 1.2:1 with the mining of the Olive pit.

Figure 16.7: **Annual Map Year -1**

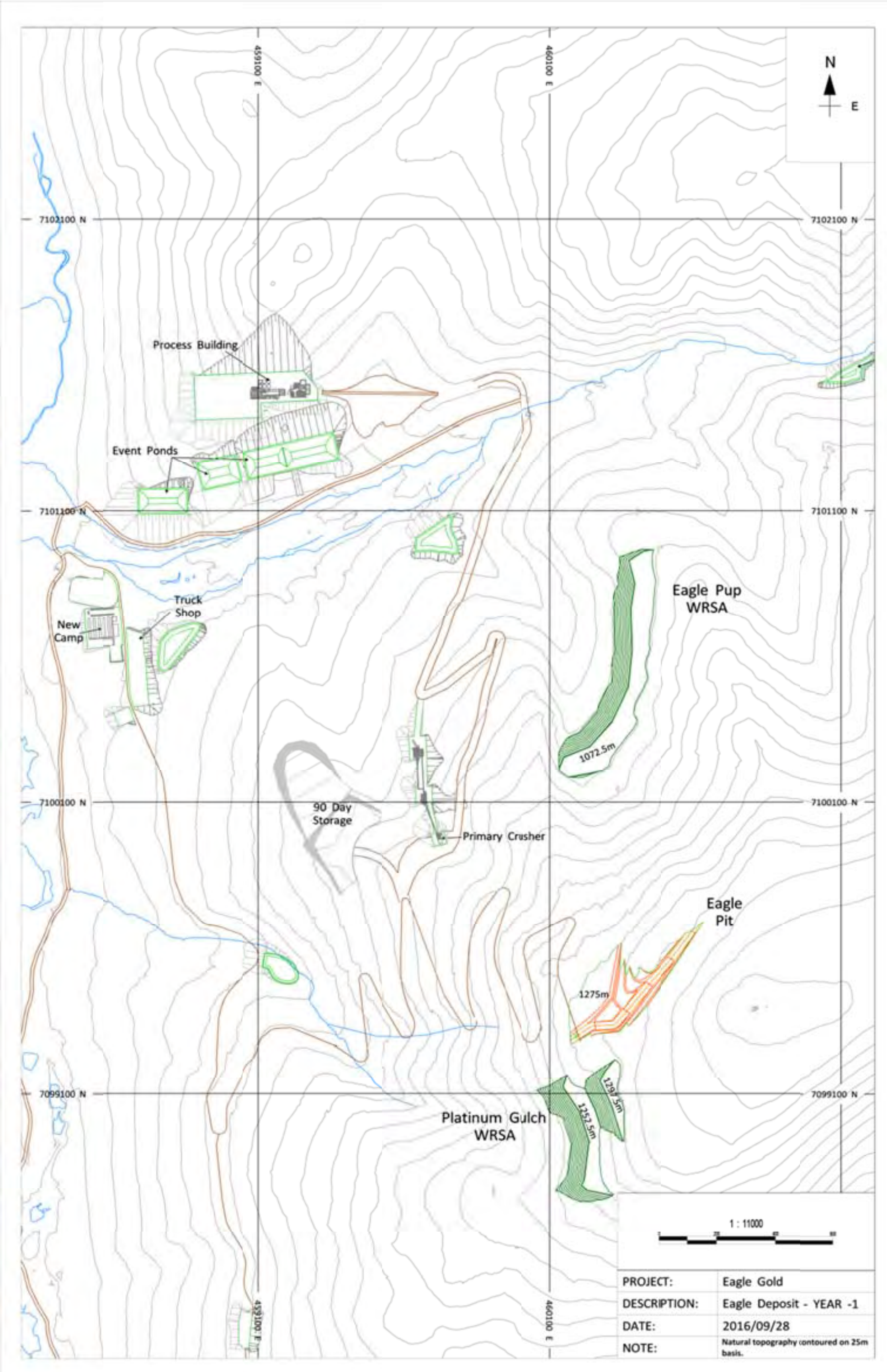


Figure 16.8: Eagle Deposit Annual Map Year 1

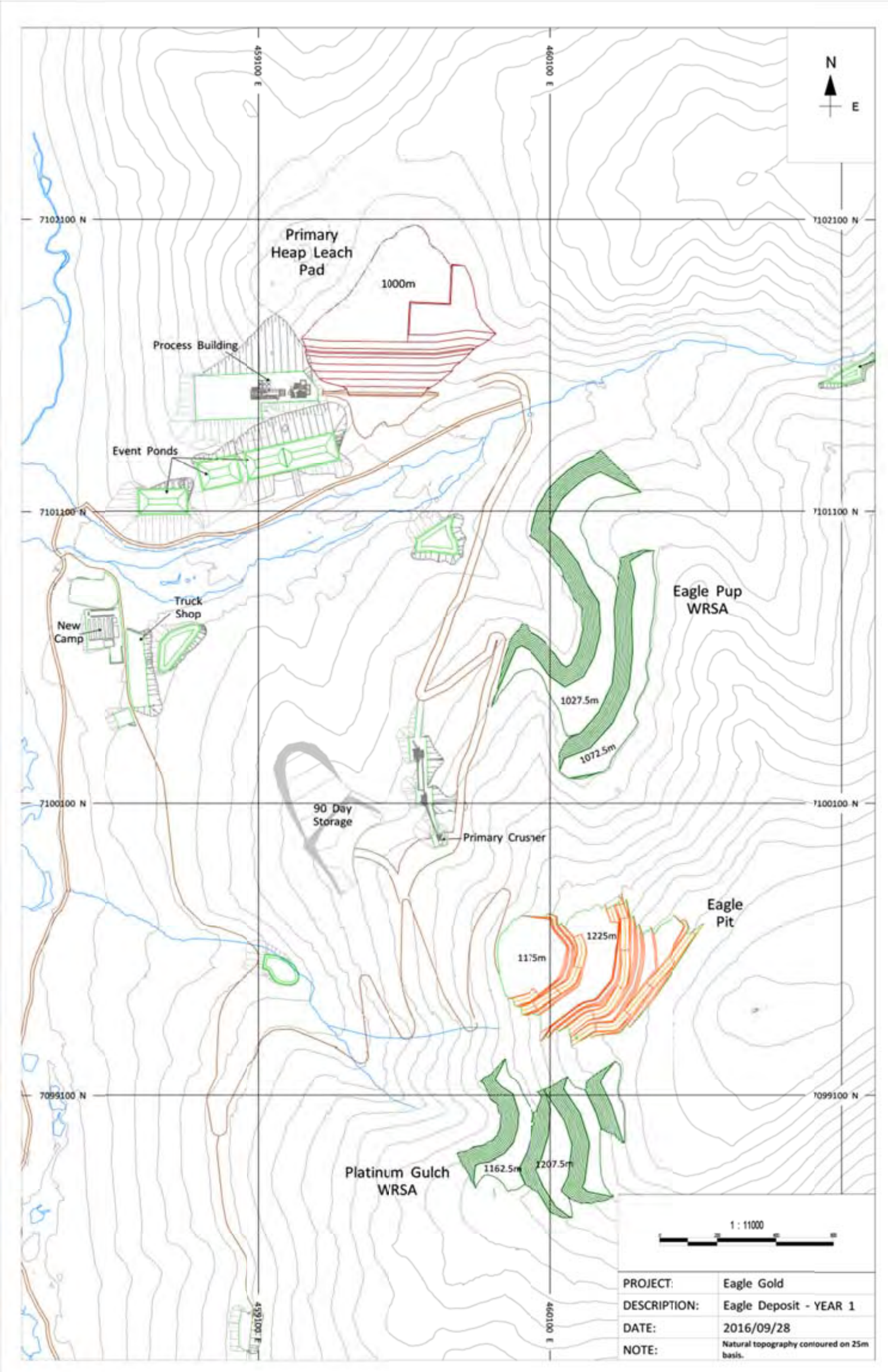


Figure 16.9: Eagle Deposit Annual Map Year 2

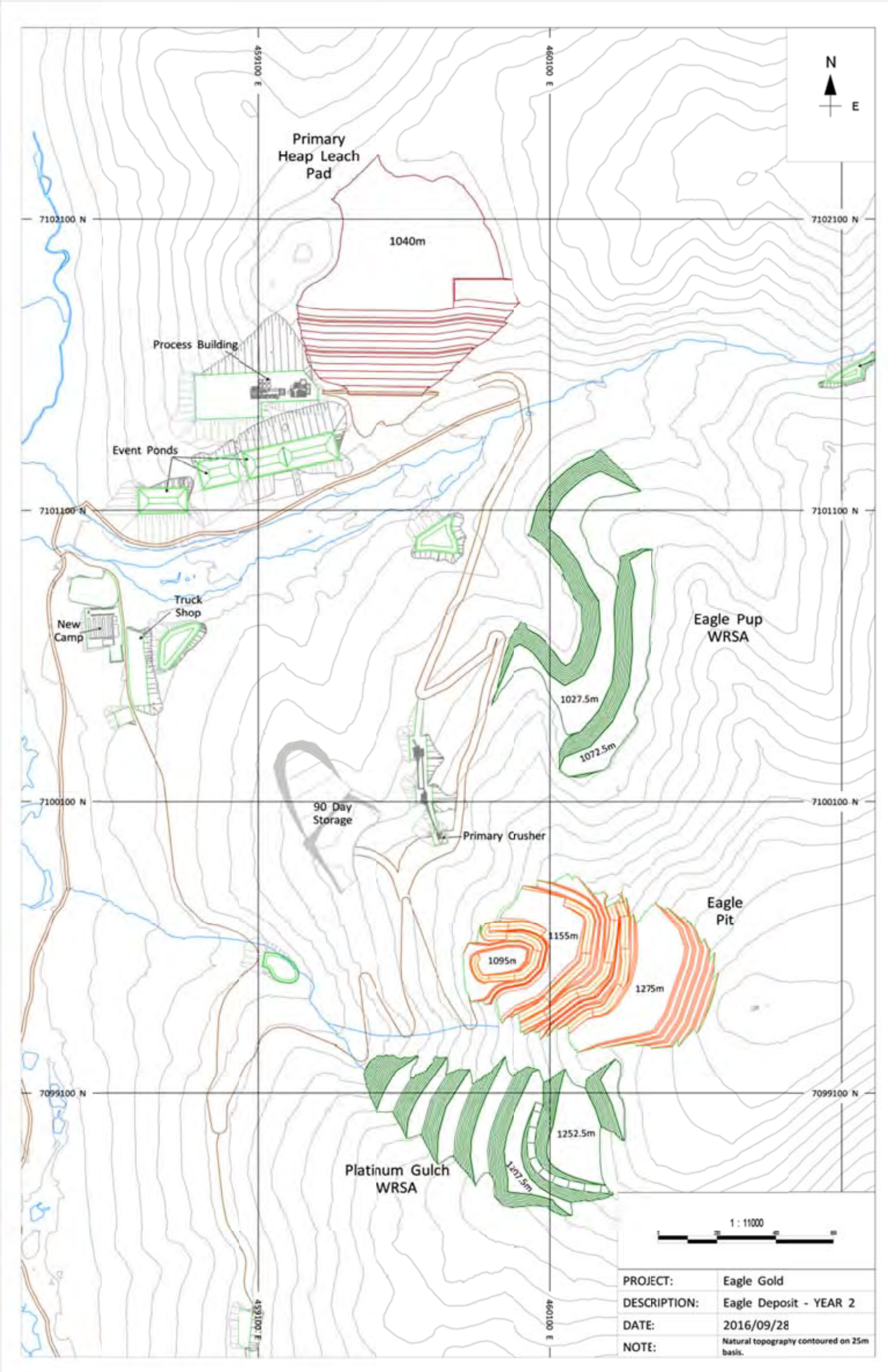


Figure 16.10: Eagle Deposit Annual Map Year 3

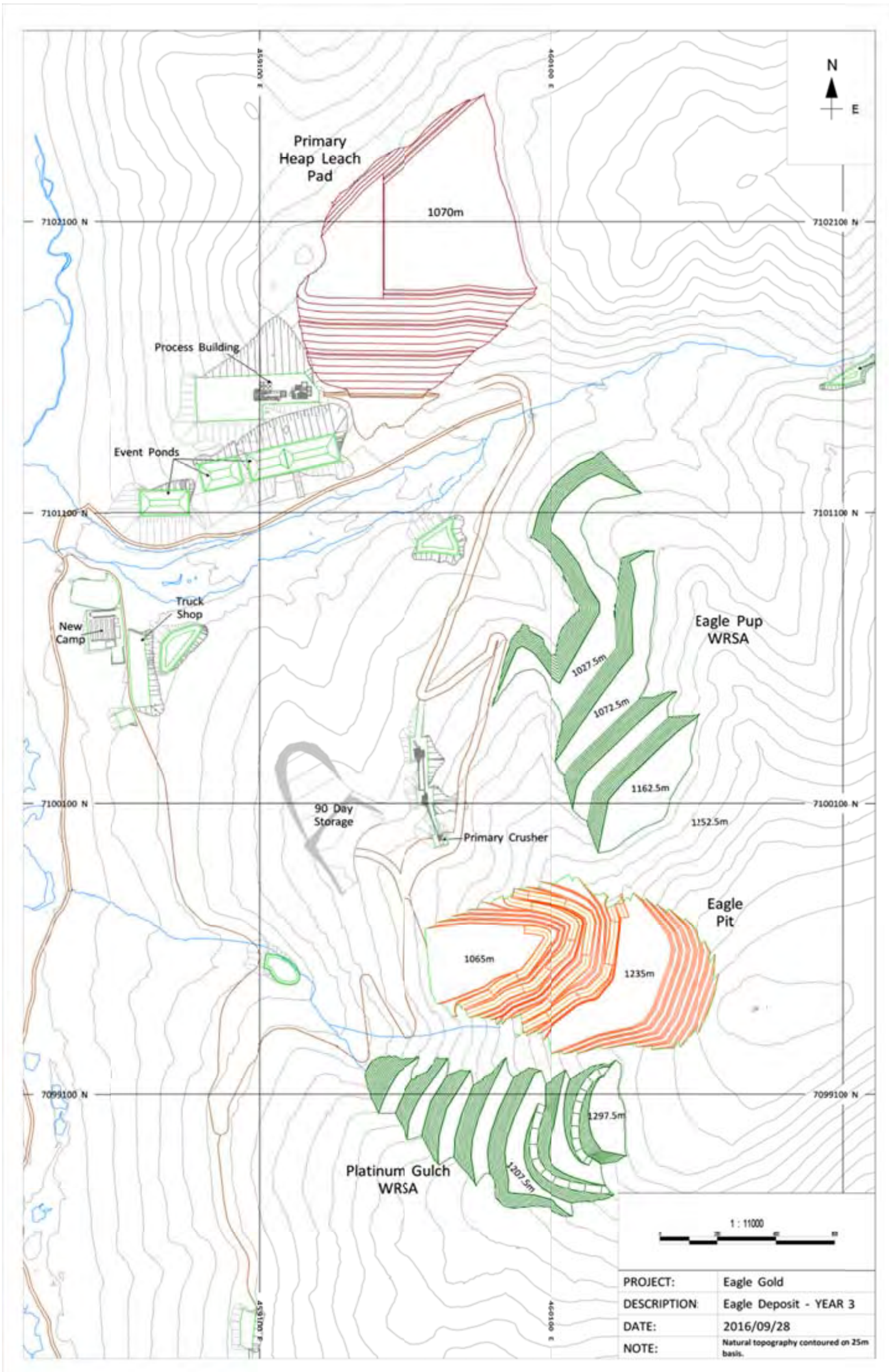


Figure 16.11: **Eagle Deposit Annual Map Year 5**

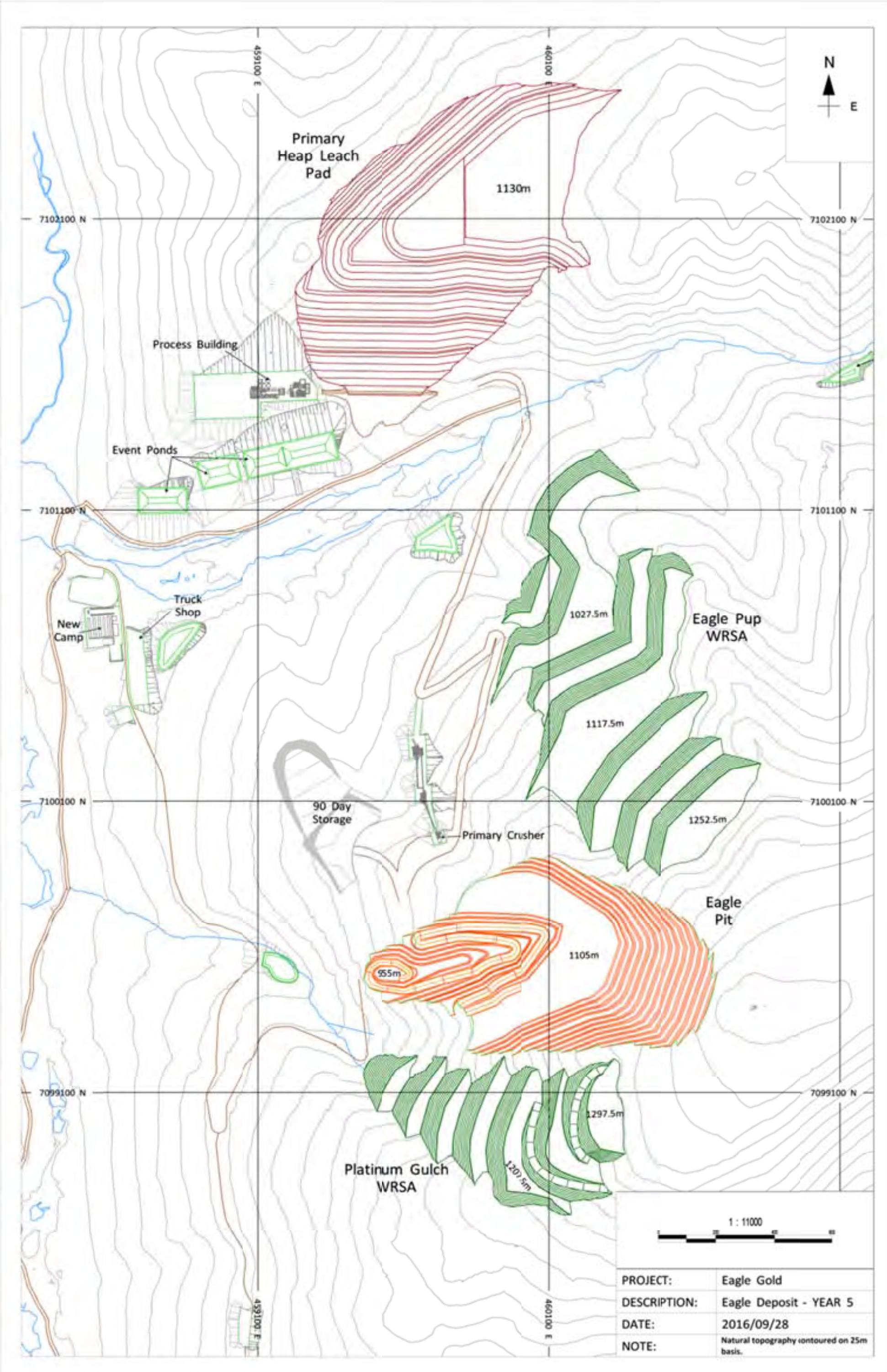


Figure 16.12: Eagle Deposit Annual Map Year 7

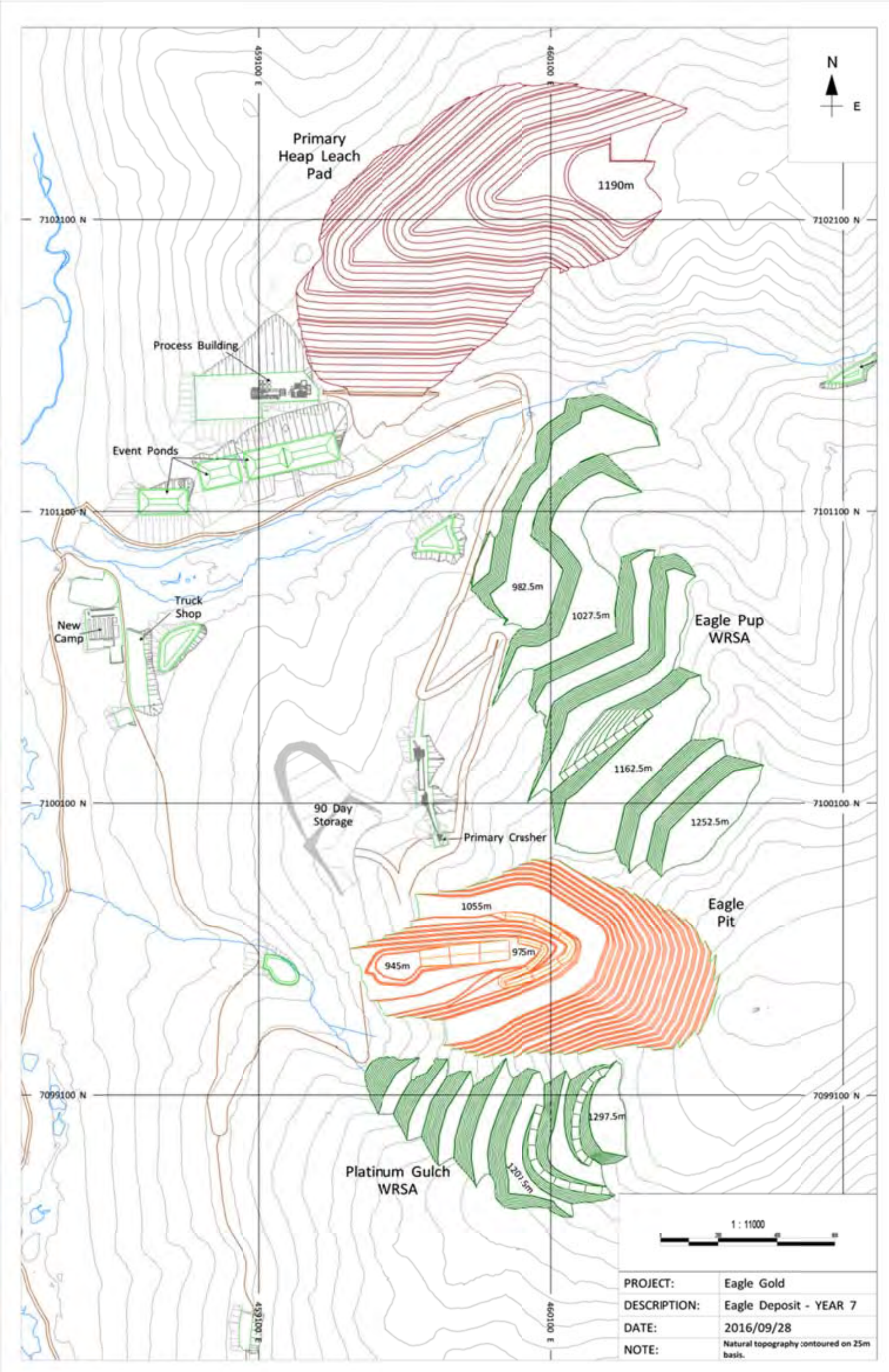


Figure 16.13: Eagle Deposit Annual Map Year 9

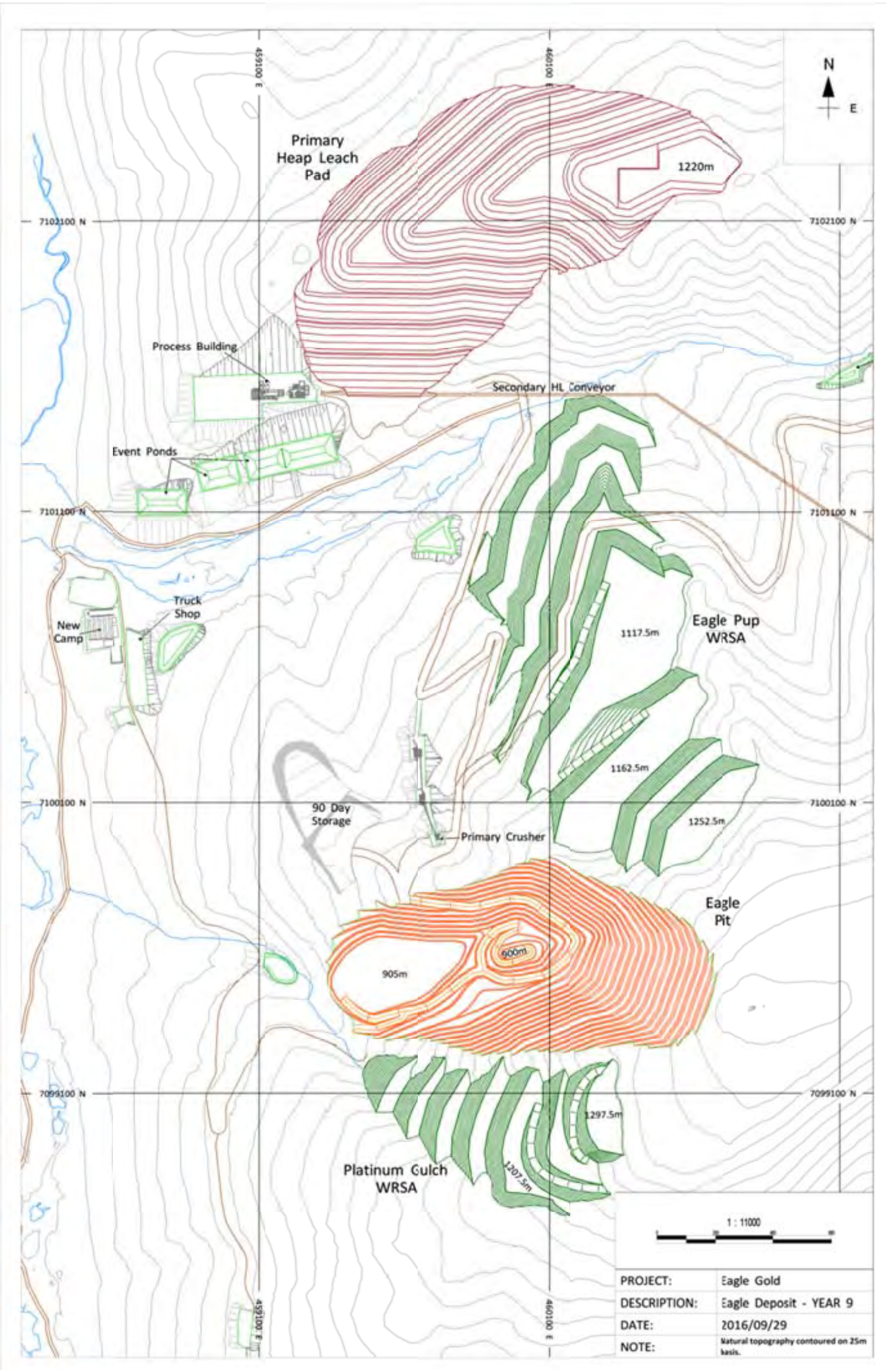


Figure 16.14: Olive Deposit Annual Map Year 9

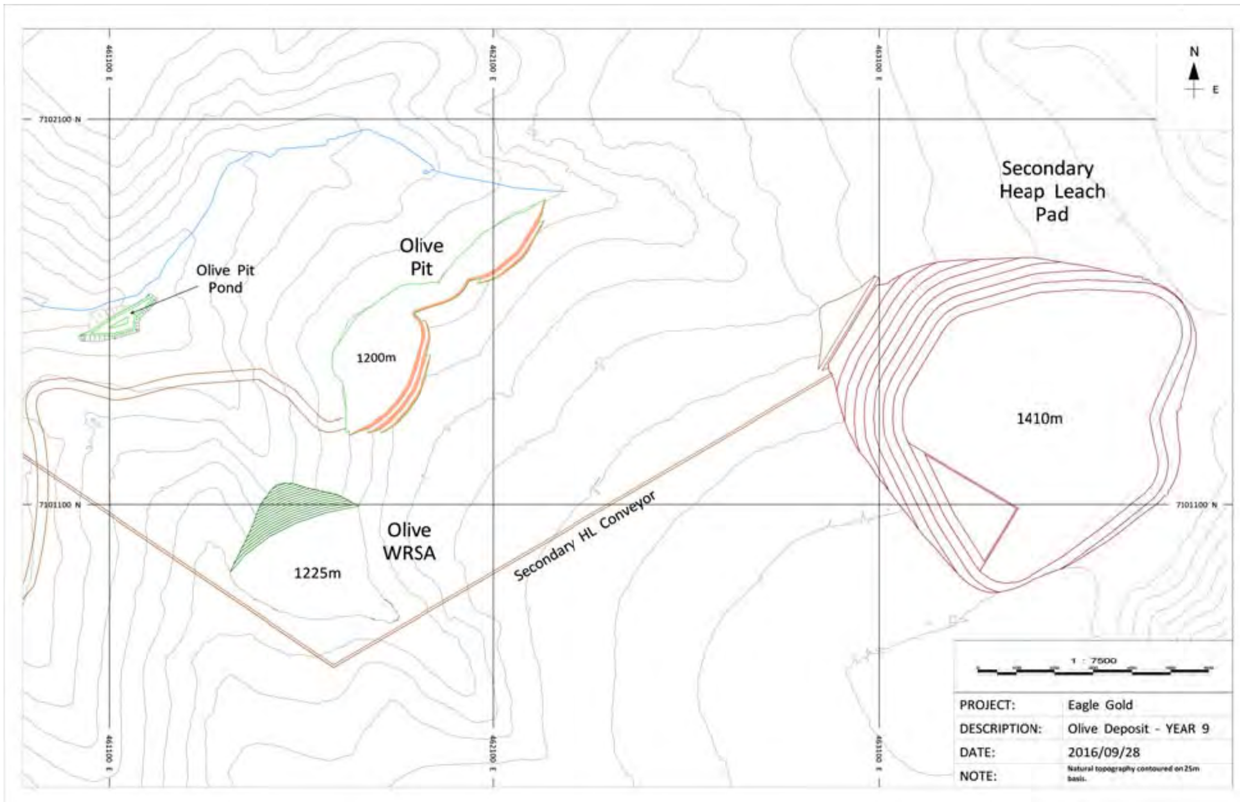


Figure 16.15: Eagle Deposit Annual Map Year 11 – End of Mine Life

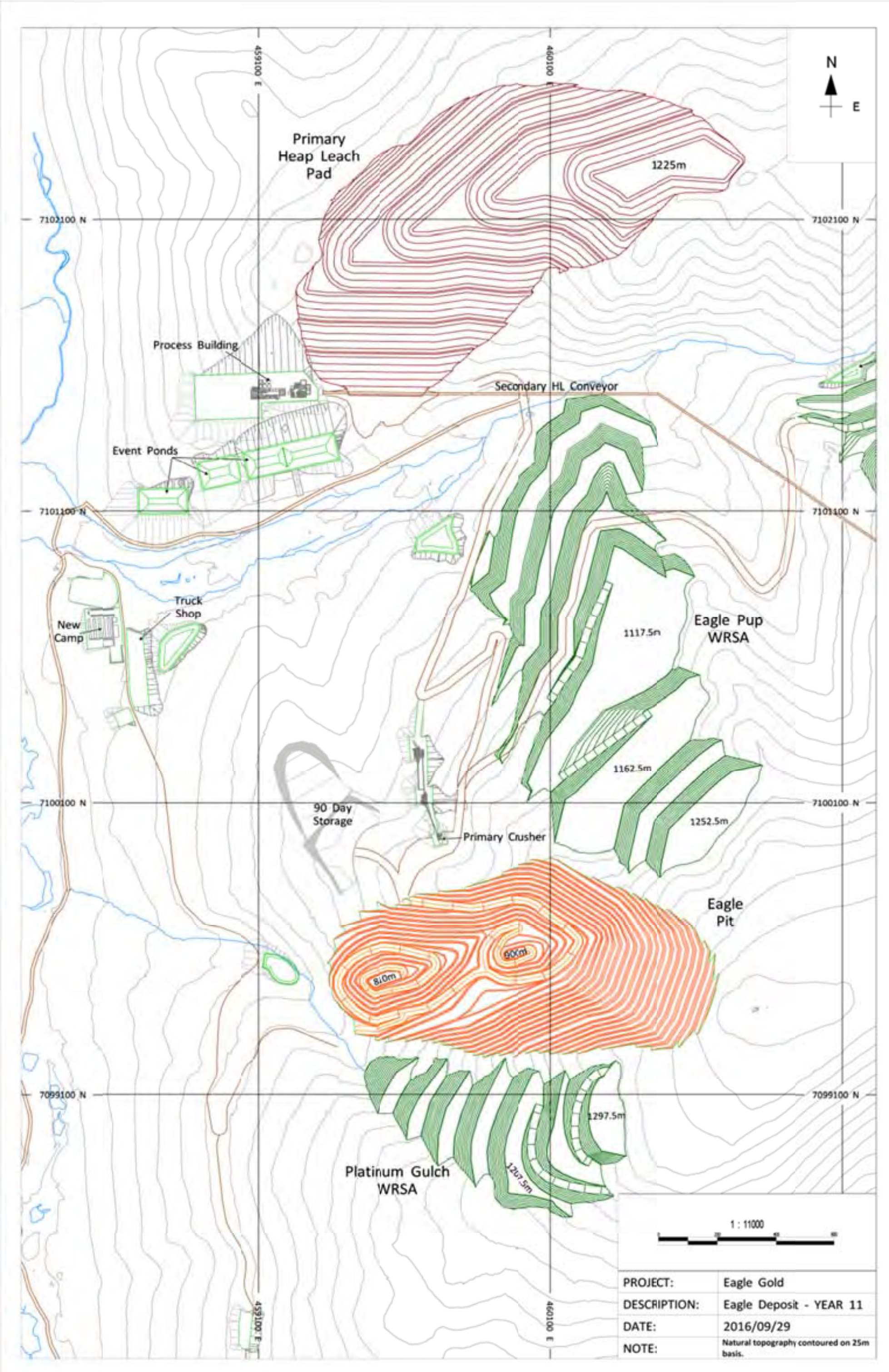
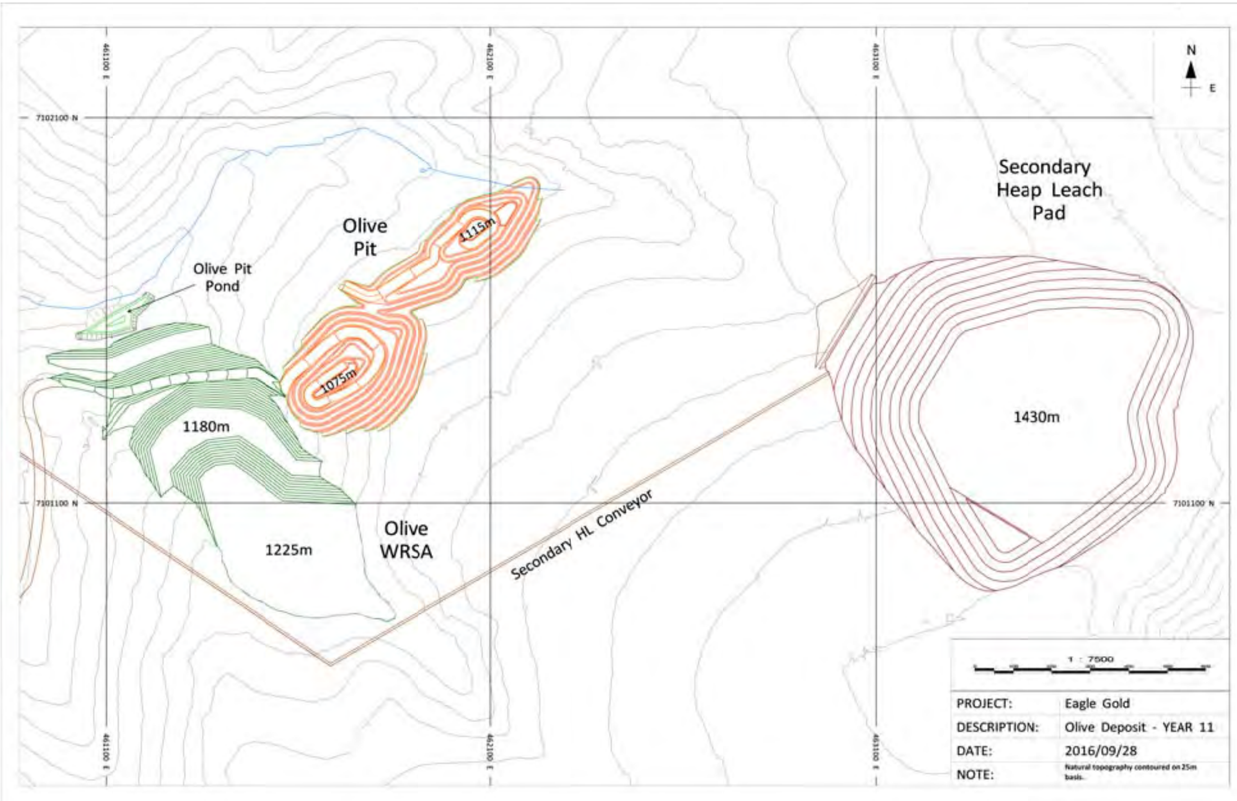


Figure 16.16: Olive Deposit Annual Map Year 11 – End of Mine Life



16.2.8.5 Waste Rock Scheduling

16.2.8.5.1 Waste Rock Scheduling

Eagle waste rock will be hauled to one of two waste rock storage areas immediately to the south (Platinum Gulch WRSA) and north (Eagle Pup WRSA) of the open pit which results in short haul distances. Olive waste rock will be hauled to a waste rock storage area immediately south-west of the open pit (Olive WRSA).

Total waste material removed from the pits totals 116 Mt. The Platinum Gulch WRSA has a design capacity of 21 Mt and Eagle Pup 78 Mt. The Olive WSRA is designed for 17 Mt. Suitable waste rock will be used for road construction, lay down areas, as well as the base of the HLP.

Each WRSA is planned to be constructed in a bottom-up approach by placing material at its natural angle of repose (approximately 1.5H:1V) with appropriate catch benches spaced approximately every 45 m vertically resulting in final slopes of 2.5:1. A 30% swell factor is assumed. Segregation of the various waste material types, if deemed necessary, will be managed given the extent of the various WRSA designs.

The Platinum Gulch WRSA is planned to have an ultimate crest elevation of 1,298 masl, a maximum height of 345 m and a footprint of 33 ha. The larger Eagle Pup WRSA has a footprint of 94 ha, an ultimate crest elevation of 1,250 masl with a maximum height of 315 m. Olive WRSA ultimate crest elevation is 1,225 masl for a maximum height of 165m and a footprint of 32 ha.

Table 16.8 summarizes annual waste volumes allocated to the various WRSAs.

Table 16.8: **Annual Waste Allocations by Destination (in millions of cubic metres)**

	Year											
Destination	**Y- 1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**	**Y10**	**Y11**
Platinum Gulch WRSA	0.8	1.3	7	1.2								
Eagle Pup WRSA	0.2	2.1		4.7	7.7	5	3.9	3.9	4.6	6	1.1	
Olive WRSA										2.1	6.1	0.1
TOTAL	**1**	**3.4**	**7**	**6**	**7.7**	**5**	**3.9**	**3.9**	**4.6**	**8.1**	**7.2**	**0.1**

Source: JDS (2016)

16.2.9 Mine Equipment Selection

16.2.9.1 Introduction

The open pit mining activities for the Eagle Gold project were assumed to be undertaken by an Owner-operated fleet with conventional drill, blast, load and haul, considering bulk excavation of waste using hydraulic excavators, and bulk-selective loading of ore using hydraulic excavators or a front-end loader. Given the overall scale of operations and equipment requirements, a diesel-powered fleet has been selected.

Supplier names and equipment types are provided for reference purposes only. Reference to particular machine types does not reflect a final recommendation of equipment supply; rather, further analysis will be carried out at the detailed engineering and procurement stages of the project.

Used mining equipment has been assumed for some of the equipment types. Results of a search in the current market for available units with low hours indicate that used equipment is readily available.

16.2.9.2 General Operating Parameters

The open pits are designed with 10 m benches in both waste and ore headings with adequate phase geometry to achieve a maximum production rate of 29 Mt/year. Given the required production rate and pit geometries, vertical advance rates average eight benches per year, with frequent requirement for ramp development and opening of new benches.

Equipment effective utilizations were based on vendor recommendations, cost services, factors and JDS experience. Effective utilizations are 65% for the drilling equipment, 57% for the loading equipment, 61% for the hauling equipment, and 65% for support and auxiliary equipment.

16.2.9.3 Blasthole Drilling and Blasting

Based on the selected bench height (drilling will occur on 10 m high benches) and the production schedule requirements, a production drill with a 229 mm hole diameter was selected for waste and 190 mm hole diameter in ore.

To ensure the recommended bench face angles (BFA) and interramp angles (IRA) are met, it was assumed that 3% of the total material to be drilled would be pre-split with a smaller drill. Operating costs were included to cover the additional cost of this small-diameter (120 mm) drill.

The blast design assumes the use of a 65% ANFO (ammonium nitrate/fuel oil) / 35% Emulsion blend for dry holes, and a 30% ANFO / 70% Emulsion blend for wet holes. Given the climatic conditions of the project area, 20% wet blast holes were assumed. An explosives supplier is planned to be contracted to provide ANFO and blasting accessories. The contractor would also supply the explosives plant and mixing equipment.

16.2.9.4 Loading

Diesel hydraulic excavators were selected as the primary loading equipment, supported by front-end loaders (FEL) and a smaller hydraulic backhoe. The main criterion for loading equipment selection is the ability to effectively load trucks with payloads of 144 t, while allowing for somewhat selective mining. As such front shovels with a 22 m^3 bucket will primarily undertake the mining of ore and waste material, while the 12 m^3 FELs and smaller excavator will complement the main shovel fleet (e.g. lower, confined benches of the open pits).

16.2.9.5 Hauling

The truck fleet for the project was selected to match the selected loading fleet, and resulted in the selection of trucks with a payload of 144 t. Haulage profiles were estimated for the mine plan for every bench over the mine life and for each material type (waste/ore). Requirements for haulage of ROM ore to the primary HLP were also accounted for. Runge Talpac software was used to determine truck requirements and productivities.

