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NI 43-101 Feasibility Study Technical Report for the Coffee Gold Project, Yukon Territory, Canada

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Prepared for:

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Kaminak Gold Corporation. 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.

Kaminak Gold Corporation 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 Kaminak Gold Corporation (Kaminak) to carry out a 43-101 Feasibility Study (FS) and Technical Report for the Coffee Gold Project (Coffee Gold or Coffee or Project), which is an advanced exploration gold project owned by Kaminak and located in the White Gold District of west-central Yukon, approximately 130 km south of Dawson. The Project contains several gold deposits within an exploration concession covering an area of more than 600 km^2.

The Coffee Gold Project, comprising four open pits called Latte, Double Double, Supremo and Kona, is proposed to be mined by conventional shovel and truck methods at an average rate of five million tonnes per annum (Mt/a) of heap leach feed. The ore will be crushed and placed onto a heap leach facility for nine months of the year. During the three coldest months of winter, run-of-mine (ROM) ore will be stockpiled. Gold will be extracted from pregnant leach solution by a 5 tonnes per day (t/d) Adsorption-Desorption Recovery (ADR) carbon plant with mercury retorting to produce a final gold doré product. A total of 1.9 Moz of gold is planned to be recovered, over an ten-year mine life.

This report presents the results of the FS, in accordance with the Canadian Securities Administrators' National Instrument (NI) 43-101 and Form 43-101F1, collectively NI 43-101, guidelines.

Five previous technical reports were prepared for the Coffee Gold Project documenting exploration work completed by Kaminak on the Coffee Gold Project in 2010, 2011, 2012, 2013 and 2014. The most recent technical report was a Preliminary Economic Assessment (PEA) dated July 8, 2014. All technical reports were filed on SEDAR.

1.2 Property Description and Ownership

The property comprises 3,021 contiguous Yukon Quartz Lease mining claims owned by Kaminak covering an aggregate area of 60,502 ha. Kaminak has a 100% interest in the property, subject to a 2% net smelter royalty (NSR), half of which may be repurchased by Kaminak for $2 million (M).

1.3 Geology and Mineralization

The Coffee Gold Project is located in the Yukon-Tanana Terrane (YTT), an accreted pericratonic rock sequence that covers a large portion of the Omineca Belt in the Yukon and extends into Alaska and British Columbia. The YTT underlies part of the Tintina gold belt and hosts multiple gold deposits, including the Sonora Gulch gold deposit, the Casino copper-gold-molybdenum porphyry, the Boulevard gold prospect, and the Golden Saddle gold deposit. The YTT also hosts volcanogenic massive sulphide (VMS) and Mississippi Valley-type (MVT) deposits.

The Project area is underlain by a package of metamorphosed Paleozoic rocks of the YTT that was intruded by a large granitic body in the Late Cretaceous. The Paleozoic rock package consists of a mafic schistose to gneissic panel which overlies the Sulphur Creek orthogneiss.

Both packages form the southwestern limb of a northwest-trending antiformal fold with limbs dipping shallowly to the northeast and southwest.

Within the schistose and gneissic mafic rock package, a thick panel of biotite (+ feldspar + quartz + muscovite ± carbonate) schist with rare lenses of amphibolite overlies a panel of amphibolite and metagabbro with arc-derived geochemical signatures. Within the schistose panel, 20 m thick slices of serpentinized ultramafic are in tectonic contact with the surrounding rocks. This rock sequence overlies the augen orthogneiss. These rocks are in contact to the southwest with the 98.2 ± 1.3 Ma Coffee Creek granite, a phase of the Dawson Range batholith. Both the Paleozoic metamorphic rocks and Cretaceous granite are cut by intermediate to felsic dykes (andesite and dacite).

Exploration drilling has led to the discovery of gold mineralization in 15 separate areas of the Coffee Gold Project: Supremo, Sumatra, Latte, Double Double, Arabica, Americano West, Americano, Espresso, Kona, Kona North, Macchiato, Cappuccino, Dolce, French Press and Sugar. Gold mineralization occurs in narrow to broad gold-bearing locally brecciated structures with quartz, dolomite, sericite, and pyrite alteration. The host rock varies between augen gneiss, granite, and biotite-feldspar schist.

The gold mineralization found to date is hydrothermal in origin and both structurally and lithologically controlled. Mineralization is associated with both polyphase brecciation and intense sulphidation of mica-rich host rocks by a CO_2-S-As-Sb-Au fluid resulting in the formation of gold-bearing arsenian pyrite. Micron-scale gold is found within arsenian pyrite and is associated with As-rich growth bands. Oxidation of the arsenian pyrite along rims and cracks results in the release of micron-scale free gold. Mineralization is constrained by sulphidized biotite within the Coffee Creek granite to <98 Ma.

1.4 History, Exploration and Drilling

The Coffee property was acquired in 2009. Exploration work has been conducted on the property each year since acquisition. The 2009 field program consisted of surficial trenching and soil sampling in the main deposit area surrounding the initial gold-in-soil anomaly. Diamond drilling was initiated in 2010 and led to the discovery of the Supremo zone in the first drill hole (17.1 g/t Au over 15.5 m; CFD0001). Latte, Double Double, Kona, and Americano were also discovered during the 2010 drilling program. The 2011 and 2012 field seasons consisted of infill and exploration drilling as well as initial metallurgical testwork which led to Kaminak's first estimate of mineral resources in January 2013. Continued infill drilling and metallurgical testwork during the 2013 field season facilitated an updated mineral resource statement and the release of a Preliminary Economic Assessment (PEA) in July 2014. The positive PEA led to an exploration program to support the Feasibility Study during the latter half of the 2014 field season and the beginning of the 2015 field season. An annual summary of the exploration programs are presented in Table 1.1.

Table 1.1: Summary of Annual Exploration Programs

Year	Drill Holes	Metres Drilled	Soil Samples	Trenching (m)	Mapping and Sampling (days)	Geophysics
2009	N/A	N/A	3,876	4,164	10	261 line-km ground magnetic survey
2010	76	16,103	8,851	4,470	10	579 line-km ground magnetic survey
2011	246	48,042	10,689	3,926	15	4,842 line-km airborne magnetic and gamma-ray spectrometric; 15.9 line-km HLEM and Ohm mapper surveys
2012	335	65,548	4,438	N/A	40	N/A
2013	302	55,478	5,027	153	2	18 days of Induced Polarization
2014	353	52,760	2,955	6,252	30	5,300 line-km airborne magnetic infill survey
2015	370	41,895	N/A	N/A	30	N/A
Total	**1,684**	**279,826**	**36,081**	**18,965**	**137**	**N/A**

Source: Kaminak 2015

The purpose of the 2014 and 2015 drilling programs was to upgrade the main Project resource area from the Inferred to Indicated Resource category and collect detailed hydrogeological and geotechnical data. Resource delineation drilling focused on the "in-pit" resources defined in the 2014 PEA at Supremo, Latte, Double Double, Kona, and Sumatra zones. Hydrogeological, metallurgical, geotechnical, and condemnation drilling programs were conducted property-wide. In addition, exploration programs were conducted at the Supremo, Latte, Kona North, Cappuccino, Macchiato, Dolce, and French Press zones.

All samples were analyzed by ALS Vancouver after sample preparation at ALS Whitehorse and in-field portable XRF analysis by Kaminak geo-technicians. Each sample was assayed for gold using conventional fire assay procedures on 30-gram charges. Samples from exploration targets were additionally analyzed for a suite of trace elements using aqua regia digestion. Samples reporting greater than 0.3 grams per tonne (g/t) Au were subsequently analyzed by cold cyanide shake test to measure the cyanide soluble gold content.

1.5 Mineral Processing and Metallurgical Testing

Metallurgical testing for the Coffee deposits began in 2011 and continued through 2015. Initial testing was conducted by Inspectorate Exploration & Mining Services Ltd. ("Inspectorate") of Richmond, British Columbia. Since 2013, almost all metallurgical testing for flowsheet development has been performed by Kappes Cassiday and Associates (KCA) of Reno, Nevada. In 2015, comminution test work was completed by ALS Metallurgy (ALS) of Kamloops, British Columbia.

Ore samples for metallurgical testing have been taken from both bulk surface samples and drill core composites. Testing included column leaching, bottle roll leaching, flotation, column percolation and drain down, multi-element head assay analyses, column leach head and tail assay screen analyses, ball mill work indices, crushing impact, and abrasion indices.

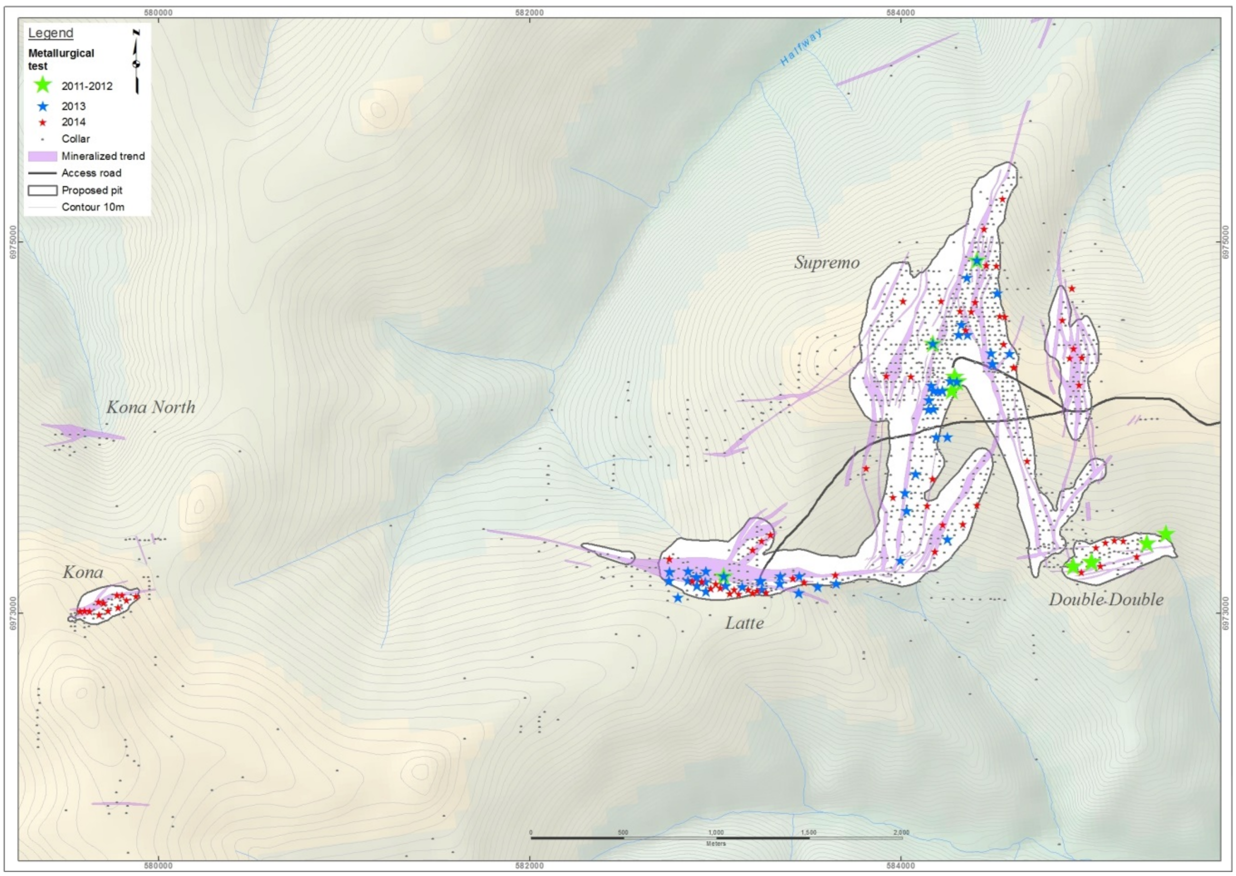
In addition to fire assays for gold and silver, analyses for carbon, sulphur, mercury and copper, semi-quantitative analyses for a series of individual elements and whole rock constituents were conducted. It is noted that no problematic elements were encountered during the head analyses.

Silver concentrations are low and any possible silver production will not be economically significant. Silver analyses are available but are not considered further in this report.

The drill core sample collection and compositing was developed to construct samples from multiple drill holes and intervals to fully represent the known areas of the different deposits and facies.

Locations of the ore samples collected are presented in Figure 1.1.

Figure 1.1: Metallurgical Sample Locations



Source: Kaminak 2016

Four Bond low impact crusher tests were conducted on the Kona, Double Double, Supremo and Latte composites. The highest value achieved was 11.5 kWh/t from the Latte composite. This is considered soft with respect to impact breakage. Bond abrasion index (AI) tests were conducted on eight samples from all mine areas and the values ranged from 0.029 to 0.097. Samples with an AI of 0.1 or less may be considered mildly abrasive. Bond Rod Mill and Ball Mill Work indices were 12.73 and 15.06 kWh/t respectively for a Latte sulphide composite sample.

Major conclusions from the test program include:

- Coffee ores generally leach very rapidly with low reagent consumption;
- Ore agglomeration is not required;
- A trade-off study to evaluate the incremental economics of crushing to a size of 80% passing a 150 mm screen size (P_{80} of 150 mm) as compared to a P_{80} of 50 mm determined the P_{80} of 50 mm to be the optimal option; and
- Cyanide soluble assays from over 14,000 samples confirmed that cyanide soluble recovery is a reliable method to map gold recovery in all known ore zones of the Coffee Gold Project deposit.

To project actual plant operating conditions, laboratory test work gold recoveries were reduced 3% and sodium cyanide consumptions were factored by 25%. However, for the Kona Middle and Lower Transition samples (which represent less than 0.05% of the mineable reserve and where there were not enough drill intervals to assemble a composite), recoveries from the cyanide soluble test work were discounted by 11% and 12% respectively. The gold recoveries and cyanide consumptions at the P_{80} crush size of 50 mm are summarized in Table 1.2.

Table 1.2: Gold Recovery, Cyanide Consumption and Lime Addition

Description	Average Laboratory Results			Application to Feasibility Study		
	Gold Recovery (%)	Reagent Consumption		Gold Recovery (%)	Reagent Consumption	
		NaCN kg/t	Lime kg/t		NaCN kg/t	Lime kg/t
Latte Oxide	93	0.6	1.27	90	0.2	1.5
Latte Upper Transition	82	0.79	1.49	79	0.2	1.5
Latte Middle Transition	63	0.76	1.52	60	0.2	1.5
Latte Lower Transition	29	0.64	1.51	26	0.2	1.5
Supremo Oxide	93	0.73	1.54	90	0.2	1.5
Supremo Upper Transition	85	0.61	1.5	82	0.2	1.5
Supremo Middle Transition	60[1]			57	0.2	1.5
Supremo Lower Transition	30[2]			27	0.2	1.5
Kona Oxide	88	0.82	1.56	85	0.2	1.5
Kona Upper Transition	72	0.76	1.53	69	0.2	1.5
Kona Middle Transition	60[3]			49	0.2	1.5
Kona Lower Transition	30[4]			18	0.2	1.5
Double Double Oxide	95	0.87	1.56	92	0.2	1.5
Double Double Upper Transition	80[5]			77	0.2	1.5
Double Double Middle Transition	60[6]			57	0.2	1.5
Double Double Lower Transition	30[7]			27	0.2	1.5

Notes:
1 Cyanide Soluble assays used for recovery - less than 1.2% of mineable reserves
2 Cyanide Soluble assays used for recovery - less than 0.5% of mineable reserves
3 Cyanide Soluble assays used for recovery - less than 0.04% of mineable reserves
4 Cyanide Soluble assays used for recovery - less than 0.01% of mineable reserves
5 Cyanide Soluble assays used for recovery - less than 0.9% of mineable reserves
6 Cyanide Soluble assays used for recovery - less than 0.5% of mineable reserves
7 Cyanide Soluble assays used for recovery - less than 0.1% of mineable reserves

Source: Kaminak 2015

The leach profile shows an initial rapid recovery of gold, followed by a slower leaching period to achieve ultimate gold extraction. From KCA test results, a standardized leach profile for all Project ore types were developed based on the solution : solids ratio. The actual period of leaching was adjusted to fit the solution : solids ratio and will be considerably longer than indicated by laboratory columns.

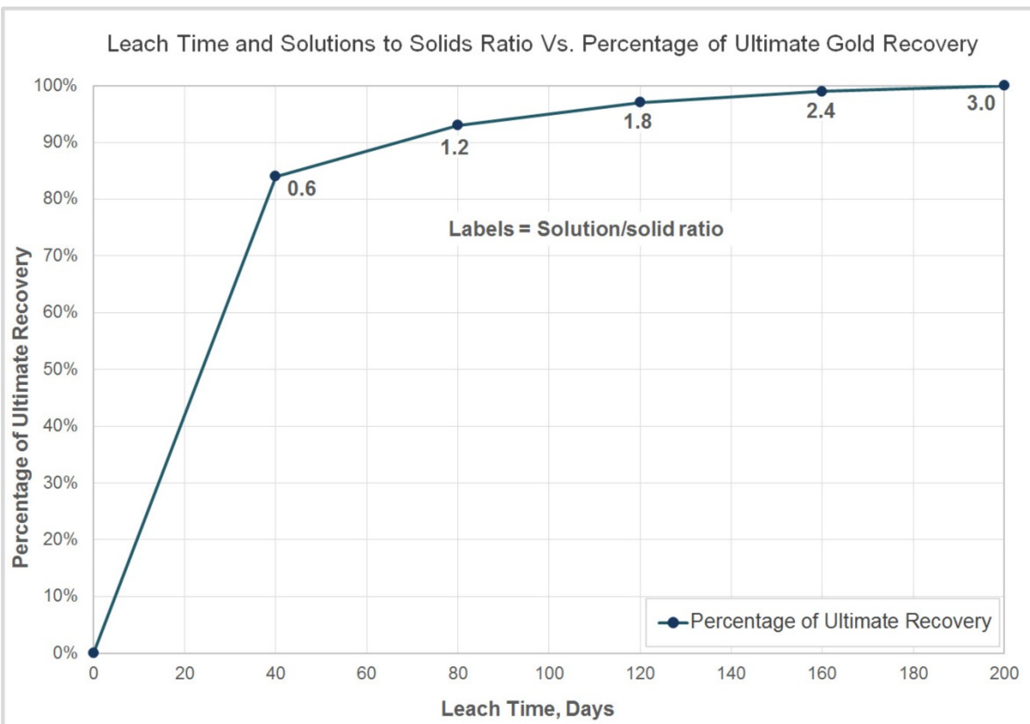
Table 1.3 presents the data for the leach recovery curve for all ores and Figure 1.2 illustrates the leach recovery cycle graphically. Recoveries presented are a percentage of the ultimate recoveries achieved in Table 1.3.

Table 1.3: Leach Recovery Curve for Coffee Ores

Solution to Solids Ratio	Days of Leaching	% of Ultimate Recovery
0	0	0
0.6	40	84
1.2	80	93
1.8	120	97
2.4	160	99
3	200	100

Source: Kaminak 2015

Figure 1.2: Leach Time and Solutions to Solids Ratio vs. Percentage of Ultimate Gold Recovery



Source: Kaminak 2015

1.6 Mineral Resource Estimate

The mineral resource estimate for the Coffee Gold Project was updated sequentially following the completion of drilling on the various deposits. The effective date of the models at Latte, Double Double and Kona is March 15, 2015 and for Supremo it is September 22, 2015. The resource block models were developed using a geostatistical approach constrained by gold mineralization wireframes.

The model considers information from 1,681 core and reverse circulation boreholes drilled by Kaminak from 2010 to 2015 (280,000 m). Four individual block models were constructed using MineSight® (v10.0-2) with limits determined on the basis of the local UTM coordinated system (Nad83 datum, zone 7). The block size in all models measure 10 m long by 2.5 m wide by 5 m high, with the long axes aligned parallel to the strike of the gold mineralization.

The boundaries of the gold mineralization were interpreted by Kaminak geologists from drilling data on vertical sections spaced at 25 to 50 m intervals. These were linked into a series of 3D domains that control the distribution and extent of gold mineralization in the resource model. Variable length borehole assay sample data were composited at 1 m lengths for geostatistical analysis and grade estimation. Potential outlier samples were examined using probability plots and a combination of capping and volume restriction was applied to these high-grade composites to restrict their influence during block grade estimation. Gold grades were estimated in model blocks using ordinary kriging and the results validated by visual inspection and statistical evaluation.

The extent and intensity of oxidation was interpreted using a combination of qualitative data collected during drill core and chip logging plus the solubility characteristics derived from a suite of samples tested for cyanide gold solubility. Five oxide types or domains have been interpreted which, in general, represent decreasing intensity of oxidation with depth below surface.

Although an extensive bulk density database was generated for the Coffee Gold Project, the lack of density measurements derived from reverse circulation drill holes did not provide sufficient data coverage in some areas to allow for direct interpolation of densities in the resource models. A general relationship between density and the intensity of oxidation is evident and, as an alternative, average density values have been used to calculate resource tonnages within each of the five oxide type domains.

Resources were classified on the basis of confidence in the geological continuity and distance from informing sample data. Block model quantities and grade estimates were classified according to the Canadian Institute of Mining (CIM) “Definition Standards for Mineral Resources and Mineral Reserves (May 2014)”. Blocks in the Indicated Resource category are delineated from multiple drill holes located on a nominal 25 m (between sections) pattern with alternating sections that have 50 m and 25 m spaced holes. In general, this is equivalent to estimating block grades using three or more drill holes within a maximum average distance of 25 to 30 m. Resources are included in the Inferred Resource category if they occur within a maximum distance of 50 m from a drill hole and exhibit a reasonable degree of geologic continuity.

The Coffee Gold Project deposits form relatively continuous, sub-vertical zones of gold mineralization extending from the surface to depths (locally) of more than 200 m. The “reasonable prospects for eventual economic extraction” were tested using a series of floating cone pit shells based on projected technical and economic assumptions. This evaluation tests the estimated block grades on a full block basis and includes adjustments for projected gold recoveries.

The depth extents of the generated pits were compared to the proportion of resources located within incremental depths of 50 m below surface. The results show that essentially all Indicated Resources and the majority of the Inferred Resources occur at depths where it is considered reasonable that present and future economic conditions would support open pit extraction. As a result, the mineral resources for the Coffee Gold Project are not constrained within a pit shell or a maximum depth below surface because, in the author's opinion, any or all of the Coffee Gold Project resource shows reasonable prospects for eventual economic extraction.

The Mineral Resource Statement for the Coffee Gold Project is presented in Table 1.4. Mineral resources are stated at cut-off thresholds that reflect the projected metallurgical characteristics at various degrees of oxidation; 0.3 g/t Au for Oxide and Upper Transition, 0.4g/t Au for Middle Transition and at 1 g/t Au for Lower Transition and Sulphide types.

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect the extraction of the mineral resource.

Table 1.4: Estimate of Mineral Resources for the Coffee Gold Project*

Area	Oxide			Upper Transition			Middle Transition			Oxide Upper & Middle Transitions			Lower Transition			Sulphide		
	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)
INDICATED																		
Supremo	37,032	1.45	1,726	3,999	1.48	190	1,147	1.675	62	42,178	1.459	1,978	431	2.262	31	24	1.855	1
Latte	8,021	1.31	338	3,913	1.139	143	2,937	1.275	120	14,871	1.258	601	1,675	1.856	100	105	2.158	7
Double Double	776	2.327	58	797	2.837	73	402	2.32	30	1,976	2.531	161	105	2.343	8	3	2.567	0
Kona	1,298	1.144	48	751	0.926	22	161	0.935	5	2,210	1.055	75	83	1.497	4	6	1.604	0
COMBINED	47,127	1.432	2,170	9,461	1.409	429	4,647	1.452	217	61,235	1.43	2,815	2,294	1.941	143	138	2.091	9
INFERRED																		
Supremo	20,775	1.087	726	4,642	1.277	191	2,506	1.797	145	27,922	1.182	1,061	1,352	2.193	95	103	2.032	7
Latte	3,435	0.91	101	4,020	0.999	129	2,951	1.243	118	10,407	1.039	348	3,964	1.828	233	2,456	1.734	137
Double. Double	529	1.135	19	1,200	1.361	53	766	2.007	49	2,496	1.512	121	138	1.684	7	107	1.55	5
Kona	146	0.883	4	373	1.024	12	337	1.13	12	856	1.042	29	940	1.738	53	55	1.962	3
Kona North	135	1.511	7	411	1.795	24	295	1.663	16	841	1.703	46	409	2.743	36	310	3.016	30
COMBINED	**25,020**	**1.065**	**857**	**10,646**	**1.192**	**408**	**6,855**	**1.543**	**340**	**42,520**	**1.174**	**1,605**	**6,803**	**1.94**	**424**	**3,030**	**1.872**	**182**

Note: Oxide and Upper Transition 0.3 g/t Au cut-off, Middle Transition 0.4 g/t Au cut-off, Lower Transition and Sulphide 1.0 g/t Au cut-off.

***This Mineral Resource Estimate includes Mineral Reserves**

Source: SIM Geological 2015

1.7 Mineral Reserve Estimate

The mineral reserve for the property is based on the mineral resource estimate completed by SIM Geological Inc. with an effective date of March 15, 2015 for Latte, Double Double and Kona and September 22, 2015 for Supremo..

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 Probable mineral reserves total 46.4 Mt at 1.45 g/t Au, containing 2.16 Moz Au (Table 1.5).

Table 1.5: Mineral Reserve Estimate for the Coffee Gold Project

Area	Classification	Tonnes (Mt)	Au (g/t)	Contained Au (koz)
Total Open Pit	Probable	46.4	1.45	2,157

Notes:
A gold price of US$1,200/oz is assumed.
A US$: C$ exchange rate of 0.87.
Dilution and recovery factors are applied as per open pit mining method.

Both mineral resource and 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.

1.8 Mining

The Coffee Gold Project deposit is amenable to development as an open pit mine. Mining of the deposit is planned to produce a total of 46.4 Mt of heap leach feed and 265 Mt of waste (at a 5.7:1 overall strip ratio) over a nine and a half year mine production life (including one year of pre-production). The current life of mine (LOM) plan focuses on achieving consistent heap leach production rates, and mining of higher value material early in the production schedule, as well as balancing grade and strip ratios.

1.8.1 Open Pit Mine Plan and Phasing

Figure 1.3 illustrates the proposed overall site layout for the Project, including the open pit, waste rock facilities, heap leach facilities, and proposed plant site locations.

The mine design process for the deposit commenced with the development of open pit optimization input parameters. These parameters included estimates of metal price, mining dilution, process recovery, off-site refining costs, geotechnical constraints (slope angles) and royalties (see Table 1.6).

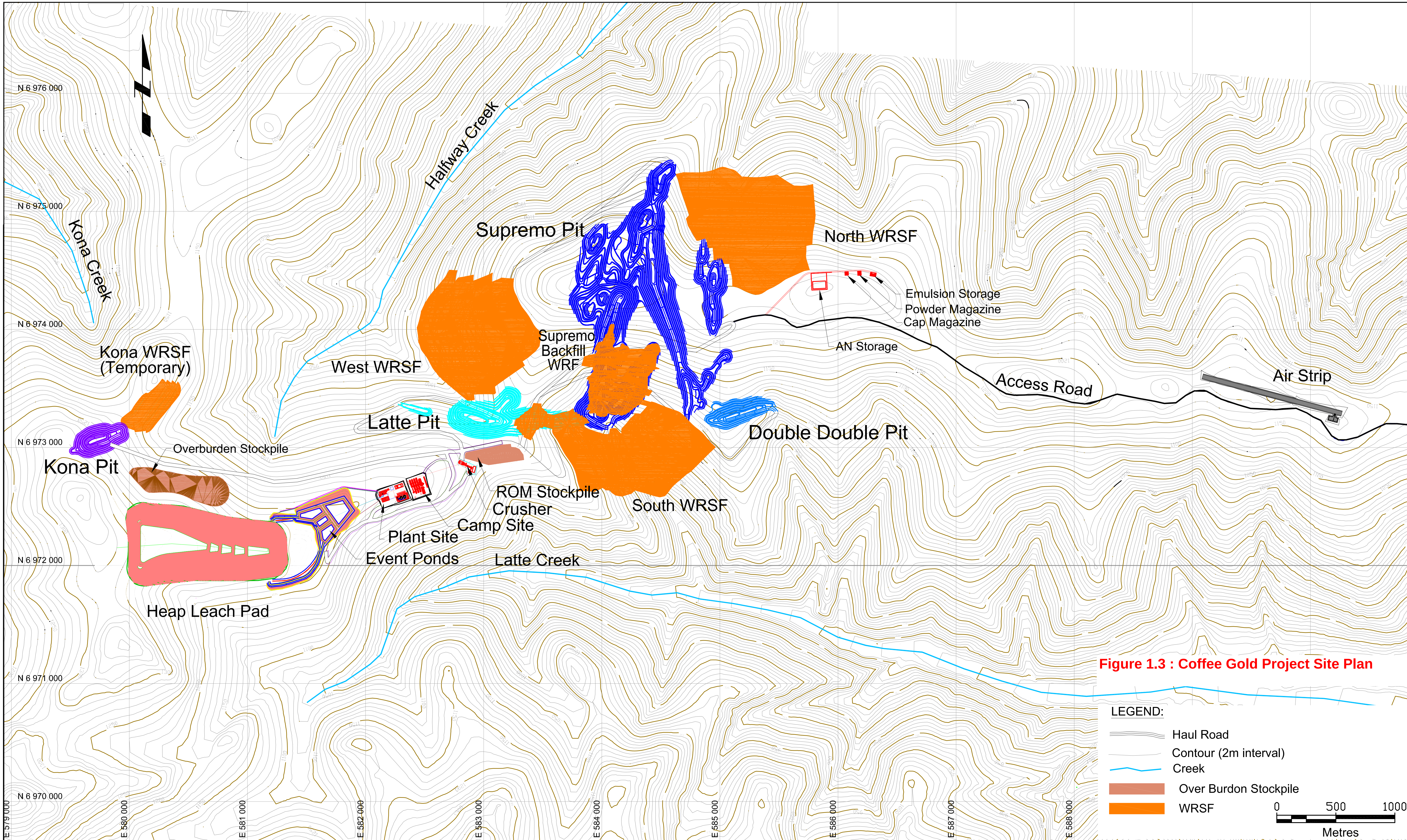


Figure 1.3 : Coffee Gold Project Site Plan

Table 1.6: Mine Planning Optimization Input Parameters*

Parameter	Unit	2016 FS Value
Revenue, Smelting & Refining		
Gold price	US$/oz Au	1,200
Exchange Rate	C$:US$	0.87
Payable metal	%	100
TC/RC/Transport	C$/oz Au	8.62
Royalty @ 1% NSR	C$/oz Au	13.79
Net gold value per ounce	C$/oz	1,357
Net gold value per gram	C$/g	43.63
OPEX Estimates		
OP Waste Mining Cost	C$/t waste mined	2.25
OP Ore Mining Cost	C$/t ore mined	2.25
Strip Ratio (estimated)	W:O	4
Open Pit Mining Cost	C$/t milled	11.25
Process Cost		
Leach Cost for all Ore Types and Deposits	C$/t leached	6.00
General & Administrative (G&A)	C$/t leached	4.00
Total OPEX Cost (excluding Mining) for all Ore Types and Deposits	**C$/t leached**	10.00
Recovery and Dilution		
External Mining Dilution – Supremo	%	7
External Mining Dilution – Double Double	%	9
External Mining Dilution – Kona	%	10
External Mining Dilution – Latte	%	5
Mining Recovery	%	95
Gold Leach Recovery		
Supremo Oxide	%	90
Double Double Oxide	%	92
Kona Oxide	%	85
Latte Oxide	%	90
Supremo Upper Transition	%	82
Double Double Upper Transition	%	77
Kona Upper Transition	%	69
Latte Upper Transition	%	79
Supremo Middle Transition	%	57
Double Double Middle Transition	%	57
Kona Middle Transition	%	49

Parameter	Unit	2016 FS Value
Latte Middle Transition	%	60
Supremo Lower Transition	%	27
Double Double Lower Transition	%	27
Kona Lower Transition	%	18
Latte Lower Transition	%	27
Cut-off Gold Grade Calculations		
Supremo Oxide	g/t Au	0.27
Double Double Oxide	g/t Au	0.27
Kona Oxide	g/t Au	0.3
Latte Oxide	g/t Au	0.27
Supremo Upper Transition	g/t Au	0.3
Double Double Upper Transition	g/t Au	0.32
Kona Upper Transition	g/t Au	0.37
Latte Upper Transition	g/t Au	0.3
Supremo Middle Transition	g/t Au	0.43
Double Double Middle Transition	g/t Au	0.44
Kona Middle Transition	g/t Au	0.51
Latte Middle Transition	g/t Au	0.4
Supremo Lower Transition	g/t Au	0.91
Double Double Lower Transition	g/t Au	0.93
Kona Lower Transition	g/t Au	1.4
Latte Lower Transition	g/t Au	0.89
Other		
Overall Pit Slope Angles	degrees	35-50
Heap Leach Production Rate	Mt/a	5

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.

Pit optimizations and analyses were conducted to determine the optimal mining shells. Detailed pit and phase designs were then generated for the Supremo, Latte, Double Double and Kona deposits and mine planning and scheduling was conducted on these detailed designs. The LOM plan proposed mineral reserves (all oxidation types) for the deposit are presented in Table 1.7.

Table 1.7: Proposed Mining Plan

Description	Unit	Value
Mine production life	yr	10
Heap leach diluted ore feed	Mt	46.4
Diluted gold grade (head grade)	g/t	1.45
Contained gold	koz	2,157
Waste	Mt	265.4
Total mined material	Mt	311.7
Strip ratio	t:t	5.7

Source: JDS 2016

1.8.2 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 grade, reduce pre-stripping requirements in the early years, and maintain the heap leach at full production capacity. The Project deposits are most economical when certain open pit phases are mined concurrently.

The open pits are projected to provide the heap leach facility feed at a nominal rate of 5.0 Mt/a for a period of ten and a half years (including the initial pre-production period). The open pit mining is envisioned to be undertaken by Kaminak, the project owner. Annual mine production of ore and waste is profiled to peak at 39.8 Mt/a, with a LOM waste to ore stripping ratio of 5.7:1. Given that the crusher and heap leach facility will only be operated between April and December of each year, a ROM stockpile will be used when necessary for stockpiling of ore from the open pit. The handling of the fine ore from the crusher to the heap leach pad is included in the open pit scheduling and operating cost estimation. Table 1.8 summarizes the LOM material movement by year for both the mine and the heap leach facility.

Mining will begin at the Coffee Gold Project in Year -1 at Latte and Double Double pits to provide waste rock for construction and enable the stockpiling of high-grade ore prior to the start of leach processing. Leachate processing will commence in Q3 Year -1 and ramps up rapidly in Q4 of Year -1. Open pit mining will then transition sequentially to the Kona and Supremo pits. Open pit mining and loading of the heap leach facility will be completed in the first half of Year 9.

Open pit mining operations will use a fleet comprising 16 m^3 shovels, 12 m^3 front-end loaders, 4 m^3 excavators, and 144 t haul trucks. This fleet will be supplemented by drills, graders, and track and rubber-tire dozers. A 5 m bench height was selected for mining in ore and waste with overall 20 m effective bench heights based on a quadruple-bench configuration.

Table 1.8: Summary of LOM Production Schedule by Year

Description	Unit	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Total Ore	kt	46,356	3,579	4,923	6,563	4,463	4,356	4,647	5,232	5,056	6,230	1,308
Total Mined grade	g/t	1.45	1.5	1.61	1.38	1.54	1.7	1.57	1.29	1.28	1.3	1.42
Total Au contain	koz	2,157	172	255	292	220	239	235	217	207	260	60
Total Waste	kt	265,361	15,125	17,601	28,036	30,800	35,163	35,015	33,828	34,730	29,715	5,349
Strip Ratio	t:t	5.7	4.2	3.6	4.3	6.9	8.1	7.5	6.5	6.9	4.8	4.1
Total Material	kt	311,717	18,705	22,524	34,598	35,263	39,519	39,662	39,060	39,785	35,945	6,656
Average Mined	t/day		51,245	61,708	94,790	96,610	108,272	108,662	107,014	109,001	98,480	24,653
Total Heap Leach (HL) feed	kt	46,356	3,500	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	2,856
HL Head grade	g/t	1.45	1.5	1.61	1.39	1.52	1.65	1.57	1.28	1.3	1.3	1.32
Au contain ounces	koz	2,157	169	258	223	244	266	252	206	209	209	121
Au recovery	%	86.3	86.6	82.0	81.4	86.1	86.2	86.8	87.8	89.6	89.0	89.1

Source: JDS 2016

1.8.3 Waste Management

Over the LOM, a total of 265 Mt of waste rock will be produced. Most waste rock from the open pits is planned to be deposited in various engineered waste rock facilities near to the pits from which the waste is sourced. Some waste rock will be backfilled into mined out pits at Latte, Supremo and Double Double in order to create causeways and facilitate ore haulage routes to the crusher. Geochemical characterization (based on static and humidity cell testing, acid-base accounting, and trace element analyses) indicates that the exposed pit walls at the Kona pit are potentially acid generating. As such, the waste rock from the Kona pit will be stored in a temporary waste rock facility adjacent to the pit during mining and then backfilled into the mined out pit.

1.9 Recovery Methods

The process flowsheet includes a two-stage crushing plant followed by a heap leach operation. Gold is extracted by an ADR carbon plant. The process flowsheet and design criteria are based on a heap leach processing rate of 5.0 million dry t/a at an average feed grade of 1.45 g/t and an anticipated overall recovery of 86.3%. The process plant will be located near the heap leach facility to minimize pumping and pipeline requirements for pregnant and barren solutions.

1.9.1 Crushing and Ore Handling

Run-of-mine ore with an approximate top size of 750 mm will be trucked from the pits and normally dumped directly into the primary jaw crusher at a rate of about 18,000 t/d. A ROM stockpile area with a capacity of approximately 1.5 Mt will allow the stockpiling of ore when the crusher is not running, particularly during the winter months of January through March. The jaw and secondary cone crusher will discharge feed onto the secondary crusher screen. The oversize will return to the secondary cone crusher and the undersize is conveyed to the crushed-ore stockpile. The final target product size is a P_{80} of 50 mm. Lime will be added to the stockpile feed conveyor for pH control on the heap leach.

Crushed ore will be reclaimed by a front-end loader from a 3,000 t crushed ore stockpile and a fleet of haul trucks will transport the material to the heap leach facility. The crushing plant and haul trucks will operate 275 days per year with no heap leach loading during the coldest period of the year.

1.9.2 Heap Leach Facility

The heap leach facility consists of a conventional, multi-lift, free-draining ridge-top leach pad, ponds, access roads, and leachate solution distribution and collection piping. Ore will be stacked with trucks on a lined leach pad in nominal 10 m thick lifts. Barren solution will be irrigated onto the heap using drip irrigation. Pregnant (gold-bearing) solution will be collected at the base of the heap leach pad by impermeable membranes and piping. The pregnant solution will flow to the process plant by gravity for gold recovery. The heap is stacked to the ultimate design height of about 80 m (vertically over the leach pad).

The leach pad will be constructed on a graded area along the ridgeline to the west of the process plant and pits as shown in the site general arrangement, Figure 1.3. The leach pad will be constructed in stages, with each stage large enough to provide ore capacity for one and a half to three years of operation. The pad will be lined with two liners: a geosynthetic clay liner (GCL) at the base directly overlain by an impermeable collection geomembrane. A network of drainage pipes within a 500-mm thick layer of permeable gravel at the base of the pad will collect and direct the pregnant solution into trunk lines on each flank of the pad and transport it by gravity to the process plant. A series of horizontal trenches or wick drains will be installed beneath the liner system to detect leakage.

Process solution (barren, pregnant and heap rinse water) will be stored in tanks located at the plant. The barren solution will be heated when necessary to ensure that the thermal integrity of the system and the leach pad is maintained. Ponds adjacent to the heap leach pad will be used to store contact and clean water generated from seasonal and storm events, heap upset conditions (e.g., power loss), and normal precipitation runoff. Ponds will generally not be used in the winter except in the event of an upset condition. Two lined events ponds will be built prior to the commencement of operations. An additional lined events pond will be built in or before Year 6 to accommodate the expanded extent of the heap leach. A lined rain water pond will be built in Year 3 to collect clean water to provide make-up water to the process circuit.

Design criteria for the heap leach facility are as follows:

- Nominal design capacity of 47 Mt with expansion capacity to at least 61 Mt;
- The heap leach pad construction will be phased to optimize capital expenditure. Stage 1 will have a capacity of 7.2 Mt and will be constructed in Year -1. Stage 2 and beyond will have incremental capacities of 8 to 20 Mt each (1.5 to 4 years of operating capacity);
- The heap leach pad is designed with a base composite liner system and graded to promote free-draining of the pregnant solution to the flanks of the leach pad and then via a pipeline to the plant;
- A number of mitigations to the extreme climate are incorporated to ensure that the heap leach pad does not freeze:
  - Loading a minimum of 3.5 Mt of ore to the pad, with 30 m of ore depth, in Year -1 in order to maintain thermal integrity;
  - No ore crushing or heap stacking during January through March of each year;
  - Barren solution heating November through March of each year; with the heating plant designed for 50% surplus capacity;
  - Temporary geomembrane covers (thermal covers) will be used beginning in Year 3 to maintain ore and solution temperatures and reduce or eliminate the need for barren solution heating during winter. The thermal covers will also minimize precipitation infiltration during spring and summer, and maximize runoff for storage or discharge depending on operational water needs;

  - Drip lines will be buried to a depth of at least 1 m before winter, and after Year -1, a backup irrigation area equal to 100% of the primary area will be provided in the event the primary lines freeze.

- Solution ponds will not be used in the winter except in upset conditions. Pregnant, barren and wash water will be stored in tanks at the plant (rather than ponds), processed, heated and recirculated back to the heap during the winter period.
- The heap water balance is designed to minimize make-up water demand from external sources and avoid the need to treat surplus water until near the end of the mine life.

The heap and ponds will be operated to facilitate progressive closure commencing in Year 4. Final closure will be designed to allow the site to transition to passive management.

1.9.3 Processing Plant

The pregnant solution from the heap leach pad will flow by gravity from the heap at a nominal rate of 455 m^3/h (design 600 m^3/h) by pipe to the pregnant solution tank located in the process plant building. The solution will then be pumped to the carbon adsorption circuit. The carbon adsorption circuit consists of a series of six cascading carbon columns. The barren solution that will discharge from the final carbon column is pumped to the barren solution tank. Cyanide solution, caustic solution and anti-scalant are added to the barren solution as needed. During the colder months, November to the end of March, a boiler will be used to heat the barren solution before it is pumped back to the leach pad. The loaded carbon from the first carbon column is advanced to the desorption circuit. The loaded carbon, 5 t/d, will be acid washed and gold recovered from the carbon-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 5 t carbon reactivation kiln.

Gold will be plated onto knitted-mesh steel wool cathodes in the electrowinning cells. The gold-bearing sludge and steel wool will be dried in an oven, mixed with fluxes and then smelted to produce gold doré and slag. The doré will be stored in a vault while waiting for transport off site to a refinery for further purification. Slag is processed to remove prills for re-melting in the furnace.A laboratory facility will be equipped to perform sample preparation and assays by atomic absorption, fire assay, and cyanide (CN) soluble analyses. A metallurgical test work area for process optimization is also included.

1.9.4 Gold Recovery

A summary of the distribution of ore types and gold recovery are presented in Table 1.9.

Table 1.9: Distribution of Ore Types and Gold Recovery

Ore Type	Ore Tonnes (kt)	Au Oz (000)		Distribution		
		Contained	Recoverable	Ore Tonnes Percentage (%)	Contained Au Percentage (%)	Recoverable Au Percentage (%)
Oxide	38,105	1,731	1,569	82.2	80.3	84.3
Upper Transition	5,326	269	215	11.5	12.5	11.5
Middle Transition	2,185	112	66	4.7	5.2	3.5
Lower Transition	740	45	12	1.6	2.1	0.6
Total Ore	**46,356**	**2,157**	**1,862**	**100**	**100**	**100**

Source: JDS 2016

The gold production model was developed from a combination of metallurgical testing data, the mine production schedule, the heap leach facility construction sequence (or stacking plan), and the leaching plan for the application of barren solution to the heaps.

The gold production model was developed on a quarterly basis for Years -1 through Year 3. At the end of each year when the heaps are no longer loaded with ore there will be an in-process inventory of recoverable gold remaining from two general areas; the recoverable gold in the ore in the heaps that has not been leached to completion and the recoverable gold contained in solution inventories, carbon, and the electrowinning/refining area that has not yet been processed into doré.

At the end of Year 3 the recoverable gold added to the heap and the gold actually produced reach equilibrium and the in-process inventory has stabilized. From Year 4 through to Year 8, the gold produced is the recoverable gold added to the heap with the inventory of gold to be recovered remaining constant. In the third quarter of Year 9 the mined ore and ROM stockpile are depleted. At that time the recovery of the in-process inventory will begin and will continue through Year 10 to completion. It is assumed that 50% of the in-process inventory will be recovered in Year 9 and the remainder in Year 10.

Table 1.10 provides a summary of the gold production schedule.

Table 1.10: Annual Gold Recovery Projections

Description	Unit	Year										
		-1	1	2	3	4	5	6	7	8	9	10
Total Gold to HL	Au (koz)	169	258	223	244	266	252	206	208	209	121	
Cumulative Total Gold to HL	Au (koz)	169	427	650	894	1,159	1,412	1,617	1,826	2,035	2,157	
Recoverable Gold to HL	Au (koz)	146	211	181	210	229	219	181	187	186	111	
Cumulative Recoverable Gold to HL	Au (koz)	146	358	539	749	978	1,197	1,377	1,564	1,751	1,862	
Gold Recovered	Au (koz)	96	204	180	217	229	219	181	187	186	137	26
Cumulative Gold Recovered	Au (koz)	96	300	480	697	926	1,145	1,326	1,512	1,699	1,836	1,862
% Cumulative Gold Recovered	%	56.5	70.3	73.9	78.0	79.9	81.1	82.0	82.8	83.5	85.1	86.3

Source: JDS 2016

1.10 Project Infrastructure

Due to the remoteness of the property, significant infrastructure would be required for the storage of consumables, power generation, and personnel accommodation. The Coffee Gold Project site is planned to have bulk fuel storage tanks with a fuel distribution system, laydown yards, a bi-fuel (liquid natural gas (LNG) and diesel) power plant, vehicle maintenance shop, accommodation camp, water and domestic waste management facilities, satellite communication and an airfield.

A 214 km all-season road from Dawson to the site will be utilized. The route comprises an existing service road which will be upgraded plus approximately 37 km of new road construction. The proposed route crosses the Stewart and Yukon Rivers. During open water season barges will be used to transport supply trucks across the river. During winter months ice bridges will be constructed across the rivers. Storage facilities for all major consumables have been sized to allow for ten weeks of road access interruptions during spring break-up and fall freeze-up.

A 1,220 m all-weather airstrip will be located approximately 7 km west of the Coffee Gold Project accommodation complex. The airstrip will be able to handle large turboprop fixed wing aircraft and will be primarily used for personnel transportation.

The major infrastructure related to the mining and processing operations at the site includes the primary and secondary crushing facilities, carbon adsorption plant, gold refinery, heap leach facilities, waste rock storage areas (WRSA), water drainage structures and storage ponds, and haul roads.

The installed power generating capacity will be 9 MW. Although the power generators will have the capability to use either diesel or LNG, only diesel is considered for the Feasibility Study. Buildings and facilities will be heated primarily by heat recovered from the power plant.

The accommodation and office complex will be portable, with modular units constructed off-site. The construction phase will accommodate up to 294 workers. This complex will be sufficient to use throughout operations as the maximum on site workforce is expected to be 225.

1.11 Environment Assessment and Permitting

In 2010, Kaminak initiated the collection of baseline environmental data and heritage information. The collection of environmental baseline information, which is ongoing, includes the disciplines of wildlife, water, hydrology (surface and groundwater), climate, vegetation, air quality and aquatic life. The data collected to date is reflective of undisturbed areas found in and around the Project area.

Kaminak has ensured the involvement of First Nations in the design, implementation and data collection of the baseline programs. Through ongoing dialogue with First Nations, environment and heritage values are readily identified and incorporated. The project area was, and continues to be, used by First Nations and it is important to Kaminak that future use be considered in Project planning.

The Project will be subject to an environmental and socio-economic assessment (ESA) under the Yukon Environmental and Socio-economic Assessment Act (YESAA), administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB). The ESA documents the potential environmental and socio-economic effects of the Project by evaluating baseline information, the proposed mine plan and by consulting widely with governments, First Nations, communities, stakeholders, experts and the public.

The ESA will also outline mitigation measures and management plans to be employed to minimize or eliminate possible negative effects resulting from the Project, and conduct research to address environmental priorities. Work has commenced on the ESA with summer 2016 being the target date for its submission to the YESAB.

Once the adequacy review is completed, Kaminak intends to submit the application for a water license from the Yukon Water Board, a Quartz Mining License under the Quartz Mining Land Use Regulation, and other authorizations.

1.12 First Nations' Considerations

The Project is located within the traditional territory of Tr'ondëk Hwëch'in and the asserted area of White River First Nation. A portion of the claim block is located within the shared traditional territory of Selkirk First Nation. The proposed road alignment is located with the traditional territory of Tr'ondëk Hwëch'in, portions of which are located within the shared traditional territories of Selkirk First Nation and the First Nation of Nacho Nyak Dun and the asserted area of White River First Nation.

Kaminak have an Exploration Cooperation Agreement with Tr'ondëk Hwëch'in, signed May 2013, and an Exploration Communication and Cooperation Agreement with White River First Nation, signed June 2014.

1.13 Capital Costs

The initial or pre-production capital cost estimate (CAPEX) is $318.4 M and sustaining CAPEX of $167.0 M, as summarized in Table 1.11. Costs are expressed in Canadian dollars with no escalation (Q4-2015 dollars).

Table 1.11: Life of Mine Capital Costs

Capital Cost	Pre-Production	Production / Sustaining	LOM
	$M	$M	$M
Mining	85.4	47.5	132.9
On-Site Development	7.7	0.9	8.7
Ore Crushing and Handling	16.4	0.0	16.4
Heap Leach	28.2	34.5	62.7
Process Plant	27.6	1.0	28.6
On-Site Infrastructure	43.1	2.8	46.0
Off-Site Infrastructure	24.3	0.0	24.3
Indirects	31.7	4.7	36.4
EPCM	18.9	1.5	20.4
Owner Costs	7.9	0.0	7.9
Reclamation/Closure	0.0	60.5	60.5
Subtotal	**291.4**	**153.4**	**444.8**
Contingency	26.1	7.2	33.3
Total Capital	**317.4**	**160.6**	**478.1**

Source: JDS 2016

Preparation of the capital cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency, and uses defined and proven project execution strategies. The estimates were developed using first principles, applying directly-related project experience, and the use of general industry factors. Almost all of the estimates used in this project were obtained from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The initial capital estimates include all pre-production mining activities in Years -3, -2 and -1 and are based on owner-performed mining. Equipment leases have not been considered in this estimate.

The CAPEX estimate includes the costs required to develop, sustain, and close the operation for the planned 10-year mine life, which includes a one year construction period. The sustaining capital estimate is based on required capital expenditure during operations for waste storage development, mining equipment acquisition and rebuilding, heap leach pad extensions, and mining infrastructure installations as defined by the mine plan. The closure and reclamation estimate is based on a preliminary estimation of a closure plan commencing in Year 4 and continuing to Year 20.

1.14 Operating Costs

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 Feasibility Study.

The total life of mine costs are summarized in Table 1.12.

Table 1.12: Life of Mine Operating Costs (excluding costs capitalized in pre-production)

Operating Cost†	Average $/t processed	LOM C$M
Mining*	15.26	707.4
Processing	4.97	230.3
Surface & Infrastructure	0.97	45.2
G&A	2.89	134.2
Total Operating Costs	**24.10**	**1,117.1**

(†): Operating Costs include the working capital during the pre-production period

(*): Average LOM Open Pit Mining Cost amounts to $2.27 /t mined at a 5.7 strip ratio
Source: JDS 2015

1.15 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.

The reader is cautioned that the gold price and exchange rate used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The gold price is based on many complex factors and there is no reliable method of predicting long term gold price.

Other economic factors considered include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario);
- Closure cost of $60.5 M which includes a 12% contingency;
- Nominal 2016 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Results are presented on 100% ownership and do not include management fees or financing costs;
- 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.); and
- Costs from operations incurred in the pre-production period have been capitalized and assumed to cover the working capital necessary.

The results of the economic analysis are shown in Table 1.13.

Sensitivities to metal prices, head grade, OPEX and CAPEX were conducted by adjusting each variable up and down 15% independently. As with most metal mining projects, the Project is most sensitive to metal price and head grade. The Project is slightly more sensitive to OPEX than CAPEX. The base case sensitivities are shown in Table 1.14.

Table 1.13: Economic Results

Parameter	Unit	Value
Au Price	US $/oz	1,150
Exchange Rate	US:CDN	0.78
Production		
Mine Life (start of commercial gold production)	Yrs	10
Au Produced	LOM k oz	1,858
	Avg k oz/yr	202
LOM NSR (after royalties)	$M	2,708
Operating Costs	LOM $M	1,094
	$/t milled	25.31
Capital Costs		
Pre-Production	$M	291
Sustaining & Closure	$M	153
Subtotal	$M	445
Contingency 11%	$M	33
Total Capital Costs	**$M**	**478**
Operating Cash Flow	$M	1,614
	$M/yr	176
Cash Cost	US$/oz	515
Cash Cost (incl. Sustaining Capital)	US$/oz	660
Economic Results		
After-Tax Free Cash Flow	$M	682.3
	Avg $M/yr	74
Discount Rate	%	5
Pre-Tax NPV$_{5\%}$	**$M**	**762**
Pre-Tax Internal Rate Of Return (IRR)	**%**	**50**
Pre-Tax Payback	**Yrs**	**1.5**
After-Tax NPV$_{5\%}$	**$M**	**455**
After-Tax IRR	**%**	**37**
After-Tax Payback	**Yrs**	**2.0**

Source: JDS 2016

1.15.1 Timing of Revenues and Working Capital

Working capital has been considered in the economic analysis by accounting for the equivalent of three months of Year -1 operating costs (for fuel and consumables) during the pre-production period. Working capital amounts to $23.0 M. The working capital is recuperated during the last year of production, Year 10.

Table 1.14: Economic Sensitivities

After-Tax $NPV_{5\%}$ ($M)			
Variable	**-15%**	**100%**	**15%**
Metal Prices	271	455	640
Head Grade	672	455	295
Operating Costs	271	455	639
Capital Costs	574	455	337

Source: JDS 2016

1.16 Conclusions

It is concluded that the Feasibility Study 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 Coffee Gold Project contains a substantial oxide 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.

There is also a possibility of improving the project economics by identifying additional mineral resources within the development area that may justify increased open pit production or extend the mine life.

The most significant potential risks associated with the Project are metallurgical recovery, climatic influences, uncontrolled dilution, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

1.17 Recommendations

It is recommended that the Project be advanced to construction through the normal process of permit acquisition, financing detailed engineering and construction. Costs for engineering and construction are provided in the capital cost of this study.

2 Introduction

2.1 Basis of Feasibility Study

This Feasibility Study Report was compiled by JDS for Kaminak. 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

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;
- Optimization of the June 2014 PEA outcomes;
- Compilation of the report, including data and information provided by other consulting companies;
- Reserve estimation and mine planning;
- Ore crushing and handling;
- ADR process plant;
- Design and location of on-site infrastructure;
- Site access road route selection and design oversight;
- Environmental permitting and community relations;
- CAPEX and OPEX estimation;
- Preparation of a financial model to enable economic evaluation;
- Interpretation of results and recommendations to improve value and reduce risks.

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

- Geotechnical assessment and design of open pits; and
- Geotechnical assessment of ground conditions for infrastructure facilities, the heap leach pad and waste rock storage facilities.

SIM Geological Inc. (SIM Geological) scope of work included:

- Project setting, history and geology description; and
- Mineral resource estimate.

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

- Implementation and supervision of the metallurgical testing program;
- Development of a conceptual flowsheet, specifications and selection of leach process equipment; and
- Establishment of gold recovery values based on metallurgical testing results.

ALS Canada (ALS) scope of work included:

- Crushing and abrasion testwork;

The Mines Group Inc. (The Mines Group) scope of work included:

- Design and construction methodology of the heap leach facility.

Fred Lightner, Kaminak Gold Corp. (Kaminak) provided:

- Oversight and review of metallurgical testwork, crushing, heap leach facility, ADR process and gold recovery model.

RRD International Corp scope of work included:

- Design and construction methodology of the heap leach facility;

Lorax Environmental Services Ltd. (Lorax) scope of work included:

- Water balance, geochemistry and source terms;

SRK Consulting (Canada) Inc. (SRK) scope of work included:

- Water management structure design;

Onsite Engineering Ltd. (Onsite Engineering) scope of work included:

- Final alignment and design of site access road.

Tetra Tech EBA (Tetra) scope of work included:

- Permafrost.

Ernst and Young's scope of work included:

- Tax model review.

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 Kaminak 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 Kaminak 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:

Table 2.1: Qualified Person Responsibilities

Qualified Person	Company	Report Sections of Responsibility
Gordon Doerksen, P.Eng.	JDS Energy & Mining Inc.	1, 2, 3, 18, (except 18.1.2, 18.1.3, 18.1.4, 18.4), 19, 20, 21, 22, 23 (except 23.6.5), 25, 26, 27, 28, 29
Robert Sim, P. Geo.	SIM Geological Inc.	4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 24
Dino Pilotto, P.Eng.	JDS Energy & Mining Inc.	15, 16 (except 16.2.4.1, 16.2.6.6.1)
Kelly McLeod, P. Eng.	JDS Energy & Mining Inc.	17 (except 17.3.4 and 17.4)
Mark Smith, P.E.	RRD International Corp.	17.3.4, 17.4
Mike Levy, P.E.	SRK Consulting (U.S.) Inc.	16.2.4.1, 16.2.6.6.1, 18.1.2, 18.1.3, 18.1.4
Tom Sharp, P.Eng.	SRK Consulting (Canada) Inc.	18.4, 25.3.6.5
Daniel W. Kappes, P.E.	Kappes, Cassiday & Associates	13

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

- Gordon Doerksen visited the Project site on May 22, 2015.
- Robert Sim visited the site several times including September 12-14, 2011, August 28-29, 2012, May 15-16, 2013 and September 24, 2014.
- Dino Pilotto visited the site on May 22, 2015.
- Kelly McLeod has not visited the site.
- Mike Levy visited the site on May 22, 2015.
- Tom Sharp has not visited the site.
- Mark Smith visited the site on May 22, 2015.
- Daniel Kappes has not visited the site.

2.4 Sources of Information

The sources of information include data and reports supplied by Kaminak 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 "Preliminary Economic Assessment Technical Report, Coffee Gold Project, Yukon Territory, Canada" prepared by JDS Energy and Mining Inc., with an effective date of June 10, 2014.

2.5 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 $).

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 Kaminak 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:

- Fred Lightner, Kaminak Gold Corp. provided oversight and review of metallurgical testwork, crushing, heap leach facility, the ADR process and gold recovery model;
- Anthony Crews, P.E. designed the heap leach facility;
- David Flather, Lorax Environmental Services Ltd. Advised on geochemistry and water balance;
- Jeremy Araki, Onsite Engineering Ltd.;
- Kevin Jones, Tetra Tech EBA (Tetra) for advice relating to permafrost;
- Ernst and Young advised on the tax 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

The Coffee Project is located in west-central Yukon, within the Whitehorse Mining District, Canada, 130 kilometers (km) south of Dawson (Figure 4.1). The Project comprises 3,021 contiguous claims covering an aggregate area of approximately 60,502 hectares (ha). Claims are summarized in Table 4.1. The Coffee property covers parts of 1:50 000 scale national topographic system (NTS) map sheets 115J-13, 115J-14, and 115J-15. The main mineralized zones at the Project are located roughly at the UTM NAD83 coordinates of 6,974,000mN and 584,000mE.

4.1 Mineral Tenure

The main Coffee Project claim block consists of 3,021 registered claims (2,927 Coffee, 68 Cream, 16 Lion, and 10 Sugar). The entire claim block covers an area measuring approximately 50 km by 12 km (Figure 4.1). The boundaries of the individual claims have not been legally surveyed. The list of claims is presented in Figure 4.2.

The mineral rights include surface rights under the Yukon Territory Quartz Mining Act, including access to the property under a Class 4 Mining Land Use Permit to undertake exploration activities (see Section 4.3) and the right to extract mineralized material from surface pursuant to the grant of a Quartz Lease (see Section 4.4).

4.2 Underlying Agreements

Kaminak's rights to the Coffee claims were acquired from prospector Mr. Shawn Ryan of Dawson, through an agreement dated April 27, 2009 (amended and restated on June 9, 2009 and further amended on March 25, 2010 and March 30, 2011). Pursuant to that agreement, in 2011 Kaminak earned a 100% legal and beneficial interest in the property by making cash payments of $400,000; issuing 2,000,000 shares; and fulfilling a $1,800,000 work commitment.

There is a 2% net smelter returns royalty (NSR) on the property, payable to Mr. Ryan, subject at any time to a 1% buy-back for C$2.0 M, with annual advance royalty payments of $20,000 commencing December 31, 2013. Subject to the 2% NSR payable to Mr. Ryan, the property is free and clear of all liens and third party interests.