16.2.9.6 Support and Auxiliary Equipment

The support and auxiliary equipment selection was made considering the size and type of the primary loading and hauling fleet, the geometries of the various open pits, and the number of roads and WRSAs that would be in operation at any given time.

The type of equipment selected was based on vendor recommendations as well as JDS experience in similar sized operations. The auxiliary equipment fleet is planned to comprise track dozers (Komatsu D375-class), rubber tire dozers (Komatsu WD600-class), graders (Komatsu GD825-class) and water trucks (90 m^3).

The following items were also included in the list of support equipment:

- Fuel trucks for the supply of diesel fuel to all the hydraulic diesel excavators, dozers, and drills;
- Lube truck for the supply of lubricants, hydraulic fluids, cooling water to all open pit equipment;
- Mobile mechanical trucks for preventative and corrective maintenance conducted in the field;
- Low-boy transporter trailer (100 t weight capacity) for transportation of dozers, drills, small back hoe and major equipment components;
- Light vehicles for supervisors/technical personnel; and
- Mobile lights for lighting of pits, waste dumps and construction areas.

16.2.10 Mine Equipment Requirements

16.2.10.1 Summary

An annual summary of the open pit fleet requirement is shown in Table 16. 9. In terms of equipment replacements, equipment suppliers provided estimates for equipment life, and where information was lacking, industry standards and JDS experience were used. Given the 10-year mine life, it is estimated that limited replacements will be necessary; these include the FEL's, track dozers, wheel dozer and grader.

All open pit equipment maintenance on-site will be carried out with Victoria Gold personnel using the company's own installations. Work on-site would consist of mainly preventative maintenance and major component exchange. Given the estimated mine life, no major rebuilds for new equipment are anticipated. However, should they be required, it is anticipated that they would be performed on-site by contractors. For the initial used haul trucks, a major rebuild has been incorporated.

Table 16.9: Open Pit Mine Primary Equipment Requirements

Type	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
D50KS Drill (152 – 229 mm)	1	2	3	3	3	3	3	3	3	3	3	1
DP1500i Drill (89 - 152 mm)	1	1	1	1	1	1	1	1	1	1	1	1
Komatsu HD1500 Truck (144t)*	3	11	11	11	11	11	11	11	11	11	11	2
Komatsu PC4000 Shovel (22m^3)	1	2	2	2	2	2	2	2	2	2	2	1
Komatsu PC800 Excavator (4.5m^3)	1	1	1	1	1	1	1	1	1	1	1	1
Komatsu WA900 Wheel Loader (11.5m^3)	1	1	2	2	2	2	2	2	2	2	2	1
Komatsu D375 Track Dozers	2	3	4	4	4	4	4	4	4	4	4	1
Komatsu WD600 Wheel Dozer	1	1	1	1	1	1	1	1	1	1	1	1
Komatsu GD825 Grader	1	2	2	2	2	2	2	2	2	2	2	2
Water Truck (90m^3)	1	1	1	1	1	1	1	1	1	1	1	1

*Note: Haulage fleet will require the addition of a few rental units sporadically through the LOM for short durations
Source: JDS (2016)

16.2.11 Mine Personnel and Organization Structure

16.2.11.1 Basis

The work schedule assumes a 24-hour/day, 7-days/week and 365-days/year mining operation. Operations and maintenance personnel will work two 12-hour shifts per day. Production, maintenance and technical services personnel are planned to be on a 2-week in/2-week out rotation.

With the exception of the blasting crew, all hourly labour and supervisory personnel will rotate between day and night shifts. Management and technical staff will work the day shift only, with the exception of grade control technicians, who share the same shift rotation as the production crews.

Equipment operator labour requirements are based on the number of equipment units, operating requirements and shift rotations. Maintenance labour requirements are based on the number of equipment units to be maintained, estimates of mechanical availability, and estimates on the ratio of maintenance labour requirements to the number of units for each open pit fleet type.

16.2.11.2 Personnel Activities

The mining operation will be headed by the mine superintendent, who will report to the general manager.

Under the direction of the mine superintendent, the mine operations department will be responsible for the mining operation. This includes drilling, blasting, loading, and hauling of ore and waste, WRSA operations, haul road construction and maintenance, and mine dewatering. Each crew will be led by a mine shift foreman.

The mine maintenance department, responsible for maintaining all open pit mine mobile equipment, will report to the mine maintenance superintendent. Maintenance crews are planned to work the same shift schedule as the production crews. Each maintenance crew will be led by a maintenance shift foreman. A mine operations and maintenance general foreman is planned. The engineering department will be led by the chief mining engineer and will be responsible for providing short, medium and long term mining plans.

The geology department under the chief geologist will be responsible for updating the resource models, calculating ore resources and reserves, and undertake ore grade control.

Annual personnel requirements are summarized in Table 16.10.

Table 16.10: Annual Personnel Requirements

Description	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y10	Y11
Mine Operations												
Drillers	4	12	12	12	12	12	12	12	13	13	14	4
Blasters	2	2	2	2	2	2	2	2	2	2	2	1
Blasting helpers	2	2	2	2	2	2	2	2	2	2	2	1
Shovel/loader operators	4	12	14	13	13	12	12	13	13	15	14	4
Haul truck drivers	9	40	44	28	42	28	28	36	36	53	38	8
Track dozer operators	8	8	11	11	11	11	11	11	11	11	11	4
R.T. Dozer Operator	4	4	3	3	3	3	3	3	3	3	3	3
Grader operators	5	6	6	6	6	6	6	6	6	6	6	6
Water/ancillary truck drivers	2	2	2	2	2	2	2	2	2	2	2	1
Labourer/trainees	2	4	4	4	4	4	4	4	4	4	4	4
Subtotal Mine Operations	**42**	**92**	**100**	**83**	**97**	**82**	**82**	**91**	**92**	**111**	**96**	**36**
Mine Maintenance												
Heavy equipment mechanics	4	6	6	6	6	6	6	6	6	6	6	4
Welders/mechanics	4	6	6	6	6	6	6	6	6	6	6	4
Electricians/instruments	2	4	4	4	4	4	4	4	4	4	4	2
Lube/PM mechanics/light duty mech.	4	6	6	6	6	6	6	6	6	6	6	2
Tiremen	3	4	4	4	4	4	4	4	4	4	4	2
Labourers/trainees	2	4	4	4	4	4	4	4	4	4	4	2
Subtotal Mine Maintenance	**19**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**16**
Technical/Supervisory												
Mine superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance superintendent	1	1	1	1	1	1	1	1	1	1	1	-
Mine shift foremen	6	12	12	12	12	12	12	12	12	12	12	4
Maintenance planner	1	1	1	1	1	1	1	1	1	1	1	-
Maintenance shift foremen	4	4	4	4	4	4	4	4	4	4	4	4
Chief mining engineer	1	1	1	1	1	1	1	1	1	1	1	1
Senior mine engineer	1	1	1	1	1	1	1	1	1	1	1	-
Mine engineers	2	3	3	3	3	3	3	3	3	3	3	2
Mine technicians	1	2	2	2	2	2	2	2	2	2	2	-
Surveyors	2	2	2	2	2	2	2	2	2	2	2	1
Survey assistants	2	2	2	2	2	2	2	2	2	2	2	1
Mine clerk	2	2	2	2	2	2	2	2	2	2	2	-
Chief geologist	1	1	1	1	1	1	1	1	1	1	1	-
Mine geologists	2	3	3	3	3	3	3	3	3	3	3	-
Technicians/ore control	1	2	2	2	2	2	2	2	2	2	2	-
Subtotal Technical/Supervisory	**28**	**38**	**38**	**38**	**38**	**38**	**38**	**38**	**38**	**38**	**38**	**14**
Total Personnel	**89**	**160**	**168**	**151**	**165**	**150**	**150**	**159**	**160**	**179**	**164**	**66**

Source: JDS (2016)

16.3 Pit Slope Geotechnical Analysis and Recommendations

Pit slope geotechnical design criteria were developed by SRK Consulting (U.S.), Inc. (SRK). A significant amount of geotechnical characterization and analyses were conducted as part of the Wardrop 2012 FS. This previous work was reviewed by SRK and served as the basis for the FS update analyses and design recommendations. No additional field data collection or laboratory testing were completed as part of this FS update.

16.3.1 Slope Stability Analyses

As part of the FS update, opportunities were identified that could advance the previous 2012 geotechnical design parameters, potentially resulting in less conservative slope angles and reduced stripping. Detailed probabilistic bench design analyses were conducted incorporating the natural variability in discontinuity properties, through statistical distributions that were defined, based primarily on the original BGC Engineering (BGC) (2012a) discontinuity characterization and strength testing information. The probabilistic analyses demonstrated that less conservative bench design parameters could be used than originally indicated in the previous 2012 FS.

Bench design analyses were accomplished using the software program SBlock (Esterhuizen, 2004) and an acceptability criteria of a maximum probability of failure of 30%, and a minimum catch bench width equal to approximately the 80th percentile cumulative catch bench width (i.e. 80% reliability). The primary conclusions from the analyses follow:

- Eagle and Olive final pit designs are not anticipated to be controlled by structural instabilities based on the structural trends identified to date. A maximum achievable bench face of 70° was assumed for the design based on data uncertainties and operational constraints;
- The highest risk of bench and possibly low interramp-scale instabilities at Eagle pit is anticipated to be in the southwest corner of the pit (northeast dipping walls) in the intrusive rock mass. Walls in this area and orientation are anticipated to have a high susceptibility to planar and wedge instabilities formed by the intersection of two discontinuity sets. However, walls oriented in this direction represent a very small portion of the overall pit design;
- West dipping bench faces in the Eagle pit metasedimentary rocks are anticipated to be controlled by the dominant foliation discontinuities, as well as potential wedges formed by the intersection of two discontinuity sets. A maximum bench face angle of 60° was determined to be appropriate for this section based on the analyses; and,
- Achievable bench face and lower interramp slope angles in the northwest Olive pit intrusive rock are anticipated to be controlled by the intersection of various combinations of two or more discontinuity sets. Although theoretically controlled by geologic structure, analysis results still indicate achievable bench face angles of equal to or greater than the maximum bench face angle criteria of 70° assumed for the project.

Bench stability analyses are based solely on geologic structure and do not directly consider effects of weathering, alteration, blasting or excavation techniques. Depending on the quality of blasting and excavation techniques, achievable BFA might be reduced from the theoretical angles determined by these analyses. When taking these operational effects into consideration, it is rare to achieve effective BFA greater than about 70° to 75°, unless there is a steeper structure controlling the bench geometry.

Increasing BFA to greater than about 70° to 75° may be achievable in some areas of the pit, but usually requires more rigorous drilling and blasting effort, and specialized controlled blasting techniques, than are commonly practiced.

Limit equilibrium slope stability analyses were then conducted using Slide (Rocscience, 2015) to confirm stability of the high interramp/overall slope angles that resulted from the detailed bench design criteria alone. Rock mass shear strengths were developed for each rock type based largely on the BGC (2012a) investigation and assuming the Hoek-Brown (Hoek, et al., 2002) rock mass shear strength criteria. Results of the overall slope stability analyses for the bench configuration based slope angles indicate safety factors between 1.3 and 1.8 for Eagle pit, and between 1.3 and 3.4 for Olive pit, which either meet or exceed the minimum acceptable safety factor of 1.3 for static loading conditions.

16.3.2 Pit Slope Geotechnical Design Criteria

Recommended pit slope design parameters are summarized in Table 16.11 and shown graphically on Figure 16.17. The recommendations in Table 16.11 are based on the dip direction of the pit wall (e.g. for an east-west trending, south facing pit wall, the slope dip direction would be 180° azimuth).

The recommendations for Eagle pit are based on full depressurization occurring to a minimum distance of 125 m behind the pit wall, as was recommended by BGC as part of the previous 2012 FS. BGC (2012a and 2014) recommends the installation of 250 m horizontal drains to accomplish the 125 m depressurized zone and provides additional specifications for horizontal drain construction and installation. Mine planning and costing for the FS update have considered the installation and operation of these drains.

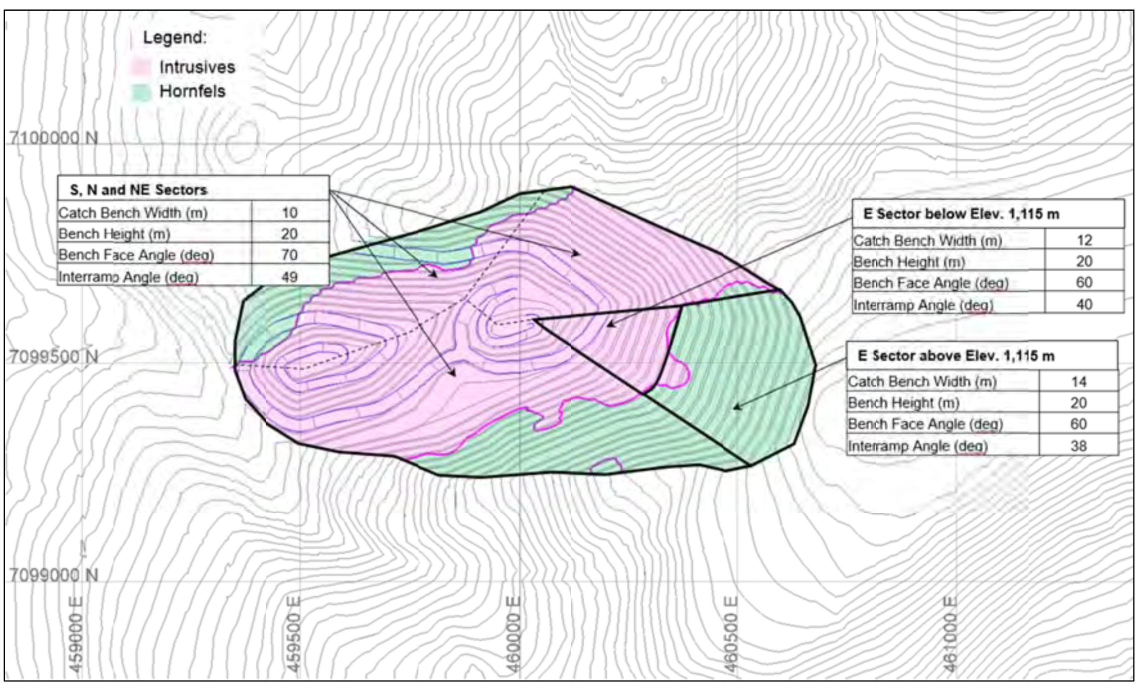
Table 16.11: Recommended Pit Slope Design Parameters

Pit	Sector	Max. Slope Height (m)	Wall Dip Direction		Bench Face Angle (°)	Bench Height (m)	Bench Width (m)	Max. ISA[1] (°)
			From (°)	To (°)				
Eagle	North	225	130	200	70	20	10	49
	Northeast	280	200	265	70	20	10	49
	East (above elev. 1,115 m)	280	265	305	60	20	14	38
	East (below elev. 1,115 m)	210	265	305	60	20	12	40
	South	375	350	85	70	20	10	49
Olive	Southeast	180	90	260	70	20	10	49
	Northwest	110	260	90	70	20	10	49

[1] ISA indicates Interramp Slope Angle

Source: SRK (2016)

Figure 16.17: Pit Slope Design Recommendations



Source: SRK (2016)

16.3.3 Recommendations for Additional Geotechnical Work

Structural geology of the project is not well understood beyond major mineralization controlling structures. Major geologic structures not currently included in the 3D structural model or whose existence is unknown may adversely impact mine stability. SRK recommends additional structural geology work be completed to develop a pit-scale 3D structural model. It is anticipated that this may be accomplished based primarily on the existing drill hole database with minimal, if any, additional drilling required. As part of the structural geologic interpretation, additional work should also be conducted to evaluate the potential for large scale faults paralleling foliation in the Eagle pit metasediments and further delineate the spatial extent of the clay-altered intrusives described by BGC (2012a).

A thorough geological and geomechanical bench face mapping program should be undertaken beginning in the early stages of development to verify that the geologic structural conditions encountered are consistent with the assumptions and estimates used in the analyses, and to identify local variations in structural conditions that might increase the risk of instabilities. The data collection should concentrate on developing a geotechnical database that will facilitate further refinement of the bench design and optimization of interramp and overall slope angles. Particularly important information will include discontinuity persistence, spacing and variations in orientation as well as assessments of blast performance.

A slope monitoring program should be designed and implemented to ensure that slopes are behaving as anticipated and provide sufficient warning should movements occur. The monitoring program should include, at a minimum, a network of survey prisms monitored that are analyzed regularly.

16.4 Waste Rock Storage Area Geotechnical Analysis and Recommendations

Geotechnical investigation and design of the Eagle pit WRSAs (Eagle Pup and Platinum Gulch) was previously carried out by BGC (2012b, 2012c and 2012d) as part of the Wardrop 2012 FS. The work program consisted of field and laboratory characterization, slope stability analyses and provision of design recommendations. The previous work was reviewed by SRK and served as the basis for the FS update analyses and design recommendations. No additional field data collection or laboratory testing were completed as part of this FS. A complete description of the geotechnical investigations and properties can be found in BGC (2012b, 2012c and 2012d). A summary of this work is presented below.

16.4.1 Foundation Conditions

A veneer of organics is widespread across the project site and typically ranges between 0.2 m and 0.3 m deep. Colluvium soils and completely weathered bedrock underlie the organics up to approximately 10 m in depth. The overburden soils are variable in composition ranging from boulders and cobble with silt and sand, to silty sand with gravel and cobble.

Frozen ground, often containing excess ice, has been identified at various locations within the proposed WRSA footprints. Depending on the thickness, initial temperature, and timing of the waste rock placement, the thermal regime of the initially frozen foundation may be altered during construction. Additional delineation of potentially thaw-unstable soils will be required at the detailed design level of the project. Depending on their extent and ice-content, such soils may require removal from the WRSA footprints and storage in the ice-rich overburden material storage area.

16.4.2 Geotechnical Design

Design acceptability criteria for the stability analyses were based on the “Mined Rock and Overburden Piles Investigation and Design Manual” prepared by the BC Mine Waste Rock Pile Research Committee (1991). It is recommended in the manual that, under static loading conditions, a minimum factor of safety of 1.3 be achieved for short term developments (e.g. during mine operations), and that a minimum factor of safety of 1.5 be applied to the long term (e.g. closure) stability of the WRSAs. Under pseudo-static seismic loading conditions, it is recommended that a minimum factor of safety of 1.1 be achieved. Using the BGC (2012d) material properties, SRK has confirmed that the safety factors of the final FS WRSA slopes meet or exceed the minimum acceptability criteria.

The Eagle Pup WRSA construction sequence was developed as recommended by BGC (2012d) buttressing the ice-rich lobate feature in the upper Eagle Pup WRSA valley to increase stability of the lifts built above the feature, should rapid melting of the feature occur. All three facilities will have rock drains constructed beneath that have been designed to handle the 200-year precipitation event.

16.4.3 Recommendations for Additional Geotechnical Work

- The presence of ice-rich soils in WRSA foundations presents risk of creep movements and/or WRSA slope instability. Additional geotechnical investigation should be conducted at the detailed design level of the project to further delineate the ice-rich soils in areas where final and interim toes of the WRSAs will be located.
- The suitability of metasedimentary and intrusive rock excavated during construction as rock drain materials requires evaluation. The susceptibility of these materials to mechanical degradation should be considered. The particle size distribution of the various waste rock sources should be better defined to confirm assumptions regarding proposed materials for rock drain construction.
- A geotechnical investigation must be undertaken at the Olive WRSA during the detailed design phase of the project to confirm similar foundation properties as encountered at the Eagle Pup and Platinum Gulch facilities by BGC (2012d). To date, no geotechnical investigations have been carried out at the Olive WRSA.
- The physical stability of the individual WRSA phases will require evaluation during the project detailed design phase to verify that a 1.3 minimum safety factor is met for each phase. Minor adjustments to the slope angles and toe locations may be necessary; however, major revisions to the phase designs or are not anticipated.
- Consideration should be given to leaving the organics layer in-place except where excavation of soils and bedrock beneath is required. Clearing and grubbing of the organics may cause degradation of permafrost in the soils beneath which may result in poor working conditions, thawing of the permafrost and increased pore water pressures.

17 Process Description/Recovery Methods

This section describes the recovery methods used for the Eagle Gold project for the crushing, heap leach and process facilities. Flowsheet development, operating parameters and design criteria were based on results from metallurgical test work presented in Section 13. The gold recovery process was designed on the basis of leaching approximately 12.5 Mt of ore per year with an average gold head grade of 0.67 g/t (ROM and crushed ore combined) at an overall gold recovery of 70.8%.

The three-stage crushing plant will operate at a nominal primary crushing rate of 30,000 t/d, 365 days per year and a secondary and tertiary crushing rate of 39,800 t/d, 275 days per year. During the coldest part of the year (January through March), fine crushing and HLP loading activities will be suspended. Barren solution, made up of a cyanide-caustic mixture, will be pumped at a nominal rate of 2,070 m^3/h to a network of supply piping and drip emitters on the HLP. Pregnant solution will be collected in a sump near the bottom of the pads and pumped to the 8 t/d ADR plant for gold extraction and the production of gold doré.

The gold ore processing facilities will include the following unit operations:

Crushing and Ore Handling

- Primary crusher: a vibrating grizzly screen and gyratory crusher in open circuit, producing a final product P_{80} of approximately 114 mm;
- Secondary crusher: a vibrating screen and cone crusher operating in open circuit, producing a final product P_{80} of 22 mm;
- Tertiary crushers: three vibrating screens and three cone crushers operating in reverse closed circuit, producing a final product P_{80} of 6.5 mm, and;
- Heap placement: crushed material will be conveyed to the heap leach pad (HLP) by overland conveyor.

Heap Leach Pad

- Crushed ore stacking and spreading by a series of grasshoppers to a radial stacker;
- Ore leaching; and
- Barren and pregnant solution delivery and recovery piping systems.

ADR Plant

- Carbon-in-Column (CIC) Adsorption: adsorption of solution gold onto carbon particles;
- Desorption: acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold-rich solution, carbon stripping to recover gold into solution and thermal regeneration of carbon to remove organic foulants; and
- Gold recovery: gold electrowinning (sludge production), filtration, drying, mercury retorting, and smelting to produce gold doré.

A process flowsheet and process plant layout are presented in Figure 17.1 and Figure 17.2.

Figure 17.1: Process Flowsheet

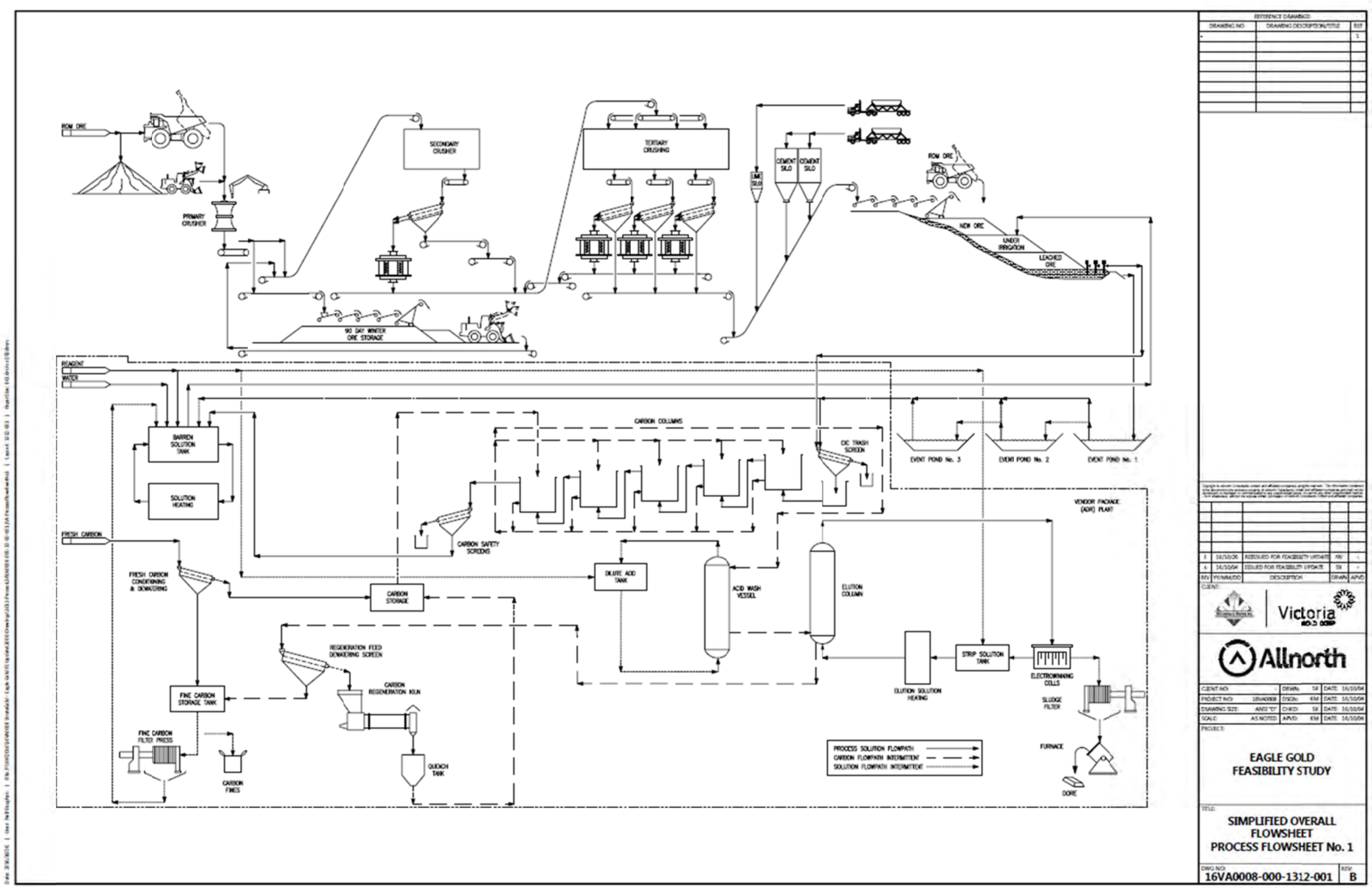
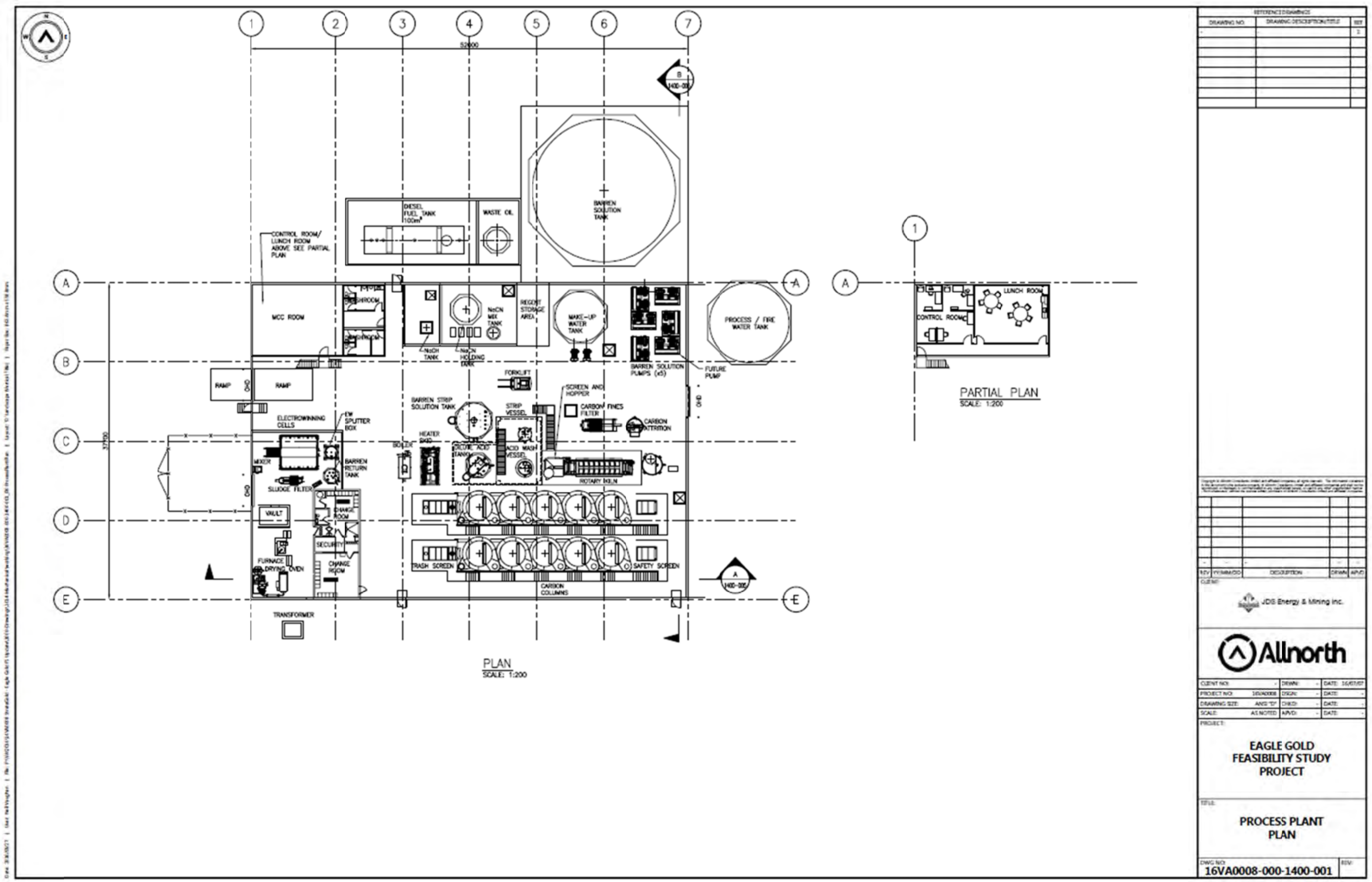


Figure 17.2: Process Plant Layout



17.1 Process Design Criteria

The process design criteria and mass balance detail the annual ore production, major flows, and plant availability. The key process design criteria are summarized in the Table 17.1.

Table 17.1: Process Design Criteria

General	Unit	Value
ROM Annual Treatment Rate	t/y	10,767,500
Crushing Plant Operation - Primary	d/y	365
	t/d	30,000
	t/h	1,500
Crushing Plant Operation – Secondary and Tertiary	d/y	275
	t/d	39,800
	t/h	1,990
Crushing Plant Operation	h/d	20
Heap Loading and Spreading Method	-	Grasshoppers and Radial Stacker
Heap Loading Operation	d/y	275
Crushed Ore - Heap Loading Operation	t/d	39,800
Crushed Ore - Heap Loading Operation	h/d	20
Crushed Ore - Design Rate Heap Loading Operation	t/h	1,990
Average LOM Feed Grade	g /t Au	0.69
Overall LOM Recovery	%	70.9
Ore Characteristics		
Specific Gravity (Average)	t/m^3	2.70
Dry Crushed HL feed Bulk Density	t/m^3	1.7
ROM Moisture	%	5
Bond Crusher Work Index (Oxide)	kWh/t (Eagle)	6.9
Abrasion Index (Oxide)	g (Eagle)	0.218
Lime Consumption	kg/t ore	1.0 to 1.5
Cement Consumption	kg/t ore	0 to 6.0
Cyanide Consumption	kg/t ore	0.35
Crushing		
Days per Week	d	7
Days per Year - Primary	d	365
Days per Year – Secondary and Tertiary	d	275
Shifts per Day	shifts	2
Shift Length	h	12
Crusher Availability	%	85
Hours per Day	h	20
Primary Crusher		
Type		Gyratory
Size	-	MK-II 50-65 or equiv.
Closed Side Setting	mm	140
Motor	kW	365
Product Size, 80% Passing	mm	114
Secondary Crushing Feed Conveyor		
Type		Belt (Covered)
Width	mm	1,372
Motor	kW	149
Secondary Crusher Screen		
Type		Vibrating, Double Deck
Size	mm	2,400 x 6,100
Screen Deck Aperture	mm	30/75
Motor	kW	55
Secondary Crusher		
Type		Standard Medium Cone Crusher

General	Unit	Value
Size		MP1250 or equivalent
Motor	kW	933
Closed Side Setting	mm	35
Tertiary Crushing Feed Conveyor		
Type		Belt (Covered)
Width	mm	2134
Motor	kW	373
Tertiary Crusher Screen		
Number		3
Type		Vibrating, Double Deck
Size	mm	4,200 x 8,500
Screen Deck Aperture	mm	18-Oct
Motor	kW	93
Tertiary Crusher		
Number		3
Type		Standard Fine Cone Crusher
Size		MP1250 or equivalent
Motor	kW	933
Closed Side Setting	mm	12
Overland Conveyors to Pad 1		
Type		Belt (Covered)
Width	mm	1,219
Motor	kW	264
Feed Size, 80 % Passing	mm	6.5
Leach Pad		
Ultimate Design	mt	123,000,000 (nominal)
		77,000,000 (Primary HLP)
		46,000,000 (Secondary HLP)
Slope Stability, factor of safety		1.3 static
		1.0 pseudo-static
Ultimate Height, toe to crest	m	150 (Primary HLP)
		120 (Secondary HLP)
Lift Height	m	10 (nominal)
Heap Slope, Overall	h:v	2.5:1
Leach Cycle, Primary	d	90
Solution Application Rate	$l/hr/m^2$	10
Solution Flow Rate	m^3/h	2,070
Area Under Leach	m^2	200,000
Ponds and Diversion Channels for Heap		
Design storm event for peak flow rate, 100-yr, 24-hr	mm	72
Design storm event for pond storage, 24-hr PMP	mm	256
Freeboard in ponds	mm	500
Pregnant Solution	-	Sump with pumping to the Plant
Pumping		
Number of Units Installed		8 (5 future)
Number of Units Operating		3 (5 future)
Type	mm x mm	450 x 760
Motor	kW	186
Barren Solution		
Tank Dimensions	m x m	17.5 x 17.5
Pumping		
Number of Units Installed		10 (7 future)
Number of Units Operating		2 (9 future)

General	Unit	Value
Type		Krebs millMAX-e™ Model 20x18-47 or equiv.
Motor	kW	932
Barren Solution Heating		
Type	-	Boiler (Diesel-fired)
Years Required	y	1 and 7
Carbon-in-Columns		
Quantity per Train	-	5
Number of Trains	-	2
Tank Dimension	m x m	4.6 x 5.2
Capacity per Train	m^3/h	1,037
Carbon Acid Wash		
Power - Pumps	kW	11
Carbon Capacity	mt	8
Dilute Acid Tank Dimensions	m x m	2.2 x 2.3
Carbon Stripping (Elution)		
Power - Pumps	kW	15
Carbon Capacity	mt	8
Barren Strip Tank Capacity	m^3	24
Barren Strip Tank Dimensions	m x m	3.0 x 4.0
Heat Skid – Strip Solution Heating		
Capacity	M btu/h	3.2
Diesel Fuel Consumption	l/h	12
Electrowinning Cells		
Number of Cells	-	2
Power	kW	15
Capacity	m^3	3.54
Regeneration Furnace		
Capacity	t	5
Power	kW	15
Fuel Consumption	l/h	5.7
Capacity and Dimensions	t (mm x mm x mm)	8 (11,500 x 2,500 x 2,500)
Drying Oven		
Power	kW	35
Doré Furnace		
Type	-	Induction (1 M btu/hr, 660 kg)
Power	kW	100

All equipment sizes and power requirements are approximate
Source: JDS (2016)

17.2 Process Description

17.2.1 Primary Crushing

ROM ore from Eagle and Olive will be trucked from the open pits and dumped directly into a primary feed hopper. The primary crusher, a 365 kW gyratory crusher, will crush ROM material from a maximum feed size of 1,000 mm down to a P_{80} of approximately 114 mm.

The primary crushing plant will operate 365 days per year at a rate of 1,500 t/h. During the winter months, January to March, the crushed material will be conveyed and stacked on the winter stockpile using a series of grasshoppers and a radial stacker. Between April, up to the end of December, the primary crusher product will be fed directly onto the secondary crushing feed conveyor. The material from the winter stockpile will be reclaimed during the summer at a rate of 473 t/h by front-end loader (FEL), and conveyed to the secondary crushing feed conveyor for a combined feed of 1,990 t/h or 39,800 t/d.

If the crushing plant is down, the mine haul trucks will dump onto the ROM stockpile. A FEL will be used to reclaim the ROM material and deliver the material to the dump pocket. The ROM stockpile will also be used to feed the crusher, if the mining operations are suspended.

17.2.2 Secondary Crushing and Screening

Ore from the secondary crushing feed conveyor will be transported to a secondary vibrating double deck screen. Screen undersize material will be conveyed to the tertiary crushing feed conveyor. The screen oversize will feed the 933 kW secondary cone crusher. The secondary cone crusher product will discharge onto the tertiary crushing feed conveyor.

17.2.3 Tertiary Crushing and Screening

Ore from the tertiary crushing feed conveyor will be transported to the tertiary crushing feed bin. The material from the bin will be reclaimed by belt feeders to three tertiary vibrating double deck screens. The oversize material from the screens will feed the tertiary crushers, each installed with 933 kW motors. The crusher product will return to the tertiary crusher feed conveyor. The undersize material, with a target P_{80} of 6.5 mm, will be transferred by overland conveyors to the HLP for stacking, by a series of grasshoppers that feed a radial stacker.

Lime will be added to the stockpile feed conveyor from the 200 t lime silo by screw conveyor for pH control, at a rate of 1 to 1.5 kg/t. Cement for belt agglomeration will be stored in two 300 t silos that will be added at a rate of 2 to 6 kg/t based on ore type. Agglomeration is planned for Year 1 and Year 7, during the initial years of stacking for the primary HLP and the secondary HLP.

17.2.4 Heap Leach Pad

The proposed primary HLP will accommodate approximately 77 Mt of ore and will be located approximately 1.2 km north of the Eagle Zone pit. The primary HLP will be located in the Ann Gulch catchment. The base of the primary HLP confining embankment will be located at an elevation of 880 masl, and at full height in Phase 3 of the primary HLP, the heap will extend up Ann Gulch to an elevation of approximately 1,225 masl at the top of the planned ore stack.

The proposed secondary HLP will accommodate the remaining estimated 46 Mt of ore (with possibly double the capacity potential) and will be located approximately 3 km east of the Eagle Zone pit near the Olive Zone satellite pit. The secondary HLP will be located in the Bawn Boy catchment. The base of the secondary HLP confining embankment is located in the upper portion of the basin at an elevation of 1,300 masl, and at full height in Phase 2, the secondary HLP will extend to an elevation of approximately 1,470 masl at the top of the planned ore stack.

Each HLP comprises a number of elements:

- An earth/rock-filled embankment, to provide stability to the base of the HLP;
- A lined storage area for the ore to be leached;
- A pregnant leach solution (PLS) collection system;
- An in-heap sump for collection and pumping of PLS;
- Events ponds to contain excess solution in extreme events; and
- Leak detection recovery and monitoring systems to ensure the containment of PLS.

The primary HLP and secondary HLP will be constructed in phases with each phase accommodating approximately 25 Mt of ore. The initial phase of the primary HLP will be constructed and operated in Year 1 of the mine plan, and the secondary HLP will be constructed and begin operations during Year 6. The primary HLP and secondary HLP are illustrated in Figure 17.3 and 17.4, respectively.