Figure 4.1: Coffee Project Location Map

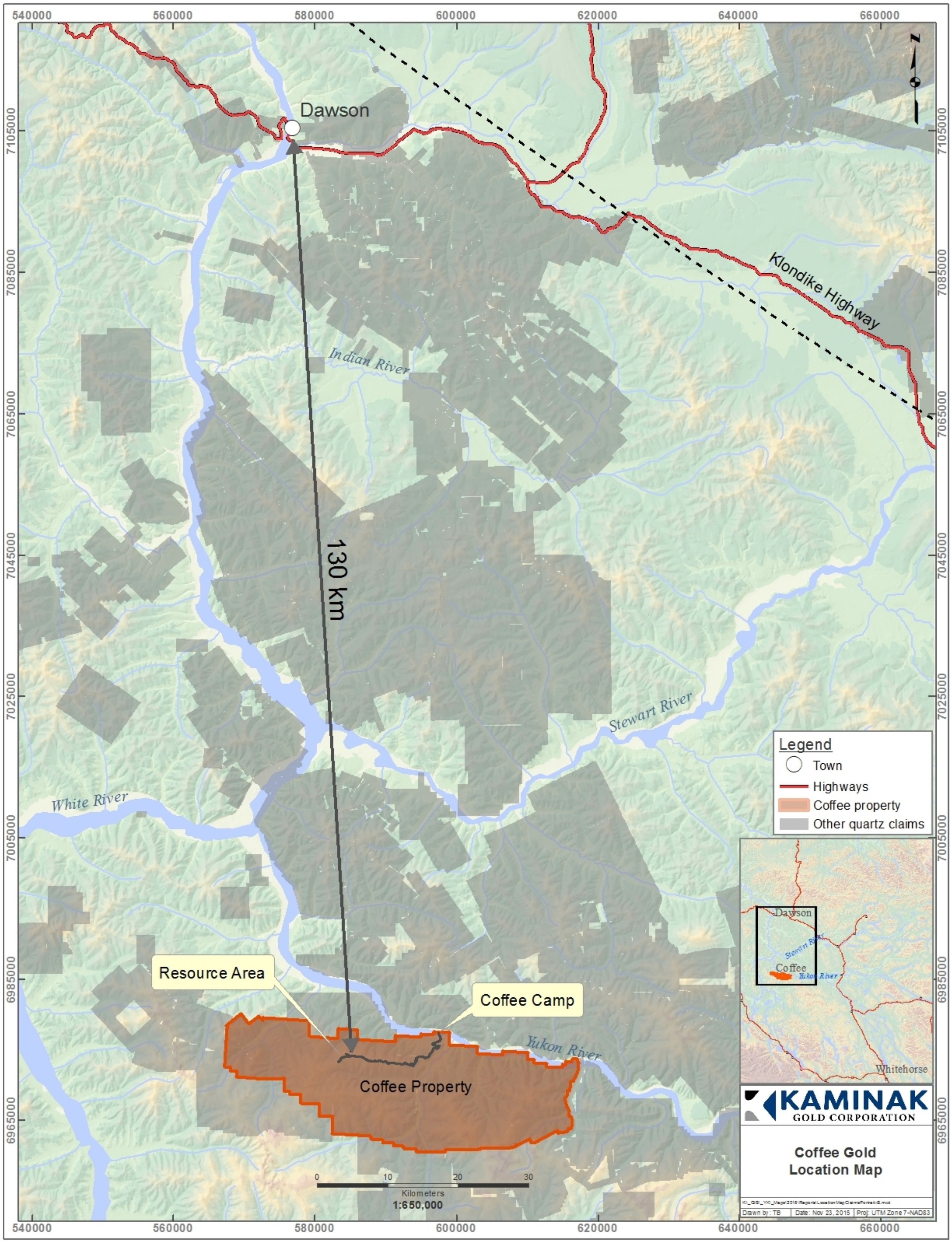


Figure 4.2: Mineral Tenure Map

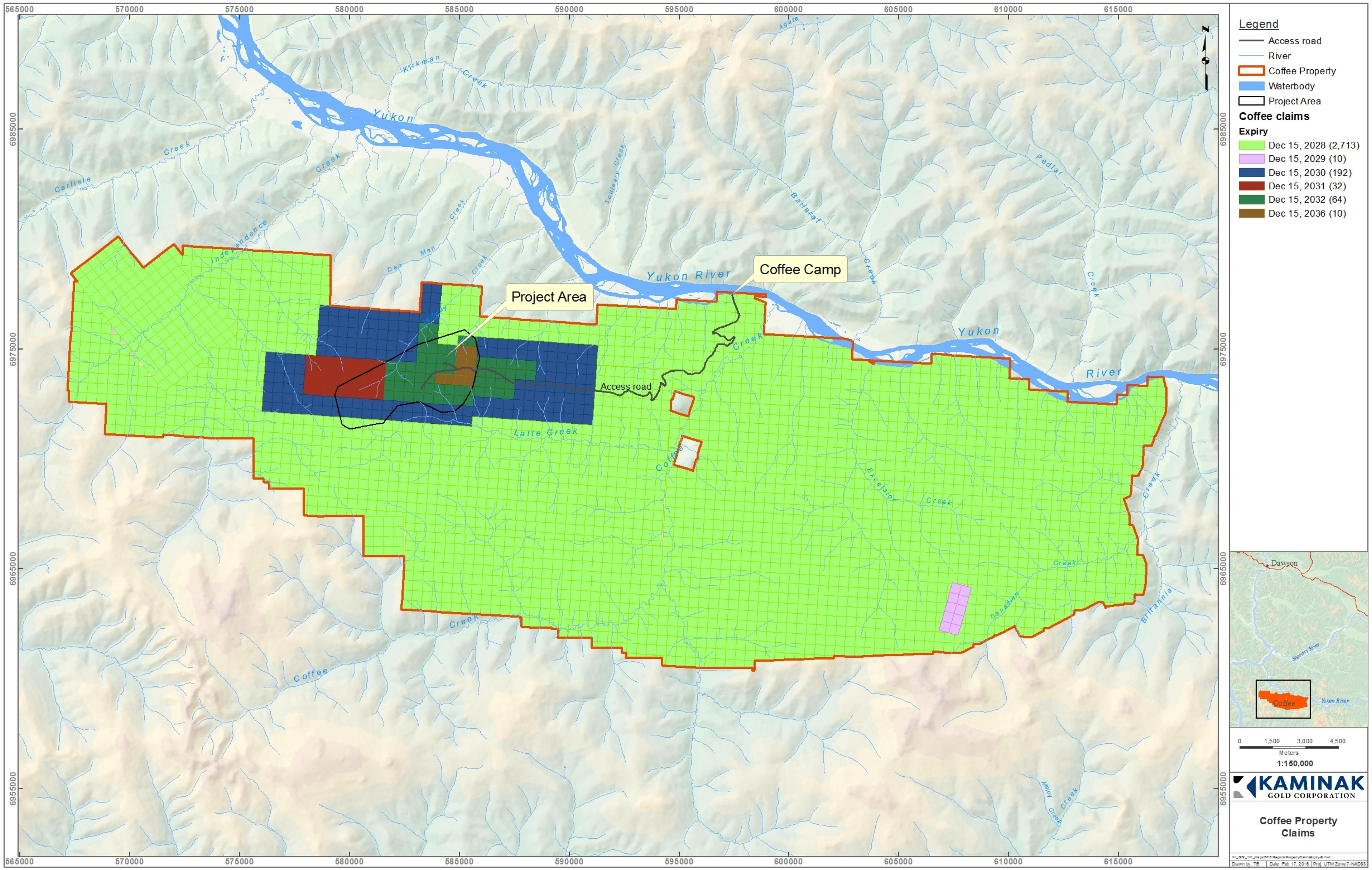


Table 4.1: Kaminak Coffee Property Claims

Claim Name	Grant No.	Expiry Date	NTS #'s	Group
COFFEE 1 - 6	YC46734 - YC46739	15-Dec-2032	115J14	HW07440
COFFEE 7	YC46740	15-Dec-2036	115J14	HW07442
COFFEE 8	YC46741	15-Dec-2036	115J14	HW07441
COFFEE 9 - 12	YC46742 - YC46745	15-Dec-2036	115J14	HW07444
COFFEE 13 - 14	YC46746 - YC46747	15-Dec-2036	115J14	HW07442
COFFEE 15	YC46748	15-Dec-2036	115J14	HW07443
COFFEE 16	YC46749	15-Dec-2036	115J14	HW07442
COFFEE 17 - 18	YC53949 - YC53950	15-Dec-2032	115J14	HW07444
COFFEE 19 - 24	YC53951 - YC53956	15-Dec-2028	115J14	HW07440
COFFEE 25 - 36	YC53957 - YC53968	15-Dec-2032	115J14	HW07441
COFFEE 37 - 39	YC54445 - YC54447	15-Dec-2032	115J14	HW07443
COFFEE 40	YC54448	15-Dec-2033	115J14	HW07443
COFFEE 41 - 54	YC54449 - YC54462	15-Dec-2032	115J14	HW07441
COFFEE 55 - 62	YC54463 - YC54470	15-Dec-2031	115J14	HW07441
COFFEE 63 - 68	YC54471 - YC54476	15-Dec-2032	115J14	HW07441
COFFEE 69 - 92	YC54477 - YC54500	15-Dec-2031	115J14	HW07441
COFFEE 93	YC60164	15-Dec-2032	115J14	HW07444
COFFEE 94	YC60165	15-Dec-2032	115J14	HW07442
COFFEE 95	YC60166	15-Dec-2032	115J14	HW07444
COFFEE 96	YC60167	15-Dec-2032	115J14	HW07442
COFFEE 97	YC60168	15-Dec-2032	115J14	HW07444
COFFEE 98	YC60169	15-Dec-2032	115J14	HW07442
COFFEE 99	YC60170	15-Dec-2032	115J14	HW07444
COFFEE 100	YC60171	15-Dec-2032	115J14	HW07442
COFFEE 101	YC60172	15-Dec-2032	115J14	HW07444
COFFEE 102	YC60173	15-Dec-2032	115J14	HW07442
COFFEE 103	YC60174	15-Dec-2032	115J14	HW07444
COFFEE 104	YC60175	15-Dec-2032	115J14	HW07442
COFFEE 105 - 112	YC60176 - YC60183	15-Dec-2032	115J14	HW07443
COFFEE 113 - 122	YC83190 - YC83199	15-Dec-2030	115J14	HW07442
COFFEE 123 - 128	YC83200 - YC83205	15-Dec-2030	115J14	HW07444
COFFEE 129 - 132	YC83206 - YC83209	15-Dec-2028	115J14	HW07440
COFFEE 133 - 136	YC83210 - YC83213	15-Dec-2030	115J14	HW07444
COFFEE 137 - 140	YC83214 - YC83217	15-Dec-2028	115J14	HW07440
COFFEE 141 - 144	YC83218 - YC83221	15-Dec-2030	115J14	HW07444
COFFEE 145 - 148	YC83222 - YC83225	15-Dec-2028	115J14	HW07440
COFFEE 149 - 152	YC83226 - YC83229	15-Dec-2030	115J14	HW07444
COFFEE 153 - 172	YC83230 - YC83249	15-Dec-2028	115J14	HW07440

Claim Name	Grant No.	Expiry Date	NTS #'s	Group
COFFEE 173 - 226	YC83250 - YC83303	15-Dec-2030	115J14	HW07441
COFFEE 227 - 276	YC83652 - YC83701	15-Dec-2030	115J14	HW07441
COFFEE 277 - 282	YC89405 - YC89410	15-Dec-2030	115J14	HW07442
COFFEE 283 - 288	YC89411 - YC89416	15-Dec-2030	115J14	HW07444
COFFEE 289 - 292	YC89417 - YC89420	15-Dec-2028	115J14	HW07440
COFFEE 293 - 298	YC89421 - YC89426	15-Dec-2030	115J14	HW07442
COFFEE 299 - 304	YC89427 - YC89432	15-Dec-2030	115J14	HW07444
COFFEE 305 - 308	YC89433 - YC89436	15-Dec-2028	115J14	HW07440
COFFEE 309 - 316	YC89437 - YC89444	15-Dec-2030	115J14	HW07441
COFFEE 317 - 328	YC89445 - YC89456	15-Dec-2030	115J14	HW07443
COFFEE 329 - 344	YC89457 - YC89472	15-Dec-2030	115J14	HW07442
COFFEE 345 - 382	YC93441 - YC93478	15-Dec-2028	115J13,115J14	HW07441
COFFEE 383 - 394	YC93479 - YC93490	15-Dec-2028	115J14	HW07443
COFFEE 395 - 404	YC93491 - YC93500	15-Dec-2028	115J14	HW07442
COFFEE 405 - 410	YC97368 - YC97373	15-Dec-2028	115J14	HW07442
COFFEE 411 - 570	YC92601 - YC92760	15-Dec-2028	115J13,115J14	HW07441
COFFEE 571 - 578	YC92761 - YC92768	15-Dec-2028	115J14	HW07443
COFFEE 579 - 586	YC92769 - YC92776	15-Dec-2028	115J14	HW07441
COFFEE 587 - 590	YC92777 - YC92780	15-Dec-2028	115J14	HW07441
COFFEE 591 - 598	YC92781 - YC92788	15-Dec-2028	115J14	HW07443
COFFEE 599 - 610	YC92789 - YC92800	15-Dec-2028	115J13,115J14	HW07441
COFFEE 611 - 616	YC93351 - YC93356	15-Dec-2028	115J14	HW07441
COFFEE 617 - 624	YC93357 - YC93364	15-Dec-2028	115J14	HW07443
COFFEE 625	YC93365	15-Dec-2028	115J13	HW07441
COFFEE 627 - 658	YC96801 - YC96832	15-Dec-2028	115J13	HW07441
COFFEE 659 - 676	YC96833 - YC96850	15-Dec-2028	115J14	HW07443
COFFEE 677 - 684	YC96851 - YC96858	15-Dec-2028	115J14	HW07441
COFFEE 685 - 718	YC96859 - YC96892	15-Dec-2028	115J14	HW07443
COFFEE 719 - 726	YC96893 - YC96900	15-Dec-2028	115J14	HW07442
COFFEE 727 - 782	YC92535 - YC92590	15-Dec-2028	115J14	HW07442
COFFEE 783	YC92591	15-Dec-2028	115J14	HW07444
COFFEE 784	YC92592	15-Dec-2028	115J14	HW07442
COFFEE 785	YC92593	15-Dec-2028	115J14	HW07444
COFFEE 786	YC92594	15-Dec-2028	115J14	HW07442
COFFEE 787	YC92595	15-Dec-2028	115J14	HW07444
COFFEE 788	YC92596	15-Dec-2028	115J14	HW07442
COFFEE 789	YC92597	15-Dec-2028	115J14	HW07444
COFFEE 790	YC92598	15-Dec-2028	115J14	HW07442
COFFEE 791	YC92599	15-Dec-2028	115J14	HW07444
COFFEE 792	YC92600	15-Dec-2028	115J14	HW07442

Claim Name	Grant No.	Expiry Date	NTS #'s	Group
COFFEE 793	YC92818	15-Dec-2028	115J14	HW07444
COFFEE 794	YC92819	15-Dec-2028	115J14	HW07442
COFFEE 795	YC92820	15-Dec-2028	115J14	HW07444
COFFEE 796	YC92821	15-Dec-2028	115J14	HW07442
COFFEE 797	YC92822	15-Dec-2028	115J14	HW07444
COFFEE 798 - 865	YC92823 - YC92890	15-Dec-2028	115J14	HW07442
COFFEE 866 - 894	YC93271 - YC93299	15-Dec-2028	115J14	HW07442
COFFEE 895 - 910	YC92801 - YC92816	15-Dec-2028	115J14	HW07442
COFFEE 911 - 928	YD12701 - YD12718	15-Dec-2028	115J14	HW07443
COFFEE 929 - 960	YD12719 - YD12750	15-Dec-2028	115J14	HW07442
COFFEE 961 - 969	YD13231 - YD13239	15-Dec-2028	115J14	HW07443
COFFEE 970 - 1040	YD13241 - YD13311	15-Dec-2028	115J14	HW07443
COFFEE 1041 - 1168	YD13312 - YD13439	15-Dec-2028	115J14	HW07442
COFFEE 1169 - 1172	YD13440 - YD13443	15-Dec-2028	115J14	HW07444
COFFEE 1173 - 1180	YD13444 - YD13451	15-Dec-2028	115J14	HW07440
COFFEE 1181 - 1184	YD13452 - YD13455	15-Dec-2028	115J14	HW07444
COFFEE 1185 - 1192	YD13456 - YD13463	15-Dec-2028	115J14	HW07440
COFFEE 1193 - 1196	YD13464 - YD13467	15-Dec-2028	115J14	HW07444
COFFEE 1197 - 1204	YD13468 - YD13475	15-Dec-2028	115J14	HW07440
COFFEE 1205 - 1208	YD13476 - YD13479	15-Dec-2028	115J14	HW07444
COFFEE 1209 - 1216	YD13480 - YD13487	15-Dec-2028	115J14	HW07440
COFFEE 1217 - 1224	YD13488 - YD13495	15-Dec-2028	115J14	HW07444
COFFEE 1225 - 1244	YD13496 - YD13515	15-Dec-2028	115J14	HW07442
COFFEE 1245 - 1252	YD13516 - YD13523	15-Dec-2028	115J14	HW07444
COFFEE 1253 - 1272	YD13524 - YD13543	15-Dec-2028	115J14	HW07442
COFFEE 1273 - 1280	YD13544 - YD13551	15-Dec-2028	115J14	HW07444
COFFEE 1281 - 1300	YD13552 - YD13571	15-Dec-2028	115J14	HW07442
COFFEE 1301 - 1308	YD13572 - YD13579	15-Dec-2028	115J14	HW07444
COFFEE 1309 - 1328	YD13580 - YD13599	15-Dec-2028	115J14	HW07442
COFFEE 1329 - 1332	YD13600 - YD13603	15-Dec-2028	115J14	HW07444
COFFEE 1333 - 1352	YD13604 - YD13623	15-Dec-2028	115J14	HW07442
COFFEE 1353 - 1356	YD13624 - YD13627	15-Dec-2028	115J14	HW07444
COFFEE 1357 - 1376	YD13628 - YD13647	15-Dec-2028	115J14	HW07442
COFFEE 1377 - 1380	YD13648 - YD13651	15-Dec-2028	115J14	HW07444
COFFEE 1381 - 1400	YD13652 - YD13671	15-Dec-2028	115J14	HW07442
COFFEE 1401 - 1412	YD13672 - YD13683	15-Dec-2028	115J14	HW07444
COFFEE 1413 - 1416	YD13684 - YD13687	15-Dec-2028	115J14	HW07442
COFFEE 1421 - 1429	YD13692 - YD13700	15-Dec-2028	115J14	HW07442
COFFEE 1430	YD42501	15-Dec-2028	115J14	HW07442
COFFEE 1435 - 1472	YD42506 - YD42543	15-Dec-2028	115J14	HW07442

Claim Name	Grant No.	Expiry Date	NTS #'s	Group
COFFEE 1473 - 1496	YD42544 - YD42567	15-Dec-2028	115J14	HW07444
COFFEE 1497 - 1714	YD42701 - YD42918	15-Dec-2028	115J14,115J15	HW07444
COFFEE 1715 - 1718	YD43085 - YD43088	15-Dec-2028	115J14	HW07442
COFFEE 1719 - 1781	YD43929 - YD43991	15-Dec-2028	115J14,115J13	HW07441
COFFEE 1782 - 1954	YD43992 - YD44164	15-Dec-2028	115J13,115J14	HW07441
COFFEE 1955 - 1986	YD16283 - YD16314	15-Dec-2028	115J14	HW07440
COFFEE 1987 - 1992	YD16315 - YD16320	15-Dec-2028	115J14	HW07444
COFFEE 1993 - 1996	YD16321 - YD16324	15-Dec-2028	115J14	HW07440
COFFEE 1997 - 2006	YD16325 - YD16334	15-Dec-2028	115J14	HW07442
COFFEE 2007 - 2012	YD16335 - YD16340	15-Dec-2028	115J14	HW07444
COFFEE 2013 - 2016	YD16341 - YD16344	15-Dec-2028	115J14	HW07440
COFFEE 2017 - 2022	YD16345 - YD16350	15-Dec-2028	115J14	HW07442
COFFEE 2023 - 2028	YD16351 - YD16356	15-Dec-2028	115J14	HW07444
COFFEE 2029 - 2032	YD16357 - YD16360	15-Dec-2028	115J14	HW07440
COFFEE 2033 - 2038	YD16361 - YD16366	15-Dec-2028	115J14	HW07442
COFFEE 2039 - 2044	YD16367 - YD16372	15-Dec-2028	115J14	HW07444
COFFEE 2045 - 2048	YD16373 - YD16376	15-Dec-2028	115J14	HW07440
COFFEE 2049 - 2054	YD16377 - YD16382	15-Dec-2028	115J14	HW07442
COFFEE 2055	YD16383	15-Dec-2028	115J14	HW07444
COFFEE 2056 - 2060	YD16384 - YD16388	15-Dec-2028	115J14	HW07444
COFFEE 2061 - 2064	YD16389 - YD16392	15-Dec-2028	115J14	HW07440
COFFEE 2065 - 2076	YD16393 - YD16404	15-Dec-2028	115J14	HW07444
COFFEE 2077 - 2080	YD16405 - YD16408	15-Dec-2028	115J14	HW07440
COFFEE 2081 - 2092	YD16409 - YD16420	15-Dec-2028	115J14	HW07444
COFFEE 2093 - 2096	YD16421 - YD16424	15-Dec-2028	115J14	HW07440
COFFEE 2097 - 2116	YD16425 - YD16444	15-Dec-2028	115J14	HW07442
COFFEE 2117 - 2124	YD16445 - YD16452	15-Dec-2028	115J14	HW07444
COFFEE 2125 - 2264	YD89255 - YD89394	15-Dec-2028	115J15	HW07444
COFFEE 2265 - 2308	YD89395 - YD89438	15-Dec-2028	115J14	HW07442
COFFEE 2309	YD89439	15-Dec-2028	115J14	HW07444
COFFEE 2310 - 2316	YD89440 - YD89446	15-Dec-2028	115J14	HW07089
COFFEE 2317 - 2346	YD89447 - YD89476	15-Dec-2028	115J14	HW07442
COFFEE 2347 - 2846	YD91501 - YD92000	15-Dec-2028	115J15,115J14	HW07440
COFFEE 2847 - 2936	YD90101 - YD90190	15-Dec-2028	115J15	HW07440
CREAM 1 - 22	YC60088 - YC60109	15-Dec-2028	115J13	HW07441
CREAM 23 - 68	YC83144 - YC83189	15-Dec-2028	115J13	HW07441
LION 1 - 16	YC83761 - YC83776	15-Dec-2028	115J14	HW07443
SUGAR 1 - 10	YC95568 - YC95577	15-Dec-2029	115J15	HW07444

4.3 Permits and Authorization

Kaminak has obtained all permits and authorizations required from governmental agencies to allow surface drilling and exploration activities on the Coffee Project.

The Energy, Mines and Resources Department of the Yukon Government issued a Class 4 Quartz Mining Permit on July 12th, 2011, amended on February 29, 2012, with an expiry date of July 11, 2016. The Class 4 Permit includes provisions for: an 80-person camp (Coffee camp) located on the Yukon River near the confluence with Coffee Creek, a 40 km access road, temporary trails to allow improved access to the property, a winter road, and surface drilling and exploration activities on the Coffee Project. The Class 4 Mining Land Use Permit (#LQ00312a) is the sole permit necessary for the exploration work currently undertaken. Permit renewal is currently underway with the Energy, Mines and Resources Department of the Yukon Government in advance of the expiry of #LQ00312a.

The Yukon Water Board issued a Class B Water Licence on April 18, 2012 (licence number MN12-014), with an expiry date of July 11, 2016. The Class B Water Licence was required when the camp numbers increased from 50 persons. Permit renewal is currently underway with the Energy Mines and Resources Department in advance of the expiry of MN12-014.

Kaminak has advised the authors of this report that it has obtained and complied with any applicable permit requirements to conduct mineral exploration on the Coffee Project claims.

Apart from those disclosed herein, the Qualified Persons are unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform the exploration work recommended for the Coffee Project.

4.4 Mining Rights in the Yukon

The Yukon mining industry is governed by the Quartz Mining Act. A basic overview of mining rights in the Yukon is given as follows from the government’s website (www.emr.gov.yk.ca/mining):

“The Quartz Mining Act [QMA] is the primary legislation governing hard rock mining activities on lands in Yukon. The purpose of the QMA is to encourage prospecting, exploration, staking and development of mineral resources by providing an orderly system of allocation of exclusive rights to minerals. Specific permission must be obtained where the surface is occupied by others.

“Mineral tenure is granted under the free entry system in Yukon. This system gives individuals exclusive right to publicly-owned mineral substances from the surface of their claim to an unlimited extension downward vertically from the boundary of the claim or lease. All Commissioner’s lands are open for staking and mineral exploration unless they are expressly excluded or withdrawn by order-in-council (e.g. parks, interim protected lands, buildings, dwelling houses, cemeteries, agricultural lands, settlement lands).”

A Mineral Claim (claim) is a parcel of land granted for hard rock mining, which also includes any ditches, water rights or other things used for mining the claim. A claim is a rectangular plot of land, which does not exceed 1,500 ft by 1,500 ft. All angles of a claim must be right angles, except for fractional claims, which consist of land found between and bounded on opposite sides by previously located mineral claims. A fractional claim does not need to be rectangular in form and the angles do not need to be right angles.

Staking a claim requires that claim tags be obtained from the Mining Recorder prior to staking in the field and that posts be placed in the ground according to specific regulated requirements. Tenure to the mineral rights is dependent on performing exploration work on the claims. To renew claims, a full report of the work done must be submitted to the Mining Recorder when work has been done on claims. Renewal of a claim requires that $100 of work be done per claim per year, based on the Schedule of Representation Work outlined in the Quartz Mining Act. Where work is not performed, a payment in lieu of work can be filed. Claims can be grouped to allow for assessed work performed on one claim to be distributed to adjoining claims.

A Quartz Lease (lease) can be acquired by upgrading claims which have known vein or lode mineral deposits. A lease is considered the most secure mineral right in Yukon. Companies contemplating production will take their claims to lease to provide secure title. Leases are issued for 21-year periods.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

The Coffee Gold Project is located, approximately 130 km south of Dawson and approximately 160 km northwest of Carmacks within the Dawson Range. The Casino copper-gold porphyry deposit (Western Copper Corporation) is located approximately 30 km southeast of the main drilled zones of the Project.

Access to the property is by airplane or helicopter from Whitehorse and/or Dawson or by barge via the Yukon River. In 2011, Kaminak constructed a 23 km road from the barge landing at the Coffee Gold Project camp to the Supremo and Latte drilling areas. This road was the main access for exploration activities from 2012 through 2015.

5.2 Local Resources and Infrastructure

There are currently no all-weather or winter roads connecting the Coffee Gold Project to any of the major communities in the Yukon. However, the Feasibility Study proposes the construction of a 214 km all-weather gravel road between Dawson and the Coffee property. Crossing of the Stewart and Yukon Rivers will be by barge in summer and ice road in winter. An airstrip is located at the Coffee Gold Project camp approximately 10 km from the areas of gold mineralization.

Currently, river transport along the Yukon River, with multiple barge access points to the Coffee Gold Project exploration camp, is available for five months during the summer period when the river is free of ice.

The proposed project infrastructure details are contained in Section 18, while the proposed access road is detailed in Section 18.3.

5.3 Climate

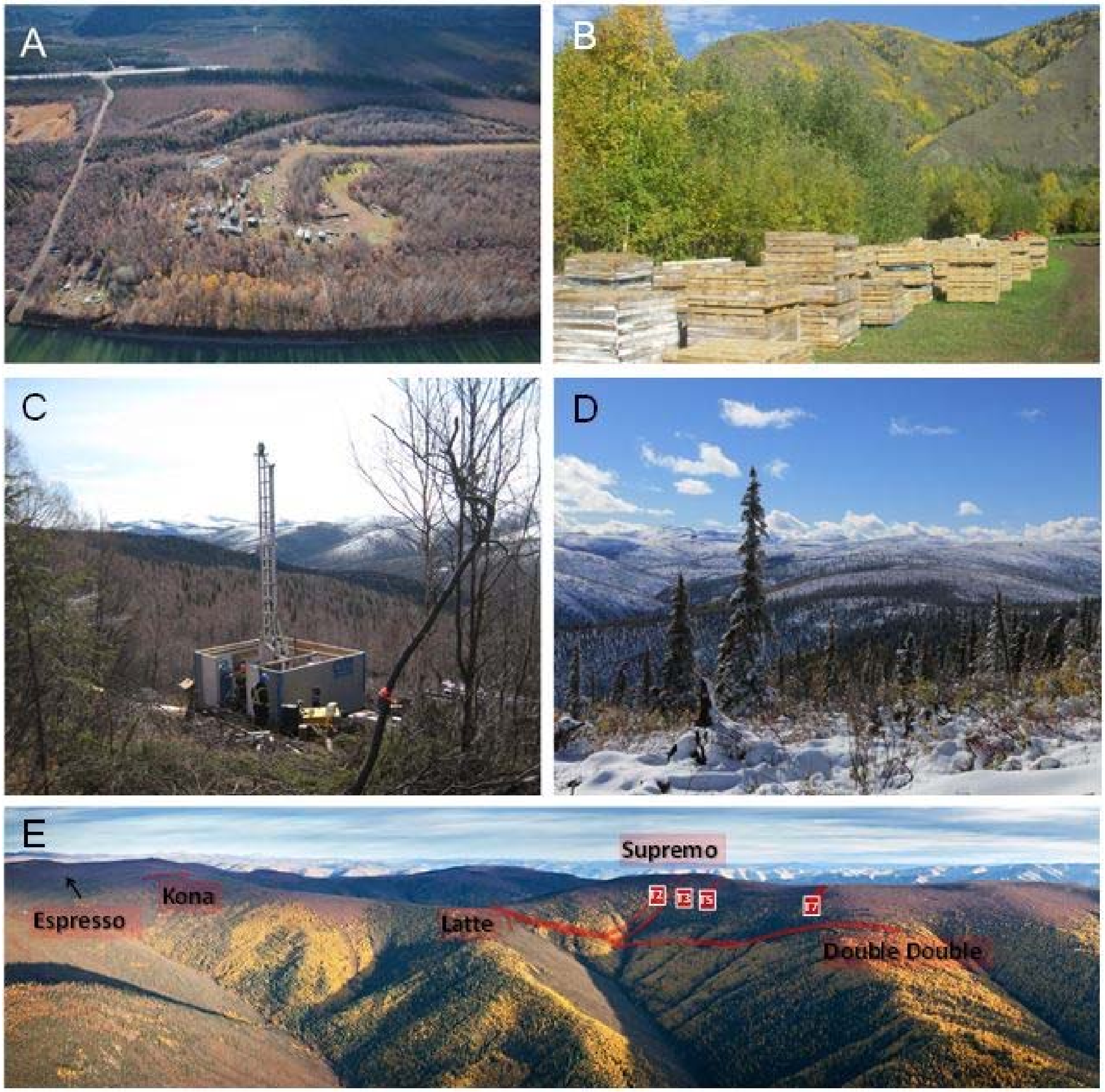
The Yukon has a subarctic, continental climate with a summer mean temperature of 10°C and a winter mean temperature of -23°C. Summer and winter temperatures can reach up to 35°C and -55°C respectively. Dawson, the nearest access point, has a daily temperature average above freezing for 180 days of the year.

5.4 Physiography

The Coffee property is located in the northern Dawson Range, forming a moderate plateau that escaped Pleistocene glaciation. As such, the topography of the area is defined by stream erosion resulting in gently rounded hills with tightly incised valleys. Across the property, elevations range from 400 to 1,500 m above sea level. The majority of the property is above tree line and contains short shrubby vegetation.

The Coffee Gold Project claims encompass an area of partially tree-covered hills on the Yukon Plateau, incised by mature drainages that are part of the Yukon River watershed. The property has local mature pine forests with thick moss cover on the ground. Bedrock exposures are scarce (Figure 5.1).

Figure 5.1: Typical Landscape in the Project Area



A. View of the Coffee Gold Project exploration camp looking south

B. Core yard at Coffee Gold Project camp looking north

C. Active core drilling at Double Double looking southeast

D. View looking west-southwest towards Latte from the Supremo Zone

E. Aerial view, looking northwest, of the Espresso, Kona, Latte, Supremo and Double Double mineralized zones.

Source: Kaminak 2016

6 Property History

The Coffee Gold Project area has a limited hard rock exploration history and only minor placer activity. The Coffee Gold Project site has experienced sporadic exploration for placer gold from the turn of the last century until 1981. Prior to 1981, hard rock exploration in the area was limited to a period of reconnaissance in the 1960s and 1970s for porphyry copper.

C.D.N. Taylor, P.Eng., reported that soil and silt samples collected from Coffee Creek, near the confluence of the Yukon River, contained “uniformly high, double digit arsenic values.” Taylor recommended that Coffee Creek be re-sampled during low water table levels (Jaworski and Meyer, 2000; Taylor, 1981).

Deltango Gold Ltd. conducted silt and soil sampling in 1999 in the area of the Coffee Gold Project claims and recommended further work, based on anomalous results (Jilson, 2000). During 1999 and 2000, a brief exploration program was conducted by Prospector International Resources. This program involved stream sediment sampling of secondary drainages, contour and ridgeline reconnaissance soil sampling, rock sampling of available outcrop and prospecting pits, and minor fluid inclusion work. The 1999 work, at a wide sample spacing, identified an open-ended soil gold anomaly. The 2000 work further delineated the extent of this anomaly to be approximately 400 by 900 m, resulting in the recommendation that further soil sampling be undertaken together with mechanized trenching (Jaworski and Meyer 2000; Jaworski and Vanwermeskerken 2001).

In 2006 and 2007, Ryanwood Exploration conducted grid sampling and ridge-top soil sampling traverses on the Coffee Gold Project claims (Ryan, 2007; Ryan, 2008).

In June 2009 Kaminak executed an option agreement with Mr. Shawn Ryan to acquire the Coffee Gold Project. Following this agreement, Kaminak expanded the soil sampling grid in the Coffee areas, developing targets at Supremo, Latte, Kona, Espresso and Double Double. Trenching, geological mapping, and prospecting were conducted at all of these target areas. Kaminak pursued drilling programs from 2010 through 2015 on Supremo, Latte, Double Double, Kona, Espresso, Americano, and Sugar.

The exploration and drilling activities completed by Kaminak from 2009 to 2015 are discussed in Section 9 and 10.

7 Geological Setting and Mineralization

This chapter is a summation of the geology and mineralization textures of the Coffee Project, with specific focus on the deposits which contribute to the resource, namely Supremo, Latte, Double Double, and Kona.

7.1 Regional Geology

The Coffee Project is located in the Yukon-Tanana Terrane (YTT), an accreted pericratonic rock sequence that covers a large portion of the Omineca Belt in the Yukon and extends into Alaska and British Columbia. The YTT underlies part of the Tintina gold belt and hosts multiple gold deposits, including the Sonora Gulch gold deposit, the Casino copper-gold-molybdenum porphyry, the Boulevard gold prospect, and the Golden Saddle gold deposit (Bennett et al., 2010; Allan et al., 2013). The YTT also hosts volcanogenic massive sulphide (VMS) and Mississippi Valley-type (MVT) deposits (Figure 7.1).

The YTT is composed of a basal metasiliclastic sequence overlain by three subsequent volcanic arcs. The oldest component of the YTT is the Snowcap assemblage which was deposited prior to the Late Devonian, which consists of metasediments including psammitic schist, quartzite, and carbonaceous schist in addition to local amphibolite, greenstone, and ultramafic rocks (Piercey and Colpron, 2009). The Snowcap assemblage was deposited on the ancient Laurentian margin in a passive marine setting (Piercey and Colpron, 2009). The beginning of eastward subduction of the paleo-Pacific plate led to the formation of a magmatic arc at approximately 365 Ma (Colpron et al., 2006a). Rapid westward slab rollback caused significant extension, which initiated the formation of the Slide Mountain Ocean back-arc basin by approximately 360 Ma (Colpron et al., 2007). Arc volcanism during the Wolverine-Finlayson magmatic cycle (365-342 Ma) deposited submarine mafic and felsic volcanic rocks of the widespread Finlayson assemblage onto the Snowcap assemblage (Colpron et al., 2006b).

A reversal of subduction polarity during the Late Permian resulted in the western margin of Slide Mountain Ocean subducting beneath the evolving YTT (Erdmer et al., 1998). This subduction initiated a magmatic arc which was active from 269-253 Ma and formed the Klondike arc assemblage, the youngest member of the outboard YTT (Allan et al., 2013; Colpron et al., 2006a). Closure of the Slide Mountain Ocean by the Latest Permian to Early Jurassic led to the obduction of the YTT onto the Laurentian margin, causing a collisional event responsible for lower amphibolite facies metamorphism in the Coffee Project area (Beranek and Mortensen, 2011). In addition, collision resulted in the development of a low-angle transpositional foliation recognized throughout the YTT (S2 of Berman et al., 2007).

Following accretion of the YTT onto Laurentia, easterly subduction caused intra-arc shortening and compressional deformation. In the Klondike and the area of the Coffee Project, thrust fault-bounded panels of Slide Mountain assemblage greenstone and serpentinized ultramafic occur within the tectonic stratigraphy of the YTT (Buitenhuis, 2014; MacKenzie et al., 2008).

These thrust-emplaced slices are generally less than 100 m in thickness, dip to the southwest, and persist for tens of kilometres in some areas (MacKenzie and Craw, 2010 and 2012). The emplacement of these slices is contemporaneous with northeast-vergent, open to tight folding dated between 195 and 187 Ma (Berman et al., 2007).

Beginning in the early- to mid-Cretaceous, localized rapid uplift and exhumation occurred throughout the YTT in Yukon and Alaska, including the Dawson Range (McCausland et al., 2006; Dusel-Bacon et al., 2002; Gabrielese and Yorath, 1991). Extension and unroofing of the rocks of the Dawson Range was accompanied by the emplacement of the Coffee Creek granite and Dawson Range batholith (~110-90 Ma; McKenzie et al., 2013; Wainwright et al., 2011; Colpron et al., 2006; Mortensen, 1992). This localized extension and exhumation is recorded by an apparent age-resetting event observed in white mica in western Yukon-Tanana at roughly 90 Ma (Douglas et al., 2002), in rhenium-osmium dates in molybdenite (92.4 Ma), and U-Pb dates in monazite (92.5 Ma) from plutons in east-central Alaskan YTT (Selby et al., 2002). At the Coffee property, this extension resulted in the activation of the Coffee Creek fault system, a set of dextral strike-slip faults and associated north-to-northeast brittle faults interpreted as splays off of the regional Big Creek fault to the south-east (Sánchez et al., 2013; Johnston, 1999).

7.2 Property Geology

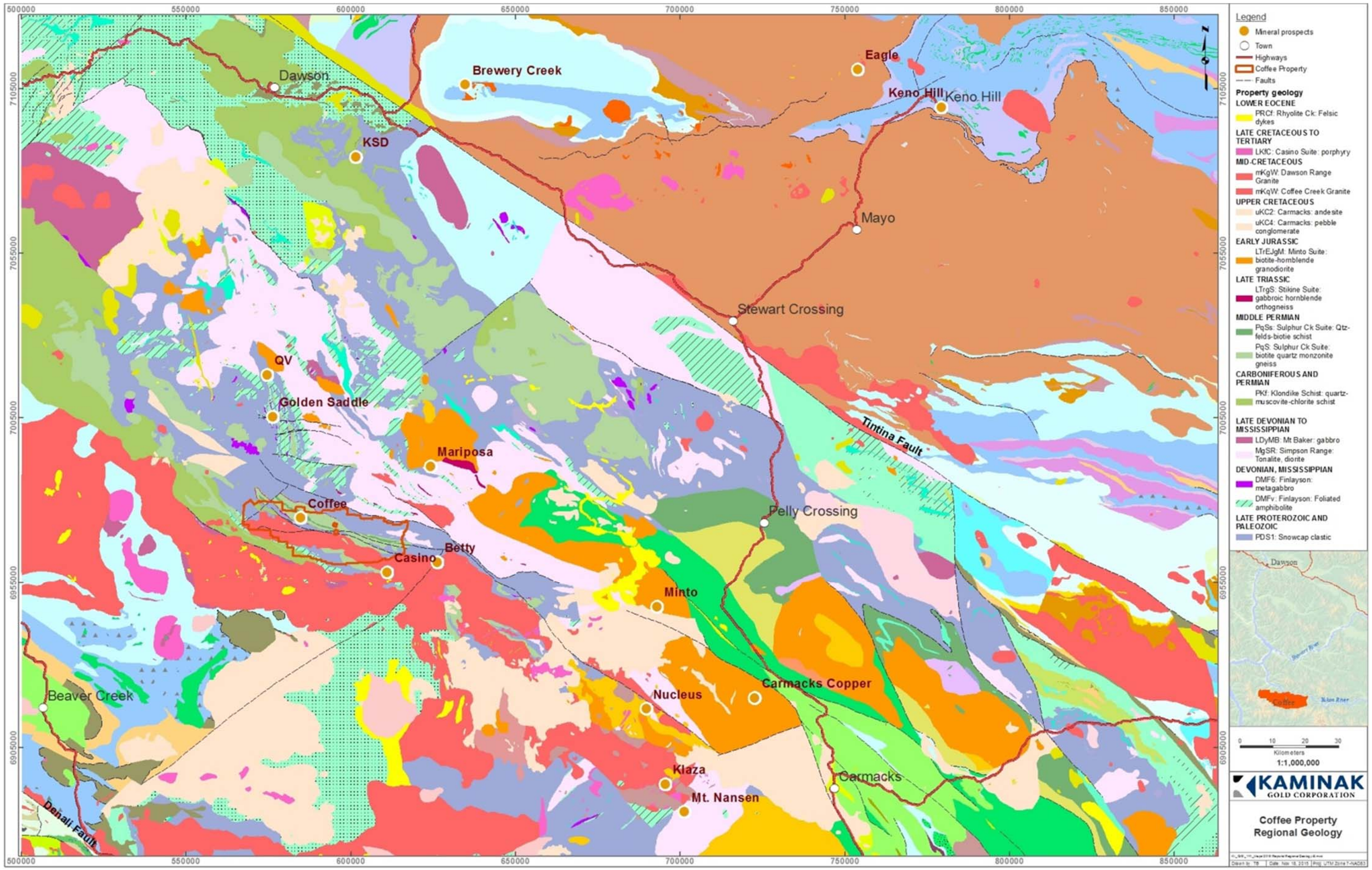
The Coffee Project area is underlain by a package of metamorphosed Paleozoic rocks of the YTT that was intruded by a large granitic body in the Late Cretaceous. The Paleozoic rock package consists of a mafic schistose to gneissic panel which overlies the Sulphur Creek orthogneiss. Both packages form the southwestern limb of a northwest-trending antiformal fold with limbs dipping shallowly to the northeast and southwest.

The schistose and gneissic mafic rock package comprises a thick panel of biotite (+ feldspar + quartz + muscovite ± carbonate) schist with rare lenses of amphibolite which overlies a panel of amphibolite and metagabbro with arc-derived geochemical signatures. Within the schistose panel, slices of 20 m thick serpentinized ultramafic are in tectonic contact with the surrounding rocks. This rock sequence overlies the augen orthogneiss. These rocks are in contact to the southwest with the 98.2 ± 1.3 Ma Coffee Creek granite. Both the Paleozoic metamorphic rocks and Cretaceous granite are cut by intermediate to felsic dykes of andesitic to dacitic composition.

Due to only rare outcrop exposure on the property (< 5%), the geological map (Figure 7.2 and Figure 7.3) has been compiled from a combination of geological traverses, bedrock mapping, borehole data, soil geochemistry, and geophysics (magnetic and radiometric).

The magnesium number from soil samples (Mg# = Mg/Mg+Fe) was used to discern mafic from felsic units with the granite being the most felsic, followed by the felsic gneiss. The mafic schist unit was further subdivided into felsic-intermediate schist, biotite schist, amphibolite, and ultramafic rocks (Table 7.1).

Figure 7.1: Geological Setting of the Coffee Gold Project Area



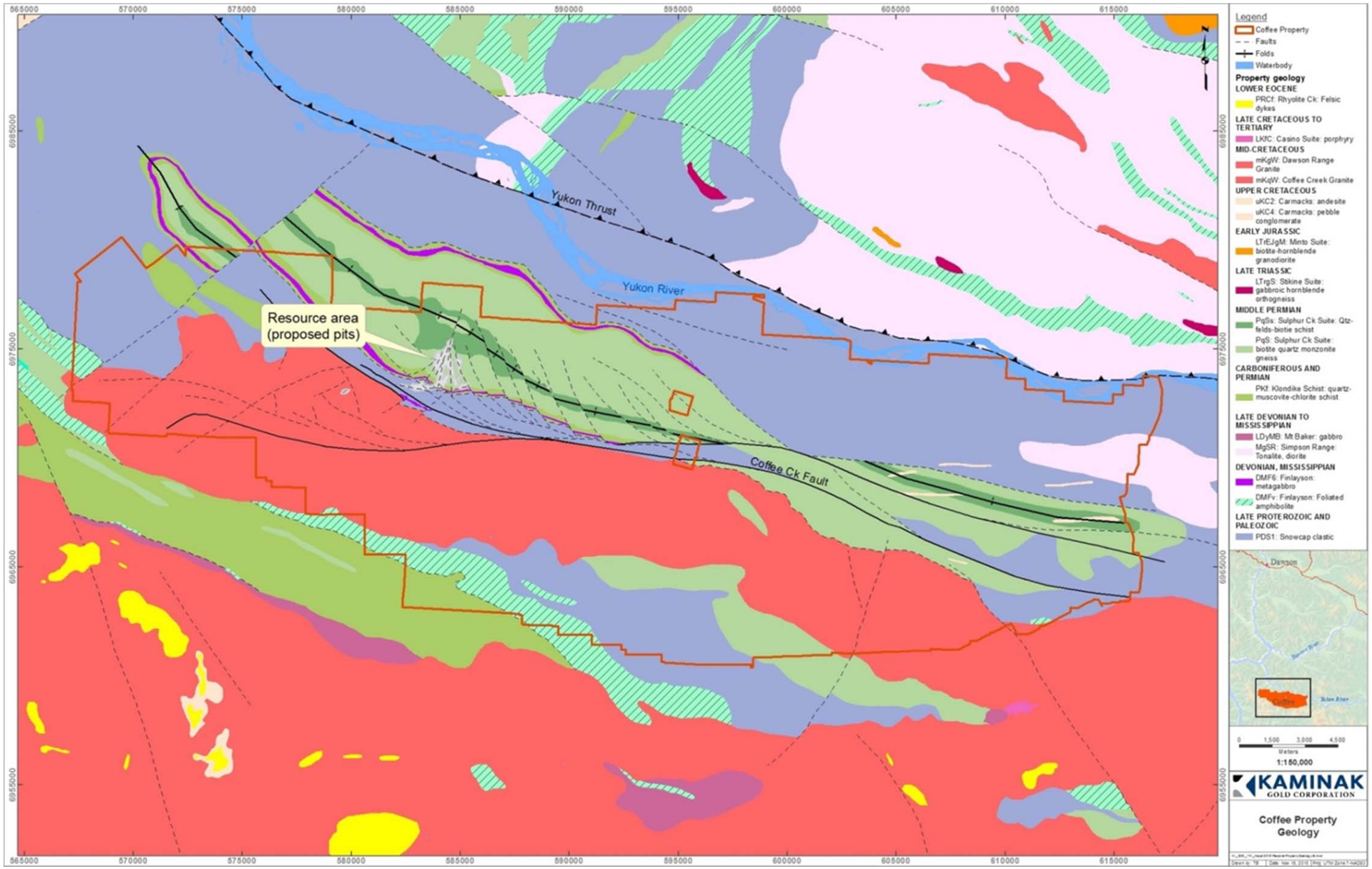
Source: Grodzicki, K. R., Allan, M. M., Hart, C.J.R., 2015. Mineral Deposit Research Unit, University of British Columbia, Yukon

Table 7.1: Main Rock Units in the Coffee Gold Project Area

Rock Unit	Description
Felsic Gneiss	Variable quartz + feldspar augen + biotite + muscovite. Typical Mg# 2-28. Low in potassium. Host to gold mineralized zones at Supremo.
Biotite Schist	Biotite+/-feldspar+/-quartz+/-muscovite+/-amphibole. Commonly carbonate-rich. High in potassium. Typical Mg# 20 - 40. Locally mylonitic. Host to gold mineralized zones at Latte.
Muscovite Schist	Mainly quartz + muscovite. Typical Mg# 10 - 20. Locally mylonitic.
Biotite Amphibolite	Amphibole + feldspar + biotite. Typical Mg# 20 - 40. Biotite and amphibole both Fe-rich. Contains up to 20% biotite.
Amphibolite	Found within the lower mafic footwall. Amphibole + feldspar ± biotite. Typical Mg# 30-50, biotite and amphibole more Mg-rich than biotite amphibolite. Contains up to 15% biotite.
Metagabbro/Amphibolite	Interleaved metagabbro with coarse magnesiohornblende + feldspar, and fine-grained, massive amphibolite with >95% magnesiohornblende. Moderate to strong retrogression to actinolite. High Mg content of biotite, amphibole.
Ultramafics	Serpentinite, pyroxenite or listwaenite. Typical Mg# 50 - 73, higher than all amphibolites and metagabbro. Very high in chromium and nickel.
Granite	Coffee Creek granite and Dawson Range batholith. Both are phases of the Whitehorse Plutonic suite and are uranium-rich. Dawson Range batholith higher in Thorium. Both are identifiable using airborne radiometrics.
Dacite Dykes	Quartz + feldspar phenocryst porphyry. Generally strongly silicified and sericitized. Strong spatial association with mineralized gold zones.
Andesite Dykes	Feldspar phenocrystic. Aphanitic in gold-bearing structures where all original textures are destroyed by intense silicification and sericitization. Strong spatial association with mineralized gold zones.

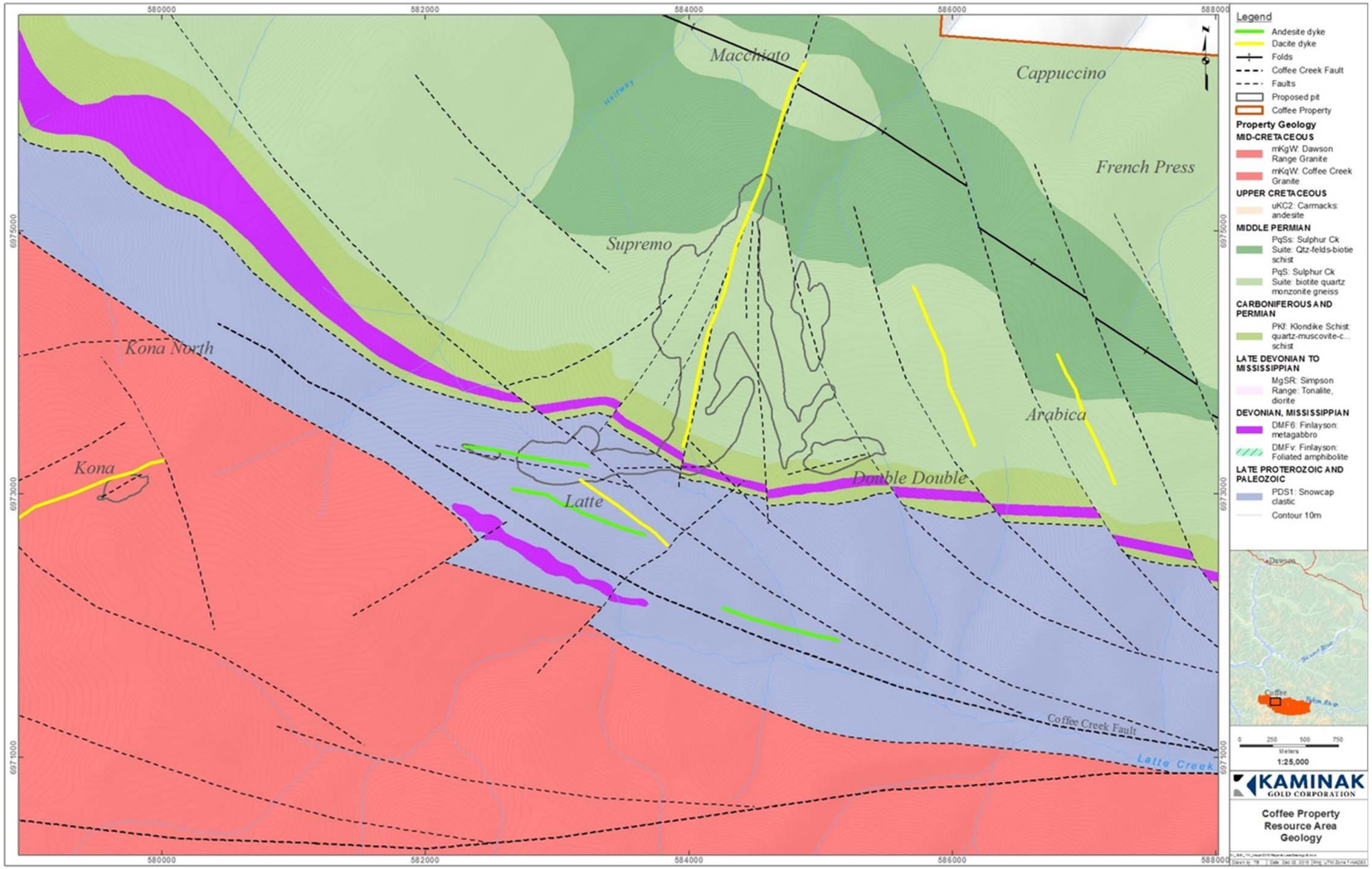
Source: Kaminak 2016

Figure 7.2: Geological Map of the Coffee Gold Project Area



Source: Grodzicki, K. R., Allan, M. M., Hart, C.J.R., and Smith, T. 2015. Geologic Map of the Coffee Gold deposit area, western Dawson Range, Yukon (MDRU Map M-9):

Figure 7.3: Geology in the Supremo, Latte, Double Double, and Kona Areas



Source: Grodzicki, K. R., Allan, M. M., Hart, C.J.R., and Smith, T. 2015. Geologic Map of the Coffee Gold deposit area, western Dawson Range, Yukon (MDRU Map M-9):

7.2.1 Structural Geology

Rocks at the Coffee Project were deformed by a series of three YTT-wide tectonic events (Table 7.2). Gold mineralization at Coffee occurred during the Cretaceous event.

Table 7.2: Tectonic Events at Coffee

Event	Age	Structures	Mineralization
Extension	Cretaceous	Brittle Fractures	Main Coffee Gold mineralization
		Dextral normal faults	
YTT-Laurentia Collision	Jurassic	East-west shears and thrust faults	Quartz veining, sericite alteration
		Slices of ultramafic rocks	
Klondike Orogeny	Pre- to late-Permian	Metamorphic gneissosity and schistosity	

Source: Buitenhuis, 2014; Mackenzie and Craw, 2013; Berman et al., 2007

7.2.1.1.1 Metamorphic Foliation

Gneissose and schistose metasedimentary rocks at Coffee contain a shallowly-to-moderately southwest dipping penetrative cleavage, as described for each lithology in section 6.2.1 (S2 foliation of Berman et al. (2007)). The foliation becomes steeper-dipping to the south. Structural data collected from oriented drill core show the following average orientations:

- Supremo: 20-40° dip to the south-southwest (190-230°);
- Latte: 35-55° dip to the south-southwest (180-210°); and
- Double Double: 35-65o dip to the south-southwest (170-200°).

7.2.1.1.2 Jurassic Shearing

As the YTT-Laurentia collision continued and the Slide Mountain Ocean was completely closed, the rocks in the Coffee area developed roughly east-west brittle-ductile shears and younger rocks were thrust north over older rocks. This deformation corresponds to the D_3 deformation of Berman et al, 2007. This deformation is best seen in the more mafic rocks of the southern schistose panel where intervals of mylonitic rocks are traceable between multiple sections.

7.2.1.1.3 Brittle Fracturing and Faulting

Following post-collision uplift and erosion in the YTT, steep-to-vertical brittle fractures and normal faulting affected all lithologies at Coffee. These brittle structures are the hosts to gold mineralization at Coffee. This deformation corresponds to the D_5 deformation of Berman et al. (2007). The faults and fractures are splays of the regional Big Creek fault to the southeast of the property. The faults may have locally followed pre-existing Jurassic shear zones. The faults both deflect along the northern edge of the Coffee Creek granite and cut the granite and therefore are syn-to-post granite emplacement (~98 Ma). Younger dacite and andesite dykes intruded into these brittle fractures.

Gold mineralized structures comprise strike-extensive planar zones exhibiting a continuum of deformation intensity from crackle breccia/stockwork fracture systems through to polyphase high-energy matrix-supported breccias with intensely altered and reworked clasts. Individual mineralized structures exhibit localized flexures, anastomosing patterns and pinch and swell geometries over scales of tens to hundreds of metres. Overall however, gold mineralization, accompanied by elevated arsenic and antimony, wallrock alteration, deformation intensity, the presence of sub-parallel pre-mineralization dykes, and post-mineral oxidation in the upper 0-300 m below surface, display continuity over hundreds of metres in strike and dip, and over 2 km along strike at Supremo T3 and Latte.

Structural measurements of vein orientations and deformation fabrics from oriented drill core provide hard evidence on the structural geometries, but are often not available in the mineralized zones due to the disaggregated nature of fractured and often clay-altered core. Where intact core is able to be measured, various structural fabrics from within mineralized zones are used to measure local orientation of mineralization and guide 3D geometric interpretation of mineralization on section, and from section to section. Fabrics measured include the dominant fracture orientation, internal fracture or shear fabric, breccia margin, and vein or dyke margin orientation.

The planar gold mineralized zones at Coffee exhibit a number of strike orientations, dominated by east-west, north-south, and east-northeast–west-southwest strike directions. Structures typically have sub-vertical dip, with the exception of western Latte which dips 60-70° south.

7.3 Mineralization

Exploration drilling completed between 2010 and 2015 led to the discovery of significant gold mineralization in over 15 separate areas of the Coffee Project: Supremo, Latte, Sumatra, Arabica, Double Double, Americano West, Americano, Espresso, Kona, Kona North, Macchiato, Cappuccino, French Press, Dolce, and Sugar. Mineralization textures are described below for the four deposits which comprise the Coffee resources and reserves, namely Supremo, Latte, Double Double, and Kona (Table 7.3).

Table 7.3: Main Mineralized Zones Investigated by Drilling on the Coffee Gold Project Area

Zone	Host Rocks	Summary Description
Supremo	Augen Gneiss	Narrow gold-bearing brittle fault structures with gold hosted in intense fracture zones, immature clast supported breccia and in zones of most intense deformation matrix-supported breccia. Mineralization commonly on the margin of and within dacite dykes which intruded along the fracture zones pre-mineralization. Gold mineralization and accompanying quartz-sericite-pyrite alteration associated with later reactivation of structures. Complete oxidation up to 250m below surface.
Latte	Biotite-feldspar Schist, Augen Gneiss	Gold is hosted in zones of brecciation and strong fracturing as well as areas with pervasive sericite alteration and strongly disseminated sulphides. Some high-grade zones associated with quartz vein breccias. Dolomite-illite/sericite-arsenian pyrite sulphidizes foliaform biotite laths. Potential remobilization of gold to other structures. Complete oxidation up to 75m below surface.
Double Double	Augen Gneiss, Biotite-feldspar Schist	Narrow gold-bearing brittle structures hosted in matrix-supported breccia including dacite porphyry fragment breccia. Anastomosing quartz vein networks and microbreccia associated with high-grade. Local intense silicification and strong disseminated sulphide mineralization. Complete oxidation up to 350m below surface.
Kona	Granite	Broad zones of fracture-controlled and disseminated pyrite associated with dacite dykes. Gold hosted in quartz-sericite altered granite. Iron oxides after disseminated pyrite, pyrite veinlets, stockworks and sooty-pyrite rich shear zones.

Source: Kaminak 2016

7.3.1 Supremo

The Supremo Zone is hosted in the augen gneiss package and consists of a number of interconnected north-to-northeast trending, steeply dipping structures (T1 to T8). The structures are variably spaced, and are known to splay and merge into one another over their strike length. This geometry of the structural zones are defined by linear gold-in-soil anomalies, topographic lineaments and magnetic linear breaks, and ultimately subsurface via extensive drilling and 3D interpretation of lode geometries supported by core oriented structural measurements.

From east to west the main drill-tested T-structures are: T1 - T2 (1,100 m strike length, open North and South), T3 (>3,500 m strike length, open to the north, merges with Latte to the south), T4 (1,650 m strike length, merges with T3 to the north and Latte to the south), T5 (1,850 m strike length, open to the north, merges with Double Double to the south), and T7 (900 m strike length, open North and South).The T-structure gold corridors are 5 to 30 m wide and mineralized intervals are associated with intense illite, kaolinite, and sericite alteration in addition to abundant (typically oxidized) pyrite.

The gold mineralization at Supremo is generally characterized by two distinct styles: brecciated mineralization and biotite replacement mineralization. The highest grades are associated with polyphase hydrothermal breccias (Figure 7.4a).

Breccia textures range from mature matrix-dominant phases with rounded fragments to immature wall rock crackle breccias. Matrix compositions range from incompetent limonite-clay material to strongly silicified material. Angular-to-subrounded clasts range from 0.5 to 3 cm in diameter and consist predominantly of highly silicified fragments and subordinate altered wall rock and dacite porphyry fragments. Brecciated clasts occur locally, indicating multiple phases of brecciation.

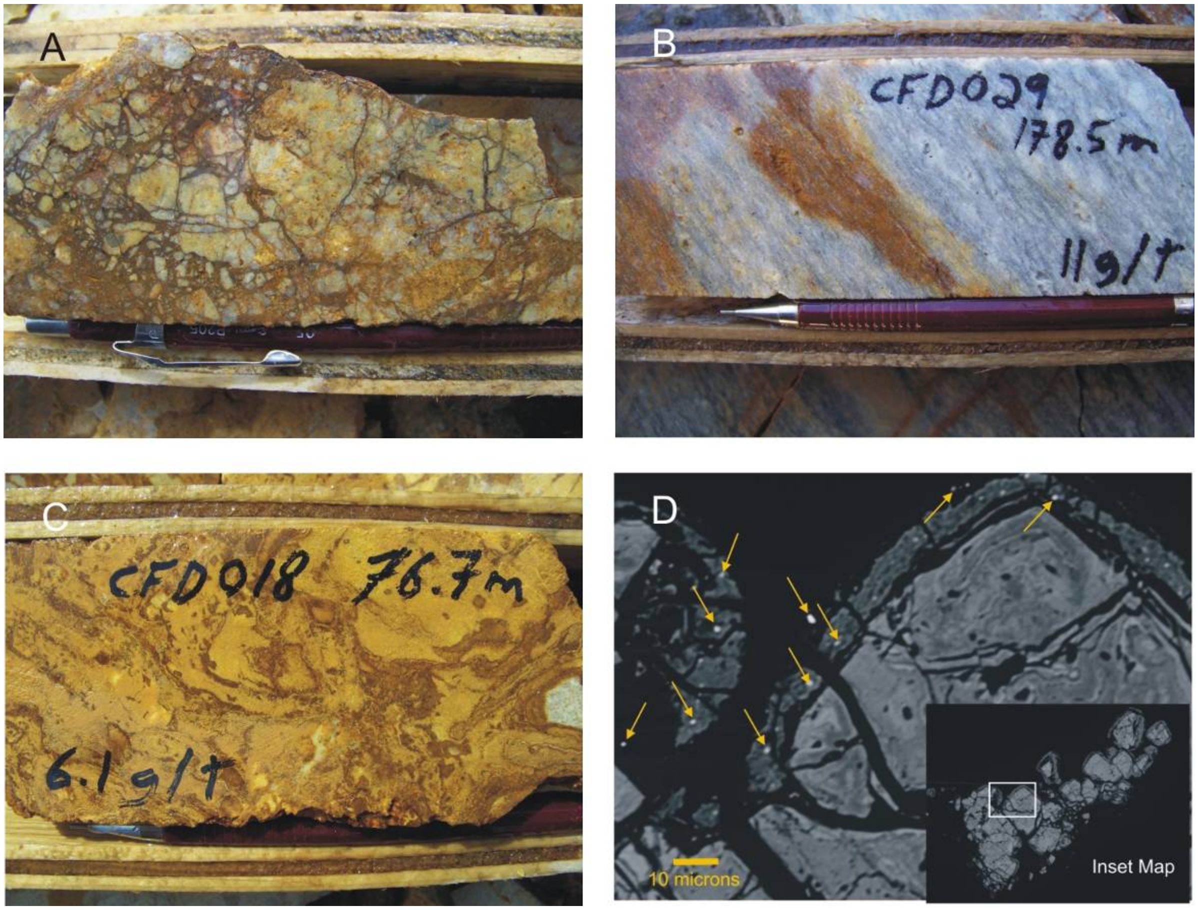
The lower grade gold mineralization is associated with pervasive hydrothermal alteration of non-brecciated gneissic host rock. Biotite is pseudomorphously replaced by pyrite and the hydrothermal alteration is characterized by an overall removal of potassium and aluminum with the addition of sulphide, carbonate, and silica (Figure 7.4b).

Andesite and dacite dykes appear to have utilized the same structures as mineralizing fluids, but they are themselves altered and locally auriferous, therefore they predate mineralization (Figure 7.4c). In other cases, altered dykes with elevated arsenic and antimony are barren. Some dyke margins appear to focus brecciation, potentially due to rheological contrast. The relationship between dykes and the auriferous hydrothermal system remains poorly constrained.

Portable infrared mineral analyser (PIMA), ASD TerraSpec portable infrared mineral spectroscope, and electron microprobe preliminary work indicate that illite and iron-carbonate comprise part of the alteration mineral assemblage associated with gold at Supremo. Micron-scale gold particles are strongly associated with pyrite and free gold grains are found within the oxidized rims and cracks within pyrite grains, in addition to various growth bands within the pyrite grains (Figure 7.4d).

The microscopy and microprobe examination also reveal micron-sized crystals of barite associated with gold and trace amounts of iron-barium arsenate, an iron-calcium-silver-phosphorus mineral phase, monazite, and zircon in alteration zones

Figure 7.4: Gold Mineralization Textures at Supremo



A. Mineralized crackle breccia. Borehole CFD0001, from 19.6 to 20.0 m with 14.35 g/t gold

B. Pervasively altered, auriferous augen gneiss. Note the “pitted” appearance of feldspar augen. Borehole CFD0029, from 178 to 179 m with 11.0 g/t gold

C. Mineralized, clay-altered dacite dyke. Borehole CFD0018, from 76 to 77 m with 6.1 g/t gold

D. Backscatter image of pyrite grain in Supremo breccia showing the extremely fine-grained nature of gold (denoted by arrows) and its association with pyrite. Linear trains of gold grains suggest gold was likely precipitated with pyrite and captured within the pyrite structure and later released during oxidation of the pyrite rim. Borehole CFD0001, from 24 to 25 m with 31.9 g/t gold.

Source: Kaminak 2016

7.3.2 Latte

Drilling across an east-west trend of gold-in-soil anomalies at Latte has intersected gold mineralization 0.5 – 1 m below surficial colluvium and soil (Figure 7.5). Latte consists of a stacked set of moderately to steeply south-southwest dipping, east-southeast striking brittle-ductile structures, whereas the Latte North structure splays off from the main Latte structure and dips moderately to steeply to the southeast with a north-easterly strike. No shear fabric or observable high strain indicators are visible in association with the steep and mineralized Latte structures. Drilling has intersected mineralization at depths of up to 450 m below surface and all structures remain open at depth.

Figure 7.5: Expression of the Latte Structure at Surface Looking East (Section 583250mE)

Source: Kaminak 2016

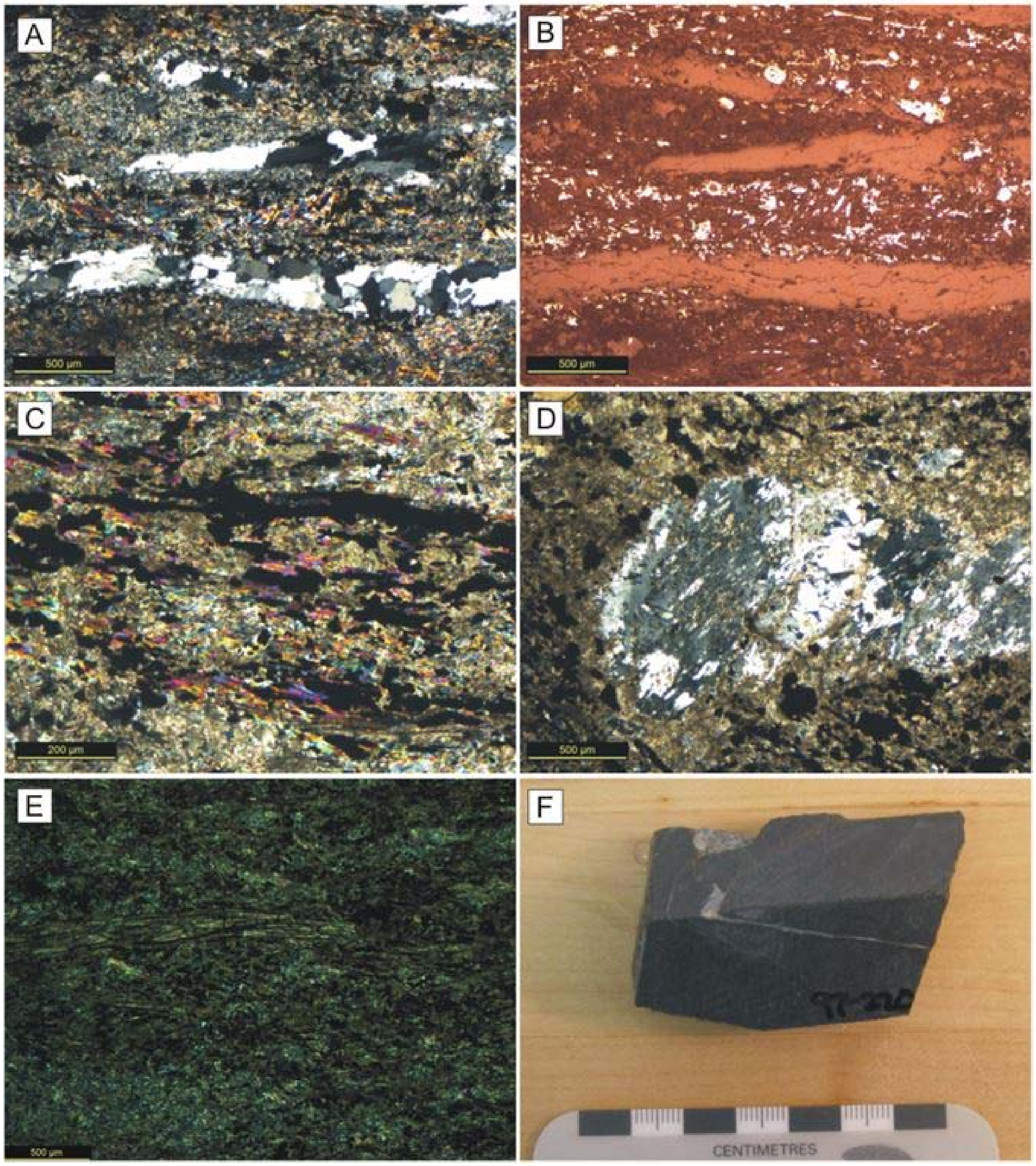
The western portion of the Latte zone is dominated by broad regions of disseminated mineralization found throughout a wide panel of biotite schist. The western structures strike approximately 100˚ and contain five or more mineralized shoots which merge and separate along strike. The structures continue to the east and eventually merge into the Connector zone, where the Supremo north-south structures and the east-west Latte and Double Double structures converge (Figure 7.3). Total traceable length of the mineralized Latte structure is currently in excess of 2,100 m.

Latte North displays identical mineralization textures as the main Latte structure. However, the structures strike at approximately 045˚ and dip approximately 60˚ to the southeast. Latte North splays away from the main Latte corridor for a minimum strike length of 275 m.

Mineralization at Latte consists of disseminated gold-bearing arsenian pyrite, overprinted by later brecciation and late fluid ingress (Figure 7.6). Mica-rich rocks are the main host for gold, with a three phase mineral reaction resulting in gold precipitation. Gold-bearing mineralizing fluid rich in CO_2-As-Sb and S reacted strongly with Fe-bearing biotite within the biotite schist at Latte. A sulphidation reaction proceeds, in which Fe within the biotite is leached to form fine-grained arsenian pyrite, illite, and dolomite which pseudomorphously replace the parent biotite grain. Titanium within the parent biotite is removed and incorporated within hydrothermal illite and rutile.

In high-grade intervals, this reaction has ran to completion with no biotite preserved. Areas which did not experience the same levels of fluid-rock interaction retain relict biotite laths.