Figure 17.3: Primary HLP

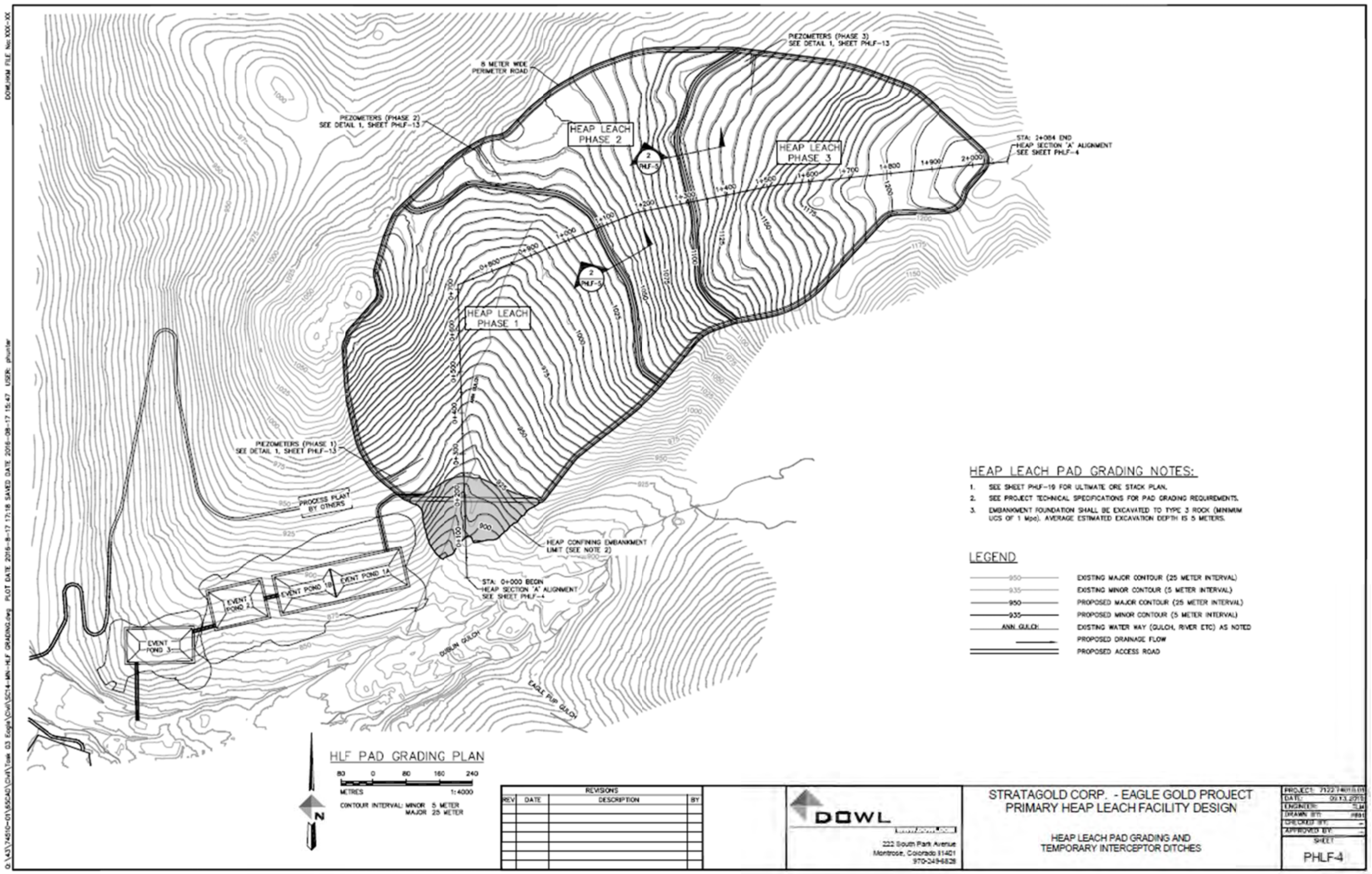
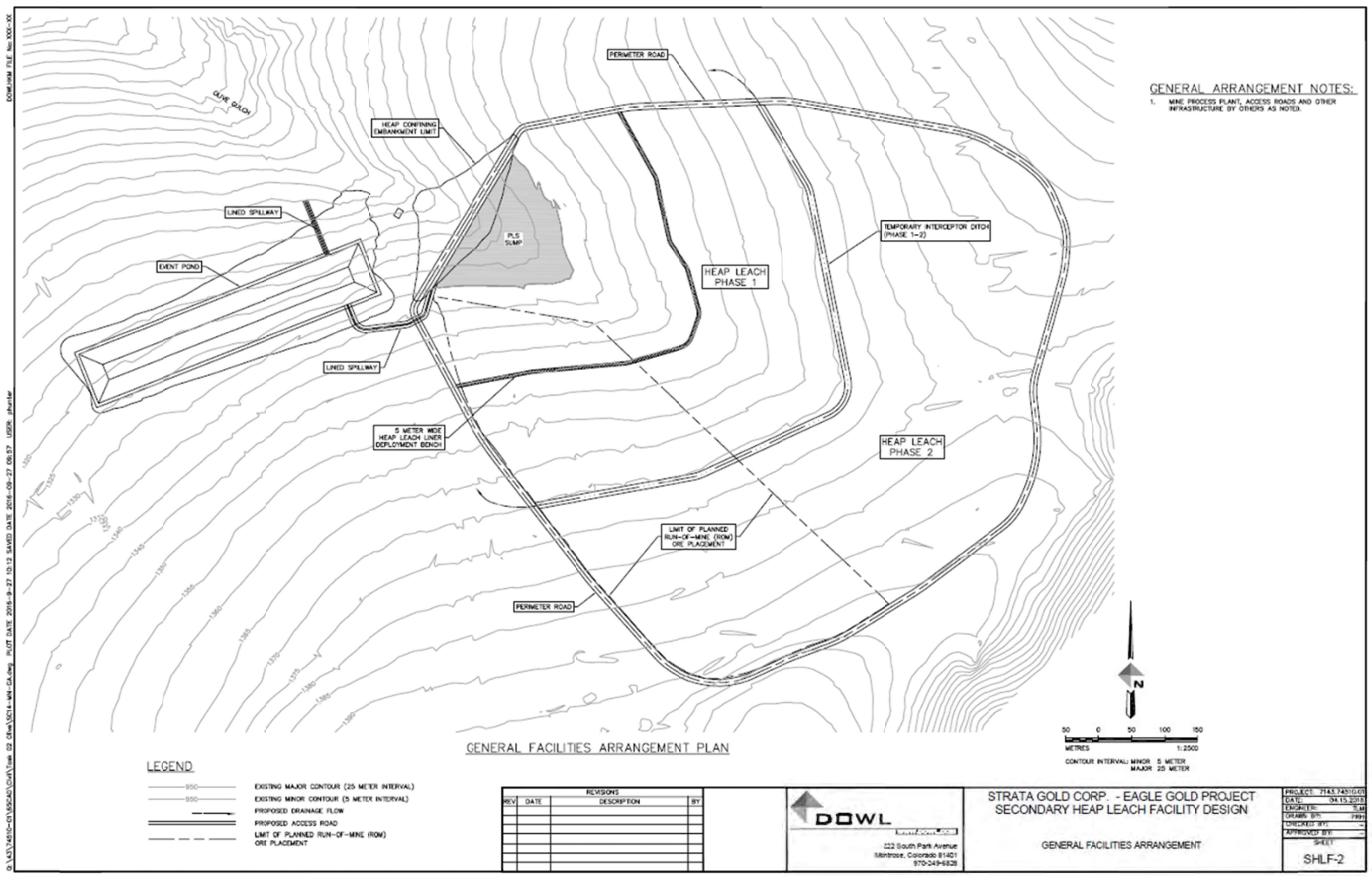


Figure 17.4: Secondary HLP



The liner for the HLPs and events pond will consist of a composite geomembrane and underlying low-permeability bedding material, which is the state-of-practice liner system for heap leach facilities. The primary purpose of the composite liner system is to prevent the loss of PLS for both environmental and economic reasons. In addition to playing a role in preventing leakage, the underliner beneath the geomembrane is necessary as a transition layer between the geomembrane and the prepared foundation.

A geosynthetic clay liner (GCL) will be used in lieu of a 300mm thick layer of compacted low-permeability material due to the lack of suitable on-site soils in sufficient quantities. The GCL soil liner provides an equivalent 300 mm minimum thickness of 1x10-6 cm/sec or lower permeability soil layer.

Free-draining granular material will be placed on top of the pad liner together with a network of collection pipes to collect and drain process solutions and storm infiltration, and to minimize hydraulic heads on the liner, thereby reducing the risk of leakage. Piezometers will be installed within the liner cover fill at the strategic locations to monitor the hydraulic head on the liner system during pad operation.

The PLS sump area to the elevation of the HLP embankment crest will have a double-geomembrane liner installed over a GCL liner together with a leak detection and recovery system (LDRS). The LDRS will be installed between the two geomembranes to monitor and contain any leaks through the top geomembrane.

The events ponds also will be lined with a double-geomembrane liner installed over a GCL liner together with a LDRS. This will allow them to contain excess solutions for short durations until the excess can be taken up by fresh ore.

Temporary runoff interceptor ditches or berms will be constructed for each phase of the HLPs in order to collect storm water runoff from entering the heap. The interceptors will be constructed and in operation before construction of each pad phase. The temporary interceptors will be constructed at the up-gradient limit of each phase of the HLP as the pad liner will tie into the access road adjacent to the ditches. Once the HLP is ready for the next phase, the temporary interceptor ditch will be filled and regraded for placement of the liner for the next phase.

The ditches are sized for the 100-year, 24-hour event, and armoured with riprap. The ditches are backfilled or removed at the end of each phase in order to tie in the HLP liner system and pipework. In the event of an emergency or other unforeseen circumstance in which pumping of solution ceases, or in the event of excessive surface runoff from the HLP, discharge of excess water or solution will be directed in a controlled manner through a lined spillway to the events pond. Solution levels within the heap leach are expected to be kept low during normal operations. However, during emergency situations, the HLP spillway will prevent overtopping of the embankment, and will maintain containment of the solution at all times. The HLP spillway is designed to safely convey the flow represented as one third between the 1,000-year event and the probable maximum flood (PMF). The event ponds will incorporate internal and outlet spillways to safely pass the PMF peak flows after attenuation through the pond. Excess water will discharge to events ponds.

The events ponds are sized to provide containment storage for the Probable Maximum Flood (PMF) Event plus 24 hours of draindown from the heap after the in-heap pond has reached its maximum capacity. Assuming fully saturated conditions (no rainfall losses) upstream of the embankment, the estimated rainfall volume reporting to the primary HLF events ponds will be 132,200 m^3.

The primary HLF events ponds will have a combined operational storage capacity of approximately 359,100 m^3 with 1 m of freeboard. The combined storage capacity of the primary HLF events ponds without freeboard will be 308,800 m^3.

17.2.5 Ore Stacking Plan

Ore will be stacked on the HLP in cells in accordance with stacking equipment capacity. The tonnage on each lift was calculated based on the tonnes per day of crushed ore, conveyed to the HLP and the lift volumes. Low grade ROM material will be dumped on the pad from the mine trucks in designated cells and spread using a loader. The total annual tonnage and volumes are listed in the tables below.

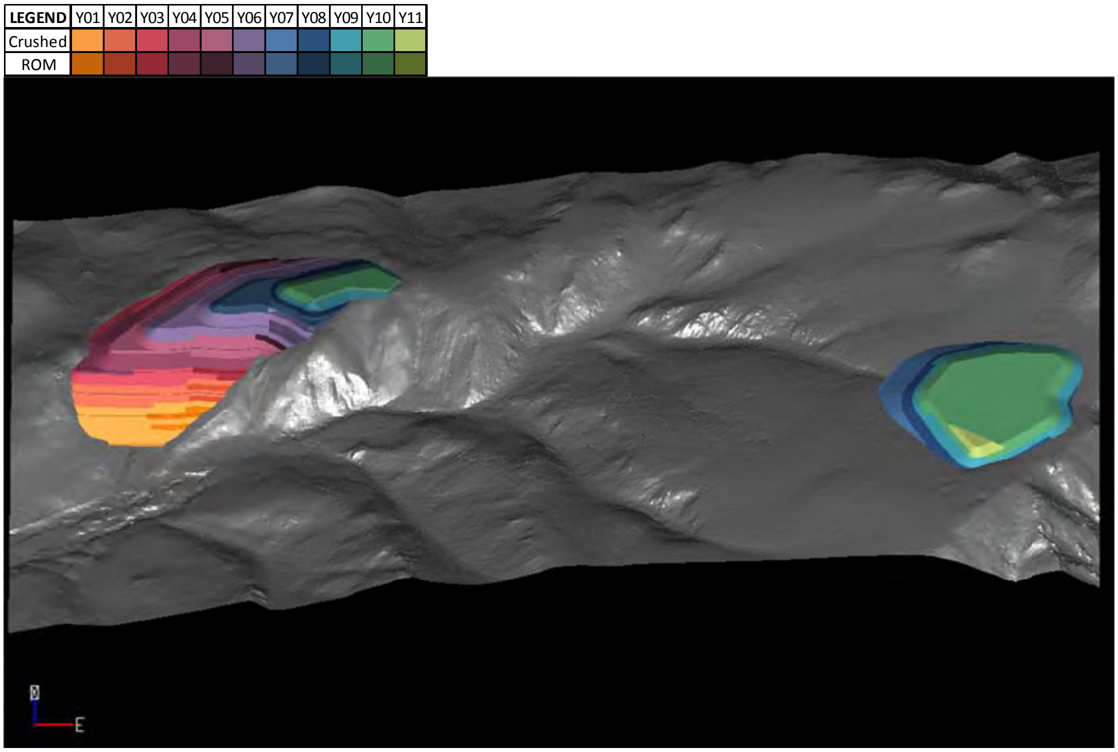
Table 17.2: Total Annual Tonnage and Volumes (m^3)

Year	Heap Leach 1 (Eagle)			Heap Leach 2 (Olive)	Total
	Crush	ROM	Subtotal	Crush	
1	8.8	1.1	9.9	-	**9.9**
2	11.0	1.6	12.6	-	**12.6**
3	10.9	1.5	12.5	-	**12.5**
4	10.9	0.4	11.3	-	**11.3**
5	10.9	1.4	12.3	-	**12.3**
6	11.0	1.7	12.7	-	**12.7**
7	-	2.2	2.2	11.0	**13.2**
8	-	1.8	1.8	10.9	**12.8**
9	-	2.5	2.5	10.9	**13.5**
10	-	0.8	0.8	11.0	**11.7**
11	-	-	-	0.5	**0.5**
Total	**63.5**	**15.1**	**78.6**	**44.3**	**122.9**

Source: JDS (2016)

The final stacking plan is shown in Figure 17.5.

Figure 17.5: Stacking Plan



Source: JDS (2016)

17.2.6 Leaching and Solution Delivery

17.2.6.1 Barren Solution

Piping and Pumping

The barren solution will be pumped by a series of four pumps to the primary HLP and five pumps to the secondary HLP from the barren tank, located in the plant in double-walled pipelines. Barren solution will be pumped at a nominal rate of 2,070 m^3/h, where it will connect into the pad distribution system. The pipeline consists of a 610 mm (24”) standard weight carbon steel carrier pipe, in a 760 mm (30”) fiberglass reinforced plastic containment pipe, and buried to a minimum of 1.5 m below grade to prevent solution freezing. A leak detection system, including a moisture sensing cable, tied into the plant distributed control system, DCS. On the pad, the barren solution will be transferred to 610 mm (24”) HDPE pipe and distributed to the on-pad drip emitter header pipes.

Barren solution will be applied to the heap using drip emitters. The emitters will be buried or covered at least 1 m to reduce the likelihood of freezing. The emitter lines will be ripped into the ground approximately 1 m apart, running along the length of each cell and on the slopes of the lifts.

The barren pipelines will be used for heap rinsing once the gold recovery process is complete.

Solution Heating

During the first year of loading for each pad, barren solution will be heated by a diesel-fired boiler, located adjacent to the plant building, to maintain the thermal balance in the HLP. The boiler is designed to provide 18.1M btu/h (British thermal unit per hour) to heat the solution during the initial loading period, before the pad mass is significant enough to maintain an internal thermal balance.

17.2.6.2 Pregnant Solution

The pregnant solution will be pumped from the collection sump at the toe of the HLPs to the plant, in a 610 mm standard dimension ratio (SDR) 7.3 carrier pipe, contained in a 760 mm SDR 17 containment pipe. The pipes have been sized for a nominal flowrate of 2,070 m^3/h. The pregnant solution pipe will be buried at a minimum depth of 1.5 m, and run for approximately 400 m to the plant from the primary HLP, and approximately 4 km from the secondary HLP. The lines will be installed with a leak detection system, which monitors air pressure in the annular void between the two pipes.

17.2.6.3 Cold Weather Considerations

A review and comparison of heap leaching operations in cold climates indicates year-round leaching operations at the Eagle Gold project site is feasible. Design provisions are incorporated to add and maintain heat in the process solutions applied to the heap.

Since ore particle size, ambient temperatures, delivered ore moisture, agglomeration requirements, and snowfall may play a role in the ability to stack in winter, the project has adopted the following mitigation measures:

- Selected an in-valley heap configuration to create a heat sink;
- Use of an in-heap solution pond for PLS storage;
- Sizing of the fine ore crushing operation to allow increased production rate during warm months;
- Incorporation of 90-day ore storage pad;
- Sizing of the starter HLP to accommodate more than one year of ore production, allowing advanced stacking for at least the first winter season;
- Provision for a D9 track dozer, equipped with a ripper assembly, to rip frozen ore prior to resuming leaching in the spring;
- Heating of barren solution;
- In-heap temperature monitoring;
- Burying drip emitter lines;
- Heat-tracing and insulating the barren tank;
- Heat-tracing and/or insulating (or burying) pipelines; and
- Generators for back-up power supply to pumps and emergency process equipment.

17.2.6.4 Events Ponds

Lined events ponds, external to the HLPs, will be constructed to temporarily store excess process solution that may occur during upset conditions, freshet, and excess precipitation events. The solution contained in these ponds will be recycled back into the heap leach circuit when normal operation resumes. The ponds have been sized to contain peak intensity storm events as well as repetitive wet years and/or periods. The ponds will be constructed to include a leak detection and recovery system underneath the main liner system.

If leach solution cannot be recycled back to the HLP, due to water balance constraints, the leach solution will be stored within the events ponds. In the unlikely event storage volume within the events ponds is at capacity, the solution will be treated in the cyanide detoxification and mine water treatment plant (MWTP), prior to discharge into Haggart Creek.

17.2.6.5 Leachate Solution Collection System

The HLP will consist of an engineered liner system in the heap leach facility. The lower section of the HLP acts as an in-heap pond for the primary storage of PLS.

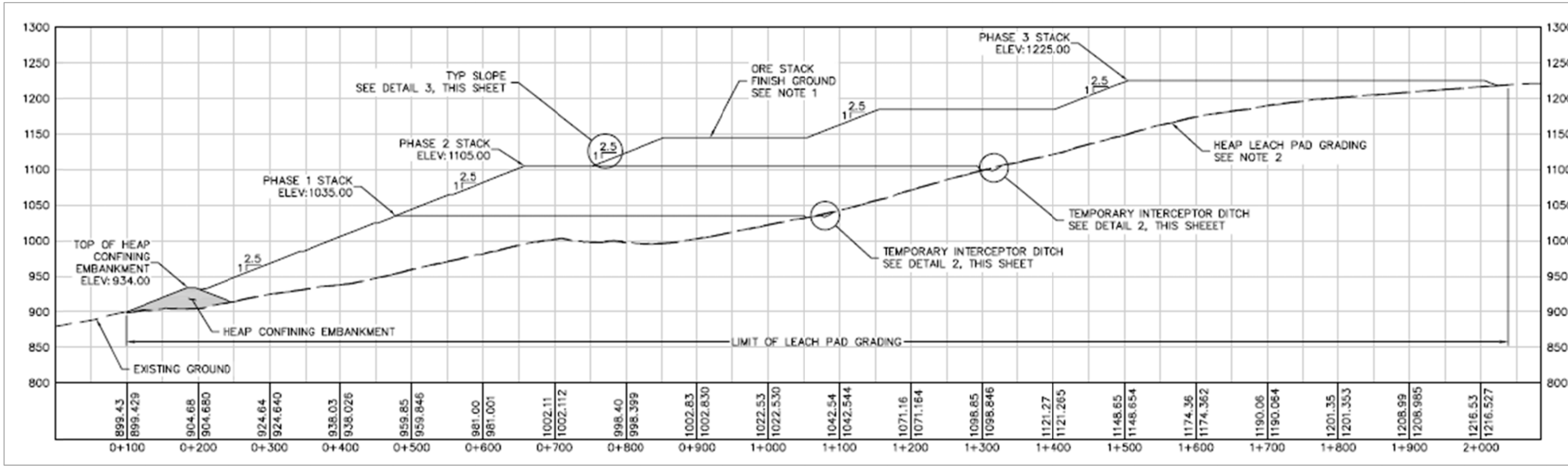
Located above this liner system is a 0.6 m (minimum thickness) layer of drainage rock (all passing 38 mm) which has been designed to transmit the PLS to a collection system. This drainage blanket serves to efficiently transmit the PLS and protect the primary liner from damage by rocks and/or equipment which might come in contact with the liner.

The leachate collection piping system consists of 450 mm, 250 mm, and 100 mm diameter corrugated, dual-wall, perforated ADS N-12 pipes, embedded within the drain rock. The collection pipe network consists of a series of 100 mm secondary drain pipes, spaced approximately 25 m on centre and arranged in a “herringbone” pattern around the larger pipes that will convey the collected fluid (i.e. PLS and storm water flows). The larger pipes consist of 250 mm collector pipes reporting to the 450 mm process lines.

Within the PLS sump, there are three vertical turbine pumps operating and two spares available. The 450 x 760 mm pregnant pumps are each installed with 186 kW motors. Each well has an outer casing, connected to the 450 mm collection lines, mechanical pump, and related electrical and control components. These wells and pumps serve to convey the pregnant solution to the process plant.

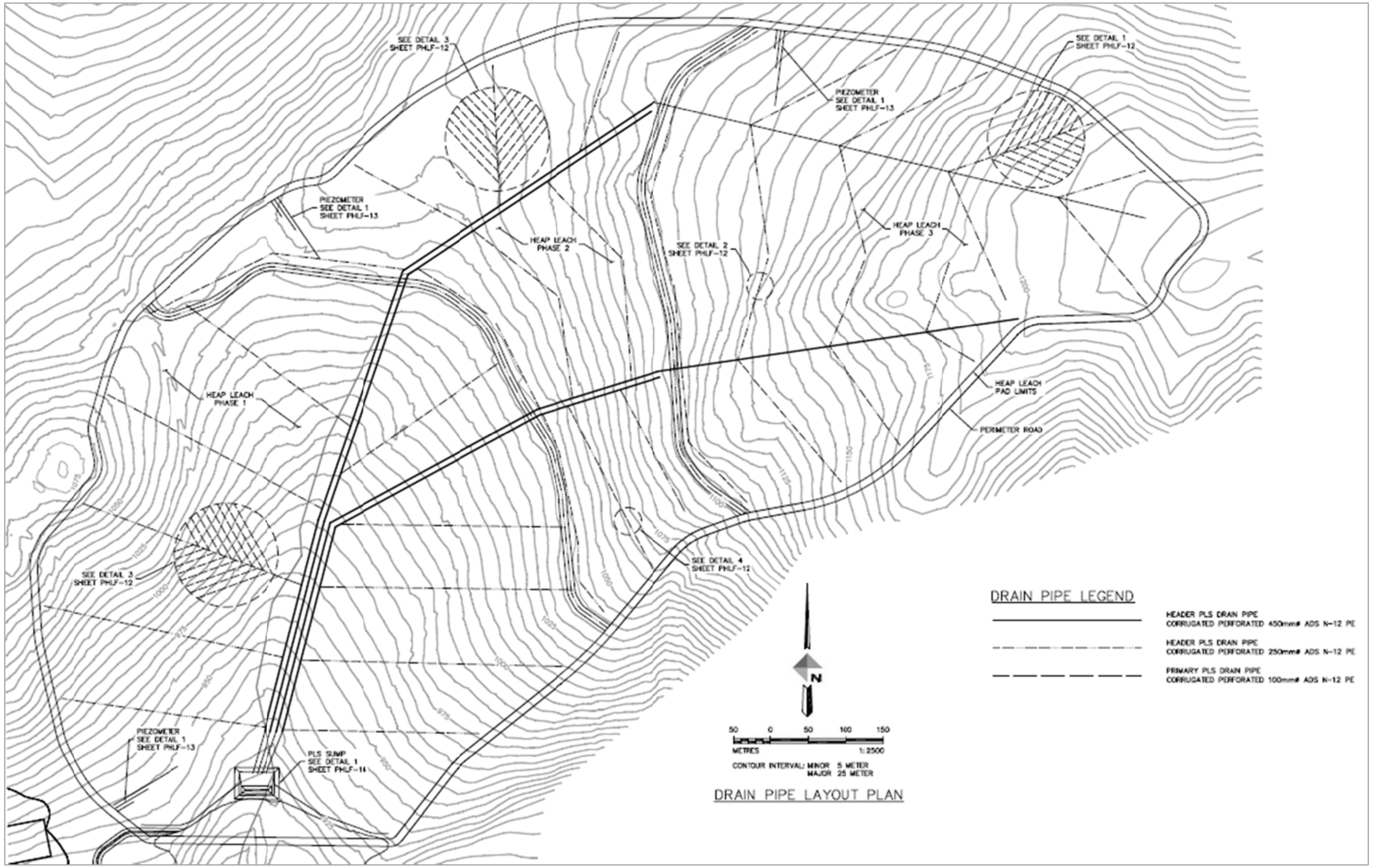
Because of the efficient capture and conveyance of fluids, the primary liner has very limited hydrostatic head exerted as a result of the process fluid. Figure 17.6 and Figure 17.7 provide an overview of the primary HLP leachate solution collection system. The design criteria call for a hydraulic head less than 1.5 m.

Figure 17.6: HLP Cross-Section



Source: Dowl (2016)

Figure 17.7: Leachate Solution Collection System



Source: Dowl (2016)

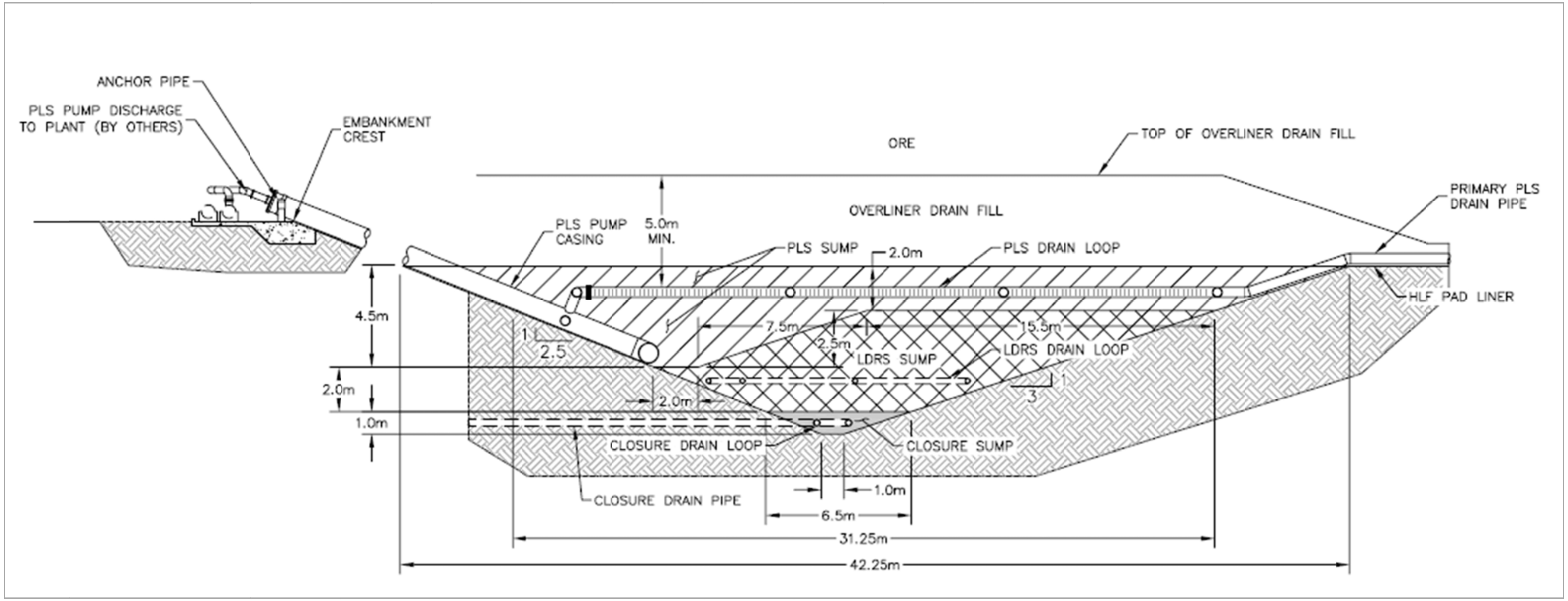
17.2.6.6 Leak Detection and Recovery System

There are two safety systems designed to detect, contain and pump back any leakage resulting from a possible liner failure before any contamination can reach the groundwater.

A leak detection and recovery system (LDRS) will be installed between the upper and lower geomembrane liners, from the sump level to the embankment crest, where the hydrostatic head is greatest (Figure 17.8). If a leak occurs, the drain system will collect the PLS via drainpipes, connected to a collection monitoring sump, located in the HLP. The sump will be installed with monitoring instruments to provide early alerts to the presence of flow. Collected solution will then be pumped back to the ADR plant or the HLP.

There will be a secondary drainage system below the liner system throughout the entire pad area. A leak will trigger an early alert via monitoring instruments; the drain system will collect the PLS, direct it to an external collection monitoring sump located downstream of the events ponds, and then pump it back to the ADR plant or the HLP.

Figure 17.8: LDRS Detail



Source: Dow (2016)

17.2.7 Process Plant

Pregnant solution will be pumped from the HLP sump to the plant. The solution will be distributed between the two trains of Carbon Adsorption Columns (CIC). The solution will pass down each train and carbon will flow countercurrent up the train. The carbon will be collecting the gold as it moves from the last to the first column, depleting the solution by the end of the train. The barren solution will be pumped from the last carbon column to the barren solution tank and back to the HLP. The 17.5 m diameter by 17.5 m high tank will provide approximately three hours of storage capacity for the barren solution.

17.2.8 Carbon Adsorption

The carbon adsorption circuit consists of two trains of five cascading carbon columns. The pregnant or gold-enriched solution will be pumped to the carbon adsorption circuit across a stationary trash screen for removal of any debris from the HLP. The solution will flow counter- current to the movement of carbon from column 1 to column 5. The solution overflow from the final column will discharge onto a screen in order to recover any carbon. The barren solution, which at this stage has had most of the gold in solution adsorbed, will discharge from the final carbon column and be pumped to the barren tank. Cyanide solution, caustic solution, antiscalant and make-up water will be added to the barren tank as needed. On average, 8 t of loaded carbon from the first carbon column will be pumped to the acid wash and stripping circuits each day. The carbon in the second column will be advanced to the first, and the process will be continued down the train. The carbon from the fifth column will advance to the fourth column, and then freshly reactivated carbon will be added.

17.2.9 Desorption and Gold Refining

17.2.9.1 Carbon Acid Wash

The loaded carbon will be transferred to the acid wash vessel and treated with 3% hydrochloric acid (HCl) solution to remove calcium, magnesium, sodium salts, silica, and fine iron particles. Organic foulants such as oils and fats are unaffected by the acid and will be removed after the stripping or elution step by thermal reactivation utilizing a kiln. The dilute acid solution will be pumped into the bottom of the acid wash vessel, exiting through the top of the vessel back to the dilute acid tank. At the conclusion of the acid wash cycle, a dilute caustic solution will be used to wash the carbon and neutralize the acidity.

A recessed impeller pump will transfer acid washed carbon from the acid wash tank into the strip or elution vessel. Carbon slurry will discharge directly into the top of the elution vessel. Under normal operation, only one elution will take place each day

17.2.9.2 Carbon Stripping (Elution)

After acid washing, the loaded carbon will be stripped of the adsorbed gold using the ZADRA process. The strip vessel holds approximately 8.0 t of carbon. During elution, solution containing approximately 1% sodium hydroxide and 0.1% sodium cyanide, at a temperature of 140°C and 450 kilopascals (kPa), will be circulated through the strip vessel. Solution exiting the top of the vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing pregnant solution will be transferred to the incoming cold barren solution.

A diesel-powered boiler will be used as the primary solution heater to maintain the barren solution at 140°C. The cooled pregnant solution will flow by gravity to the electrowinning cells. At the conclusion of the strip cycle, the stripped carbon will be pumped to the carbon-regeneration circuit.

17.2.9.3 Carbon Regeneration

The stripped carbon from the strip vessel will be pumped to the vibrating carbon-sizing screen. The kiln-feed screen doubles as a dewatering screen and a carbon-sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon-regeneration kiln-feed hopper. Screen undersize carbon will drain into the carbon-fines tank and then be filtered and bagged for disposal. A 250 kg/h diesel-fired horizontal kiln will treat 8.0 t of carbon per day at 650°C, equivalent to 100% regeneration of carbon. The regeneration-kiln discharge will be transferred to the carbon quench tank by gravity, cooled by fresh water or with carbon-fines water, prior to being pumped back into the CIC circuit.

To compensate for carbon losses by attrition, new carbon will be added to the carbon attrition tank. New carbon and fresh water are mixed to break off any loose pieces of carbon prior to being combined with the reactivated carbon in the carbon holding tank.

17.2.9.4 Refining

Pregnant solution will flow by gravity from the elution vessel to a secure gold room. The solution will flow through one of two electrowinning cells. Gold will be plated onto knitted-mesh steel wool cathodes in the electrowinning cell. Loaded cathodes will be power washed to remove the gold-bearing sludge and any remaining steel wool. The gold-bearing sludge and steel wool will be filtered to remove excess moisture and then dried in an oven. From the oven, the gold material will be mixed with fluxes consisting of borax, silica and soda ash before being smelted in an induction furnace to produce gold doré and slag. The doré will be transported to an off-site refiner for further purification. Slag will be processed to remove entrained gold prills and re-melted in the furnace. The gold bars will be stored in a vault located in the gold room prior to secure off-site transportation.

17.2.10 Reagents

Sodium cyanide briquettes will be delivered to site in containers and in 1 t super sacks contained in a wood frame. The briquettes will be mixed in the cyanide mix tank and subsequently transferred to the cyanide solution storage tank. The concentrated cyanide solution will be added to the barren tank at a rate of 0.35 kg/t of ore. Cyanide will be used in the carbon strip circuit at a concentration of 0.1%. The principles and standards of practice for the transport to site and handling of cyanide on-site will be in accordance with the guidelines set out in the International Cyanide Management Code (ICMC).

Sodium Hydroxide (caustic) will be supplied to site in 1 t totes. The caustic will be mixed and stored for distribution to the acid wash and strip circuits. The caustic will be used to neutralize the acid in the acid wash circuit. A solution of 1.0% caustic will be mixed with barren solution in the carbon strip circuit.

Hydrochloric acid and antiscalant solutions will be supplied to site in 1 t totes. The solutions will be metered directly from the totes for distribution in the plant.

Hydrated lime and cement will be delivered to the site in bulk by trucks and stored in a 200 t lime silo or 300 t cement silos. The lime will be delivered at a rate of 1 to 1.5 kg/t of ore by screw feeder onto the heap leach feed conveyor during heap loading operations. Cement will be added at a rate of 2 to 6 kg/t ore for agglomeration.

17.2.11 Laboratory

An assay and metallurgical laboratory will be equipped to perform sample preparation and assays by AA, fire assay, and cyanide (CN) soluble analyses. The facility will be equipped to prepare and analyze up to 3,600 samples per month. The laboratory facility will support exploration, mining, minor environmental sampling, total suspended solids (TSS) monitoring and processing. The majority of the environmental samples will be sent off-site to an accredited laboratory for third party reporting. The laboratory has space available for process optimization and test program.

17.3 Gold Production Model

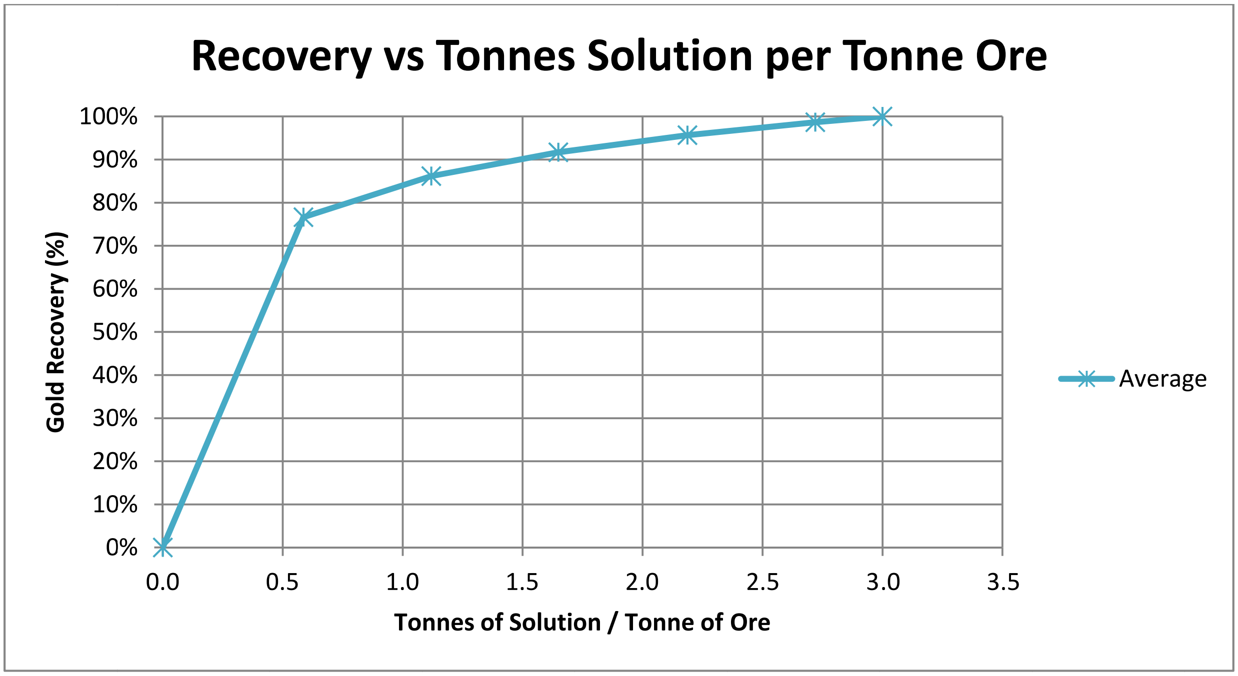
The gold production model was developed from a combination of metallurgical test work, the mine production schedule, the stacking volumes on the primary HLP and secondary HLP and the barren solution application rate.

17.4 Gold Model Development

17.4.1 Metallurgical Test Work

The raw column leach test data for each rock type was compiled to produce average recovery curves for days leached and tonnes of leach solution applied per tonne of ore. The solution ratio recorded during lab tests was scaled down to an ultimate ratio of 3.0, reflecting full-scale operations with a 10 m lift height. The leach curves for all Eagle ore types were averaged to create a master leach profile, as shown in Figure 17.9.

Figure 17.9: Master Leach Profile



Source: JDS (2016)

The grade and recovery for each rock type were provided by KCA and are discussed in Section 13. The table below shows the rock types and their assigned gold recovery.

Table 17.3: Recovery by Rock Type

Zone	Recovery (%)
Eagle	
Type 1 - A	79
Type 2 - C	73
Type 3 - B	68
Type 4 - E	73
Type 6	68
Olive	
Oxide	66
Transition	55
Sulphide	52
ROM	55

Source: KCA (2016)

17.4.2 Gold Production Schedule

The quarterly mine production schedule provided the quarterly tonnage and grade for each rock type being delivered to the HLPs. From the tonnes, grade and recovery, a weighted average grade and recovery were calculated for each quarter. The tonnage from the first quarter of each year, and the calculated grade and recovery, were distributed equally between the three quarters when stacking takes place. This accounts for the primary crushed ore being continuously reclaimed from the winter ore storage stockpile.

17.4.3 Gold Model

For Year 1 through Year 11, each quarter was divided into cells of 7.6 stacking days, representing 12 cells per quarter. The mine schedule for that quarter was then used to determine the weighted average grade and recovery for each cell. Each cell (week) of each quarter has the same tonnage, grade and recovery. An example is shown in Table 17.4 below.

Table 17.4: Year 7 – Quarter 2 Weeks 1 and 2

Annual Tonnage 10.95 Mt			
Description	**Units**	**M1 - W1**	**M1 - W2**
Type 1 - A	t	20,600	20,600
Type 2 - C	t	10,600	10,600
Type 3 - B	t	194,600	194,600
Type 4 - E	t	1,800	1,800
Type 6	t	600	600
Primary Crush	t	76,000	76,000
Total	**t**	**304,200**	**304,200**
Grade			
Type 1 - A	g/t	0.68	0.68
Type 2 - C	g/t	0.57	0.57
Type 3 - B	g/t	0.6	0.6
Type 4 - E	g/t	0.44	0.44
Type 6	g/t	0.42	0.42
Primary Crush	g/t	0.65	0.65
Ave. Grade	**g/t**	**0.61**	**0.61**
Recovery			
Type 1 - A	%	79	79
Type 2 - C	%	73	73
Type 3 - B	%	68	68
Type 4 - E	%	73	73
Type 6	%	68	68
Primary Crush	%	70	70
ROM	%	55	55
Ave. Recovery	**%**	**69**	**69**

Source: JDS (2016)

The total gold contained in each cell, and the total amount recoverable, were calculated using the tonnage, average grade and recovery. The model then calculated the gold recovered in each cell over time, depending on the amount of solution applied to that cell. Each cell experienced a primary leach time of 90 days, and the remaining gold was recovered through secondary leaching from the solution applied to the cells above. A solution application rate of 10 l/h/m^2 was used for design, and the cell area was calculated based on cell tonnage, a 10 m lift height and a 1.7 t/m^3 bulk density. As the year progresses, the model calculates the gold heap leach inventory and the gold recovered in solution.

Cells will be created every 7.6 days, and after 90 days of primary leaching, approximately 90% of extractable gold will be recovered. After approximately 200 days of combined primary and secondary leaching, 100% of extractable gold recovery will be achieved. The estimated combined 200 days of leach time varies from the 150 days, as recommended by KCA, due to secondary leaching and solution flow rate differences.

A total solution flow rate of 2,070 m^3/h will be maintained on the pad as cells progress from primary to secondary leaching. The gold recovered from each cell is calculated using the master leach profile and the amount of solution applied to the cell at that point in time. An example of the model parameters for the first few weeks of a cell is shown in Table 17.5.

Table 17.5: Cell Parameters Example

Elevation - Cell ID	Unit	Cell 940-1	Cell 940-1	Cell 940-1
Status		Stacking	Leaching	Leaching
Days leached in period	days	7.6	7.6	7.6
Total Days	days		7.6	15.3
Solution Flow Rate	m^3/hr		179	179
Tonnes of Solution	t	-	32,817	65,634
Tonnes of Ore	t	304,248	304,248	304,248
Tonnes Solution/Tonne Ore	t/t	-	0.11	0.22
Ultimate Recovery	**%**	**-**	**14**	**28**
Gold Received	oz	-	588	1,177
Gold Inventory	oz	6,016	5,428	4,839

Source: JDS (2016)

The gold inventory and gold recovered from the pads is carried from one year to the next as the leaching of each cell is completed. The total gold recovered is summarized in Table 17.6 below.