Figure 7.6: Disseminated Mineralization within the Latte Zone



A. Finely disseminated arsenian pyrite along relict micaceous foliation. CFD0164 at 464 m, XPL

Same as A, RL

B. Detail of sulphidized mica with fine arsenian pyrite along relict foliation planes. CFD0164 at 469 m, XPL

C. Relatively fresh feldspar porphyroblast surrounded by sulphidized mica and dolomite. CFD0164 at 469 m, XPL

D. Strong disseminations of fine-grained arsenian pyrite. CFD0097 at 220 m, XPL

E. Hand sample image of E

Source: Kaminak 2016

Brecciated intervals are common at Latte, with fine sulphide and clay minerals forming the matrix to angular-to-subrounded clasts of wall rock. These “sulphide-matrix” breccias are generally immature and usually appear as concentrations of very fine “sooty” arsenian pyrite with a steel grey colouration (Figure 7.7). Sulphide content within the matrix of these breccias can be ≥ 20%. These brecciated intervals are best preserved at depth, where oxidative meteoric fluids have not completely altered the matrix to clay and oxidized the contained sulphides. Thin quartz-carbonate veinlets containing extremely fine gold-bearing arsenian pyrite along their margins are interpreted to be of the same phase as the sulphide-matrix breccias.

Figure 7.7: Core photographs of Latte Zone Pyritic Faults and Sulphide-matrix Breccia

A. Pyritic fault in CFD0169 at 237.8 m

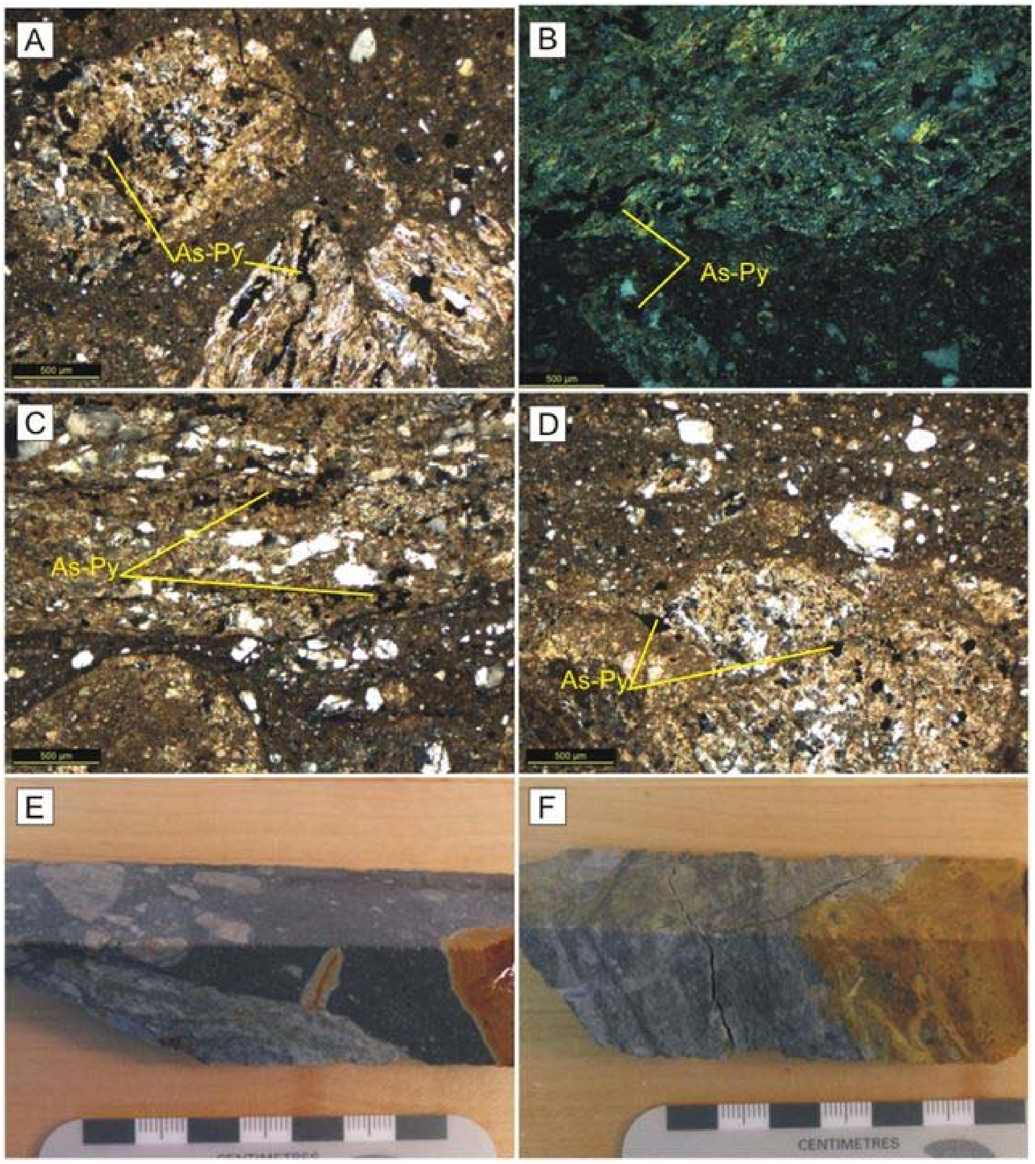
B. Sulphide-matrix breccia from CFD0010 at 122.2 m.

Source: Kaminak 2016

Late breccias are also common, with angular-to-subrounded clasts of mineralized wall rock set in a matrix of rock flour and silica (Figure 7.8). These breccias can be greatly comminuted and locally polyphase.

Within the lower mafic footwall at Latte, amphibole-rich rocks are dominant. Mineralized intervals within this panel are usually restricted to narrow, generally high-grade intersections which represent thin slivers of biotite schist hosted within amphibolite. Amphibole-dominant host rocks do not react with the mineralizing fluid, impeding wall rock sulphidation. The interconnectivity of biotite laths within the schistose rocks promotes reactive fluid flow; laths act as a channel for the sulphidizing fluid, which pervades throughout the schistose host and reacts with the biotite.

Figure 7.8: Late Brecciation of Mineralized Intervals at the Latte Zone



A. Clasts of mineralized wall rock hosted by a super-fine rock flour/silica matrix. CFD0114 at 162 m, XPL
B. Same as A, XPL
C. Breccia corridor with super-fine rock flour/silica matrix and mineralized clasts of wall rock. CFD0097 at 30 m, XPL
D. Same as C, XPL
E. Hand sample of A, B
F. Hand Sample of C, D.

Source: Kaminak 2016

7.3.3 Double Double

The Double Double zone trends east-northeast with a known strike length of 600 m. It dips steeply to the north and consists of a number of discrete, high-grade strands of mineralization up to several metres wide. Host rocks are augen-bearing gneissic rocks with interleaved biotite-feldspar-quartz (± muscovite ± amphibole) schist. The gold mineralization at Double Double is structurally controlled, and may be associated with a north-easterly trending splay off the main Latte structure.

Gold-rich intervals at Double Double are characterized by relict schistose to mylonitic textures overprinted by mottled silica and sericite alteration in addition to limonite-filled micro fracture networks and oxidized pyrite cubes. Breccia domains locally exceed 50% by volume within gold zones, characterized by silicified fragments as well as strongly altered wall rock and porphyry dyke clasts (Figure 7.9a).

Some of these fragments exhibit rounding and imbrication in addition to textures consistent with re-fragmentation of earlier breccia events (i.e. polyphase breccia). Networks of anastomosing chalcedonic silica veins with local microbreccia domains within the veins have been noted in the high-grade intervals (Figure 7.9b).

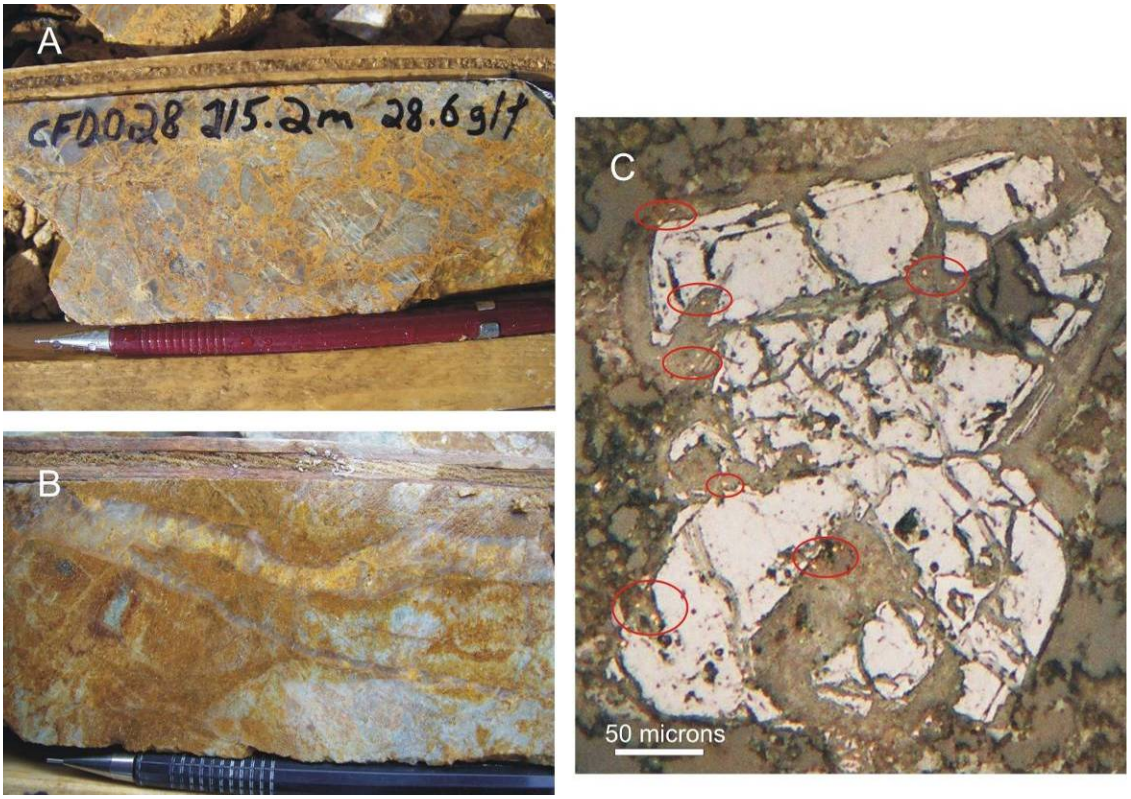
Similar to the Supremo zone, gold at Double Double is micron-scale, and illite has been detected by infrared spectroscopy within the mineralized intervals (Figure 7.9c). Other alteration minerals observed at Double Double include sericite, epidote, leucoxene, hematite, and carbonate.

7.3.4 Kona

Drilling in the Kona zone encountered both biotite replacement and breccia hosted mineralization within the Coffee Creek granite. The Kona zone is hosted within coarse grained equigranular biotite monzogranite, and consists of 1-3 east-northeast trending, steeply south-dipping fault structures. The gold structures are associated with narrow, less than 5 m wide, sparsely feldspar phenocrystic to aphanitic andesite to dacite dykes.

Alteration typically consists of sericite, clay and limonite, with illite being detected during reconnaissance PIMA work at Kona. Sulphide is dominated by sooty arsenian pyrite, which typically replaces ferromagnesian minerals (Figure 7.10a), and also occurs as veins/veinlets or fracture fill, and in sulphide-matrix fault breccias (Figure 7.10b).

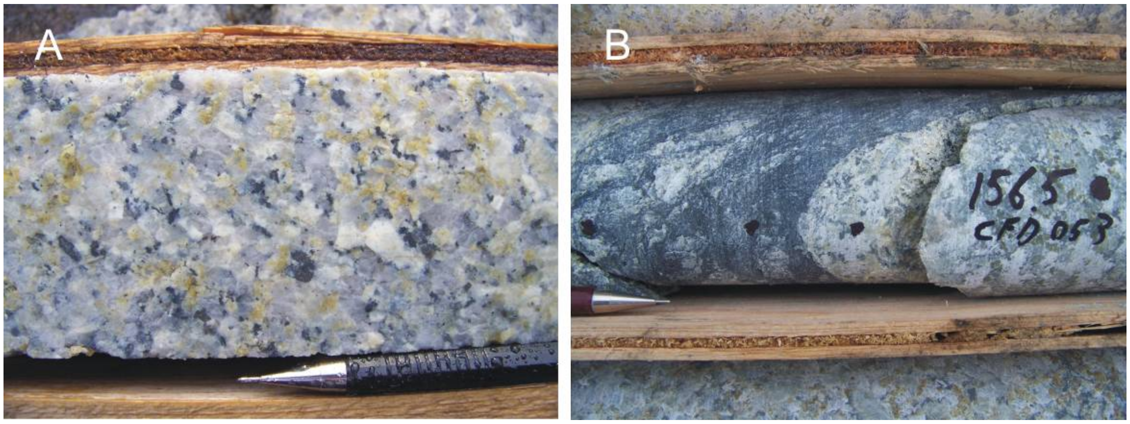
Figure 7.9: Gold Mineralization Textures at Double Double



A. Cement supported silicified-clast breccia. Borehole CFD0028, from 215 to 216 m with 28.6 g/t gold
B. Silica vein network cutting intensely silicified host rocks. Borehole CFD0090, from 105 to 106 m with 120.25 g/t gold
C. Micron-scale gold (circled in red) associated with fractures within pyrite and pyrite grain rims. Borehole CFD0027, from 156 to 157 m with 14.75 g/t gold.

Source: Kaminak 2016

Figure 7.10: Gold Mineralization Textures at Kona



A. Quartz-sericite altered granite; with sulphide mineralization (steel grey mineral) replacement of amphibole and biotite. Borehole CFD0053, from 172 to 173 m with 9.54 g/t gold

B. Sulphide-matrix fault breccia cutting granite. Borehole CFD0053, from 156 to 157 m with 0.94 g/t gold.

Source: Kaminak 2016

7.4 Coffee Weathering Profiles

The mineralized structures at the Coffee Project have undergone extensive preferential weathering and oxidation of iron-bearing minerals as a result of meteoric fluids percolating from surface downwards through the permeable structural corridors. Conversely, unfractured and unaltered country rock is typically fresh (unoxidized) at surface. As a result of this preferential weathering, oxidation is channelized along the structural corridors. Oxidation of mineralized intervals resulted in the breakdown of arsenian pyrite to Fe-oxides such as limonite and hematite. Gold which was structurally bound within the arsenian pyrite coalesces into micron-scale nuggets, which enables rapid and complete gold recovery cyanide leaching.

Oxidation appears to be channeled along the structural corridors that host the deposits. It is common to find intense oxidation at depths below 200 m from surface within these structures. Strong oxidation is present over the majority of the Supremo deposit, but it is less pervasive and more variable at Latte, Double Double, and Kona. As a result, transitional facies material forms a larger proportion of the Latte and Double Double deposits than in Supremo. Outside the interpreted mineral domains, rocks show only weak signs of near-surface oxidation.

7.4.1 Oxide Categorization

Five oxide types or domains, listed below, are designated based on metallurgical test work.

- Oxide zone: intense to pervasive oxidation (>90% oxidation);
- Upper Transition zone: moderate to intense oxidation (70-90% oxidation);
- Middle Transition zone: moderate oxidation (50-70% oxidation);
- Lower Transition zone: weak to moderately oxidized (10-50% oxidation); and
- Sulphide zone: fresh to weakly oxidized rocks (<10% oxidation).

7.4.2 Cyanide Solubility Analyses

In 2013, a comprehensive cyanide shake test re-assaying program was implemented to systematically measure cyanide solubility of gold, which provides a proxy of the degree of oxidation within mineralized rocks at Coffee. Specifically, in addition to measuring the potential cyanide gold leach characteristics of the rocks, the ratio of cyanide soluble (AuCN) gold to total gold (from fire assay) provides information regarding the degree of oxidation.

Cyanide shake tests were conducted on a series of pulp rejects retained from previous drilling campaigns. In many of these older drill holes, AuCN results are available for only select samples. However, , beginning in 2013 cyanide shake testing was routinely performed on all drill holes on a sample-by-sample basis, providing a resolution of 1m in diamond drill core and of 1.5 m in reverse circulation chips. The effective lower detection limit for cyanide shake tests is 0.3 g/t Au, making it only possible to determine the percent recoverability of mineralized samples. This limitation results in tightly constrained recoverability estimates within the mineralized portion of structures.

7.4.3 Three-Dimensional Modelling of Oxidation Surfaces

As described above, Kaminak has conducted cyanide shake testing on the majority of sample intervals that exceed a total (fire assay) gold grade of 0.3 g/t. These data are reasonably distributed but, because they exclude lower grade sample intervals, they are not sufficient to support direct estimation of AuCN estimates in the resource block model. As an alternative, the ratio of AuCn : total Au was calculated in samples where AuCN data are present. These ratios are then interpolated in the block model and are used in combination with qualitative (visual) estimates of the intensity of oxidation to provide information regarding the depth and intensity of oxidation Based on this information, domains have been interpreted that represent the physical distribution of the five oxidation types described previously.

Outside the mineral domains, there is little to no cyanide soluble sample data to assist in defining oxidation states. The type of oxidation present outside the mineral domains is interpreted based on visual observations during drill core and chip logging.

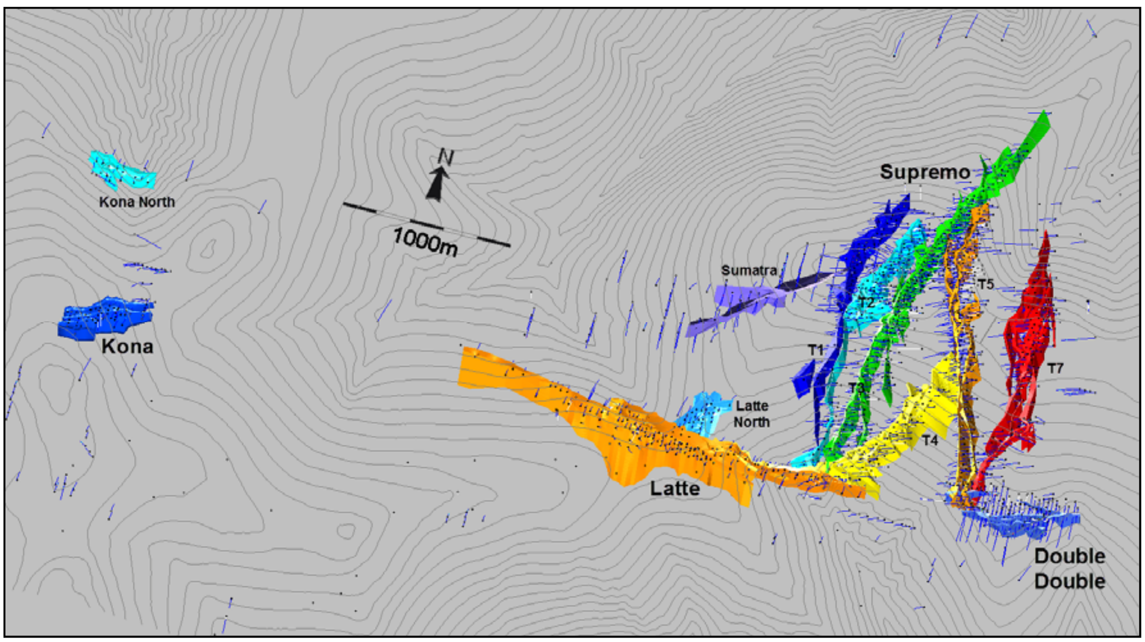
7.5 Three-Dimensional Modelling of Gold Mineral Domains

Gold mineralization at Coffee Gold Project is located within a series of steeply dipping structures that cross-cut all rock units on the property. The structural zones are identified in drill core and from surface mapping and trenching. Soil sampling has also located gold-in-soil anomalies in many areas which were subsequently drilled. Although these structural zones may exhibit faulting, brecciation, silicification, alteration, and local sulphide veining, they can be traced over strike lengths of up to 3.5 km.

Kaminak geologists interpreted a series of “mineralized” or “mineral” domains in each resource area using a combination of surface mapping, drill hole core (and reverse circulation chips) logging, and the distribution of gold grades in drill hole sample data. These domains encompass rocks that exhibit the potential to host gold mineralization and, in most cases, contain elevated gold grades. In previous resource estimations, interpreted “structural” domains relied primarily on geologic conditions that were favourable to potentially host gold mineralization. These domains locally included areas that did not contain appreciable quantities of gold. With the increased density of drilling, resulting from drill holes added during the June 2014 to June 2015 infill drilling program, the confidence in the continuity of gold mineralization between drill holes has increased and the interpretation of these domains now targets the presence of gold mineralization generally above a threshold grade of 0.1 g/t Au. This resulted in a change to the naming convention for domains; previous "structural" domains are now referred to as “mineral” domains.

The distribution of the mineral domains is shown in Figure 7.11. The individual areas at Supremo, (T1, T2, T3, T4, T5, and T7) are named after the trenches that were initially used to investigate the surface mineralization in these areas.

Figure 7.11: Distribution of Mineral Domains



Source: SIM Geological 2016

8 Deposit Types

The Coffee Gold deposit is hydrothermal in origin, structurally controlled and characterized by elevated As and Sb, ± Ag, Bi, U, Hg and Ba. Coffee is interpreted to represent a shallow-level (epizonal) structurally controlled orogenic gold deposit (Buitenhuis et al., 2015; Buitenhuis, 2014; Allan et al., 2013) based on the following criteria:

1. Although mineralization spatially coincides with the Coffee Creek Pluton and intermediate to felsic dykes there is no evidence for direct connection between hydrothermal fluids and a magmatic source;
2. There is a spatial and probably temporal relationship (from field evidence but unconstrained by geochronological data at present) to other mid-Cretaceous gold systems within the wider Dawson Range (e.g. Boulevard and Moosehorn, McKenzie et al., 2013; Joyce, 2002);
3. There is a lack of vertical alteration or metal zoning;
4. Mineralization occurs within subsidiary structures associated with the regional scale Coffee Creek –Big Creek fault system; and,
5. The deposit is characterized by a strong Au-As-Sb association that is typical globally of many orogenic gold deposits.

Recent work in the Dawson Range has demonstrated that both the nearby Boulevard gold showing (10km southwest of Coffee) and the Golden Saddle deposit (40km north of Coffee) are orogenic gold systems (Allan et al., 2013; Bailey, 2013; and McKenzie et al., 2013). Detailed study of the Latte gold zone suggests that Coffee is a shallow (epizonal), brittle stage orogenic gold deposit (Buitenhuis, 2014; Allan et al., 2013). The fluid responsible for mineralization at Latte is likely a cool (220-250˚C), shallow extension of the mineralizing fluid responsible for gold mineralization at Boulevard.

A possible paragenetic model proposed by Buitenhuis et al. (2015) comprises regional CO2-rich fluid flow powered by the anomalous geothermal gradient, in turn caused by the rapid unroofing of the Dawson Range rocks in the mid-Cretaceous. This fluid formed sheeted quartz veins within the mesozonal domain at Boulevard, where the base metal and silica content of the fluid was depleted during vein formation. Base metal depleted fluid migrated upwards into the epizonal domain where it was controlled by the structural framework of the Coffee fault system and reacted with favourable host lithologies. The fluid travelled along brittle structures and deposited gold-rich arsenian pyrite within schistose rocks through sulphidation, and high-energy pulses formed gold-rich hydrothermal breccias.

The timing of gold mineralization at Coffee is post-emplacement of the Coffee Creek granite (100 Ma), and is most likely to be syn-to-post-mineralization at Boulevard (approximately 95 Ma) based on field, geochemical and petrographic observations. The possibility exists, however, that Coffee could be related to a younger, as yet unidentified mineralizing event.

9 Exploration

Kaminak carried out exploration on the Coffee property over the course of seven separate and consecutive field seasons from 2009 to 2015. Exploration carried out from 2009 to 2013 is summarized in detail in previously published technical reports (Couture and Siddorn, 2011; Chartier and Couture, 2012; Chartier et al., 2013; Sim and Kappes, 2014; Makarenko et al., 2014) and described briefly in Table 9.1. Work completed after the 2014 Preliminary Economic Analysis, during the 2014 and 2015 field seasons in support of the Feasibility Study, is described below.

Table 9.1: Exploration Work Completed by Kaminak

Coffee Exploration Summary by Year							
Year	**Drill (m)**	**Soil Samples**	**Trenching (m)**	**Trench Samples**	**Mapping and Sampling (days)**	**Geophysics**	**Geomorphology**
2009	N/A	3,876	4,164	828	10	261 line-km ground magnetic survey	N/A
2010	16,105	8,851	4,470	826	10	579 line-km ground magnetic survey	N/A
2011	47,990	10,689	3,926	799	15	4,842 line-km airborne magnetic and gamma-ray spectrometric; 15.9 line-km HLEM and Ohm mapper surveys	Mapping
2012	65,548	4,438	N/A	N/A	40	N/A	N/A
2013	55,477	5,027	153	147	2	18 days of IP	N/A
2014	52,760	2,955	6,252	2,025	30	5,300 line-km airborne magnetic infill survey	N/A
2015	41,895	N/A	N/A	N/A	30	N/A	N/A
Totals	**279,775**	**36,081**	**18,965**	**4,625**	**137**	**N/A**	**N/A**

Source: Kaminak 2016

9.1 2014 Exploration Activities

9.1.1 Soil Sampling

A total of 2,955 soil geochemical samples were collected in 2014 by Ground Truth Exploration Inc. of Dawson, YT in order to follow up on previously-sampled ridge and spur gold-in-soil anomalies, and provide greater detail to previous sample programs.

A total of eight new soil sample grid areas were sampled. Four of these grids were sampled on 100 m lines with 50 m sample spacing to follow up on previous ridge and spur anomalies or to provide an initial indication to eliminate the possibility that underlying ore potential existed (condemnation) in the vicinity of proposed mine infrastructure. An infill sampling program was undertaken at the Espresso prospect on lines spaced at 100 m and orientated north-south with 25 m sample spacing in order to provide more detail to previously outlined anomalies. In addition, portions of the existing Macchiato, Cappuccino, and T3 North soil grids were sampled to decrease sample spacing to 25 m (Figure 9.1).

9.1.2 Trenching

Six trenches were dug to obtain metallurgical samples, while a further 57 were dug for exploration purposes. Locations of the metallurgical and exploration trenches are illustrated in Figure 9.2. Trenches were dug to provide bulk samples for the metallurgical program. Trenches were dug at the Supremo, Macchiato, Cappuccino, Arabica, French Press, Decaf, Espresso, and Double Double zones to test gold-in-soil anomalies and to generate drill targets. All trenches were laid out by Kaminak geologists and excavated by 320 or 312 Caterpillar excavators operated by JDS.

Analytical results from trench samples confirmed that gold mineralization extends to surface. Although great care and consistency was used to obtain representative samples from these trenches, the type and volume of these samples differs significantly from samples collected from drill holes. As a result, the channel sample data has not been incorporated for use in the generation of the resource model. Assay data generated from trench samples was used to generate exploration targets and the location of samples for metallurgical test work.

Exploration trenches dug early in the 2014 field season occasionally encountered permafrost which limited excavations to surficial material rather than in-situ bedrock. Trenches which were successful in reaching bedrock generally provided lithological, alteration, and structural control, which in combination with assay data was utilized to generate drill targets.

9.1.3 Mapping and Prospecting

Kaminak geologists prospected and sampled at the Supremo, Macchiato, Cappuccino, Arabica, French Press, Decaf, Espresso, and Double Double zones during the exploration trenching program. Additional transects were completed in the southern portions of the property through the Coffee Creek granite while supporting environmental and geotechnical surveys.

An external prospector was contracted for 30 days in June and July of 2014 to examine ground adjacent to areas undergoing infill soil sampling and trenching.

9.1.4 Geophysical Surveys

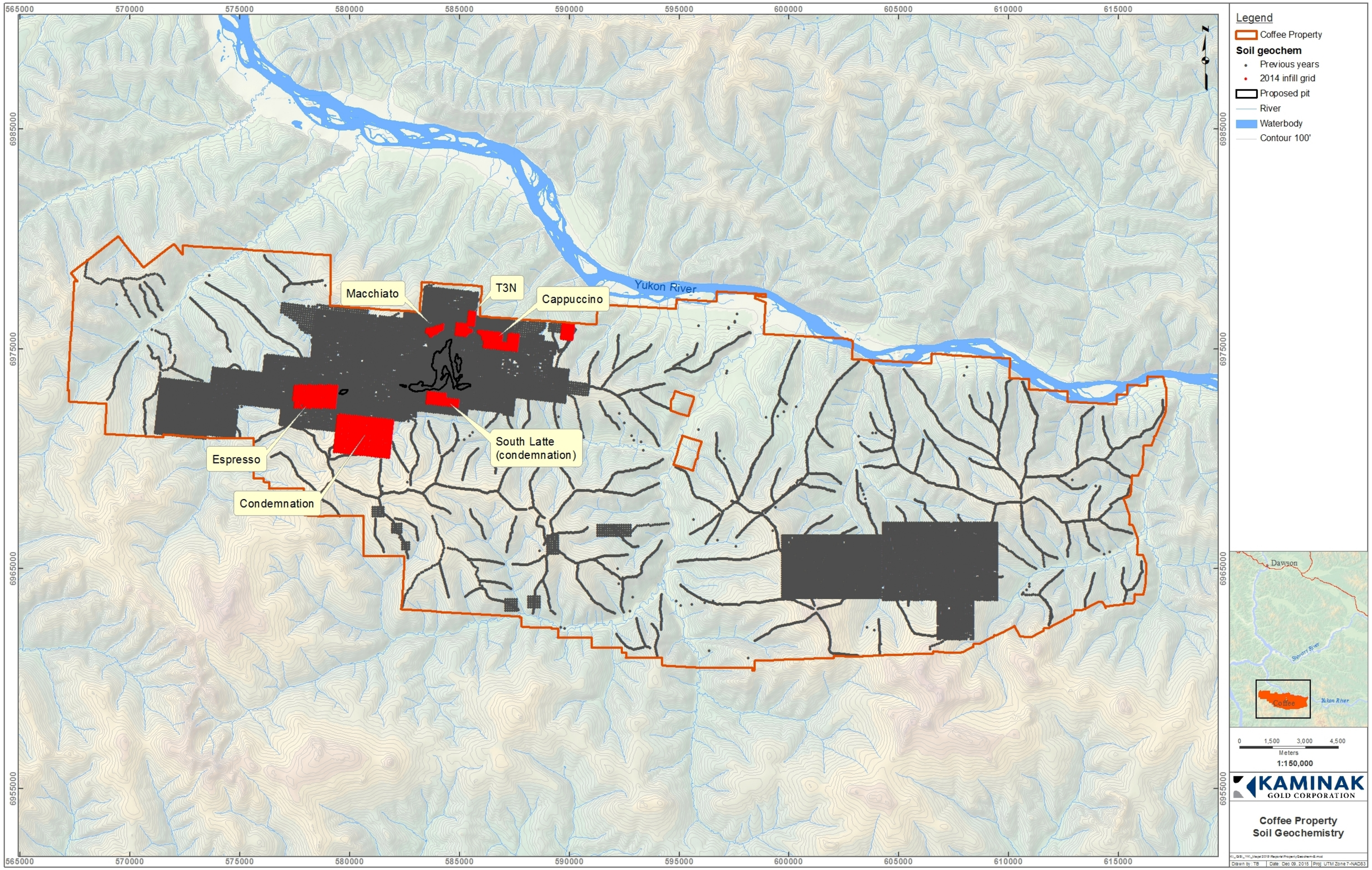
Mira Geoscience completed 5,300 line kilometres of airborne geophysics in a program which infilled the existing magnetic dataset to achieve higher magnetic survey resolution of the entire Coffee property (Figure 9.3).

9.2 2015 Exploration Activities

9.2.1 Mapping and Prospecting

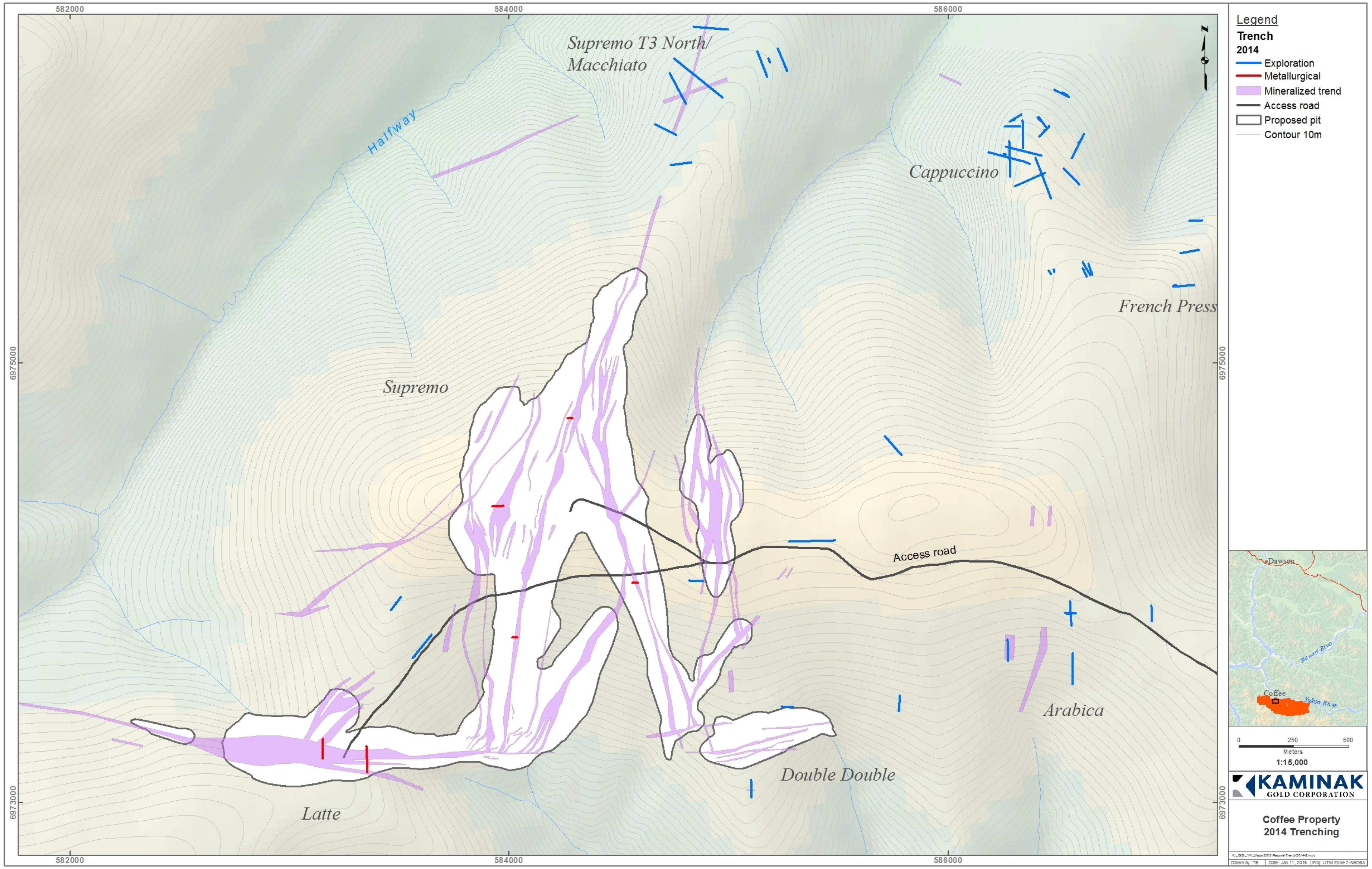
Kaminak geologists completed approximately 30 days of mapping and prospecting on the Coffee property in 2015. This work was conducted outside of the main Coffee resource area and generally followed up on ridge and spur soil anomalies to the south and east (Figure 9.4). Sampling was limited to rock samples from outcrop and colluvial float.

Figure 9.1: 2014 Soil Geochemistry Program



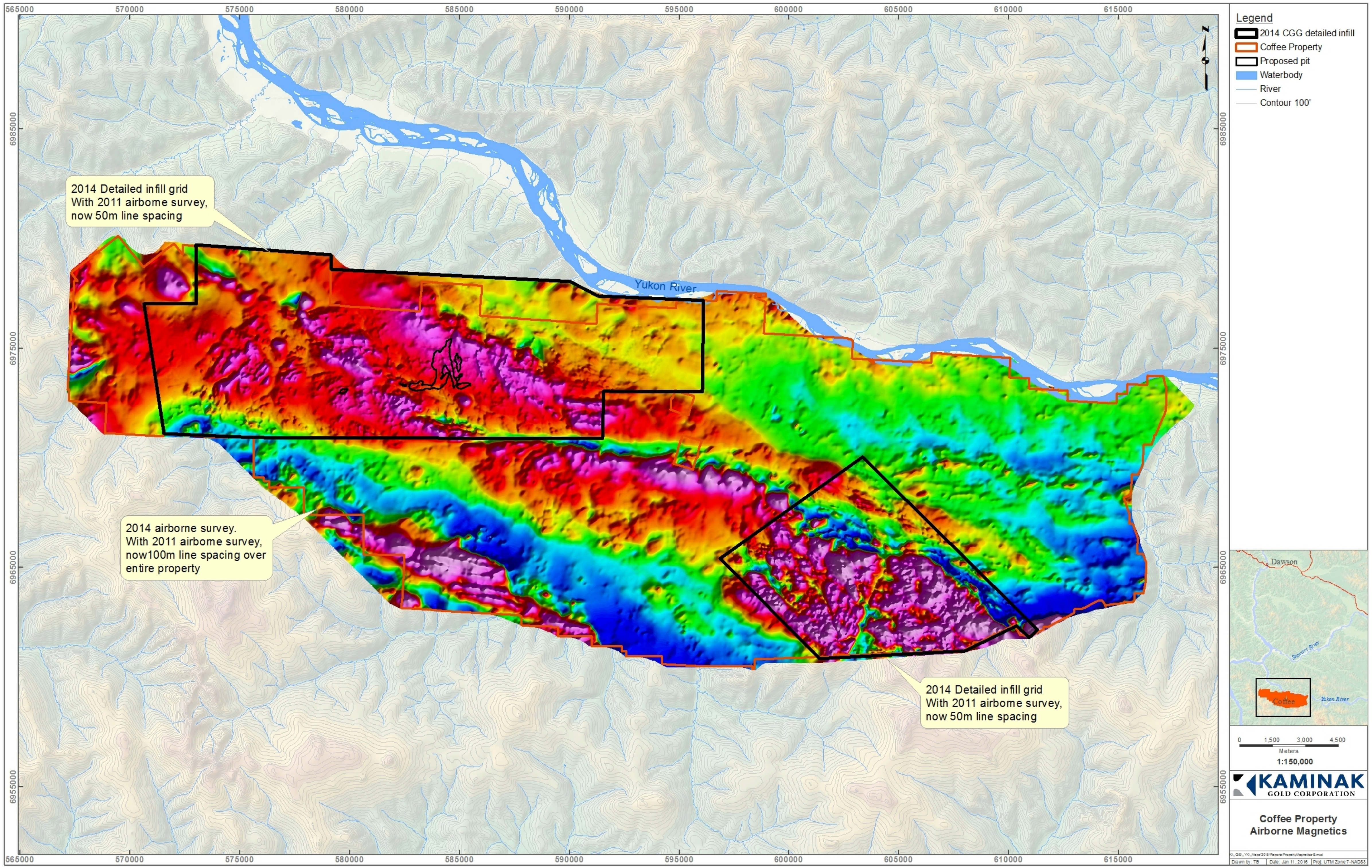
Source: Kaminak 2016

Figure 9.2: 2014 Trenching Program



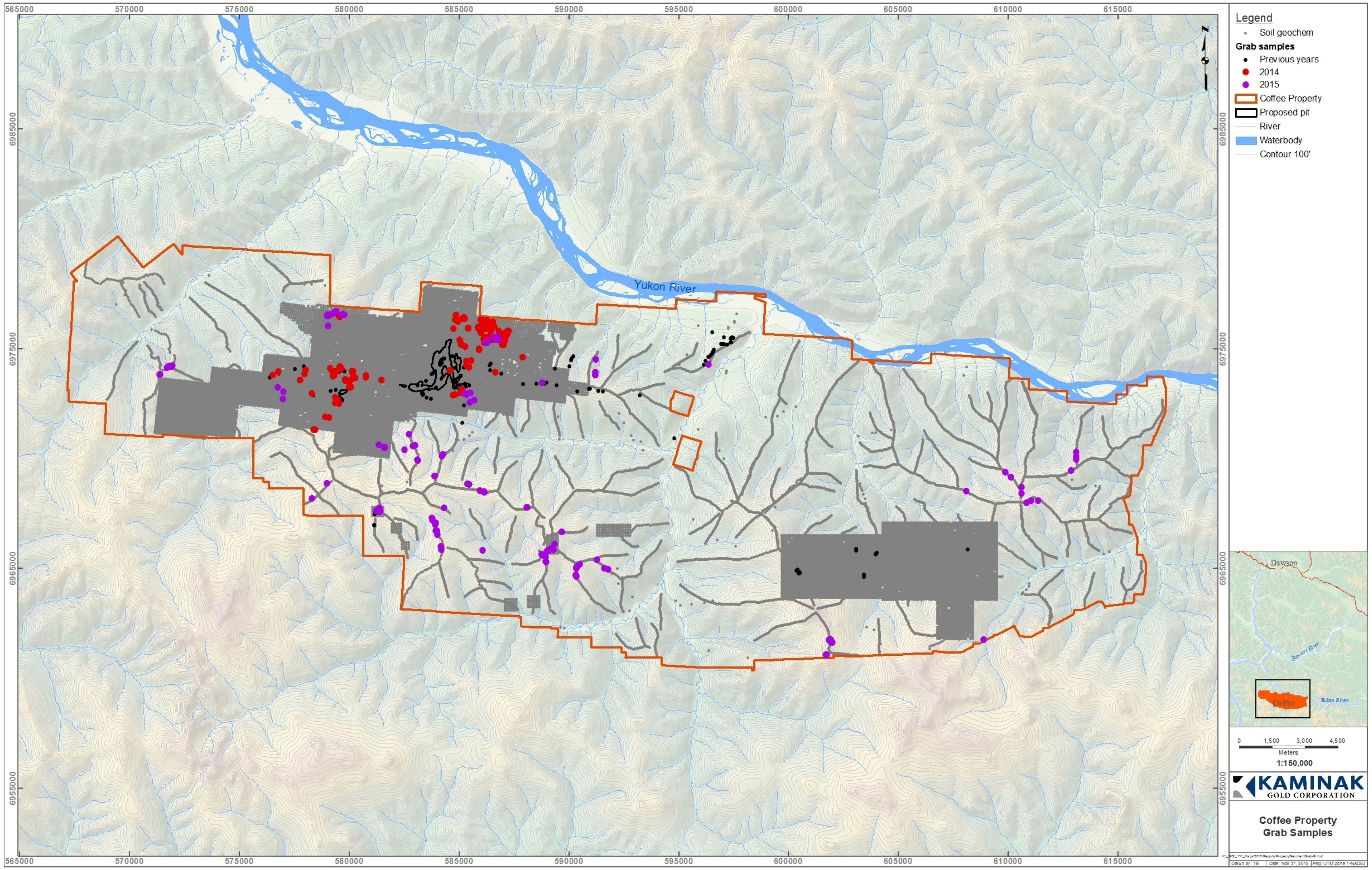
Source: Kaminak 2016

Figure 9.3: 2014 Geophysical Program



Source: Kaminak 2016

Figure 9.4: 2014 and 2015 Prospecting Sample Locations



Source: Kaminak 2016

9.3 Surface Sampling Method and Approach

Sampling undertaken by Kaminak from 2009 through 2015 was performed by experienced geological technicians under the supervision of appropriately qualified geologists. The sampling methodology and approach for the soil and rock chip samples is summarized below.

9.3.1 Soil Sampling

The purpose of the soil sampling was to map the distribution of gold and associated metals which is known to commonly overlie mineralized areas on the Coffee property.

Soil sampling was carried out by Ground Truth Exploration Inc. Soil samples were collected over a grid pattern of north–orientated lines 100 m apart with sampling stations at intervals of 50 m, or 25 m in areas where detailed soil sampling was required.

Samples were collected using a hand auger to various depths depending on the soil profile. The organic A-horizon material was discarded. Augering continued until the C-horizon rock fragments were encountered, checking for false bottoms on the A-horizon profile. Soil samples were collected over depth intervals of 60 to 70 cm, with maximum depth not exceeding the 1.25 m length of the auger. Samples were placed directly into pre-labelled bags. A field duplicate sample was collected at a rate of one in every 25 samples. Sample number, location, depth, and geological parameters were recorded directly into a handheld computer with a Global Positioning System (GPS) sample location. The sample location was marked with flagging tape and a metal tag on the closest tree.

Samples were submitted by the contractor to Acme Analytical Laboratories in Vancouver, BC.

9.3.2 Rock Grab Sampling

Collection of ‘grab’ samples from outcrop and colluvial float by qualified geologists, was undertaken sporadically across the property. Mapping supported by analysis of various lithological, alteration and structurally deformed rocks helped delineate areas for more systematic exploration.

9.3.3 Trench Sampling

The trenches excavated in 2009 to 2012 had composite rock samples taken at 5 m horizontal intervals. Representative samples were collected by chipping sub-cropping rock on the wall or base of the trench. Hand sampling may introduce sampling bias. However, the purpose was to link gold-in-soil anomalies to their bedrock origin and thereby define deeper drilling targets. In such circumstances a positive sampling bias is generally desirable.

For trenching, heavy equipment was used to excavate and rip the 2013 and 2014 metallurgical sampling trenches at 2 m intervals to 0.5 m below the bedrock surface. A similar process was used for exploration trenches.

All metallurgical composite samples in 2013 and 2014 were sampled from metallurgical samples placed alongside the trench. Kaminak geologists collected bulk representative samples from each sample pile in addition to a representative ~4 kg sample for laboratory assay.

The centroid of each sample was recorded using a handheld GPS unit. Other descriptive attributes and geological information was recorded and incorporated into the project database.

10 Drilling

10.1.1 Sampling Method and Approach

Kaminak's sampling of diamond drill core and reverse circulation cuttings, beginning in 2009 through 2015, was performed by experienced geological technicians under the supervision of appropriately qualified geologists. The drilling and sampling procedures described below, have been performed in a consistent manner throughout each drilling program. The following section summarizes the sampling methodology and approach for core and reverse circulation drill holes. Methods for sonic and rotary air blast (RAB) drilling are not discussed herein, as neither contribute to the mineral resource statement.

10.1.2 Drill Core Sampling

Drilling typically targets specific mapped geochemical or structural trends with fences of one or more core drill holes drilled perpendicular to the strike of the interpreted mineralized structures on variably spaced sections or fences. Most cross sections contain two to five drill holes that are designed to intersect the mineralized target horizon at intervals typically ranging from 25 m to 50 m, typically to maximum depths of 200 m below surface. The approach was adjusted during drilling to allow for the testing of extensions of interesting geology, or assay results on adjacent sections. Individual drill holes were completed each from a unique setup, resulting in a series of sub-parallel holes that often intersect at angles roughly perpendicular to the target horizon. The resultant fence of intersections supports a geological interpretation of the geometry and “true” thickness of mineralized zones.

Borehole locations were planned and set out by Kaminak geologists using a handheld GPS. A compass was used to determine borehole azimuth and inclination. Drill holes were drilled at an angle of between 70^{o} and 45° from the horizontal, depending upon the target. Downhole surveys were completed for all drill holes using a Reflex EZ-Shot® electronic single shot (magnetic) device. Downhole deviation of drill holes was measured using these tools at nominal interval of 30 m. Upon completion of drilling, collar locations were surveyed by Challenger Geomatics Ltd. of Whitehorse, YT with a Real Time Kinematic (RTK) GPS using five control points.

Drill core was transported daily by truck or helicopter to the logging facility at the Coffee Gold Project camp. Core was reviewed for consistency and each metre marked clearly for reference. Core recovery and rock quality designation (RQD) were measured and recorded, and the core oriented when possible. XRF analyses were performed on drill core at 1 m intervals, as close to the metre mark as possible. Core was then logged by a geologist who recorded lithology, alteration, structure, and mineralogy directly into a computer. Core photographs were taken prior to sampling. Core samples were taken from half-core sawed lengthwise with a diamond saw. Half-core samples were bagged and prepared for dispatch to ALS Minerals laboratory. The remaining half was returned to the core boxes. Commercially prepared blank and control (standard reference) samples were inserted at a rate of one for every 10 samples, alternating between a blank and a reference material sample. Following sampling, core boxes were labelled with metal tags and stored on cross-stacked pallets at the Coffee Gold Project camp for future reference and testing. Pre-numbered sample books were used to record borehole number, location, sampling interval, and date of sampling. All sample books are organized and archived at Kaminak's Vancouver office.

Diamond core recovery data is available for nearly all drill holes on the Coffee property. The overall average core recovery is 96%, with 95% of all sample intervals demonstrating recoveries greater than 80%. Approximately 1.5% of sample intervals have recoveries less than 50%. There is no apparent relationship between drill core recovery and gold content at Coffee. However, during the 2014 and 2015 infill drill programs, mineralized sections with < 80% recovery were considered for re-drill, and sections with <50% were designated automatic re-drill.

10.1.3 Reverse Circulation Chip Sampling

Reverse circulation drilling was completed on the Coffee Gold Project from 2010 through 2015. The drilling approach was similar to that employed for diamond core drilling; a series of sub-parallel holes designed to perpendicularly intersect the mineralized target horizons at (typically) 25 m to 50 m intervals, depending upon the level of geological confidence of the mineralized trend.

RC drilling produces a sample of rock cuttings rather than rock core. The downhole hammer is powered by compressed air, which also acts as the medium bringing the cuttings up to surface. Compressed air drives a pneumatic hammer attached to a rotating face sampling bit with tungsten carbide nodes. Chips and rock dust generated by the hammer are forced through openings in the face of the bit and up into the sample return tube inside the rod string. The 5-foot rods are attached to an air and sample hose that continues into a cyclone module. The sample is separated from the air in the cyclone and drops out of the bottom into a 5-gallon pail. Each sample comprises one 5-foot (1.52 m) run, with the drill hole and rods being blown out (cleaned) between each “run”. The total volume of cuttings from each run is reduced through a 1:7 riffle splitter, into a sample typically weighing 2 kg which was retained for analysis. The larger volume of reject material was retained at the drill site in plastic retention bags labelled by depth of sample.

The technician collected a small volume of sample chips, sieved from a spear sample of the retention bag for observation and records the geologic properties (lithology, texture, grain size, alteration, colour, etc.) directly into a field laptop. The chips were then logged by a geologist in camp. Sample bags collected for analysis were transported daily by truck or helicopter to the processing facility at the Coffee Gold Project camp. Each sample was then analysed on the XRF instrument before being shipped to ALS Minerals for analysis.

Reverse circulation sample recovery was closely monitored by the driller and supervising geologist or technician. If poor sample quantity or quality was encountered during drilling and if the driller was unable to reinstate the drill hole and achieve adequate sample return, the hole was abandoned and re-drilled. Intervals with poor sample quantity and/or excessive moisture content were logged as such, and the interval was not sampled. The vast majority of RC sample recoveries are generally very good, with qualitative studies showing that recovery averages >85%. While some fine dust is lost to the air or within the drill hole or voids/fractures during drilling, this represents a very small amount of sample material and is not believed to affect sample integrity to a measurable degree.

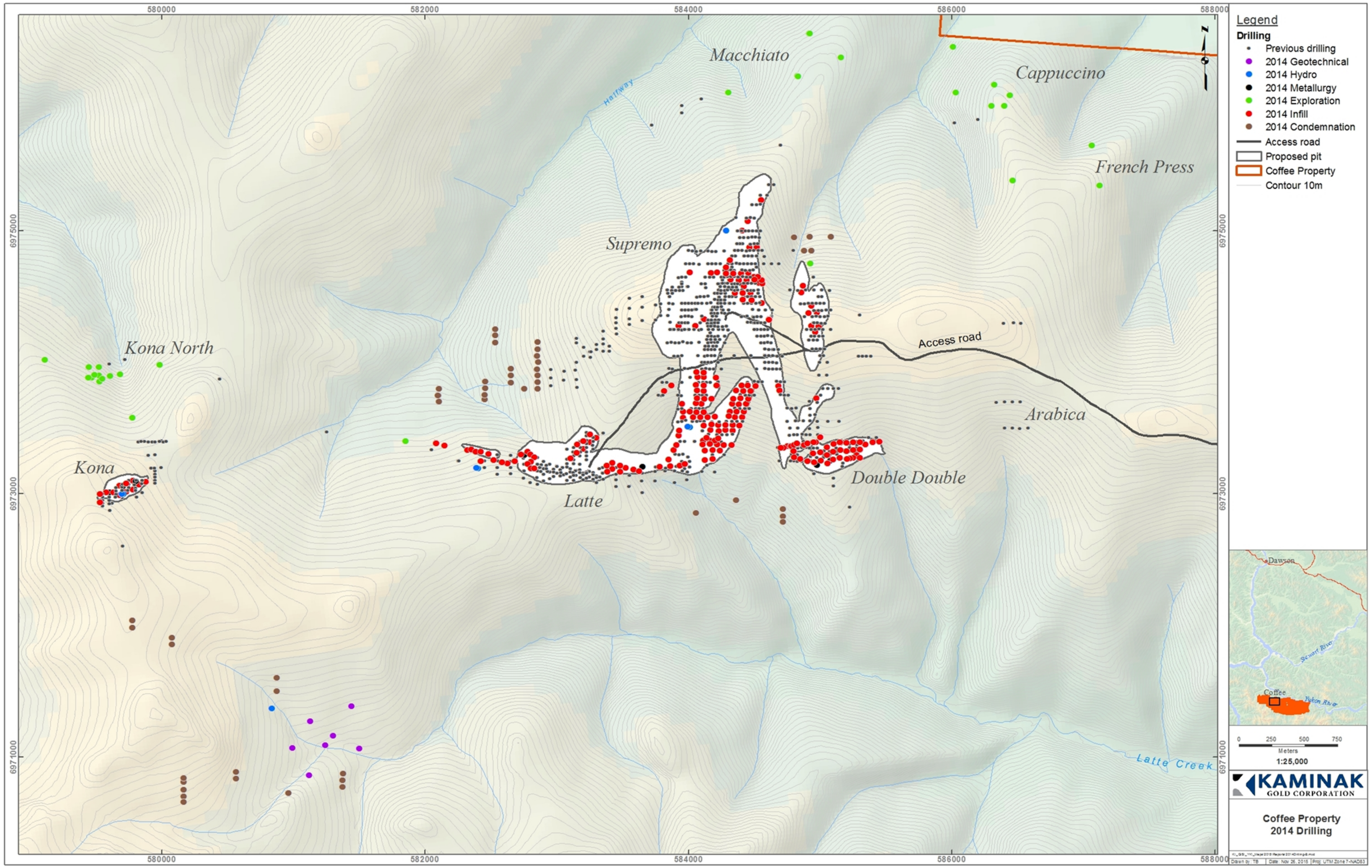
10.1.4 2014 Drilling

During 2014, 353 drill holes (52,760 m) were drilled at Supremo, Latte, Kona, Kona North, Double Double, Cappuccino, French Press, Macchiato, as well as various infrastructure locations for condemnation purposes (Figure 9.5). Of the 353 drill holes, 147 were cored (26,894 m) and 206 were reverse circulation (25,866 m).

Core drilling took place between May and October 2014 and was contracted to Cyr Drilling International Ltd. of Winnipeg, Manitoba. The vast majority of core was NQ2 (50.5 mm diameter), with minor HQ (63.5 mm) and PQ (85.0 mm) core drilled to support the metallurgical and hydrogeological programs.

RC drilling took place between August and November 2014 and was contracted to Northspan Explorations Ltd. All RC boreholes were of 92 mm diameter.

Figure 10.1: 2014 Drill Hole Plan



Source: Kaminak 2016

10.1.5 2015 Drilling

During 2015, 370 drill holes (41,895 m) were drilled at Supremo, Latte, Kona, Kona North, Double Double, and various infrastructure locations for condemnation purposes (Figure 10.2). The 370 drill holes comprised 103 core drill holes (15,840 m), 197 reverse circulation drill holes (23,702 m), 35 rotary air blast drill holes (2,197 m), and 35 sonic drill holes (156 m).

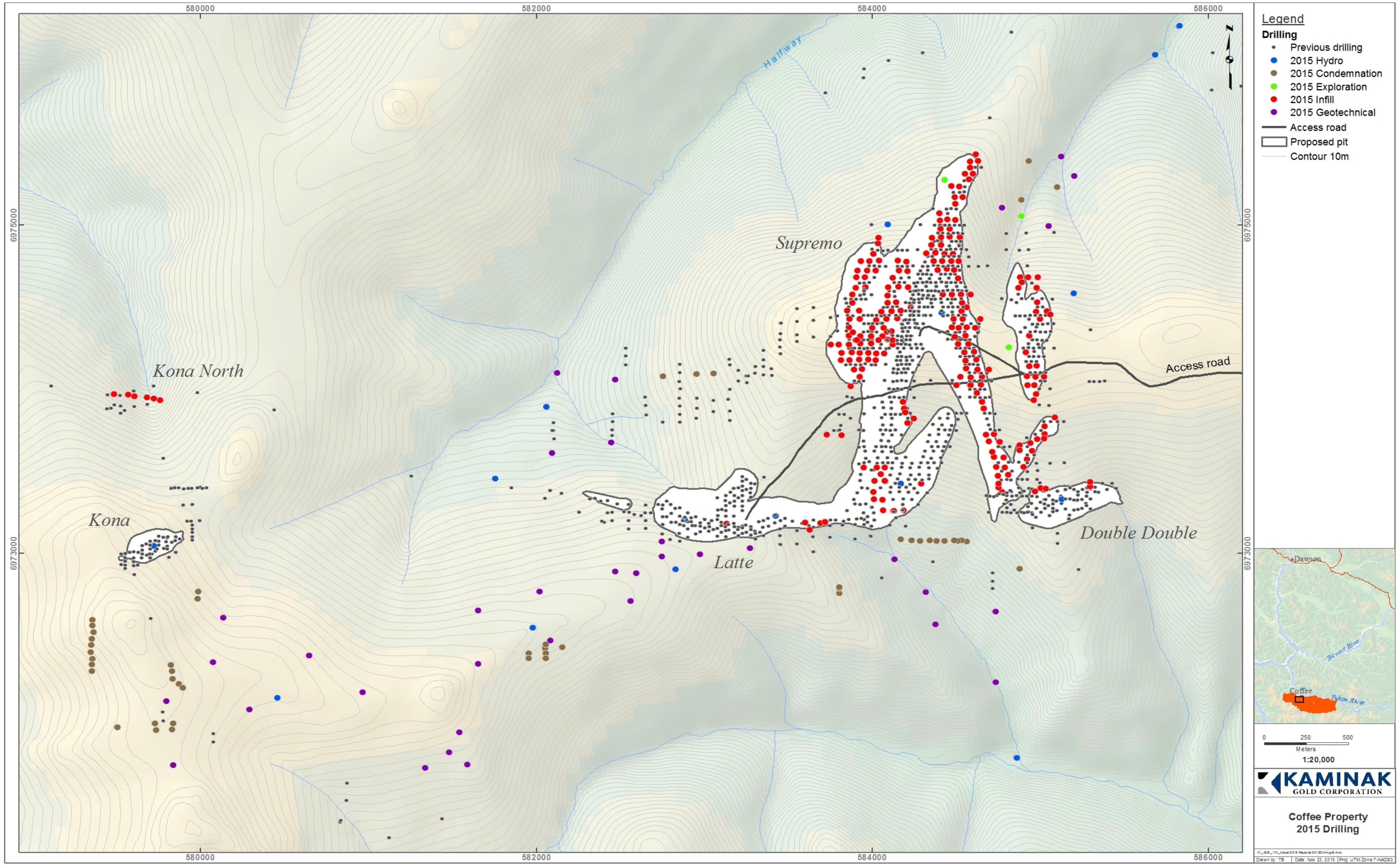
Core drilling took place between May and July 2015 and was contracted to Cyr Drilling International Ltd. The majority of core drilled was NQ2 (50.5 mm diameter), with minor HQ (63.5 mm) and HQ3 (61.1 mm) core drilled to support the hydrogeological and pit-wall geotechnical programs.

RC drilling took place between January and May 2015 and was contracted to Northspan Explorations Ltd. of Kelowna, British Columbia. All RC boreholes were of 92 mm diameter.

RAB drilling took place between February and April 2015, and was contracted to Ground Truth Exploration of Dawson, Yukon. RAB samples were used for condemnation and exploration purposes, but are not considered to have a high level of precision and were therefore did not contribute to the mineral resource.

Sonic drilling took place during the month of April, 2015, and was contracted to Boart Longyear Canada of Calgary, Alberta. All sonic boreholes were 114 mm in diameter. Sonic drill samples were used specifically for the purposes of geotechnical sampling and were not assayed.

Figure 10.2: 2015 Drill Hole Plan



Source: Kaminak 2016

10.1.6 Drilling Summary

As of the end of the 2015 field season, 1,684 drill holes for approximately 279,826 m of cumulative drilling length have been completed. A complete drilling summary by year, drilling method, and zone is provided in Table 10.1 and representative cross sections with interpreted structural zones and interpreted mineralized intervals for Supremo and Latte are shown in Figure 10.3 and Figure 10.4.

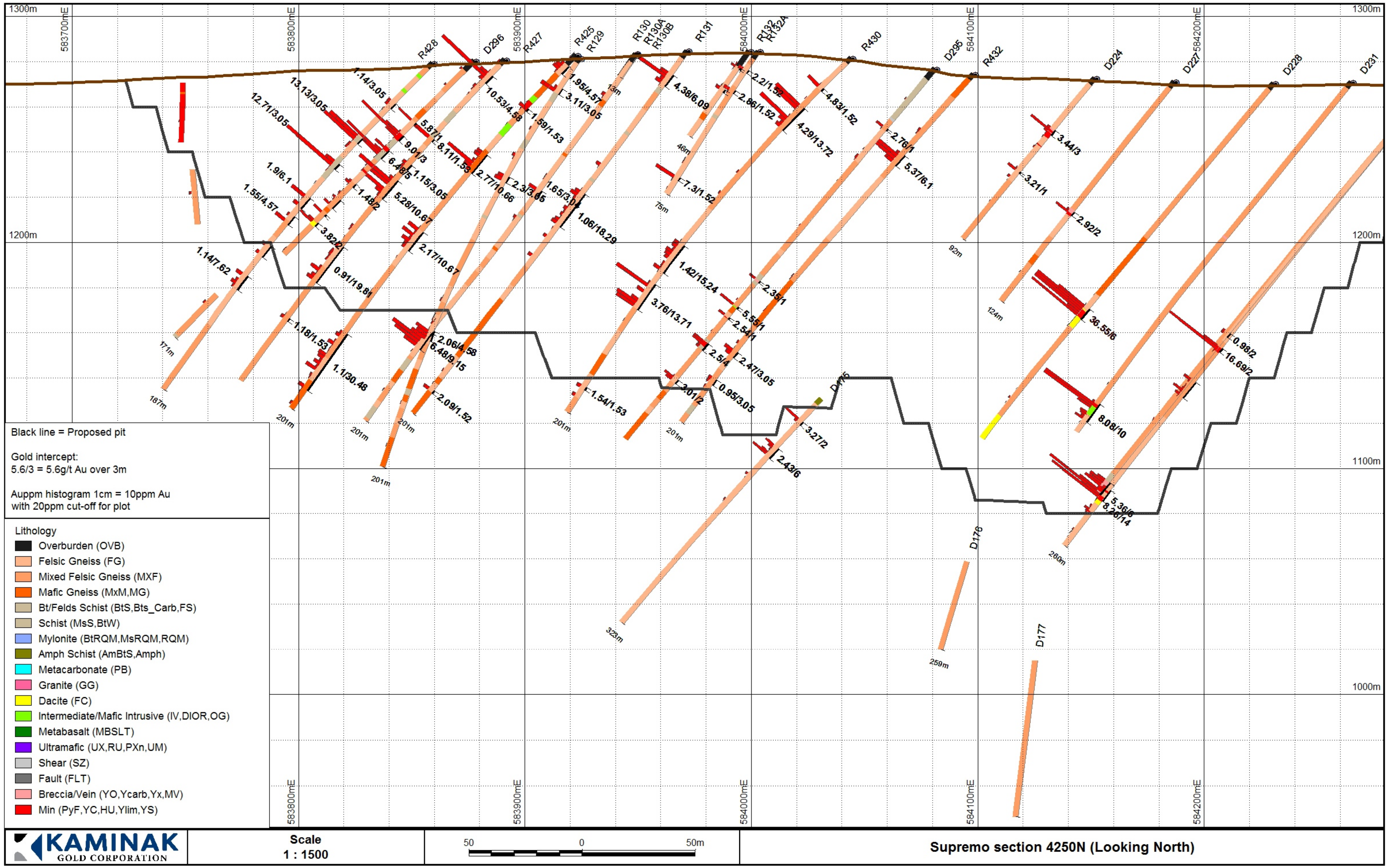
Table 10.1: Coffee Gold Project Drilling by Year

Coffee Drilling Summary by Year				
Year	**Type**	**Zone**	**Holes**	**Metres**
2010	DD	Supremo	27	5,433
		Latte	19	4,291
		Latte North	2	420
		Sumatra	1	184
		Double Double	5	1,231
		Kona	3	499
		Kona North	4	745
		Espresso	3	795
		Americano	10	1,868
		Regional	2	637
		All Zones	76	16,103
2010 Summary		**Totals**	**76**	**16,103**
2011	DD	Supremo	15	4,904
		Latte	60	15,812
		Latte North	1	229
		Double Double	11	2,742
		Kona	6	1,810
		Macchiato	4	1,191
		Cappuccino	2	602
		Americano West	4	1,222
		All Zones	101	28,515
	RC	Supremo	98	13,374
		Kona	47	6,153
		All Zones	145	19,527
2011 Summary		**Totals**	**246**	**48,042**
2012	DD	Supremo	82	17,642
		Double Double	30	8,455
		All Zones	112	26,097
	RC	Supremo	223	39,451
		All Zones	223	39,451
2012 Summary		**Totals**	**335**	**65,548**
2013	DD	Supremo	30	5,953
		Latte	19	4,225
		Sumatra	13	2,094
		All Zones	62	12,272
	RC	Supremo	142	26,339
		Latte	35	5,480
		Latte North	25	4,645
		Sumatra	25	4,682
		Double Double	2	316
		Arabica	11	1,744
		All Zones	240	43,206

Coffee Drilling Summary by Year				
Year	**Type**	**Zone**	**Holes**	**Metres**
2013 Summary		**Totals**	**302**	**55,478**
2014	DD	Supremo	84	16,860
		Latte	10	1,635
		Kona	14	2,212
		Kona North	7	1,546
		Double Double	8	1,724
		Cappuccino	7	1,210
		French Press	4	620
		Macchiato	4	716
		Condemnation	9	371
		All Zones	147	26,894
	RC	Supremo	70	9,644
		Latte	39	4,488
		Kona	3	411
		Kona North	6	1,126
		Double Double	42	5,660
		Condemnation	46	4,537
		All Zones	206	25,866
2014 Summary		**Totals**	**353**	**52,760**
2015	DD	Supremo	86	13,827
		Latte	4	565
		Double Double	2	357
		Kona	2	308
		Kona North	2	289
		Condemnation	7	494
		All Zones	103	15,840
	RC	Supremo	141	18,739
		Latte	6	707
		Kona North	10	873
		Double Double	5	512
		Hydrology	12	653
		Condemnation	23	2,218
		All Zones	197	23,702
	RAB	Supremo	1	50
		Latte	2	87
		Condemnation	32	2,060
		All Zones	35	2,197
	Sonic	Geotechnical	35	156
2015 Summary		**Totals**	**370**	**41,895**
Coffee Gold Project		**Property Totals**	**1,684**	**279,826**

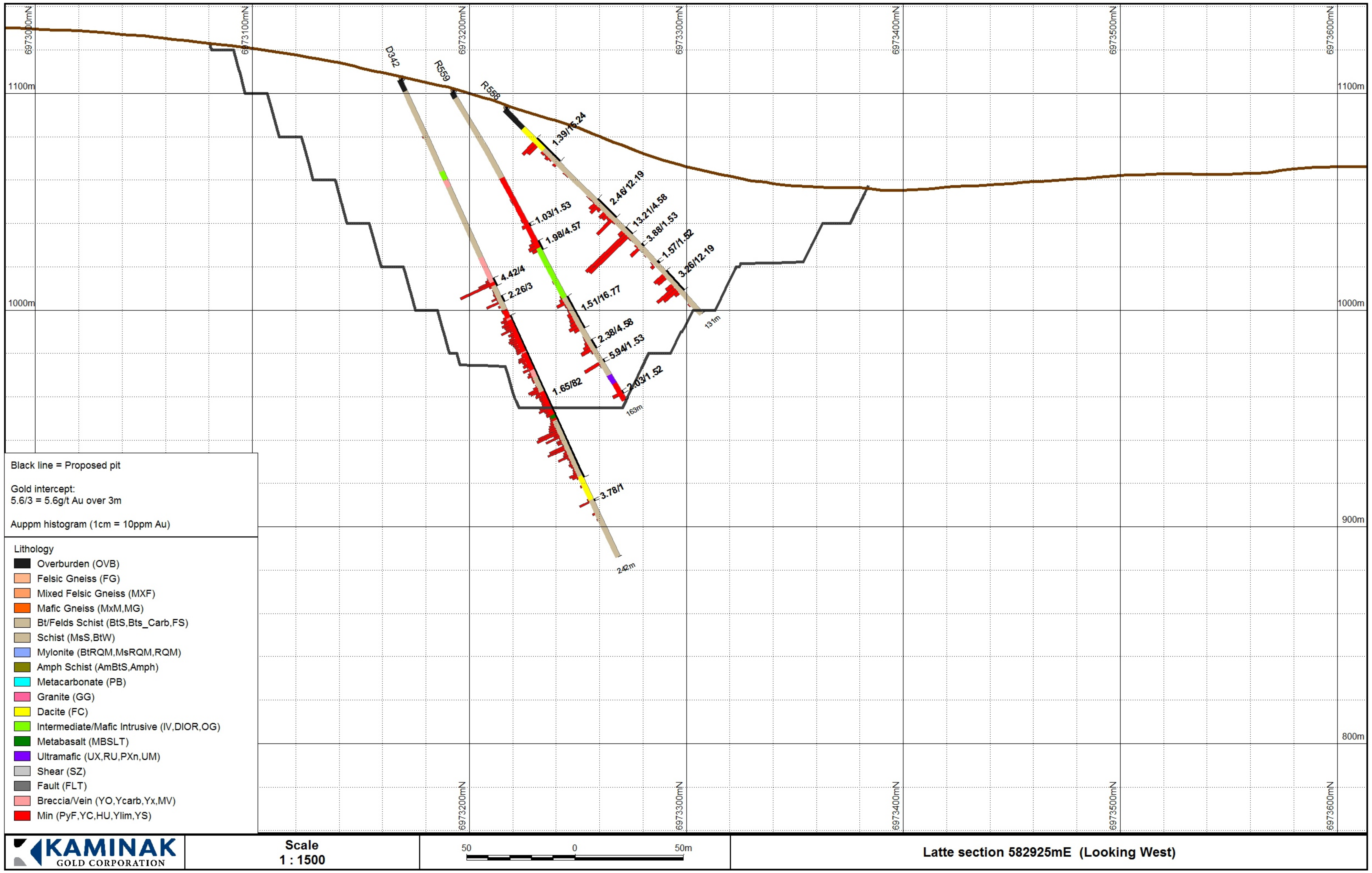
Source: Kaminak 2016

Figure 10.3: Supremo Cross Section – 6974250mN



Source: Kaminak 2015

Figure 10.4: Latte Cross Section - 582925mE



Source: Kaminak 2015

11 Sample Preparation, Analyses and Security

11.1.1 Historical Sampling

Soil samples collected by Mr. Shawn Ryan in 2007 were analyzed by Acme Analytical Laboratories (Acme) in Vancouver, BC. The Acme laboratory management system is International Standards Organization (ISO) 9001:2008 accredited by BSI America Inc. Acme implements a QA/QC system compliant with the ISO 9001 Model for Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories. Acme also participates in the CANMET and Geostats Pty. Ltd. round robin proficiency tests.