Table 17.6: Gold Recovery Summary

Description	Units	TOTAL	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10	YR 11
Eagle Ore													
Total Throughput	Mt	**101.3**	8.8	11.0	10.9	10.9	10.9	11.0	11.0	10.9	10.3	5.6	0.0
Gold Recovered From HL	koz	**1,697.3**	138.9	201.2	205.2	209.0	204.6	184.0	155.9	150.3	134.6	86.0	27.6
Year End Gold Inventory	koz	**633**	68	150	218	284	361	429	491	555	612	653	633
		72.90%											
ROM Ore													
Total Throughput	Mt	**15.1**	1.1	1.6	1.5	0.4	1.4	1.7	2.2	1.8	2.5	0.8	0.0
Gold Recovered From HL	koz	**73.0**	3.5	7.1	7.5	3.7	5.0	7.7	9.9	9.5	11.0	6.8	1.4
Year End Gold Inventory	koz	**59.7**	6.2	13.7	19.3	19.0	26.1	33.4	43.2	49.7	60.8	61.1	59.7
		55.00%											
Olive Ore													
Total Throughput	Mt	**6.5**									0.7	5.4	0.5
Gold Recovered From HL	koz	**113.3**									16.0	90.9	6.4
Year End Gold Inventory	koz	**86.3**									9.4	81.1	86.3
		56.80%											
Total Recovery													
Total Throughput	Mt	**122.9**											
Gold Recovered From HL	koz	**1,883.7**											
Year End Gold Inventory	koz	**778.6**											
		70.8%											

Source: JDS (2016)

18 Project Infrastructure and Services

18.1 General Site Arrangement

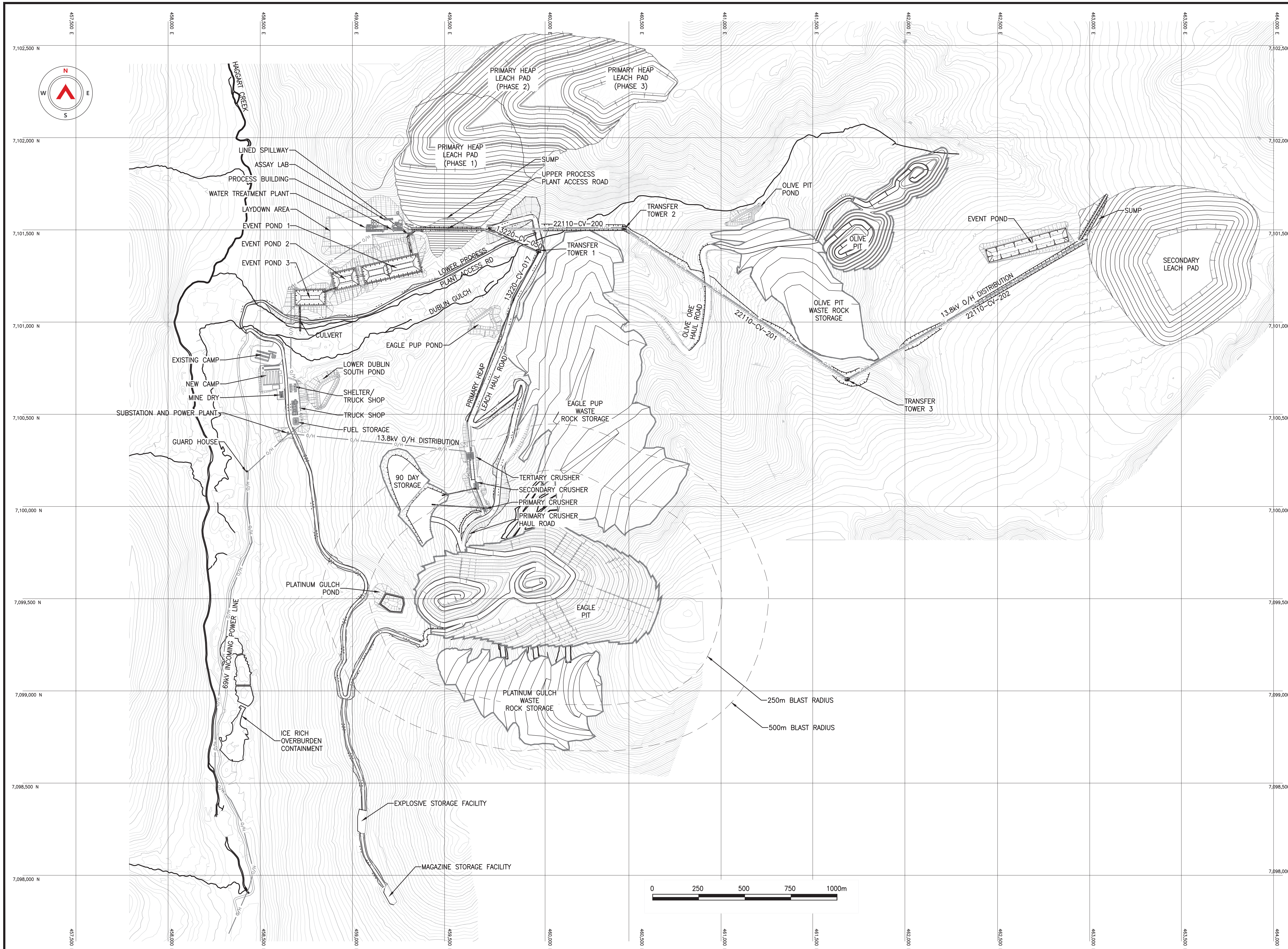
The project will require the development of various ancillary facilities and related infrastructure; their location has been selected to take advantage of local topography, to accommodate environmental considerations, and reduce capital and operating costs.

Project facilities and infrastructure will include:

- Two heap leach pads (HLP), comprised of a sump, a lined storage area, an in-heap storage area, pumping wells, events ponds, diversion ditches, leak detection, recovery and monitoring systems;
- Fresh water supply systems to treat and distribute process, fire, and potable water;
- Access and site roads, including the upgrading 23 km of the Haggart Creek access road;
- Water treatment infrastructure, including a Mine Water Treatment Plant (MWTP), cyanide detoxification capacity, and potable and sewage treatment infrastructure;
- Domestic waste disposal facilities;
- Ancillary facilities, including:
    - Warehouse/first aid;
    - Truck shop and truck shelter;
    - Mine dry;
    - Cold storage/laydown;
    - Administration building;
    - On-site fuel storage depots including diesel, gasoline & propane;
    - On-site explosive storage and magazines;
    - Assay laboratory;
    - Temporary and permanent camp accommodations complete with recreation area, commissary and laundry facilities;
    - Guard shack and entrance gate;

- Power supply and distribution, including:
    - A 43.5 km long, 69 kV power supply line from the Yukon Energy Corporation's power grid where a tap in point is available, approximately 25 km southeast of the property.
    - 13.8 kV power distribution from the mine site substation to all the facilities; and
    - Process control and instrumentation communication systems.

The location of the main project facilities is shown in Figure 18.1.



NOTES:

1. ALL ELEVATIONS ARE IN METERS.
2. CONTOUR DATA RECEIVED FROM UNDERHILL (FEBRUARY 2011) UTM NAD83.
3. 5m CONTOUR INTERVALS SHOWN.
4. 13.8kV O/H DISTRIBUTION TO FOLLOW ALL CONVEYORS EXCEPT 13220-CV-017

REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
B	16/10/19	ISSUED FOR FEASIBILITY UPDATE	JLC	NV
A	16/10/04	ISSUED FOR FEASIBILITY UPDATE	JLC	NV

CLIENT: Victoria GOLD CORP

Allnorth

CLIENT NO: #	DRWN: JLC	DATE: 16/10/19
PROJECT NO: 16VA0008	DSGN: SK/JLC	DATE: 16/10/19
DRAWING SIZE: 22 X 34	CHKD: NV	DATE: 16/10/19
SCALE: 1:10,000	APVD: SK	DATE: 16/10/19

PROJECT:

EAGLE GOLD FEASIBILITY STUDY

TITLE:

GENERAL ARRANGEMENT OPERATIONAL

DWG NO: 16VA0008-000-1011-002 REV: B

Date: 2016/10/19 | User: Jessica Currie | File: C:\Users\jbaxter\Desktop\1000-Drawings - Eagle Gold\1011-Civil\01-Production\16VA0008-000-1011-Production | Layout: 002 | Paper Size: 863.6mm x 558.8mm

18.2 Roads

18.2.1 Access Road

An existing 90 km series of paved and gravels roads—including the Silver Trail Highway (Highway 11), the South McQuesten Road and the Haggart Creek Road (HCR) currently provide access to the project site (Figure 18.2). Of these, only the 23 km HCR will require modest upgrading to accommodate equipment, supplies and materials required for the project construction and operation. The HCR will be:

- Widened to two lanes in certain areas, and to two-way single lane, radio-controlled in others with area turnouts every 500 m or as required;
- Surfaced with crushed granular material where required;
- Provided with a minimum 2% cross-fall to facilitate water shedding from road surface, to improve drainage;
- Provided with side ditches for positive water drainage;
- Provided with new or replaced culverts as necessary, to further improve drainage and/or to further improve trafficability and road safety.

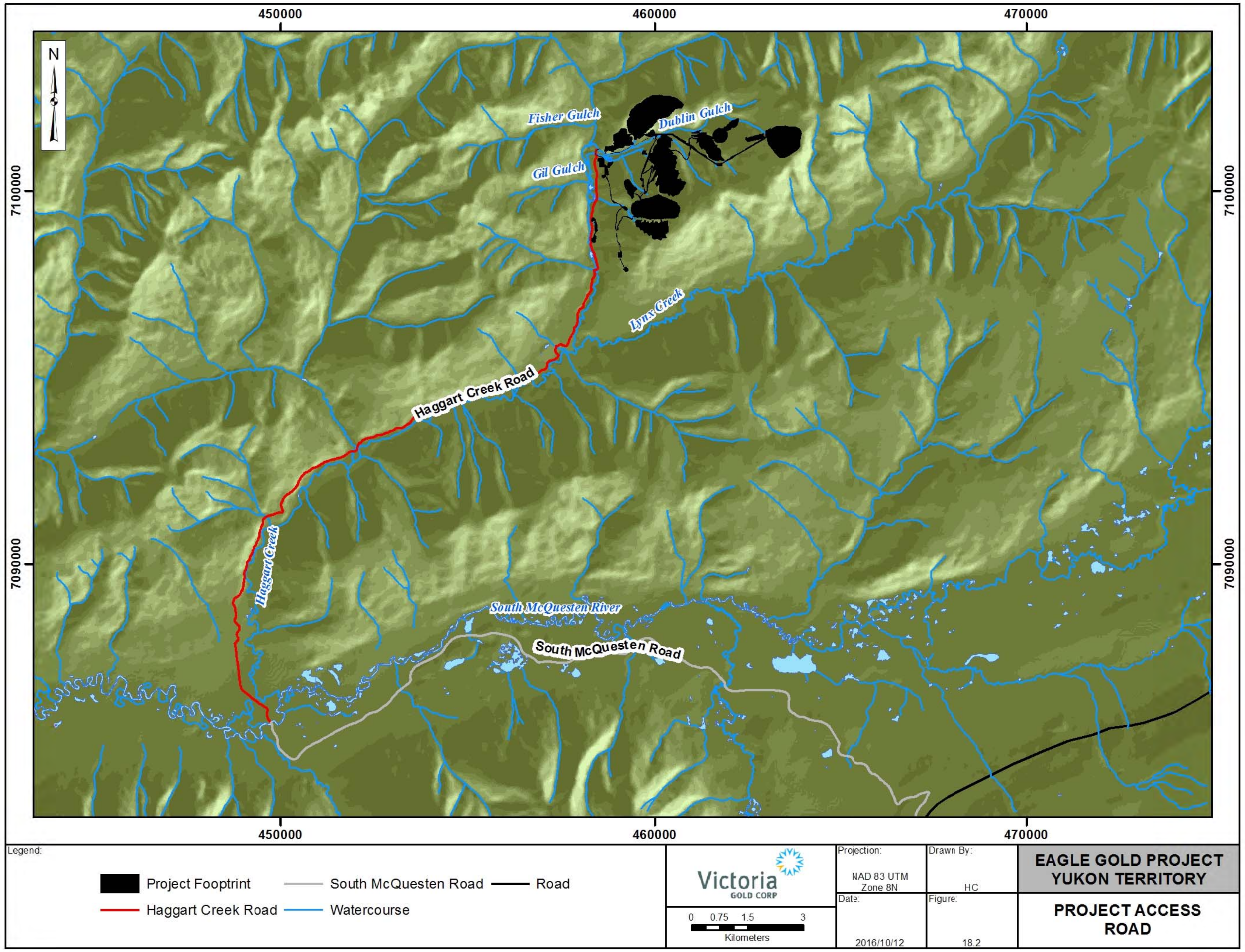
The road design will be informed by the meet the following standards:

- Geometric Design Standards for Canadian Roads and Streets, Transportation Association of Canada;
- BC Supplement to Transportation Association of Canada Geometric Design Standards.

Appropriate speed/hazard signage will be provided as necessary to ensure safe use.

Victoria Gold will assume responsibility for maintenance of the HCR during construction, throughout the LOM, and during closure; however, funding and/or work share agreements with the Yukon Department of Highways and Public Works have been utilized by Victoria Gold in prior years for maintenance activities and are likely to remain available.

Figure 18.2: Project Access Road



18.2.2 Site Roads

A network of site roads will be constructed throughout the mine site. Roads have been divided into three categories depending on use: haul roads, service roads, and access roads

Haul roads will include:

- The Eagle pit haul road – Connects the Eagle pit to the primary crusher;
- The Olive pit haul road – Connects the Olive pit to the primary crusher; and
- The primary heap haul road – Connects the Eagle pit to the primary HLP.

In addition to the named haul roads, roads will be constructed as required connecting the pits to the WRMFs in order to minimize haul distances and improve operations.

These named haul roads will have a running width of 20.7 m (three times the width of a haul truck) with a 2 m high berm, where a drop-off exists above 3 m adjacent to the road. Ditching will be provided on the side of the road for drainage. The road sub-base and base requirements will be governed by the quality of the subgrade; the maximum haul road grade is planned to be 10%. The unnamed haul roads will be constructed as required and built to suit.

A service road with a running surface of 13.8 m (two times the width of a haul truck) and 2 m high berms, where required, will connect the Eagle pit to the truck shop area for maintenance purposes. Ditching will be provided adjacent to this road as required for drainage. The maximum grade of this road will be 10%.

Access Roads, with 8.5 m running surfaces will be constructed to connect other areas of site such as the explosive and magazine storage areas and the ADR plant.

Additional access roads may be required for maintenance of overland conveyors. Conveyors have been located adjacent to existing roads where practical and conveyor corridors will include sufficient width to allow for vehicle access as well. Grades on these roads be greater than 10% for short distances to support efficient conveyor routing.

Ditching and surface grading will be provided for all site roads as required, to facilitate drainage.

18.3 Buildings and Structures

18.3.1 Diesel Storage

Diesel fuel, primarily for haul trucks, will be stored in two 750,000 L tanks within a bermed containment area located near the truck shop.

Additionally, a 10,000 L storage tank will be located at the ADR Plant to store waste oil, which in addition to diesel fuel will be used as a fuel source for the solution heating boiler.

18.3.2 Propane Storage

Three 5,000 gal propane tanks will be located adjacent to the permanent camp facilities.

18.3.3 Explosives Storage

18.3.3.1 On-site Explosives Manufacture and Storage

The explosives manufacture plant will be a pre-engineered building provided by the explosives supply contractor. The plant will be located 1 km southwest of the Eagle pit, and 600 m from the main access road to the plant. Access to the plant will be controlled by a locked gate to prevent unauthorized access.

18.3.3.2 Detonator Magazine Storage

The detonator magazine will be a pre-fabricated Sea Can-type structure provided by the explosives supply contractor. This facility will be located 300 m south the explosive manufacturing facility, and 600 m from the main access road to the plant. Like the explosives manufacture plant, access to the detonator magazine will be controlled by a locked gate.

18.3.4 Pre-Engineered Buildings

Pre-engineered buildings are used for the following facilities:

- ADR plant (52 m long x 38 m wide);
- Truck shop (77 m long x 21 m wide);
- Secondary crushing building (33 m long x 24 m wide);
- Tertiary crushing building (35 m long x 34.6 m wide); and
- Primary crusher building (17.8 m long x 10.8 m wide).

Buildings will be constructed with a structural steel frame, steel girts and purlins and intermediate structural members. Walls will be constructed of insulated metal wall panels and the roof will be a metal standing seam roof system. The envelope package will come complete with doors and all other envelope-related items. High bay lighting will also be included where applicable.

18.3.5 Modular Buildings

Modular buildings will be used for the following facilities:

- Administration facility;
- Assay lab modules (34 m long x 7 m wide);
- Gatehouse (6.1 m long x 3.7 m wide); and
- Mine dry (32.5 m long x 18 m wide).

Each building will include heating, ventilation, and air conditioning (HVAC), electrical, piping, fire detection and suppression systems ready to be connected to the site utilities. The modules are to be constructed of wood framing with insulated metal clad walls and ethylene propylene diene monomer roofing on plywood substrate. Once the modules will be in place and connected together the complex will be weather tight.

18.3.6 Hybrid Buildings

18.3.6.1 Truck Shelter / Warehouse (36 m long x 30 m Wide)

This building will be constructed with the combination of shipping containers and pre-engineered fabric roof and end walls. It will include HVAC, electrical, piping, fire detection and suppression systems ready to be connected to the site utilities. The flooring will be wood-mat flooring with a geomembrane.

18.4 Power

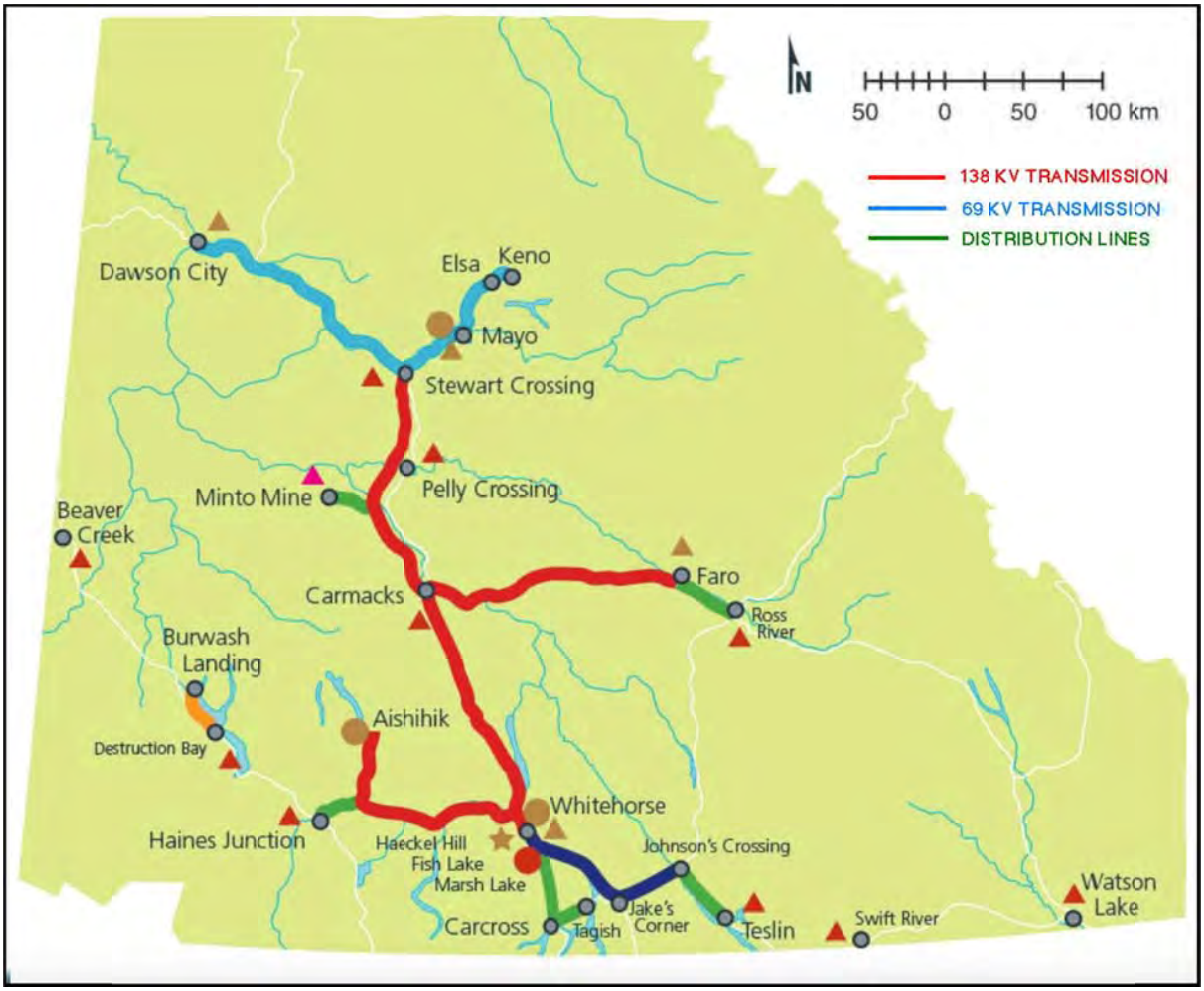
18.4.1 Utility Power Supply

The planned source of primary electric power for the Eagle Gold project will be from the Yukon Energy Corporation (YEC) Grid. YEC currently generates most of the Yukon's electricity supply, and sells wholesale power to ATCO Electric Yukon, and directly to customers in several cities, plus directly supplies large industrial customers. YEC’s primary source of power is hydro generation with facilities including the Whitehorse hydro plant, the Aishihik hydro plant, located about 110 km northwest of Whitehorse, and the Mayo A and B hydro plants.

YEC has back-up diesel generation primarily in Whitehorse but also has facilities in Faro, Dawson and Mayo. YEC also has two new LNG fueled generators in Whitehorse with a total capacity of 8.8 MW, with provision for the convenient installation of a third generator.

YEC owns and operates the Yukon power grid. The grid runs from Dawson YT, its most northern region on the grid, to Whitehorse, its most southern connection. Refer to Figure 18.3 below.

Figure 18.3: Yukon Electricity Grid (Map by YEC)



18.4.2 Project Transmission Line

Electric power for the project will be provided from a tap point on the existing YEC 69 kV transmission line between Mayo and Keno. This line, plus the line from Stewart Crossing to Mayo are currently being redesigned by YEC consultants as 138 kV circuits. In addition, YEC is currently having the new McQuesten substation designed. This is to be located between Mayo and Keno which is the location selected as the tap point for the transmission line extension to the Eagle Gold project. It is believed that a reconstructed YEC transmission line from Mayo to Keno, although designed for 138 kV, would initially be energized at 69 kV.

If the line voltage is raised to 138 kV, then step-down transformation would be added at the McQuesten substation. Utility metering will be located at the McQuesten substation. If the Mayo to Keno line is initially to be energized at 138 kV, which is not expected, the tap line and the mine site substation could be built at 138 kV, with a modest cost increase.

The proposed 69 kV transmission line for the mine will run 43.5 km from the McQuesten tap location near the Silver Trail Highway, to a 69 to 13.8 kV step down substation at the mine site. The line will generally be constructed parallel with the existing access road. It will follow the South McQuesten Road to the crossing of the South McQuesten River, and then along the Haggart Creek Road to the mine site. There are several selected locations where the line will deviate somewhat from the road in order to improve constructability, contribute to safety, and improve the long term reliability of the circuit, while at the same time reducing costs.

The terrain along the transmission line route is generally mild undulating ground and light boreal forest vegetation. Ground conditions are expected to be a mixture of permafrost and till with small pockets of near surface bedrock. Single wood pole structures and 266.7 MCM “Partridge” ACSR conductor have been used to develop preliminary material costing. Horizontal line post tangent structures and guyed angle structures have been used for preliminary structure spotting and layout. This configuration provides for average spans of 90 to 100 m. At the detailed design stage, it is likely that two or three structures may be designed as H-Frames in order to allow for longer spans and increased foundation stability. Appropriate pole foundations have been allowed for in permafrost locations. The line preliminary design has avoided the necessity of locating any structures in wetlands.

The proposed line route was ground truthed, and using PLS-CADD software, a preliminary transmission plan and profile drawing set was prepared from available 1 m contour data and orthographic imagery. The design conforms to the applicable CSA standards with special consideration given to the anticipated icing conditions. This preliminary design formed the basis of a material take-off and the construction capital cost estimate. The current design built off earlier preliminary design work undertaken by Stantec Ltd., as outlined in their 2010 document titled “Eagle Gold Project, High Voltage Transmission Facilities.”

18.4.3 Eagle Gold Main Substation

The site main 69 kV step-down substation will contain an incoming line termination structure, a main incoming circuit switcher (combined breaker and motorized isolating switch) and areal 69 bus work to deliver 69 kV power to two step-down transformers, each with a primary circuit switcher. The transformers will be connected to the secondary 13.8 kV metalclad switchgear via cable bus. This switchgear, located in the diesel power plant modular E-house, will include the transformer main secondary circuit breakers, and in addition to the diesel plant generator circuit breakers, it will include circuit breakers for site 13.8 kV power distribution, via overhead lines to the crushing and processing plants, pumping installations, and ancillary facilities.

The two main power transformers will be outdoor, oil filled type, designed to CSA Standard C88. They will be each rated 69 kV to 13.8 kV, 10/13.5/15 MVA, ONAN/ONAF1/ONAF2 with automatic on-line tap changers. At their maximum fan cooled rating, full redundant capacity is provided. Included are 69 kV station class surge arresters for each transformer and for the incoming line.

Each main power transformer will have a secondary neutral grounding resistor, and thus the 13.8 kV distribution system is 3-wire high resistance grounded, providing for both increased safety and system availability, as is standard for mining installations. All loads on the 13.8 kV system must be 3-phase 3-wire, or if single phase, must utilize two bushing transformers with a 13.8 kV primary rating.

The E-house will also mount a 120 volt DC battery bank for use by both the power plant and substation switchgear, and will house the main substation control and protection panel. Note that the utility metering will be located at McQuesten.

A substation ground grid is included and the planned installation design and cost estimate also includes station fencing and fence grounding.

An automatically switched 13.8 kV power factor correction capacitor bank will be fed from the E-house to provide power factor correction as required by YEC, and to assist in voltage control, particularly during starting of large motors.

The substation will utilize pre-cast equipment and structure foundations in order to reduce site labour and speed construction.

18.4.4 Diesel Generation

Two modular, 1800 revolutions per minute (rpm) diesel generator sets and associated modular E-houses will be installed at the project site to provide 5 MW continuous power for standby (emergency power) to critical loads such as for the accommodations, offices etc. and for essential process loads, in particular to provide freeze protection. The generators will also provide supplemental generation as may be required, especially in winter.

The diesel generator set modules are fully insulated with Arctic rated heating and ventilating systems. Each unit includes a day tank, staring batteries, a local control panel, motor control centres (MCC) and other accessories. The E-house contains the generator circuit breakers and the master protection and control equipment. The engines radiators will be field mounted on top of the modules and will connect to factory installed piping and wiring.

The diesel generating station central E-house also includes the main substation transformer secondary 13.8 kV circuit breakers, site distribution circuit breakers, the station 120 volt battery bank, the station protection and control panels, and the generating plant and substation combined station service transformers. Separate switched grounding resistors are provided for emergency operation. The station 13.8 kV switchgear is split into two sections (with a normally closed tie breaker) to provide additional emergency serviceability. Mounting the substation equipment in the generating plant E-house eliminates the need for a second E-house and reduces field wiring and installation costs.

The diesel plant is PLC controlled and designed for automatic unattended operation, with power import / export controls for paralleling with the YEC system. Human Machine Interface (HMI) operator stations will be included as well as hard wired emergency operator controls. A fibre optic connection is to be provided to the process plant so that a remote monitoring HMI can be located there. It is to be noted that the station will include the additional functionality as required, to operate continuously in parallel with the utility, as may be required at times.

The station modules are designed to be mounted on rig mats, included in the estimate, which in turn rest on compacted pads. Concrete foundations are not required, except for small exhaust stack foundations. The modules include fire detection, inert gas fire protection and leak containment facilities.

18.4.5 Site Power Distribution

18.4.5.1 General

Large-capacity power loads will be serviced by pad-mounted transformers and dedicated electrical buildings housing switchgear, MCCs and control systems equipment. Small-capacity power loads will be serviced by pole-mounted transformers and electrical and control equipment installed in rooms within the administration, camp and other buildings, or in outdoor-rated enclosures.

18.4.5.2 13.8 KV

Power will be distributed through the site at 13.8 kV via overhead power lines to the following facilities:

- Crushing Area;
- Camp and shops area;
- Process building and water treatment plant; and
- Primary & Secondary HLP.

18.4.5.3 4160 V

Large motors such as crushers, barren and pregnant solution pumps and large conveyors will be fed from 4,160 V. This voltage will also be used for distribution to local 600 V MCCs via transformers.

18.4.5.4 600 V

Low voltage MCC, switchboards and panels will be provided as appropriate. Process loads are powered via 600 V MCCs which are located close as reasonable to the associated equipment.

18.4.6 Annual Site Wide Power Demands

The power supply infrastructure is designed to operate at full capacity, year-round. A number of processing-related equipment items will not operate during the 90-days winter period (approximately January through March). Accordingly, power consumption will vary seasonally.

Power consumption will also increase over the LOM, as the size of the HLP increases over its two expansion phases, requiring greater pumping and conveying capacity.

Table 18.1 outlines the estimated summer and winter energy demand through the various phases.

Table 18.1: Estimated Energy Consumption

Description	Summer (MWh)	Winter (MWh)	Total kWh/period
Pre-production		1,420	1,420
Year 1 - 4	213,083	5,358	218,441
Year 5 to 6	120,601	5,429	126,030
Year 7	94,898	2,714	97,612
Year 8+	357,817	21,767	379,584
Total			**823,087**

Source: Allnorth (2016)

18.5 Water

18.5.1 Water Supply

18.5.1.1 Water Supply Infrastructure

The fresh water system will supply fresh water to the ADR facility area, the truck shop area, and crushing area.

Fresh water will be pumped from an aquifer via ground wells located in the Dublin Gulch valley, to a common fire water tank located near the process plant building. Water will then be pumped to a fire water main at each facility throughout the site, providing required flow and pressure for the fire protection system. Fresh water will be pumped from the upper portion of the tank to a potable water treatment system. Treated water stored in a tank supplies the heating solution boiler, elution solution boiler, and shower and eyewash stations in the facility plant. Distribution of potable water to other facilities will be delivered via truck. Fresh water not requiring treatment is piped from the holding tank by gravity to the ADR facility for distribution.

18.5.2 Water Management

The water management infrastructure will include all structures related to the collection, diversion, conveyance and storage of surface water passing through the project footprint.

The water management infrastructure items are listed in the following sections, followed by water management performance objectives, functional requirements, and design basis.

18.5.2.1 Non-contact Water Diversion Structures

Sources of water that have not been influenced by mining activities (non-contact water) be diverted around mining disturbances. Separation of contact and non-contact waters reduces the downstream impact of mining activities and it is more economical to minimize the quantity of water that requires treatment by physical, chemical, or biological means (i.e. passing through the MWTP).

The network of diversion structures will include long term fixed diversion ditches, and temporary diversion ditches. Each of these structures is described below.

18.5.2.1.1 Non-contact Water Diversion Ditches

A number of diversion ditches will be established during the initial construction phase to divert non-contact runoff. These channels are typically V-shaped or trapezoidal in cross-section, with rock or vegetated channel lining to prevent erosion. Additional erosion protection will be required at slope breaks and channel bends.

These ditches will generally be sized to convey the 10-year 24-hour peak storm for the watershed size; however, ditches located upslope of key mine infrastructure will be sized to convey the runoff from a 100-year 24-hour storm event.

18.5.2.1.2 Temporary Non-contact Water Diversion Ditches

During construction and though expansion phases of mine operations, temporary (generally consisting of six months to a year) diversions will be required. Construction and maintenance of these structures should be consistent with that of permanent diversion structures.

18.5.2.1.3 Contact Water Interception Structures

Contact water will be intercepted down gradient of areas that have been disturbed by construction and mining activities. These facilities are optimally located at converging topographic low points to facilitate drainage by gravity. However, they may consist of side-hill ditches that intercept overland sheet flow. These channels are typically V-shaped or trapezoidal in cross-section with rock or geosynthetic channel lining to prevent erosion. Additional erosion protection will be required at slope breaks and channel bends.

Similar to diversion ditches, both permanent and temporary interceptor ditches will be required.

18.5.2.1.4 Contact Water Interceptor Ditches

Typical interceptor ditches include roadside swales that intercept sediment-laden water from heavily trafficked areas. Runoff collected by these interceptor ditches is generally routed to one of the permanent pond facilities located at topographic low points that facilitate gravity drainage. The function of these ponds is described in Section 18.5.3.

18.5.2.1.5 Temporary Contact Water Interceptor Ditches

During construction and though expansion phases of mine operations, temporary (generally consisting of six months to a year) interceptor ditches will be required. Construction and maintenance of these structures should be consistent with that of permanent interceptor structures.

18.5.2.1.6 Water Storage

The permanent ponds for the project are designed to:

- Accumulate all contact runoff and seepage generated in the areas disturbed by mining activities;
- Provide quiescent storage to promote sedimentation; and
- Harvest contact water for re-use in the heap leach process circuit.

The ponds planned to be located immediately downstream of the WRSA will ultimately redirect accumulated contact water to the heap leach circuit via the Lower Dublin South Pond. This routing scheme will also allow for WRSA area-sourced contact waters to be directed to the Mine Water Treatment Plant (MWTP) as required.

Each pond will be equipped with a primary riser-pipe outlet to prevent the release of sediment-laden water, prior to discharge to the environment. Manually operated slide gates will allow the mine operators to hold back collected runoff when poor water quality prohibits discharge to the environment, or when additional site water will be required for process make-up. Secondary discharge capability will be provided by riprap-armoured, broad-crested weirs and spillways notched into each pond. The outlet works will provide the capability to safely discharge pond water that will accumulate during extreme runoff events or emergency events.

This approach is consistent with general industry standard practices for mine water management.

Descriptions of each of the major site ponds and their functionality are provided below.

18.5.3 Storage Ponds

The Eagle Mine site will contain four storage ponds. The purpose of these ponds is to accumulate seepage and runoff generated from the various pits and WRSAs. The four ponds are listed below:

The Eagle Pup pond is located north of the Eagle pup WRSA. It will have a total storage capacity of 35,300 m^3.

The Platinum Gulch pond is located west of the Eagle pit. It will have a total storage capacity of 41,200 m^3.

The Lower Dublin South pond is located east of the camps and truck shops. It will have a total storage capacity of 38,700 m^3.

The Olive pit pond is located north of the Olive pit WRSA. It will have a total storage capacity of 25,700 m^3.

18.5.4 Water Treatment

Active (mechanical and chemical) water treatment facilities to be provided as a part of the project include:

- Potable water treatment plant (PWTP);
- Septic system with leach field for sanitary sewage;
- Cyanide detoxification to treat excess water discharged from the HLP; and
- MWTP to treat site drainage collected at the Lower Dublin South pond and to further treat HLP discharge after it is processed through the CDP.

Cyanide detoxification and MWTP will operate until the water quality of site and HLP drainage is suitable for discharge through passive treatment facilities while maintaining compliance with water quality discharge standards. Water management practices will be used to provide for compliance with water quality discharge standards.

18.5.5 Mine Water Treatment Plant (MWTP)

To meet throughput needs and to provide redundancy, the MWTP will be constructed as two essentially independent trains, each capable of treating up to 300 m^3/h. is the MWTP is scheduled to be constructed in Year 3 of the mine life, so that it can be available to treat flows in Year 4, which is the first year predicted by modeling in which excess water may be released under the median scenario of the water balance model. The MWTP will primarily be a metals removal plant and is intended to treat the site drainage collected at the Lower Dublin South Pond, as well as to provide additional treatment for excess water from the HLP. Treated effluent from the MWTP will drain by gravity to an outfall at Haggart Creek.

The MWTP is intended to operate until heap rinsing, closure, and capping produces water quality at the HLP and Lower Dublin South Pond that will allow discharge either directly or through passive treatment while maintaining compliance with water quality standards. For purposes of this study, it has been estimated that the operations phase of the mine life will cease at the end of Year 9 and that the MWTP will be operated through Year 18.

Treatment goals for the MWTP are based on achieving compliance with Water Use Licence discharge standards and the metal mining effluent regulations (MMER) criteria for end of pipe concentrations.

The treatment at the MWTP will consist of several processes: oxidation, high pH precipitation (lime softening using lime), low pH coagulation (using ferric), pH adjustment, and dechlorination. Filter presses will dewater the solids produced by the high pH precipitation step, and the low pH coagulation step.

18.5.5.1 Potable Water Treatment

Based on an average usage rate of 300 L/d per person, and a camp population of up to 400, the camp water requirements will be approximately 120 m^3/d during construction, and less during operation. The potable water system is not intended to provide process water or as a source of water for firewater.

For redundancy, two potable water wells will be constructed to provide water for the potable uses of the camp and other buildings. A hypochlorite solution storage and feed system will be provided to dose chlorine into the water pipeline, as water is pumped from the groundwater wells to the storage tank. A flow metre on the influent line to the tank will be used to pace the chlorine application rate as needed to maintain a 2 mg/L chlorine residual in the storage tank. The storage tank will be sized at 120 m^3 to provide up to 24-hours water storage. A packaged booster pumping system will be provided to supply water from the storage tank to the camp water system, at a minimum pressure of 70 psi. A small metal building will be constructed to house the hypochlorite solution system and booster pumps.

18.5.5.2 Sewage Treatment

The camp population will generate sanitary sewage at a flow rate less than the potable water supply rate. The current in-situ septic field located at the 100-man camp will be expanded as needed to accommodate the increase to up to 400 personnel during construction.

18.5.5.3 Ice-Rich Material Storage Area

The site design has been optimized to minimize the amount of ice-rich soils that will be disturbed during construction. In addition, ice-rich soil will be re-used as much as possible. However, soil requiring long term storage will be placed in existing excavated containment areas within the placer tailings along Haggart Creek, several kilometres south of the Eagle Creek area. New storage areas located close to the existing excavated containment areas may be required if ice-rich material in excess of the available storage is encountered.

18.6 Process Control and Instrumentation

18.6.1 Overview

The plant control system will consist of a Distributed Control System (DCS) with PC based Operator Interface Stations (OIS) located in three separate control rooms:

- Primary crusher control room;
- Tertiary crusher control room; and
- Central control room (ADR plant).

The DCS, in conjunction with the OIS, will perform all equipment and process interlocking, control, alarming, trending, event logging, and report generation. DCS Input/Output (I/O) cabinets will be located in electrical rooms throughout the plant and interconnected via a plant wide fibre optic network.

Field instrumentation will consist of microprocessor based “smart” type devices. Instruments will be grouped into process areas and wired to local field instrument junction boxes located within those areas. Signal trunk cables will connect the field instrument junction boxes to DCS I/O cabinets.

Intelligent type MCCs will be located in the electrical rooms throughout the plant. MCC remote operation and monitoring will be via Profibus (or other approved industrial communications protocol) interface to the DCS.

Programmable logic controllers or other third party control systems supplied as part of mechanical packages will be interfaced to the plant control system via Ethernet network interfaces.

18.6.2 Communication

A number of integrated systems will be provided for on- and off-site communication at the Eagle Gold project site.

A trunked radio system consisting of hand held, mobile and base radios will provide wide area coverage for on-site communication by operations. The trunked radio system will be interfaced to the on-site voice over internet protocol (VoIP) telephone system.

The VoIP telephone system will feature four-digit dialing within the mine site, access code-based long distance calling, and voice mail services. For connectivity, the telephone system will utilize the site local area network (LAN).

A site LAN will be provided to consolidate services into a single network infrastructure. Computers, cameras, telephones and any IP device requiring connection to the corporate network will utilize the LAN. Further to the hardwired portion of the LAN, wireless access points will be placed in common areas such as the recreation hall, administration area, dining area and construction office.

Voice and data communications to the mine site will be established via a microwave radio link. A tower mounted microwave antenna and radio equipment at the site, along with a repeater station that is proposed to be installed at Mount Haldane, will be utilized to establish a voice and data link to Mayo where Total North has an established communication network.

18.7 Mobile Support Equipment

Mobile site support equipment provides support to operations at the Eagle site. A list of site support equipment is provided in Table 18.2. The support equipment fleet is based on similar equipment utilized at other northern Canadian mining operations.

Table 18.2: Site Support Equipment

Equipment Description	Quantity	Comments
Light Vehicles & Passenger Movement		
1 T Diesel Crew Cab Pick-up	8	
44 Passenger Bus	1	
Cranes		
50T Rough Terrain Crane	1	Larger crane will be rented as/when required
Mine Support Equipment		
Tire manipulator (966 attachment)	1	
Mechanics Truck	2	
Welding Service Truck	1	
Site Services Equipment		
Skid Steer Loader (1Cu.M)	2	
65ft Man-Lift	1	
Plow/Sand/Dump Truck	1	Including attachments
Large Tool Carrier (Cat 966K)	1	
Roll-off truck incl. Accessories	1	
Snowcat	1	
Materials Management		
Small Tool Carrier	1	
3 T Forklift - Warehouse	1	
Winch Tractor	1	
Flat Deck Trailer	2	
20T Flat deck truck w/ rigid boom crane	1	
Misc. Small Equipment		
Portable Diesel Heaters	3	
25kW Generators	2	
Rescue Vehicles		
Ambulance / Rescue - Ford F450	1	
Fire Truck	1	

Source: JDS (2016)

18.8 Manpower

The site support manpower crew (Table 18.3) will provide support to the operations and will be responsible for the following activities:

- Infrastructure facilities maintenance and repairs;
- Transferring of freight from the storage areas to the warehouse and operation centres;
- Personnel/baggage handling between the camp and busses;
- Inbound and outbound freight handling;
- Waste management duties (incineration, water treatment, hazardous waste handling);
- Plant site snow removal; and
- Site surface water management.

Table 18.3: Facilities Operation & Maintenance Manpower

Position	Staff	Rotation	Average On-Site
Facilities Manager	1	4x3	1
Site Services Foreman	2	2x2	1
Electrician	2	2x2	1
Carpenters	2	2x2	1
Multi-Equipment Operator	4	2x2	2
Skilled Labourers	4	2x2	2
Total Site Support	15		8

Source: JDS (2016)

19 Market Studies and Contracts

19.1 Market Studies

Detailed market studies on the potential sale of gold from the Eagle Gold project were not completed. JDS confirmed the refining and payable terms with a leading industry entity in order to determine indicative terms with respect to the doré to be produced. The terms were reviewed and found to be acceptable by QP Gord Doerksen, P.Eng.

No contractual arrangements for shipping, port usage, or refining exist at this time. Table 19.1 outlines the terms used in the economic analysis.

Table 19.1: NSR Assumptions Used in the Economic Analysis

Assumptions	Unit	Value
Au Payable	%	99.5
Au Refining Charge	US$/oz	10

Source: JDS 2016

19.2 Contracts and Royalties

There are no known significant contracts entered into by StrataGold that would impact the results of this study.

The Dublin Gulch property is subject to three underlying agreements, two of which are material to the Eagle Gold project.

The Eagle deposit falls entirely within claims that are subject to a royalty historically known as the Mar Gold Zone Royalty. This royalty requires minimum annual royalty payments of $20,000 or a production royalty of 2% of the gross returns received from the sale of all metals produced from the claims to a maximum of $1,000,000 after which the royalty reverts to 1% with no end price.