Soil samples were prepared using a conventional preparation procedure and analyzed for a suite of 36 elements using aqua regia digestion followed by inductively coupled plasma-atomic emission spectrometry (ICP-AES) on 15 g sub-samples (method code 1DX2).

There is no historical litho-geochemical (rock) sample data for the Coffee Gold Project.

11.1.2 Sampling by Kaminak 2009-2015

Kaminak conducts two main types of sampling; “soil” samples that are primarily used to direct exploration efforts across the property and “rock” samples, mainly derived from diamond and reverse circulation drilling, which are primarily used for the estimation of mineral resources. There are also a relatively small number of additional “grab” samples and samples collected from trenches for exploration and evaluation purposes, these are not used directly in the estimation of mineral resources. Kaminak used two primary laboratories for assaying samples collected during the 2009 through 2015 programs. One secondary laboratory was used for umpire or “check” assaying of drill samples only, which is described further below.

Soil samples collected between 2009 and 2014 were submitted to the Acme laboratory. The samples were prepared and assayed using the same methodology used to assay samples submitted in 2007. Soil samples were prepared using standard preparation procedures and analyzed for a suite of 36 elements using aqua regia digestion followed by ICP-AES on 15 g sub-samples (method code 1DX2).

All drill core, reverse circulation, trench, and grab samples collected from 2010 through 2015 were submitted to ALS Minerals for preparation and assaying. The management system of the ALS Group of laboratories is ISO 9001:2000 accredited by QMI Management Systems Registration. Samples were crushed and pulverized by the ALS Whitehorse preparation facility and shipped to ALS North Vancouver for assaying. The North Vancouver laboratory is ISO/IEC 17025:2005 accredited by the Standards Council of Canada for certain testing procedures, including those used to assay samples submitted by Kaminak. ALS Minerals participates in international proficiency tests such as those managed by CANMET and Geostats Pty. Ltd.

All drill samples were individually sealed in polyore bags on site and shipped by commercial fixed wing charter aircraft (operated by Alkan Air Ltd. and Great River Air Ltd.) to Whitehorse or Dawson, then via road transport by expeditor or Kaminak personnel directly to ALS Minerals' preparation facility in Whitehorse. Samples were conveyed in rice sacks sealed and uniquely numbered with security tags to minimize tampering. Security tags were tracked throughout transportation until receipt by ALS Minerals. No samples were reported tampered with from 2010 through to 2015.

Rock and core samples were prepared for assaying at the ALS Minerals preparation facility using a conventional preparation procedure (dry at 60° Celsius, crushed and sieved to 70% passing 10 mesh ASTM, pulverized to 85% passing 75 micron or better). Prepared samples were then transferred to ALS Minerals laboratory in North Vancouver where they were assayed for gold using a conventional fire assay procedure (ICP-AES) on 30 g sub-samples (50 g samples were used in 2010). In 2010 and 2011, all samples were also analyzed for 35 elements using an aqua regia digestion and ICP-AES finish on 5 g sub-samples. In 2012, samples from only select drill holes (54 boreholes in total) were submitted for the 35-element analysis. In 2013, samples from 87 drill holes were submitted for the 35-element analysis. In 2014, samples from 50 drill holes were submitted for the 35-element analysis. No samples received 35-element analysis in 2015.

Fire-assayed samples with grades in excess of 10 g/t gold were re-assayed from a second 30 g split (50 g split in 2010) using a fire assay procedure and a gravimetric finish. From 2012 to 2015, samples with grades in excess of 20 g/t gold were submitted for screened fire assay from a 1,000 g coarse reject split. The screened fire assay was passed through a 100 micron mesh, with the oversize fraction) undergoing gravimetric analysis following fusion, whereas the undersize fraction was split into two 50 g samples and analyzed using atomic absorption. The average between the two minus fractions was then combined with the plus fraction to provide the total weighted average gold.

From 2013 to 2015 samples with grades greater than 0.3 g/t gold were submitted for cyanide soluble gold assay. For this analysis, a 30 g sub-sample was weighed in a closed 100 ml plastic vessel. 60 ml of sodium cyanide solution (0.25% NaCN, 0.05% NaOH) was then added and the sample shaken until homogenized. Following homogenization, the solution was rolled for an hour before an aliquot was taken and centrifuged. Finally, the sample was analyzed by atomic absorption spectrometry. In 2013, 8,016 sample pulps from previous drilling programs (2010 through 2012 inclusive) were subjected to cyanide leach analyses. A total of 5,539 samples were analyzed by cyanide leaching during the 2014 and 2015 drilling campaigns.

In 2010, samples with a silver grade of more than 100 g/t (two samples) were re-assayed using either an "ore grade" digestion followed by ICP-AES or by conventional fire assay with gravimetric finish on 50 g charges. Two samples from 2011 and two samples from 2012 reported more than 100 g/t silver, but were not re-assayed. No samples from 2013 to 2015 drilling returned greater than 100 g/t silver.

Approximately 1% of all master pulps from core and reverse circulation samples, submitted to ALS Minerals in 2010 through 2015, were submitted at the conclusion of each exploration season to Acme Labs, now under the operating name Bureau Veritas Commodities Canada, for umpire check assaying.

Bureau Veritas' Vancouver laboratory is certified ISO9001:2008 by BSI Group America Inc. for the provision of assays and geochemical analyses. Bureau Veritas used the same methods to analyze the umpire samples as described above, including developing a customized cyanide leach method in 2014 to replicate the one used by ALS. The number of umpire samples analyzed by Bureau Veritas each year is detailed in Table 11.1.

All zones drilled in a given year were represented in the check-assay samples. Although samples covered a wide range of assay results (from detection limit to greater than 20 g/t gold), preference was given to individual samples that displayed greater than 0.3 g/t gold in order to provide an accurate test of laboratory performance and avoid analyzing a large number of near-detection level samples. Kaminak did not use an umpire laboratory to verify the assay results for soil samples.

Table 11.1: Umpire Samples by Year

Year	Umpire Samples
2010	178
2011	425
2012	672
2013	441
2014	448
2015	448

Source: Kaminak 2016

11.1.3 Specific Gravity Data

Specific gravity measurements were made using the water immersion method. In 2011, measurements were made at nominal 10 m intervals in non-mineralized rock and at nominal 5 m intervals in structural zones or apparent gold mineralized rock. From 2012 to 2015, measurements were selected at a rate of one sample per mineralized zone, and one sample per major lithology in non-mineralized rock. In areas of multiple mineralized zones separated by non-mineralized intervals less than 10 m wide, specific gravity was measured for the mineralized zones only.

Samples were weighed dry in air, coated with paraffin wax and weighed immersed in water. A standard was measured roughly every ten samples in order to measure instrumental drift. Results were recorded directly into a Microsoft Excel spreadsheet. A total of 6,734 specific gravity (SG) measurements have been collected.

Specific gravity measurements less than 2.40 or greater than 3.50 were re-weighed by technicians to ensure accuracy. Independent specific gravity testing was also conducted on a randomly selected batch of 35 samples in 2011, 30 samples in 2012, 26 samples in 2013, and 37 samples in 2015 by ALS Minerals in North Vancouver, BC in order to verify the accuracy of the on-site methodology. ALS Minerals results are in close agreement with field measurements, and, therefore, indicate good reproducibility.

11.1.4 Quality Assurance and Quality Control Programs

The exploration work conducted by Kaminak was carried out using a quality assurance and quality control (QA/QC) program meeting industry best practices for exploration properties. Standardized procedures were used in all aspects of the exploration data acquisition and management including mapping, surveying, drilling, sampling, sample security, assaying, and database management.

During 2009, Kaminak did not implement specific analytical quality control measures to monitor the assay results delivered by Acme. The 2009 exploration program involved primarily soil sampling and trenching. Kaminak relied on the laboratory internal analytical quality control measures to monitor the reliability of assay results delivered by Acme.

At the commencement of core drilling in 2010, Kaminak began implementing external analytical quality control measures, in addition to choosing an ISO accredited primary laboratory. The analytical quality control measures involved the use of control samples (certified reference material, blanks, field duplicates) and independent check assaying at an umpire laboratory.

Certified reference materials were sourced from CDN Resource Laboratories Ltd. (CDN) of Langley, BC. Typically six unique standards and one blank were used in each sampling program completed. In 2015, Kaminak used six standards, with certified assay values ranging from 0.628 to 10 g/t gold and one blank with a certified assay value of less than 0.01 g/t gold (Table 11.2). For 2010 rock samples, certified reference materials were inserted approximately at a rate of one every 30 samples. For 2011, 2012, 2013, 2014, and 2015 drill core and reverse circulation samples, and for 2011, 2013, and 2014 trench samples, blanks and certified reference materials were alternated and inserted at a rate of one every ten samples.

Field and laboratory duplicates were also inserted within the samples submitted for assaying. Field duplicate samples were collected by splitting the remaining half-core in half Reverse circulation sample duplicates were collected by running the retention bag of the original sample through the riffle splitter, splitting a second sample from the original sample directly at the drill site. Laboratory duplicates are repeat assays on pulverized samples originally assayed by ALS Minerals.

In 2013, additional laboratory duplicates of cyanide shake test samples were taken at a rate of 1:50 total analyzed samples.

Table 11.2: Specifications of the Certified Control Samples Used by Kaminak in 2015

Reference Material	Gold (g/t)	Standard Deviation (g/t)	Number of Samples
CDN-BL-10	<0.01	-	1,495
CDN-GS-P7K	0.694	0.033	289
CDN-GS-1L	1.06	0.05	95
CDN-GS-1M	1.07	0.045	217
CDN-GS-3L	3.18	0.11	279
CDN-GS-6D	6.09	0.29	307
CDN-GS-9A	9.31	0.345	384

Source: Kaminak 2016

11.1.5 Comments

The Qualified Person reviewed the field procedures and analytical quality control measures used by Kaminak. In the opinion of the Qualified Persons, Kaminak personnel used care in the collection and management of field and assaying exploration data.

In the opinion of the Qualified Person, the sample preparation, security, and analytical procedures used by Kaminak are consistent with generally accepted industry best practices and are, therefore, adequate for the purpose of mineral resource estimation.

12 Data Verification

12.1.1 Verification by Kaminak

The exploration work carried out on the Coffee Gold Project was conducted by Kaminak personnel and qualified subcontractors. Kaminak implemented a series of routine verifications to ensure the collection of reliable exploration data. All work was conducted by appropriately qualified personnel under the supervision of qualified (P.Geo.) geologists. In the opinion of Mr. Robert Sim, P.Geo., of SIM Geological, (APEGBC#24076), the field exploration procedures used at Coffee consistently meet industry practices.

The quality assurance and quality control program implemented by Kaminak was comprehensive and supervised by qualified personnel. Exploration data were recorded digitally to minimize data entry errors. Core logging, surveying, and sampling were monitored by qualified geologists and verified routinely for consistency. Electronic data were captured and managed using an internally-managed Microsoft Access database, and backed up daily. Data from 2010 were managed by Maxwell Geoservices Inc. (Maxwell), and later in 2010 were managed by Kaminak personnel using Maxwell data management applications. In early 2011, the 2010 data were migrated to an internally-managed and internally-designed Microsoft Access database.

Assay results were delivered by the primary laboratory electronically to Kaminak and were examined for consistency and completeness. Kaminak personnel reviewed assay results for analytical quality control samples using bias charts to monitor reliability and detect potential assaying problems. Batches under review for potential failures were recorded in a quality control spreadsheet, investigated and corrective measures were taken when required.

The failure threshold for control samples was set at two times the standard deviation, based on recommended values provided by CDN Resource Laboratories Ltd. Quality control samples exceeding that threshold were investigated. Batches of barren samples containing a quality control failure were not re-assayed. Batches of samples containing more than one quality control failures were re-assayed completely. In batches containing one control sample failure, samples surrounding the failed control sample were re-assayed. After review, Kaminak requested either partial or complete batches of samples be re-assayed by ALS Minerals (Table 12.1). Once it was confirmed that the re-assayed batches passed the quality control failure thresholds, they were accepted and the assay database was updated accordingly.

Table 12.1: Count of Batch Re-runs by Year

Year	Number of Sample Batches Partially or Wholly Re-assayed
2010	44
2011	28
2012	31
2013	19
2014	21
2015	21

Source: Kaminak 2016

12.1.2 Verifications by the Authors of this Feasibility Study

12.1.2.1 Site Visits

In accordance with National Instrument 43-101 (NI 43-101) guidelines, the Qualified Person visited the property on several occasions during active drilling. Robert Sim, P. Geo. (APEGBC#24076) visited the property on four separate occasions; September 12-14, 2011, August 28-29, 2012, May 15-16, 2013 and September 24, 2014. Each visit was similar in process and Mr. Sim was given unfettered access to all aspects of the project and all questions were satisfactorily addressed.

Exploration activities were reviewed with site personnel and the nature of the ongoing interpretation of the geologic environment was discussed with Kaminak geologists. Drill core handling and sampling procedures were reviewed and inspected. Mr. Sim visited a series of drill sites and inspected ongoing diamond drilling and reverse circulation drilling activities. Recent trenches, opened during the summer of 2014, were visited and the nature of the mineralized zones was observed.

During the 2011 site visit, Mr. Sim randomly selected three representative samples from previously sawed drill core intervals. These samples were collected by Mr. Sim, transported to Vancouver and submitted to ALS Minerals laboratory for analysis. The resulting gold grades were similar to those present in Kaminak's sample database. It is Mr. Sim's opinion that Kaminak operates the Coffee Gold Project in a very organized and disciplined manner that follows accepted industry standards.

12.1.3 Verification of Analytical Quality Control Data

Kaminak made available to the authors of this report exploration data in the form of a Microsoft Access database. This database aggregated the assay results for the quality control samples received to date, and was accompanied by comments from Kaminak personnel. The analysis of analytical quality control data produced by Kaminak prior to 2014 was discussed in previous technical reports (Couture and Siddorn, 2011, Couture and Chartier, 2012, Couture et al., 2013, Makarenko et al., 2014) and is not reproduced here.

Kaminak personnel aggregated the assay results for the external quality control samples for further analysis. Sample blanks and certified reference materials data were summarized on time series plots to highlight the performance of the control samples.

Paired data (field duplicate and check assays) were analyzed using bias charts, quantile-quantile and relative precision plots. The analytical quality control data produced by Kaminak in 2014 and 2015 are summarized in Table 12.2 and Table 12.3. The external quality control data produced on this Project represents 13.26% of the total number of samples submitted for assaying in 2014, and 13.47% of the total number of samples submitted for assaying in 2015.

Table 12.2: Summary of Analytical Quality Control Data Produced by Kaminak in 2014

	Reverse Circulation Samples	(%)	Core Samples	(%)	Total	(%)	Comment
Sample Count	16,650		19,187		35,837		
Blanks		5.68		5.64		5.66	
CDN-BL-10	945		1,083		2,028		<0.01 g/t Au
CDN-GS-P7K	163		113		276		
CDN-GS-P7H	18		109		127		
CDN-GS-1L	185		217		402		
CDN-GS-3L	183		219		402		
CDN-GS-6D	209		212		421		
CDN-GS-9A	184		217		401		
Field Duplicates	321	1.93	373	1.94	694	1.94	
Total QC Samples	2,208	13.26	2,543	13.25	4,751	13.26	
Check Assays							
Acme Labs (FA)	188	1.13	260	1.36	448	1.25	Umpire Lab Testing Fire Assay
Acme Labs (CN)	52	0.31	74	0.39			Umpire Lab Testing Cyanide Leach
Total Acme Labs	188	1.13	260	1.36	448	1.25	All CN also included in FA

Source: Kaminak 2016

Table 12.3: Summary of Analytical Quality Control Data Produced by Kaminak in 2015

	Reverse Circulation, Sonic, & RAB Samples	(%)	Core Samples	(%)	Total	(%)	Comment
Sample Count	15,975		10,484		26,459		
Blanks		5.67		5.64		5.65	
CDN-BL-10	904		591		1,495		<0.01 g/t Au
CDN-GS-P7K	177		112		289		
CDN-GS-1M	102		115		217		
CDN-GS-1L	91		4		95		
CDN-GS-3L	162		117		279		
CDN-GS-6D	179		128		307		
CDN-GS-9A	193		118		311		
Field Duplicates	353	2.21	219	2.09	572	2.16	
Total QC Samples	2,161	13.53	1,404	13.39	3,565	13.47	
Check Assays							
Acme Labs (FA + CN)	252	1.58	196	1.87	448	1.69	Umpire Lab Testing

Source: Kaminak 2016

Overall, the performance of the control samples (certified reference materials including both blanks and reference material) inserted with samples and submitted for assaying that were used by Kaminak is very good (example presented in Figure 12.1a). With only very few exceptions, ALS Minerals delivered assay results for the certified reference materials within two standard deviations of the mean for all six reference material tested, and less than the recommended value for the blank sample.

Assay results delivered by ALS Minerals of reference material in 2014 reported less than 2.4% of the results, falling outside of two standard deviations of the certified values for each standard, which is below the 5% threshold typically used for this type of analysis. Some of these failures are attributed to sample mislabeling. Only 0.6% of blanks in 2014 returned assay values above the threshold grade of 0.01 g/t gold.

Assay results from 2015 reported less than 2.4% of the results, falling outside of two standard deviations of the certified values for each standard which is, again, well below the acceptable limits. Some of these failures are due to sample mislabeling. There is no evidence of contamination in the lab procedures, as only 0.6% of blanks returned assay values above 0.01 g/t gold (Figure 12.2b).

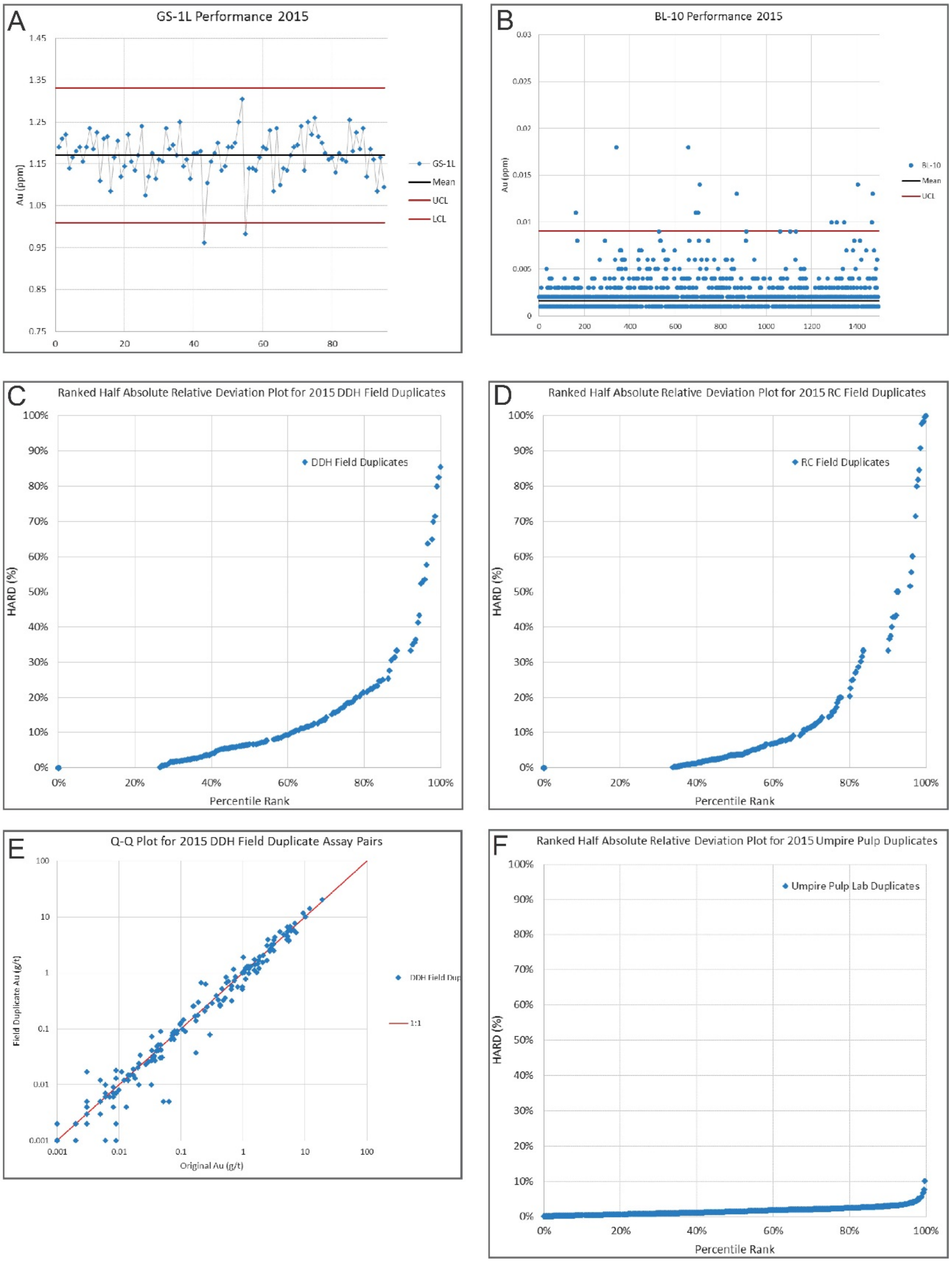
Paired assay data for field duplicates suggest that gold grades demonstrate a natural spread typical of structurally hosted hydrothermal gold systems. Ranked Half Absolute Relative Deviation (HARD) plots from 2014 suggest that 60% of the core field duplicate sample pairs and 50% of the reverse circulation field duplicate sample pairs have HARD below 10%. Ranked HARD plots from 2015 suggest that 61% of core field duplicate samples and 68% of reverse circulation field duplicate samples pairs have HARD below 10% (Figure 12.1c and Figure 12.1d).

These levels of reproducibility of field duplicate results are common in many gold deposits, reflecting the natural variability in gold grade over short distances. These results also show that 80% of the time the field duplicates are within 20% of each other. The correspondence between field duplicates is consistent with the variogram nugget effects that range from 15-25% of the total variability. The distribution on a QQ plot (Figure 7.9e) shows good correspondence of coarse duplicate samples, especially above the projected cut-off threshold grade of (approximately) 0.3g/t gold.

Results from umpire laboratory testing of pulp duplicates in 2014 indicate good reproducibility and no significant deviation or bias in results between labs. HARD plots show that 98% of the check fire assay sample pairs have HARD below 10%. Results from umpire laboratory testing in 2015 also show good reproducibility with no significant deviation or bias in results. HARD plots show that 99% of the check fire assay sample pairs have relative differences below 10% (Figure 12.1f).

In the opinion of the Qualified Persons, the analytical results delivered by ALS Minerals are of sufficient accuracy and precision to support the estimation of mineral resources.

Figure 12.1: Selected 2015 QAQC Plots



12.1.4 Database Verification

Following the completion of the mineral resource models, the sample data from 27 randomly selected drill holes, representing approximately 5% of the data, was exported from the MineSight® for validation purposes. This includes holes completed during each of the drilling programs conducted at Coffee since 2011. The gold grades were manually compared to the values listed in certified assay certificates provided from the lab. Of the 4,634 samples checked, no errors were identified. Similar validation studies were conducted for the previous resource estimates in January 2014 and November 2012 with similar results. These results indicate that the database is sound and sufficient to support the estimation of mineral resources.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Metallurgical testing for the Coffee Gold deposits began in 2011 and continued through 2015. Initial testing was conducted by Inspectorate Exploration & Mining Services Ltd. (Inspectorate) of Richmond, British Columbia. Since 2013 almost all metallurgical testing for process flowsheet development has been performed by Kappes, Cassiday and Associates (KCA) of Reno Nevada. In 2015 comminution test work was completed by ALS Metallurgy Kamloops (ALS) of Kamloops, British Columbia.

Samples for metallurgical testing were derived from both bulk surface samples and drill core composites. Testing included column leaching, bottle roll leaching, flotation, column percolation and drain down, multi-element head assay analyses, column leach head and tailings assay screen analyses, ball mill work indices, crushing impact, and abrasion indices.

13.2 Preliminary Metallurgical Testing

Preliminary test work was undertaken during 2011 and 2012.

Kaminak commissioned SRK Consulting Inc. (SRK) to supervise preliminary metallurgical testing on core sample rejects collected for the Coffee Gold Project. The metallurgical testing work was conducted by Inspectorate. Mr. John Starkey, P.Eng. of Starkey & Associates Inc. (an SRK associate metallurgist) supervised the testing program.

Inspectorate conducted preliminary cyanide leaching tests on two composite drill core samples from the Supremo and Latte oxidized gold zones. Bottle roll, carbon-in-leach (CIL) and carbon-in-pulp (CIP) leaching tests yielded gold extractions of 96.3% to 98.5% respectively. The results from this test work suggest that the oxidized gold mineralization is amenable to conventional cyanide leaching. During the first quarter of 2012 drill core composite samples from Supremo and Latte (samples from the 2011 drilling campaign) were crushed to 12.5 mm (0.5 inch) for simulated heap leach gold extraction via column leach test work. Test results indicated 90.4% gold recovery after 80 days of leaching, with 83.2% recovery after the first 15 days.

During the fourth quarter of 2012 three additional samples from each of Double Double, Supremo and Latte were submitted to Inspectorate for cyanide leach test work. Testing consisted of 72 hour bottle roll cyanidation utilizing CIL and CIP with the same test parameters as the previous test work. The Double Double sample returned gold recoveries between 96.0% and 96.9%. The Supremo sample was a mixture of oxide and transitional material and yielded gold recoveries of 90.7% to 92.4%. The Latte sample was sulphide ore and achieved gold recoveries of 2.0% to 5.3%.

13.3 Metallurgical Testing to Support the PEA

This test work took place between 2013 and 2014.

Kaminak engaged KCA to undertake comprehensive metallurgical testing on core and bulk samples from the Coffee Gold deposit. KCA issued two (2) reports:

- Coffee Gold Project, Report of Metallurgical Test Work, November 2013, Prepared for: Kaminak Gold Corporation, Prepared by: Kappes, Cassiday & Associates, Reno, Nevada 89506, Report I.D.: KCA0130085_COF01_02
- Coffee Gold Project, Report of Metallurgical Test Work, April 2014, Prepared for: Kaminak Gold Corporation, Prepared by: Kappes, Cassiday & Associates, Reno, Nevada 89506, Report I.D.: KCA0130149_COF02_01

Much of the information presented below is summarized from these KCA reports.

13.3.1 Bulk Sample Metallurgical Test Work

13.3.1.1 Sample Selection for the Bulk Samples

A Supremo oxide sample was collected from a surface trench across the T3 mineralized structure at Location 6974250mN.

A Latte oxide sample was collected from a surface trench across the Latte mineralized structure at Location 583250mE.

Trenches were excavated to bedrock and sampling was undertaken across strike at 2 m intervals. The Supremo oxide sample comprised 59 sub-samples and the Latte oxide sample comprised 58 sub-samples. Each sample comprised approximately 1500 kg of material. KCA homogenized and split each sample for the testing program. Each composite sample was assigned a unique sample number and tested for head analyses, head screen analyses with assays by size fraction, bottle roll leaching, agglomeration, and column leaching. All preparation, assaying and metallurgical studies were performed utilizing industry-standard procedures.

13.3.2 Head Analyses

For each sample, portions of the head material were ring and puck pulverized and analysed for gold and silver content by standard fire assay and wet chemistry methods. Head material was also assayed semi-quantitatively for an additional series of elements and for whole rock constituents. In addition to these semi-quantitative analyses, the head material was assayed by quantitative methods for carbon, sulphur and mercury contents. A cyanide shake test was conducted on a portion of the pulverized head material.

In addition to the analyses on pulverized head material, portions of material from select crush sizes were utilized for head screen analyses with assays by size fraction. A summary of the head analyses for gold content is presented in Table 13.1.

Table 13.1: Summary of Head Analyses – Bulk Samples

KCA Sample No.	Description	Average Assay Au (g/t)	Weighted Average* Head Assay Au (g/t)
69580	Latte Oxide	1.61	1.15
69581	Supremo Oxide	4.08	4.37

*Values are the average of two head screen analyses with assays by size fraction

Source: KCA 2015

13.3.3 Bottle Roll Leach Test Work

Bottle roll leach testing was conducted on a portion of material from each composite. For each test, a 1,200 g portion of head material was ring and puck pulverized to a target size of 80% passing 0.075 mm. A 1,000 g portion of the pulverized material was then used for a 96 hour bottle roll leach test conducted and maintained at a target concentration of 1.0 g sodium cyanide per litre of solution.

A summary of the gold extraction from the bottle roll leach test work is presented in Table 13.2.

Table 13.2: Summary of Bottle Roll Leach Test Work – Bulk Samples

KCA Sample No.	Description	Calculated Head Au (g/t)	Extraction Au (%)	Reagent Consumption	
				NaCN	Ca(OH)2
				(kg/t)	(kg/t)
69580	Latte Oxide	1.47	94%	0.06	2.5
69581	Supremo Oxide	3.59	96%	1.62	2.5

Source: KCA 2015

13.3.4 Agglomeration Test Work

Preliminary agglomeration test work was conducted on sub-samples of crushed material. Percolation tests were undertaken to evaluate the permeability of the agglomerated material at various cement contents. The percolation tests were conducted at a range of cement contents, with no compressive load applied, utilizing 2 kg portions of material crushed to the target size of 100% passing 31.5 mm. For agglomeration test work conducted by KCA, the parameters that are typically examined are slump, maximum flow rate, agglomerate pellet break down (when material is agglomerated) and discharge solution colour and clarity.

All agglomeration tests passed the criteria put forth by KCA.

13.3.4.1.1 Column Leach Test Work

A total of four column leach tests were conducted utilizing composite sample material crushed to target sizes of 100% passing either 175 mm or 31.5 mm. The column tests, utilizing material crushed to 100% passing 175 mm, were conducted at ambient temperature (22°C). The column tests utilizing material crushed to 100% passing 31.5 mm were conducted at a temperature of 4°C. A summary of the column leach test extractions for gold are presented in Table 13.3.

Table 13.3: Summary of Column Leach Tests – Bulk Samples

KCA Sample No.	Description	Crush Size P100 (mm)	Calculated Head Au (gpt)	Extraction Au (%)	Days of Leach	Reagent Consumption	
						NaCN	Ca(OH)2
						(kg/t)	(kg/t)
69580	Latte Oxide	175	1.28	88%	100	0.56	1.01
69580	Latte Oxide	31.5	1.12	92%	100	1.08	1
69581	Supremo Oxide	175	4.31	85%	152	0.91	1.52
69581	Supremo Oxide	31.5	3.44	92%	100	0.93	1.51

Source: KCA 2015

13.3.5 Drill Core Composite Metallurgical Test Work

13.3.5.1 Sample Selection for the Drill Core Composite Samples

Seven drill core composite samples were used for the test program. Each composite sample was assigned a unique sample number a2015nd tested for head analyses, head screen analyses with assays by size fraction, bottle roll leaching, agglomeration, and column leaching. The Latte sulphide sample was also subjected to preliminary flotation testing. All preparation, assaying and metallurgical studies were performed utilizing industry-standard procedures.

A summary of the samples utilized for each composite sample is presented in Table 13.4.

Table 13.4: Summary of Drill Core Composite Samples

KCA Sample No.	Description	Number of Individual Sub-samples	Number of Drill Holes Sampled
68151	Supremo, Oxide	150	16
68152	Supremo, Upper Transition	130	12
68153	Supremo, Lower Transition	112	6
68154	Latte, Oxide	128	13
68155	Latte, Upper Transition	99	8
68156	Latte, Lower Transition	96	7
68157	Latte, Sulphide	73	1

Source: Kaminak Gold 2015

13.3.6 Head Analyses

Portions of the head material from each composite were ring and puck pulverized and analyzed for gold by standard fire assay methods. Head material was also assayed semi-quantitatively for an additional series of elements and for whole rock constituents. In addition to these semi-quantitative analyses, the head material was assayed by quantitative methods for carbon, sulphur and mercury contents. A cyanide shake test was also conducted on a portion of the pulverized head material.

In addition to the pulverized head material analyses, portions of material from select crush sizes were utilized for head screen analyses with assays by size fraction.

For the Latte sulphide composite an additional portion of head material was submitted to Phillips Enterprises, LLC for comminution test work.

A summary of the head analyses for gold are presented in Table 13.5.

Table 13.5: Coffee Gold Project Summary of Head Analyses – Drill Core Composites

KCA Sample No.	Description	Average Assay Au (g/t)	Weighted Average* Head Assay Au (g/t)
68151	Supremo, Oxide	1.46	1.5
68152	Supremo, Upper Transition	1.23	1.48
68153	Supremo, Lower Transition	1.57	1.6
68154	Latte, Oxide	1.49	1.56
68155	Latte, Upper Transition	1.48	1.45
68156	Latte, Lower Transition	1.66	1.31
68157	Latte, Sulphide	2.47	2.33

Source: KCA 2015

*Values are the average of two head screen analyses with assays by size fraction

13.3.7 Flotation Test Work

Sub-samples from the Latte sulphide composite were subjected to a two- phase kinetic flotation test program. Flotation tests were conducted in a laboratory-scale Denver flotation apparatus utilizing municipal tap water. The products from each flotation test were individually assayed for gold, silver, copper, lead and total sulphur contents.

A total of four reagent scoping tests were conducted utilizing various reagent combinations and concentrations. Each test was conducted utilizing material milled in a laboratory rod mill to the target size of 80% passing 0.075 mm. Applying various reagent schemes, the reagent scoping tests showed that between 62% and 69% of the gold was concentrated into a rougher flotation concentrate comprising between 8.0% and 8.9% of the sample weight.

Utilizing the results from the reagent scoping test work, four grind size optimization tests were then conducted. The tests were conducted utilizing material milled to target grind sizes of 80% passing 0.150, 0.075, 0.053 and 0.045 mm respectively. At the various grind sizes the grind size optimization tests showed that between 58% and 72% of the gold was concentrated into a rougher concentrate containing 9.8% and 10.5% of the flotation feed sample weight.

13.3.8 Comminution Test Work

A portion of the head material from the Latte sulphide composite was submitted to Phillips Enterprises, LLC in Golden, Colorado for comminution testing. Test work demonstrated Bond Rod Mill and Ball Mill Work indices of 12.7 kWh/t and 15.1 kWh/t respectively.

13.3.9 Bottle Roll Leach Test Work

Bottle roll leach testing was conducted on a sub-sample of each composite. For each test, a 1,000 g portion of head material was milled in a laboratory rod mill to a target size of 80% passing 0.075 mm. The milled slurry was then subjected to a 96 hour bottle roll leach test maintained at a target concentration of 1.0 g sodium cyanide per litre of solution.

A summary of the gold extractions from the bottle roll leach test work is presented in Table 13.6.

Table 13.6: Summary of Bottle Roll Leach Test Work – Drill Core Composites

KCA Sample No.	Description	Calculated Head Au (g/t)	Extraction Au (%)	Reagent Consumption	
				NaCN (kg/t)	Ca(OH)2 (kg/t)
68151	Supremo, Oxide	1.44	94%	1.29	1.5
68152	Supremo, Upper Transition	1.45	78%	2.12	1
68153	Supremo, Lower Transition	1.64	53%	1.45	1
68154	Latte, Oxide	1.57	92%	1.27	1.5
68155	Latte, Upper Transition	1.37	51%	1.15	1.5
68156	Latte, Lower Transition	1.46	38%	1.57	1.5
68157	Latte, Sulphide	2.46	13%	1.35	1.5

Source: KCA

13.3.10 Agglomeration Test Work

Preliminary agglomeration test work was conducted on portions of crushed material from each composite except the Latte sulphide composite. The purpose of the percolation tests was to examine the permeability of the material at various cement contents. The percolation tests were conducted in 75 mm inside-diameter columns at a range of cement levels with no compressive load applied.

For the Supremo oxide and Latte oxide composites agglomeration tests were conducted utilizing 2 kg portions of the material crushed to the target sizes of 80% passing 25 and 12.5 mm, and agglomerated with 0, 2, 4 and 8 kg cement per tonne of dry ore.

For the Supremo, Upper and Lower Transition and Latte, Upper and Lower Transition composites agglomeration tests were conducted utilizing 2 kg portions of the material crushed to the target size of 80% passing 12.5 mm, and agglomerated with 0, 2, 4 and 8 kg cement per tonne of dry ore.

All agglomeration tests passed the criteria put forth by KCA. It was determined from this testwork that the column leaching would be undertaken without the use of agglomeration.

13.3.11 Column Leach Test Work

A total of ten column leach tests were conducted utilizing material crushed to a target size of 80% passing 25 or 12.5 mm. During testing the material was leached for 40 or 42 days with a sodium cyanide solution. Tests were conducted at a temperature of 4°C. A single test was conducted at ambient temperature (approximately 22°C).

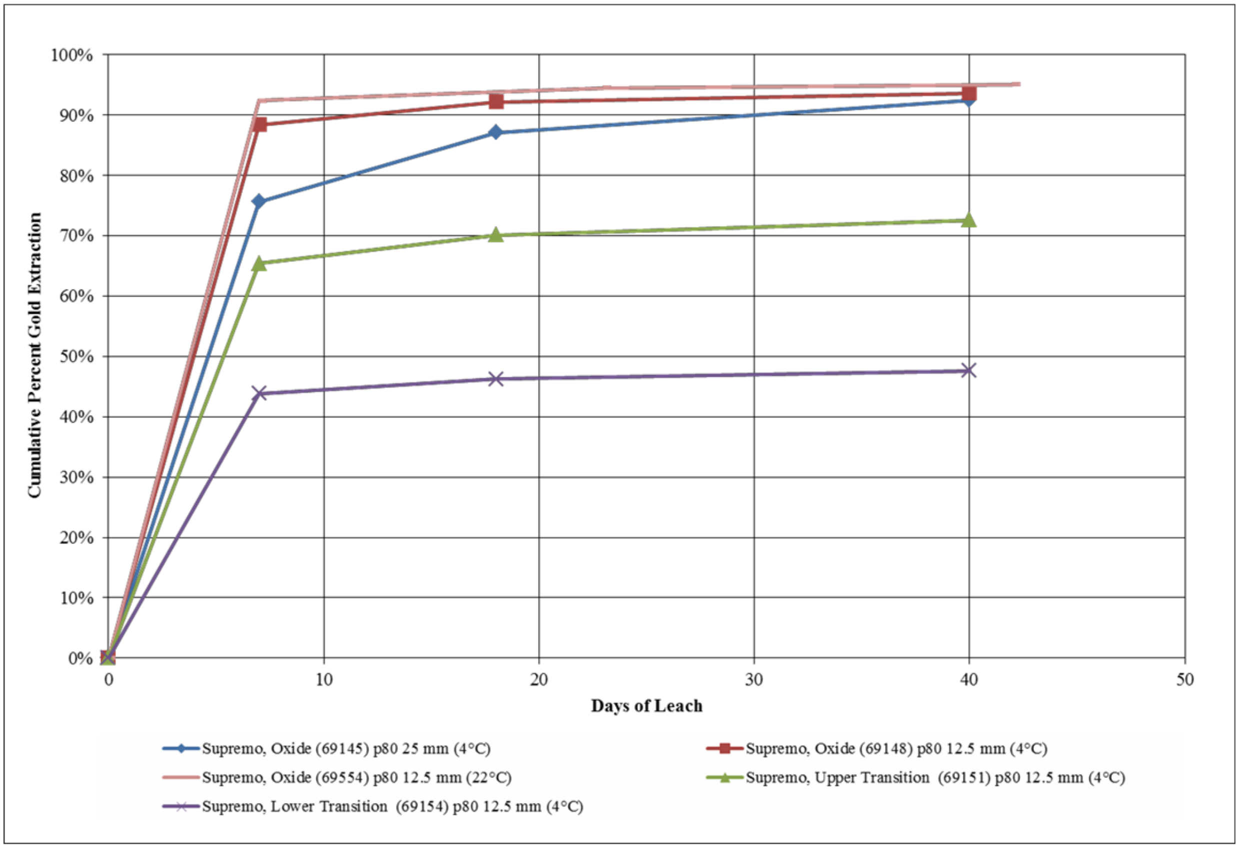
A summary of the column leach test work is presented in Table 13.7. For each group of samples (Supremo and Latte), gold extraction over time is presented graphically in Figure 8.1 and Figure 8.2.

Table 13.7: Summary of Column Leach Test Work – Drill Core Composites

KCA Sample No.	Description	Crush Size p100 (mm)	Temperature (ºC)	Calculated Head Au (g/t)	Extraction Au (%)	Reagent Consumption	
						NaCN (kg/t)	$Ca(OH)^2$ (kg/t)
68151	Supremo Oxide	31.5	4	1.57	92%	0.17	1.51
68151	Supremo Oxide	16	4	1.44	94%	0.28	1.5
68151	Supremo Oxide	16	22	1.55	95%	0.52	1.57
69152	Supremo, Up Transition	16	4	1.49	73%	0.31	1
68153	Supremo, Low Transition	16	4	1.67	48%	0.38	1
68154	Latte, Oxide	31.5	4	1.62	90%	0.19	1.51
68154	Latte, Oxide	16	4	1.54	90%	0.27	1.51
68155	Latte, Upper Transition	16	4	1.54	47%	0.46	2.01
68156	Latte, Lower Transition	16	4	1.42	29%	0.64	1.51
68157	Latte, Sulphide	16	4	2.37	5%	0.46	1.51

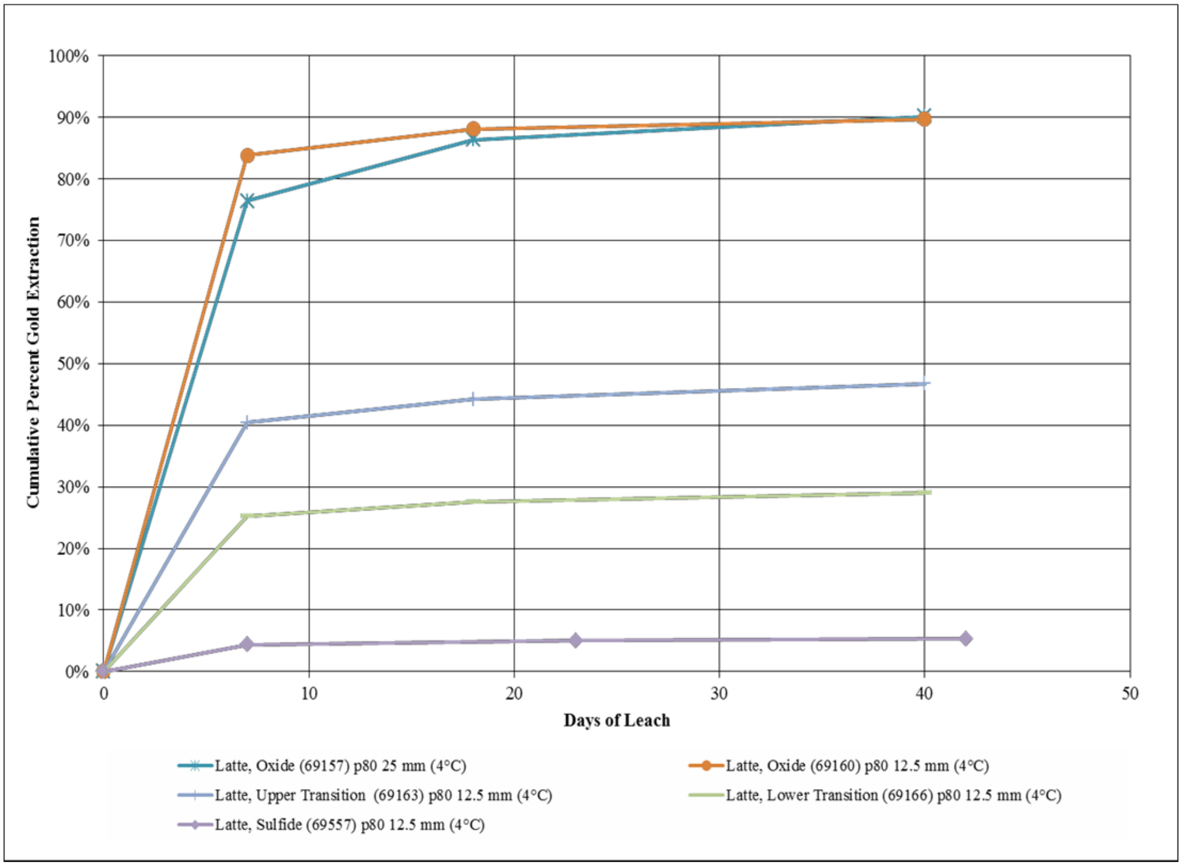
Source: KCA 2015

Figure 13.1: Supremo Core Composites – Column Leach Tests Gold Extractions



Source: KCA 2015

Figure 13.2: Latte Core Composites – Column Leach Tests Gold Extractions



Source: KCA 2015

13.3.12 Discussion

To evaluate leach efficiency as a function of temperature, gold extraction from the head screen versus tailings screen analyses, and comparisons from similar size fractions were conducted. These showed only minor variations, typically in the order of 1 to 2%, which demonstrates that there are no obvious differences in leaching kinetics between tests conducted at 4°C versus those conducted at ambient temperature (22°C).

The head values for gold obtained in this test program compared well, showing an overall agreement between head grades and calculated head grades sourced from the various parts of the test program.

Column test extraction accuracy was confirmed by comparing carbon assays to the calculated head extraction (carbon assays plus tailings assays). For the column leach tests, the calculated head grades derived from carbon and solution assays compared well with each other.

For the column leach tests conducted on each group of samples (Supremo and Latte), a comparison of the gold extraction percentages with sulphide sulphur content showed a reduction in gold extraction with increasing sulphide sulphur content.

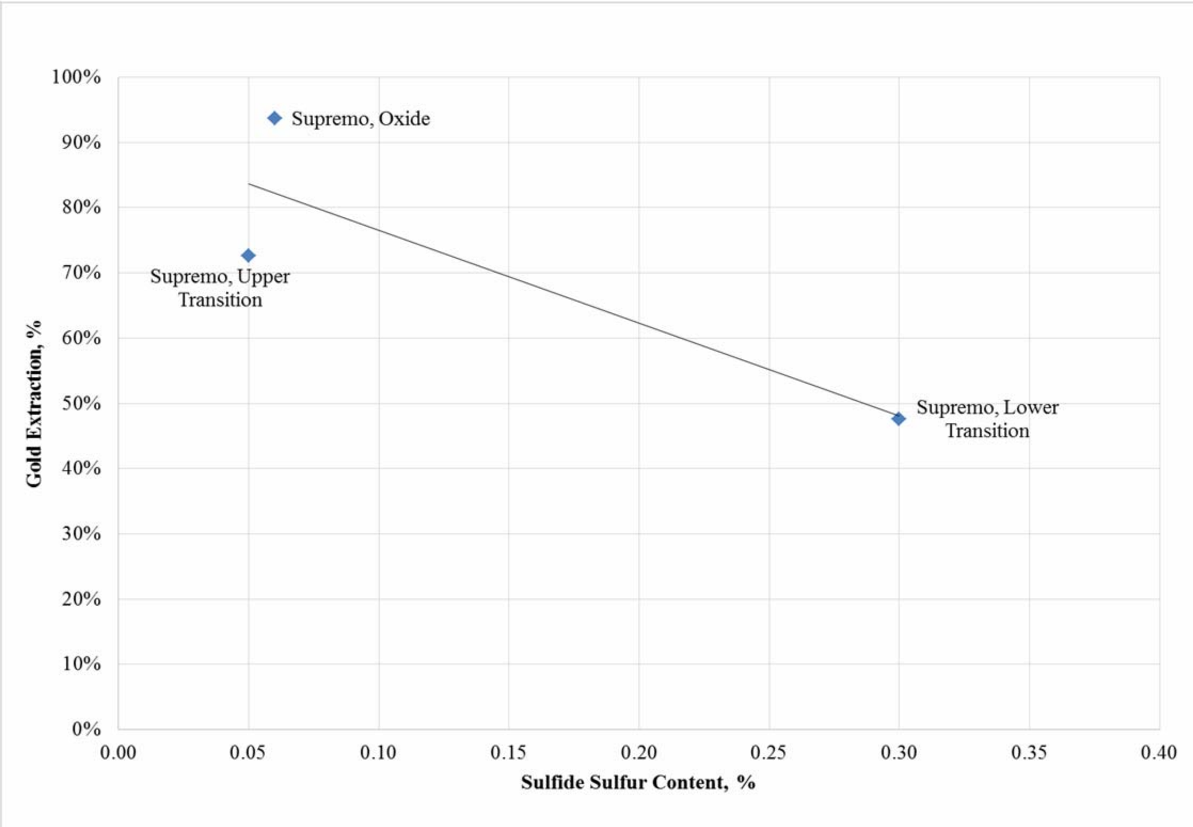
A summary comparing gold extraction and sulphur speciation is presented in Table 13.8. Gold extraction versus sulphide content is presented graphically in Figure 13.3 and Figure 13.4.

Table 13.8: Column Leach Test Gold Extraction verse Sulphur Speciation

KCA	Description	Column Extracted	Total Sulphur%	Sulphide Sulphur	Sulphate Sulphur
Sample No.		% Au		%	%
68151	Supremo, Oxide	94%	0.1	0.06	0.04
68152	Supremo, Upper Transition	73%	0.15	0.05	0.09
68153	Supremo, Lower Transition	48%	0.47	0.3	0.17
68154	Latte, Oxide	90%	0.16	0.02	0.14
68155	Latte, Upper Transition	47%	1.03	0.7	0.33
68156	Latte, Lower Transition	29%	1.27	0.93	0.34
68157	Latte, Sulphide	5%	1.55	1.32	0.23

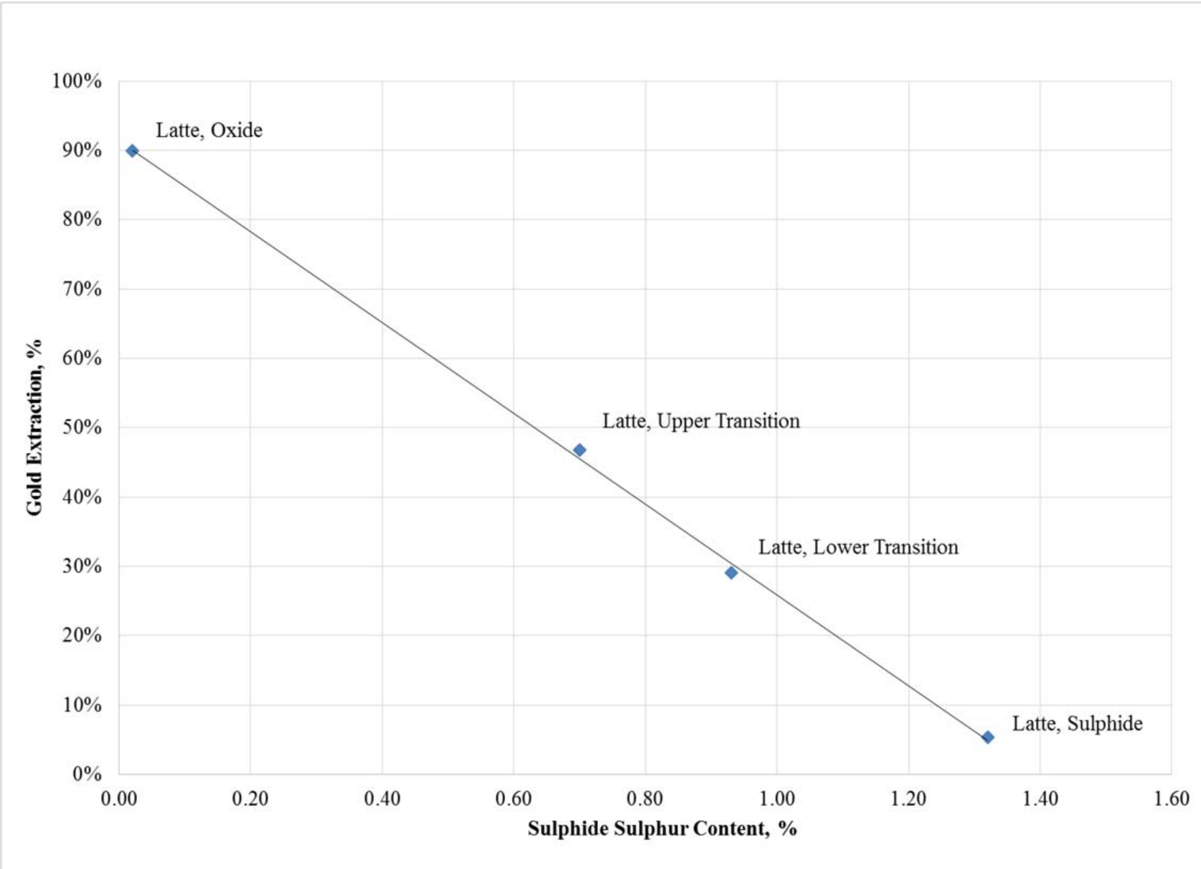
Source: KCA 2015

Figure 13.3: Supremo Core Composites Gold Extraction verse Sulphide Content



Source: KCA 2015

Figure 13.4: Latte Core Composites Gold Extraction vs. Sulphide Content



Source: KCA 2015

13.3.13 Cyanide Soluble Gold Test Work

Categorization of oxidation has in the past been undertaken by visual estimation of the proportion of oxidized sulphur and sulphide in the ore. The degree of oxidation at Coffee Deposit varies both with depth and along the mineralized structures. Thus, the visual estimation of oxidation is unlikely to be sufficiently accurate or consistent to quantify the degree and distribution of oxidation for metallurgical purposes. In particular, the internal variability within Transitional material can be considerable. Presently, the Transitional is simply divided into an 'Upper' and 'Lower' zone based on ≥50% oxidized material and ≤50% oxidized material respectively. As can be seen from Table 13.8, although the visual estimate of the degree of oxidation for the Upper and Lower transitional material at both Supremo and Latte were targeted to be similar, the actual sulphide content of the Latte samples was much higher.

The lower gold extractions from the Latte Transitional material are considered to be a result of a visual underestimation of sulphide in those samples.

In order to better evaluate the amenability and variability of Transitional material to metallurgical cyanide leach recovery, over 8,000 samples, representing 70% of all samples above a fire assay value of 0.3 g/t Au within mineralized intercepts drilled from the 2010 up to and including 2013, were subjected to cyanide soluble assay tests. The cyanide soluble assays were performed by ALS Laboratories (Au-AA13 method).

The difference between the cyanide soluble assay and the original fire assay, on an individual assay by assay basis, or across composites made up of equivalent samples, may be used to provide an indication of the gold within the samples that are amenable to cyanide leach. By extension, it also indicates the degree of oxidation of the sample. The cyanide soluble proxy gold recovery is the percentage of the fire assay value actually reporting to the leach solution.

A comparison of the column leach test recoveries from the KCA testing program and the cyanide soluble assays from the same samples as used in the testing composites illustrate a strong correlation. This correlation of the cyanide soluble recovery and the actual column leach test recovery indicates that cyanide soluble recovery is a reliable method to map the metallurgical recovery throughout the Oxide, Transitional and Sulphide zones of the Coffee Deposit.

13.4 Metallurgical Testing to Support the Feasibility Study

This test work was conducted between 2014 and 2015.

An additional metallurgical testing program to support the Feasibility Study was outlined in the summer of 2014. Both bulk samples from surface trenches and core drill hole composite samples were selected and sent to KCA. Samples from the Kona and Double Double deposits were included in order to expand the testing to all potential mine areas. Also, to better quantify the metallurgical response of transitional materials to cyanide leaching the use of samples with varying ranges of cyanide soluble gold values were generated.

Results from the KCA program are summarized within this document. All of the detailed procedures and results are contained in the KCA report entitled:

- Coffee Gold Project, Report of Metallurgical Test Work, June 2015, Prepared for: Kaminak Gold Corporation, Prepared by: Kappes, Cassiday & Associates, Reno, Nevada 89506, Report I.D.: KCA0140138_COF04_03

KCA tested eight bulk and ten drill core composite samples. Each sample or sample composite was submitted for head analyses, head screen analyses with assays by size fraction, bottle roll leaching, agglomeration, and column leaching at three different crush sizes, 175, 62.5 and 16 ml. Also samples from each mine area were sent to ALS Kamloops for crushing impact and abrasion testing. Results are summarized from the ALS report entitled:

- Comminution Test Work On Samples From The Coffee Gold Project, Kaminak Gold Corporation, KM4668, April 23, 2015, ALS Metallurgy Kamloops

13.4.1 Sample Receipt and Preparation

Between September 30, 2014 and December 10, 2014 KCA received a total of five supersacks and 376 bags of samples. The supersacks contained 148 bags of trench and surface bulk samples from the Latte and Supremo areas. The bags of samples contained drill core samples from the Supremo, Latte, Double Double and Kona areas. All sample preparation, assaying and metallurgical studies were performed using industry-standard procedures.

A summary of the samples received at KCA is presented in Table 13.9.

Table 13.9: Summary of Samples Received at KCA

KCA Sample No.	Sample Description	Sample Type
72101	Latte 583150mE Trench	Bulk
72102	Latte 583350mE Trench	Bulk
72103	Supremo 6974000mN Trench	Bulk
72130	Supremo 6974350mN Trench	Bulk
72131	Supremo 6973750mN Trench	Bulk
72132	Supremo 6974750mN Trench	Bulk
72133	Latte Mine Block	Bulk
72134	Supremo Mine Block	Bulk
72142	Supremo T2-T4 Composite**	Bulk
72143	Supremo T3 Composite****	Bulk
72135	Latte Oxide West	Drill Core Composite
72136	Latte Oxide East	Drill Core Composite
72137	Latte 80% CN Soluble	Drill Core Composite
72138	Latte 60% CN Soluble	Drill Core Composite
72139	Kona Oxide	Drill Core Composite
72140	Kona 80% CN Soluble	Drill Core Composite
72141	Double Double Oxide	Drill Core Composite
72144	Supremo Oxide West	Drill Core Composite
72145	Supremo Oxide East	Drill Core Composite
72146	Supremo 80% CN Soluble	Drill Core Composite

** Composite of Samples 72103 & 72130
****Composite of Samples 72131 & 72132
Source: KCA 2015

13.4.2 Sample Selection for the Bulk Samples

The Supremo Bulk samples were collected from four separate surface trenches across the Supremo mineralized structures.. From the four trenches, two composite samples were generated. The Latte bulk samples were collected from two separate surface trenches across the Latte mineralized structures. Trenches were excavated to bedrock and sampling was completed at 2 m intervals. A total of approximately 3,000 kg of material collected from the four Supremo trenches and 2,600 kg of material collected from the two Latte trenches was sent to KCA where it was homogenized prior to splitting.

The Supremo and Latte mine block samples were taken from trenches with a continuous zone of contiguous two metre samples included and with no blending of lower grade material to target a specific average grade.

13.4.3 Sample Selection for the Drill Core Composite Samples

Ten core drill hole composite samples were used for the test program. Each composite sample was assigned a unique sample number and submitted for head analyses, head screen analyses with assays by size fraction, bottle roll leaching, agglomeration, and column leaching. All preparation, assaying and metallurgical studies were performed utilizing industry-standard procedures.

A summary of the samples utilized for each composite is presented in Table 13.10.

Table 13.10: Summary of Drill Core Composite Samples

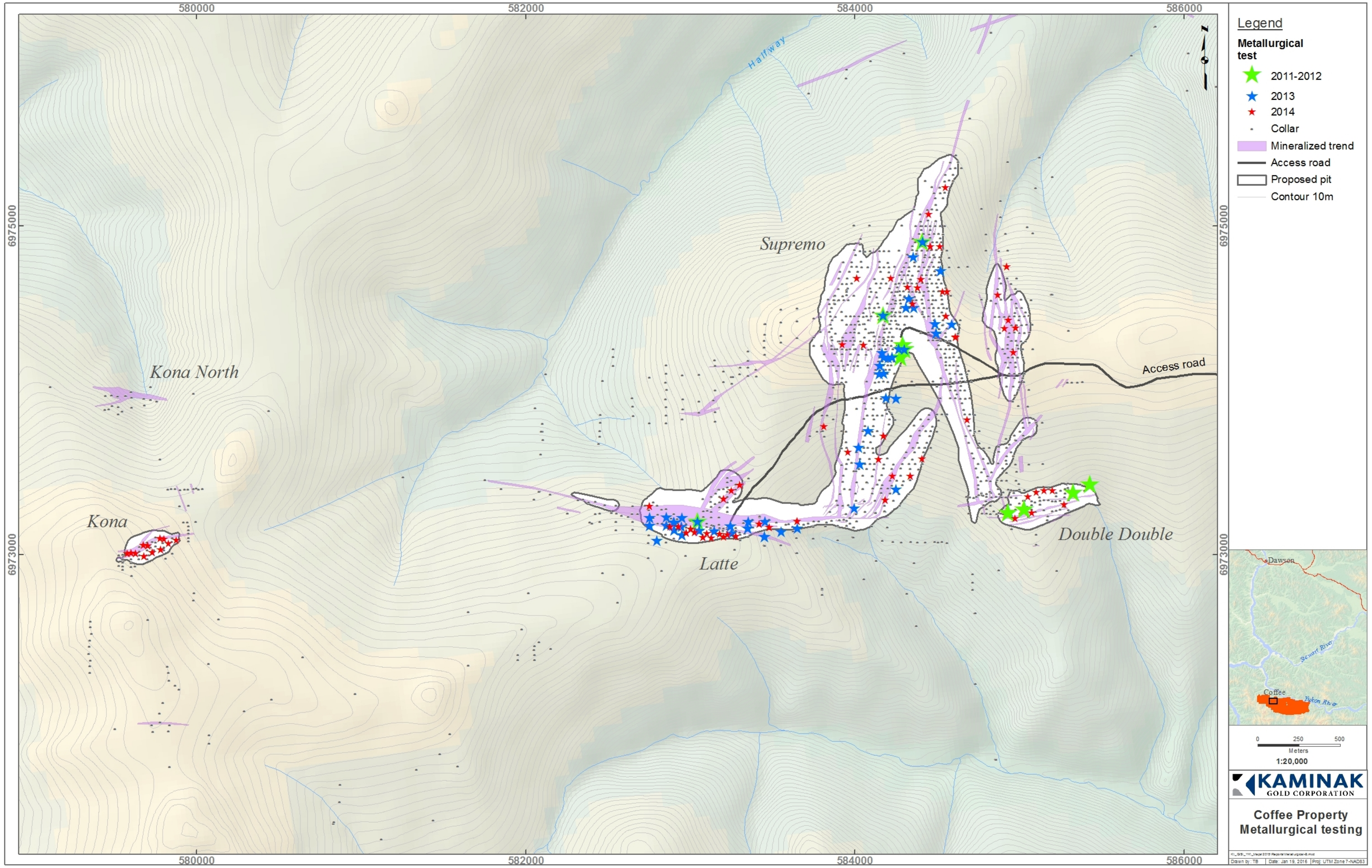
KCA Sample Number	Sample Description	Number of Individual samples within composite	Number of Drill Holes Sampled	Weight kg
72135	Latte Oxide West	67	5	254
72136	Latte Oxide East	56	8	270
72137	Latte 80% CN Soluble	48	8	251
72138	Latte 60% CN Soluble	38	12	223
72139	Kona Oxide	42	11	279
72140	Kona 80% CN Soluble	48	10	263
72141	Double Double Oxide	67	7	253
72144	Supremo Oxide West	73	8	396
72145	Supremo Oxide East	98	12	388
72146	Supremo 80% CN Soluble	44	18	255

Source: KCA and Kaminak 2015

It should be noted that the selection of the cyanide soluble samples was designed to more adequately define the Transitional zones from oxide to sulphide. The absence of some 60% CN soluble and some 80% CN soluble samples for certain mine areas was due to a lack of enough sub-samples available to generate a composite sample.

Figure 13.5 illustrates the location of the samples used for the metallurgical test work.

Figure 13.5: Metallurgical Sample Location Map



Source: Kaminak 2015

13.4.4 Head Analyses

For each sample, splits of the head material were ring and puck pulverized and analyzed for gold and silver content by standard fire assay and wet chemistry methods. Head material was also assayed semi-quantitatively for an additional series of elements and for whole rock constituents. In addition to these semi-quantitative analyses, the head material was assayed by quantitative methods for carbon, sulphur and mercury content. A cyanide shake test was also conducted on a portion of the pulverized head material.

In addition to the analyses on pulverized head material, portions of material from each crush size utilized for column leach test work were utilized for head screen analyses with assays by size fraction.

No problematic elements were discovered during the overall head analyses.

The results of the head analyses for gold and silver content are summarized in Table 13.11.

Table 13.11: Summary of Head Analyses – Gold and Silver

KCA Sample No.	Description	Average Assay Au g/t	Average Assay Ag g/t	Avg. Head Assay Au g/t**	Avg. Head Assay Ag g/t**
72101	Latte 583150mE Trench	1.11	0.62	1.21	0.5
72102	Latte 583350mE Trench	0.93	0.62	0.9	0.5
72133	Latte Mine Block	9.62	0.89	9.64	0.36
72135	Latte Oxide West	1.15	0.62	1.01	0.39
72136	Latte Oxide East	1.02	0.41	1.01	0.21
72137	Latte 80% CN Soluble	2.28	0.62	2.03	0.57
72138	Latte 60% CN Soluble	1.16	0.41	1.24	0.32
72142	Supremo T2-T4 Composite	1.7	0.62	1.91	0.33
72143	Supremo T3 Composite	2.56	0.41	2.33	0.32
72134	Supremo Mine Block	5.25	0.31	5.5	0.32
72144	Supremo Oxide West	1.31	0.41	1.49	0.25
72145	Supremo Oxide East	1.81	0.41	1.69	0.25
72146	Supremo 80% CN Soluble	1.02	0.41	1.05	0.28
72139	Kona Oxide	1.37	0.21	1.43	0.18
72140	Kona 80% CN Soluble	1.32	0.21	1.34	0.15
72141	Double Double Oxide	3.08	0.41	3.18	2.32

** Average of all head assay screen analyses

Silver content of the head samples is low and any silver production will not be economically significant. Silver is therefore omitted from further consideration in this document but all silver analytical data can be found in the KCA 2015 report.

Head analyses for carbon and sulphur were conducted. In addition to total carbon and sulphur analyses, speciation for organic and inorganic carbon and speciation for sulphide and sulphate sulphur were conducted. Head analyses for mercury and copper content as well as semi-quantitative analyses were conducted for a series of individual elements and whole rock constituents. All of these analyses are contained in the KCA 2015 report.

No problematic elements were discovered during the overall head analyses.

13.4.5 Bottle Roll Leach Test Work

Bottle roll leach testing was conducted on a portion of material from each sample. For each test, a 1,200 g portion of head material was ring and puck pulverized to a target size of 80% passing 0.075 mm. A 1,000 g portion of the pulverized material was then utilized for a 96 hour leach test maintained at a target concentration of 1.0 g sodium cyanide per litre of solution.

The results of the bottle roll leach test work are summarized in Table 13.12.

Table 13.12: Summary of Bottle Roll Leach Test Work

KCA Sample No.	KCA Test No.	Description	Calc. Head Au g/t	Au Extraction %	Reagent Consumption	
					NaCN Kg/t	Lime Kg/t
72101	72191A	Latte 583150mE Trench	1.08	94%	0.18	3
72102	72191B	Latte 583350mE Trench	0.87	96%	0.08	3
72133	73050A	Latte Mine Block	8.88	94%	0.08	3.5
72135	73043A	Latte Oxide West	1.13	95%	0.08	2.5
72136	73043B	Latte Oxide East	0.98	95%	0.06	2.5
72137	73043C	Latte 80% CN Soluble	2	86%	0.15	2.5
72138	73043D	Latte 60% CN Soluble	1.18	61%	0.16	2.5
72142	73050C	Supremo T2-T4 Comp.	1.66	98%	0.01	3
72143	73050D	Supremo T3 Comp.	2.4	97%	0.8	2
72134	73050B	Supremo Mine Block	5.02	97%	0.03	2.75
72144	73051A	Supremo Oxide West	1.24	96%	0.02	2
72145	73051B	Supremo Oxide East	1.78	97%	0.1	1.75
72146	73051C	Supremo 80% CN Soluble	0.99	86%	0.12	2
72139	73044A	Kona Oxide	1.23	90%	0.31	2.5
72140	73044B	Kona 80% CN Soluble	1.31	81%	1.33	2.75
72141	73044C	Double Double Oxide	2.64	96%	0.15	2.5

Source: KCA 2015

Silver recoveries ranged between 28 and 62%.

The carbon and sulphur assays for the Latte 80% CN soluble, Latte 60% CN soluble, Supremo 80% CN soluble and Kona 80% CN soluble samples were reviewed in conjunction with the gold extractions from the cyanide shake tests and the bottle roll leach tests. The organic carbon levels did not impact on gold leachability. However, the sulphide sulphur level trended more closely with the leachability of the material; an increase in sulphide sulphur content led to a decrease in gold extraction in both the cyanide shake and bottle roll leach tests. It should also be noted that the indications given by the sample labels where presented were a strong indication of the gold extraction.

13.5 Column Leach Test Work

Column leach tests were conducted on Latte and Supremo material crushed to 100% passing 175, 62.5 or 16 mm. Column leach tests were conducted utilizing the remaining material types crushed to 100% passing 62.5 or 16.0 mm.

All column leach tests conducted on material crushed to 100% passing 16 mm were undertaken under temperatures to simulate cold-climate heap leach conditions (4ºC).

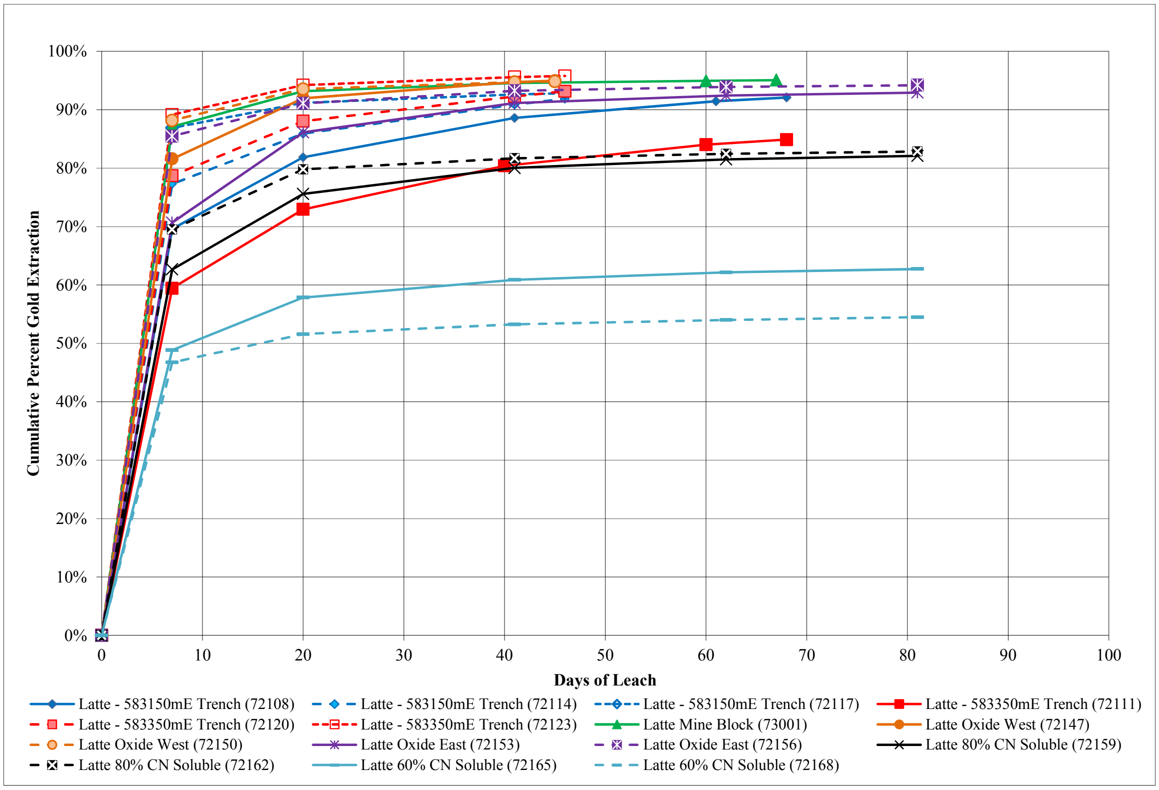
A summary of the column leach test work is presented in Tables 13.13. Figures 13.6, 13.7, and 13.8 illustrate the gold extraction versus days of leach.

Table 13.13: Summary of Column Leach Test Work

KCA Sample No.	KCA Test No.	Description	Crush Size mm	Calc. Head Au g/t	Extraction Au %	Days of Leach	Reagent Consumption NaCN Kg/t	Reagent Consumption Lime Kg/t
72101	72108	Latte 583150mE Trench	175	1.24	92%	68	0.4	0.99
72101	72114	Latte 583150mE Trench	62.5	1.36	92%	46	0.5	1.01
72101	72117	Latte 583150mE Trench	16	1.2	93%	46	0.54	1
72102	72111	Latte 583350mE Trench	175	1.14	85%	68	0.41	0.99
72102	72120	Latte 583350mE Trench	62.5	0.89	93%	46	0.55	1.01
72102	72123	Latte 583350mE Trench	16	0.97	96%	46	0.55	0.98
72133	73001	Latte Mine Block	16	9.54	95%	67	0.58	0.99
72135	72147	Latte Oxide West	62.5	1.17	95%	45	0.48	1.52
72135	72150	Latte Oxide West	16	1.12	95%	45	0.17	1.53
72136	72153	Latte Oxide East	62.5	1.03	93%	81	0.87	1.53
72136	72156	Latte Oxide East	16	0.96	94%	81	0.3	1.52
72137	72159	Latte 80% CN Soluble	62.5	1.46	82%	81	0.79	1.49
72137	72162	Latte 80% CN Soluble	16	1.93	83%	81	0.44	1.51
72138	72165	Latte 60% CN Soluble	62.5	1.47	63%	81	0.76	1.52
72138	72168	Latte 60% CN Soluble	16	1.06	54%	81	0.3	1.5
72142	73007	Supremo T2-T4 Comp.	175	2.1	92%	67	0.53	1.49
72142	73010	Supremo T2	62.5	2.05	93%	67	0.96	1.51
72142	7308	Supremo T2	16	1.9	97%	67	0.59	1.49
72143	73016	Supremo T3 Comp.	175	2.86	82%	67	0.46	1.51
72143	73019	Supremo T3 Comp.	62.5	2.28	93%	67	0.72	1.59
72143	73022	Supremo T3 Comp.	16	2.26	95%	67	0.29	1.51
72134	73004	Supremo Mine Block	16	5.55	98%	67	0.68	0.99
72144	73025	Supremo Oxide West	62.5	1.31	90%	67	0.49	1.51
72144	73028	Supremo Oxide West	16	1.28	95%	67	0.84	1.5
72145	73031	Supremo Oxide East	62.5	1.67	95%	67	0.74	1.53
72145	73034	Supremo Oxide East	16	1.78	96%	67	0.32	1.49
72146	73037	Supremo 80% CN Soluble	62.5	1.04	85%	67	0.61	1.5
72146	73040	Supremo 80% CN Soluble	16	0.93	81%	67	0.43	1.2
72139	72171	Kona Oxide	62.5	1.42	88%	81	0.82	1.56
72139	72174	Kona Oxide	16	1.36	91%	81	0.37	2.06
72140	72177	Kona 80% CN Soluble	62.5	1.36	72%	81	0.76	1.53
72140	72180	Kona 80% CN Soluble	16	1.35	76%	81	0.5	2.01
72141	72183	Double Double Oxide	62.5	4.33	95%	81	0.87	1.56
72141	72186	Double Double Oxide	16	2.99	95%	81	0.32	2.02

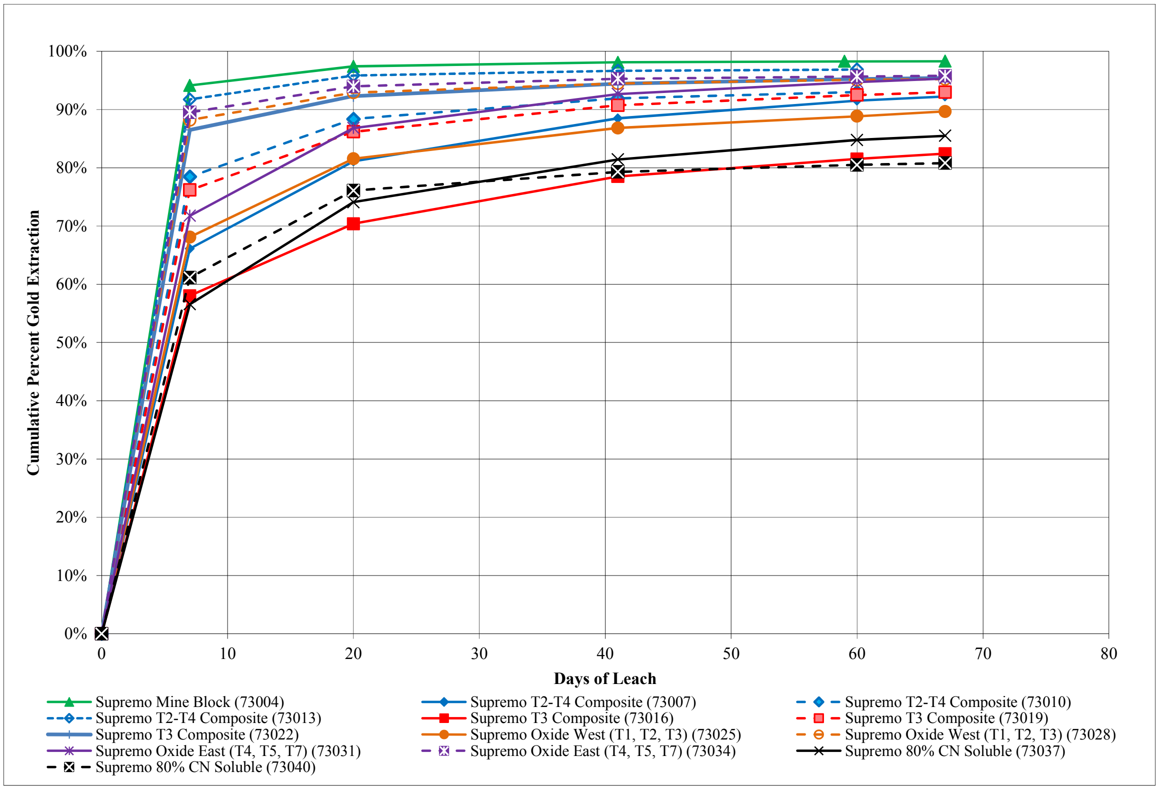
Source: KCA

Figure 13.6: Latte: Gold Extraction versus Days of Leach



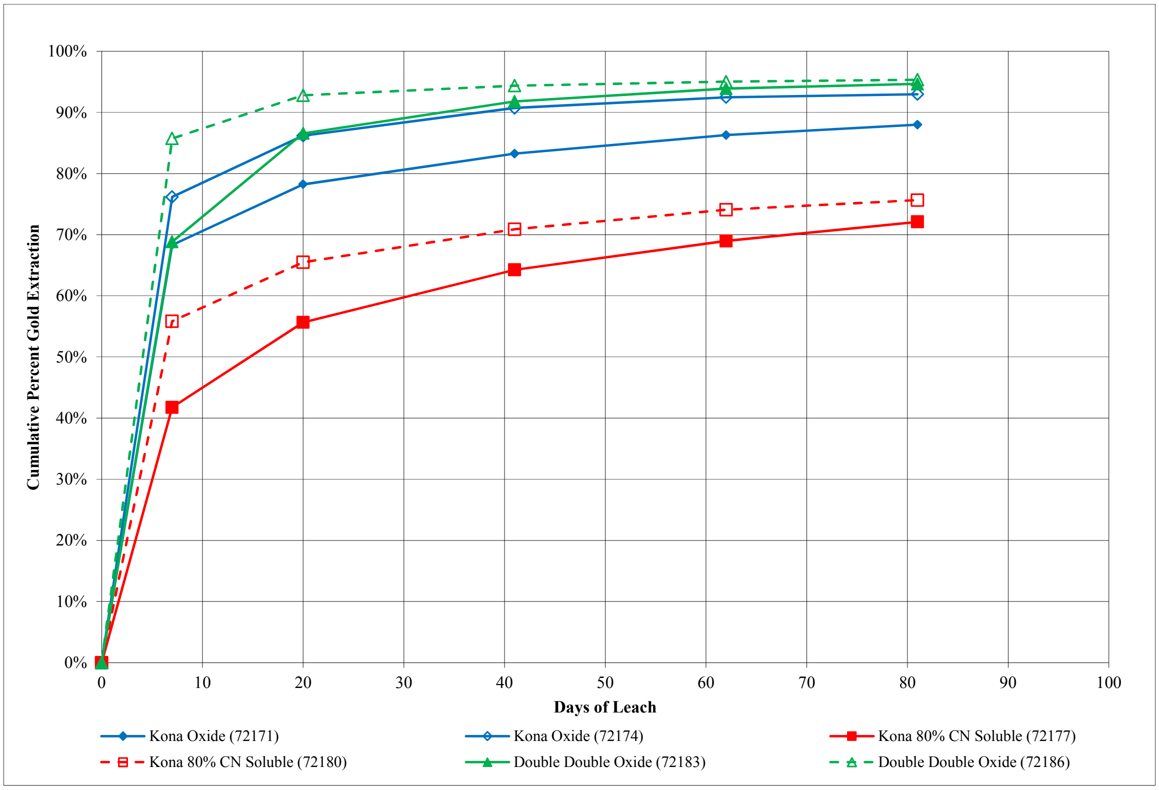
Source: KCA 2015

Figure 13.7: Supremo: Gold Extraction versus Days of Leach



Source: KCA 2015

Figure 13.8: Kona and Double Double: Gold Extraction versus Days of Leach



Source: KCA 2015

Silver recoveries ranged from 13% to 63%. The height, slump and final apparent bulk density from the column leach tests were all favourable and it was determined that no agglomeration would be necessary.

The temperature of the test did not have an effect on the rate of gold extraction or the overall gold extraction. Cyanide consumption was not affected by temperature.

When an outside party submits samples, KCA can estimate gold extraction for an ore body based upon the assumption that the ore to be mined will be similar to the samples tested. For the Coffee material, KCA would discount laboratory gold extractions by three percentage points when estimating plant recoveries.

Based upon KCA's experience with mostly clean, non-reactive ores, cyanide consumption for the Coffee material would be only 25% of the laboratory column test consumptions.

13.5.1 Cyanide Soluble Gold Test Work

The cyanide soluble testing program initiated for the PEA was continued into the Feasibility Study program. Over 14,000 samples above a fire assay value of 0.3 g/t Au within mineralized intercepts were subjected to a cyanide soluble assay tests by ALS Laboratories (Au-AA13 method).

A comparison of the column leach test recoveries from the KCA testing program, and the cyanide soluble assays from the same samples as used in the composites illustrate a strong correlation and confirms that cyanide soluble recovery is a reliable method to map the metallurgical recovery through all zones of the Coffee Deposit.

13.5.2 Crushing and Abrasion Test Work

Four Bond low impact crusher tests were conducted on the Kona, Double Double, Supremo and Latte composites. The Kona and Double Double composites were received as HQ core cut in half along the length of the core cylinder. Supremo and Latte composites were received as bulk samples. The low impact crusher tests on the Kona and Double Double composites may not be accurate due the possibility that the initial sample was too fine. The Bond low impact crusher index for the Kona and Double Double samples averaged 5.8 kWh/t. The Supremo and Latte composites averaged 11.1 kWh/t which is considered to be soft with respect to impact breakage.

Bond abrasion tests were conducted on eight samples, six of which were received as coarse crush material. The average Bond abrasion index (Ai) value for the eight samples was 0.056; with values ranging from 0.029 to 0.097. Samples with an Ai value of 0.1 or less may be considered mildly abrasive.

13.6 Metallurgical Data Input to the Feasibility Study

13.6.1 Selection of Crush Size

From the KCA testing results at the crush sizes of 80% passing (P_{80}) 150 mm and 80% passing (P_{80}) 50 mm an approximate 6% gold recovery increase was indicated by the finer crush for Supremo and an approximate 4% gold recovery increase was indicated by the finer crush for Latte. These recovery increases are offset by capital and operating requirements for the crushing circuit. Crushing to a P_{80} of 150 mm requires a single crusher whereas crushing to a P_{80} of 50 mm requires primary and secondary crushing with an associated increase in capital and operating costs.

A trade-off study conducted by JDS to evaluate the incremental economics of crushing to a $P_{80}$of 150 mm compared to a P_{80} of 50 mm was undertaken. An economic evaluation of crushing to a P_{80} of 12.5 mm was not undertaken since the recovery gains by finer crushing were not material. Crushing to P_{80} of 12.5 mm would have a higher capital and operating cost due to the three stage crushing plant that would be required and it would have more difficult operating conditions in the Yukon climate with the potential for possible heap percolation issues.

Based on a high level estimate, it was determined that the incremental operating cost of crushing to a P_{80} of 50 mm is approximately $0.56/t higher than to a P_{80} of 150 mm. At an average production rate of 5 Mt/a, this translates to an annual incremental cost of $2.8M per year. The incremental pre-production capital cost of using a P_{80} of 50 mm crushing is $8.8M.

The incremental annual operating costs and upfront capital expenditure required to proceed with P_{80} of 50 mm crushing are compensated by the increase in recoveries. Based on the assumptions used in the trade-off study, it was recommended that Kaminak implement two-stage crushing to a P_{80} of P_{80} mm. Although this results in an increased initial capital investment of approximately $8.8M, this is offset by significant increases in net present value (NPV) and IRR over the life of the mine.

13.6.2 Ultimate Recovery and Reagent Consumptions

Using the KCA recommended reduction in gold recovery of 3% and the 25% factoring for sodium cyanide consumption, estimated operating recoveries and cyanide consumptions for use in the Feasibility Study are summarized in Table 13.14. For consistency all cyanide consumptions were conservatively standardized at 0.2 kg/t and all lime consumptions were standardized at 1.5 kg/t.

Table 13.14: Gold Recoveries and Reagent Consumptions Applied to the Feasibility Study

KCA Test No.	Description	Crush Size mm	Laboratory Recovery Au %	Reagent Consumption	
				NaCN kg/t	Lime kg/t
Latte Oxides					
72114	Latte 583150mE Trench	62.5	92	0.5	1.01
72120	Latte 583350mE Trench	62.5	93	0.55	1.01
72147	Latte Oxide West	62.5	95	0.48	1.52
72153	Latte Oxide East	62.5	93	0.87	1.53
	Average		93.25	0.6	1.27
	Ultimate Recovery For FS		**90**	**0.2**	**1.5**
72159	**Latte 80% CN Soluble**	62.5	82	0.79	1.49
	Ultimate Recovery For FS		**79**	**0.2**	**1.5**
72165	**Latte 60% CN Soluble**	62.5	63%	0.76	1.52
	Ultimate Recovery For FS		**60**	**0.2**	**1.5**
Supremo Oxides					
73010	Supremo T2	62.5	93	0.96	1.51
73019	Supremo T3 Comp.	62.5	93	0.72	1.59
73025	Supremo Oxide West	62.5	90	0.49	1.51
73031	Supremo Oxide East	62.5	95	0.74	1.53
	Average		92.75	0.73	1.54
	Ultimate Recovery For FS		**90**	**0.2**	**1.5**
73037	**Supremo 80% CN Soluble**	62.5	85	0.61	1.5
	Ultimate Recovery For FS		**82**	**0.2**	**1.5**
72171	**Kona Oxide**	62.5	88	0.82	1.56
	Ultimate Recovery For FS		**85**	**0.2**	**1.5**
72177	**Kona 80% CN Soluble**	62.5	72	0.76	1.53
	Ultimate Recovery For FS		**69**	**0.2**	**1.5**
72183	**Double Double Oxide**	62.5	95	0.87	1.56
	Ultimate Recovery For FS		**92**	**0.2**	**1.5**

13.6.3 Recovery, Solution/Solids Ratio and Leach Time

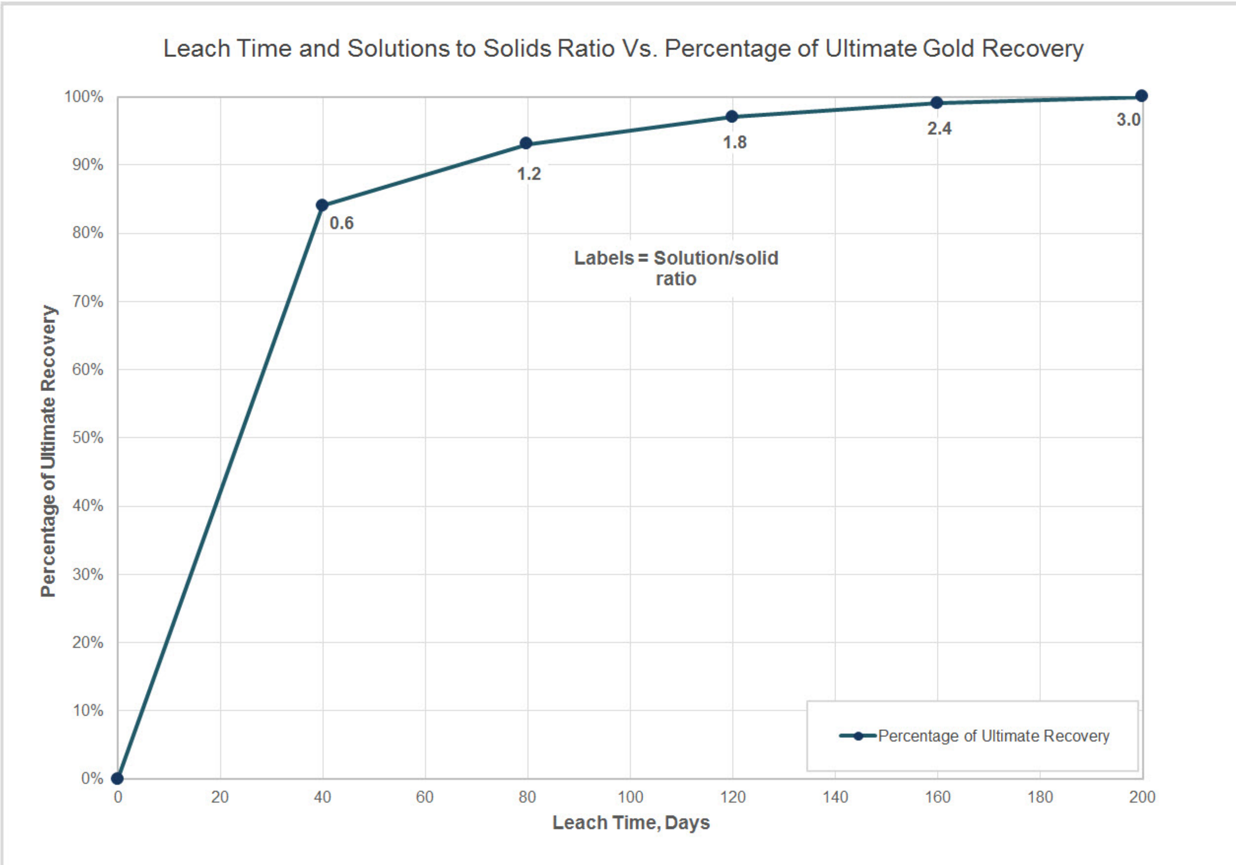
From Figure 13.8, Figure 13.9, and Figure 13.10, the leach profile for the recovery of gold shows an initial rapid recovery of the majority of gold followed by a slower leaching period to achieve ultimate gold extraction. Based on data from KCA the solution to solids ratios (tonnes of leach solution applied to tonnes of ore under leach) versus gold recovery was the basis for the development of a standardized leach profile for all Coffee ore types. This leach profile is considered the best variable for the prediction of leach cycle. The leaching period has been adjusted to fit the solution to solids ratio. For operational scale projections leaching time will be considerably longer than indicated by laboratory columns. Using a solution to solids ratio of 3.0 to achieve ultimate extraction indicates a leach time of 200 days; much longer than the laboratory leach times of between 45 and 81 days used in the test program. Table 13.16 presents the data for the leach recovery curve for all Coffee ores and Figure 13.9 illustrates the leach recovery cycle graphically. Recoveries presented are a percentage of the ultimate recoveries presented in Table 13.15.

Table 13.15: Leach Recovery Curve for Coffee Ores

Days of Leaching	Solution to Solids Ratio	% of Ultimate Recovery
0	0	0%
40	0.6	84%
80	1.2	93%
120	1.8	97%
160	2.4	99%
200	3	100%

Source: KCA and Kaminak 2015

Figure 13.9: Leach Time and Solutions to Solids Ratio verse Percentage of Ultimate Gold Recovery



Source: KCA and Kaminak 2015

Figure 13.9 in the combination with the mine plan, the heap stacking plan, and the addition rate of barren leach solution were used to calculate the annual gold production.

14 Mineral Resource Estimate

14.1 Introduction

The mineral resource statement presented herein represents the third mineral resource evaluation prepared for the Coffee Gold Project in accordance with the Canadian Securities Administrators' National Instrument (NI) 43-101. The previous resource estimates are described in technical reports dated January 10, 2013 (Chartier et al, 2013) and March 12, 2014 (Sim & Kappes, 2014).

The mineral resource estimation process was a collaborative effort between Kaminak and SIM Geological Inc. (SIM Geological). The geologic model was interpreted by Kaminak and reviewed and edited as required by SIM Geological. The geologic model was used to define resource domains to constrain gold grade estimations in the resource block models. The geostatistical analysis, variography, selection of resource estimation parameters, construction of the block model, and the conceptual pit optimization work were completed by Mr. Robert Sim, P.Geo. of SIM Geological, with the assistance of Bruce Davis, FAusIMM of BD Resource Consulting Inc. Mr. Sim is a Qualified Person pursuant to National Instrument 43-101 based on his education, work experience that is relevant to the style of mineralization, the deposit type under consideration, and to the activity undertaken, and membership to a recognized professional organization. Mr. Sim is independent from Kaminak.

The Feasibility Study drilling programs, initiated in mid-2014 and completed in mid-2015, were designed to delineate each of the gold deposits with sufficient coverage to upgrade the Inferred Resources included in the 2014 (PEA April 2014) mine plan to the Indicated category. The resource block models for the Latte, Double Double and Kona deposits were completed on March 15, 2015 and the block model for the Supremo deposit was completed on September 22, 2015.. Five additional holes were drilled during the summer of 2015 in the eastern part of the Latte deposit – an area that could not be accessed in 2014. These results are similar to the previous, adjacent drill holes and, as a result, would not materially affect the Latte resource estimate. The Latte resource model described in this report does not include these five new drill holes.

This section of the technical report describes the resource estimation methodology and summarizes the key assumptions considered by SIM Geological to prepare the resource models for the gold mineralization at the Coffee Gold Project. In the opinion of the Qualified Person, the resource evaluation reported herein is a reasonable representation of the gold mineralization found in the Coffee Gold Project at the current level of sampling. The mineral resource has been estimated in conformity with generally accepted CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (November 23, 2003) and is reported in accordance with the Canadian Securities Administrators' NI 43-101. Mineral resources are not mineral reserves and they do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into a mineral reserve upon application of modifying factors.

Estimates of mineral resources for the Supremo, Latte, Double Double and Kona deposit areas are prepared using three-dimensional block models based on geostatistical applications, and are created using commercial mine planning software (MineSight® v10.0-2). The project limits are based on the local UTM coordinate system (NAD83 Zone7).

The block size in all deposit areas measures 10 x 2.5 x 5 m with the long axis of the blocks aligned with the strike of the zone, and the shorter dimension aligned perpendicular to strike. At Latte, Double Double and Kona the long axis is oriented east-west and at Supremo the long axis is oriented north-south, consistent with the dominant strike-orientation of each mineralized zone respectively. The drilling database was developed by Kaminak from exploration programs conducted in the spring through fall field seasons from 2010 through 2015.

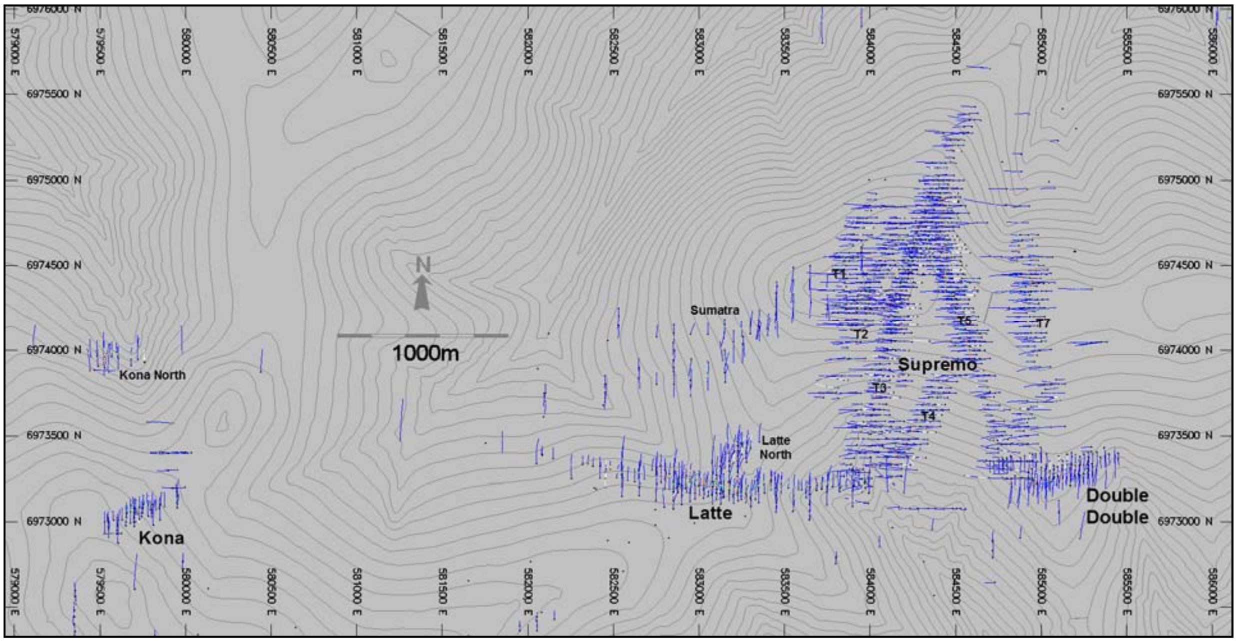
Mineral resource estimates are generated using drill hole sample assay results and the interpretation of a geologic model that relates to the spatial distribution of gold in the deposits. Interpolation characteristics were defined based on a combination of the geology, drill hole spacing, and geostatistical analysis of the data. The mineral resources are classified according to their proximity to the sample locations and are reported, as required by NI 43-101, according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014).

14.2 Available Data

There are a total of 1,682 individual drill holes in the project database with a total of 279,731 m of drilling. Of this, 636 holes (125,827 m) are diamond drill core holes and 1,046 holes (153,904 m) are reverse circulation drill holes.

The majority of the drilling is located on north-south or east-west oriented fence lines and is designed to intersect the mineralized zones at right angles. Where holes are fanned from single setup locations, the intersection angle between the drill hole and the typically steep-dipping target horizon becomes more acute with depth. In such cases, Kaminak frequently drilled parallel holes on-section from individual setup locations. Overall, drilling was conducted on a systematic pattern throughout the majority of the areas containing mineral resources. The distribution of drill holes is shown in plan in Figure 14.1.

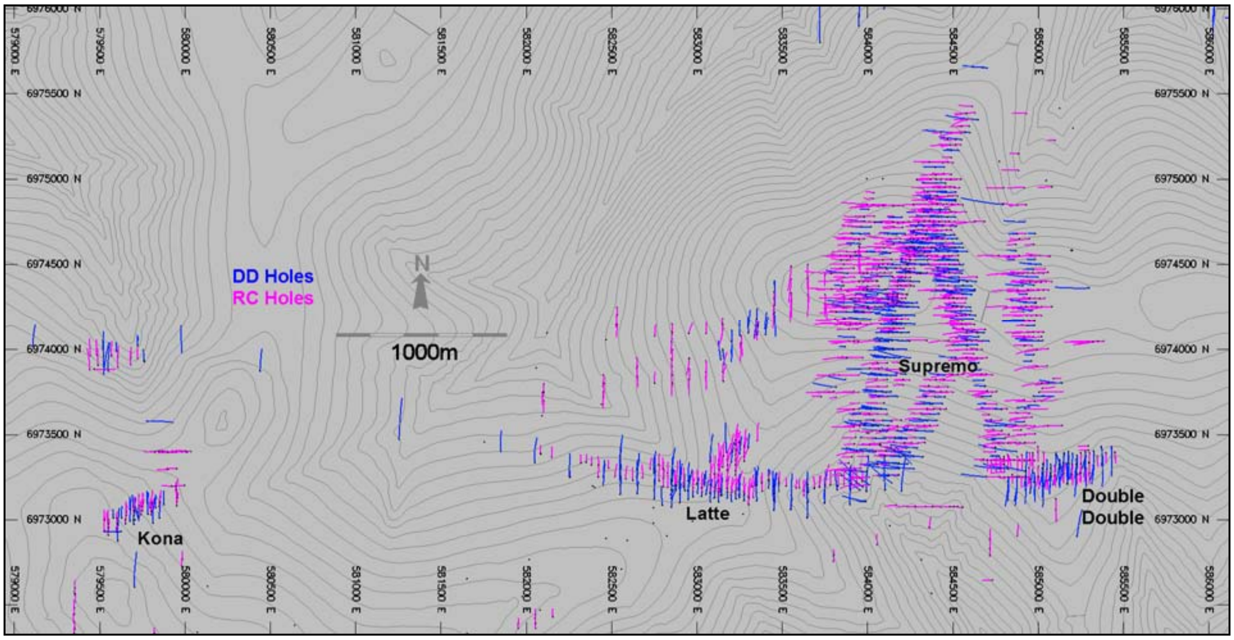
Figure 14.1: Plan View Showing the Distribution of Drill Holes and Deposit Areas



Source: SIM Geological 2016

Analysis of gold assay data shows that there is no apparent difference in grade distribution between diamond drill and reverse circulation samples. The distribution of diamond drill holes and reverse circulation drill holes is shown in Figure 14.2. Locally, there are areas where diamond drill holes are more prevalent and other areas where reverse circulation drilling was more widely used but, overall, there is a reasonably good mix of both types of drill hole.

Figure 14.2: Distribution of Diamond Drill (DD) and Reverse Circulation (RC) Drill Holes



Source: SIM Geological 2016

The project database includes resource delineation drill holes, plus other drill holes that test surrounding exploration targets, test for geotechnical or hydrogeological properties, or condemnation holes for areas where infrastructure is planned. A summary of the drill holes used in the resource models for the four deposit areas is shown in Table 14.1; this table includes only drill holes that have been used in the interpretation of the mineralized domains and therefore contribute to the development of the resource models.

Table 14.1: Summary of Drilling Used in Each Model Area to Estimate Mineral Resources

Deposit	Number of Drill Holes	Drilling (m)
Supremo	1,014	174,766
Latte	262	50,958
Double Double	118	22,671
Kona (incl. Kona North)	84	13,031

Source: SIM Geological 2016

As described previously, drilling at Latte, Double Double, and Kona is distributed on north-south oriented fence lines, and drilling at Supremo is distributed on east-west oriented fence lines. In most areas the fence lines are spaced at 25 m intervals with on-section holes intersecting the target horizon at 25 to 50 m intervals down the dip plane.

Drill hole collar locations were surveyed using a differential GPS. The collar location of each drill hole correlates very well with the local digital terrain (topographic) surface.

Fire assay (total) gold grade data were extracted from the assay database and imported into MineSight® for use in the development of the resource models. The statistical summary of the available gold grade data for each deposit area is shown in Table 14.2.

Table 14.2: Statistical Summary of Gold Assay Data by Deposit Area

Deposit	Sample Count	Total Length	Minimum	Maximum	Mean(1)	Std. Dev.
		(m)	(Au g/t)	(Au g/t)	(Au g/t)	
Supremo	120,245	156,931	0.001	86.8	0.219	1.432
Latte	51,195	47,210	0.001	65.5	0.253	1.309
Double Double	25,032	21,321	0.001	121	0.247	2.737
Kona	9,921	12,308	0.001	36.5	0.262	1.202

(1)Statistics are weighted by sample length.

Source: SIM Geological 2015

During the 2013 field season, Kaminak began testing for cyanide soluble gold (AuCN) using ALS method Au-AA13 (cold cyanide shake test). These samples were conducted on pulp rejects and target fire assay grades greater than 0.3 g/t Au as this reflects the lower detection limit of this type of analysis. This selective sampling provides a data set that is biased towards higher grade material and, as a result, does not support the ability to directly estimate AuCN grades in model blocks. However, ratios of AuCN/Total Au are used to support the interpretation of a series of oxide zones that represent domains with differing metallurgical properties.

Additional data used in the interpretation of the geologic model included lithologic designations obtained during geologic logging of the drill core and reverse circulation chips. Surface geologic mapping, trenching, contouring of soil geochemical sampling, aeromagnetic interpretation and topographic lineament interpretation contributed to the interpretation of the geological structures at surface.

Kaminak provided a topographic digital terrain surface as a gridded point file (x, y, z coordinates) that was originally produced using contour lines spaced at 10 m intervals. These data were derived from a LiDAR survey of the area conducted by Eagle Mapping Ltd. in 2010.

Individual sample intervals range from 0.1 m to 7.62 m in length and average 1.25 m. The standard sample interval for a diamond drill hole is 1 m, except at Double Double where the 2012 drill holes were sampled at 0.5 m intervals. Reverse circulation drilling was sampled at 1.52 m (5-foot) intervals.

Specific gravity (bulk density) determinations were conducted on 6,734 samples in the database. Samples for specific gravity measurements were typically selected at 10 m intervals down most of the diamond drill holes. The frequency of specific gravity measurements increases in some drill holes in the areas within the interpreted mineralized domains.

Core recovery for essentially all diamond drill holes averaged 96%. Ninety-six percent of the sample intervals showed core recoveries greater than 80%, and only about 1% of sample intervals had recoveries below 50%. Kaminak's standard procedure during infill drilling was to re-drill the hole if an expected mineralized intercept returned less than 50% recovery. There is no apparent relationship between recovery and gold content in the diamond drill holes.

Recovery data are not available for reverse circulation drilling but in-situ inspection, by the Qualified Person, of the reverse circulation drilling and sampling procedures indicates that recoveries are very good. There is a loss of very fine dust generated during drilling, but this represents a very small volume of material, and it is not believed to affect the samples to any measurable degree. Numerous reverse circulation reject samples were observed in the field; they show very consistent sample sizes which is a reflection of the consistent nature of reverse circulation recoveries throughout the drilling process. There were no adjustments or omissions to the database in response to diamond drill or reverse circulation sample recoveries.

14.3 Geologic Model and Estimation Domains

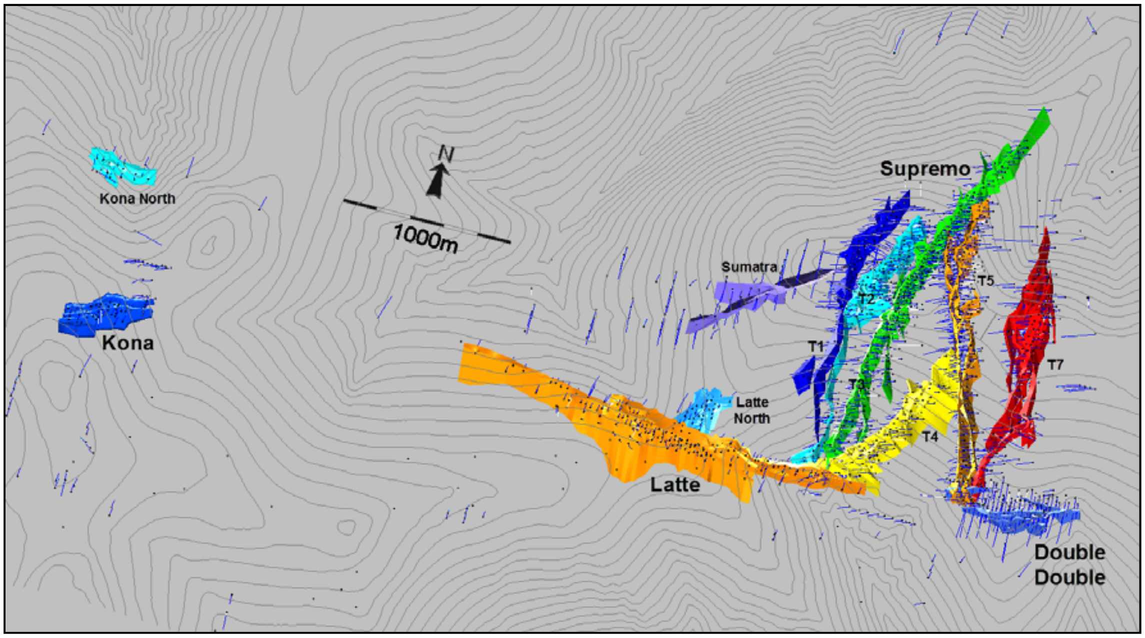
14.3.1 Mineral Domains and Mineralized Zones

Gold mineralization at Coffee Gold is located within a series of steeply dipping structures that cross-cut all rock units on the property. The structural zones are identified in the drill core and from surface mapping and trenching. Soil sampling has also located gold-in-soil anomalies in many areas which were subsequently drilled. Although the nature of these structural zones can exhibit a variety of characteristics, including faulting, brecciation, silicification, alteration, and local sulphide veining, they can be traced over strike lengths up to 2.5 km.

Kaminak geologists interpreted a series of "mineralized" or "mineral" domains in each resource area using a combination of surface mapping, geologic core (and reverse circulation chips) logging, and the distribution of gold grades in drill hole sample data. These domains encompass rocks that exhibit geologic conditions with the potential to host gold mineralization and, in most cases, contain elevated gold grades. In previous resource estimations, interpreted "structural" domains relied primarily on geologic conditions that were favourable to potentially host gold mineralization. These domains locally included areas that did not contain appreciable quantities of gold. With the increased density of drilling, resulting from drill holes added during the June 2014 to June 2015 infill drilling program, the confidence in the continuity of gold mineralization between drill holes has increased and the interpretation of these domains now targets the presence of gold mineralization generally above a threshold grade of 0.1 g/t Au. This resulted in a change to the naming convention for domains: the previous application of "structural" domains was replaced by the current application of "mineral" domains.

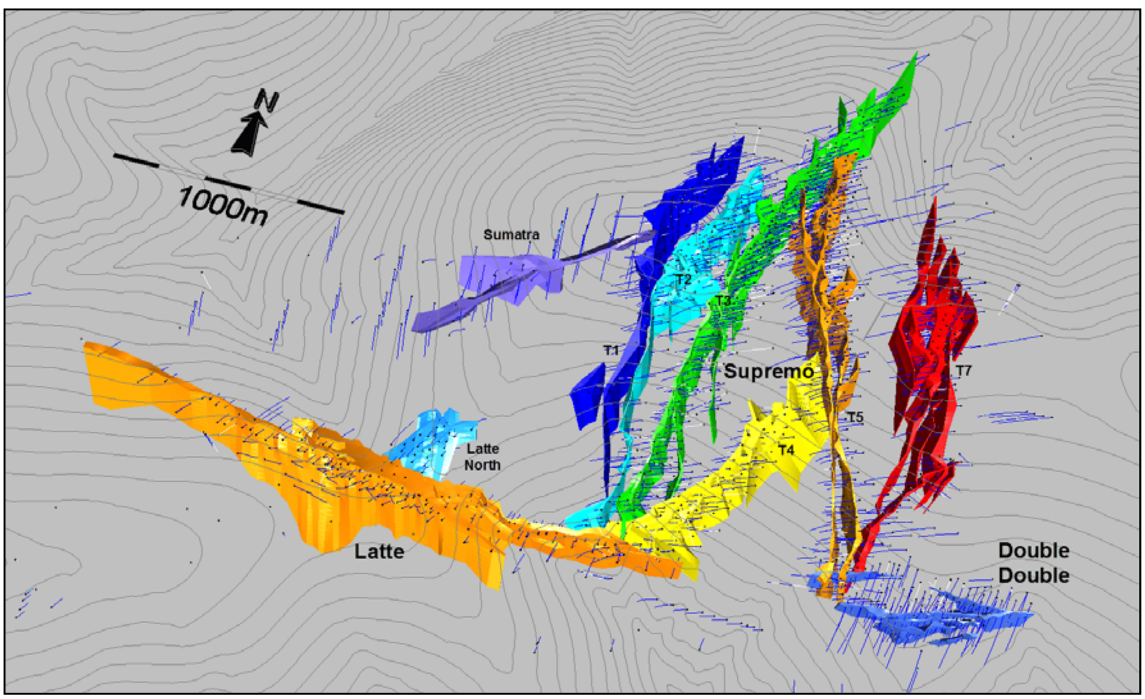
The distribution of the mineral domains is shown in Figure 14.3. The mineral domains at Latte, Supremo, and Double Double are shown in more detail in Figure 14.4. The individual areas at Supremo, (T1, T2, T3, T4, T5, and T7) are named after the trenches that were initially used to investigate the surface mineralization in these areas.

Figure 14.3: Mineral Domains at Supremo, Latte, Double Double, and Kona



Source: Source: SIM Geological 2015

Figure 14.4: Mineral Domains at Supremo, Latte, and Double Double

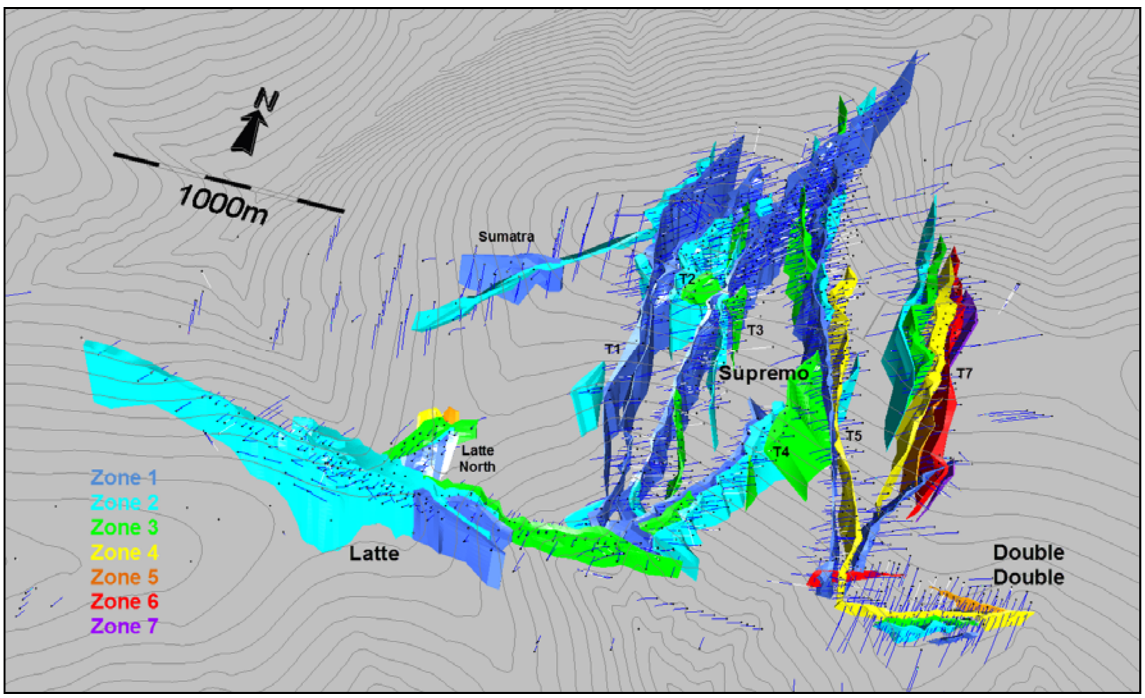


Source: SIM Geological 2015

Each deposit area comprises a series of sub-parallel, often braided mineralized zones that coalesce and bifurcate along the general strike-orientation of the mineral domain. Individual mineralized zones have been subdivided for modelling purposes and, within each zone, a three-dimensional plane has been interpreted that represents the overall trend of the gold mineralization for that part of the deposit. These trend planes are then used to orient search directions so that samples of a similar nature are related during grade interpolation in the block model. This approach introduces a dynamic, anisotropic search process during block-grade interpolation that reproduces the locally complex, undulating, and banded nature of the gold mineralization in the resource block models that would otherwise be impossible to achieve using traditionally oriented search ellipses.

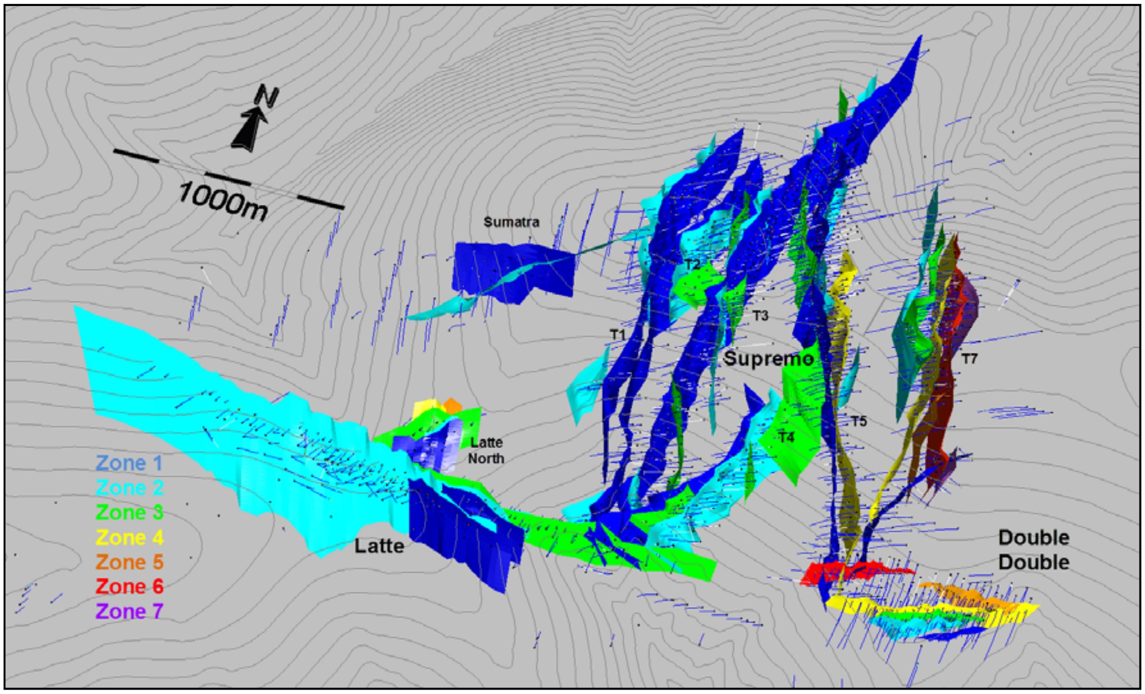
Figure 14.5 shows the individual mineralized zones defined at Supremo, Latte, and Double Double, and Figure 14.6 shows the trend planes defined for each individual mineralized zone.

Figure 14.5: Individual Mineralized Zones Defined at Supremo, Latte, and Double Double



Source: SIM Geological 2015

Figure 14.6: Planes Representing Trends of Mineralization in each Mineralized Zone



Source: SIM Geological 2015

14.3.2 Oxidation Model

Since the 2013 field season Kaminak has conducted a re-assaying program on all laboratory pulps that tests the cyanide soluble characteristics of the gold in the deposit. Samples selected for AuCN analysis are restricted to samples where total fire assay gold grades are greater than 0.3 g/t, which is the lower limit of detection for the cyanide leach analytical method (the cyanide leach analytical method is described in more detail in Section 9 – Exploration). This data is reasonably distributed but, because they exclude lower grade sample intervals, they are not sufficient to support direct estimation of AuCN estimates in model blocks. As an alternative, the ratio of AuCn/total Au was calculated in samples where AuCN data are present. These ratios are then interpolated in the block model and are used in combination with qualitative (visual) estimates of the intensity of oxidation to provide information regarding the depth and intensity of oxidation in the mineralized domains.

Oxidation appears to be channeled along the structural corridors that host the deposits. It is common to find intense oxidation at depths below 200 m from surface within these structures. Strong oxidation is present over the majority of the Supremo deposit, but it is less pervasive and more variable at Latte, Double Double, and Kona. Outside the interpreted mineral domains, rocks show only weak signs of near-surface oxidation.

AuCN/Total Au ratios are interpolated into model blocks located inside the mineral domains using the dynamic anisotropy controlled by the trends of the mineralization. The resulting ratios are used to define the following five oxide types (the designation of these five oxide types are based on metallurgical test work).

- Oxide: intense to pervasive oxidation (AuCN/Total Au > 0.90);
- Upper Transition: moderate to intense oxidation (AuCN/Total Au = 0.70–0.89);
- Middle Transition: moderate oxidation (AuCN/Total Au = 0.50–0.69);
- Lower Transition: weak to moderate oxidization (AuCN/Total Au = 0.10–0.49); and
- Sulphide: fresh to weak oxidization (AuCN/Total Au < 0.10).

Outside the mineral domains, there is little to no cyanide soluble sample data to assist in defining oxidation states. The type of oxidation present outside the mineral domains is interpreted based on visual observations during drill core and chip logging.

A surface representing the top of bedrock has been interpreted using the thickness of overburden intersected in drilling. Although overburden is present across most of the deposit areas, it is typically less than 5 m thick.

14.4 Compositing

Compositing drill hole samples standardizes the database for further statistical evaluation. This step eliminates any effect the sample length may have on the data.

To retain the original characteristics of the underlying data, a composite length that reflects the average original sample length is selected: an excessively long composite can sometimes result in a degree of smoothing that can mask certain features of the data. The majority of samples were taken at two standard lengths: 1.00 m in diamond drilling, and 1.52 m (5 feet imperial) in reverse circulation drilling, with an average of 1.18 m. A standard composite length of 1.00 m was used for geostatistical analysis and grade estimation.

Drill hole composites are length-weighted and are generated “down-the-hole”, meaning composites begin at the top of each drill hole and are generated at 1 m intervals down the length of the drill hole. Composites honour the structural domain contacts (individual composites begin and end at the point where a drill hole crosses the domain boundary). Several drill holes were randomly selected and the composited values were checked for accuracy. No errors were found.

14.5 Exploratory Data Analysis

Exploratory data analysis (EDA) involves statistically summarizing the database to better understand the characteristics of the data that may control gold grade. One of the main purposes of EDA is to determine if there is evidence of spatial distinctions in gold grade. This would require the separation and isolation of domains during interpolation.

The application of separate domains prevents unwanted mixing of data during interpolation, and the resulting grade model will better reflect the unique properties of the deposit. However, applying domain boundaries in areas where the data are not statistically unique may impose a bias in the distribution of gold grades in the model.

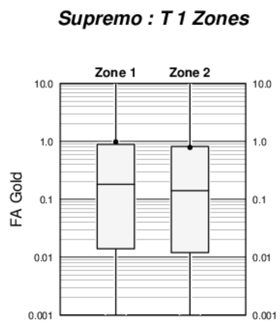
A domain boundary, which segregates the data during interpolation, is typically applied if the average grade in one domain is significantly different from another. A domain boundary may also be applied where a significant change in the grade distribution exists across the contact.

14.5.1 Basic Statistics by Domain

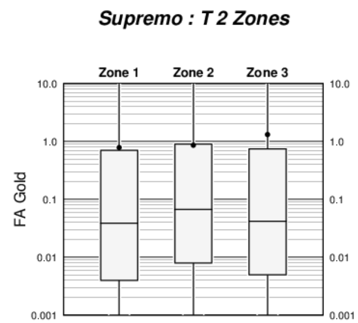
Frequency distributions and associated summary statistics are compared using a series of boxplots. The boxplots compare the assay frequency distributions by individual mineralized zones in each model area. Examples from the four deposit areas are shown in Figure 14.7, Figure 14.8, Figure 14.9, and Figure 14.10.

There are differences between the individual structural zones, and these typically show higher gold content compared to the surrounding samples. Note the variability between some of the individual structural zones. Some of the interpreted zones contain relatively low amounts of gold.