A portion of the Olive deposit falls within a claim that is subject to the Queenstake Mar Tungsten Royalty. This royalty is a 1% net smelter return royalty payable only upon the commencement of production.

Other than the two royalties described above, the project is free and clear of any liens or third party interests.

Total third party royalties for the project amount to $ 30M over the LOM.

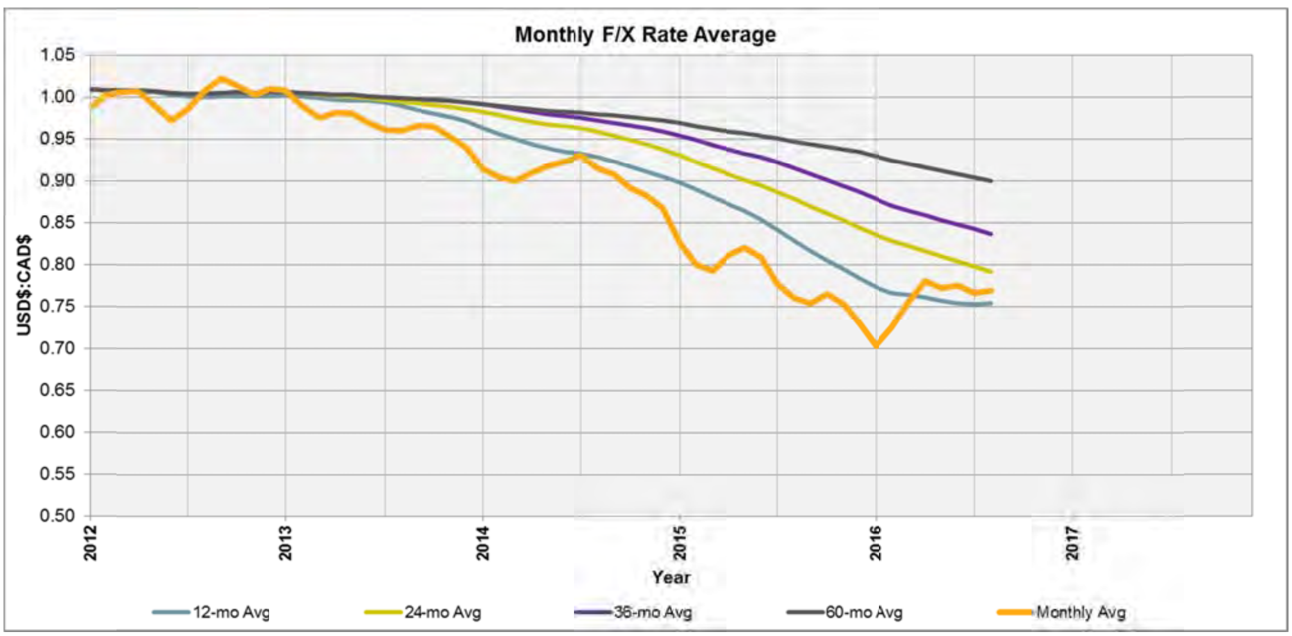
19.3 Metal Prices

The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical gold prices are shown in Figure 19.1 and demonstrate the change in metal prices from 2000 to 2016. Historical average US$:C$ exchange rates are shown in Figure 19.2.

Figure 19.1: Historical Gold Price

Source: Kitco 2016

Figure 19.2: Monthly Average US$:C$ Foreign Exchange Rate – Bank of Canada



Source: JDS 2016

The gold price used in the economic analysis is based on the 6-month trailing average spot rate during July 2016 sourced from Kitco Metals Inc. The US$:C$ exchange rate used in the economic analysis is based on the 3-month trailing average as at June 2016. A sensitivity analysis was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Table 19.2 outlines the metal price and exchange rate used in the economic analysis.

Table 19.2: Metal Price and Exchange Rate used in the Economic Analysis

Assumptions	Unit	Value
Gold Price	US$/oz	1,250
Exchange Rate	US$:C$	0.78

Source: JDS 2016

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Assessment and Permitting

20.1.1 Overview

Prior to construction or operational activities taking place in Yukon, a mining project essentially has to complete three major steps: the collection of a robust environmental and socio-economic baseline dataset; the successful completion of an assessment and a positive record of decision regarding potential effects of the project on valued environmental and socio-economic components; and, the application for and acquisition of regulatory approvals.

Victoria Gold concluded all three major steps for the project and has received positive Decision Documents upon the completion of the assessment of the project under the Yukon Environmental and Socio-Economic Assessment Act (YESAA) in 2013. Victoria Gold also holds both a Quartz Mining Licence and a Type A Water Use Licence that collectively allow for the construction, operation and closure of the Eagle Gold project.

20.1.2 Completed Environmental Assessment

In December 2010, Victoria Gold submitted a project proposal to the Yukon Environmental and Socio-Economic Assessment Board (YESAB) to begin the environmental assessment process of the Eagle Gold Project. The Project assessed by the YESAB included consideration of the construction and operation of the Eagle open pit, the Ann Gulch HLP, the two WRSAs located north and west of the open pit, and all facilities and activities required to support mining operations.

On February 19, 2013 the Executive Committee of the YESAB concluded its assessment of the project pursuant to the YESAA. As a result of the assessment, the Executive Committee recommended to the Decision Bodies that the project be allowed to proceed without a review, subject to the terms and conditions identified in the Screening Report and Recommendation for Project Assessment 2010-0267.

On April 6, 2013 Yukon Government (YG) exercised its authority as per YESAA s.75 or s.76 to issue a Decision Document for the project. The YG Decision Document, premised on the commitments made by Victoria Gold as detailed in the Screening Report and Recommendation, agreed with the 123 terms and conditions (recommendations) proposed by the YESAB Executive Committee without variation.

On April 19, 2013 a consolidated Decision Document was completed by federal decision bodies as required under YESAA s.74(1). Fisheries and Oceans Canada, Natural Resources Canada and Transport Canada, in their capacity as the federal decision bodies identified for the project and pursuant to s.76(1)(a) of YESAA, issued a Decision Document which accepted the recommendation that the project be allowed to proceed without a review, subject to the terms and conditions identified in section 19.0 of the Screening Report and Recommendation.

The federal decision bodies were in agreement with the rationale for the recommendation as expressed in the Screening Report and Recommendation.

The completion of the environmental assessment allowed Victoria Gold to enter the regulatory phase for the project.

20.1.3 Quartz Mining License

On September 20, 2013, the YG Department of Energy, Mines and Resources issued a Quartz Mining License (QML) for the project. The scope of authorization involves the development, production, reclamation and closure of an open pit mine and gold extraction through heap leaching involving ore crushing, cyanide leaching and a carbon adsorption, desorption and recovery in accordance with the terms and conditions set out in the QML and the approved plans listed in the QML. The QML was subsequently amended on March 24, 2016 to align with the timeline and requirements of a Type A Water Use License issued for the project.

20.1.4 Type A Water Use License

On December 3, 2015, the Yukon Water Board issued a Type A Water Use License (WUL) for the project. The Type A WUL specifies the quantity of water that can be used for all aspects of the project and includes criteria that must be met for discharge of water from the project site. The Type A WUL includes the approval of a range of plans and activities that are also contemplated in the QML and affirms that the plan for construction, operation and closure of the project represents industry standard practice and can move forward subject to certain terms and conditions.

Both the Type A WUL and QML require the submission of detailed reclamation and closure plans that describe the measures an applicant will take to return the mine site to functional and sustainable ecosystems. Victoria Gold has submitted these reclamation and closure plans which described the covering and revegetation of all disturbance land surfaces, except for the open pits, the draindown, rinsing and treatment of the HLP, the treatment of mine contact waters, and the subsequent monitoring of the project to ensure closure objectives are met.

The reclamation and closure planning required by the regulatory agencies also requires the submission of estimates for a third party to undertake the proposed reclamation activities.

20.1.5 Additional Environmental Assessment and Permitting

The YESAA includes certain triggers related to the alteration of a project which subsequently require additional assessment of a project to ensure environmental and socio-economic values can be protected. The inclusion of the Olive pit, the olive WRSA, and the secondary HLP in the mine plan will mean that these activities have to be assessed by the YESAB.

The assessment of these activities does not impact Victoria Gold's ability to commence with previously assessed and licensed work (i.e. mining the Eagle Zone and the use of the WRSAs and HLP primarily associated with Eagle material). Based on the current mine plan, the completion of the three major steps to mine approval in Yukon can feasibly be accomplished well in advance of these facilities being required.

20.1.6 Additional Federal and Territorial Permits, Licenses and Authorizations

Table 20.1 provides a list of the federal and territorial act, regulations and guidelines that may apply to the project at various stages of development, operations and closure.

Table 20.1: List of Relevant Federal and Territorial Acts, Regulations and Guidelines

Applicable Legislation/Regulations	Permit – Approval	Responsible Agency	Expiry Date
Quartz Mining Act	Quartz Mining License	Energy Mines and Resources, Yukon Government	September 20, 2040
Quartz Mining Act Quartz Mining Land Use Regulations	Class IV Mining Land Use Approval	Energy Mines and Resources, Yukon Government	May 10, 2021
Waters Act Waters Regulation	Water License – Type A	Yukon Water Board	September 10, 2040
Waters Act Waters Regulation	Water License – Type B	Yukon Water Board	September 10, 2040
Fisheries Act Metal Mining Effluent Regulations	Section 35(2) Authorization	Fisheries and Oceans Canada	
Navigable Waters Protection Act	Section 5(2) Approval – to be determined	Transport Canada	
Yukon Public Utilities Act	Energy Certificate and Operating Certificate	Yukon Government	
Highways Act Highways Regulations	Work in Highway Right of Way Permit, Access Permit	HPW, Yukon Government	March 31, 2017
Territorial Lands (Yukon) Act Land Use Regulations Quarry Regulations	Land Use Permit Quarry Permit Timber Permit	Energy Mines and Resources, Yukon Government	
Environment Act Air Emission Regulations Special Waste Regulations Solid Waste Regulations Storage Tank Regulations Contaminated Sites Regulations	Air Emissions Permit Special Waste Permit Land Treatment Facility Permit Storage Tank Systems Permit	Environment Yukon, Yukon Government Community Services, Yukon Government	December 31, 2016
Forest Protection Act Forest Protection Regulations	Burning Permit	Community Services, Yukon Government	
Highways Act Bulk Commodity Haul Regulations Highways Regulations	Highways Hauling Permit	HPW, Yukon Government	
Yukon Historic Resources Act	Archaeological Sites Permit	Tourism and Culture, Yukon Government	
Dangerous Goods Transport Act	Permit – certificate for transport of dangerous goods	HPW, Yukon Government	

Applicable Legislation/Regulations	Permit – Approval	Responsible Agency	Expiry Date
Explosives Act and Regulations	Blasting permit, Magazine License, Factory License, ANFO Certificate, Purchase and Possession Permit, Permit to Transport Explosives	Natural Resources Canada, Explosives Regulatory Division and Minerals and Metals Sector	
Occupational Health and Safety Act Occupational Health & Safety Regulations	Blaster's Permit	Workers' Compensation Health and Safety Board	
Species at Risk Act	N/A	Environment Canada	
Wildlife Act	N/A	Environment Yukon, Yukon Government	
Canadian Environmental Protection Act	N/A	Environment Canada and Health Canada	
Migratory Birds Convention Act Regulations Respecting the Protection of Migratory Birds	N/A	Environment Canada	
Building Standards Act Electrical Protection Act	Building Permit, Plumbing Permit	Community Services, Building Safety, Yukon Government	Granted for 100 person camp.
Gas Burning Devices Act	Gas Installation Permit Gas Burning Devices Permit	Community Services, Building Safety, Yukon Government	
Boiler and Pressure Vessel Act	Pressure Vessel Boiler Permit	Community Services, Building Safety, Yukon Government	
Yukon Public Health and Safety Act Regulations Respecting Public Health	Compliance with Public Health Regulations	Health and Social Services, Environmental Health Services	

Source: Victoria Gold (2016)

20.2 Environmental and Socio-economic Baseline Studies

From 2007 onwards, Victoria Gold and its predecessor, StrataGold has prepared (and now maintains) a comprehensive set of baseline studies for climate, hydrology, soils, surficial geology, vegetation, wildlife, groundwater, water quality, aquatic ecology, socio-economic conditions, historical, and paleontology resources. Additional water and climate data is currently being collected. The baseline characterization included historical data sets collected from 1993 to 1996 and was supported by regional analyses.

The baseline data collection covered various geographical extents, depending on the component under study. In general, each technical discipline defined local and regional study areas to frame the spatial scope of their assessment. Data collection was focused within the footprint and surrounding areas of the project for the local study areas; regional study areas were defined based on information such as species ranges, watershed boundaries, geologic units, and community administrative boundaries, depending on the component under study.

To support the assessment and licensing of components of the project related to the Olive Zone, it is anticipated that some additional work in the disciplines of hydrology, water quality, groundwater flow and groundwater quality will be required.

Victoria Gold commenced the additional data collection in 2016 to support the inclusion of the Olive Zone and associated facilities in future amendments to their existing permits.

20.2.1 Climate

The Dublin Gulch 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 between 30 to 40% of the annual precipitation. Lower elevations are typically snow-free before May, while snow remains in higher elevations until mid-June. Frost action may occur at any time during the spring, summer or fall.

Regional climatic data are available from several stations in the area including Mayo, Keno Hill, Dawson, Klondike and Elsa, as well as other relevant long-term data from other areas within Yukon (e.g. Whitehorse). Historical climatic information of the Project site was available from 1993 to 1996. Climate data collection was renewed in August 2007 at the Potato Hills climate station site (1,420 masl), and a second climate station (Camp station - 778 masl) was installed in August 2009 near the existing camp Climate data from the Potato Hills and Camp climate stations are collected at 15-minute intervals and data are available for the period from August 2007 through April 2016 and August 2009 through April 2016, respectively.

20.2.1.1 Temperature

The recorded mean annual temperatures have ranged from -2.0 to -5.1°C. July is typically the warmest month with mean July temperatures at the Camp station ranging from 12.6 to 13.6°C and from 8.1 to 11.6°C at the Potato Hills station during the period of record. The coldest temperatures are generally experienced in January and the Camp station recorded a range of monthly mean temperatures from -17.1 to -25.2°C and the Potato Hills station recorded a range of monthly mean temperatures of -15.5 to -19.8°C for the month of January.

During the period in which the Potato Hills and Camp stations have collected data simultaneously, the higher Potato Hills station has generally reported colder temperatures than the lower Camp station; however, autumn and winter temperature inversions do occur at the site as is common in mountainous regions, and the Camp station has a much larger range in recorded temperature. The maximum recorded temperature on-site was 29.3°C in August 2010 at the Camp station and the minimum recorded temperature was -42.8°C in January 2012 also at the Camp station.

20.2.1.2 Precipitation

The estimated mean annual rainfall at the Project site ranges from 223 mm and 255 mm for the Camp and Potato Hills stations respectively. Snow water equivalent values, calculated during site snow surveys, show annual maximum values of 79 mm to 161 mm for the camp station and 167 mm to 410 mm near the Potato Hills station. Rainfall, snowfall, and surface lying moisture and snow are natural dust suppressants and as such, the area is not prone to prolonged dusty periods.

Based on the regional and local data, monthly precipitation totals are highest in July and lowest in February. Snowfall typically begins in late September and continues until May.

20.2.1.3 Wind Speed and Direction

The predominant wind direction at the site climate stations is from the north, and west-northwest, for the Camp and Potato Hills stations, respectively. Wind speeds average 1.2 m/s at the Camp station, and 2.3 m/s at the Potato Hills station, on an annual basis. The maximum recorded gust speed at the Camp station was 15.1 m/s, and 23.9 m/s at the Potato Hills station (Lorax 2016).

20.2.2 Surficial Geology and Soils

20.2.2.1 Surficial Geology

The surficial geology of the project area has been substantially affected by historic glaciation over 200,000 years ago, including two major glaciation episodes in the Quaternary period; the pre-Reid (~2.5Ma-400ka BP) and the Reid (~200 ka BP) (Bond 1997; 1998a; b). Glacial limits are provided in Figure 3.1-1. In each case, ice likely originated from the Ogilvie and Wernecke Mountains, with glaciations being more extensive during the pre-Reid period.

Preservation of pre-Reid glacial deposits and landforms is rare. A few intact deposits and diorite erratics at high elevations are the only records left (Bond 1998a). Glacial deposits from the Reid glaciation are moderately preserved. Colluvium, alluvium, and small areas of shallow organics drape the Reid glacial sediments and the interglacial sediments throughout the area.

Dominant surficial materials within the project area are weathered bedrock and colluvium. Competent bedrock outcrops are rare, as sufficient geologic time has passed to allow extensive weathering of exposed rock.

20.2.2.2 Soils

The largest influence on soil development in the area of the project is climate, and the resulting permafrost which is discontinuous throughout the area. Despite over 200,000 years of soil development, pedogenic processes have been slow due to the cold climate and to the short growing season for vegetation, resulting in a predominance of ice-affected and relatively undeveloped soils (Cryosols and Brunisols).

Non-frozen soils encountered in the area of the Project include Brunisols, minor areas of Luvisols (on fine-textured till), and Gleysols (on poorly and imperfectly drained materials). The majority of the soil textures in the area are sandy-silt to silty sand loam matrix with angular or tabular coarse fragments ranging from gravels to boulders.

Soil in the project area is limited for reclamation suitability primarily by high coarse-fragment content, due to development of soils from weathered bedrock. Rooting depths are on average 50 cm, but can reach depths of over 120 cm.

20.2.2.3 Permafrost

The project site is located in a region of widespread discontinuous permafrost (Brown, 1979). On the regional scale, permafrost distribution is typically controlled by mean annual temperature and precipitation, whereas on a local scale it is controlled by vegetation, surface sediments, soil moisture, slope aspect, and snow depth. Within the project area, frozen ground occurs typically on north- and east-facing slopes, and within poorly drained areas lower in the valleys. The distribution and thickness of frozen ground is highly variable across the site.

Frozen ground, when observed, is generally encountered immediately below the organic cover. Ground temperatures have been measured with thermistors installed on-site in 1995-1996, and 2009-2012. The measured ground temperatures showed the frozen ground to be relatively warm when observed, typically between 0°C and -1°C.

20.2.3 Hydrology

The hydrology of the region is generally characterized by large snowmelt runoffs during freshet in May, which quickly taper off to low summer stream flows interspersed with periodic increases in stream flow associated with intense rainfall events during July and August. The pattern of low stream flows punctuated by high stream flows associated with rain fall events continues throughout the summer to autumn when freeze up beings in October. In larger streams, base flows are maintained below river/creek ice throughout the winter by groundwater contributions. Smaller streams tend to dry up during the late summer or fall, as flow generally goes subsurface when the groundwater table drops to seasonally low levels. Aufeis (or overflow) ice may build in certain places in stream channels if groundwater emerges during winter.

20.2.4 Surface Water Quality and Aquatic Biota

The water quality study area includes the Haggart Creek, Dublin Gulch, Eagle Creek basins, which have been subject to placer mining in the past and Lynx Creek basin, which has not been subject to placer mining. A total of 21 monitoring stations have been sampled within the study area during the StrataGold and Victoria Gold baseline data collections program. Sites within the Haggart Creek, Dublin Gulch, and Eagle Creek drainage basins were selected upstream and downstream of the proposed project footprint, where possible. Lynx Creek drains a large catchment to the south of the project area that will be unaffected by development activities.

All sites, except those located in Dublin Gulch, had high acid buffering capacity, as indicated by high alkalinity, calcium, and hardness. Turbidity and total suspended solids levels tended to be low with some exceptions noted at several sites depending on season and year. Nutrient levels tended to be low and suggestive of oligotrophic levels, with measurable amounts of nitrate, and low levels of phosphate and dissolved organic carbon.

Metals levels and naturally high arsenic concentrations in water and sediment, in addition to abundances and taxonomic compositions of periphyton and benthic invertebrates are consistent with a mineralized area and reflect previous disturbance of substrates during placer mining.

Metals data for the fine (less than 63 μm) sediment fraction were similar to the water quality data in terms of high levels of arsenic at all sites as well as cadmium, chromium, copper, lead, mercury, and zinc at certain times. For periphyton, chlorophyll levels suggest oligotrophic conditions highest richness, diversity, and evenness indices were recorded in Haggart Creek, suggesting better water quality than in Dublin Gulch, Eagle Pup, or Lynx Creek.

20.2.5 Groundwater

20.2.5.1 Hydrogeologic Setting

There are two principal water-bearing units in the Project area: deeper relatively low-permeability bedrock and the near surface moderately permeable surficial deposits. Surficial material at the Project site consists of a thin veneer of organic soils underlain by colluvium (i.e., a loose heterogeneous mass of soil material), glaciofluvial (i.e., originating from rivers associated with glaciers) deposits, or till (a glacial deposit). Below these clastic (or transported broken fragments of rock) units are either metasedimentary or Granodiorite bedrock, which is deeply weathered in places. The elongated Granodiorite stock (ore bearing unit) has intruded the surrounding host metasediment. The surficial material thickness and physical properties varies significantly throughout the area. Recorded depths to bedrock in the project area range from 0 m to greater than 20 m.

The Dublin Gulch valley contains large amounts of fluvial (i.e., river deposited) materials that were considerably reworked by placer mining operations. Extensive stockpiles of placer deposits comprised of sub-rounded metasediment and Granodiorite clasts, ranging in size from sands to boulders, and fine-grained material (i.e., that are located in former placer settling ponds) are present adjacent to the Dublin Gulch and Eagle Creek watercourses. A till blanket covered with a colluvial veneer is located along the south valley wall in Dublin Gulch valley and extends southward in the Haggart Creek valley. A recent alluvial (i.e., a water-laid clastic deposit) fan is present where Dublin Gulch meets Haggart Creek.

Discontinuous permafrost is also present, especially on the north-facing slopes and affects the connectivity between the deep and shallow water-bearing zones in places.

20.2.5.2 Groundwater Occurrence

Generally groundwater has been observed deeper (approximately >6 m below ground) at higher elevations and shallow to artesian in lower elevations and in valley bottoms. Springs and seeps have been observed in a few locations where valley bottoms have narrowed. These are typically associated with the re-emergence of a stream from channel deposits (i.e., a gaining reach). In these instances (e.g., Eagle Pup, Stewart Gulch), thin alluvium overlying shallow bedrock is the likely cause of the emergence. Groundwater levels within the lower Dublin Gulch valley have been observed to have seasonally delayed trends due to higher groundwater levels during spring freshet and/or associated with rainstorms and lower groundwater levels during dry summer periods.

20.2.5.3 Groundwater Flow

Groundwater flow in the bedrock occurs in fractures and fault zones, while preferentially flowing through more permeable (and porous) sediments within the surficial deposits. General orientation of groundwater flow contours mimic the topography of the site as groundwater flows from the highest areas to lowest. Throughout most of the area the groundwater divides of each sub-basin approximately coincide with the surface water divides (i.e., groundwater from the Eagle Pup and Suttles Gulch drain to Eagle Creek, while groundwater from Ann and Stewart Gulch Basins drain to Dublin Gulch). In the lower Dublin Gulch valley the groundwater divide between the Eagle Creek and Dublin Gulch basins in the placer tailings is not clearly defined.

Field observations suggest that at times the divide migrates across the valley so that groundwater from the Dublin Gulch basin may flow towards Eagle Creek. This shifting is seasonal and also due in part to the variability in the timing of the freshet and/or rainfall events across the entire watershed.

Groundwater recharge occurs at higher elevations throughout the Dublin Gulch-Eagle Creek drainage basin and ultimately discharges to surface water (in some cases as seeps and springs) at lower elevations in the valley or directly to surface streams, or ultimately into Haggart Creek. The main groundwater flow in conjunction with the highest groundwater elevations is expected to occur during the snowmelt in late spring (e.g., May to June) after thawing of the shallow sediment.

20.2.5.4 Surface Water - Groundwater Connectivity

Base flow values represent the groundwater contributions to streams. Groundwater contributes to stream flows where the groundwater table elevation intersects the ground surface, typically these intersections are located in stream channel inverts (e.g., Eagle Pup appears in mid-channel where the valley is well confined by bedrock); however, they also appear as seepage from slopes within the placer deposits of the lower Dublin Gulch valley. Groundwater from the lower Dublin Gulch valley likely contributes a measureable portion of the baseflow to Haggart Creek. The baseflow contributions to the streams maintain flow in the larger creeks during the drier months of the year (including winter flows).

20.2.5.5 Groundwater Flow Properties

Hydraulic conductivities ranged from 10-3 m/s to 10-7 m/s in the surficial material, and from 10-5 m/s to 10-8 m/s in the bedrock. The hydraulic conductivity of the colluvial, alluvial, and till deposits was generally higher than that of the placer material, and the variable hydraulic conductivity seen in the bedrock is typical of fractured crystalline rock, which showed decreasing hydraulic conductivity with depth. The test data did not demonstrate a measureable difference in the hydraulic conductivities of Granodiorite and metasedimentary rock. This suggests that the flow properties of both rock types are similar.

20.2.6 Groundwater Quality

The groundwater quality data suggests that the chemical composition of groundwater depends on the local and up-gradient rock-types. The following parameters naturally exceeded the CCME and/or CSR guidance (used for reference only) in the project area: aluminum, arsenic, cadmium, copper, iron, lead, molybdenum, nickel, selenium, silver, and/or zinc. The CSR guideline values apply to both surface and groundwater, whereas the CCME guidelines only apply to surface water.

However, as groundwater ultimately discharges to surface water bodies, the CCME guideline values were considered for reference.

The groundwater samples were classified based on their major ion chemical composition, taking into account the major anions and cations. Calcium is the dominating cation in most groundwater samples from the site; however, some sampling locations magnesium concentrations exceeded calcium. Carbonate was the dominating anion in all samples, and was particularly high in some samples.

The exceedances do not imply that the groundwater at the site is contaminated; only that background concentrations of these parameters are higher than typically found in other natural sites in Canada, and merely reflect the natural geologic and hydrogeologic conditions within these specific areas of the project area.

Comparison of the multiple years of groundwater data indicated that groundwater quality parameters were generally in the same range and that seasonal trends were not apparent over the years sampled.

20.2.7 Fisheries

Baseline fish and fish habitat information was gathered from existing consultant reports, government databases, and the results of field studies conducted for the project prior to StrataGold's claim ownership. Field studies were completed for watercourses located within the local project area to obtain biophysical habitat data, determine fish presence and abundance, and characterize fish populations (i.e., size, age, and tissue metal concentrations).

At least 11 fish species are known to occur in the South McQuesten River watershed, including Chinook salmon (Oncorhynchus tshawytscha), Arctic grayling (Thymallus arcticus), northern pike (Esox lucius), longnose sucker (Catostomus catostomus), Arctic lamprey (Lampetra camtschatica), burbot (Lota lota), slimy sculpin (Cottus cognatus), round whitefish (Prosopium cylindraceum), inconnu (Stenodus leucichthys), lake whitefish (Coregonus clupeaformis), and rainbow trout

(Oncorhynchus mykiss) (DFO 2010). No freshwater fish species on Schedules 1 or 2 of the Federal Species at Risk Act (SARA) are present in the South McQuesten River watershed or the entire Yukon Territory (Government of Canada 2012). Haggart and Lynx creeks are both known to contain five fish species: Chinook salmon, Arctic grayling, round whitefish, burbot, and slimy sculpin (DFO 2010). Ironrust Creek, Dublin Gulch and Eagle Pup are known to be inhabited by Arctic grayling and slimy sculpin (Hallam Knight Piésold 1996b, DFO 2010).

Fish tissues (from both Arctic grayling and slimy sculpin) were tested for metal concentrations in three of the fish bearing water courses. Although metal concentrations in tissues were high, they did not, with the exception of selenium concentrations in Arctic grayling liver, exceed the lower-limits set by BC Guidelines for the Protection of Aquatic Life.

20.2.8 Wildlife

The project site is located in the Mayo Lake-Ross River Ecoregion and contains two ecological zones, Subalpine and Forested. Both of these zones serve as habitat for wildlife. To characterize wildlife use of these areas, existing literature, field studies and discussions with wildlife biologists in the region and with the NND was conducted.

A total of 31 individual species were recorded using data from all sources. Mammals present include two ungulate species (moose, woodland caribou), two bear species (black bear, grizzly bear), and an assortment of small to medium size mammals including gray wolf, wolverine, red fox, American marten, snowshoe hare, and lemming. Moose was the most commonly detected mammal species. It was found across all survey types and a wide range of habitat types, indicating a relatively wide distribution in the area. Most detections were in lower-elevation forested habitat zones likely used all year long.

These areas contain riparian areas, marshes, and deciduous forest stands which contain preferred food sources and offer thermal protection in winter. The study's moose detections are consistent with the reports from the NND—the area provides winter habitat for moose and is important for moose hunting. Aerial and ground surveys and telemetry data suggest that while woodland caribou make some use of the study area, it does not represent core habitat for them.

Snowshoe hare, red squirrel, and ptarmigan were the most commonly detected mammal species after moose. This is of interest as all three species represent potential prey for a range of larger mammals (e.g. lynx, wolf, and red fox), and raptor species such as Golden Eagle. While formal bird surveys have not been carried out, eighteen bird species were detected in the study area including Golden Eagle, Gyrfalcon, Trumpeter Swan, Dusky Grouse, Common Raven, Ptarmigan, and Grey Jay.

20.2.9 Vegetation

Two ecological zones were delineated in the baseline study areas: the Subalpine zone and the Forested (Boreal) zone. The majority of project activities occur in the Forested zone. The Subalpine zone occurs on the ridge tops and high plateaus above approximately 1,225 masl. Tree cover is discontinuous or absent at this elevation, and the vegetation is dominated by dwarf birch, willows, ericaceous shrubs, herbs, mosses, and lichens. The highest points within the three study areas are 1,520 masl. These upper elevations are dominated by dwarf-shrub, heath and lichen communities.

The Forested zone, which is part of the northern boreal forest (Boreal Cordillera Ecoregion), includes the valley bottoms, and the slopes of the mountains below the treeline. The elevation range of this zone in the three study areas is 600 masl up to the Subalpine zone, about 1,225 masl. Open canopy stands of black spruce are generally present on moist sites and on the lower portions of north-facing slopes. However, coniferous dominated forests consisting of white and black spruce are found along creeks and rivers and on well drained sites. Ericaceous shrubs and feather mosses are most common in the understory of the coniferous forests. On the upper slopes, open subalpine fir stands are predominant with trees becoming smaller and more spread out with increasing elevation; the cover of willows, dwarf birch and ericaceous shrubs increase as the canopy opens. Mixed forests, consisting of white spruce, trembling aspen, and Alaska birch are also present on warm aspects or near-mesic sites that have been disturbed by forest fire. Small deciduous stands dominated by aspen (warm aspects) and Alaska birch are also occasionally present in the study area.

While no existing rare plants were found through queries of government databases past surveys, one rare plant, island purslane (Koenigia islandica), was found in the study area, a 2 m by 2 m patch of Koenigia islandica L. (island purslane). This plant is considered “imperiled” in Yukon. All foliar samples analyzed contained metal concentrations below levels considered toxic for cattle.

20.2.10 Social Environment

20.2.10.1 First Nation of Na-Cho Nyäk Dun

The FNNND (which translates as Big River People) represents the most northerly community of the Northern Tutchone language and culture group in the Yukon. In the Northern Tutchone language, the Stewart River is called Na-Cho Nyäk, meaning Big River. The FNNND is culturally affiliated with the Northern Tutchone people of the Pelly Selkirk, and the Carmacks Little Salmon First Nations; these three First Nations form the Northern Tutchone Tribal Council. The FNNND constitutes much of the community of Mayo, and their Traditional Territory covers 162,456 km2 of land (131,599 km2 in Yukon and 30,857 km2 in Northwest Territories). Under the 1993 land claims agreement, the First Nation owns 4,739.68 km2 of settlement lands.

Traditionally, FNNND citizens lived and trapped throughout the area surrounding Mayo.

As a self-governing First Nation (under the FNNND Final Agreement and Self-Government Agreements), the FNNND has the ability to make laws on behalf of their citizens and their lands. Under their Final Agreement, FNNND owns the minerals under all Category A Settlement Lands, and receives royalties from any mining on this land. For mining activity elsewhere in the FNNND Traditional Territory, including on Category B Settlement Lands, the FNNND Government shares in a portion of any mineral royalties collected by the Yukon Government.

20.2.10.2 Comprehensive Cooperation and Benefits Agreement

VGC and the FNNND signed a comprehensive Cooperation and Benefits Agreement (CBA) on October 17, 2011. The CBA replaced an earlier Exploration Cooperation Agreement and applies to the Eagle Gold mine development and exploration activities conducted by VGC (including subsidiaries) anywhere in FNNND Traditional Territory located south of the Wernecke Mountains.

The objectives of the CBA are to:

- Promote effective and efficient communication between VGC and the FNNND in order to foster the development of a cooperative and respectful relationship and FNNND support of VGC's exploration activities and the project;
- Provide business and employment opportunities, related to the project, to the FNNND and its citizens and businesses in order to promote their economic self-reliance;
- Establish a role for the FNNND in the environmental monitoring of the project and the promotion of environmental stewardship;
- Set out financial provisions to enable the FNNND to participate in the opportunities and benefits related to the project; and
- Establish a forum for VGC and the FNNND to discuss matters related to the project and resolve issues related to implementation of the CBA.

20.2.10.3 Village of Mayo

The village of Mayo is located 407 km north of Whitehorse and 235 km east of Dawson City. Mayo is situated at the confluence of the Mayo and Stewart Rivers within the Traditional Territory of the FNNND. Historically, the site of Mayo was used as a traditional camp by the FNNND.

Prior to becoming a service centre for significant mining activity in the area, Mayo was established as a river settlement as it was the farthest navigable point up the Mayo and Stewart Rivers by steamboat. The permanent community of Mayo Landing was established in 1903 (Bleiler, et al. 2006), and was incorporated as a village in 1984.

The administration of the village of Mayo consists of a mayor, a Chief Administrative Officer, and four councilors. For planning purposes, the village of Mayo uses a population of 466 persons (although this figure includes those who live outside the village boundaries). This figure also includes both the Aboriginal population (FNNND citizens and other Aboriginal people) and the non-Aboriginal population. In 2010, the village had an annual budget of approximately $3.4 million and employed seven full-time and two part-time staff. In the summer season, as many as 12 to 15 other individuals are employed by the village, including students.

Property taxes and grants in lieu provided by other levels of government comprise some of the municipal revenue of the village of Mayo.

20.2.10.4 Employment and Economic Opportunities

There are a number of quartz mining claims, exploration projects, and proposed mining projects in the region. Minerals of interest include gold, silver, zinc, lead, and copper. Recently, the Mayo area has experienced a surge in mineral exploration and development (e.g., Alexco Resource Corporation's proposed Bellekeno Mine [silver] and other Keno Hill Silver District interests; ATAC Resources' Rau Gold project), and the Elsa Reclamation and Redevelopment Company's (a subsidiary of Alexco Resource Corp.) reclamation and closure of historical mines in the district.

Placer mining continues to be a major contributor to the economy of the area. The majority of Mayo area placer mining operations are family-run, some for three or more generations. Following the mining downturn in the 1980s, it was realized that diversification to include tourism, outfitting, recreation, and other economic activities would reduce Mayo‘s reliance on a mineral-based economy.

Mayo‘s economy is beginning to focus on the provision of various services, including government services, to its residents and to individuals living in the surrounding area (village of Mayo 2006). Tourism is becoming a growing segment of the local economy.

20.2.10.5 Traditional Activities and Culture

The FNNND has prepared a 5-year strategic heritage development plan (FNNND 2007) that identifies priorities relating to traditional knowledge, language, heritage sites and special places, a cultural centre, governance policy and guidelines development. An implementation plan was also prepared. While FNNND staff noted that the plan is somewhat dated, it is still used as a planning guide by FNNND.

At community meetings, FNNND citizens noted the importance of several areas in the vicinity of the project for traditional activities including hunting, fishing, trapping, and gathering. FNNND elders and staff indicated that citizens still rely on traditional foods—berries, fish, moose, deer, small game, and birds—as a significant portion of their diet. These traditional foods are shared with those who may not be able to obtain it directly (e.g., single mothers, elders).

Hunting, fishing, and harvesting are also very important aspects of Northern Tutchone culture and diet, and for continued monitoring of the land. Northern Tutchone people have always relied heavily on the foods of the forests and the rivers. Moose, caribou, sheep, grouse and fish, as well as many types of plants and berries are harvested and preserved to last through the seasons.

The FNNND also offers a number of on the land programs, including day-trips for medicine gathering, fishing and hunting camps for youth, and an archaeological camp, as well as some longer trips. Programs for jigging, beading and other craft work are also offered.

Ongoing activities organized by the FNNND include:

- Traditional food lunches at the school;
- Teacher cultural orientation;
- Participation at other First Nation events (Moosehide Gathering, May Gathering);
- Traditional pursuits funding to assist people to get out on the land;
- Old Village Day, Aboriginal Day, Self-Government Day; and
- Elders in the school and daycare.

Recent initiatives include:

- Renewed linkages with Fort Good Hope (NWT) families;
- Hide tanning workshop;
- Knife making workshop; and
- Wind River canoe trip.

20.2.10.6 Historic and Paleontological Resources

An archaeological and historic assessment was conducted in 1995 for the then-proposed Dublin Gulch Mine site (Greer 1995). The study included a field assessment on a large project area that encompassed the proposed mine location. During the studies, no archaeological or historic period sites were identified; all areas favourable for pre-contact human occupation were deemed to have been destroyed by the extensive placer mining activity in the area, and all structures identified within the project area were all determined to be related to mining activities over the past 50 years.

Field surveys found that most of the valley fill at Dublin Gulch and Haggart Creek has been reworked by placer mining. There is no sign of any remaining source layer for the Dublin Gulch Pleistocene fossil locality, and no additional fossil vertebrate material was found.

Organic layers at the top of the surficial sequence in Dublin Gulch contain plant and arthropod material and yielded conventional (calibrated) radiocarbon ages of approximately 10,000 to 13,000 years before present. These late Pleistocene to early Holocene dates indicate the sediments were deposited during climatic warming following the McConnell Glaciation. A large piece of wood recovered from intact surficial deposits along the access road yielded a conventional (calibrated) radiocarbon age of approximately 2,700 years before present, which is late Holocene.

Remnant intact surficial deposits that have not been disturbed by placer gold mining occur along the south side of Dublin, along Ann Gulch, and at Secret Creek (along access road).

21 Capital Cost Estimate

21.1 Introduction

The estimated $370M initial capital costs for the development of the mine, crushing and processing facilities, and all associated costs have been prepared to a level of accuracy considered to be within -5 to +15%.

The FS team provided design quantities for bulk materials, including earthworks, concrete and structural steel, capital equipment costs, and some costs for bulk items such as mining and power.

Where possible, costs were derived to establish labour and material costs, productivity costs and logistics.

The estimate considers comparisons to recent mining activities in the North, including productivity, sources for labour and associated travel, weather restrictions and seasonal fluctuations in the construction environment.

21.1.1 Summary

The CAPEX consists of four main components:

- Direct costs;
- Indirect costs;
- Owner's costs; and
- Contingency.

Contributors to the capital and sustaining cost estimate included:

- JDS – mining, process equipment, Owner's, camp, G&A;
- DOWL – heap leach pad;
- Brazier – power transmission line and main transformer;
- Allnorth - roads;
- Victoria Gold - closure; and
- Merit – overall CAPEX lead.

The pre-production capital cost summary and distribution is shown in Table 21.1 and is based on Q3 2016 costs. The CAPEX is subject to some qualifications, assumptions, and exclusions, all of which are detailed herein.

Table 21.1 Summary of Pre-Production Capital Cost Estimate

Area	Cost (M$)
Direct Costs	
Mining and Pre-production Development	34.5
Site General	17.6
Process and Material Handling	101.3
Ancillaries	22.2
Power Supply and Distribution	15.1
Water Management	5.7
Heap Leaching	56.3
Total Direct Costs	**252.8**
Indirect Costs	
Owner’s costs	8.6
Indirects	72.9
Total Indirect Costs	**81.5**
Subtotal	334.4
Contingency (10.5%)	35.2
Total Project	**369.6**

Source: Merit (2016)

21.1.1.1 Basis of Estimate

The direct costs are based on the following information:

- Process flow diagrams, site layout and general arrangement drawings, equipment list, etc.;
- Budget submissions for the design and supply of new major and secondary equipment were provided by vendors in accordance with specifications and/or datasheets developed by the engineering groups involved;
- Prices for permanent materials were based on supplier quotations and current in-house data;
- Quantity take-offs for materials were provided by Allnorth, DOWL and JDS;
- Labour rates were provided by local and regional construction contractors. The rates used in this study have been derived from a strategy combining the research of labour pool locations and availability, input from unaffiliated local contractors, input from experienced union contractors affiliated with the Construction Labour Relations Association (CLRA) and alternative union contractors affiliated with the Christian Labour Alliance of Canada (CLAC);
- Productivities for installing equipment and materials were provided by local and regional construction contractors who are familiar with the project location and local conditions. Productivities that can be expected for this location were discussed and compared with actual results from other projects, as well as with in-house data;
- Supply and installation prices were provided by experienced vendors of pre-engineered and modular buildings;
- A freight allowance equaling 7.5% of the total material cost, and 10% of the total mechanical equipment cost, were included for all procured items;
- Topographic data and HLP geotechnical information/recommendations were provided by DOWL;
- An Engineering, Procurement and Construction Management (EPCM) project execution strategy and project schedule for the derivation of time-based indirect costs and sizing of site person-loading dependent infrastructure;
- Camp accommodation and catering prices were based on quotations and actual costs from similar recent projects

No allowances for currency fluctuations were included in the estimate. Table 21.2 lists the exchange rates used for this study.

Table 21.2: Exchange Rates used for the Eagle Gold Project FS

Exchange	Value
US$:C$	0.76
ZAR : C$ (for ADR plant)	10.2

Source: Merit (2016)

21.1.1.2 Direct Costs

Direct costs include all equipment, materials and installation associated with:

- Crushing and material handling;
- The ADR process;
- Infrastructure roads and site preparation;
- Power supply and distribution;
- Pre-production mining;
- The leach pad conveying system, and the solution recovery and distribution;
- Truck shelter;
- Yard services and other utilities;
- Control and communications systems;
- Plant mobile equipment;
- Fuel storage; and
- Explosives storage.