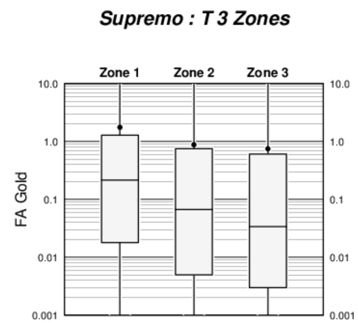
Figure 14.7: Boxplot for Gold in Mineralized Zones at Supremo



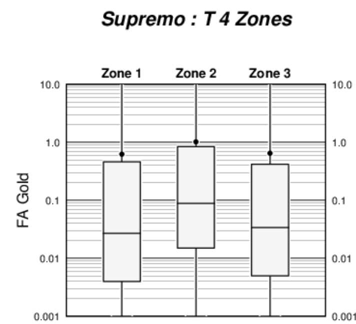
	Zone 1	Zone 2
Number of data	1315	722
Mean	0.987	0.791
Variance	5.493	3.249
Skewness	5.945	5.642
Coef of Variation	2.375	2.277
Maximum	36.2	21.9
Upper quartile	0.899	0.82
Median	0.182	0.142
Lower quartile	0.014	0.012
Minimum	0.001	0.001



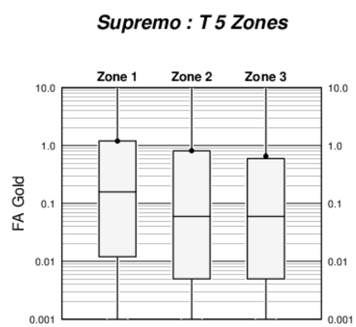
	Zone 1	Zone 2	Zone 3
Number of data	2361	1644	186
Mean	0.788	0.863	1.321
Variance	5.0	4.197	16.791
Skewness	10.596	6.227	6.835
Coef of Variation	2.839	2.374	3.103
Maximum	57.7	31.3	39.8
Upper quartile	0.707	0.904	0.743
Median	0.039	0.067	0.042
Lower quartile	0.004	0.008	0.005
Minimum	0.001	0.001	0.001



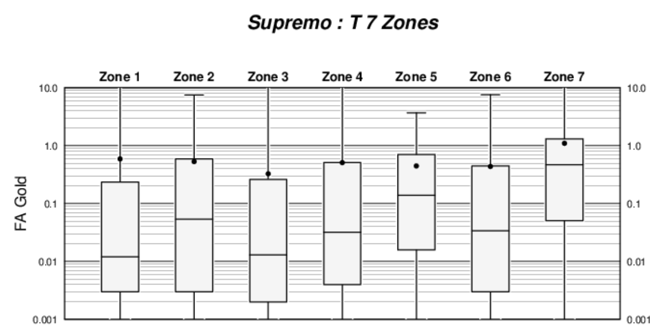
	Zone 1	Zone 2	Zone 3
Number of data	5689	1306	383
Mean	1.767	0.885	0.751
Variance	27.53	4.945	3.975
Skewness	7.222	5.688	7.633
Coef of Variation	2.969	2.513	2.655
Maximum	84.8	33.4	27.6
Upper quartile	1.284	0.751	0.612
Median	0.216	0.067	0.034
Lower quartile	0.018	0.005	0.003
Minimum	0.001	0.001	0.001



	Zone 1	Zone 2	Zone 3
Number of data	1433	942	1093
Mean	0.626	1.022	0.65
Variance	2.574	7.931	2.689
Skewness	5.519	6.341	4.449
Coef of Variation	2.562	2.756	2.524
Maximum	20.4	33.8	17.25
Upper quartile	0.461	0.843	0.42
Median	0.027	0.089	0.034
Lower quartile	0.004	0.015	0.005
Minimum	0.001	0.001	0.001



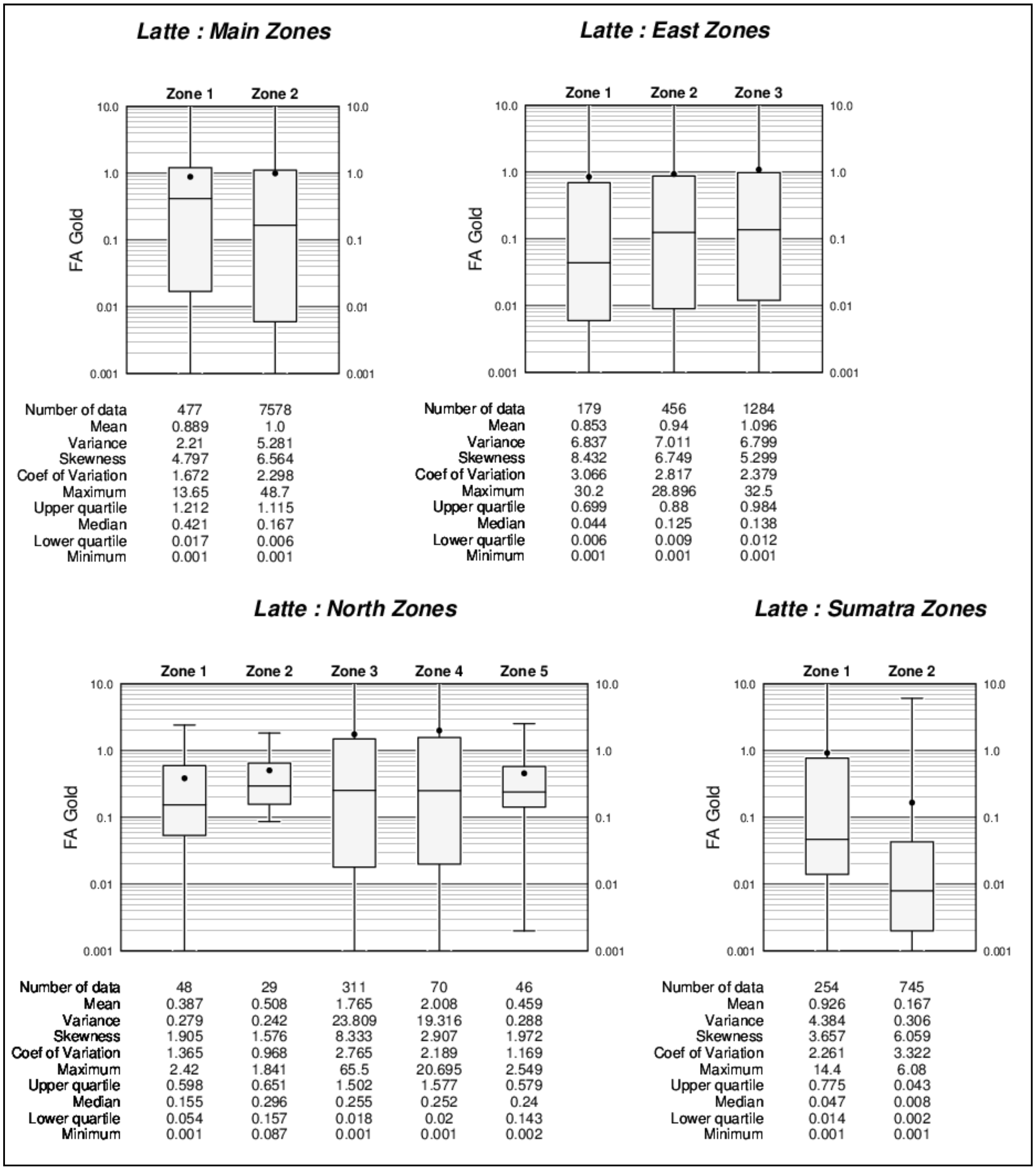
	Zone 1	Zone 2	Zone 3
Number of data	4601	1585	997
Mean	1.203	0.82	0.66
Variance	6.732	2.928	3.973
Skewness	5.171	3.9	8.866
Coef of Variation	2.156	2.086	3.019
Maximum	50.3	18.0	31.588
Upper quartile	1.21	0.812	0.601
Median	0.159	0.06	0.06
Lower quartile	0.012	0.005	0.005
Minimum	0.001	0.001	0.001



	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	Zone 6	Zone 7
Number of data	564	654	999	2676	74	625	94
Mean	0.592	0.533	0.328	0.511	0.448	0.439	1.102
Variance	3.483	1.027	0.755	1.15	0.406	0.784	2.877
Skewness	5.091	2.921	5.947	4.117	2.506	3.386	2.861
Coef of Variation	3.154	1.903	2.647	2.1	1.422	2.017	1.538
Maximum	17.55	7.5	12.05	14.3	3.72	7.6	10.75
Upper quartile	0.237	0.591	0.262	0.514	0.702	0.447	1.31
Median	0.012	0.054	0.013	0.032	0.139	0.034	0.471
Lower quartile	0.003	0.003	0.002	0.004	0.016	0.003	0.051
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001

Source: SIM Geological 2016

Figure 14.8: Boxplot for Gold in Mineralized Zones at Latte



Latte : Main Zones

	Zone 1	Zone 2
Number of data	477	7578
Mean	0.889	1.0
Variance	2.21	5.281
Skewness	4.797	6.564
Coef of Variation	1.672	2.298
Maximum	13.65	48.7
Upper quartile	1.212	1.115
Median	0.421	0.167
Lower quartile	0.017	0.006
Minimum	0.001	0.001

Latte : East Zones

	Zone 1	Zone 2	Zone 3
Number of data	179	456	1284
Mean	0.853	0.94	1.096
Variance	6.837	7.011	6.799
Skewness	8.432	6.749	5.299
Coef of Variation	3.066	2.817	2.379
Maximum	30.2	28.896	32.5
Upper quartile	0.699	0.88	0.984
Median	0.044	0.125	0.138
Lower quartile	0.006	0.009	0.012
Minimum	0.001	0.001	0.001

Latte : North Zones

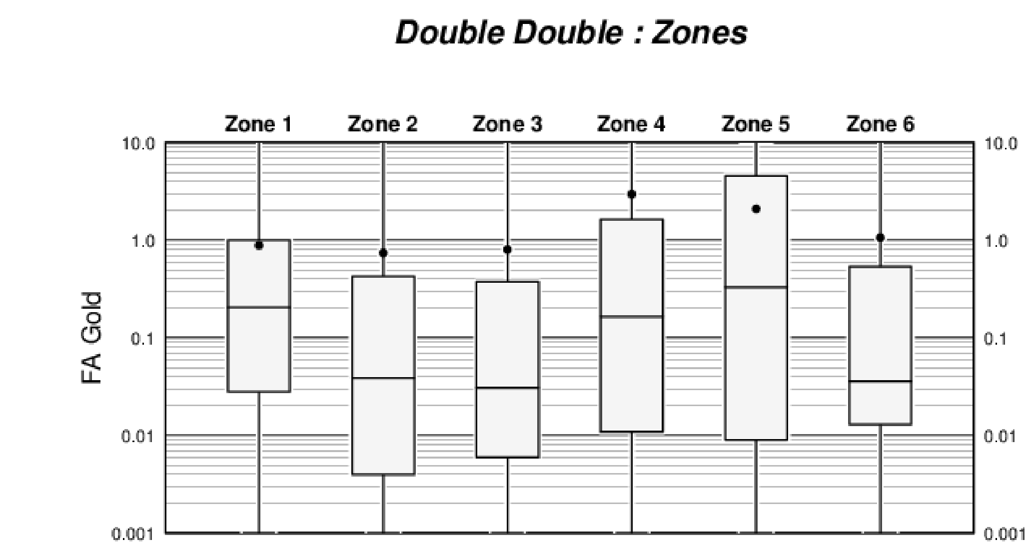
	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
Number of data	48	29	311	70	46
Mean	0.387	0.508	1.765	2.008	0.459
Variance	0.279	0.242	23.809	19.316	0.288
Skewness	1.905	1.576	8.333	2.907	1.972
Coef of Variation	1.365	0.968	2.765	2.189	1.169
Maximum	2.42	1.841	65.5	20.695	2.549
Upper quartile	0.598	0.651	1.502	1.577	0.579
Median	0.155	0.296	0.255	0.252	0.24
Lower quartile	0.054	0.157	0.018	0.02	0.143
Minimum	0.001	0.087	0.001	0.001	0.002

Latte : Sumatra Zones

	Zone 1	Zone 2
Number of data	254	745
Mean	0.926	0.167
Variance	4.384	0.306
Skewness	3.657	6.059
Coef of Variation	2.261	3.322
Maximum	14.4	6.08
Upper quartile	0.775	0.043
Median	0.047	0.008
Lower quartile	0.014	0.002
Minimum	0.001	0.001

Source: SIM Geological 2016

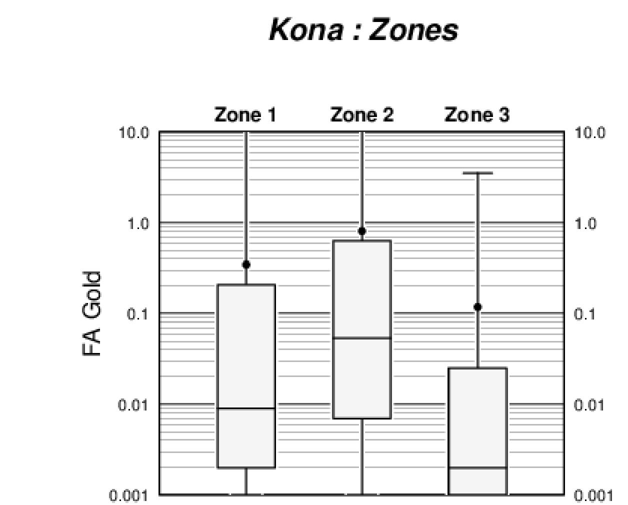
Figure 14.9: Boxplot for Gold in Mineralized Zones at Double Double



	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	Zone 6
Number of data	134	398	463	1140	48	466
Mean	0.887	0.744	0.804	2.965	2.108	1.065
Variance	3.237	4.702	11.791	104.306	8.778	22.855
Skewness	4.13	6.531	9.2	7.16	1.252	10.357
Coef of Variation	2.028	2.916	4.273	3.444	1.406	4.489
Maximum	13.6	26.4	47.5	121.0	10.35	66.78
Upper quartile	1.002	0.429	0.377	1.638	4.576	0.536
Median	0.207	0.039	0.031	0.165	0.329	0.036
Lower quartile	0.028	0.004	0.006	0.011	0.009	0.013
Minimum	0.001	0.001	0.001	0.001	0.001	0.001

Source: SIM Geological 2016

Figure 14.10: Boxplot for Gold in Mineralized Zones at Kona

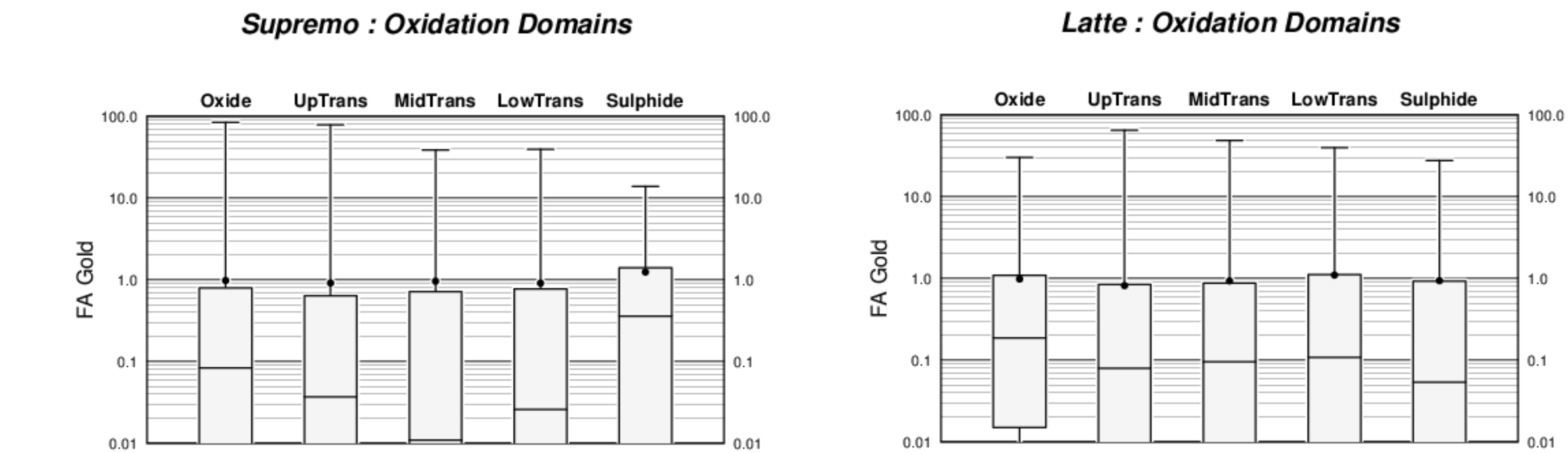


	Zone 1	Zone 2	Zone 3
Number of data	1231	1899	409
Mean	0.348	0.814	0.118
Variance	0.833	4.335	0.151
Skewness	5.323	6.465	5.459
Coef of Variation	2.621	2.557	3.304
Maximum	10.9	36.5	3.541
Upper quartile	0.208	0.632	0.025
Median	0.009	0.054	0.002
Lower quartile	0.002	0.007	0.001
Minimum	0.001	0.001	0.001

Source: SIM Geological 2016

The boxplot in Figure 14.11 shows the distribution of total gold by oxide domain at Supremo and Latte. Gold grades do not vary much between these domains. The majority of samples at Supremo comprise pervasively oxidized rocks whereas the distribution of oxidation at Latte is more variable.

Figure 14.11: Boxplot for Total Gold in Oxide Domains at Supremo and Latte



Supremo	Oxide	UpTrans	MidTrans	LowTrans	Sulphide
Number of data	27199	1979	761	954	115
Mean	0.982	0.916	0.964	0.913	1.247
Variance	8.497	10.625	7.498	6.251	5.357
Skewness	9.808	13.169	6.684	7.572	3.413
Coef of Variation	2.968	3.56	2.84	2.737	1.856
Maximum	84.8	78.7	38.9	39.8	13.9
Upper quartile	0.798	0.64	0.72	0.779	1.411
Median	0.084	0.037	0.011	0.026	0.361
Lower quartile	0.008	0.003	0.002	0.002	0.006
Minimum	0.001	0.001	0.001	0.001	0.001

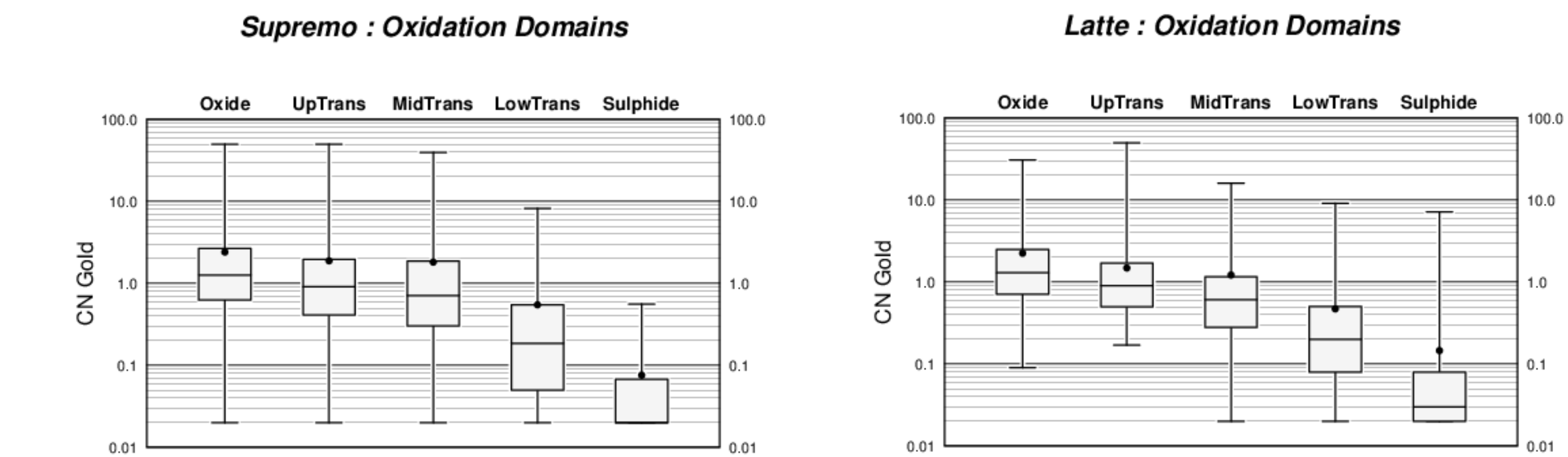
Latte	Oxide	UpTrans	MidTrans	LowTrans	Sulphide
Number of data	4604	1985	1314	2331	1229
Mean	0.987	0.819	0.936	1.102	0.942
Variance	5.087	5.795	6.246	7.051	5.302
Skewness	6.176	13.254	8.399	5.608	5.722
Coef of Variation	2.286	2.94	2.669	2.41	2.446
Maximum	30.2	65.5	48.7	39.9	27.9
Upper quartile	1.095	0.847	0.878	1.12	0.933
Median	0.187	0.08	0.096	0.109	0.054
Lower quartile	0.015	0.005	0.003	0.003	0.004
Minimum	0.001	0.001	0.001	0.001	0.001

Source: SIM Geological 2016

The distribution of AuCN data by oxide domain is shown in Figure 14.12: although total gold does not vary much between oxide domains, the AuCN grades consistently decrease with decreasing intensity of oxidation.

Figure 14.12 shows the ratio of cyanide soluble gold versus total gold in each of the five oxide domains. There are distinct differences evident between domains, especially in the less-oxidized types, where there is little to no overlap in the distribution of these ratios in contained sample data.

Figure 14.12: Boxplot for Cyanide Soluble Gold in Oxide Domains at Supremo and Latte



Supremo : Oxidation Domains

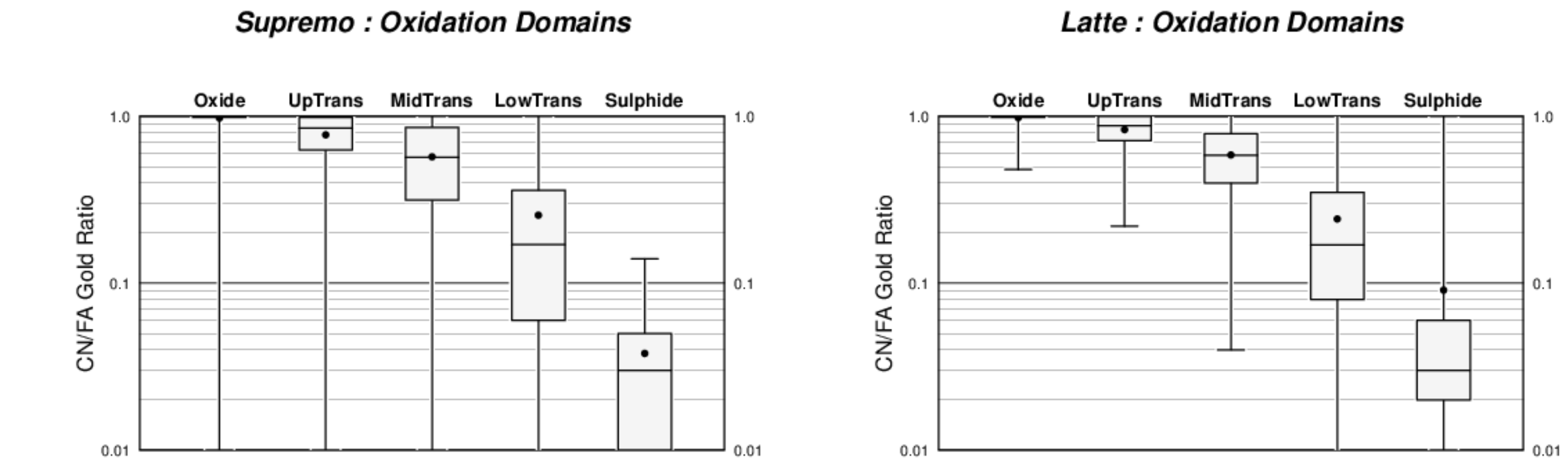
	Oxide	UpTrans	MidTrans	LowTrans	Sulphide
Number of data	9888	583	225	294	60
Mean	2.415	1.887	1.82	0.549	0.076
Variance	14.11	11.445	13.839	0.983	0.014
Skewness	5.608	7.05	6.287	3.824	2.789
Coef of Variation	1.556	1.793	2.044	1.805	1.551
Maximum	50.0	50.0	39.7	8.24	0.56
Upper quartile	2.68	1.972	1.871	0.551	0.068
Median	1.26	0.917	0.71	0.185	0.02
Lower quartile	0.63	0.41	0.304	0.05	0.02
Minimum	0.02	0.02	0.02	0.02	0.02

Latte : Oxidation Domains

	Oxide	UpTrans	MidTrans	LowTrans	Sulphide
Number of data	1945	589	390	805	453
Mean	2.249	1.487	1.22	0.471	0.146
Variance	9.554	6.733	3.755	0.751	0.296
Skewness	4.623	12.203	4.033	5.277	8.513
Coef of Variation	1.374	1.745	1.588	1.838	3.725
Maximum	30.8	50.0	16.05	9.15	7.15
Upper quartile	2.5	1.71	1.16	0.502	0.08
Median	1.3	0.9	0.61	0.2	0.03
Lower quartile	0.71	0.5	0.28	0.08	0.02
Minimum	0.09	0.17	0.02	0.02	0.02

Source: SIM Geological 2016

Figure 14.13: Boxplot for Ratio of AuCN/Au in Oxide Domains at Supremo and Latte



Supremo

	Oxide	UpTrans	MidTrans	LowTrans	Sulphide
Number of data	10011	591	225	294	60
Mean	0.979	0.776	0.573	0.256	0.038
Variance	0.005	0.055	0.096	0.066	0.001
Skewness	-7.083	-1.047	-0.153	1.42	1.235
Coef of Variation	0.07	0.303	0.542	1.007	0.782
Maximum	1.0	1.0	1.0	1.0	0.14
Upper quartile	1.0	0.99	0.86	0.36	0.05
Median	1.0	0.85	0.57	0.171	0.03
Lower quartile	0.99	0.63	0.315	0.06	0.01
Minimum	0.01	0.01	0.01	0.0	0.01

Latte

	Oxide	UpTrans	MidTrans	LowTrans	Sulphide
Number of data	1945	589	390	805	453
Mean	0.983	0.834	0.588	0.243	0.091
Variance	0.002	0.032	0.067	0.049	0.042
Skewness	-4.681	-1.058	-0.001	1.418	3.868
Coef of Variation	0.047	0.213	0.439	0.907	2.256
Maximum	1.01	1.01	1.33	1.05	1.0
Upper quartile	1.0	1.0	0.79	0.35	0.06
Median	1.0	0.88	0.585	0.17	0.03
Lower quartile	0.99	0.717	0.4	0.08	0.02
Minimum	0.48	0.22	0.04	0.0	0.01

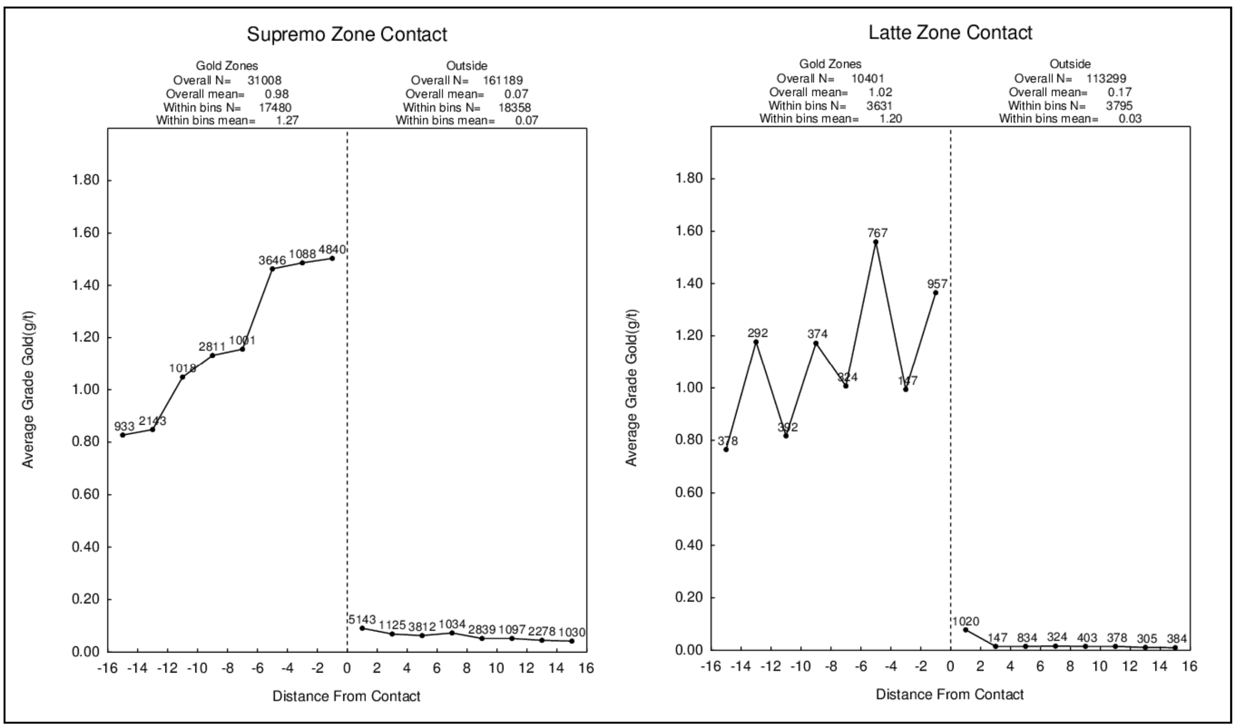
Source: SIM Geological 2016

14.5.2 Contact Profiles

Contact profiles evaluate the nature of gold grade trends between two domains. They graphically display the average grades at increasing distances from the contact boundary. Those contact profiles that show a marked difference in gold grades across a domain boundary indicate that the two datasets should be isolated during interpolation. Conversely, if a more gradual change in grade occurs across a contact, the introduction of a hard boundary (in other words, segregation during interpolation) may result in much different trends in the grade model; in this case, the change in grade between model domains is often more abrupt than the trends seen in the raw data. Finally, a flat contact profile indicates that there are no grade changes across the boundary; in this case, hard or soft domain boundaries will produce similar results in the model.

A series of contact profiles were generated that compare sample data inside and sample data outside the interpreted mineral domains. Figure 14.14 shows examples from Supremo and Latte. There is a marked difference in gold grade between samples inside the mineral domains compared to the surrounding data. This trend is similar for all deposit areas.

Figure 14.14: Contact Profile Comparing Samples Inside/Outside the Mineral Domains at Supremo and Latte



Source: SIM Geological 2016

14.5.3 Modelling Implications

Boxplots show that similarities and differences exist in the gold content of the individual mineralized zones in each of the deposit areas. Most mineral domains contain one continuous, main mineralized zone that encompasses the majority of the sample data and, in most cases, has the highest average grade. The contact profiles show that the interpreted mineral domains contain samples with significantly higher grades and that the change in grade often occurs abruptly at the domain boundary. It was concluded that the interpreted mineral domains contain gold grades that are sufficiently different than surrounding samples, and that these data should be segregated during model grade interpolations.

Although the results show that some differences exist between individual mineralized zones, they tend to be somewhat subtle. The individual mineralized zones represent individual bands of mineralization. The segregation of these zones is primarily based on differences in the trends and continuity of the mineralization rather than differences in grade between zones. Therefore, segregation of these zones allows for better reproduction of the interpreted trends of gold mineralization in the resource model and ensures that sample data between individual trends are not mixed during block-grade interpolation.

At Latte, Double Double, and Kona, the degree of oxidation can be quite complex and variable as it transitions from true oxide to sulphide material. At Supremo, pervasive intense oxidation is present throughout the majority of the mineral domains. Gold solubility ratios have been used to help interpret these domains and they represent areas with similar oxidation properties. The generalization of the degree of oxidation through the transition zones is considered appropriate to provide a basis for estimating metallurgical recoveries from the resource model. There is little difference in (total) gold grade based on varying degrees of oxidation and, as a result, the oxide domains are not used during interpolation of gold grade in the resource model.

14.5.4 Conclusions

Each deposit area contains two or more individual mineralized zones that are used as hard-boundary domains during the development of the resource model. This means that data are not mixed between zones during block-grade interpolation. The resulting mineral domains and mineralized zones are summarized in Table 14.3. Note that the area outside the structural zone is essentially barren and shows no potential for economic gold resources. No grade estimates were conducted outside the mineral domains.

Table 14.3: Summary of Estimation Domains

Deposit/Mineral Domain	Comments
Supremo	
T1 area	Two mineralized zones. Sub-vertical trending 10° azimuth.
T2 area	Three mineralized zones. The main zone has a strike length of over 1.6 km. All with approximately 15° azimuth and -80° dip to the east.
T3 area	Three mineralized zones. The main zone extends over 2.5 km with 20° azimuth and -85° dip to the east. T3 contains some of the higher grade resources on the property.
T4 area	Three mineralized zones with 35° azimuth and -70° to -80° dip to the south-east. The largest of the three zones has a strike length of about 800 m.
T5 area	Four mineralized zones. The main zone has a strike length of almost 1.8 km. In general, these have 345° azimuth and -85° dip to the east. The north end of T5 swings sub-parallel to T3.
T7 area	Seven mineralized zones that trend north-south and dip steeply to the east.
Latte	
Main area	Two mineralized zones. A thicker main zone with 100° azimuth and -65° dip to the south. A second zone is a “splay” extending from the east end of the Main zone.
East area	Three mineralized zones extending eastward from the east end of the Main area. Sub-vertical, east-west trending, zones with variable strike length up to 1 km. One of these zones is interpreted to intersect the south end of Supremo T2, T3, and T4 at what is referred to as the “connector” zone.
North area	Five mineralized zones interpreted as splays from the Main zone with 55° azimuth and -55° dip SE.
Sumatra	Two mineralized zones approximately 700 m north of the Main Latte deposit. Trend 55° and dip -60° north-west. Interpreted to intersect with Supremo T1 to the east.
Double Double	Six mineralized zones with 255° azimuth and -85° dip to the north.
Kona	Three mineralized zones with 70° azimuth and -85° dip to the south. Strike length approximately 500 m.
Kona North	3 mineralized zones with 100° azimuth and -55° dip to the south. Strike length about 400 m.

Source: SIM Geological 2016

14.6 Specific Gravity Data

The methodology used to generate the specific gravity database is described in detail in Section 10 of the January 2014 technical report (Sim & Kappes, 2014).

There are 6,734 samples tested for specific gravity (SG). Samples for SG measurements were typically selected at 10 m intervals down most of the diamond drill holes. In some drill holes the frequency of SG measurements increases within the interpreted mineralized domains. Approximately 26% of these samples occur inside one of the interpreted mineral domains and the remaining 74% occur outside the interpreted mineral domains. SG data are only collected from diamond drill (core) holes. Although the SG database is relatively large, the distribution of SG data is locally sparse and therefore is insufficient to support direct interpolation into model blocks.

A relationship exists between the density of rocks and the intensity of oxidation, a common feature in deposits of this type. Average densities have been determined by oxide type and are assigned to model blocks as shown in Table 14.4. The assigned values differ slightly by deposit area. There is a relative lack of density data in some oxide domains at Double Double and Kona; average values derived from available data at Latte have been used for these two areas.

Table 14.4 lists the average specific gravity values used to calculate tonnages in each of the oxide zones.

Table 14.4: Summary of Specific Gravity Data by Oxide Type

Oxide Type	Latte, DD, Kona Inside Mineral Domains (t/m^3)	Supremo Inside Mineral Domains (t/m^3)	Outside Mineral Domains (t/m^3)
Oxide	2.54	2.48	2.54
Upper Transition	2.57	2.54	2.57
Middle Transition	2.61	2.61	2.61
Lower Transition	2.65	2.65	2.67
Sulphide	2.7	2.7	2.7

Source: SIM Geological 2016

Near-surface blocks, in which greater than 50% of the block volume is in overburden, have been assigned a density of 1.90 t/m^3.

14.7 Evaluation of Outlier Gold Grades

Histograms and probability plots were generated to show the distribution of gold in each individual mineralized zone. These were used to identify the existence of potentially anomalous high-grade samples in the composite database. The physical locations of these potential outlier samples were reviewed in relation to the surrounding data. It was decided that, in most cases, the effects of these high-grade samples would be controlled through a combination of traditional top-cutting and the use of outlier limitations during block-grade interpolation. An outlier limitation limits samples above a defined threshold to a maximum distance of 30 m of influence during grade estimation. The various thresholds and the resulting effects on the model areas are shown in Table 14.5.

The reduction in contained gold in all areas, by the application of this methodology, is considered reasonable. The relatively high reduction in contained gold at Double Double is due to the smaller size of the deposit and the presence of several very high-grade composite samples.

Table 14.5: Summary of Capping Levels and Outlier Limitations Applied

Domain	Maximum (Au g/t)[(1)]	Top-cut Limit (Au g/t)	Outlier Limitation (Au g/t)[(2)]	% Metal Lost [(3)]
Supremo				
T1 - Zone1	36.2	20	15	T1 -6%
T1 - Zone2	21.9	-	10	
T2 - Zone1	57.7	35	20	T2 -8%
T2 - Zone2	31.3	-	15	
T2 - Zone3	39.8	15	7	
T3 – Zone1	84.8	70	50	T3 -3%
T3 – Zone2	33.4	20	15	
T3 – Zone3	27.6	10	6	
T4 – Zone1	20.4	-	15	T4 -6%
T4 – Zone2	33.8	-	20	
T4 – Zone3	17.25	-	10	
T5 - Zone1	50.3	30	20	
T5 – Zone2	18	-	10	T5 -3%
T5 – Zone3	31.59	-	15	
T5 – Zone4	11.44	-	6	
T7 - Zone1	17.55	-	10	T7 -4%
T7 – Zone2	7.5	-	5	
T7 – Zone3	12.05	-	5	
T7 – Zone4	14.3	-	10	
T7 – Zone5	3.72	-	2	
T7 – Zone6	7.6	-	5	
T7 - Zone7	10.75	-	6	
All Supremo domains combined				**-5%**
Latte				
Main - Zone1	13.65	-	4	-2%
Main – Zone2	48.7	35	25	
East – Zone1	30.2	-	6	-15%
East – Zone2	28.9	-	15	
East - Zone3	32.5	-	20	
North - Zone1	2.42	-	-	-15%
North - Zone2	1.84	-	-	
North - Zone3	65.5	35	15	
North - Zone4	20.7	-	15	
North - Zone5	2.55	-	-	
Sumatra - Zone1	30	-	13	-6%
Sumatra – Zone2	6.08	-	-	
Latte Combined (Incl. Sumatra)				**-5%**
Double Double				
Zone1	13.6	-	7	-15%
Zone2	26.4	-	15	
Zone3	47.5	30	15	
Zone4	121	75	50	
Zone5	10.35	-	8	
Zone6	66.78	50	15	
Kona				
Zone1	10.9	-	7	-4%
Zone2	36.5	20	12	
Zone3	3.54	-	2	

Domain	Maximum (Au g/t)[(1)]	Top-cut Limit (Au g/t)	Outlier Limitation (Au g/t)[(2)]	% Metal Lost [(3)]
Kona North				
Zone1-3	33.58	-	6	-10%

([1]) 1 m composites
([2]) Influence of composites above threshold limited to maximum 30 m during grade interpolation
([3]) Loss in contained gold metal in resource model limited to blocks in Indicated and Inferred categories

Source: SIM Geological 2016

14.8 Variography

The degree of spatial variability and continuity in a mineral deposit depend on both the distance and direction between points of comparison. Typically, the variability between samples is proportionate to the distance between samples. If the variability is related to the direction of comparison, then the deposit is said to exhibit anisotropic tendencies which can be summarized by an ellipse fitted to the ranges in the different directions. The semi-variogram is a common function used to measure the spatial variability within a deposit.

The components of the variogram include the nugget, the sill, and the range. Often samples compared over very short distances, including samples from the same location, show some degree of variability. As a result, the curve of the variogram often begins at a point on the y-axis above the origin; this point is called the nugget. The nugget is a measure of not only the natural variability of the data over very short distances, but also a measure of the variability which can be introduced due to errors during sample collection, preparation, and assaying.

Typically, the variability between samples increases as the distance between the samples increase. Eventually, the degree of variability between samples reaches a constant or maximum value; this is called the sill, and the distance between samples at which this occurs is called the range.

The spatial evaluation of the data was conducted using a correlogram instead of the traditional variogram. The correlogram is normalized to the variance of the data and is less sensitive to outlier values; this generally gives cleaner results.

Correlograms were generated for the distribution of gold in the various areas using the commercial software package Sage 2001© developed by Isaaks & Co. Due to a lack of available information in some areas, sample data from each mineral domain are combined to generate correlograms. Multidirectional correlograms were generated from composited drill hole samples and the results are summarized in Table 14.6. Due to a lack of data, variograms were not produced for Kona North.

Correlograms were generated using relative distances from the trend planes rather than the true sample elevations. This approach essentially flattens out each structural zone during interpolation relative to the defined trend plane. A variety of correlograms were generated, including raw 1 m composites, capped distributions, indicator variograms and correlograms produced using 3 m composite samples. Properties of these various models contributed to the final parameters believed to appropriately represent the spatial distribution of gold in the deposit areas.

Table 14.6: Gold Variogram Parameters

				1st Structure			2nd Structure		
Area/Domain	**Nugget**	**S1**	**S2**	**Range (m)**	**AZ**	**Dip**	**Range (m)**	**AZ**	**Dip**
Supremo T1	0.135	0.598	0.267	39	180	85	184	0	0
			Spherical	8	90	0	10	90	0
				5	0	5	6	0	90
Supremo T2	0.154	0.751	0.095	31	0	-9	1152	180	66
	Spherical			10	90	0	141	0	24
				6	0	81	12	90	0
Supremo T3	0.245	0.583	0.173	51	0	-48	425	0	49
	Spherical			13	90	0	208	0	-41
				6	0	42	12	90	0
Supremo T4	0.229	0.679	0.091	19	0	-74	64	0	-8
	Spherical			10	90	0	32	0	82
				5	0	16	10	90	0
Supremo T5	0.138	0.710	0.152	19	0	-21	216	0	-8
	Spherical			11	90	0	87	0	82
				7	0	69	12	90	0
Supremo T7	0.163	0.663	0.173	16	180	12	237	0	-60
	Spherical			8	90	0	47	0	30
				6	0	78	12	90	0
Latte Main	0.200	0.650	0.150	11	0	0	55	90	37
	Spherical			5	270	-45	54	0	0
				5	270	45	48	90	-53
Latte East	0.250	0.400	0.350	11	0	0	38	90	-6
			Spherical	8	270	46	26	90	84
				4	270	-44	7	0	0
Latte North	0.250	0.600	0.150	24	90	-47	194	90	-22
			Spherical	7	0	0	47	270	-68
				2	90	43	32	0	0
Sumatra	0.250	0.600	0.150	24	90	7	67	90	-29
			Spherical	5	0	0	30	90	61
				4	270	83	5	0	0
Double Double	0.350	0.606	0.044	10	90	43	531	90	-32
	Spherical			10	0	0	100	270	-58

				1st Structure			2nd Structure		
Area/Domain	Nugget	S1	S2	Range (m)	AZ	Dip	Range (m)	AZ	Dip
				6	270	47	10	0	0
Kona	0.300	0.580	0.120	11	90	82	143	90	-21
	Spherical			6	0	0	69	90	69
				4	90	-8	6	0	0

Note: Correlograms modelled using sample data composited to 1 m intervals.
Source: SIM Geological 2016

14.9 Model Setup and Limits

Five block models were initialized in MineSight®. The dimensions are shown in Table 14.7. None of the models are rotated.

The block size used in the models measures 10 m along strike, 2.5 m across strike and 5 m vertical. The determination of the block size considered the current drill hole spacing and the selective mining unit (SMU) size that is considered appropriate for deposits of this type and scale. These deposits are generally narrow and relatively high-grade and, although a block dimension of 2.5 m may be considered small from mining selectivity and productivity aspects, it is applied in order to retain the high-grade nature of the deposit in the block model. The author believes that this is the scale of selectivity that is required to retain the relatively high-grade nature of the deposit during mining.

Table 14.7: Block Model Limits

Direction	Minimum[(1)] (m)	Maximum[(1)] (m)	Block Size (m)	Number of Blocks
Supremo				
East	583,550	585,200	2.5	660
North	6,973,100	6,975,800	10	270
Elevation	650	1320	5	134
Latte				
East	581,700	584,300	10	260
North	6,972,800	6,974,500	2.5	680
Elevation	500	1,300	5	160
Double Double				
East	584,550	585,500	10	95
North	6,973,050	6,973,550	2.5	200
Elevation	600	1,150	5	110
Kona				
East	579,450	580,050	10	60
North	6,972,850	6,973,300	2.5	180
Elevation	950	1,320	5	74
Kona North				
East	579,200	580,000	10	80
North	6,973,650	6,974,250	2.5	240
Elevation	800	1,200	5	80

[(1)] UTM coordinates (Nad83 datum, zone 7), elevation relative to mean sea level

Source: SIM Geological 2015

Using the mineral domain wireframes, blocks in the model are assigned “area” codes and, using the mineralized zone domains, blocks are assigned individual “zone” codes. Blocks that straddle two or more domains are assigned codes on a majority basis.

The proportion of blocks within the mineral domains is also calculated and stored in the model as a percentage. These percentages are used as a weighting factor to determine the in-situ volume and tonnage estimates.

Blocks are also assigned oxide codes on a majority basis.

14.10 Interpolation Parameters

The block model grades for gold were estimated using ordinary kriging in all models, except at Kona North where inverse distance squared (ID^2) estimates were made. The kriging models were validated using the Hermitian Polynomial Change of Support model (Journel and Huijbregts, 1978), also known as the Discrete Gaussian Correction. The ordinary kriging models were generated with a relatively limited number of composites to match the change of support or Herco (Hermitian correction) grade distribution. This approach reduces the amount of smoothing (also known as averaging) in the model. While there may be some uncertainty on a localized scale, this approach produces reliable estimates of the potentially recoverable gold and tonnage for the deposit. The interpolation parameters are summarized by domain in Table 14.8.

Table 14.8: Interpolation Parameters

Area/ Domain	Search Ellipse Range (m) [1]			Number of Composites			Other
	X	Y	Z	Minimum	Maximum	Maximum Per Hole	
Supremo T1 Zone1	2	200	200	1	9	3	
T1 Zone2	3	200	200	1	12	4	
Supremo T2 Zone1	2	200	200	1	9	3	1 hole per quadrant
T2 Zone2	2	200	200	1	9	3	1 hole per quadrant
T2 Zone3	2	200	200	1	9	3	1 hole per quadrant
Supremo T3	3	200	200	1	16	4	
T3 Zone2	3	200	200	1	20	5	
T3 Zone3	3	200	200	1	20	5	
Supremo T4 Zone1	3	200	200	1	12	4	1 hole per quadrant
T4 Zone2	3	200	200	1	12	4	1 hole per quadrant
T4 Zone3	3	200	200	1	12	4	1 hole per quadrant
Supremo T5 Zone1	2	200	200	1	9	3	1 hole per quadrant
T5 Zone2	3	200	200	1	12	4	1 hole per quadrant
T5 Zone3	3	200	200	1	12	4	1 hole per quadrant
T5 Zone4	3	200	200	1	12	4	1 hole per quadrant
Supremo T7 all Zones	3	200	200	1	12	4	1 hole per quadrant
Latte Main Zone1	150	3	150	3	15	5	1 hole per octant
Main Zone2	150	4	150	3	18	6	1 hole per octant
Latte East Zone1	150	2	150	3	12	4	1 hole per octant
East Zone2	150	2	150	3	12	4	1 hole per octant
East Zone3	150	2	150	3	12	4	1 hole per octant
Latte North Zone1	150	4	150	3	18	6	
North Zone2	150	4	150	3	18	6	
North Zone3	150	4	150	3	18	6	
North Zone4	150	4	150	3	18	6	
North Zone5	150	4	150	3	18	6	

Area/ Domain	Search Ellipse Range (m) (1)			Number of Composites			Other
	X	Y	Z	Minimum	Maximum	Maximum Per Hole	
Sumatra Zone1	150	3	150	3	18	6	
Sumatra Zone2	150	3	150	3	18	6	
Latte	150	3	150	1	15	5	1 hole per quadrant
Double Double All Zones	150	2	150	1	12	4	
Kona All Zones	150	3	150	1	9	3	
Kona North All Zones	200	3	200	3	15	5	ID2 estimate

(1) The longer ranges are oriented parallel to the mineralization trend planes. The shortest range is perpendicular to the plane of mineralization.

Source: SIM Geological 2015

During grade estimation, search orientations were designed to follow a mineralization trend surface interpreted to represent the general trend of the mineralization in each of the structural zone domains (as described in Section 1.3.1). The distance from this trend plane is assigned to all composited drill hole samples and model blocks and is used to replicate the undulating and banded nature of the deposit.

14.11 Block Model Validation

The block models were validated using several methods: a thorough visual review of the model grades in relation to the underlying drill hole sample grades, comparisons with the change of support model, comparisons with other estimation methods, and grade distribution comparisons using swath plots.

14.11.1.1.1 Visual Inspection

A detailed visual inspection of the block models was conducted in both section and plan to compare estimated grades with the underlying sample data. This includes confirmation of the proper coding of blocks within the respective domains. The distribution of block grades was compared relative to the drill hole samples to ensure the proper representation in the model.

14.11.1.1.2 Model Checks for Change of Support

The relative degree of smoothing in the block estimates was evaluated using the Hermitian Polynomial Change of Support model. Using this method, the distribution of the hypothetical block grades can be directly compared to the estimated ordinary kriging model through the use of pseudo-grade/tonnage curves. Adjustments are made to the block model interpolation parameters until an acceptable match is made with the Herco distribution.

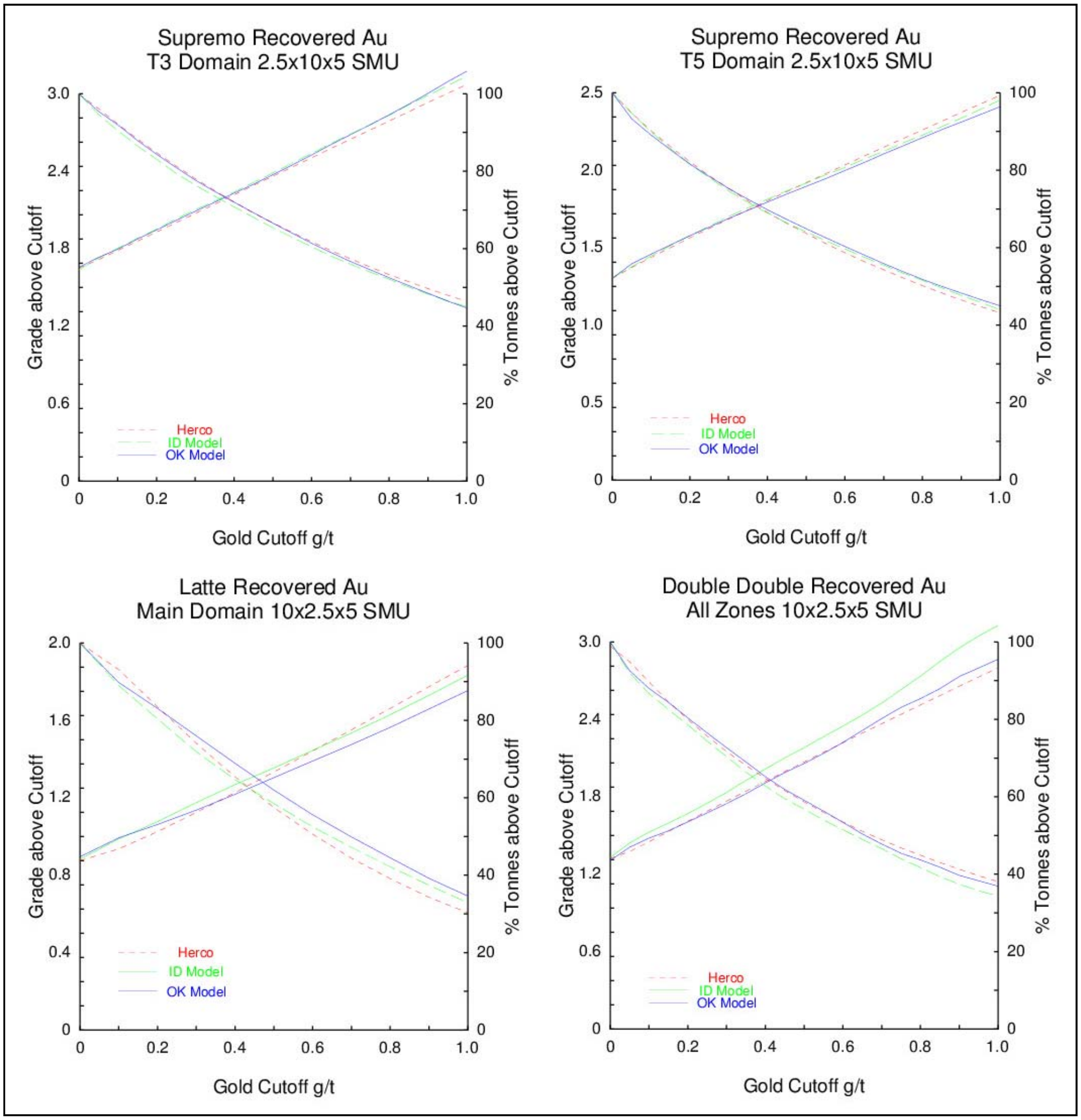
In general, the estimated model should be slightly higher in tonnage and slightly lower in grade when compared to the Herco distribution at the projected cut-off grade. These differences account for selectivity and other potential ore handling issues which commonly occur during mining.

The Herco distribution is derived from the declustered composite grades which have been adjusted to account for the change in support moving from smaller drill hole composite samples to the larger blocks in the model. The transformation results in a less skewed distribution, but with the same mean as the original declustered samples. Examples of Herco plots from some of the models are shown in Figure 14.15.

Overall, the desired degree of correspondence between models has been demonstrated. The results indicate that the gold models are realistic representations of the gold grade distributions for the defined scale of selectivity (i.e., minimum mining unit size).

It should be noted that the change of support model is a theoretical tool intended to direct model estimation. There is uncertainty associated with the change of support model, and its results should not be viewed as an absolutely correct value. In cases where the model grades are greater than the change of support grades, the model is relatively insensitive to any changes to the modelling parameters. Any extraordinary measures to make the grade curves change are not warranted.

Figure 14.15: Examples of Herco Plots



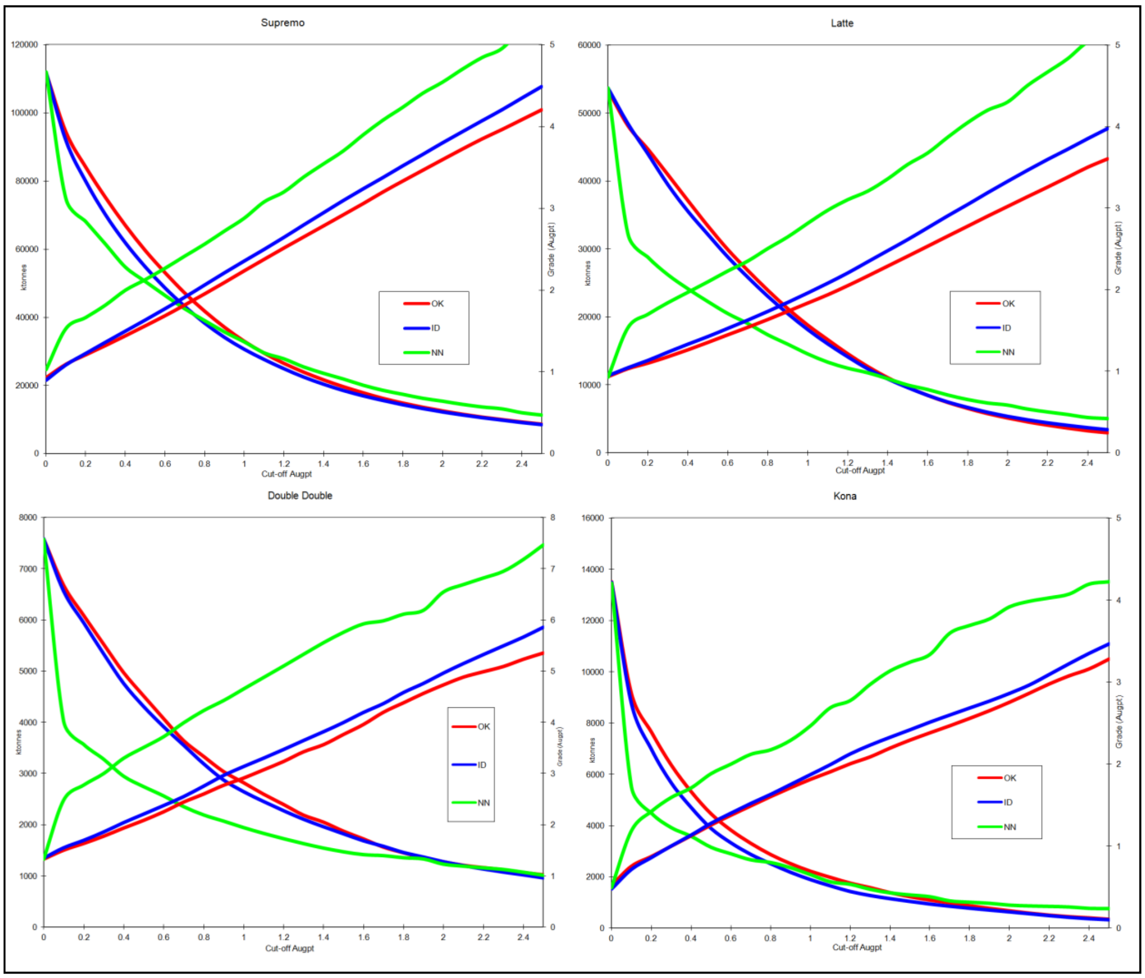
Source: SIM Geological 2015

14.11.1.1.3 Comparison of Interpolation Methods

For comparison purposes, additional grade models were generated using the inverse distance weighted (ID2) and nearest neighbour (NN) interpolation methods. The nearest neighbour model was created using data composited to lengths equal to the short block axis. The results of these models are compared to the ordinary kriging (OK) models at various cut-off grades in a series of grade/tonnage graphs shown in Figure 14.16.

There is good correspondence between the OK and ID^2 models in all areas. The reduced grades in the OK and ID2 models compared to the NN model result from the smoothing that takes place during grade interpolation. These differences are not unique to Coffee but are typical of gold deposits that exhibit a relatively high degree of grade variability.

Figure 14.16: Comparison of Ordinary Kriging (OK), Inverse Distance (ID2) and NN Models



Source: SIM Geological 2015

14.11.1.1.4 Swath Plots (Drift Analysis)

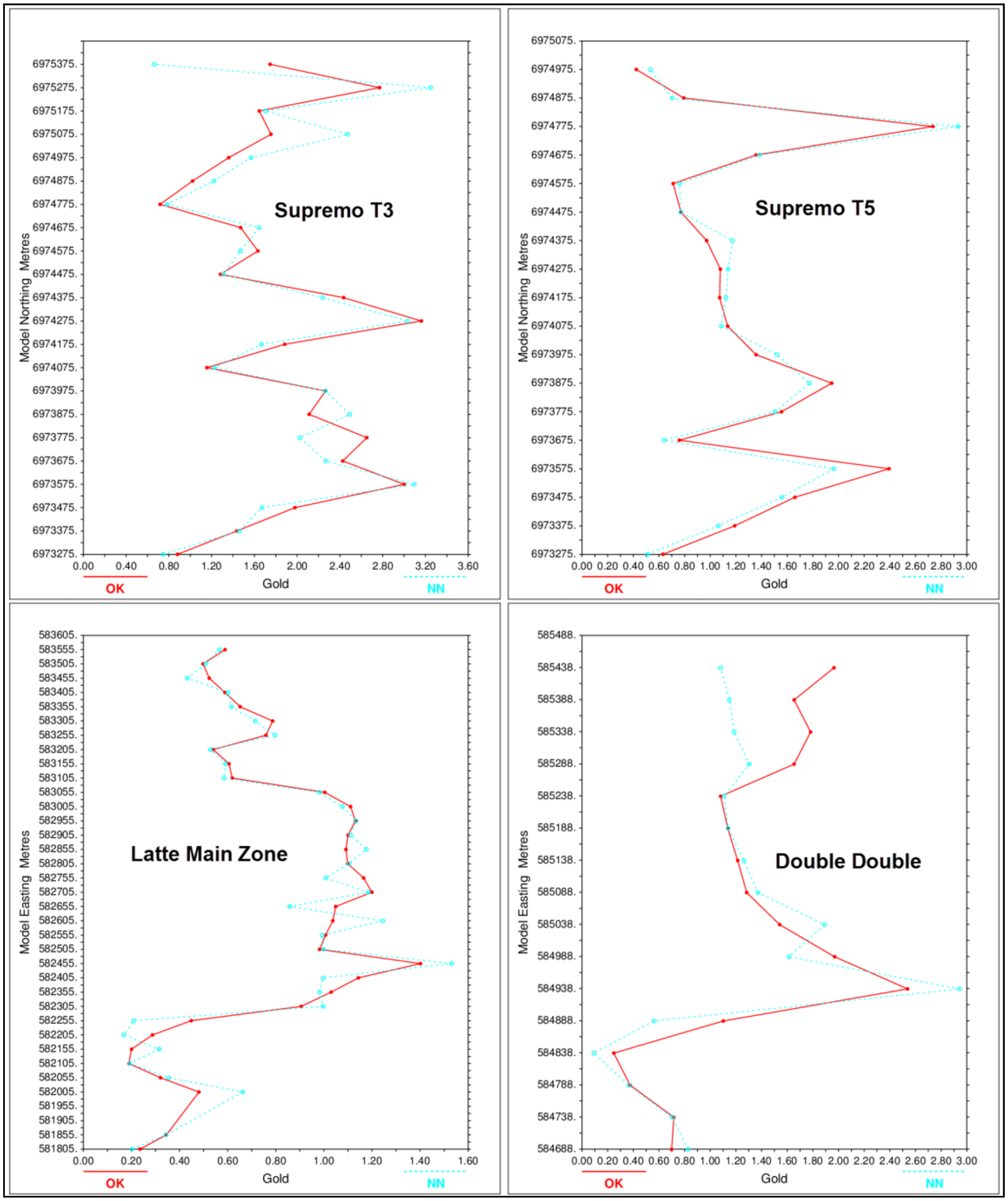
A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions throughout the deposit. Using the swath plot, grade variations from the ordinary kriging model are compared to the distribution derived from the declustered nearest neighbour grade model.

On a local scale, the nearest neighbour model does not provide reliable estimations of grade, but, on a much larger scale, it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the ordinary kriging model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the nearest neighbour distribution of grade.

Swath plots were generated in three orthogonal directions that compare the ordinary kriging and nearest neighbour gold estimates. Several examples of swath plots oriented along the general trend of each deposit area are shown in Figure 14.17.

There is good correspondence between the models in all areas. The degree of smoothing in the ordinary kriging model is evident in the peaks and valleys shown in the swath plots.

Figure 14.17: Examples of Swath Plots



Source: SIM Geological 2015

14.12 Resource Classification

The mineral resources were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). The classification parameters are defined relative to the distance between sample data and are intended to encompass zones of reasonably continuous mineralization that exhibit the desired degree of confidence. These parameters are based on visual observations and statistical studies.

Resources are included in the Indicated category if the projected volume, equivalent to one year of production, is within +/-15% of projected tonnage and grade from the model, 90% of the time. The reliability, or uncertainty, of the resource estimates can be measured using the spatial variation exhibited in the variogram. This work suggests that these levels of reliability can be achieved when drilling is spaced on a maximum, nominal 40 m grid pattern. These reliability predictions have been supported by the results of conditional simulation studies and analysis of resources generated in areas of significant close-spaced drilling.

Kaminak began drilling the majority of the deposit with holes spaced at 25 m intervals on fence lines spaced 50 m apart. The nominal 40 m drill grid pattern described above has been modified to integrate with the current distribution of drill holes. Infill drilling comprises 50 m spaced drill holes on the 25 m sections located between the initial 50 m drill sections. This results in an alternating sequence of 25 m and 50 m spaced drill holes on 25 m sections.

Resources are included in the Measured category if the projected volume, equivalent to one quarter of a year of production, is within +/-15% of projected tonnage and grade, 90% of the time. At this stage, the author can only project the drilling pattern to achieve the relatively high level of confidence in Measured class resources using the current drilling which has holes spaced at 25 m intervals or greater. Based on these projections, resources in the Measured category require drilling on a nominal grid pattern of holes spaced at 10m to 15 m intervals. It is felt that the generally thicker and more consistent mineralized zones in the centre of the Latte deposit will require a wider spaced grid of holes where the more typical, somewhat narrow, mineralized zones will require closer-spaced drilling to delineate resources in the Measured category.

To further investigate the continuity of mineralization, indicator variograms were produced for samples inside the mineral domains. The indicator variograms are based on an indicator variable designed to generally delineate the presence of mineralization and roughly based on mineralized intercepts of at least 5 m greater than 0.4 g/t Au. The variogram ranges show that most of the continuity in gold grades above the defined threshold deteriorates somewhere between 10 m and 40 m, but the ultimate ranges extend well beyond 100 m. The uncertainty calculation results, described above, are consistent with the indicator variogram results and these observations provide additional support for the resource classification criteria.

The mineral resource categories are defined as follows:

14.12.1.1 Measured Resources

At this stage, only drilling patterns to achieve the relatively high level of confidence in Measured class resources using the current drilling (which has holes spaced at 25 m intervals or greater) may be projected. Based on these projections, resources in the Measured category require drill holes on a nominal 10 m to 15 m grid pattern.

14.12.1.2 Indicated Resources

Indicated Resources are delineated from multiple drill holes located on a nominal 25 m spacing between sections, and comprising alternating sections that have 50 m spaced holes and 25 m spaced holes respectively. In general, this is equivalent to estimating block grades using three or more drill holes within a maximum average distance of 25 to 30 m.

14.12.1.3 Inferred Resources

Inferred Resources include any material that does not fall into the Indicated category, but is within a maximum distance of 50 m from a drill hole. This means that holes may be drilled on a nominal 100 m pattern to delineate resources in the Inferred category.

The spacing distances are intended to define contiguous volumes and allow for some irregularities due to actual drill hole placement. Some manual smoothing of these criteria is conducted that includes areas where the drill hole spacing locally exceeds the desired grid spacing but still retains continuity of mineralization or, conversely, excludes areas where the mineralization does not exhibit the required degree of continuity. This process results in a series of three-dimensional domains that are used to assign resource classification codes into model blocks.

14.13 Mineral Resources

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) provides the following definition:

A mineral resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

The "reasonable prospects for eventual economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade which takes the extraction scenarios and processing recovery into account.

The Coffee Gold deposits form relatively continuous, sub-vertical zones of gold mineralization extending from the surface to depths (locally) of more than 200 m. The "reasonable prospects for eventual economic extraction" were tested using a series of floating cone pit shells based on projected technical and economic assumptions. Before running pit shells, the estimated in-situ block grades were expanded to a full block basis, meaning that partial blocks that straddle a mineral domain boundary are recalculated to include the portion of the block outside the domain (included at zero grade). The resulting block grades are then adjusted for metallurgical recoveries based on the oxide type designation.

The following technical and economic parameters were assumed:

- Metallurgical recoveries: Oxide 90%, Upper Transition 80%, Middle Transition 60%, Lower Transition 45%, and Sulphide 10%;
- Mining cost: $2/t;
- Process cost: $6.65/t;
- General and administration: $3.80/t;
- Pit slope: 45 degrees;
- Gold price: A series of shells produced at prices from US$1,000 to US$1,700/oz gold; and
- No adjustments were made for mining recovery or dilution.

These pit optimization evaluations are used solely for the purpose of testing the “reasonable prospects for eventual economic extraction,” and do not represent an attempt to estimate mineral reserves. The optimization results are used to help prepare a mineral resource statement and to select appropriate reporting assumptions.

Pit shells generated at a gold price of US$1,400/oz often extend to depths of 200 m and, locally, beyond 250 m below surface. Approximately 65% of the Indicated and Inferred Resources occur within a pit shell using a gold price of US$1,400/oz and this proportion increases to 75% at a gold price of US$1,700/oz. When restricted to oxide types amenable to cyanide leaching, Oxide plus Upper Transition and Middle Transition, more than 70% of the Indicated plus Inferred resources are inside a US$1,400/oz Au shell, and more than 80% are inside a US$1,700/oz Au shell. These results show that, over the projected lifespan of this deposit, it is reasonable to assume that economic conditions could occur such that the majority of the total resource has a reasonable chance of economic viability.

Further studies segregating resources contained within 50 m increments below surface show that 72% of the Indicated Resource is within 100 m of surface, and 91% is within 150 m of surface. Approximately 70% of the Inferred Resource is located within 200 m of surface. These results show that essentially all of the Indicated Resource and the majority of the Inferred Resource occur at relatively shallow depths and, as a result, are potentially amenable to open pit extraction methods.

The studies described here show that parts of the resource may never be economically viable, but these include primarily Inferred Resources that tend to comprise the deeper Lower Transition and Sulphide oxide types and represent only a minor proportion of the total resource. Rather than sterilizing these relatively small parts of the deposit at this stage, it is believed that it is better to retain them for engineering-level analysis to determine economic viability.

Gold mineralization delineated by drilling at Coffee is primarily located within a maximum distance of 200 m below surface. Condemning portions of the deposit from the mineral resource using assumed technical and economic factors affects only a minor proportion of the resource at this stage of project evaluation. Future detailed engineering studies are required to demonstrate the true economic viability of the resource.

As a result, the mineral resources for the Coffee Gold Project are not constrained within a pit shell, or a maximum depth below surface, because, in the author's opinion, any or all of the mineralization at Coffee shows reasonable prospects for eventual economic extraction.

The Mineral Resource Statement for the Coffee Gold Project is shown in Table 14.9. Mineral resources are stated at cut-off thresholds that reflect the projected metallurgical characteristics of the various degrees of oxidation; 0.3 g/t Au for Oxide and Upper Transition, 0.4g/t Au for Middle Transition and at 1 g/t Au for Lower Transition and Sulphide types. The distribution of base case resources in the deposit areas is shown in Figure 14.18 and Figure 14.19.

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect the mineral resource.

Table 14.9: Estimate of Mineral Resources for the Coffee Gold Project

Area	Oxide			Upper Transition			Middle Transition			Oxide Upper & Middle Transitions				Lower Transition		Sulphide		
	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)
INDICATED																		
Supremo	37,032	1.45	1,726	3,999	1.48	190	1,147	1.67	62	42,178	1.46	1,978	431	2.26	31	24	1.85	1
Latte	8,021	1.31	338	3,913	1.14	143	2,937	1.27	120	14,871	1.26	601	1,675	1.86	100	105	2.16	7
Double. Double.	776	2.33	58	797	2.84	73	402	2.32	30	1,976	2.53	161	105	2.34	8	3	2.57	0
Kona	1,298	1.14	48	751	0.93	22	161	0.93	5	2,210	1.05	75	83	1.50	4	6	1.60	0
COMBINED	47,127	1.43	2,170	9,461	1.41	429	4,647	1.45	217	61,235	1.43	2,815	2,294	1.94	143	138	2.09	9
INFERRED																		
Supremo	20,775	1.09	726	4,642	1.28	191	2,506	1.80	145	27,922	1.18	1,061	1,352	2.19	95	103	2.03	7
Latte	3,435	0.91	101	4,020	1.00	129	2,951	1.24	118	10,407	1.04	348	3,964	1.83	233	2,456	1.73	137
Double. Double.	529	1.13	19	1,200	1.36	53	766	2.01	49	2,496	1.51	121	138	1.68	7	107	1.55	5
Kona	146	0.88	4	373	1.02	12	337	1.13	12	856	1.04	29	940	1.74	53	55	1.96	3
Kona North	135	1.51	7	411	1.80	24	295	1.66	16	841	1.70	46	409	2.74	36	310	3.02	30
COMBINED	25,020	1.06	857	10,646	1.19	408	6,855	1.54	340	42,520	1.17	1,605	6,803	1.94	424	3,030	1.87	182

Note: Oxide and Upper Transition 0.3 g/t Au cut-off, Middle Transition 0.4 g/t Au cut-off, Lower Transition and Sulphide 1.0 g/t Au cut-off.