Existing infrastructure will be used for the warehouse, 200-man camp and admin facilities.

21.1.1.3 Indirect Costs

Indirect costs include the following:

- Temporary construction facilities including worker's camp, lay down area, warehousing, etc.;
- Freight;
- Vendor representatives;
- First fills and capital spares;
- EPCM services;
- Quality assurance;
- Surveying;
- Owner's costs; and
- Start-up and commissioning allowance.

21.1.1.4 Qualifications and Assumptions

The following assumptions were made in preparing the estimate:

- Budget quotes from vendors for equipment and materials are valid to within ±5% of the purchase price;
- Concrete aggregate and suitable backfill material will be available locally, and suitable sources were identified by the Owner's geotechnical consultants;
- Soil conditions will be adequate for foundation bearing pressures;
- Development of the project will be executed as per the schedule described in Section 25.1;
- Labour productivities are valid having been adjusted for northern locations and having been established with input from experienced contractors and Merits' in-house database for current projects; and
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.

21.1.1.4.1 Taxes

No taxes are included.

21.1.1.4.2 Project Currency, Estimate Base Date and Foreign Exchange

All project capital costs are expressed in Canadian dollars with the following provisions:

- Costs are expressed in Q3 2016 dollars with no provision for escalation beyond this date;
- Costs submitted in other currencies have been converted to Canadian dollars. Foreign currency exchange rates applied to the CAPEX relative to the Canadian dollar are presented in Table 21.2;
- No provision has been made for variations in the currency exchange; and
- No provision has been made for any taxes or fees applicable to currency exchanges.

21.1.1.4.3 Accuracy

The CAPEX, including contingency, for the mine, process plant, HLP and infrastructure has been prepared to a level of -5% to +15%.

21.1.1.4.4 Project Execution

The CAPEX is based on the assumption that Victoria Gold will follow the project execution plan described in Section 25.1. Any deviation from this plan may have an impact on both project schedule and costs.

21.1.1.5 Exclusions

The CAPEX does not include allowances for:

- Escalation during construction;
- Scope changes;
- Interest during construction;
- Schedule delays and associated costs such as those caused by:
    - Scope changes;
    - Unidentified ground conditions;
    - Extraordinary climatic events;
    - Labour disputes;
    - Permit applications;
    - Receipt of information beyond the control of EPCM contractors; and
    - Schedule recovery or acceleration.
- Cost of financing;
- Sunk costs;
- Research and exploration drilling;
- Sustaining capital;
- Permitting costs; and
- Working capital (included within economic model, refer to Section 23).

21.1.2 Project Direct Costs

21.1.2.1 Quantities and Unit Pricing

Engineering material take-offs have been based on quantities derived by the engineering groups from project drawings, sketches, similar projects and from the previous 2012 Wardrop FS.

21.1.2.1.1 Earthworks

Earthwork quantities were derived from the general arrangement and layout drawings. Quantities have been based on topographic drawings at 1 m or better contour intervals.

Site preparation and site roads earthwork quantities were provided by JDS and Allnorth. Heap leach earthworks quantities were provided by DOWL.

Earthworks costs were mainly based on 2012 FS estimates escalated annually at a rate of 2% for a total of 8.24% applied to the original unit rate.

The rates include the rental of earthmoving equipment, operators, fuel, mobilization/demobilization and construction indirect costs.

It has been assumed that concrete aggregates, structural backfill, granular base, road base, and sub-base will be supplied from local borrow pits and from pre-stripping material within the boundary of the Eagle pit. The unit costs associated with these materials include borrow pit development (crushing and screening) and transport costs.

21.1.2.1.2 Concrete, Formwork and Reinforcing Steel

Concrete quantities were determined from feasibility stage drawings and experience from previous projects. The unit rates for concrete placement and finishing were provided by regional industrial contractors. Aggregate will be sourced locally for use with an on-site batch plant to be located near the primary crusher location.

Formwork was estimated for each type of concrete classification and includes local and regional material supply, form oil, accessories, shoring and stripping. The price of formwork is included in the concrete unit rate.

Reinforcing steel was calculated based on the estimated weight per cubic metre of concrete, for each type of classification. The unit price includes for the supply of material, cutting, accessories and installation. The price of reinforcing steel is included in the concrete unit rate.

21.1.2.1.3 Structural Steel

Structural steel quantities were determined from feasibility stage drawings and experience from similar projects. The unit rates have been provided by regional industrial contractors. The weights shown include allowances for connections and base plates. The steel unit costs include:

- Material supply, fabrication and surface treatment, where required;
- Erection at site, based on estimated installation man-hours and unit labour costs and including final touch-up of surface coating; and
- Connection steel, weldments, and bolts.

ADR plant internal steel materials are included in the ADR plant vendor package. Installation man-hours and productivity rates were calculated using contractor's rates.

21.1.2.1.4 Architectural

Architectural quantities were derived from the general arrangement and layout drawings.

Costs for pre-engineered secondary and tertiary crushing buildings were based on vendor budgetary quotations for design, supply and erection, based on the 2012 FS and escalated accordingly.

The cost for the design, supply and erection of the pre-engineered ADR plant building was provided by Allnorth.

The cost for the design, supply and erection of the modular mine dry complex building was provided by Allnorth. Provision for the modification of the existing structures is included to construct the site administration offices.

Permanent truck shop building costs were transferred to sustaining capital. An allowance for a temporary truck shelter, built with Owner's supply containers and a roof structure complete with electrical installations was included in the CAPEX. The roof structure and electrical allowance was provided by Allnorth.

Estimates for the architectural finishes were based on the floor area of the finished space. Pricing was sourced from Merit's in-house data.

21.1.2.1.5 Mechanical Equipment

Budget quotations were obtained for all major mechanical equipment based on the mechanical equipment list and preliminary specifications. Installation hours were estimated based on pricing from similar North American projects by regional industrial contractors. Vendor representatives are planned to be engaged to oversee the installation of the larger equipment. Minor mechanical equipment was estimated based on pricing obtained from similar recent projects.

21.1.2.1.6 Mechanical (Plate Work and Tanks)

Plate work weights were calculated with allowances made for any necessary stiffeners, weirs and launders, etc. The unit prices included locally available plate purchase, detailing, fabrication and installation, based on contractor's pricing.

ADR plant plate work materials are included in the ADR plant vendor package. Installation man-hours and productivity rates were calculated using contractor's rates.

21.1.2.1.7 Piping

Quantities for piping within the process plant, with the exception of the ADR plant, were derived from general arrangement drawings and the process flow diagrams. Supply pricing and installation hours were based on contractor's pricing and include the following:

- Supply of material;
- Piping installation productivity and material units to include for:
    - Scaffolding;
    - Pipe supports & hangers;
    - Pipe labelling;
    - Pipe handling;
    - Pipe tagging;
    - Pipe cropping; and
    - Pipe testing.

ADR plant piping materials are included in the ADR plant vendor package.

In-heap leach pipeline quantities were provided by DOWL. Pregnant and barren solution piping quantities from the pad to the ADR plant were derived by JDS. Unit prices included the supply, shop and field fabrication, and installation and testing, with allowances for pipe fittings, pipe excavation and backfill.

21.1.2.1.8 Electrical

On-site Electrical

Quantities for all electrical materials and equipment are based on one-line diagrams and connected loads, as detailed in the flow sheets. Database developed installation productivities were applied for in-plant electrical systems, which included material supply (excluding freight) and installation.

Major electrical equipment and electrical material prices were based on budget quotations from the 2012 Wardrop FS and were escalated accordingly.

Transmission cable quantities for overhead power lines were estimated from the overall site plan and database unit pricing was applied to the estimated quantities.

Power Transmission to Site

The 43.5 km long transmission line from the McQuesten tap point is planned based on 69 kV single circuit wood pole construction utilizing line post insulators and 266.8 kcmil ACSR conductor. The line generally follows the Eagle Gold project access road with average spans of 90 to 100 m. A preliminary set of design drawings were prepared to facilitate a material and labour detailed estimate. This estimate covers all construction costs including line clearing, but does not cover land tenure or environmental monitoring costs during construction. Much of the line construction would best be carried out during the winter months. The construction force would be housed at the Eagle Gold project construction camp. Engineering and construction management costs are budgeted elsewhere.

The transmission line estimate is based on vendor material quotations and actual construction labour and construction equipment cost from several recent similar projects and as per published industry rates. The general overall estimated costs have been reviewed with a transmission line contractor with Yukon experience.

Main Substation

The mine site 69 kV to 13.8 kV substation will be a conventional outdoor air insulated installation consisting of 69 kV circuit switchers for high voltage switching and protection with two outdoor 3 phase oil filled transformers rated 10/13.5/15 MVA, ONAN/ONAF1/ONAF2 with automatic on-line tap changers. Switched capacitors will be included for power factor correction and to limit voltage drop during motor starting. This design is in line with the requirements of the previous YEC System Impact Study.

The substation cost includes pre-cast foundations for all equipment, a termination structure for the incoming 69 kV line, station class surge arresters, overhead lightning protection conductors, 13.8 kV system grounding resistors, etc. A station ground grid and station fence are also included in the estimate. Detailed equipment proposals were received. Engineering and construction management costs are budgeted elsewhere.

The construction estimate is based on (a) vendor equipment quotations that in turn were based on detailed equipment specifications issued for budget quotations, (b) standard electrical estimating hours (RS Means) as adjusted for the remote location and hours of work and (c) database and experience from similar projects.

Standby Power

Standby diesel generation is planned as two large modular units with a total capacity of 5 MW continuous at 0.8 PF, 13.8 kV. Provision is included to easily add additional generator units. The modular design includes an E-house with a 13.8 metal clad switchgear for generator control, plant site power distribution and includes the 69 to 13.8 kV substation secondary transformer main circuit breakers. In addition, the substation protection and control panel and 120 volt battery bank will be housed in this room. The E-house thus will form part of both the power plant and the main substation resulting in cost savings. The diesel plant will be PLC controlled and designed for automatic unattended operation with power import / export controls for paralleling with the YEC grid. Detailed proposals were obtained for this equipment. Engineering and construction management costs are budgeted elsewhere.

The modules are designed to mount on rig mats that in turn rest on compacted backfill pads, all included in the estimates. The modular "plug and play" design will greatly limit the required site installation time and labour.

The estimated modular power station cost is based on detailed vendor proposals received in response to a detailed equipment specification as issued for budget tender. These costs were confirmed by comparison to a number of other actual equipment purchases on file.

21.1.2.1.9 Instrumentation

Quantities for all instrumentation equipment and material, with the exception of the ADR plant, were provided by the engineering team.

Installation rates and productivities were sourced from Merit's in-house data and other similar projects.

Instrumentation equipment and material prices were based on budget quotations from the 2012 Wardrop FS and were escalated accordingly.

ADR plant instrumentation materials were included in the ADR plant vendor package.

21.1.2.2 Direct Field Labour

Labour rates (Table 21.3) were provided by local and regional construction contractors. The following hourly rates, by discipline, were developed and used throughout the estimate based on a 3-week in and 1-week out schedule for workers on a 70 hour/week basis.

Table 21.3: Labour Rates

Crew	Rate ($/hr)
Concrete Composite Crew	106.30
Structural Composite Crew	101.50
Mechanical Composite Crew	98.04
Piping Composite Crew	98.04
Electrical/Instrumentation Composite Crew	106.58
Material Handling Composite Crew	95.14
Architectural Composite Crew	107.29

Source: Merit (2016)

The blended average labour rate was $104.20.

The labour rates include:

- A base labour wage rate;
- Overtime premiums;
- Benefits and burdens;
- Workers compensation premiums;
- Travel allowances;
- Transportation to and from accommodations;
- Appropriate crew mixes;
- Small tools and consumables allowance;
- Field office overheads;
- Home office overheads; and
- Contractors' profit.

21.1.2.3 Direct Field Materials

Bulk material components constituted domestically available and imported quantities priced as free on board (FOB) manufacturer. Freight costs to transport materials to site are included in the indirect costs. Pricing was based on quantities derived by the engineering groups associated with the study.

21.1.2.4 Off-site Infrastructure

Off-site infrastructure includes the main Haggart Creek access road improvements as well as the extension to the utility transmission line.

21.1.3 Project Indirect Costs

21.1.3.1 Temporary Construction Facilities and Services

All indirect costs for contractors were included in the direct costs in the form of "all-in" unit rates for labour, construction equipment, productivities, and material costs, including but not limited to:

- Contractors' mobilization and demobilization;
- Miscellaneous construction equipment;
- Construction field offices, furnishings, equipment;
- Contractor travel and accommodations;
- Temporary power supply;
- Temporary water supply;
- Temporary heating and hoarding;
- Warehouse and lay down costs;
- Temporary toilets;
- Temporary communications;
- Ongoing and final clean-up;
- Yard maintenance;
- Janitorial services; and
- Site safety personnel and training.

Indirect costs included in the CAPEX provided for the following:

- Construction management;
- Construction camp;
- Construction accommodations and catering;
- Engineering and procurement;
- Start-up and commissioning;
- First fills and warehouse inventory; and
- Freight.

21.1.3.2 Construction Camp and Catering

Accommodation for 210 persons will be available immediately when construction starts. The costs for these facilities are considered sunk costs and include the existing 100-man camp and a recently purchased used 110-man camp. Each camp has its' own kitchen. The combined camp will be expanded to accommodate 500 people, including the pre-production mining personnel. No additional kitchen is needed and the cost for the expansion is included at a rate of $10,000 per room.

An average catering cost of $59.50 per camp man-day is based on prices by experienced national catering contractors, who provided a scale of man-day costs based on various levels of camp occupancy, received in the 2012 Wardrop FS and escalated accordingly

21.1.3.3 First Fill and Spare Parts

An allowance of 2% of the costs of the mechanical equipment was included for first fills.

An allowance of 5% of the costs of the mechanical equipment, less the ADR plant and the main substation were included for spare parts. The ADR plant and main substation were included separately based on vendor quotations.

Mining first fills and spare parts costs are included within the direct costs related to the purchase of the mobile equipment fleet.

21.1.3.4 Start-up and Commissioning

The requirements for vendor representatives, to supervise the installation of equipment, or to conduct a checkout of the equipment, prior to start-up of the equipment, as deemed necessary for equipment performance warranties, was calculated and included in the estimate.

An allowance was made for the retention of vendor representatives for start-up, as well as for a selection of eight people from the contractor's crews, for a period of approximately eight weeks. Engineering support was included in the Owner's costs.

An allowance of 1.5% of the mechanical equipment costs was included for commissioning spares.

21.1.3.5 Freight

A freight allowance equaling 7.5% of the total material cost, and 10% of the total mechanical equipment cost was included for all procured items. Materials and equipment quoted delivered to site within the direct costs were excluded from the factored freight calculation.

21.1.3.6 Engineering, Procurement and Construction Management

Engineering and procurement costs are based on the estimated number of drawings and specifications required.

Construction and project management costs are derived from a staffing plan against the project schedule and the application of prevailing EPCM consultant rates.

21.1.3.7 Duties

No customs duties were included.

21.1.4 Owner's Costs

Owner's costs include:

- Victoria Gold project management staff;
- Pre-production general & administration staff;
- Pre-production processing staff;
- Personnel transportation for owners staffing including the mining group;
- Camp costs related to owners personnel including the mining group;
- Owners health, safety, medical, and first aid costs;
- Owners surface support equipment operations;
- Main access road maintenance during construction;
- Environmental costs;
- Human resources costs;
- Construction insurance;
- Legal and regulatory costs; and
- Owners site office costs.

21.1.5 Contingency

Contingency is a provision for project costs that will likely occur, but cannot be accurately defined or estimated. Including project contingency in a capital cost estimate is necessary to determine the most likely project cost.

The contingency amount of 10.5% of the total direct and indirect costs covers unforeseeable costs within the scope of the estimate as shown in Table 21.4.

Contingency has been estimated by discipline, taking into account items that have been quoted, estimated or factored, the cost risks, and level of engineering definition for each area. Contingency is a subjective allowance based on the degree of confidence that study contributors feel should be applied to their work.

Table 21.4: Contingency

CAPEX Cost Centre	Allocated Contingency (%)
7 - Mining - Pre-Production Development	10
8 - Mining - Equipment	5
10 - Bulk Earthworks	25
12 - Roads	25
13 - Civil - Heap Leach (Earthworks, liners, piping)	15
20 - Concrete	15
30 - Structural Steel	10
40 - Architectural	10
50 - Mechanical Equipment Supply Only	5
50 - Mechanical	10
55 - Platework	10
57 - Building Services	15
58 - Plant Mobile Equipment	5
60 - Piping	15
70 - Electrical	15
75 - Power Supply - transmission line, substation and auxiliary power	15
80 - Instrumentation & Controls	10
91 - Construction Indirects	10
92 - Initial Fills	5
93 - Spares	5
94 - Freight & Logistics	10
95 - Commissioning and Start-up	10
96 - EPCM	5
97 - Vendors Assistance	5
98 - Owner's Costs	10

Source: Merit (2016)

Contingency is not intended to be used for scope changes or project exclusions that would otherwise be added or subtracted from the budget. Nor is it intended to cover such items as labour disputes, currency fluctuations, escalation, force majeure or other project uncontrollable risk factors. It should be assumed that the contingency amount will be spent over the engineering and construction period.

No escalation was included in the CAPEX.

21.1.6 Sustaining and Closure

The sustaining capital costs including mining equipment and closure costs for the project are estimated at $218M. The same basis of estimate described above, and used to determine the CAPEX, was used to determine the sustaining capital costs for which a summary is provided in Table 21.5. Contingency and escalation throughout the period of sustaining development was not included in the sustaining capital cost estimate.

Table 21.5: Sustaining and Closure Capital Estimate

Area	Cost (M$)
Mining and Pre-production Development	45.6
Site General	9.9
Process and Material Handling	-
Ancillaries	30.3
Power Supply and Distribution	0.8
Water Management	15.0
Heap Leach Facilities	81.6
Owner's Costs	-
Indirects	-*
Closure (Net of Salvage)	35.0
Subtotal	218.1
Contingency	-
Total Project	**218.1**

* Sustaining capital indirect costs are included in direct costs for each area.

Source: Merit (2016)

22 Operating Cost Estimate

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies. The estimate was developed using first principles and applying direct applicable project experience, thus avoiding the use of general industry factors. The operating cost is based on the Owner owning and operating the mining and services fleet. Minimal use of permanent contractors is assumed except where value is provided through expertise and/or the provision of seasonal services. Most estimates were derived from engineers, contractors, and suppliers who have provided similar services to existing operations (particularly in northern Canada) and have demonstrated success in executing the plans set forth in this study.

The target accuracy of the operating cost is -10/+15%, which represents a FS Budget Class 3 Estimate.

The operating cost estimate is broken into three major sections:

- Open Pit Mining;
- Crushing, Heap Leach and Processing; and
- General & Administrative.

22.1 Operating Cost Summary

Operating costs are reported only for the operating life of the mine and exclude those costs incurred during the pre-production phase. Mine production costs up to the end of Q4 Year -1 are capitalized; subsequent periods commencing in Q1 Year 1 are reported as operating costs. Some of the costs incurred during the pre-production period relate to the purchase of items such as consumables required for the following year of production. The timing of these costs has been accounted for in the economic analysis.

Operating costs are expressed in Canadian dollars with a fixed exchange rate of US$:C$ = 0.78. No allowance for inflation has been applied.

The total operating unit cost is estimated to be $10.54/t leached. Average annual, total LOM and unit operating cost estimates are summarized in Table 22.1.

Figure 22.1 illustrates the operating cost distribution. Annual operating costs by area are outlined in Table 22.3.

Table 22.1: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t leached	LOM (M$)
Mining*	51.6	4.19	515
Processing	60.7	4.93	606
G&A	17.5	1.42	175
Total	**129.7**	**10.54**	**1,295**

Source: JDS (2016)
*Average LOM Mining cost amounts to $2.17/t mined at a 0.95:1 strip ratio (excluding pre-production tonnes mined).

The main OPEX component assumptions are outlined in Table 22.2.

Table 22.2: Main OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.11
Overall power consumption (all facilities)	kWh/t processed	7.45
Sodium Cyanide cost (delivered)	$/t	2,863
Diesel cost (delivered)	$/litre	0.76
LOM average manpower (including contractors, excluding corporate)	employees	351

Source: JDS (2016)

Table 22.3: Annual Operating Cost by Area

	Unit	LOM	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Annual Operating Cost													
Mining	M$	**515**	44	58	49	55	49	46	48	49	63	49	4
Processing	M$	**606**	51	55	53	55	55	55	66	61	65	69	21
G&A	M$	**175**	17	17	17	17	17	17	17	17	17	17	5
Total Operating Costs	M$	**1,295**	**112**	**130**	**119**	**126**	**121**	**118**	**131**	**127**	**146**	**136**	**31**
Total Ore Leached	Mt	**122.9**	9.9	12.6	12.5	11.3	12.3	12.7	13.2	12.8	13.5	11.7	0.5
Unit Operating Cost by Year													
Mining	$/t leached	**4.19**	4.43	4.62	3.94	4.83	3.95	3.65	3.67	3.83	4.7	4.19	9.09
Processing	$/t leached	**4.93**	5.15	4.35	4.27	4.82	4.44	4.33	5.02	4.79	4.82	5.91	43.62
G&A	$/t leached	**1.42**	1.7	1.34	1.35	1.5	1.37	1.33	1.28	1.32	1.27	1.44	11.14
Total Unit Operating Costs	$/t leached	**10.54**	**11.27**	**10.31**	**9.56**	**11.15**	**9.76**	**9.31**	**9.97**	**9.94**	**10.8**	**11.54**	**63.85**

*totals may not add due to rounding

Source: JDS (2016)

22.2 Operations Labour

This section provides an overview of total workforce and the methods used to compile the labour rates.

Table 22.4 summarizes the total planned workforce during project operations.

Table 22.4: Summary of Personnel

Department	Total Persons Employed (Peak)
Mining	169
Processing	105
General & Administration*	94
Total Personnel - All Areas	**368**

*Includes both Owner and Contractor personnel
Source: JDS (2016)

Labour base rates were determined by reference to other active northern Canadian operations and benchmarked against Costmine (Canadian Mine Salaries, Wages, Benefits 2015 Survey Results). Labour burdens were assembled using first principles. The following items are included in the burdened labour rates:

- Scheduled overtime costs based on individual employee rotation;
- Unscheduled overtime allowance of 10% for hourly employees;
- CPP, EI, WCB as legislated;
- Statutory holiday allowance of 6% of scheduled hours;
- Vacation pay allowance of 6% of scheduled hours;
- Pension allowance of 5% of scheduled hours; and
- Flexible benefits package of $2,500 annually per employee.

22.3 Mine Operating Cost Estimate

22.4 Open Pit Mine Operating Costs

Open pit mining activities were assumed to be undertaken by the Owner. Costs are presented in 2016 Canadian dollars and do not include allowances for escalation or exchange rate fluctuations.

The mining unit rate was calculated from first principles based on equipment required for the mining configuration of the operation as described in this report, as well as a comparison to similar sized open pit mining operations in the region. Local labour along with quotes from equipment suppliers and explosives suppliers were taken into consideration in determining the mining cost. The open pit mining costs include pit operations, road maintenance, mine supervision, and technical services.

The average open pit operating costs for the LOM plan are presented in Table 22.5 and Table 22.6, both by mining activity and category. These costs are based on the LOM schedule presented in this report and account for the material tonnages mined and their associated costs (exclusive of the pre-production period).

Table 22.5: Open Pit Operating Cost Estimate – by Activity

Activity	$/t mined*
Drill and Blast	0.68
Load and Haul	1.12
Mine General	0.12
Mine Maintenance	0.18
Technical Services	0.08
Total Open Pit Operating Cost	**2.17**

*Excludes pre-production period tonnes
Source: JDS (2016)

Table 22.6: Open Pit Operating Cost Estimate – by Category

Category	$/t mined*
Labour	0.65
Power and Fuel	0.45
Parts & Repair	0.37
Lubrication	0.03
Wear Items	0.17
Tires	0.06
Explosives	0.38
Services	0.07
Total Open Pit Operating Cost	**2.17**

*Note: Excludes pre-production period
Source: JDS (2016)

22.4.1 Basis of Estimate

22.4.1.1 Open Pit Mobile Equipment

A summary of open pit equipment requirements can be found in Section 16, Mining Methods. Operating costs for each piece of equipment were calculated taking into account operating hours per year, fuel consumption, lube, overhaul, and maintenance costs. Parts, consumables, and miscellaneous operating costs were based on the mining fleet requirements including detailed haul profiles calculations, major equipment requirements and the LOM material schedule.

22.4.1.2 Open Pit Labour

Mining labour costs were calculated using the personnel numbers summarized in Section 16, Mining Methods. Positions were broken into three major groups: technical services, mine operations, and maintenance. Technical services includes engineering and geology positions which support mine activities, mine operations refers to equipment operators and supervisory roles, and maintenance positions deal exclusively with mine equipment. The number of maintenance personnel required was based on the number of units operating during each time period. The labour requirements are further divided into salaried and hourly personnel.

Local labour rates are based on information gathered regarding salaries of various skill levels. Quotes from equipment and explosives suppliers were also taken into consideration as well as mining cost service information and factors based on experience were taken into consideration. Each estimate incorporates fully burdened labour rates and was benchmarked against similar operations. Table 22.7 summarizes the open pit mining workforce labour rates.

Table 22.7: Open Pit Labour Rates

Position	Shift Rotation	Salaried/Hourly	Loaded Salary per Year/ Hourly Rate ($)
Supervision			
Mining/Maintenance Superintendent	2&2	Salaried	144,100
Mine/Maintenance Shift Foreman	2&2	Salaried	104,100
Operations			
Driller, Blast Hole	2&2	Hourly	44
Blaster	2&2	Hourly	44
Blasting Helper	2&2	Hourly	32
Shovel/Loader Operator	2&2	Hourly	49
Truck Driver	2&2	Hourly	42
Track Dozer Operator	2&2	Hourly	44
R.T. Dozer Operator	2&2	Hourly	44
Grader Operator	2&2	Hourly	44
Water/Ancillary Truck Diver	2&2	Hourly	42
Labourer/Trainee	2&2	Hourly	32
Maintenance			
Heavy Equipment Mechanic	2&2	Hourly	55
Welder/Mechanic	2&2	Hourly	55
Electrician/Instrument	2&2	Hourly	55
Lube/PM Mechanic/Light Duty Mechanic	2&2	Hourly	55
Tireman	2&2	Hourly	44
Labourer/Trainee	2&2	Hourly	32
Technical Services			

Position	Shift Rotation	Salaried/Hourly	Loaded Salary per Year/ Hourly Rate ($)
Maintenance Planner	2&2	Salaried	99,100
Chief Mining Engineer	2&2	Salaried	174,100
Senior Mine Engineer	2&2	Salaried	139,100
Mine/Ore Control Engineer	2&2	Salaried	99,100
Mine Technician/Surveyor	2&2	Salaried	83,800
Surveyor Assistant	2&2	Salaried	73,500
Clerk	2&2	Salaried	73,500
Chief Geologist	2&2	Salaried	174,100
Mine Geologist	2&2	Salaried	99,100

Source: JDS (2016)

22.4.1.3 Mine Consumable Requirements

Consumable cost estimates were assembled from equipment suppliers, cost services, factors and JDS experience.

The diesel fuel price of $0.76/L includes delivery to the site and storage.

Estimates of costs for ground engagement tools, parts and equipment spares were based on inputs from equipment suppliers. Explosive costs were supplied by a local explosives supplier.

The major consumable cost drivers are diesel fuel, tires, maintenance parts and repairs. A breakdown of the consumables costs is provided in Table 22.8 with unit costs for tires, lubes and explosives.

Table 22.8: Open Pit Consumable Cost Detail

Item	Unit	Average Cost
Diesel fuel	$/litre	0.76
Lube cost	$/litre	6.00
Tires - haul trucks	$/ea	22,300
Blasting Supplies		
AN cost	$/kg	1.10
Emulsion cost	$/kg	1.60
Handidets_12m	$/ea	10.05
Handidets_15m	$ ea	12.21
Trunk_Line_4m	$/ea	5.03
Trunk_Line_6m	$/ea	6.03
Boosters	$/ea	6.86

Source: JDS (2016)

22.4.2 Drill and Blast Operating Cost

The average LOM (excluding pre-production period) drilling and blasting operating cost is $0.68 t/mined for a total of $161M.

Drilling and blasting costs include:

- Labour;
- Diesel fuel;
- Oils and lubricants;
- Repair and maintenance parts;
- Wear items (drill bits, undercarriage, structures, drill consumables);
- Explosives and accessories; and
- Contract services.

Average annual ANFO consumption is 4,000 t, while emulsion is estimated at 2,900 t. The overall powder factor is estimated at 0.29 kg/t. Table 22.9 summarizes LOM drill and blasting costs for the project.

Table 22.9: Drill and Blast Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost (M$)*
Labour	0.06	15
Fuel (drill and blasting requirements)	0.05	11
Maintenance & Operating Consumables		
Parts & Repair	0.05	11
Lubrication	0.00	1
Wear Items (bits, undercarriage, etc.)	0.07	18
Explosives and Accessories	0.40	95
Services	0.04	10
Total	**0.68**	**161**

*Note: Excludes pre-production period
Source: JDS (2016)

22.4.3 Load and Haul Operating Cost

The average LOM loading and hauling operating cost is $1.12 t/mined for a total of $266M and represents approximately 50% of the total mine operating cost. The load and haul cost includes delivery of the ore to the crusher and the ROM HLP, as well as re-handling costs associated with the stockpiles.

Loading and hauling costs include:

- Labour;
- Diesel fuel;
- Tires and rims;
- Oils and lubricants;
- Repair and maintenance parts;
- Major rebuilds for initial used haul trucks;
- Wear items (buckets, teeth, undercarriage structures); and
- Leases and Rentals (haulage fleet will require the addition of a few rental units sporadically through the LOM for short durations).

The estimated cost of loading and hauling over the LOM is shown in Table 22.10.

Table 22.10: Load and Haul Cost

Category	Average LOM Cost ($/t mined)*	Total LOM Cost (M$)*
Labour	0.28	65
Fuel	0.37	88
Maintenance &Operating Consumables		
Parts & Repair	0.29	70
Lubrication	0.02	5
Wear Items (buckets, teeth, undercarriage, etc.)	0.10	23
Tires	0.05	13
Contract Services (Haulage truck rentals)	0.01	2
Total	**1.12**	**266**

*Note: Excludes pre-production period
Source: JDS (2016)

22.4.4 Mine General Operating Cost

The average LOM mine general operating cost is $0.12 t/mined for a total of $28M. This encompasses labour costs of senior mine operations personnel and front line supervisors, as well as operating costs of small excavators and various mine service vehicles.

The mine general costs include:

- Labour;
- Tires and rims;
- Oils and lubricants;
- Repair and maintenance parts; and
- Wear items (GET, undercarriage, structures).

The estimated cost of mine general over the LOM is shown in Table 22.11.

Table 22.11: Mine General Cost

Category	Average LOM Cost ($/t mined)*	Total LOM Cost (M$)*
Labour	0.06	14
Support Equipment	0.05	12
Tools, Supplies	0.01	2
Total	0.12	28

*Note: Excludes pre-production period
Source: JDS (2016)

22.4.5 Mine Maintenance Operating Cost

The average LOM mine maintenance cost is $0.18 t/mined for a total of $42M. This encompasses labour costs (including supervision) required for the maintenance of all open pit mobile equipment fleets. All maintenance on-site will be carried out with Victoria Gold personnel using the company's own installations.

The LOM mine maintenance cost is summarized in Table 22.12.

Table 22.12: Mine Maintenance Cost

Category	Average LOM Cost ($/t mined)*	Total LOM Cost (M$)*
Labour	0.17	40
Tools, Supplies	0.01	2
Total	**0.18**	**42**

*Note: Excludes pre-production period

Source: JDS (2016)

22.4.6 Technical Services Operating Cost

The average LOM technical services cost is $0.08 t/mined for a total of $18M. This encompasses labour costs required for the technical services group, including all mine engineering staff and the mine geology group.

Table 22.13 summarizes the LOM technical serves operating cost.

Table 22.13: Technical Services Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost (M$)*
Labour	0.08	18
Total	**0.08**	**18**

*Note: Excludes pre-production period

22.5 Processing Operating Cost Estimate

The processing operating cost estimate includes operating and maintenance costs for:

- Crushing;
- ADR process plant;
- HLP piping and drip emitter installation and maintenance;
- Barren and pregnant solution handling between the HLP and plant; and
- Water treatment plant.

ROM loading, hauling and spreading is included in the mining operating costs.

Mobile equipment costs are included in infrastructure costs described in Section 18.

A summary of the process plant operating cost is presented in Table 22.14.

Table 22.14: LOM Processing Operating Cost Estimates by Activity

Category	$/t processed
Labour	0.84
Power & Fuel	0.81
Maintenance & Operating Consumables	3.22
Services	0.06
Total	**4.93**

Source: JDS (2016)

22.5.1 Process Labour

The proposed process plant labour structure and costs for both salaried and hourly personnel are based on an annual basis as shown in Table 22.15. The number of personnel required for the process plant was developed from similar projects and operating mines. Total labour costs are $0.83/t of ore processed and account for 17% of the overall process operating costs.

Table 22.15: Process Labour Complement and Rates

Position	Manpower Complement	Manpower On-Site	Salaried/Hourly	Fully Loaded Rate $/a
Process Plant				
Process Superintendent	1	1	Salaried	144,113
Operations Shift Foreman	4	2	Salaried	104,113
Plant Metallurgist	1	1	Salaried	109,113
Metallurgist Technician	2	1	Salaried	83,796
ADR Plant Operations				
Crusher Operator 275	8	4	Hourly	91,097
Crusher Operator 365	4	2	Hourly	97,148
Carbon Plant Operator	8	4	Hourly	106,224
EW/Gold Room Operator	2	1	Hourly	115,301
Helpers	8	4	Hourly	69,522
Helpers (Crushing Plant)	8	4	Hourly	69,522
HL Operations				
Leach Pad Foreman	2	1	Hourly	106,224
HL PAD Operators	12	6	Hourly	97,148
Laboratory				
Chief Assayer	1	1	Salary	94,113
Assayer	8	4	Hourly	91,097
Sample Prep/Trainee	8	4	Hourly	69,522
Water Treatment Plant				
Water Treatment / Potable / Incinerator Operator	4	2	Hourly	106,224
Process Maintenance				
Maintenance Foreman	2	1	Salary	104,113
Maintenance Planner	2	1	Salary	99,113
Millwrights/Welders	8	4	Hourly	121,352
Electricians/Instrumentation	4	2	Hourly	121,352
Apprentice	8	3	Hourly	91,097
Total	**105**	**54**		

Source: JDS (2016)

22.5.2 Power and Fuel

The power and fuel costs for the crushing and process plants account for approximately $0.81/t or 17% of the total process operating cost. Power for the process plant will be supplied at $0.11/kWh. A load list was developed from the installed power, motor efficiencies and operating time of each piece of equipment to estimate the annual power consumption and cost.

Diesel will be supplied and delivered at a cost of $0.76/L. Diesel costs are for the operation of the kiln and boiler in the plant as well as the boiler for heating barren solution.

22.5.3 Maintenance and Operating Consumables

Maintenance and consumables account for 65% of the process operating costs. The maintenance and consumable costs are summarized in Table 22.16.

Table 22.16: Maintenance and Operating Consumables

Consumables	$/t processed
HL Operations - Piping, Drip Emitters and Liners	0.16
Liners and Misc. Spares	0.69
Reagents	2.19
Maintenance	0.18
Total	**3.22**

Source: JDS (2016)

22.5.3.1 Heap Leach Operations

The cost to provide piping and drip emitters on each lift of the HLP is accounted for under heap leach operations. The piping and drip emitter costs were based on a cost per square metre of lift area ($2.45/m^2). The cost for HL operations is $0.16/t of ore processed.

22.5.3.2 Liners and Misc. Spares

Gyratory and cone crusher liner costs were based on a combination of vendor data and experience. Crusher liner costs were provided by the vendor. An additional $150,000 per year was allocated for miscellaneous spares. A total cost of $0.69/t processed was included for crusher liners and miscellaneous spares.

22.5.3.3 Reagents

The reagent consumption and cost summary is presented in Table 22.17. The quantity of reagents required for the operation is based on test work, vendor information, and empirical data. Reagent costs are considered a variable cost that changes with plant throughput. Chemical reagent costs delivered to site were obtained from vendors.

Table 22.17: Reagent Consumption Costs

Reagents	Annual Usage (t)	Cost per tonne including freight ($)	Processing Cost ($/t)
Lime - Hydrated	17,998	630	0.87
Cement	23,998	365	0.34
NaCN	3,619	2,869	0.87
HCL	321	792	0.02
NaOH	346	1,128	0.03
Carbon	88	3,292	0.02
Antiscalant	12	3,920	0.00
Hydrogen Peroxide	30	1,150	0.00
Total	**46,412**	**14,175**	**2.19**

Source: JDS (2016)

22.5.3.4 Maintenance

The spare parts and consumables cost for the process plant was estimated at 4% of the total purchased mechanical equipment cost. Mill maintenance labour will be responsible for equipment repair and part installations. The maintenance costs were estimated to be $0.18/t processed.

22.5.3.5 Services

The cost of operating the assay laboratory was provided by vendors. The cost per year was estimated to be $373k. Services also include water treatment and consumables. Together these items account for approximately $0.06/t.

22.6 General and Administration Operating Cost Estimate

The general and administrative cost is estimated to be $1.42 per tonne processed and can be attributed to four categories:

- Labour;
- On-site items;
- Personnel Transportation; and
- Off-site items.

Table 22.18 summarizes the annual G&A operating costs.

Table 22.18: Summary of G&A Costs

Cost Category	$/t Processed
Labour	0.41
Travel Costs	0.25
On-site Items	0.76
Off-site Items	0.00
Total G&A Costs	**1.42**

Source: JDS (2016)

23 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variation in metal price, foreign exchange rate, head grades, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits, to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Sections 21 and 22 of this report. They are presented in 2016 Canadian dollars (C$). The economic analysis has been run with no inflation (constant dollar basis).

23.1 Assumptions

All costs and economic results are reported in Canadian dollars (C$), unless otherwise noted. Gold pricing is reported in US dollars (US$). Table 23.1 outlines the planned LOM tonnage and grade estimates.

Table 23.1: LOM Plan Summary

Parameter	Unit	Value
Mine Life	Years	10
Total Ore	Mt	122.9
Strip Ratio	w:o	0.9
Processing Rate	ktpd	33
Average Au Head Grade	g/t	0.67
Au Payable	LOM koz	1,884
	Average koz/yr	189

Source: JDS (2016)

Other economic factors used in the economic analysis include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario);
- NPV calculated assuming a mid-year accounting period;
- Closure cost of $35M (net of salvage value);
- Nominal 2016 dollars;
- No inflation;
- No taxes or duties other than federal and territorial income tax rates of 15% each (discussed in Section 23.4) and territorial mining tax;
- Numbers are presented on a 100% ownership basis and do not include management fees or financing costs;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Costs from operations incurred in the pre-production period have been capitalized and
- Exclusion of all pre-development and sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.) However, pre-development and sunk costs are utilized for tax deductions.

23.2 Revenues & NSR Parameters

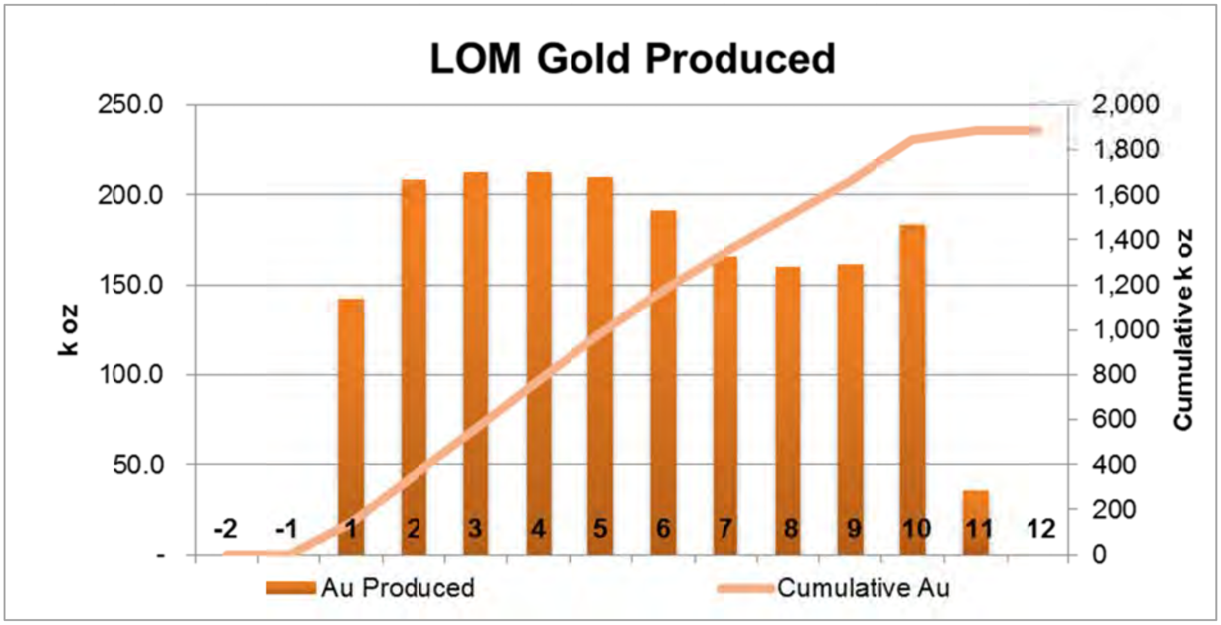
Mine revenue is derived from the sale of gold doré into the international marketplace. No contractual arrangements for refining exist at this time. However, the parameters used in the economic analysis were verified against other industry comparable projects. These details can be found in Section 19 (Market Studies) of this report. Gold production and sale is assumed to begin in Q2 of Year 1 and continue for 10 years. Table 23.2 outlines the market terms used in the economic analysis. Figure 23.1 illustrates the annual payable gold and cumulative payable gold by project year.

Table 23.2: NSR Assumptions used in the Economic Analysis

Assumptions	Unit	Value
Au Payable	%	99.5
Au Refining Charge	US$/oz	10

Source: JDS (2016)

Figure 23.1: Annual and Cumulative Payable Gold Production



Source: JDS (2016)

23.3 Timing of Revenues and Working Capital

Working capital has been accounted for in the economic analysis due to the timing difference between cash outflows and cash inflows with respect to the operating costs. The following describes how the working capital was scheduled in the economic analysis:

- Materials and consumables are purchased two months in advance on 30-day terms; and
- Labour costs are assumed to be incurred as they are paid.

A total of $27M has been considered as working capital. It was calculated by adding the following:

- Y1Q1 operating costs ($12M);
- Sustaining CAPEX in Y1Q1 ($3M);
- 1/3 of OPEX in Y1Q2 ($9M);
- Working Capital Spares ($3M); and
- Payback of working capital occurs in two installments: $24M in Y1 Q2, and $3M in Y11 (at the end of production).

23.4 Taxes

The project has been evaluated on an after-tax basis in order to reflect a more indicative, but still approximate, value of the project. Both Yukon Mineral Tax and Federal and Territorial Income Tax were applied to the project. A detailed tax analysis was completed by Wentworth Taylor in order to derive the after-tax valuation of the project. Specific assumptions and methodology in the analysis includes the following:

Yukon Mineral Royalties

- Yukon Mining Quartz Tax has been evaluated as part of the after-tax analysis. The Crown royalty applies to all ore, minerals, or mineral bearing substances mined in the Yukon on a calendar year basis;
- The royalty is calculated based on the value of the output mine which is the value of minerals produced exceeded by the various deductions allowable; and
- The royalty rate ranges from 0% to 12% based on the taxable revenue from saleable gold minus deductions.

Federal and Territorial Corporate Income Tax

- Federal tax rate of 15% and a Yukon 15% rate were used to calculate income taxes.

Mineral Property Tax Pools

- Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with appropriate opening balances to calculate income taxes.

Capital Cost Allowance (CCA)

- Specific capital cost class CCA rates were applied and used to calculate the appropriate CCA the company can claim during the entire life of the project.

Total LOM taxes for the project amount to $353M.

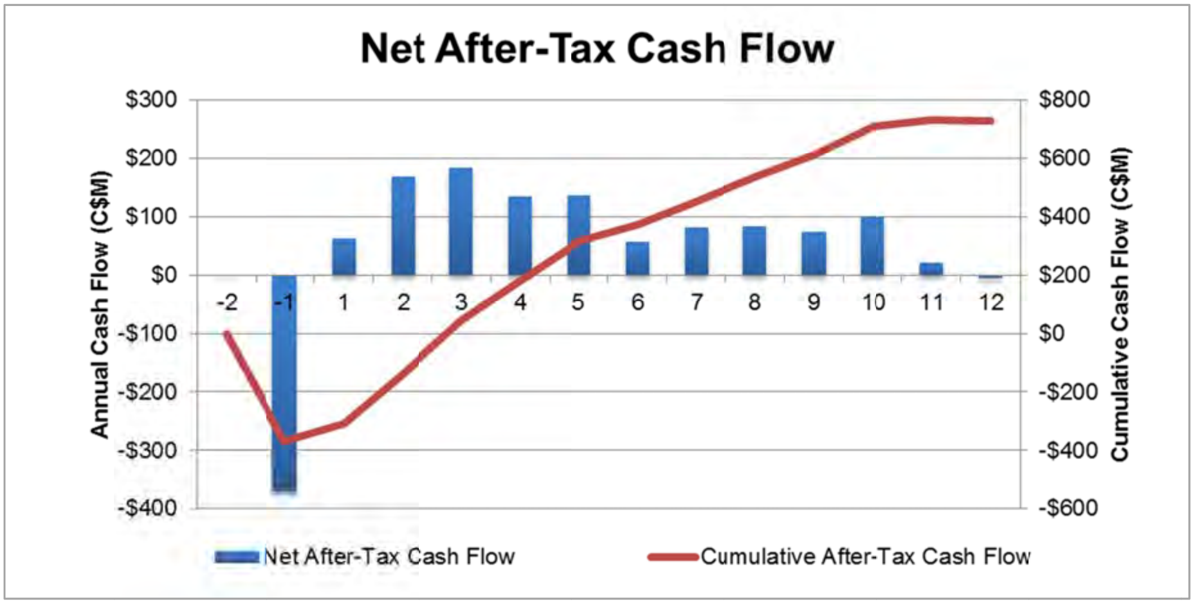
23.5 Third Party Royalties

Third party royalties have been considered in the economic analysis. A total of $30M of third party royalties are payable over the LOM. Details related to the third party royalties are outlined in Section 19 of the report.

23.6 Results

The project is economically viable with an after-tax internal rate of return (IRR) of 29.5% and a net present value at 5% ($NPV_{5\%}$) of $508M. Figure 23.2 shows the projected cash flows used in the economic analysis. Table 23.3 shows the detailed results of this evaluation.

Figure 23.2: Annual and Cumulative After-Tax Cash Flows



Source: JDS (2016)

Table 23.3: Summary of Eagle Project Economic Results

Category	Unit	Value
Net Revenues	M$	2,950
Operating Costs	M$	1,295
Cash Flow from Operations	M$	1,655
Capital Costs*	M$	588
Operating Cost	US$/oz	539
Cash Cost°	US$/oz	561
All-in Sustaining Cost †	US$/oz	638
Net Pre-Tax Cash Flow	M$	1,067
Pre-Tax NPV $_{5\%}$	M$	778
Pre-Tax IRR	%	37.1
Pre-Tax Payback	Years	2.6
Break-Even Pre-Tax Gold Price	US$/oz	844
Total Taxes	M$	353
Net After-Tax NPV$_{5\%}$	M$	508
After-Tax IRR	%	29.5
After-Tax Payback	Years	2.8
Break-Even After-Tax Gold Price	US$/oz	845

(*) Includes pre-production, sustaining, closure and reclamation capital costs

° Cash Cost formula: (Refining Costs + Third Party Royalties + Operating Costs) / Payable Au oz

† AISC formula: (Refining Costs + Third Party Royalties + Operating Costs + Sustaining Capital Costs (excluding closure)) / Payable Au oz

Source: JDS (2016)

23.7 Sensitivities

A sensitivity analysis was performed to test project value drivers on the project's NPV using a 5% discount rate. The results of this analysis are demonstrated in Table 23.4 and Table 23.5 and illustrated in Figure 23.3. The project proved to be most sensitive to changes in gold price followed by head grade, foreign exchange rate, and operating costs. The project showed least sensitivity to capital costs.

Table 23.4: Pre-Tax NPV $_{5\%}$ Sensitivity Results

Pre-Tax NPV$_{5\%}$ (M$)							
	-15%	**-10%**	**-5%**	**100%**	**5%**	**10%**	**15%**
Gold Price	425	543	660	778	896	1013	1131
F/X Rate	1190	1037	901	778	667	566	474
Head Grade	428	545	661	778	895	1011	1128
OPEX	930	879	829	778	727	677	626
CAPEX	859	832	805	778	751	724	697

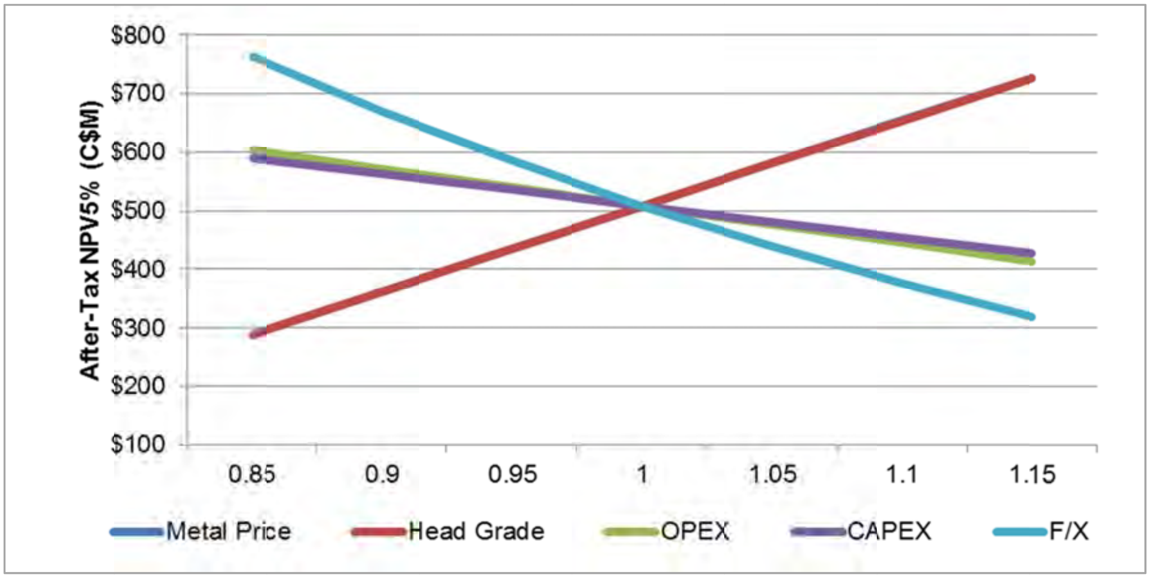
Source: JDS (2016)

Table 23.5: After-Tax NPV $_{5\%}$ Sensitivity Results

	After-Tax NPV$_{5\%}$ (M$)						
	-15%	**-10%**	**-5%**	**100%**	**5%**	**10%**	**15%**
Gold Price	287	3612	435	508	582	655	728
F/X Rate	764	670	585	508	439	376	318
Head Grade	289	363	436	508	581	654	726
OPEX	603	572	540	508	477	445	414
CAPEX	590	563	536	508	481	454	427

Source: JDS (2016)

Figure 23.3: After-Tax NPV $_{5\%}$ Sensitivity



Source: JDS (2016)

Pre-tax and after-tax results were evaluated using a wider range of sensitivities to gold price and exchange rate. The sensitivities were calculated between gold prices from $1,000 to $2,000/oz and exchange rates between 0.75 to 1.00 US$:C$. The results are presented in Tables 23.6 and 23.7.

Table 23.6: Gold Price Sensitivity

Au US $/oz	Pre Tax NPV$_{5\%}$ (M$)	Pre-Tax IRR (%)	Pre-Tax Payback	After Tax NPV$_{5\%}$ (M$)	After-Tax IRR (%)	After-Tax Payback
1,000	308	19.5	3.8	212	15.9	4.0
1,100	496	27.0	3.2	332	21.7	3.4
1,200	684	33.8	2.8	450	27.0	2.9
1,250	**778**	**37.1**	**2.6**	**508**	**29.5**	**2.8**
1,300	872	40.3	2.5	567	32.0	2.6
1,400	1,060	46.5	2.2	684	36.8	2.4
1,500	1,248	52.5	2.0	800	41.4	2.2
1,600	1,437	58.3	1.9	917	45.9	2.0
1,700	1,625	64.1	1.7	1,034	50.4	1.9
1,800	1,813	69.7	1.6	1,150	54.8	1.7
1,900	2,001	75.3	1.5	1,266	59.0	1.7
2,000	2,189	80.8	1.4	1,382	63.1	1.6

Source: JDS (2016)

Table 23.7: F/X Rate Sensitivity

F/X Rate	Pre Tax $NPV_{5\%}$ (M$)	Pre-Tax IRR (%)	Pre-Tax Payback	After Tax $NPV_{5\%}$ (M$)	After-Tax IRR (%)	After-Tax Payback
0.65	1,245	52.4	2.0	798	41.3	2.2
0.70	1,045	46.0	2.2	674	36.4	2.4
0.75	871	40.3	2.5	567	32.0	2.6
0.78	**778**	**37.1**	**2.6**	**508**	**29.5**	**2.8**
0.80	720	35.1	2.7	472	27.9	2.9
0.90	467	25.9	3.3	314	20.8	3.4
1.00	265	17.7	4.0	184	14.5	4.3

Source: JDS (2016)

A sensitivity analysis of the pre-tax and after-tax results was performed using various discount rates. The results of this analysis are demonstrated in Table 23.7. The cash flow model is shown in Table 23.9.

Table 23.8: Discount Rate Sensitivity Test Results on NPV

Discount Rate (%)	Pre-Tax NPV (M$)	After-Tax NPV (M$)
0	1,067	714
5	778	509
7	687	443
10	570	360

Source: JDS (2016)

Item		Unit	Pre-Production	Production	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
METAL PRICE & EXCHANGE RATE																									
Au	link	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Exchange Rate	link	US$:C$	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78
PRODUCTION SCHEDULE																									
Eagle																									
TOTAL EAGLE																									
Ore	calc	ktonnes	25	116,350	116,375	-	-	25	9,864	12,592	12,466	11,317	12,341	12,663	13,164	12,764	12,795	6,386	-				-	-	-
Grade	calc	g/t	0.42	0.66	0.66	-	-	0.42	0.70	0.74	0.71	0.77	0.74	0.65	0.56	0.56	0.52	0.66	-				-	-	-
Olive																									
TOTAL OLIVE																									
Ore	calc	ktonnes	-	6,524	6,524	-	-	-	-	-	-	-	-	-	-	-	690	5,353	481				-	-	-
Grade	link	g/t	-	0.95	0.95	-	-	-	-	-	-	-	-	-	-	-	1.15	0.94	0.75				-	-	-
TOTAL MINED																									
Ore	calc	ktonnes	25	122,874	122,899			25	9,864	12,592	12,466	11,317	12,341	12,663	13,164	12,764	13,485	11,739	481	-	-	-	-	-	-
Grade	calc	g/t	0.42	0.67	0.67			0.42	0.70	0.74	0.71	0.77	0.74	0.65	0.56	0.56	0.55	0.79	0.75	-	-	-	-	-	-
Waste	link	ktonnes	2,065	114,244	116,309			2,065	6,886	14,070	12,122	15,320	10,026	7,745	7,791	9,291	16,296	14,472	224	-	-	-	-	-	-
Strip Ratio	calc	w:o	83.5	0.93	0.95			83.5	0.7	1.1	1.0	1.4	0.8	0.6	0.6	0.7	1.2	1.2	0.5	-	-	-	-	-	-
Mining Rate	calc	ktpd	26	66	66			11	46	73	67	73	61	56	57	60	82	72	2	-	-	-	-	-	-
Rehandle	link	ktonnes	16	25,785	25,801			16	665	2,738	2,737	2,738	2,738	2,739	2,738	2,737	2,737	2,738	481	-	-	-	-	-	-
Total Mined	**calc**	**ktonnes**	2,090	237,118	239,208			2,090	16,751	26,663	24,588	26,636	22,367	20,408	20,954	22,055	29,781	26,211	705	-	-	-	-	-	-
HEAP SCHEDULE																									
Contained Au	calc	koz	0	2,663	2,663	-	-	0	222	301	285	279	294	266	238	229	240	298	12	-	-	-	-	-	-
Eagle Crushed Ore																									
Ore to Heap	link	ktonnes	-	101,283	101,283			-	8,776	10,950	10,949	10,950	10,950	10,950	10,951	10,949	10,260	5,597	-	-	-	-	-	-	-
Au Recovered	link	koz	-	1,697	1,697				139	201	205	209	205	184	156	150	135	86	28	-	-	-	-	-	-
Olive Ore																									
Ore to Heap	link	ktonnes	-	6,524	6,524				-	-	-	-	-	-	-	-	690	5,353	481	-	-	-	-	-	-
Au Recovered	link	koz	-	113	113				-	-	-	-	-	-	-	-	16	91	6	-	-	-	-	-	-
ROM																									
Ore to Heap	link	ktonnes	-	15,092	15,092				1,113	1,642	1,517	367	1,391	1,712	2,213	1,814	2,535	788	-	-	-	-	-	-	-
Au Recovered	link	koz	-	73	73				3	7	7	4	5	8	10	10	11	7	1	-	-	-	-	-	-
TOTAL																									
Ore to Heap	link	ktonnes	-	122,899	122,899	-	-	-	9,889	12,592	12,466	11,317	12,341	12,663	13,164	12,764	13,485	11,739	481	-	-	-	-	-	-
Au Recovered	link	koz	-	1,884	1,884	-	-	-	142	208	213	213	210	192	166	160	162	184	35	-	-	-	-	-	-
PAYABLE METALS																									
Total Recovered Au	link	koz	-	1,884	1,884	-	-	-	142	208	213	213	210	192	166	160	162	184	35	-	-	-	-	-	-
Payable Au	link	%	99.5%	99.5%	99.5%	-	-	-	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	-	-	-	-	-	-
	calc	koz	-	1,874	1,874	-	-	-	142	207	212	212	209	191	165	159	161	183	35	-	-	-	-	-	-
	calc	US$M	-	2,342.8	2,342.8	-	-	-	177.1	259.1	264.5	264.5	260.7	238.4	206.2	198.8	201.0	228.4	44.0	-	-	-	-	-	-
	calc	C$M	-	3,003.6	3,003.6	-	-	-	227.1	332.2	339.1	339.1	334.2	305.7	264.4	254.9	257.7	292.9	56.4	-	-	-	-	-	-
Refining Costs	link	US$/payable oz	10.00	10.00	10.00	-	-	-	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	10.00	-	-	-	-	-	-
	calc	C$M	-	24.0	24.0	-	-	-	1.8	2.7	2.7	2.7	2.7	2.4	2.1	2.0	2.1	2.3	0.5	-	-	-	-	-	-
NIV	calc	C$M	-	2,979.6	2,979.6	-	-	-	225.2	329.5	336.4	336.4	331.6	303.2	262.3	252.8	255.7	290.5	55.9	-	-	-	-	-	-
Royalties	calc	C$M	-	29.8	29.8	-	-	-	2.3	3.3	3.4	3.4	3.3	3.0	2.6	2.5	2.6	2.9	0.6	-	-	-	-	-	-
Net Smelter Return (NSR) after Royalties	**calc**	**C$M**	**-**	**2,949.8**	**2,949.8**	**-**	**-**	**-**	**223.0**	**326.2**	**333.0**	**333.0**	**328.2**	**300.2**	**259.7**	**250.3**	**253.1**	**287.6**	**55.4**	**-**	**-**	**-**	**-**	**-**	**-**
OPEX																									
Mining	calc	C$/t mined			2.15	-	-	-	2.62	2.18	2.00	2.05	2.18	2.26	2.31	2.22	2.13	1.87	6.20	-	-	-	-	-	-
	link	C$M	0.0	514.8	514.8		-	-	43.8	58.2	49.1	54.7	48.7	46.2	48.4	48.9	63.4	49.1	4.4	-					
Processing	calc	C$/t leached			4.93				5.15	4.35	4.27	4.82	4.44	4.33	5.02	4.79	4.82	5.91	43.62	-	-	-	-	-	-
	link	C$M	0.0	605.7	605.7		-	-	50.9	54.8	53.3	54.5	54.8	54.8	66.0	61.1	65.0	69.4	21.0	-					
G&A	calc	C$/t leached			1.42				1.70	1.34	1.35	1.50	1.37	1.33	1.28	1.32	1.27	1.44	11.14	-	-	-	-	-	-
	link	C$M	0.0	174.5	174.5		-	-	16.8	16.9	16.9	17.0	16.9	16.9	16.9	16.9	17.2	16.9	5.4	-	-	-	-	-	-
Total Opex	**calc**	**C$M**	**0.0**	**1,295.0**	**1,295.0**	**-**	**-**	**-**	**111.5**	**129.8**	**119.2**	**126.2**	**120.5**	**117.8**	**131.3**	**126.9**	**145.6**	**135.5**	**30.7**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**C$/payable oz**	**0**	**691**	**691**	**-**	**-**	**-**	**787**	**626**	**563**	**596**	**578**	**618**	**796**	**798**	**905**	**741**	**872**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**C$/t leached**	**0.00**	**10.54**	**10.54**	**-**	**-**	**-**	**11.27**	**10.31**	**9.56**	**11.15**	**9.76**	**9.31**	**9.97**	**9.94**	**10.80**	**11.54**	**63.85**	**-**	**-**	**-**	**-**	**-**	**-**
Net Operating Cashflow	**calc**	**C$M**	**0.0**	**1,654.8**	**1,654.8**	**-**	**-**	**-**	**111.5**	**196.4**	**213.8**	**206.8**	**207.8**	**182.3**	**128.4**	**123.4**	**107.6**	**152.1**	**24.7**	**-**	**-**	**-**	**-**	**-**	**-**
CAPEX																									
Mining Equip. and Pre-Production Development	link	C$M	34.5	45.6	80.1			34.5	30.2	4.3	0.1	0.1	2.2	4.6	4.0	-	-	-	-	-	-	-	-	-	-
Site General	link	C$M	17.7	9.9	27.5			17.7	8.2							1.7									
Process	link	C$M	101.3	-	101.3			101.3																	
Ancillaries	link	C$M	22.2	30.3	52.5			22.2						30.3											
Power Supply and Distribution	link	C$M	15.1	0.8	16.0			15.1						0.8											
Water Management	link	C$M	5.7	15.0	20.7			5.7			15.0														
Heap Leach Facilities	link	C$M	56.3	81.6	137.9			56.3	2.9	9.8		21.9		39.2	7.8										
Owner's Costs	link	C$M	8.6	-	8.6			8.6																	
Indirects	link	C$M	72.9	-	72.9			72.9																	
Closure (Net of Salvage)	link	C$M	-	35.0	35.0		-	-	-	-	-	-	-	-	-	-	-	-	2.0	10.0	15.0	2.0	2.0	2.0	2.0
Subtotal	**calc**	**C$M**	**334.4**	**218.1**	**552.5**	**-**	**-**	**334.4**	**41.3**	**14.1**	**15.1**	**22.0**	**2.2**	**74.9**	**11.9**	**1.7**	**-**	**-**	**2.0**	**10.0**	**15.0**	**2.0**	**2.0**	**2.0**	**2.0**
Contingency	calc	C$M	35.2	-	35.2		-	35.2																	
CAPEX incl. Contingency	**calc**	**C$M**	**369.6**	**218.1**	**587.7**	**-**	**-**	**369.6**	**41.3**	**14.1**	**15.1**	**22.0**	**2.2**	**74.9**	**11.9**	**1.7**	**-**	**-**	**2.0**	**10.0**	**15.0**	**2.0**	**2.0**	**2.0**	**2.0**
CAPEX Breakdown																									
Pre-Production	*calc*	*C$M*	369.6	-	369.6	-	-	*369.6*																	
Sustaining & Closure	*calc*	*C$M*	-	218.1	218.1				*41.3*	*14.1*	*15.1*	*22.0*	*2.2*	*74.9*	*11.9*	*1.7*	*-*	*-*	*2.0*	*10.0*	*15.0*	*2.0*	*2.0*	*2.0*	*2.0*

WORKING CAPITAL																									
Working Capital																									
Working Capital	*calc*	C$M	-	-	-	-	-	-	*3.3*	-	-	-	-	-	-	-	-	-	*(3.3)*	-	-	-	-	-	-
TAXES																									
Income Taxes																									
Income Taxes	*link*	C$M	-	**242.4**	**242.4**				-	-	-	*36.8*	*54.8*	*44.3*	*28.4*	*27.3*	*24.6*	*37.6*	*3.9*	*(5.0)*	*(9.7)*	*(0.8)*	-	-	-
Yukon Quartz Tax	*link*	C$M	-	**110.5**	**110.5**				*4.1*	*13.7*	*15.5*	*14.3*	*14.4*	*7.0*	*6.4*	*10.7*	*9.1*	*14.6*	*0.7*	-	-	-	-	-	-
Total Taxes	***calc***	**C$M**	-	**353.0**	**353.0**	-	-	-	***4.1***	***13.7***	***15.5***	***51.1***	***69.2***	***51.3***	***34.9***	***38.0***	***33.7***	***52.3***	***4.6***	***(5.0)***	***(9.7)***	***(0.8)***	-	-	-
CASH FLOWS																									
Pre-Tax																									
Net Pre-Tax Cashflow	**calc**	**C$M**	**(369.6)**	**1,436.7**	**1,067.1**	-	-	**(369.6)**	**66.9**	**182.3**	**198.7**	**184.9**	**205.5**	**107.4**	**116.6**	**121.7**	**107.6**	**152.1**	**26.0**	**(10.0)**	**(15.0)**	**(2.0)**	**(2.0)**	**(2.0)**	**(2.0)**
	calc	US$M	(288.3)	1,120.6	832.3	-	-	(288.3)	52.2	142.2	155.0	144.2	160.3	83.8	90.9	94.9	83.9	118.7	20.3	(7.8)	(11.7)	(1.6)	(1.6)	(1.6)	(1.6)
Cumulative Net Cashflow	**calc**	**C$M**				-	-	**(369.6)**	**(302.6)**	**(120.3)**	**78.3**	**263.2**	**468.8**	**576.2**	**692.7**	**814.4**	**922.0**	**1,074.1**	**1,100.1**	**1,090.1**	**1,075.1**	**1,073.1**	**1,071.1**	**1,069.1**	**1,067.1**
	calc	US$M				-	-	(288.3)	(236.1)	(93.9)	61.1	205.3	365.6	449.4	540.3	635.3	719.2	837.8	858.1	850.3	838.6	837.0	835.5	833.9	832.3
After-Tax																									
Net After Tax Cash Flow		**C$M**	**(369.6)**	**1,083.7**	**714.2**	-	-	**(369.6)**	**62.9**	**168.6**	**183.2**	**133.8**	**136.4**	**56.1**	**81.7**	**83.7**	**73.8**	**99.9**	**21.4**	**(5.0)**	**(5.3)**	**(1.2)**	**(2.0)**	**(2.0)**	**(2.0)**
		US$M	**(288.3)**	**845.3**	**557.0**	-	-	**(288.3)**	**49.0**	**131.5**	**142.9**	**104.3**	**106.4**	**43.8**	**63.7**	**65.3**	**57.6**	**77.9**	**16.7**	**(3.9)**	**(4.2)**	**(1.0)**	**(1.6)**	**(1.6)**	**(1.6)**
Cumulative After-Tax Cash Flow	**calc**	**C$M**				-	-	**(369.6)**	**(306.7)**	**(138.1)**	**45.0**	**178.8**	**315.2**	**371.3**	**453.0**	**536.7**	**610.5**	**710.4**	**731.7**	**726.7**	**721.4**	**720.2**	**718.2**	**716.2**	**714.2**
		US$M				-	-	**(288.3)**	**(239.2)**	**(107.8)**	**35.1**	**139.4**	**245.8**	**289.6**	**353.3**	**418.6**	**476.2**	**554.1**	**570.8**	**566.8**	**562.7**	**561.7**	**560.2**	**558.6**	**557.0**
ECONOMIC INDICATORS																									
Pre-Tax																									
Pre-Tax IRR	**calc**	**%**			**37.1%**																				
Pre-Tax Payback Period	**calc**	**Years**			**2.6**																				
Pre-Tax NPV @ 5%	**calc**	**C$M**			**778.0**																				
	calc	**US$M**			**606.8**																				
Pre-Tax NPV @ 0%	**calc**	**C$M**			**1,067.1**																				
	calc	**US$M**			**832.3**																				
After-Tax																									
After-Tax IRR	**calc**	**%**			**29.5%**																				
After-Tax Payback Period	**calc**	**Years**			**2.8**																				
After-Tax NPV @ 5%	**calc**	**C$M**			**508.5**																				
	calc	**US$M**			**396.6**																				
After-Tax NPV @ 0%	**calc**	**C$M**			**714.2**																				
	calc	**US$M**			**557.0**																				

24 Adjacent Properties

There are no adjacent properties pertaining to this project.

25 Other Relevant Data and Information

25.1 Project Execution Plan

25.1.1 Introduction

This Project Execution Plan (PEP) has been developed for the Eagle Gold project FS based on the latest FS information available and best practices updated from the 2012 study. It describes the strategy for moving forward on the engineering, procurement, construction and environmental activities.

25.1.1.1 Plan Objective

The development of a practical PEP at this stage of the project is integral to the success of the next phase of the work as it enters the detailed engineering stage. It helps form the basis for the ongoing work as it provides the blueprint upon which assumptions were made during the FS used to best understand rationale behind developing the FS study.

While many assumptions were made, they were based on actual northern construction experience and know-how.

The PEP includes discussion of how the following activities will be managed:

- Detailed engineering;
- Long lead delivery equipment;
- Freight;
- Construction field requirements;
- Ordering bulk materials;
- Site environmental requirements;
- Site safety requirements;
- Site security requirements;
- Construction resources;
- Accommodation for construction and operating work force; and
- Commissioning the plant and handover to Owner.

25.1.1.2 Pre-Construction Phase

Once the initial FS is completed, a program of continued development will be prepared. The program includes activities to be undertaken between the period leading up to the project approval to proceed with development. Continued development will encompass the following:

- Additional geotechnical drilling and test pitting to support final design;
- Continuation of environmental monitoring;
- Revised permit application support;
- Project design and construction optimization;
- Researching local resource availability;
- Sourcing used equipment as appropriate;
- Negotiation with long delivery vendors;
- Establishing the availability and suitability of contractors;
- Establishing the Owner in the fabrication line-up for long delivery equipment;
- Finalizing Owner commitments to the project including the mine plan;
- Improving site access;
- Providing construction access to the major construction zones with in the project footprint;
- Providing construction drainage in and around the leach pad facilities; and
- Camp expansion.

25.1.1.3 Basic Engineering Phase

Basic Engineering work can start when the Owner releases sufficient funding. It is envisaged that certain items will be finalized with a view to prepare for the detailed engineering and construction stage, including:

- Developing the project management control document;
- Flow sheet finalization;
- Long delivery equipment ordering; crushers, ADR plant, main transformers and power plant generators;
- Water balance finalization;
- General arrangement drawings fixed;
- Constructability reviews;
- Bulk earthwork drawings brought up to a level for construction;
- HLP Phase 1 drawings brought up to a level for construction;
- Overhead power line design route surveyed and fixed;
- Testing site aggregate for concrete;

- Tendering for the clearing and grubbing;
- Tendering for the road upgrade work;
- Establishing the expansion of the existing camp;
- Establishing the boiler plate for contracts and purchase orders;
- Finalizing the project schedule based on all the information gathered to that point;
- Establishing the cost reporting and control system;
- Establishing the field survey contract;
- Establishing the quality assurance contract;
- Arranging the freight forwarding contract; and
- Arranging the temporary construction facilities including fuel and water.

25.1.1.4 Detailed Engineering (Design) and Procurement

Once the Owner has established the Eagle Gold project engineering and procurement (EP) team and sufficient financing is in place, that part of the development program can start. It is expected that some basic engineering, as generally described previously, will be undertaken before full financing is available so that the detailed work can start in earnest to produce construction drawings for the earthworks and civil phases of the project when the time comes.

Capital equipment purchases will be made based on the flow sheet and performance specifications for all items that have been assigned an equipment number, right down to instruments.

The EP team will develop packages of drawings (deliverables) into groupings that follow the construction contracting strategy; concrete, buildings, structural steel, and piping, etc. For the Eagle Gold project, it is intended that work packages will be bid and awarded based on availability of engineering information, and is expected to be a progressive situation.

25.1.2 Health, Safety, Environmental and Security

Health, Safety and Environmental (HSE) programs and initiatives are essential to project success. A fully-integrated program will be implemented to help achieve a “zero-harm” goal. To achieve this, key project stakeholders will be asked to share in this responsibility by providing the leadership and commitment to attain the highest standards and values. A high level of communication, motivation and involvement will be required in the development of HSE practices, including alignment with site contractors on topics such as safety training, occupational health and hygiene, hazard and risk awareness, safe systems of work and job safety analysis. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

All contractors will be required to pay particular attention to construction safety, and to provide individual safety programs and safety plans to the satisfaction of the construction management team.

The design of the Eagle Gold project includes sound environmental protective approaches, including the location of mine, WRSAs, crushing facilities, HLPs, event ponds, sediment control ponds, process plant, and related structures in order to capture and treat surface water runoff.

All design and engineering stages incorporate criteria for responsible management of process flows, effluent and waste products to meet established capture and containment guidelines. The design also incorporates basic clean plant design standards, including operational safety and maintenance access requirements. A Hazard and Operability Analysis (HAZOP) will be conducted by the project design team during the detailed design stage for each area of the plant. This analysis will strive to eliminate hazards identified during the design phase. This systematic team approach will identify hazards associated with operability that requires attention in order to eliminate undesirable consequences. Environmental protection will be incorporated in both the design of the main processes of the plant, as well as in the transportation, storage and disposal of materials within and outside the boundaries of the plant.

The Owner will provide a well-equipped first aid facility, ambulance and fire protection for project wide use. The first aid facility will normally be staffed 12-hours per day, with on-call services available to ensure continuous coverage. The first aid staff will live at the camp. Contractors will be expected to provide basic first aid-stations for their workers at the site.

Access to the site will be controlled at the principal road entrance, and will be limited to personnel who have attended induction training, as well as approved visitors.

25.1.3 Execution Strategy

The execution strategy for the successful monitoring and control of the Eagle Gold project reflects a traditional approach to project execution, with field construction contractors, under the direction of a construction management (CM) team commencing work after engineering tasks are well advanced, and long lead times for the delivery of major equipment have been established combined with the Victoria Gold corporate objectives for sustainable development as earlier described.

25.1.3.1 Management Procedures

Although this document refers to the EPCM Consultants as a single entity, the Owner may issue separate contracts for engineering and procurement services, and the CM services.

The project team will combine the talents of the Owner, engineer and construction managers who will be charged with the responsibility for bringing the project in on time and within budget, using the strategies outlined here.

A comprehensive project procedures manual (the procedures), developed by the EPCM group, in conjunction with the Owner, will outline the procedures and requirements for the execution of the administrative activities, as well as Owner and EP and CM contractor's rights, authorities and obligations of the project.

25.1.3.2 Document Control

Effective document management is crucial to the successful implementation of the project. A collaborative document control system will be implemented that provides status and version control for all documents issued on the project. The system will be capable of publishing documents, text, drawings, photographs or 3D models to the internet. The documents, particularly vendor drawings and manuals, will be linked to the equipment database, in order to have an organized accessible control system during the project which can be turned over to the operations group at the completion of the design and construction phases.

25.1.3.3 Cost Management System

The project Work Breakdown Structure (WBS) defines the elements of project scope, each of which can stand alone with estimate, cost, schedule and accountability. The first step in the project implementation process will be to confirm the WBS and distribute the control estimate to this structure.

At the end of basic engineering, the project will produce a definitive estimate, which will become the control estimate for the execution of the project. Budgets will be cast for the scope approved at the time and will become the baseline control document against which the project will be measured.

25.1.3.4 Risk Management

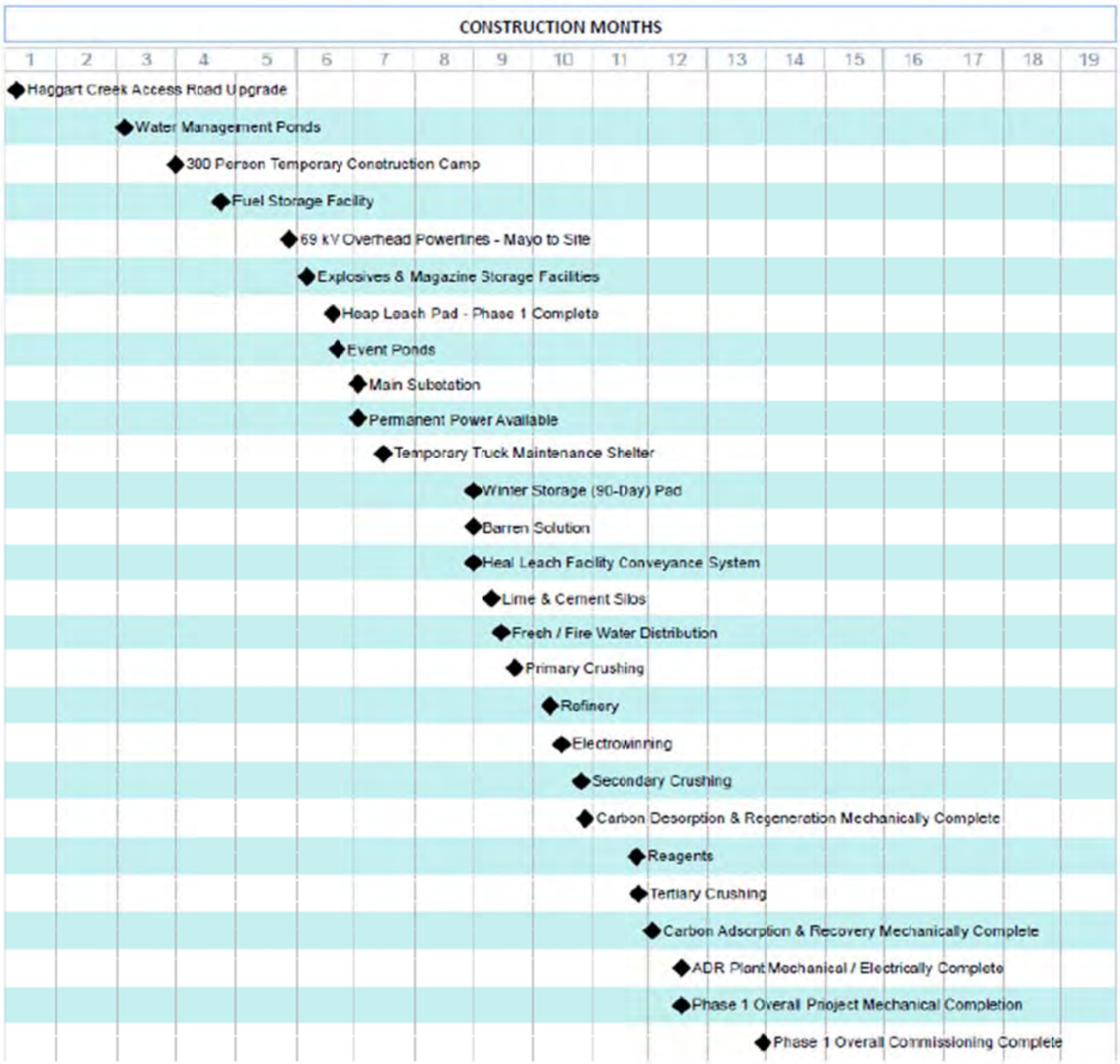
The formal risk management program began during the FS phase, and will continue, through to commissioning. The project team will review all aspects of the project throughout the developmental stage, inclusive of environmental, technical, health and safety, community, business and project delivery issues. These reviews will identify the relevant risks and or opportunities associated with this project, assess those risks and opportunities against the outcome objectives and determine the best way to eliminate or control those risks or take advantage of opportunities that may present themselves.

25.1.3.5 Project Scheduling and Progress Reporting

The overall project schedule (schedule) identifies the preferred critical sequences and target milestone dates that need to be managed for the project to be executed successfully. While executive level reports provide an overview of project status and forecasts, the detailed schedules track the planned and actual progress throughout the duration of the project using information provided by the engineering groups, contractors, vendors, the field management staff and the Owner.

The project construction duration assumes commencement of field activity in early spring of Year-1 to mechanical completion in 12.5 months.

Figure 25.1: PEP Schedule



Source: Merit (2016)

The following basic project tasks will need to be attended to as early as possible after the project has been approved to proceed in order to guarantee planning certainty and maintain a proper monitoring program for all long lead items and engineering deliverables

- Selecting the general EPCM group(s);
- Selecting the EP consultant for the final design of the Heap leach pad (HLP);
- Selecting the EP consultant for the final design of the high voltage power line;

- Ordering long delivery capital equipment;
- Establishing the cost reporting system based on the approved CAPEX;
- Verifying the schedule to account for when the project will actually start;
- Establishing the project procedures; and
- Establishing the standard formats for purchasing and contracts.

25.1.4 Construction Labour Requirement

The schedule has been based on a 70-hour work week with some double shifting as required. Crew rotations are planned to be three weeks on-site and one week off-site.

It is anticipated that there are about 780,000 man-hours of direct construction labour associated with the project construction, excluding mine pre-development and engineering with construction manpower peaking at about 380 direct construction workers on-site.

25.1.5 Construction Camp

The permanent camp with 208 bedrooms and all supporting infrastructure will be in place prior to the start of construction, This facility will be shared with all personnel supporting the project, The approximately 300 additional beds required to support the entire construction workforce will be made available by adding temporary rental dorms and other facility modules as required. The growth of the camp will continue in phases as requirements demand to accommodate up to 500 persons overall during peak construction activities including Owner's Reps and Visitors.

The camp will be single occupancy, and will be provided with recreational facilities and a commissary.

The camp will eventually accommodate the Owner's operating personnel (including mining pre-stripping operators) other than those who will reside within the local communities. The construction crews working on the high voltage power to site line and the Haggart Creek access road upgrades may opt to provide their own mobile camps to be located where convenient along the route of their work, however their manpower levels are currently included in the aforementioned construction camp sizing.

The CM team will manage the camp and catering contractor (by Contract) to ensure that quality service is provided including hygiene, food storage and handling, menus and nutritional value, and staff qualifications.

25.1.6 Housekeeping and Hazardous Waste Management

Specific procedures for waste management and spill response will be implemented for the construction period. These procedures will be defined in the project procedures and cover compliance, auditing and reporting requirements. Procedures regarding ongoing clean-up and rubbish removal, as well as safe handling, storage and disposal of batteries, fuels, oil and hazardous materials, will be established and observed for the duration of the construction phase. Waste will be recycled to the extent feasible.

Ongoing dust suppression and rain water management programs will also be established and observed for the duration of the construction phase. Specific procedures and storage areas will be designated for construction waste prior to recycling or removal from the plant. Solid waste will be disposed of in designated pits, while biologically degradable wastes will be incinerated or removed to a suitable off-site disposal site.

25.1.7 Construction Equipment

Construction equipment will generally be the responsibility of individual contractors. Contractor equipment must comply with the requirements of the Mine Safety Branch in so far as safety and operability are concerned and spot checks will be made by the Owner's safety personnel to ensure compliance. No mobile equipment is permitted to operate on-site unless it complies with the mining regulations, and no cranes are allowed to operate without current inspections. Any modifications to equipment have to be certified fit for operation – especially where welding is concerned.

The Owner may choose to supply the large construction cranes to be managed by the CM team.

25.1.8 Communication

The Owner will determine the appropriate temporary (for construction) and permanent microwave telecommunications technologies for the project, with input by the CM team where needed. Requirements will include voice and data link technologies to support growth in both construction and plant operation needs.