Source: SIM Geological 2015

Figure 14.18: Isometric View of the Distribution of Base Case Resources

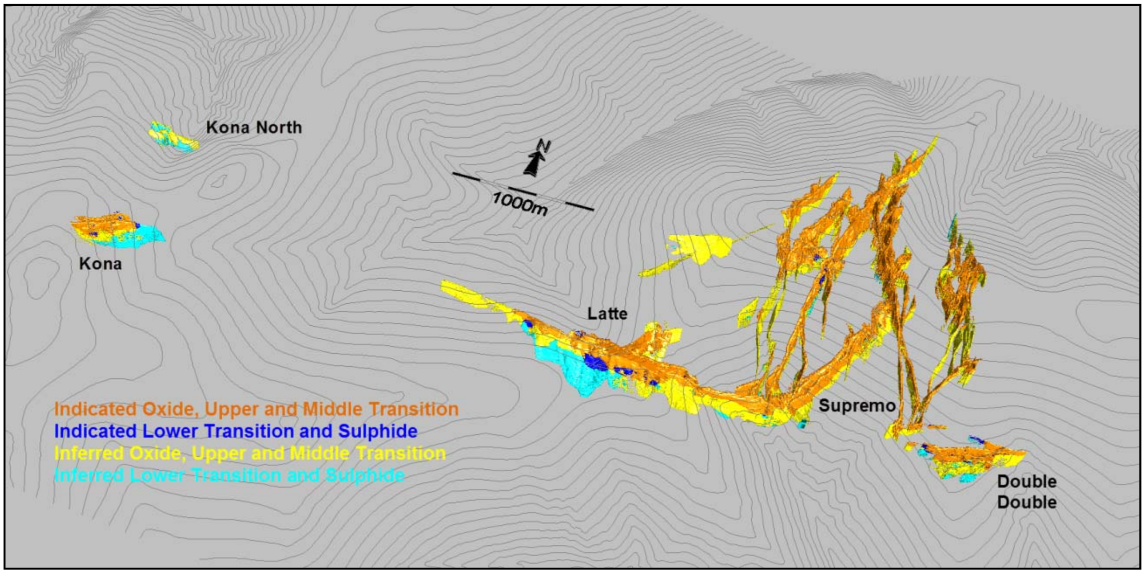
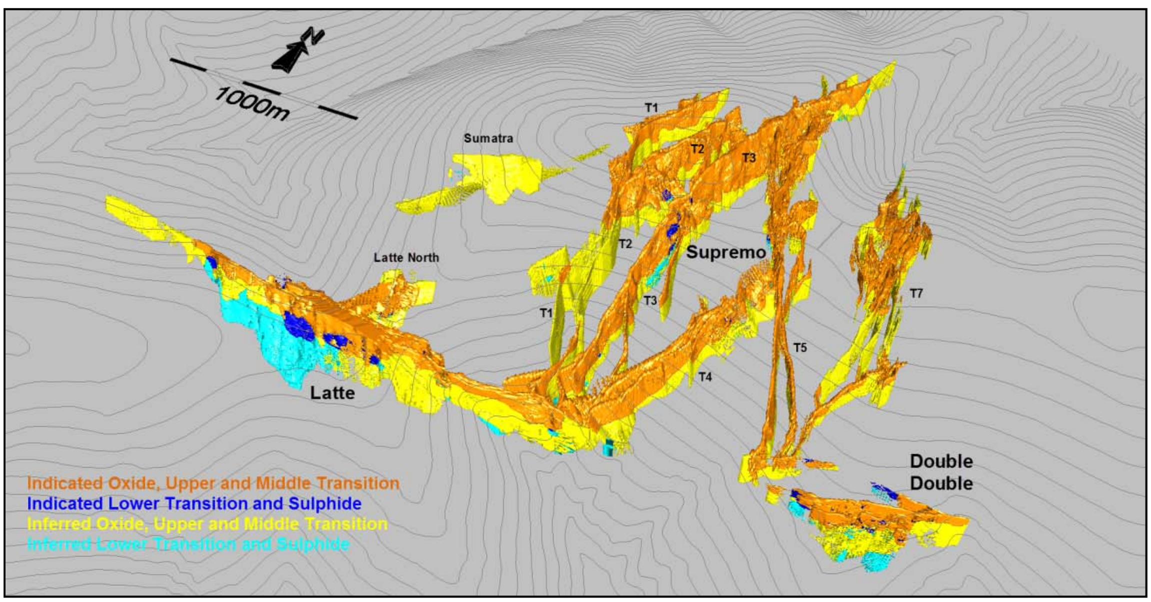


Figure 14.19: Isometric View of the Distribution of Base Case Resources at Supremo, Latte, and Double Double



14.14 Sensitivity of Mineral Resources

The sensitivity of mineral resources to cut-off grade is demonstrated by listing resources at a series of cut-off thresholds as shown in Table 14.10, Table 14.11, and Table 14.12. Higher relative gold grades at Double Double and Supremo result in a higher proportion of remaining resources at the higher cut-off thresholds.

Table 14.10: Estimate of Mineral Resources at 1.0 g/t Gold Cut-off

Area	Oxide			Upper Transition			Middle Transition			Oxide Upper & Middle Transitions				Lower Transition			Sulphide	
	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)
INDICATED																		
Supremo	17,445	2.40	1,347	1,999	2.35	151	654	2.431	51	20,097	2.40	1,549	431	2.30	31	24	1.85	1
Latte	4,171	1.95	259	1,824	1.70	102	1,601	1.768	91	7,596	1.85	452	1,675	1.85	100	105	2.15	7
Double. Double.	446	3.60	52	498	4.20	67	251	3.31	27	1,194	3.80	145	105	2.30	8	3	2.60	0
Kona	507	2.00	33	213	1.90	13	52	1.514	3	772	2.00	49	83	1.50	4	6	1.60	0
COMBINED	22,567	2.35	1,691	4,534	2.30	333	2,558	2.084	171	29,659	2.30	2,195	2,294	1.95	143	138	2.10	9
INFERRED																		
Supremo	7,378	2.00	472	2,006	2.15	139	1,621	2.418	126	11,005	2.10	737	1,352	2.20	95	103	2.00	7
Latte	1,123	1.60	58	1,327	1.80	78	1,292	2.003	83	3,742	1.80	219	3,964	1.80	233	2,456	1.70	137
Double. Double.	210	1.90	13	581	2.20	41	479	2.822	43	1,270	2.40	97	138	1.70	7	107	1.55	5
Kona	48	1.75	3	149	1.77	8	157	1.7	9	354	1.70	20	940	1.75	53	55	2.00	3
Kona North	59	2.60	5	204	3.00	20	171	2.433	13	434	2.70	38	409	2.75	36	310	3.00	30
COMBINED	**8,818**	**1.95**	**551**	**4,267**	**2.10**	**286**	**3,721**	**2.297**	**275**	**16,806**	**2.05**	**1,111**	**6,803**	**1.94**	**424**	**3,030**	**1.90**	**182**

Note: 1 g/t cut-off for all oxide types

Source: SIM Geological 2015

Table 14.11: Estimate of Mineral Resources at 1.5 g/t Gold Cut-off

Area	Oxide			Upper Transition			Middle Transition			Oxide Upper & Middle Transitions			Lower Transition			Sulphide		
	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)
INDICATED																		
Supremo	10,716	3.15	1,082	1,236	3.05	121	423	3.10	42	12,374	3.15	1,245	242	3.10	24	11	2.55	1
Latte	2,354	2.48	187	860	2.30	63	796	2.30	59	4,010	2.40	310	898	2.40	69	58	2.95	5
Double. Double.	343	4.30	47	371	5.20	62	159	4.50	23	873	4.70	133	61	3.20	6	2	3.00	0
Kona	301	2.60	25	122	2.40	9	21	1.90	1	445	2.50	36	29	2.00	2	3	2.00	0
COMBINED	13,714	3.00	1,342	2,589	3.10	256	1,399	2.80	125	17,702	3.00	1,724	1,230	2.60	101	74	2.85	7
INFERRED																		
Supremo	3,872	2.70	335	1,102	2.90	104	1,116	2.95	106	6,090	2.80	545	851	2.80	75	83	2.20	6
Latte	524	2.05	35	665	2.45	52	686	2.70	59	1,874	2.40	146	2,031	2.40	157	1,152	2.30	86
Double. Double.	114	2.50	9	362	2.75	32	334	3.50	38	811	3.00	79	70	2.10	5	52	1.95	3
Kona	27	2.10	2	84	2.15	6	90	2.05	6	201	2.10	14	498	2.20	35	42	2.20	3
Kona North	41	3.25	4	151	3.60	17	120	2.95	11	312	3.30	33	299	3.30	32	211	3.85	26
COMBINED	**4,577**	**2.60**	**385**	**2,365**	**2.80**	**211**	**2,346**	**2.90**	**220**	**9,288**	**2.75**	**816**	**3,749**	**2.50**	**304**	**1,539**	**2.50**	**124**

Source: SIM Geological 2015

Table 14.12: Estimate of Mineral Resources at 2.0 g/t Gold Cut-off

Area	Oxide			Upper Transition			Middle Transition			Oxide Upper & Middle Transitions			Lower Transition			Sulphide		
	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)	Quantity (ktonnes)	Grade Au (g/t)	Metal Au (koz)
INDICATED																		
Supremo	7,082	3.90	881	808	3.75	97	277	3.80	34	8,167	3.85	1,012	161	3.80	19	7	2.95	1
Latte	1,380	3.0	133	421	2.90	39	390	2.90	37	2,192	3.00	209	462	3.05	45	42	3.40	5
Double. Double.	272	5.00	43	280	6.30	57	122	5.40	21	673	5.60	121	44	3.75	5	2	3.40	0
Kona	186	3.20	19	69	2.90	6	7	2.40	1	262	3.10	26	10	2.50	1	1	2.35	0
COMBINED	8,920	3.75	1,077	1,578	3.95	200	795	3.60	92	11,294	3.80	1,369	676	3.70	71	53	3.30	6
INFERRED																		
Supremo	2,214	3.40	244	666	3.70	80	773	3.50	87	3,652	3.50	410	562	3.30	59	41	2.75	4
Latte	211	2.60	17	375	3.00	36	420	3.30	44	1,006	3.00	98	972	3.15	98	623	2.80	57
Double. Double.	64	3.10	6	205	3.60	23	245	4.20	33	513	3.80	63	23	3.0	2	15	2.70	1
Kona	9	3.00	1	41	2.60	3	39	2.50	3	89	2.60	7	284	2.50	23	21	2.70	2
Kona North	30	3.80	4	104	4.40	15	76	3.60	9	210	4.00	27	211	3.95	27	161	4.50	23
COMBINED	**2,528**	**3.35**	**272**	**1,390**	**3.50**	**158**	**1,552**	**3.55**	**176**	**5,470**	**3.45**	**606**	**2,052**	**3.20**	**210**	**860**	**3.15**	**87**

Source: SIM Geological 2015

14.15 Comparison with the Previous Estimate of Mineral Resources

The previous resource estimate was generated in January 2014 (Sim & Kappes, 2014). It is difficult to make direct comparisons between the previous and current resource estimates because there have been changes in the definition of resource (oxide) types and the applied base cut-off limits. In January 2014 the various oxide types were similarly defined using AuCN/Au ratios based on the following thresholds; Oxide > 0.90–0.95, Upper Transition 0.50–0.90, Lower Transition 0.10–0.50 and Sulphide < 0.10. Based on recent metallurgical test work the previous Upper Transition has now been subdivided into Upper Transition and Middle Transition types at a ratio of 0.70 AuCN/Au. In January 2014, the base cut-off for Oxide, Upper and Lower Transition was 0.5 g/t Au and for Sulphide it was 1 g/t Au. In the current estimate, the base case cut-off grade for Oxide and Upper Transition resources is 0.3 g/t Au, 0.4 g/t Au for Middle Transition and is 1 g/t Au for Lower Transition and Sulphide resources.

Table 14.13 compares the current base case resources (with the effective dates of the models at Latte, Double Double and Kona of March 15, 2015 and for Supremo of September 22, 2015) with the January 2014 estimate. There is a significant increase in Indicated Resources as a result of the tighter-spaced delineation drilling completed since the previous estimate. Resources have also increased as a result of the reduction in the cut-off thresholds for Oxide, Upper Transition and Middle Transition material.

Table 14.13: Comparison of Base Case Combined Resources March and September 2015 vs. January 2014

Area	December 2015 (Feasibility Study)			January 2014 (PEA)		
	ktonnes	Au g/t	koz Au	ktonnes	Au g/t	koz Au
42,633						
16,650	1.32	709	10,361	1.38	461	258
2,084	2.52	169	0	0	0	461
2,299	1.07	79	0	0	0	0
63,666	1.45	2,968	14,357	1.56	719	0
Inferred						719
29,376						
16,827	1.33	717	17,599	1.51	853	2,156
2,740	1.52	134	4,139	2.32	309	853
1,851	1.42	85	2,706	1.34	116	309
1,559	2.24	112	0	0	0	116
52,354	**1.31**	**2,212**	**78,591**	**1.36**	**3,434**	0
42,633	1.47	2,011	3,997	2.01	258	**3,434**

Note: March and September 2015 resource includes Oxide and Upper Transition at 0.3 g/t Au cut-off, Middle Transition at 0.4 g/t Au cut-off and Lower Transition and Sulphide at 1 g/t Au cut-off. January 2014 includes OX+UT+LT at 0.5 g/t Au cut-off and Sulphide at 1 g/t Au cut-off.

Source: SIM Geological 2015

It is difficult to quantify the overall changes that have taken place because of the transfer of relatively large volumes of resources between categories. Comparison of combined Indicated + Inferred class resources is not allowed under NI 43-101. In general, there has been an increase in resources at Coffee since the previous estimates and these increases are not only the result of reductions in the cut-off thresholds. The largest increases have occurred in the Supremo deposit with the addition of highly oxidized resources. At Latte there is a moderate increase in Oxide, Upper and Middle Transition resources and a minor decrease in Lower Transition and Sulphide resources. There is essentially no change in the resources at Double Double and only a minor increase in resources at Kona.

The changes in resources are due primarily to the following four factors:

Interpretation. Mineral domains are now more grade based interpretations rather than the geology-based interpretations of the "structural" domains used in the previous resource estimates. This results in less internal dilution and higher overall average grades in most areas.

Classification. New drilling was primarily designed to upgrade resources from the Inferred to Indicated category and, as a result, there is little change to the physical extent of resources in the Inferred category at Latte, Double Double, and Kona. New drilling at Supremo, primarily at depth, has added approximately 250 koz of contained gold in the Inferred category.

Oxide type designation. Additional cyanide soluble gold data allow for interpolation of AuCN/Au ratios along the trends of the mineral domains. These ratios are used to define five oxide types representing varying degrees of oxidation. This approach shows that oxide type distributions inside the mineral domains can be quite variable, especially at Double Double and Kona and, to a lesser extent, at Latte. A relative lack of AuCN data available for the January 2014 model resulted in a more generalized interpretation of the various oxide domains. The new (and improved) approach to oxidation has increased the amount of oxidized resources significantly at Supremo and moderately at Latte.

Cut-off grades. The reduced cut-off limits for Oxide, Upper and Middle Transition type material has had a significant impact on the amount of resources at Coffee.

15 Mineral Reserve Estimate

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 January 6, 2016.

Table 15.1: Summary of Mineral Reserves

Area	Reserve Category	Diluted Ore ('000s tonne)	Diluted Gold Grade (Au g/t)	Contained Gold ('000s ounce)
Total Open Pit	Probable	46,356	1.45	2,157

Notes:
A US$:C$ exchange rate of 0.87 is assumed.
Dilution and recovery factors are applied as further described in Section 16, Mining Methods.
Source: JDS 2016

15.1 Open Pit Mineral Reserve

15.1.1 Open Pit Mineral Reserve Basis of Estimate

The mineral reserve estimate for the Coffee property is based on the mineral resource estimate completed by SIM Geological Inc. with the effective dates of the models at Latte, Double Double and Kona of March 15, 2015 and for Supremo of September 22, 2015.

The Mineral Reserves were developed by examining each deposit to determine the optimum and practical mining method. Only Measured and Indicated Mineral Resources were included in the optimization process (no Measured Resources were defined in the models). Inferred resources were considered as waste. COGs were then determined based on appropriate mine design criteria and the adopted mining method. A thorough analysis of the optimized shells was then conducted in order to select the shells to be used as guides to detailed design and a shovel and truck open pit mining method was selected for the various deposits.

15.1.2 Mining Method and Mining Costs

The deposits at the Coffee Gold site are amenable to extraction by open pit methods. For the purposes of preliminary optimization, mining costs of $2.25/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 four deposits at the Coffee Gold site (Latte, Double Double, Kona and Supremo) 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 blocks which were contained within a given zone (in this case a resource classification of Indicated) 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 m for each edge.

The analysis resulted in external dilutions of 5%, 9%, 10% and 7% being applied to the Latte, Double Double, Kona and Supremo deposits, respectively.

15.1.4 Geotechnical Considerations

SRK Consulting (US) Inc. (SRK) carried out field investigations and analyses designed to characterize geotechnical and hydrogeological conditions required for feasibility-level open pit designs. 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, inter-ramp and overall slope angles for each design sector as shown in Table 15.2.

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, along with an ongoing commitment to geomechanical data collection and analyses over the life of mine operation.

15.1.5 Lerchs-Grossman Optimization

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

Table 15.2: Pit Optimization Parameters

Parameter	Unit	2016 FS Value
Revenue, Smelting & Refining		
Gold price	US$/oz Au	1,200
Exchange Rate	C$:US$	0.87
Payable metal	%	100
TC/RC/Transport	C$/oz Au	8.62
Royalty @ 1% NSR	C$/oz Au	13.79
Net gold value per ounce	C$/oz	1,357
Net gold value per gram	C$/g	43.63
OPEX Estimates		
OP Waste Mining Cost	C$/t waste mined	2.25
OP Ore Mining Cost	C$/t ore mined	2.25
Strip Ratio (estimated)	W:O	4
Open Pit Mining Cost	C$/t milled	11.25
Process Cost		
Leach cost for all ore types and deposits	$/t leached	6.00
G&A	C$/t leached	4.00
Total OPEX Cost (Excluding Mining) for All Ore Types and Deposits	$/t leached	10.00
Recovery and Dilution		
External Mining Dilution – Supremo	%	7
External Mining Dilution – Double Double	%	9
External Mining Dilution – Kona	%	10
External Mining Dilution – Latte	%	5
Mining Recovery	%	95
Gold Leach Recovery		
Supremo Oxide	%	90
Double Double Oxide	%	92
Kona Oxide	%	85
Latte Oxide	%	90
Supremo Upper Transition	%	82
Double Double Upper Transition	%	77
Kona Upper Transition	%	69
Latte Upper Transition	%	79
Supremo Middle Transition	%	57
Double Double Middle Transition	%	57
Kona Middle Transition	%	49
Latte Middle Transition	%	60
Supremo Lower Transition	%	27

Parameter	Unit	2016 FS Value
Double Double Lower Transition	%	27
Kona Lower Transition	%	18
Latte Lower Transition	%	27
Other		
Overall Pit Slope Angles	degrees	35-50
Heap Leach Production Rate	Mt/a	5

Source: JDS 2016

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 Latte and Supremo deposits only) 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 Cut-off Grades by Deposit and Oxidation Type

Item	Unit	COG
Incremental Cut-off Grade - OP		
Latte Pit		
Oxide Gold Cut-off Grade	g/t Au	0.27
Upper Transition Gold Cut-off Grade	g/t Au	0.3
Middle Transition Gold Cut-off Grade	g/t Au	0.4
Lower Transition Gold Cut-off Grade	g/t Au	0.89
Double Double Pit		
Oxide Gold Cut-off Grade	g/t Au	0.27
Upper Transition Gold Cut-off Grade	g/t Au	0.32
Middle Transition Gold Cut-off Grade	g/t Au	0.44
Lower Transition Gold Cut-off Grade	g/t Au	0.93
Kona Pit		
Oxide Gold Cut-off Grade	g/t Au	0.3
Upper Transition Gold Cut-off Grade	g/t Au	0.37
Middle Transition Gold Cut-off Grade	g/t Au	0.51
Lower Transition Gold Cut-off Grade	g/t Au	1.4
Supremo Pit		
Oxide Gold Cut-off Grade	g/t Au	0.27
Upper Transition Gold Cut-off Grade	g/t Au	0.3
Middle Transition Gold Cut-off Grade	g/t Au	0.43
Lower Transition Gold Cut-off Grade	g/t Au	0.91

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 the pit design.

Table 15.4: Pit Design Parameters

Description	Value
Ultimate Pit Design Parameters – All Pits	
Bench Height	5 m (single, working)
	20 m (quadruple; final pit)
Face Angle	65° to 75° (double bench, final pit)
Berm Width	10 m
Inter-ramp Angle (IRA)	46° to 52°
Ramp Width – Double Lane	27 m (total excavation)
Ramp Width (Single lane -lower benches)	20 m
Ramp gradient	10%
Overall Angle (OSA)	35° to 50°

Source: JDS 2016

15.1.8 Open Pit Mineral Reserves Estimate Statement

The Coffee Gold open pit mineral reserve is presented in Table 15.5 and further summarized by oxidation type in Table 15.5.

Table 15.5: Coffee Gold Open Pit Mineral Reserve Estimate

Deposit	Reserve Category	Total Ore ('000s t)	Gold Cut-off Grade* (g/t)	Gold Grade (g/t)	Contained Gold ('000s oz)
Latte	Probable	11,548	0.27	1.30	484
Double Double	Probable	1,057	0.27	3.24	110
Kona	Probable	868	0.30	1.20	33
Supremo	Probable	32,883	0.27	1.45	1,530
Total Mineral Reserve	**Probable**	**46,356**		**1.45**	**2,157**

*Note: Gold cut-off grade for Oxide shown

Source: JDS 2016

Table 15.6: Coffee Gold Mineral Reserves by Oxidation Type

Description	Unit	Supremo	Latte	Double Double	Kona	Total
Oxide	(ktonnes)	29,735	7,191	444	736	38,105
	Au (g/t)	1.43	1.27	3.06	1.20	1.41
	Au (koz)	1,365	293	44	28	1,731
Upper Transition	(ktonnes)	2,385	2,454	373	114	5,326
	Au (g/t)	1.56	1.25	3.84	1.22	1.57
	Au (koz)	120	99	46	4	269
Middle Transition	(ktonnes)	534	1,441	194	17	2,185
	Au (g/t)	1.68	1.43	2.70	0.84	1.6
	Au (koz)	29	66	17	0	112
Lower Transition	(ktonnes)	230	462	47	1	740
	Au (g/t)	2.09	1.71	2.43	2.22	1.88
	Au (koz)	15	25	4	0	45
Total	**(ktonnes)**	**32,883**	**11,548**	**1,057**	**868**	**46,356**
	Au (g/t)	**1.45**	**1.30**	**3.24**	**1.20**	**1.45**
	Au (koz)	**1,530**	**484**	**110**	**33**	**2,157**

Source: JDS 2016

16 Mining Methods

16.1 Introduction

The Coffee Gold Project comprises the Latte, Double Double, Supremo and Kona pits 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 46.4 Mt of ore at an average gold head grade of 1.45 g/t. The contained gold is estimated to be 2.2 Moz.

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

Capitalized mining volumes are reported up to the end of Q3 Year -1. Subsequent project and production years are aligned with the calendar year.

16.2 Open Pit Mining

16.2.1 Introduction

This section outlines the parameters and procedures used to design the open pit mine and establish a practical mining schedule for the Coffee Gold Project. This section also presents the methodology for the selection of open pit equipment and the estimation of manpower requirements.

16.2.2 Mine Design Methodology and Design Criteria

16.2.2.1 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 model;
- 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 including the four oxidation types for each area;
- Calculation of economic ultimate pit limits for the various deposits using the DataMine NPV Scheduler (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 process plant feed and mine production constraints;
- Development of waste rock storage facility (WRSF) 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; and
- 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;
- 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.

16.2.2.2 Site Topography and Climatic Conditions

Mine planning for the Project considered the northern climate environment, the annual resupply period, permafrost, and a fly-in-fly-out camp situation, as well as the physical characteristics of the ore and waste rock.

The extent and intensity of oxidation has been interpreted using a combination of qualitative data collected during drill core and chip logging plus the solubility characteristics derived from a suite of samples tested for cyanide gold solubility. Four oxide types or domains have been interpreted (oxide, upper transition, middle transition, lower transition) which, in general, represent decreasing intensity of oxidation and decreasing leach recoveries with depth below surface.

16.2.2.3 Topographic and Resource Model Description

16.2.2.3.1 Topography

Mine topography, including the WRSF and heap leach pad areas, was provided digitally by Kaminak in UTM NAD83, Zone 7 coordinates. Topography was supplied as LiDAR with contour intervals of 2 m. This was used for all pit design calculations and engineering estimates. Volumetric estimates were derived from design surfaces intersecting the topographic surface.

16.2.2.3.2 Resource Model

The 3D resource block models for the various deposits used in this Feasibility Study were prepared by Sim Geological Inc. 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, and gold grade.

16.2.2.4 Geomechanical Characterization

A field data collection program was designed with the primary objective of rock mass characterization and discontinuity orientation to serve as the basis of geomechanical model development. The program was designed to fill characterization gaps that were identified in Kaminak's existing geomechanical database which consisted of data collected on select resource drill holes. The 2015 field data collection consisted of geomechanical core logging and discontinuity orientation, point load testing and laboratory rock strength testing. A total of 6 HQ diameter core holes were logged and tested between the four deposits for a total of 833 m in length.

Results of the geomechanical characterization program suggest that, with the exception of the oxide materials which will be mostly mined and processed, the rock mass at Coffee is generally of good geomechanical quality. Table 16.1 contains a summary of rock mass characteristic data derived from the 2015 geomechanical drilling program for each of the three primary lithology types at Coffee.

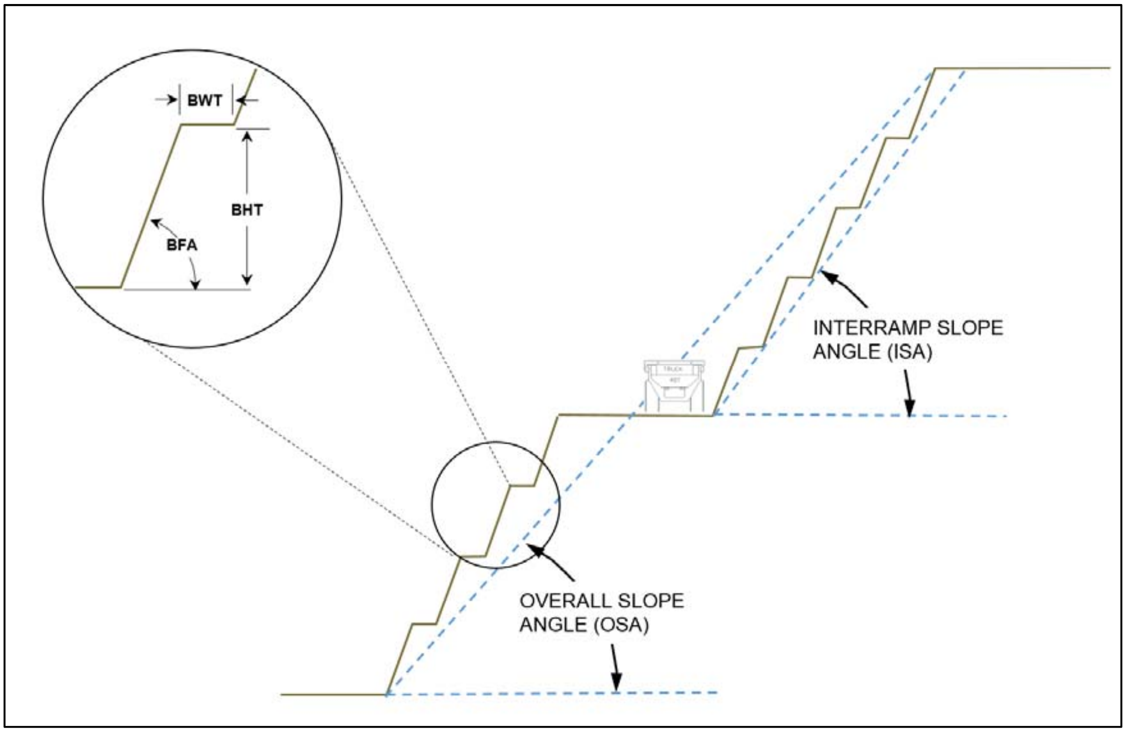
Table 16.1: Summary of Rock Mass Characteristics

Lithology	Pit	Average UCS (MPa)	No. Valid UCS Tests	Average RMR (1989)	Average RQD (%)	Total Core Length (m)
Gneiss	Supremo & Double Double	90	9	64	81	515
Schist	Latte	94	4	64	87	227
Granite	Kona	130	2	76	95	96

16.2.2.4.1 Slope Stability Analyses

Slope design involves analysis of the three major components of an open pit slope, i.e., bench configuration, interramp slope angle (ISA) and overall slope angle (OSA) as shown in Figure 16.1. The bench configuration, which is defined by the bench face angle (BFA), bench height (BHT), and berm width (BWT), defines the interramp slope angle. The overall slope angle consists of interramp slope sections separated by wide step-outs for haul roads, mine infrastructure or geotechnical purposes. The overall slope angles at Coffee will be approximately equal to the corresponding interramp angles except in areas where a haul road exists.

Figure 16.1: Pit Slope Design Components



Given the overall good rock mass quality anticipated to comprise the pit walls at Coffee, the bench configuration was analyzed first to determine the maximum achievable bench face angle based on geologic structure alone. Bench scale and lower interramp slopes are most realistically assessed using stochastic models that evaluate structurally controlled failure mechanisms. This was accomplished using the software program SBlock (Esterhuizen, 2004) as well as SRK proprietary routines compiled using functions available in Oracle's Crystal Ball statistical add-on to Microsoft Excel. The following was concluded from the analyses:

- Due mostly to the north-south orientation of the pit, benches at Supremo are not anticipated to be significantly impacted by structurally controlled instabilities. As such, a maximum achievable bench face of 75° was estimated for Supremo based on operational constraints as discussed below;
- The stability of benches on the north Latte and Double Double pit walls will likely be controlled by the dominant southerly (inward) dipping foliation discontinuities. The analyses indicate a maximum achievable bench face angle of approximately 65° for the Latte and Double Double north walls. Stability of the south and end pit wall benches are not anticipated to be governed by geologic structure controls and were, therefore, estimated to have a maximum achievable bench face of 75° based on operational considerations; and,
- Benches on the west Kona pit wall have a slightly higher probability of structurally controlled instabilities fostered by a moderately eastward dipping joint set. The analyses indicate a maximum bench face angle of approximately 70° is achievable for this portion of the pit. The

remaining pit areas are not anticipated to be significantly impacted by geologic structure controlled instabilities. A maximum achievable bench face of 75° was estimated for the remainder of the pit based on operational constraints.

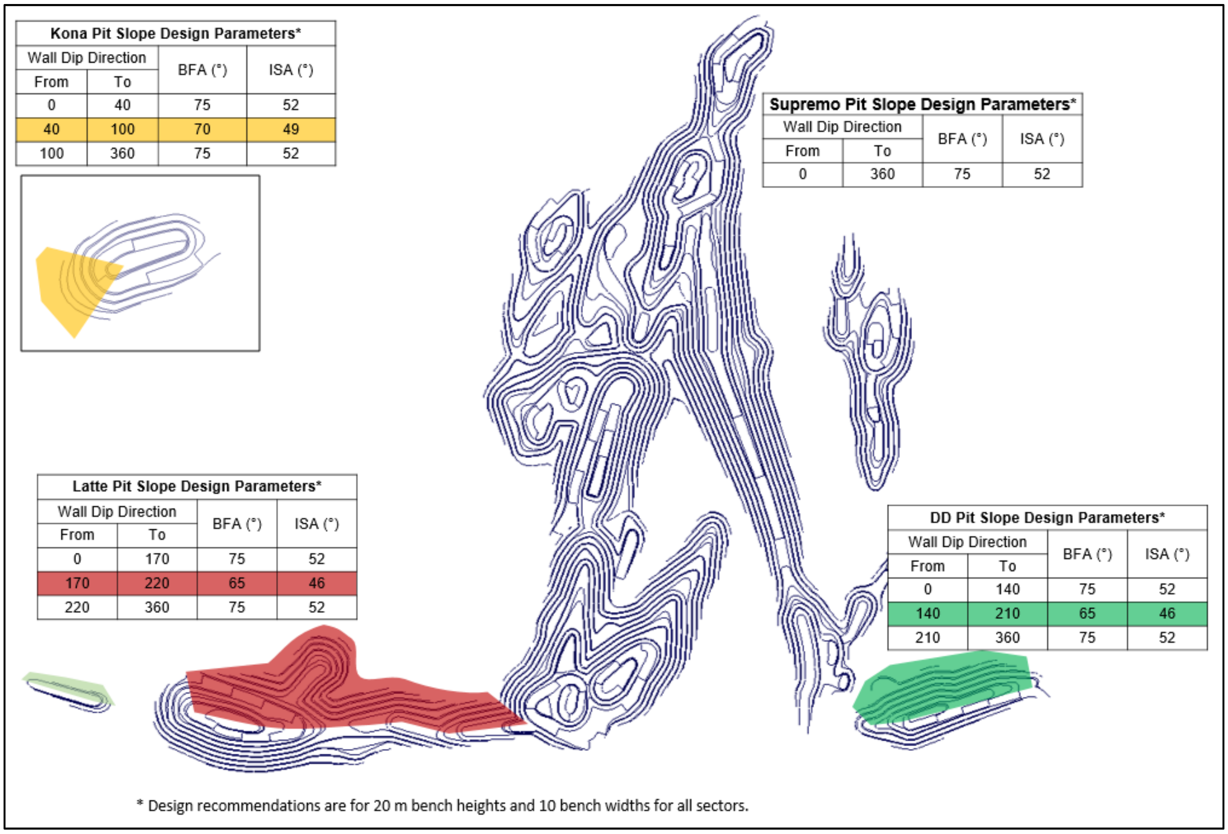
It should be noted that the bench stability analyses are based solely on orientations of geologic structure and do not directly consider effects of weathering, alteration, blasting or excavation techniques. Depending on the quality of blasting and excavation techniques, achievable bench face angles might be reduced from the theoretical angles determined by these analyses. When taking these operational effects into consideration, it is rare to achieve bench face angles greater than about 75º unless there is a steeper structure controlling the bench geometry. Increasing bench face angles to greater than about 75º may be achievable but usually require more rigorous drilling and blasting effort and specialized controlled blasting techniques than are commonly practiced.

Based on the conclusion of the bench design analyses, bench configurations and the resulting maximum interramp slope design parameters were provided by SRK and an initial detailed pit was designed by JDS incorporating necessary ramps and infrastructure. Stability of the overall slopes in the detailed design was then confirmed using the limit equilibrium slope stability modeling software package, Slide (Rocscience, 2015). Rock mass shear strengths were developed for each rock type based on the results of the field and laboratory test work assuming the Hoek-Brown (2002) rock mass shear strength criteria. Several variations of rock mass strengths were evaluated for each cross section to investigate the amount of influence different joint lengths may have on the overall rock mass strength. Results of the overall slope stability analyses demonstrate that the bench configuration based slope angles either meet or exceed the acceptable safety factor of 1.3.

16.2.2.4.2 Pit Slope Design Recommendations

The resulting geomechanical pit slope design recommendations are shown in Figure 16.2. The recommendations in Figure 16.2 are based on dip direction of the pit wall (e.g. for an east-west trending wall, facing south, the slope dip direction would be 180° azimuth).

Figure 16.2: Pit Slope Design Recommendations



Source: JDS 2015

A 75º bench face angle is recommended for the Latte and Double Double south walls as well as the Supremo pit and a majority of Kona based on the dip and dip directions of the structures relative to the slope orientation. The geomechanical advantage of the 75º bench face angle is improved rockfall control based on the anticipation that the 75º face angle can be successfully achieved without requiring exceptional care in excavation practices. It is recommended that trials in non-critical areas of the pit be implemented in order to determine the operational requirements that will be required to achieve this design.

Double benching is recommended as being more favorable in fresh, competent rock. The double, 20 m high benching will permit the incorporation of more adequately-sized berms for rockfall control, provided the drilling and, to a greater extent, blasting practices meet best practice standards, thereby reducing the number of crests and toes that are subject to potential damage.

16.2.2.4.3 Recommendations for Additional Geomechanical Work

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 localized instabilities. The data collection should concentrate on developing geotechnical databases 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.

16.2.3 Open Pit Optimization and Sensitivity Analysis

16.2.3.1 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.

Only Indicated Mineral resources were included in the pit optimization process. No part of the resource was classified in the Measured Resource category.

Table 16.2 summarizes the parameters used for each deposit, along with incremental cut-off grade calculations and mining dilution for both 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.

Table 16.2: Open Pit Optimization Parameters Used for Cut-off Grade Calculation

Parameter	Unit	FS Optimization Value
Revenue, Smelting & Refining		
Gold price	US$/oz Au	1,200
Exchange Rate	US$:C$	0.87
Payable metal	%	100
TC/RC/Transport	$/oz Au	8.62
Royalty @ 1% NSR	$/oz Au	13.79
Net gold value per ounce	$/oz	1,357
Net gold value per gram	$/g	43.63
OPEX Estimates		
Open Pit Waste Mining Cost	$/t waste mined	2.25
Open Pit Ore Mining Cost	$/t ore mined	2.25
Strip Ratio (estimated)	W:O	4
Open Pit Mining Cost	$/t leached	11.25
Process Cost		
Leach cost for all ore types and deposits	$/t leached	6
G&A	$/t leached	4
Total OPEX Cost (excluding mining) for all ore types and deposits	$/t leached	10
Recovery and Dilution		
External Mining Dilution – Supremo	%	7
External Mining Dilution – Double Double	%	9
External Mining Dilution – Kona	%	10
External Mining Dilution – Latte	%	5
Mining Recovery	%	95
Gold Recovery		
Supremo Oxide Leach Gold Recovery	%	90
Double Double Oxide Leach Gold Recovery	%	92
Kona Oxide Leach Gold Recovery	%	85
Latte Oxide Leach Gold Recovery	%	90
Supremo Upper Transition Leach Gold Recovery	%	82
Double Double Upper Transition Leach Gold Recovery	%	77
Kona Upper Transition Leach Gold Recovery	%	69
Latte Upper Transition Leach Gold Recovery	%	79
Supremo Middle Transition Leach Gold Recovery	%	57
Double Double Middle Transition Leach Gold Recovery	%	57
Kona Middle Transition Leach Gold Recovery	%	49
Latte Middle Transition Leach Gold Recovery	%	60
Supremo Lower Transition Leach Gold Recovery	%	27

Parameter	Unit	FS Optimization Value
Double Double Lower Transition Leach Gold Recovery	%	27
Kona Lower Transition Leach Gold Recovery	%	18
Latte Lower Transition Leach Gold Recovery	%	27
Cut-off Grade Calculations		
Supremo Oxide Gold Cut-off Grade	g/t Au	0.27
Double Double Oxide Gold Cut-off Grade	g/t Au	0.27
Kona Oxide Gold Cut-off Grade	g/t Au	0.3
Latte Oxide Gold Cut-off Grade	g/t Au	0.27
Supremo Upper Transition Gold Cut-off Grade	g/t Au	0.3
Double Double Upper Transition Gold Cut-off Grade	g/t Au	0.32
Kona Upper Transition Gold Cut-off Grade	g/t Au	0.37
Latte Upper Transition Gold Cut-off Grade	g/t Au	0.3
Supremo Middle Transition Gold Cut-off Grade	g/t Au	0.43
Double Double Middle Transition Gold Cut-off Grade	g/t Au	0.44
Kona Middle Transition Gold Cut-off Grade	g/t Au	0.51
Latte Middle Transition Gold Cut-off Grade	g/t Au	0.4
Supremo Lower Transition Gold Cut-off Grade	g/t Au	0.91
Double Double Lower Transition Gold Cut-off Grade	g/t Au	0.93
Kona Lower Transition Gold Cut-off Grade	g/t Au	1.4
Latte Lower Transition Gold Cut-off Grade	g/t Au	0.89
Other		
Overall Pit Slope Angles	degrees	35-50
Heap Leach Production Rate	Mt/a	5

Source: JDS 2016

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 (waste rock storage facilities, heap leach pads, 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.

16.2.3.2 Open Pit Optimization Results

A series of optimized shells were generated for the various deposits at Coffee 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 maximum throughput rate of 5.0 Mt/year. 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 Latte and Supremo deposits only) 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 Coffee Gold, are summarized in Table 16.3.

Table 16.3: Pit Optimization Results – all deposits

Description	Unit	Supremo (#42)	Latte (#36)	Double Double (#37)	Kona (#35)	Total
Oxide	(kt)	30,978	7,522	480	790	39,770
	Au (g/t)	1.42	1.26	2.91	1.24	1.4
	Au (koz)	1,413	305	45	31	1,794
Upper Transition	(kt)	2,495	2,651	376	139	5,660
	Au (g/t)	1.61	1.26	4.1	1.23	1.6
	Au (koz)	129	107	49	5	291
Middle Transition	(kt)	576	1,506	183	22	2,287
	Au (g/t)	1.81	1.46	2.87	0.88	1.65
	Au (koz)	33	71	17	1	122
Lower Transition	(kt)	241	437	46	1	725
	Au (g/t)	2.08	1.86	2.5	2.37	1.97
	Au (koz)	16	26	4	0	46
Total Heap Leach Feed	(kt)	34,290	12,116	1,084	951	48,442
	Au (g/t)	1.44	1.31	3.29	1.23	1.45
	Au (koz)	1,592	509	115	38	2,253
Waste	(kt)	218,298	27,731	9,728	3,406	259,164
Total Material	(kt)	252,588	39,847	10,812	4,358	307,605
Strip Ratio	(t:t)	6.4	2.3	9	3.6	5.4

Source: JDS 2016

16.2.4 Open Pit Design Parameters

16.2.4.1 Geotechnical and Hydrogeological Characterization

SRK Consulting's recommendations for slope angles based on their analysis of geotechnical and hydrogeological conditions are described in Appendix C of the full FS report.

16.2.4.2 General Design Parameters

The general design parameters used in the various detailed pit and phase designs, including the geotechnical data described above, are:

- Bench height, single bench mining 5 m
- Height between catch benches 20 m
- Bench face angle 65° to 75° (variable)
- Berm width 10 m
- Total road width allowance 27 m
- Running surface on final two-way roads 20.7 m
- Berms and ditches 6.4 m
- Maximum ramp grades 10%
- Single lane road allowance 20 m
- Minimum operating width 50 m to 60 m
- Minimum pit bottom width 25 m to 30 m
- Pit bottom subout depth 5 m

16.2.4.3 Haul Road and Ramp Design Parameters

Coffee Gold site roads fall into two categories as summarized in Table 16.4. All site roads are considered private roads and access will be controlled by Kaminak.

Table 16.4: Coffee Gold 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 m	8%

Source: JDS 2015

The primary haulage roads are required between the various open pit deposits and the primary ore crusher, waste rock facilities, 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 waste rock storage facilities are designed to be all-fill construction. Roads are proposed to be constructed of non-potentially acid generating (NPAG) material from the open pits. Dust control on the roads will be done using water trucks, with the addition of chemical suppressants as needed.

The main in-pit haul roads and ramps are designed to be 27 m in width. 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 16.5.

Table 16.5: 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
Total Road Allowance	27

Source: JDS 2015

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).

16.2.5 Open Pit Designs

Detailed mine designs were undertaken on all four proposed open pits and the approximate dimensions are shown in Table 16.6, with plan views of each open pit design shown in Figure 16.3 to Figure 16.6.

Table 16.6: Open Pit Dimensions

Open Pit	Length (m)	Width (m)	Depth (m)
Latte	1,300	275	150
Double Double	560	200	100
Kona	330	155	80
Supremo	2,100	400	140

Source: JDS 2015

Figure 16.3: Latte Pit Design

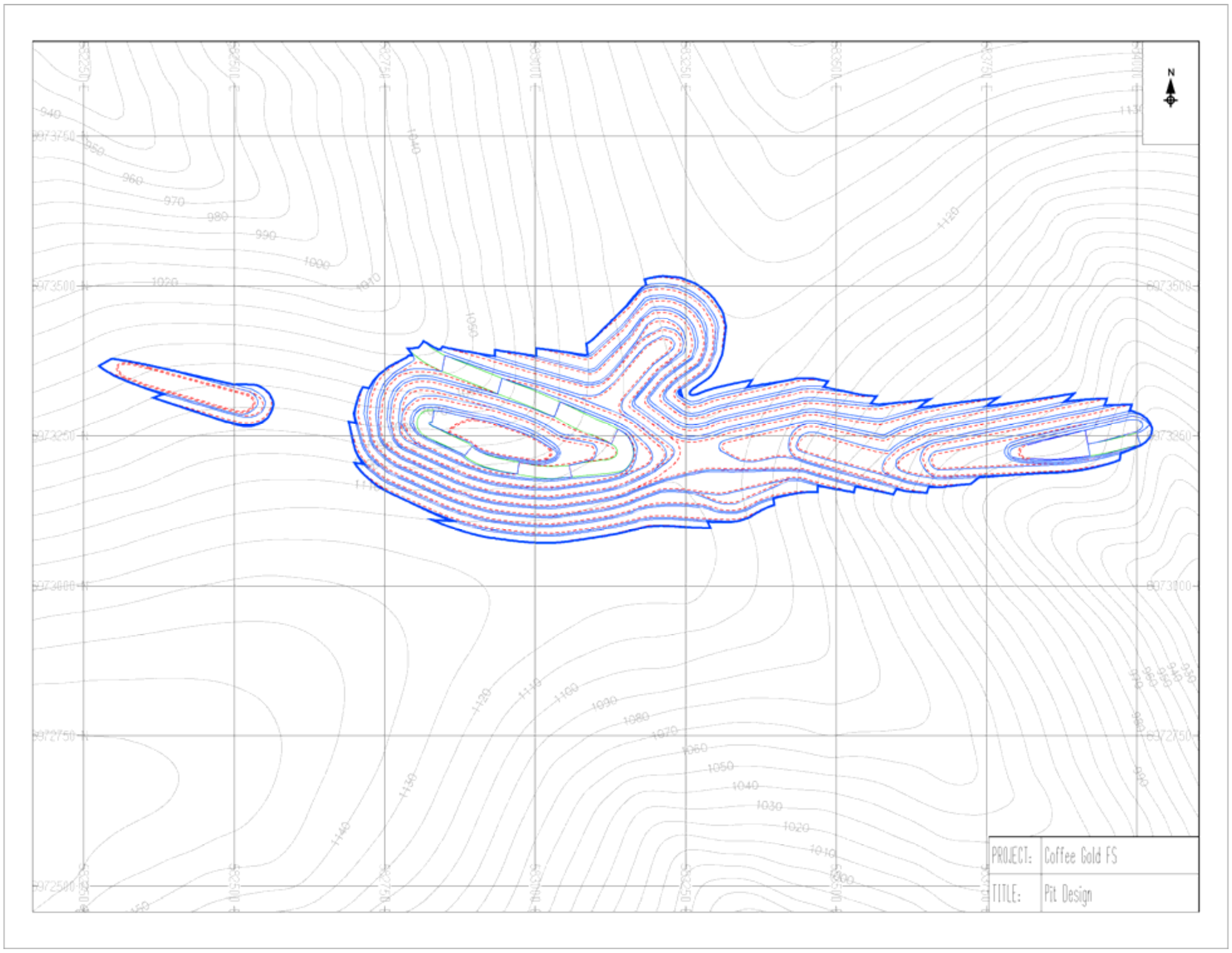


Figure 16.4: Double Double Pit Design

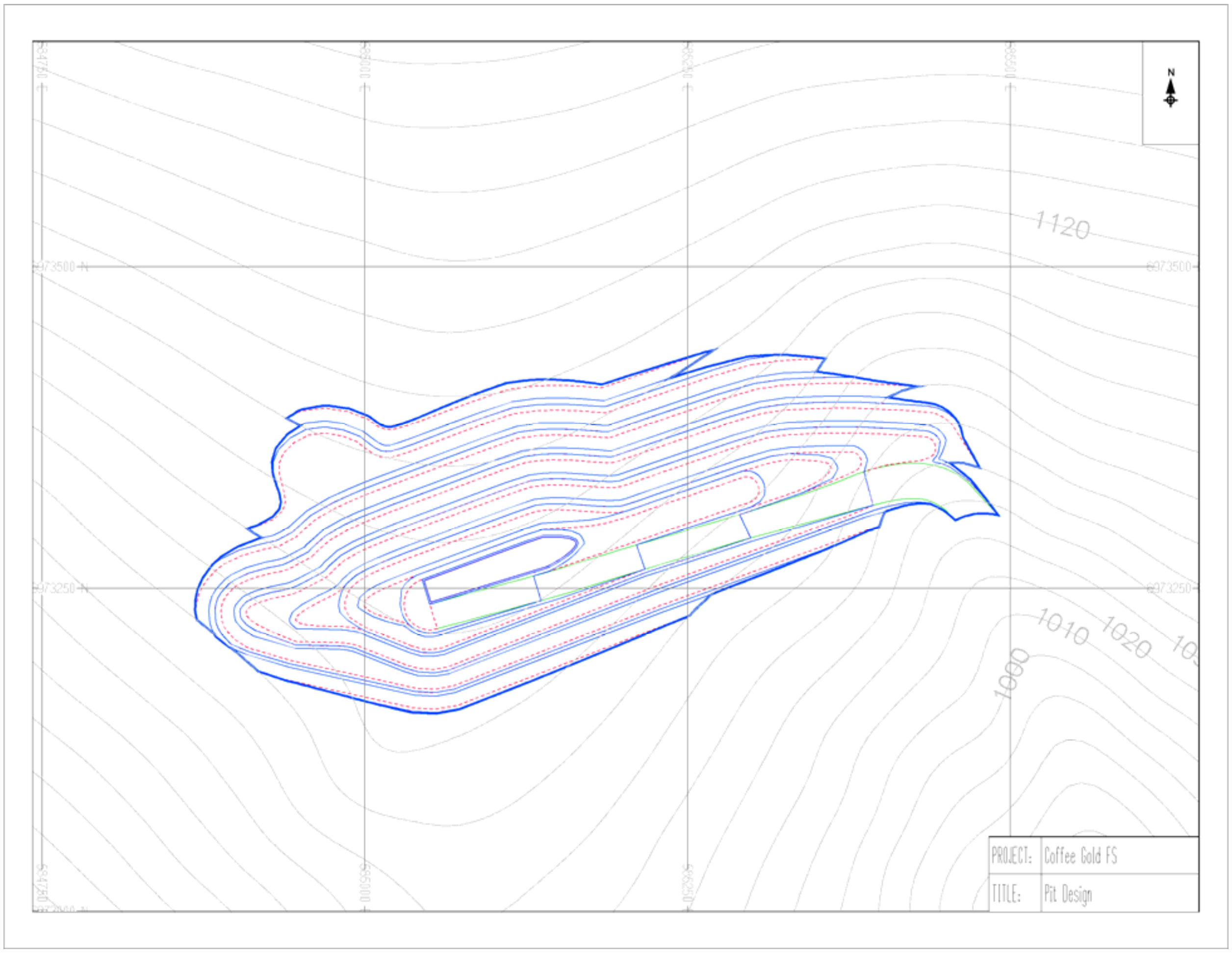


Figure 16.5: Kona Pit Design

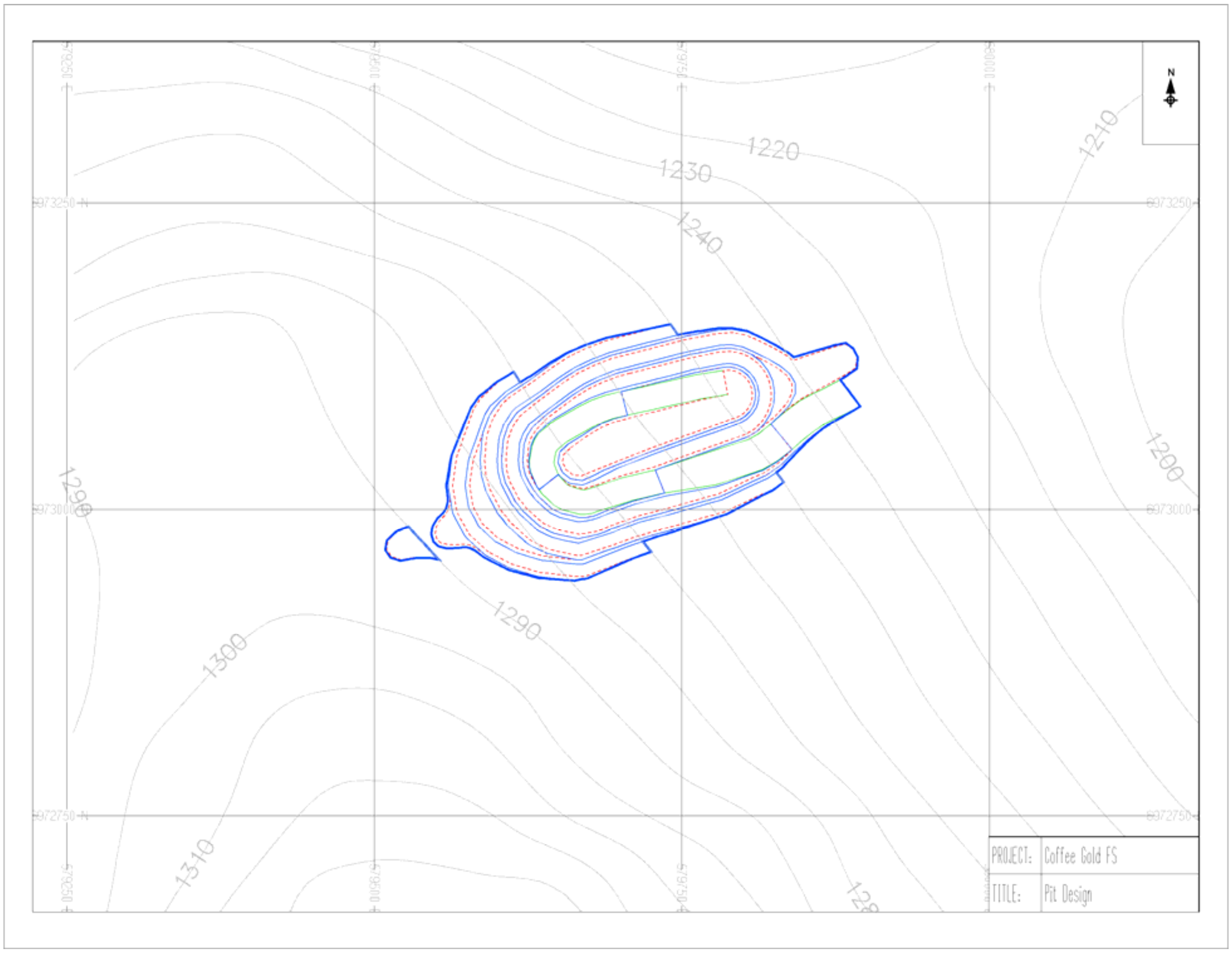
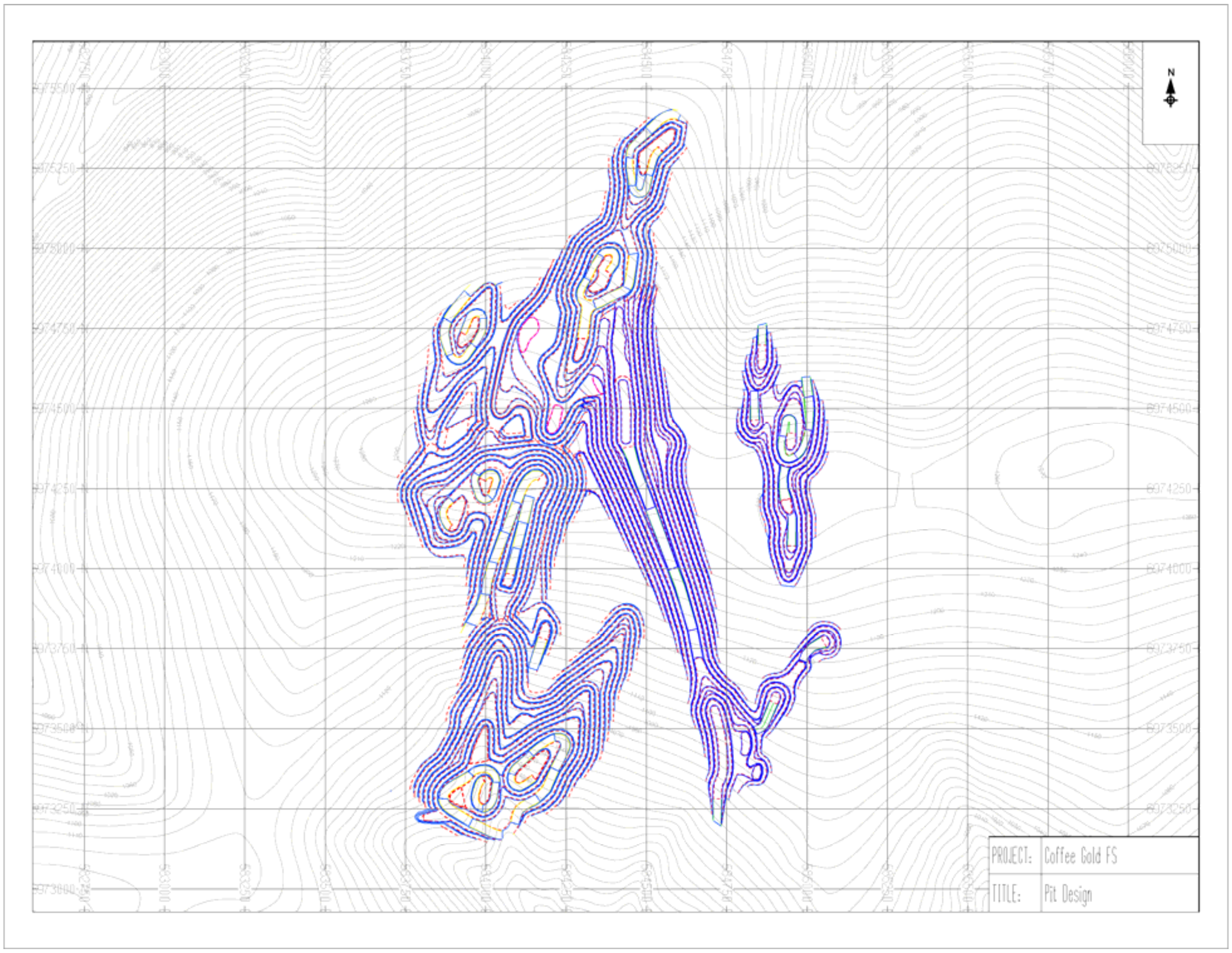


Figure 16.6: Supremo Pit Design



The detailed pit designs, along with calculated COGs, were used in the determination of the mineral reserve estimate for each deposit as summarized in Table 16.7 and Table 16.8.

Table 16.7: Coffee Gold Open Pit Mineral Reserve Estimate

Deposit	Reserve Category	Total Ore	Gold Cut-off Grade*	Gold Grade	Contained Gold
		(kt)	(g/t)	(g/t)	(koz)
Latte	Probable	11,548	0.27	1.30	484
Double Double	Probable	1,057	0.27	3.24	110
Kona	Probable	868	0.3	1.20	33
Supremo	Probable	32,883	0.27	1.45	1,530
Total Mineral Reserves	Probable	46,356		1.45	2,157

*Note: Gold cut-off grade for Oxide shown

Source: JDS 2015

Table 16.8: Mineral Reserves by Oxidation Type (all Probable Reserve Category)

Description	Unit	Supremo	Latte	Double Double	Kona	Total
Oxide	kt	29,735	7,191	444	736	38,105
	Au (g/t)	1.43	1.27	3.06	1.20	1.41
	Au (koz)	1,365	293	44	28	1,731
Upper Transition	kt	2,385	2,454	373	114	5,326
	Au (g/t)	1.56	1.25	3.84	1.22	1.57
	Au (koz)	120	99	46	4	269
Middle Transition	kt	534	1,441	194	17	2,185
	Au (g/t)	1.68	1.43	2.70	0.84	1.60
	Au (koz)	29	66	17	0	112
Lower Transition	kt	230	462	47	1	740
	Au (g/t)	2.09	1.71	2.43	2.22	1.88
	Au (koz)	15	25	4	0	45
Total Heap Leach Feed	kt	32,883	11,548	1,057	868	46,356
	Au (g/t)	1.45	1.3	3.24	1.20	1.45
	Au (koz)	1,530	484	110	33	2,157

Source: JDS 2015

16.2.5.1 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 16.9. Total ore tonnages in the final open pit designs are 4.3% lower than the optimized shells, with the corresponding waste material 2.4% higher. These 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 within acceptable limits.

Table 16.9: Material in Optimized Shell versus Final Pit Designs

Description	Total Heap Leach Feed			Waste Quantity (kt)	Total Quantity (kt)	Strip Ratio (t:t)
	Quantity (kt)	Grade Au (g/t)	Metal Au (koz)			
Pit Optimization Results						
Supremo	34,290	1.44	1,592	218,298	252,588	6.4
Latte	12,116	1.31	509	27,731	39,847	2.3
Double Double	1,084	3.29	115	9,728	10,812	9.0
Kona	951	1.23	38	3,406	4,358	3.6
Total	48,442	1.45	2,253	259,164	307,605	5.4
Mineral Reserves (Final Pit Design)						
Supremo	32,883	1.45	1,530	223,035	255,919	6.8
Latte	11,548	1.3	484	28,203	39,751	2.4
Double Double-	1,057	3.24	110	10,885	11,942	10.3
Kona	868	1.2	33	3,237	4,105	3.7
Total	**46,356**	**1.45**	**2,157**	**265,361**	**311,717**	**5.7**
Difference Reserve vs. Optimization	-4.30%	0.00%	-4.30%	2.40%	1.30%	

Source: JDS 2015

16.2.5.2 Waste Materials

Waste material to be mined from the Coffee Gold open pits was categorized by Lorax Environmental Services Ltd. (Lorax) into potentially acid generating (PAG) or NPAG material based on geochemical characterization of the material and its acid generating potential. For mine planning purposes it was assumed that the two different materials would be identified during mining. Only the exposed pit walls at the Kona Pit was categorized as PAG and, as such, the Kona pit will be backfilled at the end of mine life.

Table 16.10 summarizes the waste material to be mined by open pit and by material type. Note that no significant amounts of overburden are expected within the various open pits.

Table 16.10: Open Pit Waste Rock Summary

Pit	NPAG
Latte	28,203
Double Double	10,885
Kona	3,237
Supremo	223,035
Total	262,123

Source: JDS 2015

16.2.5.3 Open Pit Phase Design

For Double Double and Kona pits no additional pushbacks or phases were allowed for in the mine plan development due to their relatively small footprints. For Latte pit, one initial pushback has been designed and for the much larger Supremo pit, a total of five phases or pushbacks are envisaged in order to optimize the mine schedule and maximize the project value.

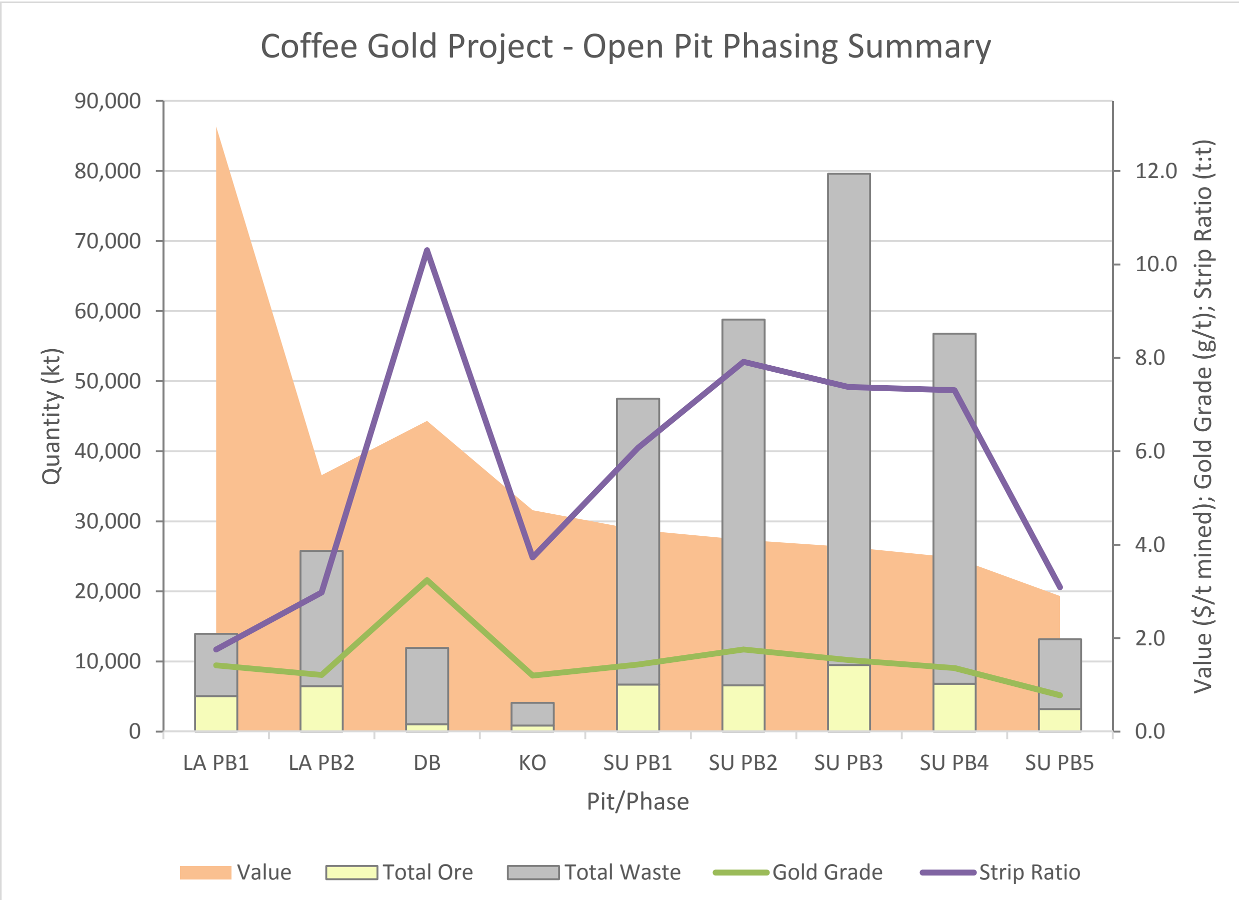
The mining schedule maximizes economic returns and achieves the target throughput of 5.0 Mt/a through concurrent mining of the various deposits. The open pit mining sequence, which is based on mining higher value material early on in the mine life, begins with the Latte pit, followed by Double Double, Kona and finally Supremo pit.

Figure 16.7 further summarizes the pit and phase designs for each of the deposits, illustrating ore and waste 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.

The waste rock generated for each of the individual deposits is planned to be placed into various waste rock storage facilities adjacent to the final pit limits or ultimately backfilled into mined out pits. All heap leach feed material will be either direct dumped to the crusher itself or hauled to the run-of-mine ore stockpile adjacent to the crusher.

The mining operation will be responsible for loading the crushed ore, hauling it to the heap leach pad and spreading the crushed ore on the pad.

Figure 16.7: Open Pit Summary



Source: JDS 2015

16.2.6 Mine Production Schedule

16.2.6.1 Summary

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

- Maximize NPV;
- Maximize the grade and value in the early years of the operation through the use of ROM stockpiles and concurrent open pit mining of the various pits/phases;
- Ensure heap leach ore loading of 5.0 Mt/a;
- Ensure adequate waste material suitable for construction is produced from the open pits in the pre-production period;
- Schedule on a quarterly basis;
- Plan up to two open pits active in production during early years (deposits are most economical when the open pit mines are mined concurrently);

- Maximum 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 39.8 Mt (LOM average 33.0 Mt/year);
- Capitalize pre-stripping tonnage (11.2 Mt total material, of which 2.2 Mt is ore, to be mined to end of Q3 in Year -1 using owner-operated equipment and resources);
- Establish a run-of-mine (ROM) stockpile to accommodate mining throughout the year including the winter period (January through March) when the crusher is not operational and there is no ore stacking of the heap leach pad;
- The mining operation will be responsible for the operation and maintenance of the equipment used to load, haul and spread crushed ore; and
- Plan on operating the open pit mine 365 days per year (mine cost model allows for ten non-operating days per year due to weather delays).

16.2.6.2 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. The heap leach pad is to be operated 275 days per annum. No ore crushing and stacking occurs during the coldest months of the year, January through March. An average annual throughput of 5.0 Mt has been assumed.

16.2.6.3 Pit Sequencing

The pit sequencing corresponds to the detailed pit designs described in Section 16.2.4. Pit sequence focuses on achieving the required heap leach feed production rate, mining of higher value material early in the mine life, while balancing gold grade and strip ratios.