The communications framework for management offices will be installed early in the construction period. The system will be supplemented with the installation of telephones in common areas, and for individual room internet access

25.1.9 Construction Power

Construction power will be supplied by portable generators situated close to the major work areas; approximately 0.5 MW of construction power will be required, with an additional 0.75MW for the camp. This power will be supplied by low noise, low emission, temporary generator sets. Permanent power will be available by 3rd quarter of Year -1 and will supply power for all mine equipment and peak construction power loads for the balance of the construction phase through winter. The temporary units will be kept on-site until at least the end of the construction period and maintained as emergency power units or operated in some of the more remote environments to replace the contractor's construction units.

25.1.10 Commissioning

Except for the mining activities, the EPCM team will have responsibility for the installation of the facilities until mechanical completion.

The sequence of system commissioning is vital to shifting the construction schedule from general area completion to more specific system completion to suit the commissioning and start-up of the entire facility.

System identification and prioritization must be expedited to allow for the construction schedule adjustments and the completion of the work, in order to satisfy the established commissioning sequence

During the latter part of construction the Owner will develop the commissioning plan in conjunction with the CM forces. The systems will be identified and scheduled for delivery by priority. Packages will be assembled for each system that has to be commissioned to include all sign off and test documentation, drawings and vendor information.

As the various systems are completed, and determined by the CM team to be free of deficiencies that would prevent safe operation they will be transferred to the Owner’s operations team. The Owner’s team will consist of plant operators and maintenance staff who will enlist the help of vendors, contractors and CM personnel as needed to Dry Run and then Wet Run the systems until they are finally accepted by the Owner’s operations management. The transfer of systems will be formally documented and include all mechanical/electrical testing documents and vendor’s information.

25.1.11 Construction Methods

25.1.11.1 CM Key Objectives

- Conduct HSE policy training and enforcement for all site and contractor staff. Site hazard management tools and programs will be employed to achieve the no harm/zero accident objective;
- Implement the contracting and construction infrastructure strategies to support the project PEP;
- Develop and implement a construction-sensitive and cost-effective master project schedule;
- Establish a project cost control system to ensure effective cost reporting, monitoring and forecasting as well as schedule reporting and control. A cost trending program will be instigated whereby the EP and CM contractors will be responsible for evaluating costs on an ongoing basis, and provide comparisons to budget and actual project trending for the cost report on monthly basis;
- Establish a field contract administration system to effectively manage, control and coordinate the work performed by the contractors;
- Manage the catering contractor (by contract) to ensure quality service is up to expected standards for the facilities, staff qualifications, hygiene standards, food handling, storage and provision of meals;
- Apply effective field constructability program, as a continuation of the constructability reviews performed in the design office; and
- Organize purchases of bulk materials.

In addition, they will assemble contract tendering documents, establish qualified bid lists, tender the work, analyze and make recommendations to the Owner for the most suitably qualified contractors, and prepare the executed contracts for issue.

The Site Materials Management group will receive, inspect, and log all incoming materials, assign storage locations, and maintain a database of the status of all materials received and dispensed to the contractors. Ongoing reconciliation with the procurement system, including reconciliation to the freight consolidation point, will confirm that the materials ordered for the project were correctly received, and that the suppliers were paid. An allowance for lease or purchase of offloading equipment, forklifts, storage racks or other equipment required during construction has been included in the construction budget.

Develop a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.

25.1.12 Construction Management (CM) Responsibilities

The construction management (CM) group will be responsible for the management of all field operations. The Construction Manager will be responsible to the Owner to effectively plan, organize, and manage construction quality, safety, budget, and schedule objectives.

The CM Field Engineering Team will employ independent quality assurance specialists, qualified to CSA, to ensure the implementation and success of the contractor's quality control.

Detailed CM responsibilities include, but are not limited to:

Project Management

- Camp management (by contract);
- Camp catering (by contract);
- Camp installation – supported by Owner;
- Insurance, WCB, General Liability, Third Party and Auto;
- Labour Relations plan and site work rules – supported by Owner;
- Freight logistics and deliveries – assist Owner;
- Overall project cost system;
- Scheduling;
- Site offices – assist Owner;
- Site topographical survey – assist Owner and EP; and
- Site utilities for field offices – assist Owner.

Design

- Concrete mix design – with assistance from EP;
- Commissioning – assist EP and Owner;
- Communications system for construction – assist Owner;
- Document control – general project and construction; and
- Constructability reviews – with support from EP and Owner.

Purchasing and Expediting

- Spare parts – start-up and commissioning – assist Owner;
- Vendor reps (erection support and commissioning) – with assistance from EP and Owner;
- Vendor reps – coordination.

Construction

- HSE policy implementation and enforcement;
- Site construction management;
- Warehouse and laydown area;
- Security personnel – assist Owner;
- Contracting Plan;
- Contract bid Documents;
- Contract tendering – post tender meetings and recommendations;
- Contract execution and administration;
- Earthworks and civil site supervision;
- Mechanical and piping site supervision;
- Structural site supervision;
- Electrical and instrumentation site supervision;
- Commissioning – assist Owner and EP;
- On-site monitoring of construction equipment condition and safe operating capability;
- Survey and Layout (Contract) – assist Owner;
- Site quality control (Contract);
- Cost reporting and controls – with EP and Owner support; and
- As-built drawings (by contractors).

25.1.12.1 Critical Path and Installation Methodology

The schedule has been presented in association with Section 25.3. There is only one critical path where there is zero float. Excluding procurement constraints, it currently runs through the construction of the ADR plant.

However, there are other items associated with the overall schedule that need to be carefully managed since the impact of them not being successfully undertaken could have serious consequences on the schedule.

A number of the most important elements of the construction phase of the project were examined in detail during the FS. The reviews were performed by a joint team representing the Owner, engineer, construction management, and sometimes contractors and vendors.

25.1.12.1.1 Heap Leach pad Earthworks

Construction is constrained by work that restricts the placement of certain general and rock fills to the warm and dry months of the year. The completion of the HLP Phase 1 has been scheduled governed in part by the engineer's quality specifications which will determine "no-build" restrictions during the winter.

Activities critical to the construction of the HLP include construction of the Lower Dublin Gulch (South) Sediment Control Pond and the Event ponds #1&2 which are a source of construction materials.

25.1.12.1.2 Construction Aggregates & Fill Materials

Current on-site estimates indicate there would be quality materials available at the pit which would be hauled to and stockpiled at the crushing/ screening plant to be located near the concrete batch plant pad which would be in the general vicinity of the primary crusher. The contractor operated facility will be tasked with producing crushed and screened materials suitable for concrete aggregates.

In addition, the following materials will be sourced and processed from the on-site placer tailings suitable for various fills such as:

- Clean topping for Substation;
- Engineered backfill for retaining walls;
- Site road dressing;
- Piping, electric conduit and cable trench protective backfill layers; and
- The portable crushing/screening plant will also supply the HLP over-liner materials from suitable waste or low grade ore supplied by the mining pre-production effort.

25.1.12.1.3 Permanent Power

Installation of 69kv permanent power to the main substation at site and 13.8kv site distribution will be available for bumping the crusher motors. Pre-commissioning and commissioning of electrical distribution power lines and equipment will commence thereafter.

The general contractors and sub-contractors will be responsible for:

- Provision of all construction labour;
- Provision of all construction equipment;
- Transportation of their workers;
- Site offices and temporary services;
- Site management;
- Contractor surveying;
- Quality Control Program - in accordance with the construction technical specifications and the applicable codes and standards;
- Contract scheduling;
- Safety;
- Environmental safeguarding;
- Security for their tools and equipment;
- Supplying permanent materials as required by contract;
- Meaningful project procedures;
- Commissioning assistance.

25.1.13 Construction Equipment Supply Philosophy

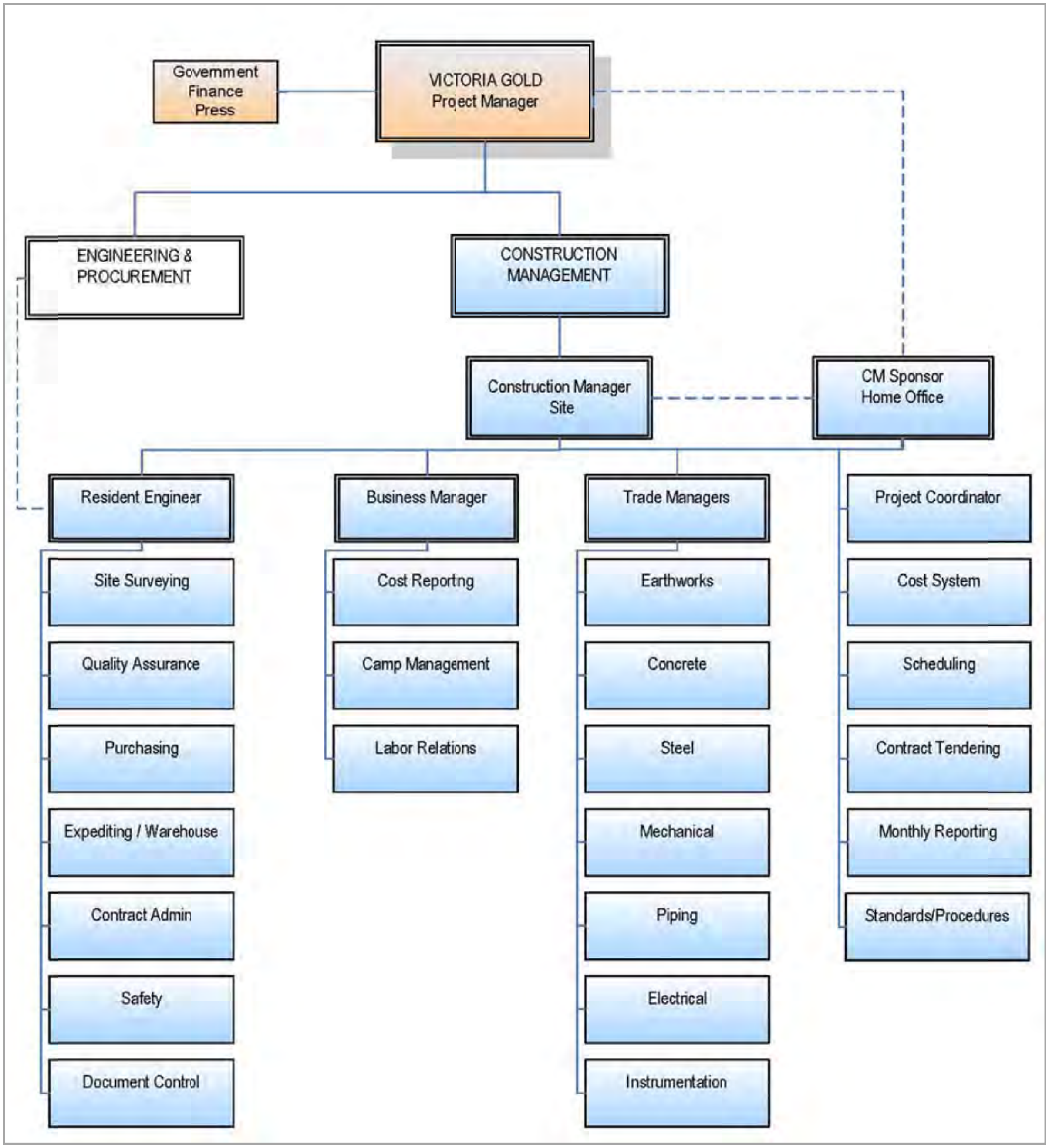
Generally the construction contractors would be responsible for the supply of all equipment required for construction. However, there are cost savings and efficiencies to consider if the Owner provides cranes and manages their use among the contractors. While there are some pieces of equipment included in the budgeted Owner’s mobile fleet, it is intended they be used by the Owner during construction for miscellaneous purposes such as road grading, ditching and the like. This fleet is not available to the construction contractors.

25.1.14 Project Team Responsibilities

Understanding the relationship and responsibilities of the various groups that contribute to making up the project team during the engineering and construction phase of the project is fundamental to the success of the project. There are accompanying methods of establishing inter-relationships including communications matrices, organization charts for communications and reporting and the like, but the two fundamental pieces of the management structure are the overall responsibility matrix and the project organization chart. Both these are presented here.

Selection of the EP and CM providers are the Owner's responsibility. The Owner will establish an Owner's team to oversee the project execution. An organizational chart of the potential project construction team is shown in Figure 25.2

Figure 25.2: Construction Management Organization Chart



Source: Merit (2016)

26 Interpretations and Conclusions

The economic results of this FS demonstrate that the project has positive economics and warrants development. It is recommended that the project proceed to detailed design and construction.

Standard industry practices, equipment and processes were used in this study. The authors of this report are not aware of any unusual or significant risks, or uncertainties that could affect the reliability or confidence in the project based on the data and information made available.

26.1 Risks

Most mining projects are exposed to risks that might impact the economics of the project to varying degrees. Most risks are external and largely beyond the control of the project proponents. They can be difficult to anticipate and mitigate although, in many instances, some reduction in risk might be achieved by regular reviews and interventions over the life of the project.

External risks are things such as the political situation in the project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects.

Table 26.1 summarizes the significant project risks for the Eagle Gold project, including the potential impacts, and possible mitigations. A formal review of the risk likelihood and consequence ratings and pre- and post-mitigation rankings was not conducted: this will be performed during detailed engineering.

The typical risks associated with open pit mining related to dilution, geotechnical and hydrogeological conditions, equipment availability and productivity, and personnel productivity are generally similar to those expected at similar operations.

Although measures to mitigate many of these issues have been identified and applied in the FS, risk identification and review of mitigations will continue to be a priority during project development, construction and operations.

Table 26.1 Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success.	Further cost estimation accuracy with detailed engineering as well as establishing an incentive-based EPCM contract as well as further investigation of cost reduction measures will mitigate over-runs.
	If OPEX increases then the NSR cut-off would increase and, all else being equal, the size of the mineable resource would reduce yielding fewer mineable tonnes.	
Engineering Assumptions	Geotechnical and hydrogeological assessments of pit, HLPs and infrastructure stability are important and if conditions are worse than assumed are encountered modifications may impact ore tonnes, WRSA capacity, strip ratio etc.	Some additional infrastructure geotech work is required as part of detailed designs. Ongoing monitoring of slopes will be conducted and adaptive management plans utilized as needed.
	Additionally, crushing, HLP and ADR plant performance and all other unit operations have all been designed using engineering analyses that are based on small but representative samples that may be different than reality.	Ongoing consideration of potential variances in engineering assumptions during detailed engineering .
Metallurgical Recoveries	Need to increase agglomeration could lead toincreased processing costs, and/or changes to the processing circuit design.	Additional sampling, test work and operational experience during the first year of operations.
Not securing a Power Purchase Agreement in time for operations	Insufficient power or higher cost to support production	Rent or purchase additional generator capacity for on-site power generation
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the project.	Current manpower market conditions are favourable and establishing quality construction and operations teams should not be an issue.
	High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	

Source: JDS (2016)

26.2 Opportunities

Several opportunities have been identified during the FS and merit further investigation. The main opportunities are summarized in Table 26.2.

Table 26.2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of Mineable Resources	*The Mineral Resource has not been fully delineated at Olive and there is an opportunity to expand the mineable resource.* *At Eagle and Olive, there is a considerable amount of mineralized material left in the floor and walls of the pits that could be mined should the COG drop as a result of lower OPEX or increased gold price (in C$ terms),*	*Increased mine life.*
Increased Production	*Increased production may be possible based on stacking the HLPs all year long rather than seasonally for nine months of the year.*	*If stacking is performed year-round (as it is at Kinross's Fort Knox mine near Fairbanks, Alaska) it would provide an additional $50M-$60M/y in net revenue although the mine life would be shortened unless additional ore could be defined.*
Optimize Mine Plan	*Further optimization of the mine plan including equipment and manpower may yield improved results in terms of grade profile, waste removal timing, etc.*	*Improved mining costs and or improved grade profile*
Contract Mining	*Contract mining instead of Owner mining.*	*Reduce CAPEX (but increase OPEX)*
Used Equipment	*Currently there is significant availability of used mining and processing equipment on the market. This FS has only taken advantage of using low-hour used equipment for mining ancillary vehicles and part of the haul truck fleet.*	*Used and "new-used" equipment offers great potential to reduce equipment CAPEX but more importantly reduce procurement time and engineering costs*

Source: JDS (2016)

27 Recommendations

Due to the positive, robust economics, it is recommended to expediently advance the Eagle Gold project to construction and development, and then production. The recommended development path is to continue efforts to advance key activities that will reduce or de-risk the project execution timeline. Associated project risks are manageable, and identified opportunities can provide enhanced economic value.

Value engineering and recommended fieldwork should be advanced in preparation of project financing in order to de-risk the construction schedule and minimize or validate costs.

From project risks and opportunities, the following were identified as critical actions that have the potential to strengthen the project and further reduce risk and should be pursued as part of the early project development plan. The costs for these activities are included in the overall initial capital costs described in Section 21.

The cost for the Engineering & Procurement (EP) is estimated at $7.4M.

The cost for the Construction Management (CM) is estimated at $13.1M

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29 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
'	Minute (Plane Angle)
"	Second (Plane Angle) or Inches
°	Degree
°C	Degrees Celsius
3D	Three-Dimensions
A	Ampere
a	Annum (Year)
AA	Atomic Absorption
ac	Acre
ADR	Adsorption-Desorption-Recovery
AES	Atomic Emission Spectroscopy
amsl	Above Mean Sea Level
ANFO	Ammonium Nitrate/Fuel Oil
ARD	Acid Rock Drainage
Au	Gold
BD	Bulk Density
BFA	Bench Face Angles
BTU	British Thermal Unit
BV/h	Bed Volumes Per Hour
C$	Dollar (Canadian)
Ca	Calcium
CBA	Cooperation And Benefits Agreement
CCA	Capital Cost Allowance
CDE	Canadian Development Expense
CDP	Cyanide Detoxification Plant
CEE	Canadian Exploration Expense
CF	Cumulative Frequency
cfm	Cubic Feet Per Minute
CHP	Combined Heat And Power Plant
CIC	Carbon-In-Column
CIM	Canadian Institute Of Mining And Metallurgy
CIM	Canadian Institute Of Mining
cm	Centimetre
CM	Construction Management
cm^2	Square Centimetre
cm^3	Cubic Centimetre
COG	Cut-Off Grades
Cr	Chromium
CSA	Canadian Securities Administrators
Cu	Copper
CV	Coefficient of Variation
d	Day
d/a	Days per Year (Annum)

Symbol/Abbreviation	Description
d/wk	Days per Week
dB	Decibel
dBa	Decibel Adjusted
DCS	Distributed Control System
DGPS	Differential Global Positioning System
dmt	Dry Metric Ton
EA	Environmental Assessment
EDA	Exploratory Data Analysis
EMR	Energy, Mines and Resources
EP	Engineering and Procurement
EPCM	Engineering, Procurement and Construction Management
FEL	Front-End Loader
FISS	Fisheries Information Summary System
FOB	Free On Board
FOC	Fisheries and Oceans Canada
FS	Feasibility Study
ft	Foot
ft^2	Square Foot
ft^3	Cubic Foot
ft^3/s	Cubic Feet Per Second
g	Gram
G&A	General And Administrative
g/cm^3	Grams Per Cubic Metre
g/L	Grams Per Litre
g/t	Grams Per Tonne
gal	Gallon (Us)
GCL	Geosynthetic Clay Liner
GJ	Gigajoule
GPa	Gigapascal
gpm	Gallons Per Minute (US)
GSC	Geological Survey of Canada
GTZ	Glacial Terrain Zone
GW	Gigawatt
h	Hour
h/a	Hours Per Year
h/d	Hours Per Day
h/wk	Hours Per Week
ha	Hectare (10,000 M2)
HCR	Haggart Creek Road
HG	High Grade
HLP	Heap Leaching Pads
HMI	Human Machine Interface
hp	Horsepower
HPGR	High-Pressure Grinding Rolls
HPW	Highways And Public Works
HQ	Drill Core Diameter Of 63.5 Mm

Symbol/Abbreviation	Description
HSE	Health, Safety and Environmental
HVAC	Heating, Ventilation, and Air Conditioning
Hz	Hertz
ICMC	International Cyanide Management Code
ICP	Inductively Coupled Plasma
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
in	Inch
in^2	Square Inch
in^3	Cubic Inch
IP	Internet Protocol
IRR	Internal Rate Of Return
JDS	JDS Energy & Mining Inc.
K	Hydraulic Conductivity
k	Kilo (Thousand)
KCA	Kappes, Cassiday & Associates
KE	Kriging Efficiency
kg	Kilogram
kg	Kilogram
kg/h	Kilograms Per Hour
kg/m^2	Kilograms Per Square Metre
kg/m^3	Kilograms Per Cubic Metre
km	Kilometre
km/h	Kilometres Per Hour
km^2	Square Kilometre
KNA	Kriging Neighbourhood Analysis
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
KV	Kriging Variance
kVA	Kilovolt-Ampere
kW	Kilowatt
kWh	Kilowatt Hour
kWh/a	Kilowatt Hours Per Year
kWh/t	Kilowatt Hours Per Tonne
L	Litre
L/min	Litres Per Minute
L/s	Litres Per Second
LAN	Local Area Network
LDD	Large-Diameter Drill
LDRS	Leak Detection And Recovery System
LG	Low Grade
LG	Lerchs- Grossman
LOM	Life Of Mine
m	Metre
M	Million
m/min	Metres Per Minute
m/s	Metres Per Second

Symbol/Abbreviation	Description
m^2	Square Metre
m^3	Cubic Metre
m^3/h	Cubic Metres Per Hour
m^3/s	Cubic Metres Per Second
Ma	Million Years
mamsl	Metres Above Mean Sea Level
MAP	Mean Annual Precipitation
masl	Metres Above Mean Sea Level
Mb/s	Megabytes Per Second
mbgs	Metres Below Ground Surface
mbs	Metres Below Surface
mbsl	Metres Below Sea Level
MCC	Motor Control Centres
mg	Milligram
mg/L	Milligrams Per Litre
min	Minute (Time)
mL	Millilitre
Mm^3	Million Cubic Metres
MMER	Metal Mining Effluent Regulations
mo	Month
MPa	Megapascal
MRE	Mineral Resource Estimate
Mt	Million Metric Tonnes
MVA	Megavolt-Ampere
MW	Megawatt
MWMT	Meteoric Water Mobility Tests
MWTP	Mine Water Treatment Plant
NAD	North American Datum
NG	Normal Grade
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm^3/h	Normal Cubic Metres Per Hour
NPVS	NPV Scheduler
NQ	Drill Core Diameter of 47.6 Mm
NRC	Natural Resources Canada
OIS	Operator Interface Stations
OP	Open Pit
ORE	Ore Research And Exploration
OREAS	Ore Research & Exploration Assay Standards
OSA	Overall Slope Angles
oz	Troy Ounce
P.Geo.	Professional Geoscientist
Pa	Pascal
PAG	Potentially Acid Generating
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFS	Preliminary Feasibility Study

Symbol/Abbreviation	Description
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	Probable Maximum Flood
ppb	Parts Per Billion
ppm	Parts Per Million
psi	Pounds Per Square Inch
QA/QC	Quality Assurance/Quality Control
QKNA	Qualitative Kriging Neighbourhood Analysis
QMA	Quartz Mining Act
QML	Quartz Mining License
QMS	Quality Management System
QP	Qualified Person
QQ	Quartile-Quartile
RC	Reverse Circulation
RMR	Rock Mass Rating
ROM	Run-Of-Mine
rpm	Revolutions Per Minute
RQD	Rock Quality Designation
s	Second (Time)
S.G.	Specific Gravity
SARA	Species At Risk Act
Scfm	Standard Cubic Feet Per Minute
SEDEX	Sedimentary Exhalative
SG	Specific Gravity
SMR	South Mcquesten Road
SRK	SRK Consulting Services Inc.
SVOL	Search Volume
t	Tonne (1,000 Kg) (Metric Ton)
t/a	Tonnes Per Year
t/d	Tonnes Per Day
t/h	Tonnes Per Hour
TCR	Total Core Recovery
tph	Tonnes Per Hour
ts/hm^3	Tonnes Seconds Per Hour Metre Cubed
TSS	Total Suspended Solids
US	United States
US$	Dollar (American)
UTM	Universal Transverse Mercator
V	Volt
VEC	Valued Ecosystem Components
VoIP	Voice Over Internet Protocol
VSEC	Valued Socio-Economic Components
w/w	Weight/Weight
WAD	Weak-Acid-Dissociable
WBS	Work Breakdown Structure
wk	Week
wmt	Wet Metric Ton

Symbol/Abbreviation	Description
WRSA	Waste Rock Storage Area
WUL	Water Use License
YEC	Yukon Energy Corporation
YESAA	Yukon Environmental And Socio-Economic Assessment Act
YESAB	Yukon Environmental And Socio-Economic Assessment Board
YG	Yukon Government
µm	Microns
µm	Micrometre

Scientific Notation	Number Equivalent
1.0E+00	1
1.0E+01	10
1.0E+02	100
1.0E+03	1,000
1.0E+04	10,000
1.0E+05	100,000
1.0E+06	1,000,000
1.0E+07	10,000,000
1.0E+09	1,000,000,000
1.0E+10	10,000,000,000

APPENDIX A
QP CERTIFICATES

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CERTIFICATE OF AUTHOR

I, Gordon Doerksen, P.Eng., do hereby certify that:

1. I am currently employed as V.P. Technical Services with JDS Energy & Mining Inc. with an office at Suite 900-999 West Hastings Street, Vancouver, BC, V6C 2W2;

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Professional Mining Engineer (P.Eng. #32273) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Mining Engineer in Yukon Territory. I am a Member of the Canadian Institute of Mining and Metallurgy and a Registered Member of the Society of Mining Engineers of the AIME.

   I am a graduate of Montana Tech with a B.Sc. in Mining Engineering (1990). I have been involved in mining since 1985 and have practiced my profession continuously since 1990. I have held senior mine production and mine technical positions in mining operations in Canada, the US and in Africa. I have worked as a consultant for over eight years and have performed mine planning, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person, for a significant number of engineering studies and technical reports many of which were located in Latin America.
   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 visited the Victoria Gold Project site on September 22 and 23, 2016;

5. I am responsible for Section numbers 1, 2, 3, 4, 5, 6, 18.5, 18.6, 18.7, 18.8, 19, 20, 22, 23, 24, 26, 27, 28 and 29 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of the Technical Report;

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. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Gordon Doerksen, P.Eng."

Gordon Doerksen, P.Eng.

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CERTIFICATE OF AUTHOR

I, Dino Pilotto, P.Eng., do hereby certify that:

1. I am currently employed as Mine Engineering Lead with JDS Energy & Mining Inc. with an office at Suite 900-999 West Hastings Street, Vancouver, BC, V6C 2W2;
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");
3. I am a Professional Mining Engineer (P.Eng. #14782) registered with the Association of Professional Engineers, Geologists of Saskatchewan. I am also a registered Professional Mining Engineer in British Columbia, Alberta, Northwest Territories and Nunavut. I am a graduate of the University of British Columbia with a B.Sc. in Mining and Mineral Process Engineering (1987). I have practiced my profession continuously since June 1987. I have been involved with mining operations, mine engineering and consulting covering a variety of commodities at locations in North America, South America, Africa, and Eastern Europe.
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 have visited the Victoria Gold Project site on May 26, 2016;
6. I am responsible for Section numbers 15 and 16 (except 16.3 and 16.4) of the Technical Report;
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the 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. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Dino Pilotto, P.Eng."

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I, Kelly S. McLeod, P. Eng., do hereby certify that:

1. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Professional Metallurgical Engineer (P.Eng. #15868) registered with the Association of Professional Engineers, Geologists of British Columbia;

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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I did not visited the Eagle Gold Project site;

6. I am responsible for Section 17 (except 17.2.4 and 17.2.6.3 to 17.2.6.6) of this Technical Report;

7. I have had no prior involvement with the property that is the subject of this Technical Report;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Kelly McLeod, P.Eng."

Kelly S. McLeod, P. Eng.

Allan V Moran Consulting LLC
62463 E. Northwood Rd
Tucson, Arizona, U.S.A. 85739
Phone : 520-403-8318
Email : allan@avmc.us

CERTIFICATE of AUTHOR

I, Allan V. Moran, a Registered Geologist and a Certified Professional Geologist, do hereby certify that:

1. I am currently employed as Manager of Allan V Moran Consulting LLC, an independent geological consulting company providing services to the mining and mineral exploration industry, with an office address of 62463 E. Northwood Rd., Tucson, Arizona, USA, 85739.

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Registered Geologist in the State of Oregon, USA, # G-313, and have been since 1978. I am a Certified Professional Geologist through membership in the American Institute of Professional Geologists, CPG - 09565, and have been since 1995.

   I graduated with a Bachelor's of Science Degree in Geological Engineering from the Colorado School of Mines, Golden, Colorado, USA; May 1970.I have been employed as a geologist in the mining and mineral exploration business, continuously, for the past 45 years, since my graduation from university. 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. The Technical Report is based upon my personal review of the information provided by the issuer. My relevant experience for the purpose of the Technical Report is:
   - Principal Consultant - Geology, SRK Consulting (U.S.) Inc., 2005-2013
   - Manager, Exploration North America for Cameco Gold Inc., 1998-2002
   - Vice President and U.S. Exploration Manager for Independence Mining Company, Reno, Nevada, 1990-1993
   - Exploration Geologist for Freeport McMoRan Gold, 1980-1988
   - Experience in the above positions working with and reviewing resource estimation methodologies, in concert with resource estimation geologist and engineers, on exploration, development, and feasibility level gold projects
   - As a consultant, I completed several NI 43-101 Technical reports, 2003-2014 relating to gold deposits in North and South America.
4. I have visited the Victoria Gold Project site on September 22 and 23, 2011, and all day on May 26, 2016;

5. I am responsible for Section numbers 7,8,9,10, 11, and 12 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had prior involvement with the property that is the subject of the Technical Report; including responsibility for geology, geological modeling, and inputs to the Mineral Resource

Estimate included in the 2012 Feasibility Study completed for the Project by Wardrop; and geological modeling for unpublished interim resource estimations of the Eagle Zone in 2013, and the Olive Zone in 2015.

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. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective date: October 26, 2016.
Signing Date: October 26, 2016

(original signed and sealed) "Allan V. Moran, CPG."

Allan V. Moran, CPG

Kappes, Cassiday & Associates
7950 Security Circle Reno, Nevada 89506
Telephone: (775) 972-7575 FAX: (775) 972-4567

CERTIFICATE OF AUTHOR

I, Carl E. Defilippi, M.Sc., C.E.M., do hereby certify that

1. I am currently employed as Senior Engineer for Kappes, Cassiday & Associates located at 7950 Security Circle, Reno, Nevada 89506;

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I graduated with a Bachelor of Science degree in Chemical Engineering from the University of Nevada in 1978 and a Master of Science degree in Metallurgical Engineering from the University of Nevada in 1981. I am a Registered Member of the Society for Mining, Metallurgy and Exploration (775870 RM) and have worked as a Metallurgical Engineer for 35 years;

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 have not visited the Eagle Gold property;

6. I am responsible for Section 13;

7. I am independent of Victoria Gold Corporation and related companies applying all of the tests in section 1.5 of National Instrument 43-101;

8. I participated in previous Pre-feasibility and Feasibility Studies on Eagle Gold from 2009 to 2015. Other than that those studies, I have had no prior involvement with the Eagle Gold Project;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the part of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.

Effective Date: September 12, 2016
Signing Date: October 26, 2016

original signed and sealed

Carl E. Defilippi

Ravindra Kumar Sharma
BMRC
Level 41, Emirates Towers
Phone : +971 509772510
Email : ravi@bmrc-emirates.com

CERTIFICATE of AUTHOR

I, Ravindra Kumar Sharma, a registered member of SME and MAusIMM CP, do hereby certify that:

1. I am currently employed as Managing Director and Principal Consultant with Bedrock Mineral Resource Consulting, an independent geological consulting company providing services to the mining and mineral exploration industry, with an office address of Level 41, Emirates Towers, Dubai, UAE.

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Geologist and Registered member of SME (4042817) member since 2009, and CP member of AusIMM (991544) since 2007.

   I did post-graduation with a Master's of Science Degree (Msc) in Geology from the Lucknow University, India; September 1989. I have been employed as geologist in mining and mineral exploration industry continuously for the past 27 years after my post-graduation from university in 1989. 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. The Technical Report is based upon my personal review of the information provided by the issuer. My relevant experience for the purpose of the Technical Report is:

   - Geologist and Dy Manager- Resource Geology ACC Ltd, India 1992-2001.
   - Chief geologist, Vedanta Resources Zod Gold Mines, Armenia 2001-2005.
   - Chief Geologist, Bulyanhulu Gold Mines, Barrick Gold Tanzania, 2005-2007.
   - Manager Resource, Tournigan Energy ltd (formerly Tournigan Gold), Denver, Colorado, USA, June 2007- September-2011.
   - Associate Principal Consultant (Resource Geology), SRK, US mining group, October 2011- 2013.
   - Managing Director and Principal Consultant- Geology with BMRC since 2011.
   - Experience in the above positions working with mineral resource estimation, experience on exploration, development, and feasibility level gold projects

4. I am responsible for Section numbers 14.1-14.11 and 14.23 of the Technical Report;

5. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have had prior involvement with the property that is the subject of the Technical Report; including responsibility for geological modeling and unpublished interim resource estimations of the Eagle Zone in 2013.

7. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

8. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Ravindra Kumar Sharma*, MAusIMM(CP), RM-SME*."

(Ravindra Kumar Sharma)

SME No – 4042817

AusIMM -991544

Frank Daviess
1549 Genesee Vista Rd
Golden, Co 80401
Phone : 303.906.6362
Email : fdaviess@comcast.net

CERTIFICATE of AUTHOR

I, Frank Daviess, Registered SME, do hereby certify that:

1. I am currently as an independent geologist providing services to the mining and mineral exploration industry, with an office address of 1549 Genesee Vista Rd, Golden Colorado.

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Registered member of the SME. 742250

   I have worked as a geologist for a total of 40 years since my graduation from university and I have specialized in the estimation, assessment and evaluation of mineral resources since 1975. I am qualified as a competent person for the resource estimation of many commodities under the JORC/CIM guidelines.

   I have not visited the Victoria Gold Project site.

4. I am responsible for Section numbers 14.14 through 14.23 of the Technical Report;

5. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have had no prior involvement with the property that is the subject of the Technical Report.

7. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

8. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Frank Daviess RM SME."

Frank Daviess RM SME

CERTIFICATE OF QUALIFIED PERSON

I, W. Neil Brazier, P.Eng., of Richmond, BC, do hereby certify that:

- I am currently a Principal with W.N. Brazier Associates Inc. with a business address at #8–3471 Regina Ave., Richmond, BC.
- I am a graduate of the University of Saskatchewan (B.Sc. Electrical Engineering, 1969) and I have practiced my profession continuously since graduation.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#8337). I am a member in good standing of Association of Professional Engineers of Yukon #1931 and have a Permit to Practice # PP295.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer").
- My relevant experience includes design engineering, estimating, construction supervision, and commissioning of a large number of diesel and combustion turbine power plants, high voltage transmission lines, substations and plant power and control systems for mining applications.
- I have visited the Victoria Gold Project site from May 25 to 27, 2016.
- I am responsible for Sections 18.4.1, 18.4.2, 18.4.3 & 18.4.4, and for capital costs estimates related to these sections.
- I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
- I have had previous involvement with the property that is the subject of the Technical Report, having studied power supply options for Victoria Gold Inc. in 2015.
- I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the date of this certificate, 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.

Effective Date: September 12, 2016

Signing Date: October 26, 2016, 2016 at Richmond, BC

Original signed and sealed by W. Neil Brazier, P. Eng.

CERTIFICATE OF AUTHOR

I, Rui Adanjo, P.Eng., do hereby certify that:

1. I am currently employed as an Electrical Engineer with Allnorth Consultants Limited with an office at Suite 1200-1100 Melville Street, Vancouver, BC, V6E 4A6;

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Professional Engineer (P.Eng.), Association of Professional Engineers and Geoscientists of BC (APEGBC), Professional Engineer (P.Eng.), Association of Professional Engineers of Ontario (PEO), Professional Engineer (P.Eng.).

4. I am a graduate Bachelor of Engineering (B.Eng.), Electrical, from the University of Pretoria, South Africa, and am also hold an Instrument Mechanic Trade Certificate, of Industrial Instrumentation and Process Control, from the , Department of Manpower, South Africa, I have more than 19 years of experience as an Electrical Engineer with four years hands-on experience in the instrumentation field. My technical experience ranges from industrial power, control and instrumentation in a variety of processes which include bulk material handling, mine mills, batch brewing and others. My engineering experience includes operational maintenance, detail design and site commissioning environments. In addition, I am experienced in overall project management and project support in discipline lead roles and discipline assistance with feasibility, conceptual and detail design projects.
   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 have not visited the Victoria Gold Project site.;

6. I am responsible for Section numbers 18.4.5 and 18.4.6 of the Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the 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. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: September 12, 2016

Signing Date: October 26, 2016

(original signed and sealed) "Rui Adanjo, P.Eng."

Rui Adanjo, P.Eng.

CERTIFICATE OF AUTHOR

I, Farhad Riahi, P.Eng. do hereby certify that:

1. I am currently employed as Sr. Structural Engineer with Allnorth Consultants Limited with an office at Suite 1200-1100 Melville Street, Vancouver, BC, V6E 4A6;

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Professional Engineer (P.Eng.), Association of Professional Engineers and Geoscientists of BC (APEGBC), Professional Engineer (P.Eng.), Association of Professional Engineers of Ontario (PEO), Professional Engineer (P.Eng.), Association of Professional Engineers and Geoscientists of AB (APEGA), and Iranian Engineers Organization, Registered Professional.

4. I am a graduate Bachelor of Science (B.A.Sc.), Structural Engineering, Tehran University, Tehran Iran, 1984. Master of Science (M.Sc.), Structural Engineering, Tehran University, Tehran Iran, 1986. I have training in the following: Concrete Structures, 1999, Behaviour of Timber Structures, 1999, Dynamic Structures, 1999, Bridge Design and Construction, 1999 and Reliability and Structural Safety, 1999. I am a Senior Structural Engineer with over 30 years of experience in the mining, forestry and heavy industrial building sector. My areas of expertise include the analysis and design of many light and heavy steel and concrete structures.
   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 have visited the Victoria Gold Project site on May 26, 2016;

6. I am responsible for Section numbers 18.1, 18.2 and 18.3 of the Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the 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. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: September 12, 2016

Signing Date: October 26, 2016

(original signed and sealed) "Farhad Riahi, P.Eng."

Farhad Riahi, P.Eng.

SRK Consulting (U.S.) Inc.
1125 17th Street, Suite 600
Denver, Colorado
80202

denver@srk.com
www.srk.com

Tel: 303.985.1333
Fax: 303.985.9947

CERTIFICATE OF QUALIFIED PERSON

Michael Levy, P.E., P.G.

I, Michael E Levy, P.E., P.G., do hereby certify that:

1. I am a Professional Engineer, employed as a Principal Geotechnical Engineer with SRK Consulting (U.S.), Inc. with an office at Suite 600, 1125 17th Street, Denver, CO, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");
3. I am a registered Professional Engineer in the states of Colorado (#40268), California (#70578) and Arizona (#61372) and a registered Professional Geologist in the state of Wyoming (#3550). I am a current member of the International Society for Rock Mechanics (ISRM) and the American Society of Civil Engineers (ASCE).

   I received a bachelor's degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004. I have practiced my profession continuously since March 1999 and have been involved in a variety of geotechnical projects specializing in advanced analyses and design of soil and rock slopes.
4. I have visited the Eagle Project site on May 26, 2016.
5. I am responsible for preparation of sections 16.3 and 16.4 of the Technical Report.
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the National Instrument 43-101;
7. I have had no prior involvement with the property that is the subject of the Technical Report;
8. 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.
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. As of the effective date of the Technical Report and the date of this certificate, 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.

Effective Date: September 12, 2016
Signing Date: October 26, 2016

"Original Signed and Sealed"

Michael E. Levy, P.E, P.G.

Group Offices:	Canadian Offices:		U.S. Offices:	
Africa	Saskatoon	306.955.4778	Anchorage	907.677.3520
Asia	Sudbury	705.682.3270	Denver	303.985.1333
Australia	Toronto	416.601.1445	Elko	775.753.4151
Europe	Vancouver	604.681.4196	Fort Collins	970.407.8302
North America	Yellowknife	867.445.8670	Reno	775.828.6800
South America			Tucson	520.544.3688

CERTIFICATE OF AUTHOR

I, Troy L Meyer, P. Eng., do hereby certify that:

1. I am currently contracted as Senior Engineer, Geotechnical, with DOWL with an office in Montrose, Colorado;
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");
3. I am a Professional Engineer (P.Eng. #2010) registered with the Association of Professional Engineers of Yukon;
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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
5. I visited the Eagle Gold Project site on Sept 14th through Sept 17th, 2016.
6. I am responsible for Section 17.2.4 and 17.2.6.3, to 17.2.6.6 of this Technical Report;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Troy Meyer, P.Eng."

Troy L Meyer, P. Eng.

CERTIFICATE OF AUTHOR

I, Jay Collins, P.Eng., do hereby certify that:

1. I am currently employed as President of Merit Consultants International (a Division of Cementation Canada Inc.) with an office at Suite 401-750 West Pender Street, Vancouver, BC, V6C 2T8;

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Eagle Gold Project, Yukon Territory, Canada", with an effective date of September 12, 2016, (the "Technical Report") prepared for Victoria Gold Corp. ("the Issuer");

3. I am a Professional Mining Engineer (P.Eng. License #12741) registered in good standing with the Association of Professional Engineers, Geologists of British Columbia.

   I am a graduate of Portsmouth University, UK (Civil/Structural, 1974). My relevant experience is associated with the project and construction management of mining projects around the world. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument"). 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 visited the Victoria Gold Project site on August 22 and 23, 2011;

5. I am responsible for Section numbers 21 and 25 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of the Technical Report other than to contribute to the 2012 Feasibility Study in the same manner as the contribution to this update;

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. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: September 12, 2016
Signing Date: October 26, 2016

(original signed and sealed) "Jay Collins, P.Eng."

Jay Collins, P.Eng.