16.2.6.4 Pre-production Development Schedule

The pre-production period is up to the end of Q3 Year -1. Open pit mining activities during this period are scheduled to provide sufficient ore exposure for heap leach start-up. To this end, heap leach thermal modelling indicates that at least 3.5 Mt of ore needs to be placed on the heap leach pad by the end of December in Year -1 in order to ensure thermal integrity of the ore. Mining also focuses on providing sufficient waste rock for construction (site roads, laydown areas, etc.). Ore mined during the pre-production period is planned to be stockpiled and re-handled during crusher and heap leach stacking operations. Mining in the pre-production period will also create a high-grade ROM stockpile in order to maximize heap leach head grade in the early part of the production schedule.

A total of 9.0 Mt of waste and 2.2 Mt of ore are scheduled to be mined from the Latte and Double Double open pits to the end of Q3 Year -1 and will be carried out with the owner's mining fleet.

Clearing and grubbing of the various pit areas, heap leach pads and waste rock facilities were included in the open pit capital cost estimate. The estimate was based on total area to be cleared and assumes clearing/grubbing to be undertaken with owner-operated fleet prior to full scale production taking place.

16.2.6.5 Mine Plan and Open Pit Production Schedule

Table 16.11 is a summary of ore and waste rock movement by year and by pit for the LOM mine production schedule along with the heap leach feed schedule and ROM stockpile balance. Note that Year-1 comprises the pre-production period to the end of Q3 as well as the start of operations in Q4 Year -1**.** Gold recoveries are as-mined and do account for heap leach pad solution dynamics.

Figure 16.8 and Figure 16.9 summarize ore/waste tonnages, grade, strip ratio and value by quarter.

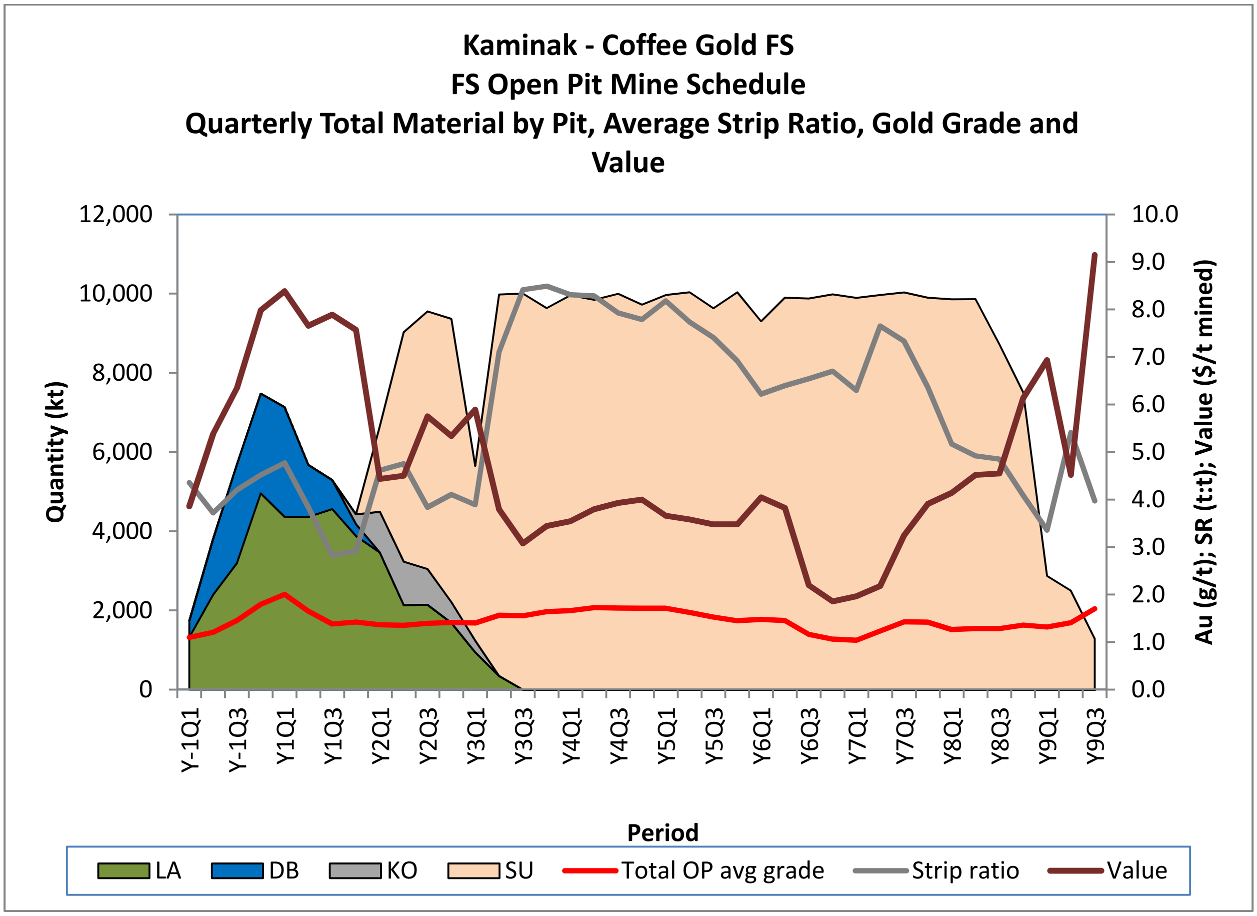
Figure 16.10 illustrates quarterly bench advance rates from each open pit and phase. Figure 16.11 summarizes annual waste tonnages (by lithology type) along with average gold grades and strip ratio.

Table 16.11: LOM Production Schedule – Coffee Gold Deposits

Description	Unit	Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Latte												
Total Ore	kt	11,548	3,167	4,250	3,484	647	0	0	0	0	0	0
Total grade	g/t	1.30	1.26	1.38	1.30	1.04	0	0	0	0	0	0
Total Au contain	koz	484	128	189	145	22	0	0	0	0	0	0
Total Waste	kt	28,203	8,705	12,914	5,941	644	0	0	0	0	0	0
Strip Ratio	t:t	2.4	2.7	3	1.7	1						
Total Material	kt	39,751	11,872	17,164	9,424	1,291	0	0	0	0	0	0
Double Double												
Total Ore	kt	1,057	412	645	0	0	0	0	0	0	0	0
Total grade	g/t	3.24	3.33	3.18	0	0	0	0	0	0	0	0
Total Au contain	koz	110	44	66	0	0	0	0	0	0	0	0
Total Waste	kt	10,885	6,421	4,465	0	0	0	0	0	0	0	0
Strip Ratio	t:t	10.3	15.6	6.9								
Total Material	kt	11,942	6,833	5,109	0	0	0	0	0	0	0	0
Kona												
Total Ore	kt	868	0	28	719	121	0	0	0	0	0	0
Total grade	g/t	1.20	0	0.63	1.11	1.84	0	0	0	0	0	0
Total Au contain	koz	33	0	1	26	7	0	0	0	0	0	0
Total Waste	kt	3,237	0	222	2,841	174	0	0	0	0	0	0
Strip Ratio	t:t	3.7		7.9	3.9	1.4						
Total Material	kt	4,105	0	250	3,560	295	0	0	0	0	0	0
Supremo												
Total Ore	kt	32,883	0	0	2,360	3,695	4,356	4,647	5,232	5,056	6,230	1,308
Total grade	g/t	1.45	0	0	1.59	1.61	1.70	1.57	1.29	1.28	1.30	1.42
Total Au contain	koz	1,530	0	0	121	192	239	235	217	207	260	60
Total Waste	kt	223,035	0	0	19,254	29,982	35,163	35,015	33,828	34,730	29,715	5,349
Strip Ratio	t:t	6.8			8.2	8.1	8.1	7.5	6.5	6.9	4.8	4.1
Total Material	kt	255,919	0	0	21,614	33,677	39,519	39,662	39,060	39,785	35,945	6,656
Total Open Pits												
Total Ore	kt	46,356	3,579	4,923	6,563	4,463	4,356	4,647	5,232	5,056	6,230	1,308
Total Mined grade	g/t	1.45	1.50	1.61	1.38	1.54	1.70	1.57	1.29	1.28	1.30	1.42
Total Au contain	koz	2,157	172	255	292	220	239	235	217	207	260	60
Total Waste	kt	265,361	15,125	17,601	28,036	30,800	35,163	35,015	33,828	34,730	29,715	5,349
Strip Ratio	t:t	5.7	4.2	3.6	4.3	6.9	8.1	7.5	6.5	6.9	4.8	4.1
Total Material	kt	311,717	18,705	22,524	34,598	35,263	39,519	39,662	39,060	39,785	35,945	6,656
Average Mined	t/day		51,245	61,708	94,790	96,610	108,272	108,662	107,014	109,001	98,480	24,653
Total HL feed	kt	46,356	3,500	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	2,856
HL Head grade	g/t	1.45	1.50	1.61	1.39	1.52	1.65	1.57	1.28	1.30	1.30	1.32
Au contain ounces	koz	2,157	169	258	223	244	266	252	206	209	209	121
Au recovery (mined ore)	%	86.3	86.6	82.0	81.4	86.1	86.2	86.8	87.8	89.6	89.0	89.1
Stockpile Balance												
ROM Stockpile	kt		79	2	1,565	1,027	383	30	262	318	1,548	
ROM grade	g/t		1.17	1.96	1.38	1.39	1.53	1.66	1.48	1.11	1.24	
ROM contain ounces	koz		3	0	69	46	19	2	12	11	62	

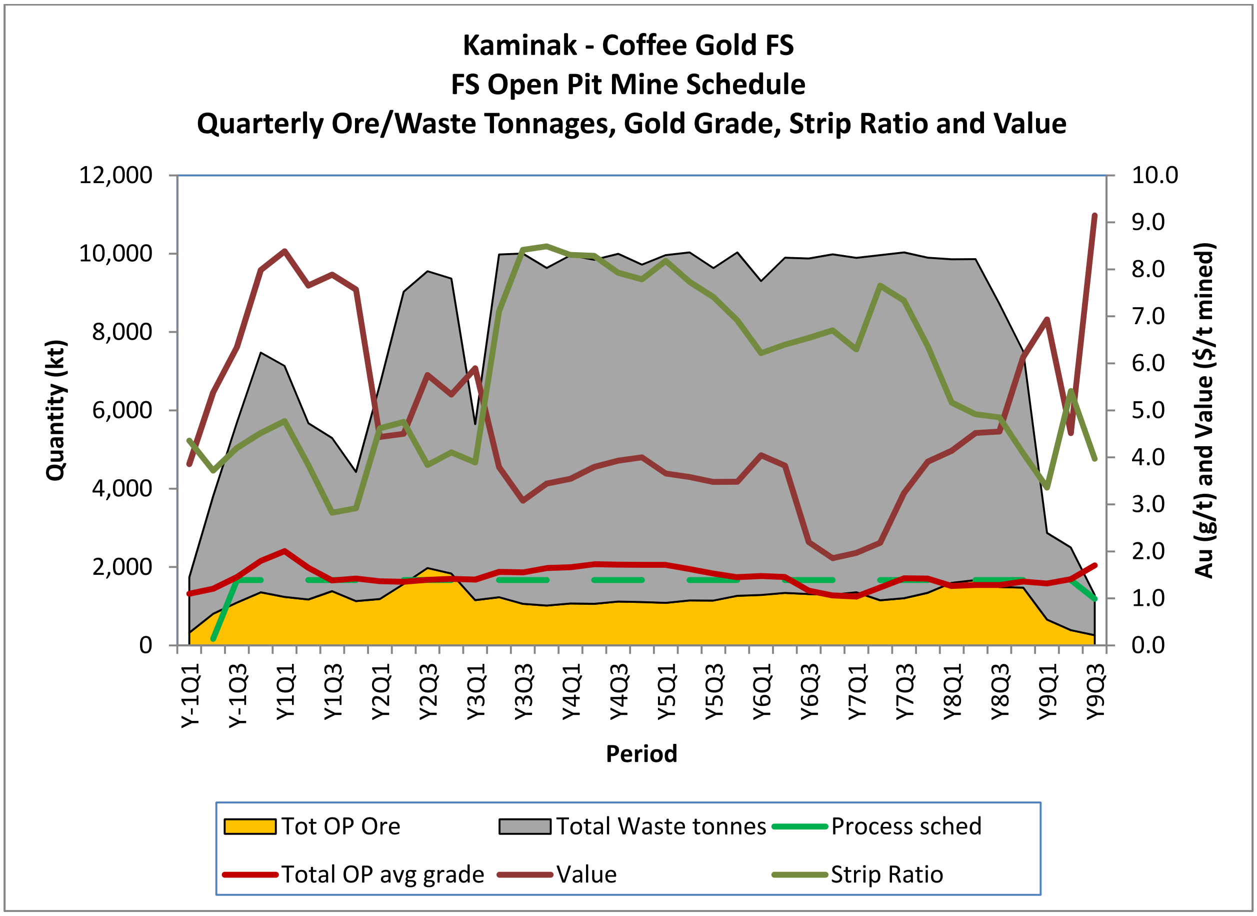
Source: JDS 2016

Figure 16.8: Total Mine Ore and Waste Tonnages, Gold Grade, Strip Ratio and Value



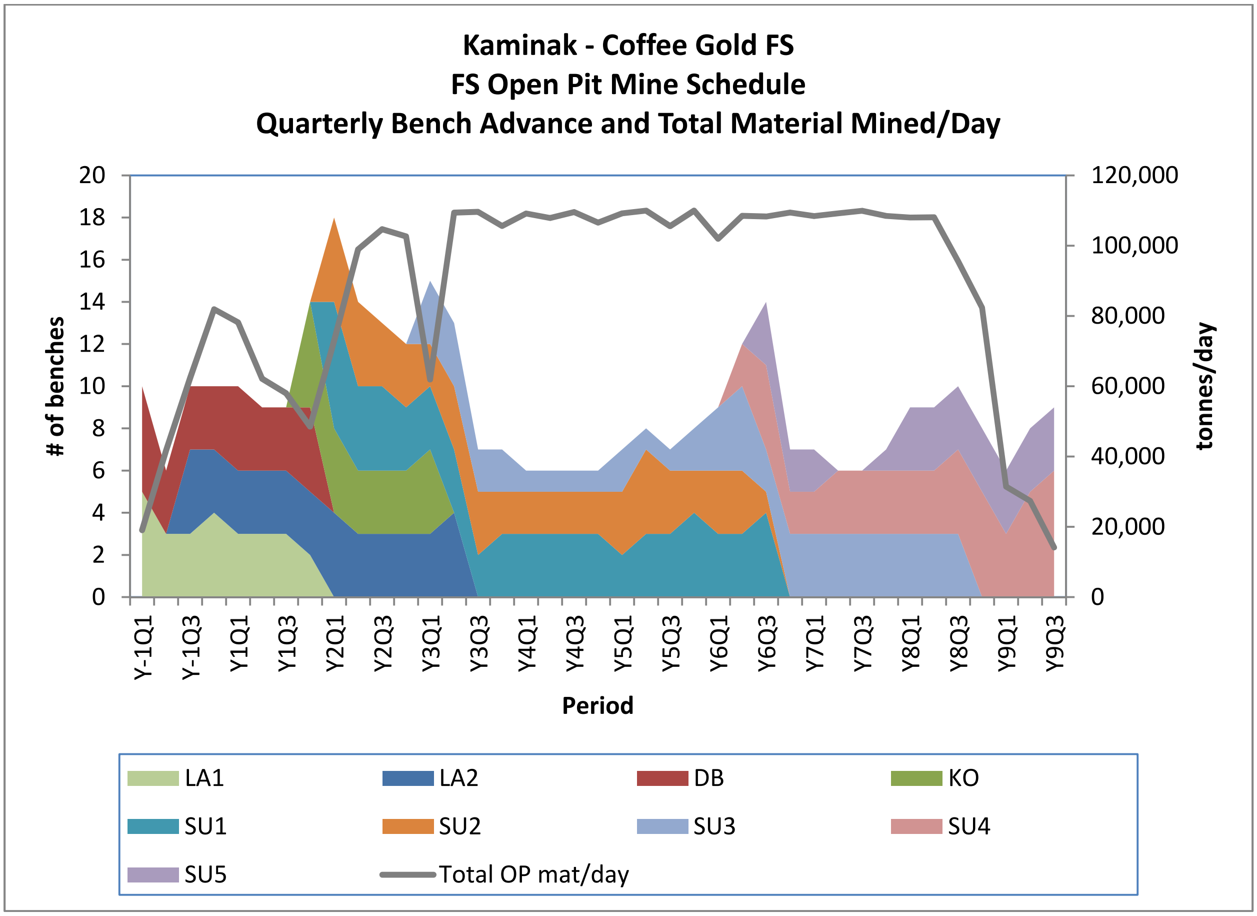
Source: JDS 2015

Figure 16.9: Mine Ore and Waste Tonnages, Grade, Strip Ratio and Value



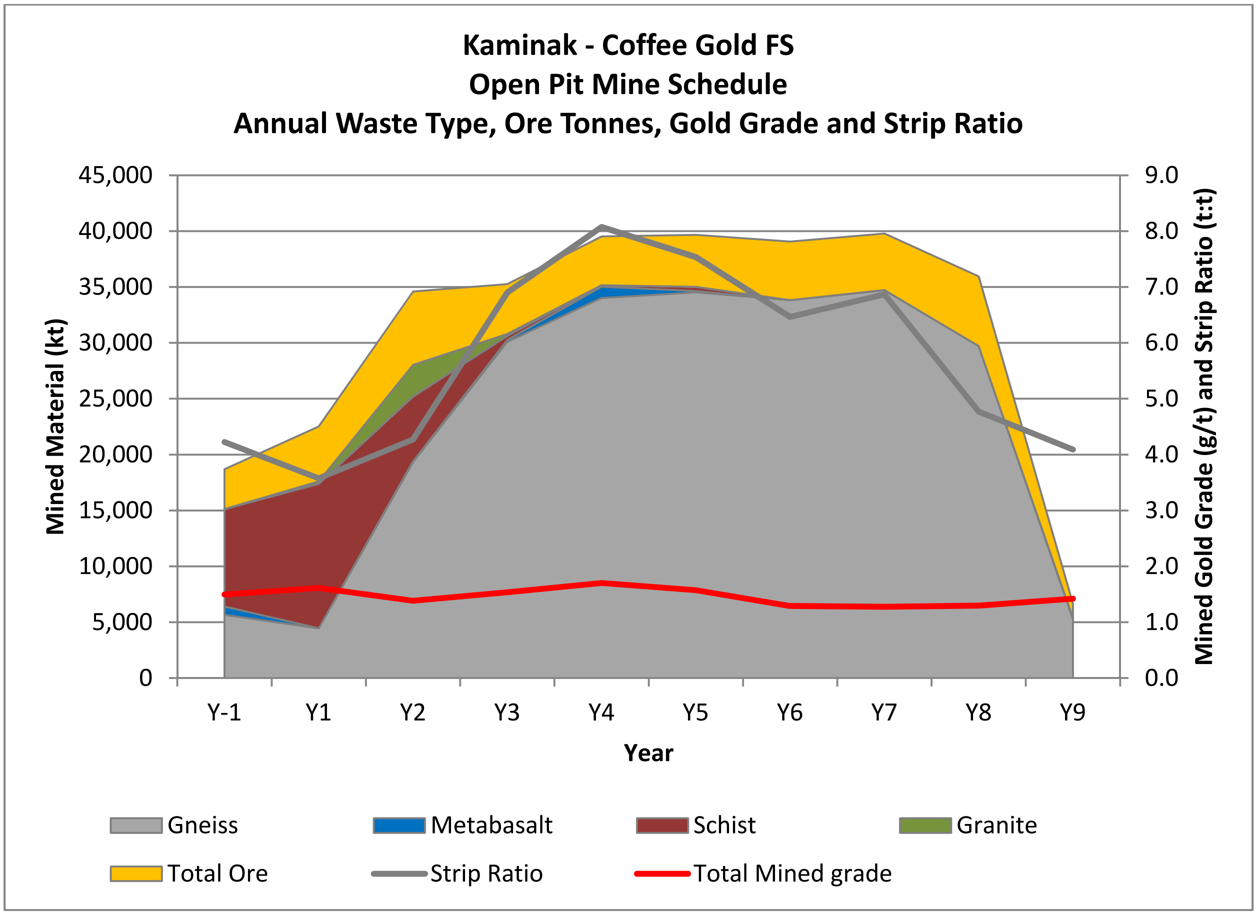
Source: JDS 2015

Figure 16.10: Quarterly Open Pit Bench Advance by Pit/Phase



Source: JDS 2015

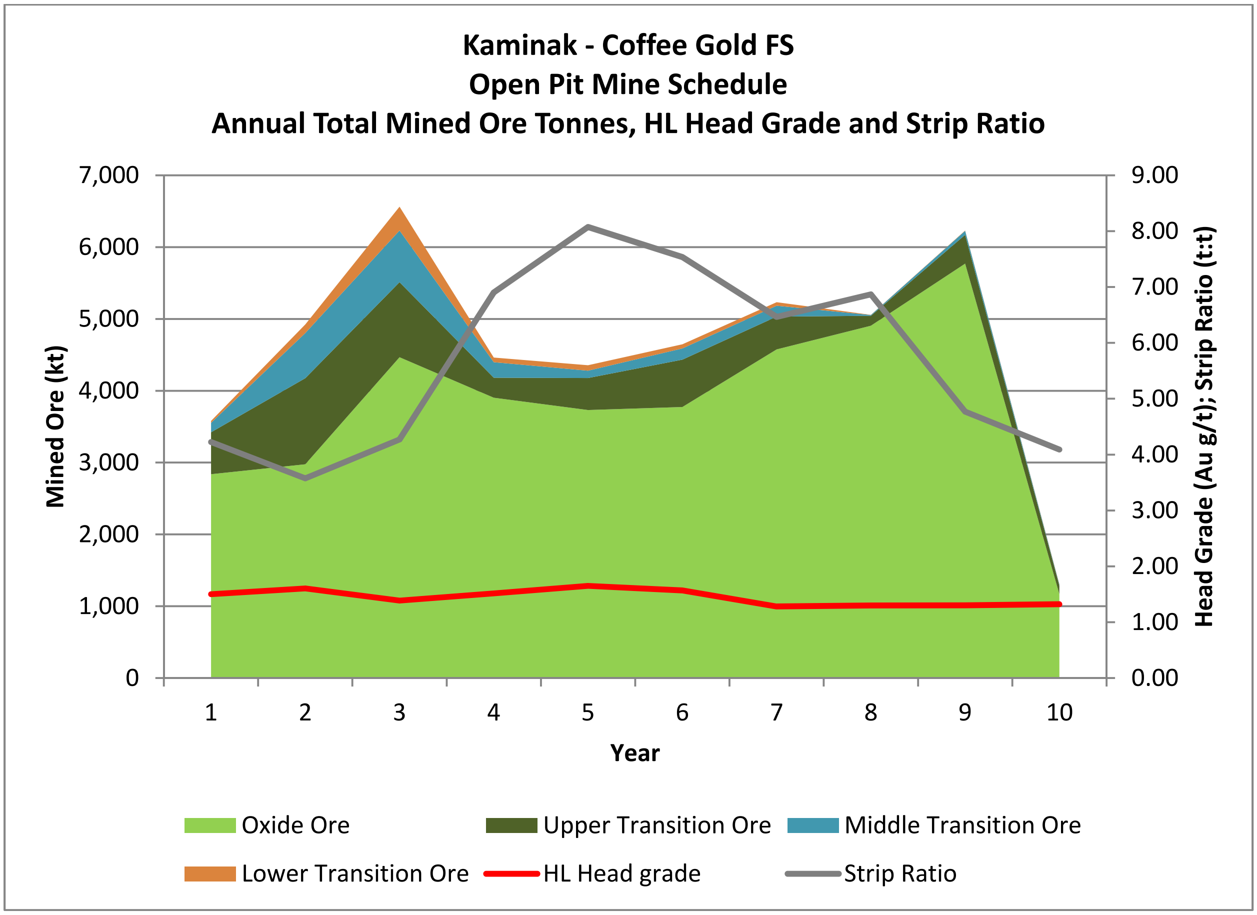
Figure 16.11: Annual Waste Tonnages (by type), Gold Grade and Strip Ratio



Source: JDS 2015

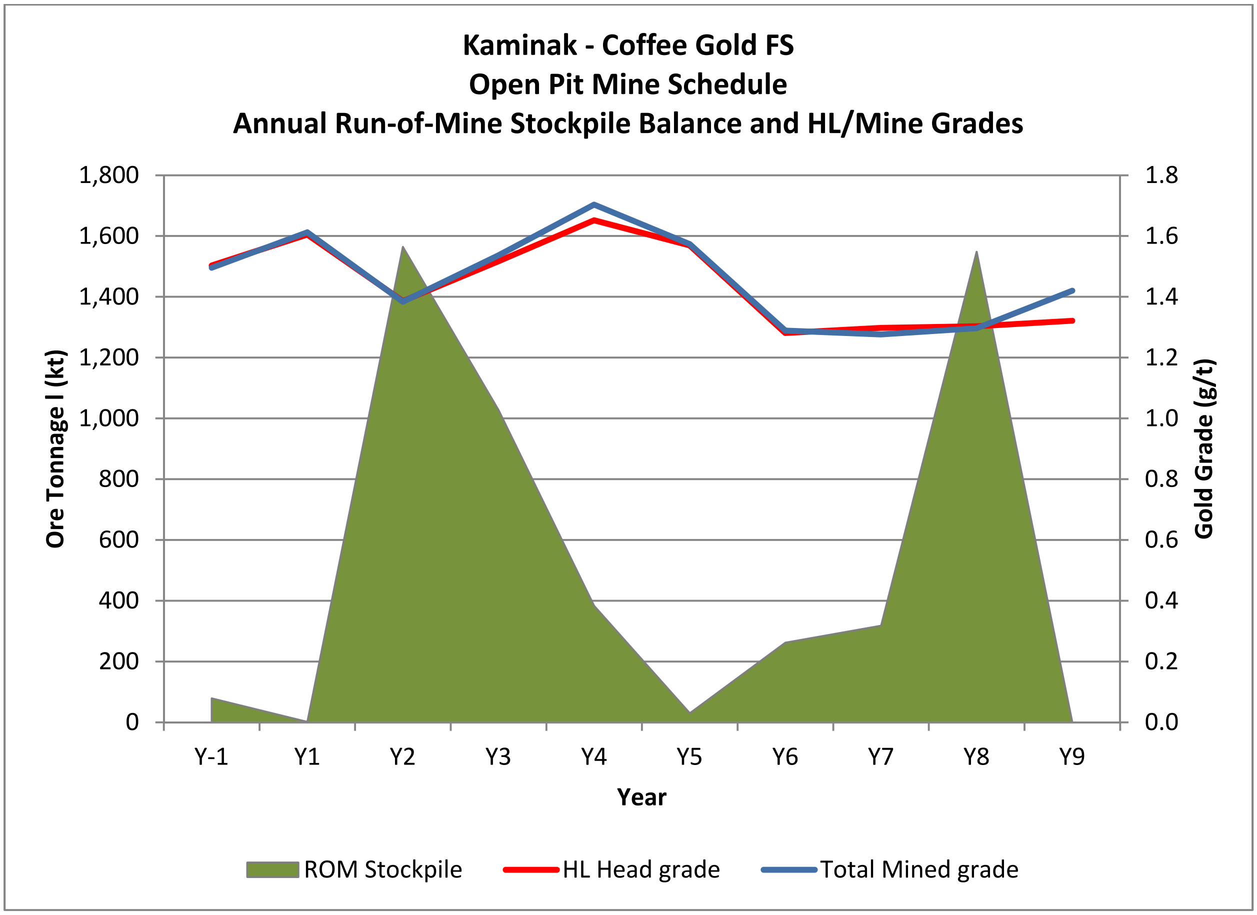
Figure 16.12 summarizes annual mined ore tonnages by oxidation type, while Figure 16.13 illustrates the annual ROM stockpile closing balances. The seasonal nature of the heap leach stacking requirements versus the continuous mining operation results in variable ROM closing stockpile balances.

Figure 16.12: Annual Mined Ore Tonnes by Oxidation Type



Source: JDS 2015

Figure 16.13: ROM Stockpile Balance and Gold Grades

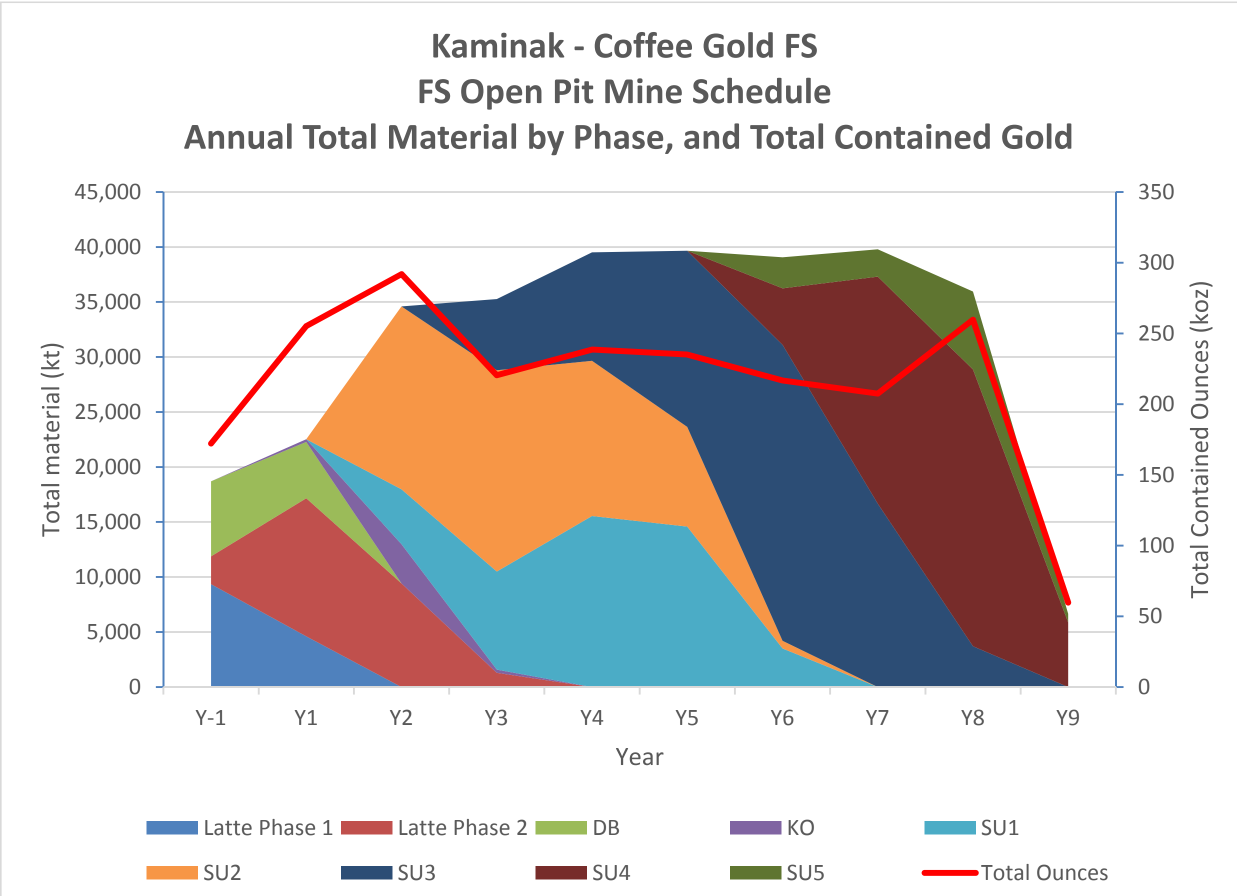


Source: JDS 2015

To further illustrate the progression of mining of the Coffee Gold open pit deposits, Figure 16.14 provides the annual pit and phase progression in terms of total material mined along with total contained ounces.

Figure 16.14 to Figure 16.24 provide layout drawings with the status of the open pit configuration, waste rock facility, as well as heap leach facility advance, at the end of each year.

Figure 16.14: Total Material Mined by Phase



Source: JDS 2015

Open Pit Development

Year -1: This period covers the pre-production period to end of Q3 as well as start of operations in Q4.Open pit mining commences with development of the Latte and Double Double open pits. Suitable waste rock is planned to be used for construction (roads, laydown areas and the heap leach pad). A total of 15.1 Mt of waste mined in this period and by the end of the calendar year, a total of 3.5 Mt of ore will have been delivered to the heap leach pad.

Year 1: First full year of processing. Open pit mining at Latte continues with Double Double pit completed. The Kona pit commences later in the year. A total of 4.9 Mt of ore is scheduled to be mined in the year. Mined gold grade for the year will average 1.6 g/t. Waste rock totaling 17.6 Mt will be produced for a strip ratio of 3.6:1.

Year 2: Mining in Latte and Kona pits will continue and Supremo pit will commence. Average gold head grade over the period is expected to be 1.38 g/t at a production rate of 5.0 Mt/a. A total of 6.6 Mt of heap leach ore feed and 28.0 Mt of waste rock is planned to be mined

Years 3 to 5: Mining at the Latte and Kona pits will be completed in Year 3, with Supremo pit mining continuing over the entire period. Gold head grade is expected to average 1.60 g/t. The waste produced over the three year period is planned to total 101.0 Mt with a total of 13.5 Mt of heap leach ore feed for an average strip ratio of 7.5:1.

Years 6 to 9: All mining occurs in the various phases of the Supremo pit over this 4-year period. A total of 17.8 Mt of heap leach feed will be mined. Gold head grades are estimated to average 1.3 g/t Total waste produced from the open pits is estimated to be 103.6 Mt.

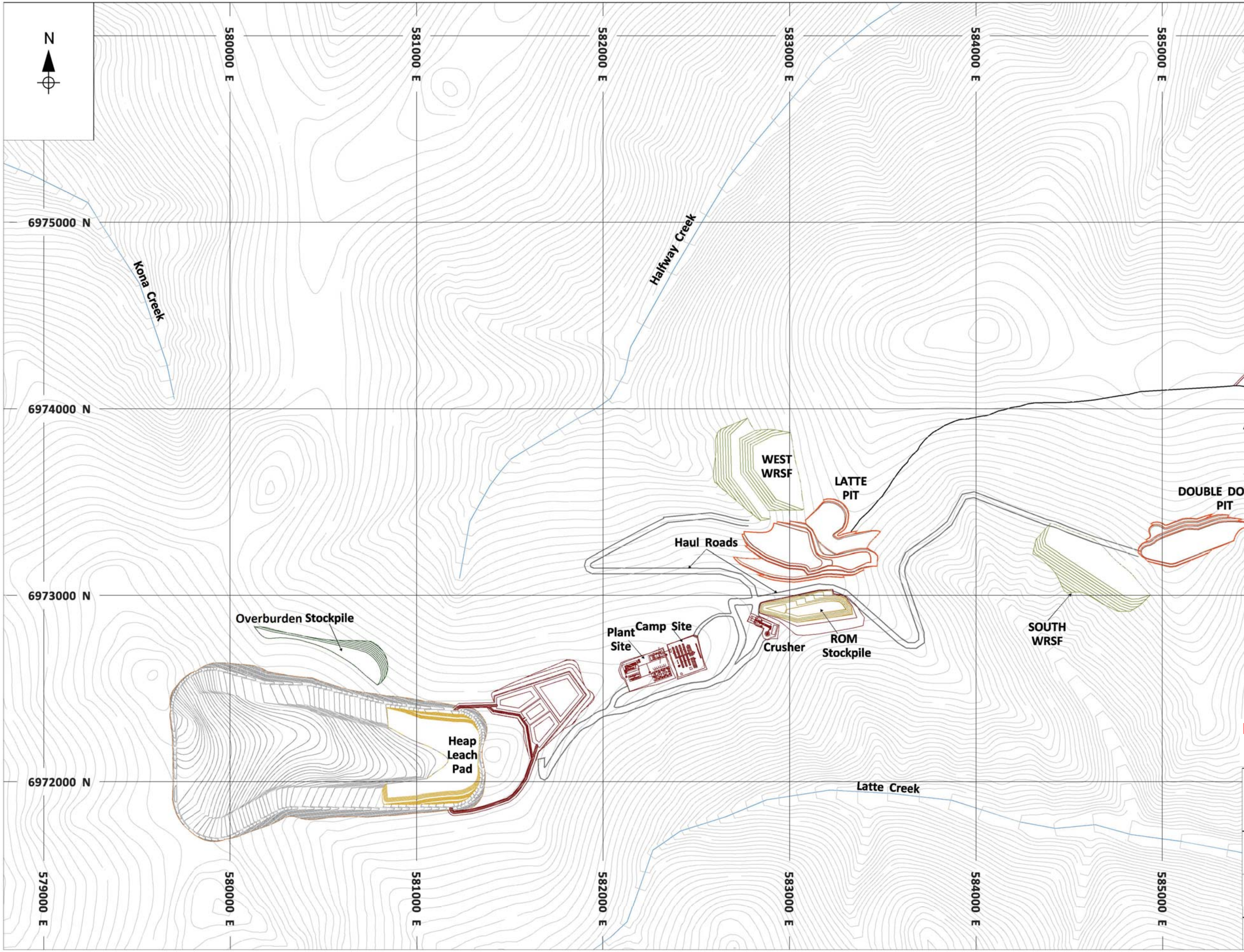


Figure 16.15: Annual Map - Year -1

PROJECT:	Coffee Gold Feasibility Study
TITLE:	Annual Map - End of Year -1

NOTE: Topography contoured on a 10m basis

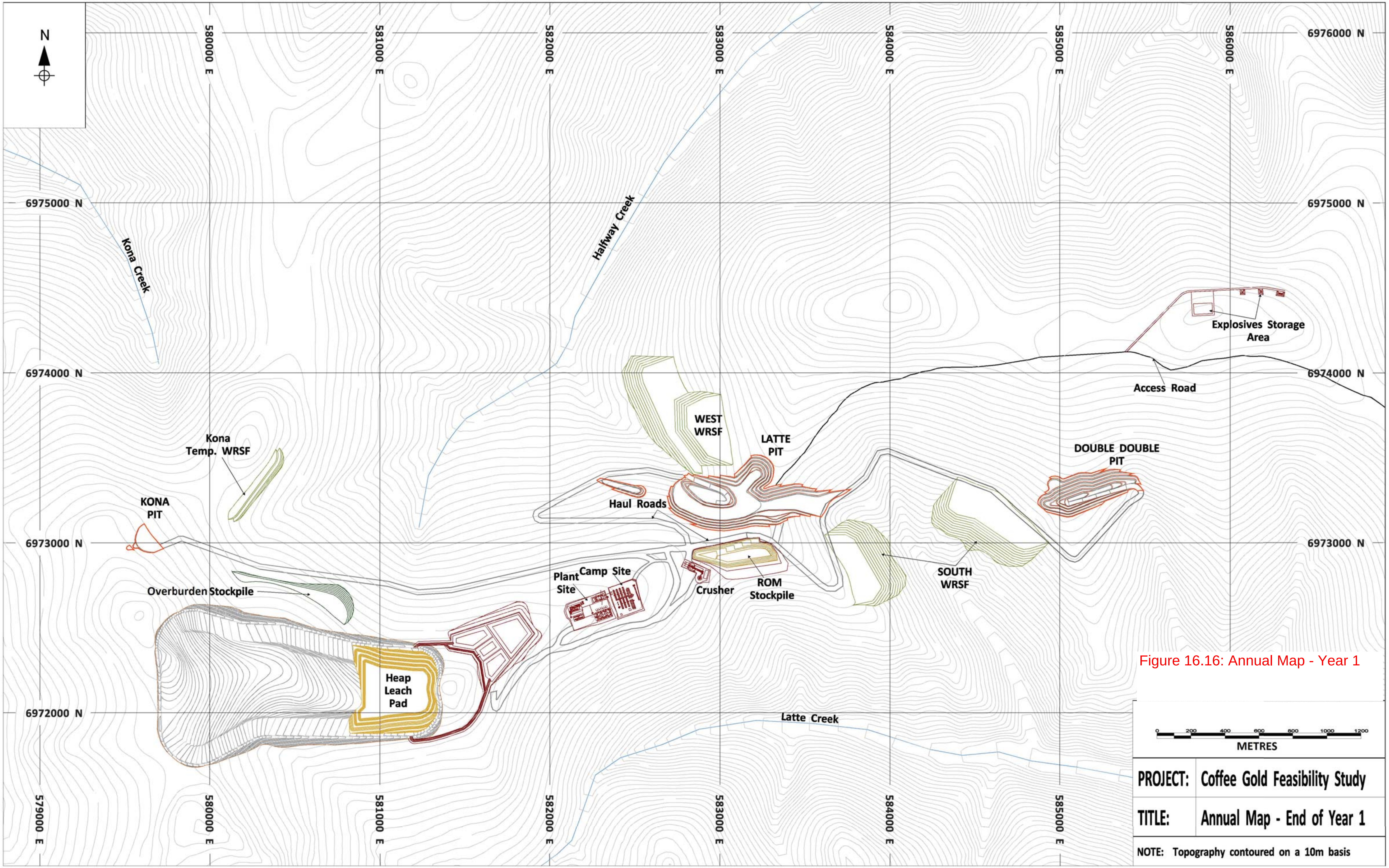


Figure 16.16: Annual Map - Year 1

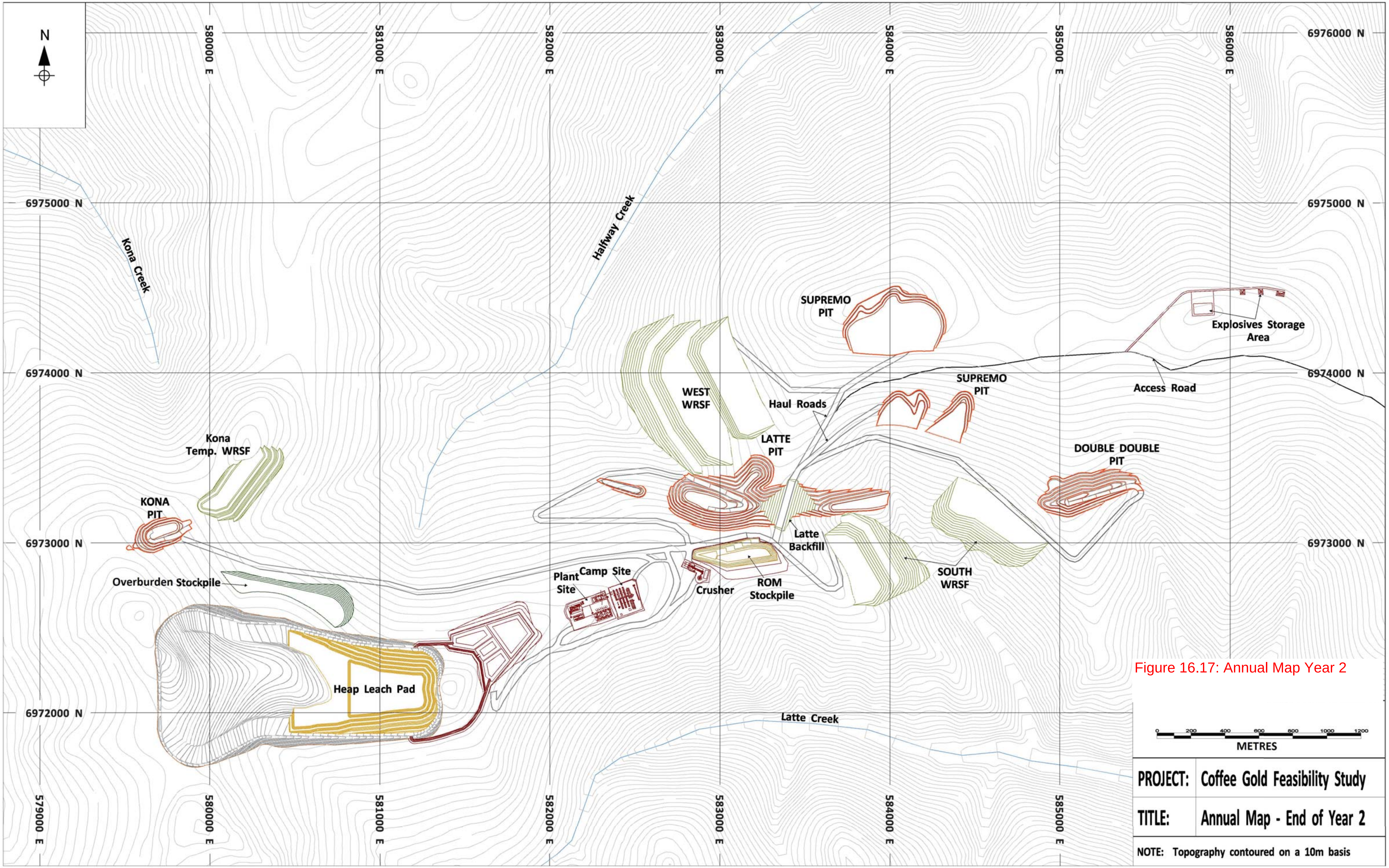


Figure 16.17: Annual Map Year 2

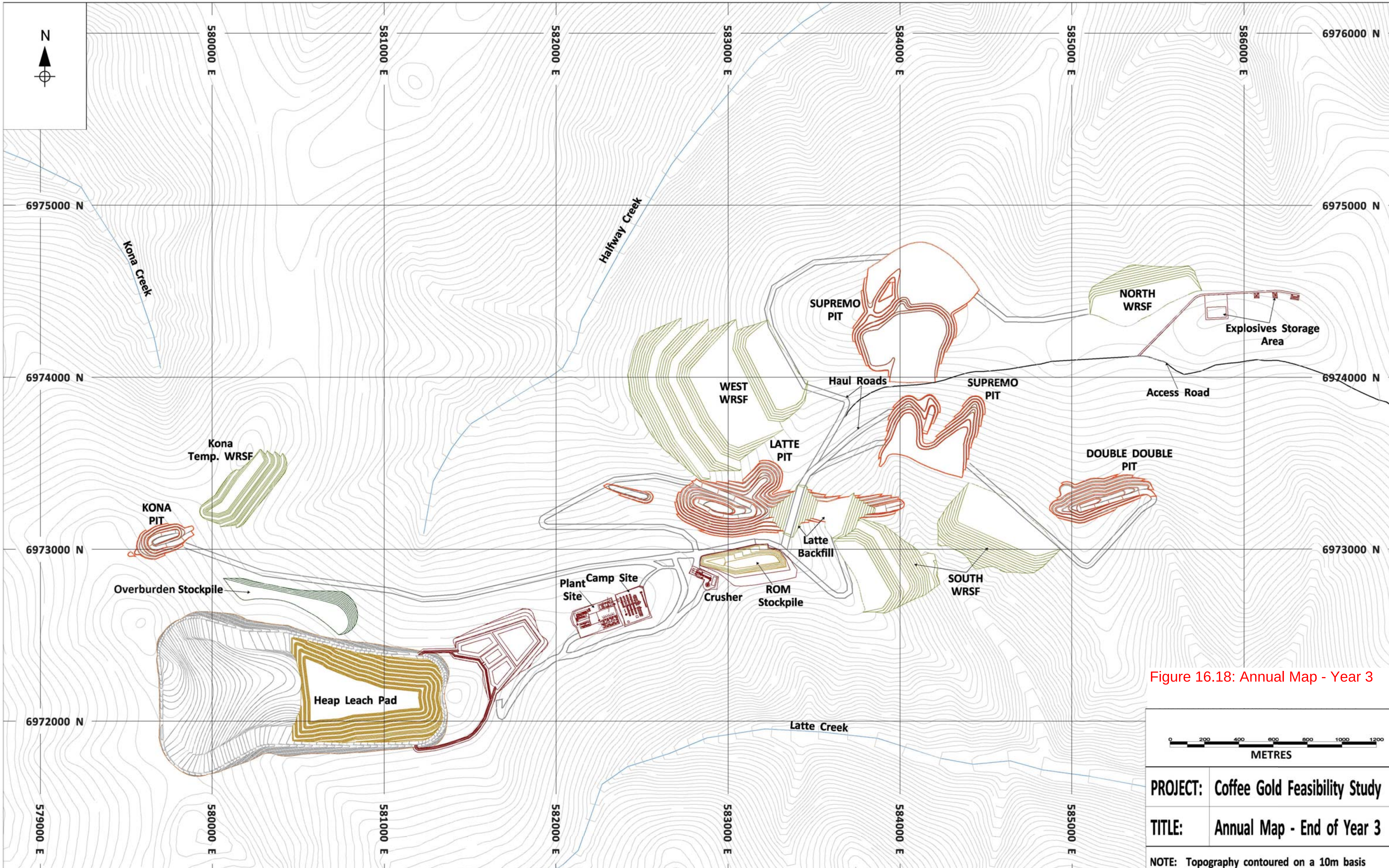


Figure 16.18: Annual Map - Year 3

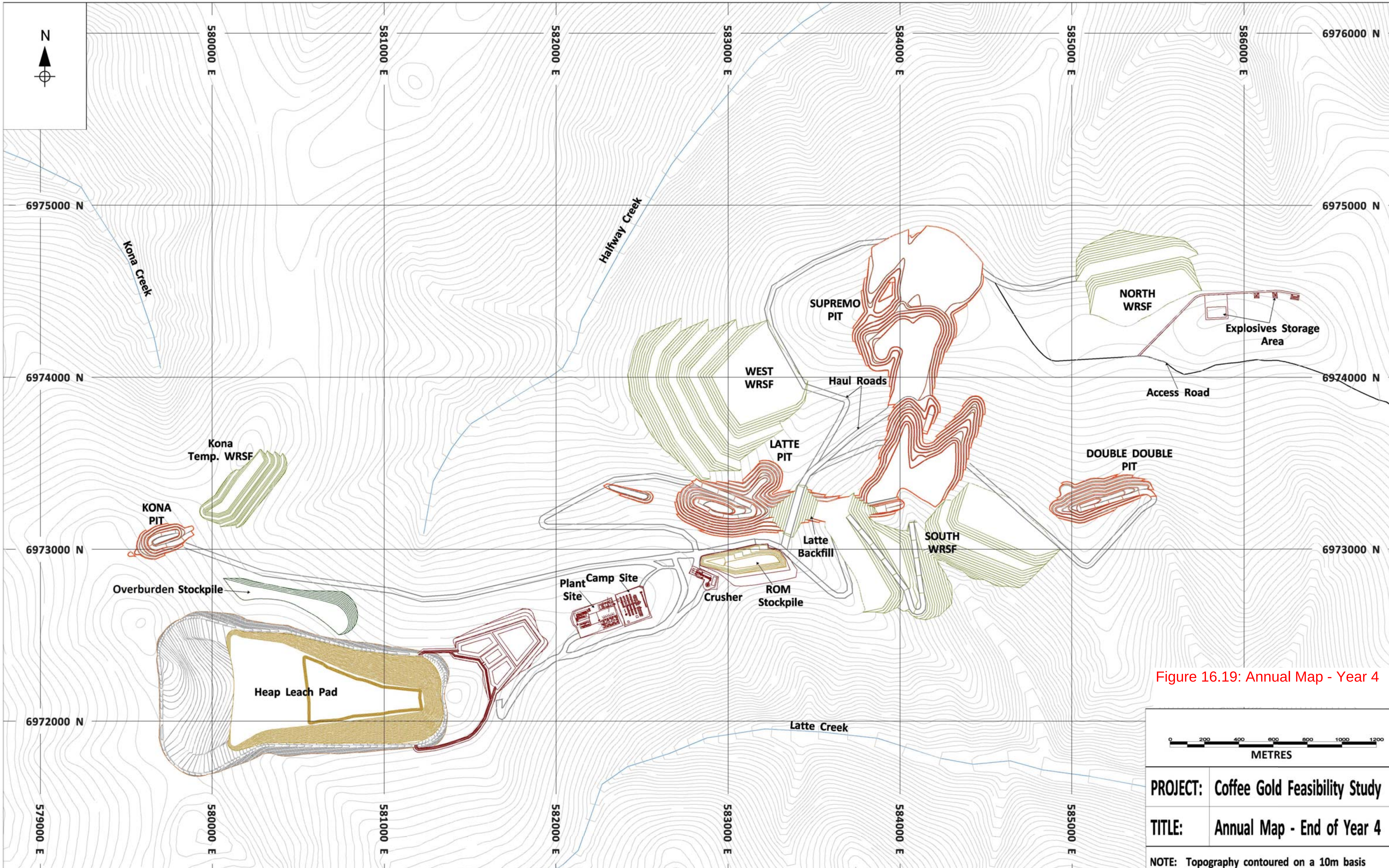


Figure 16.19: Annual Map - Year 4

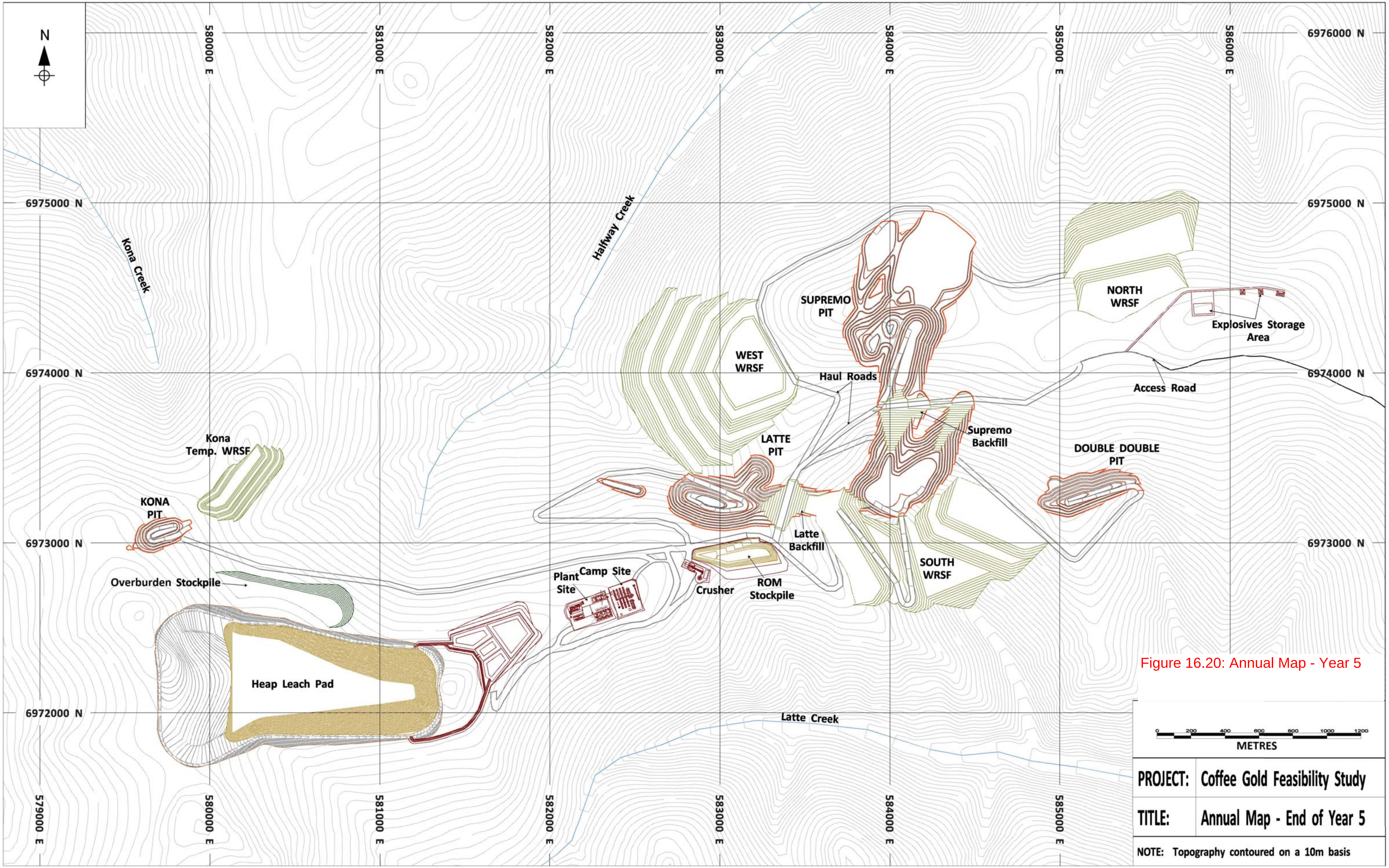


Figure 16.20: Annual Map - Year 5

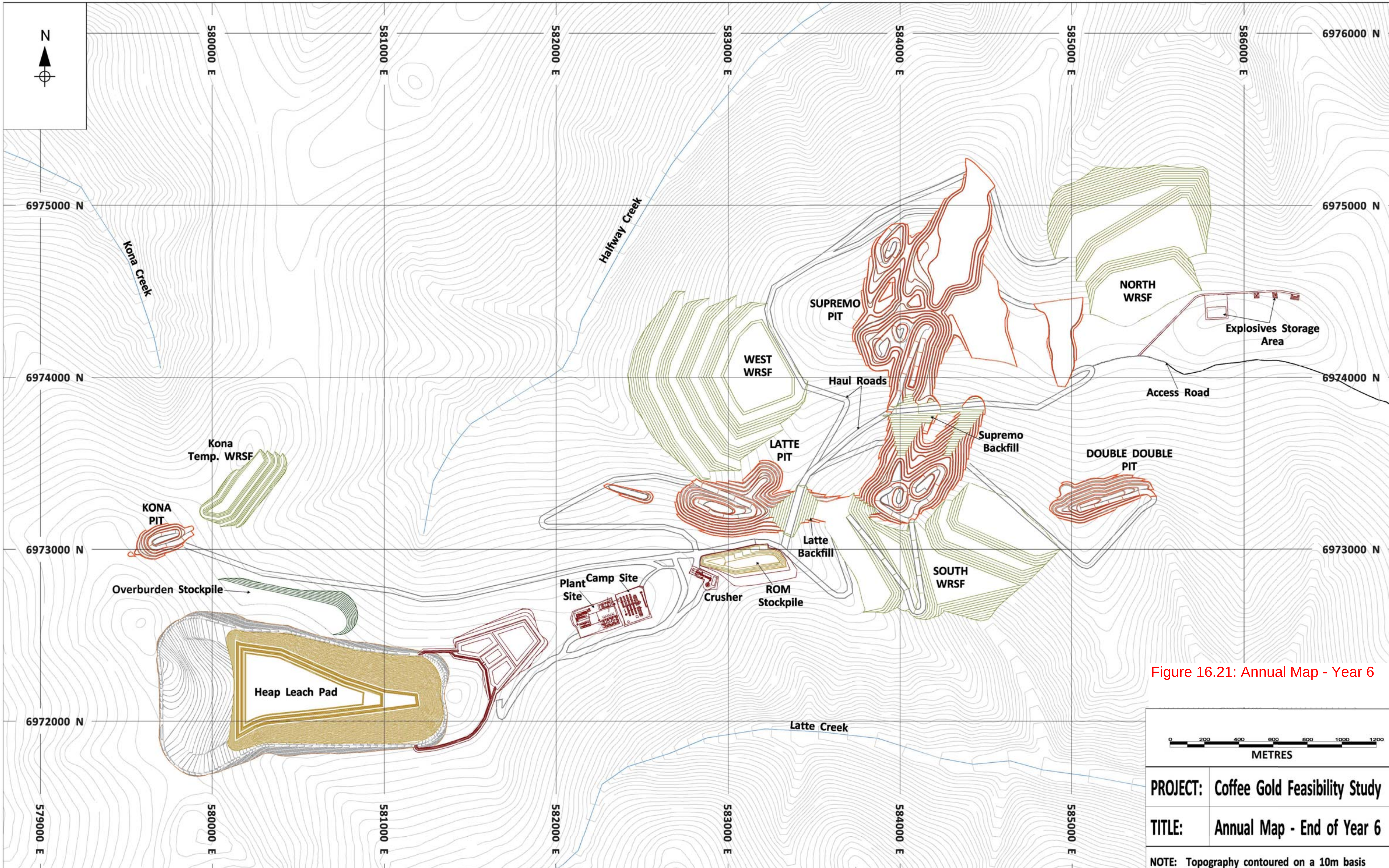


Figure 16.21: Annual Map - Year 6

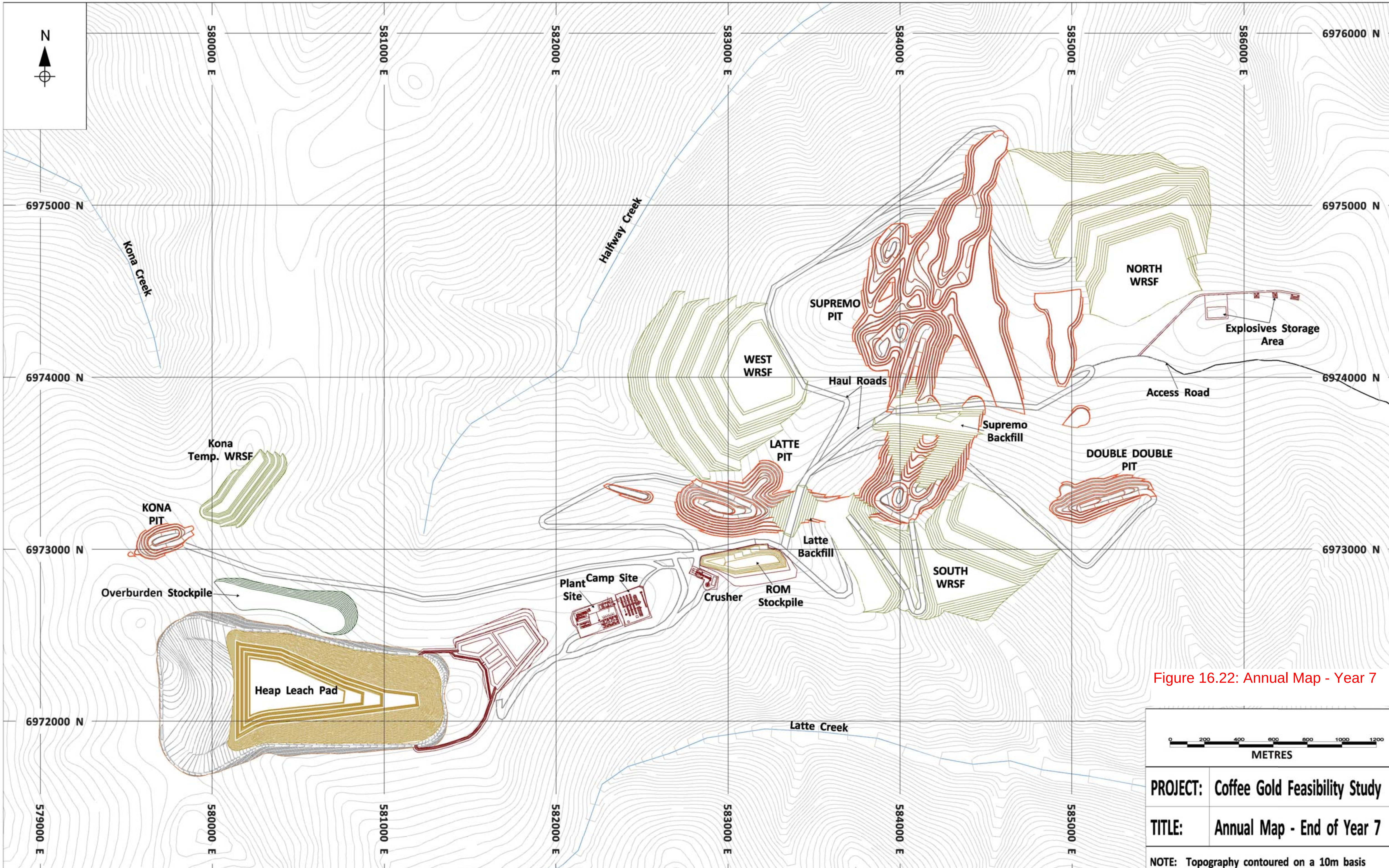


Figure 16.22: Annual Map - Year 7

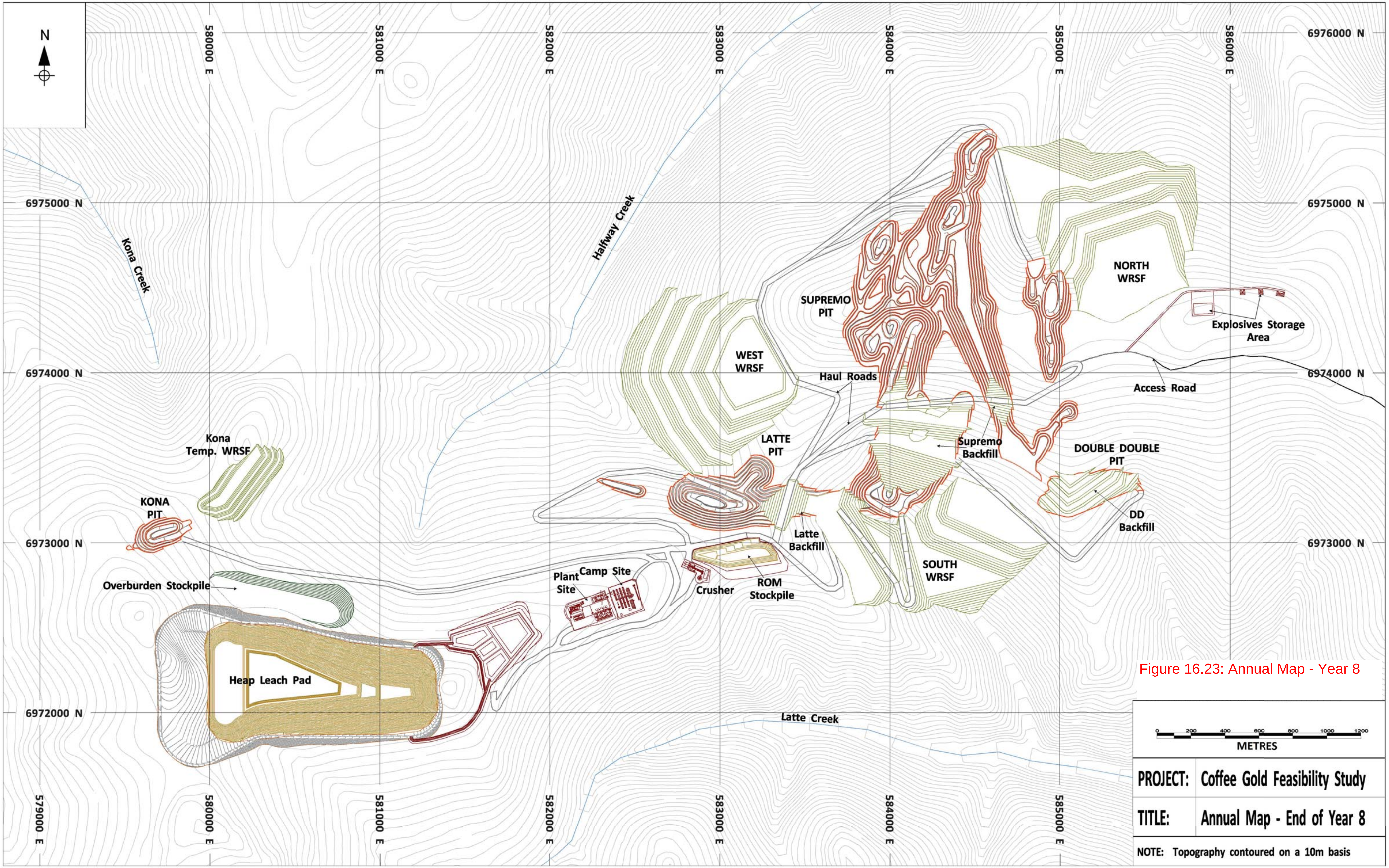


Figure 16.23: Annual Map - Year 8

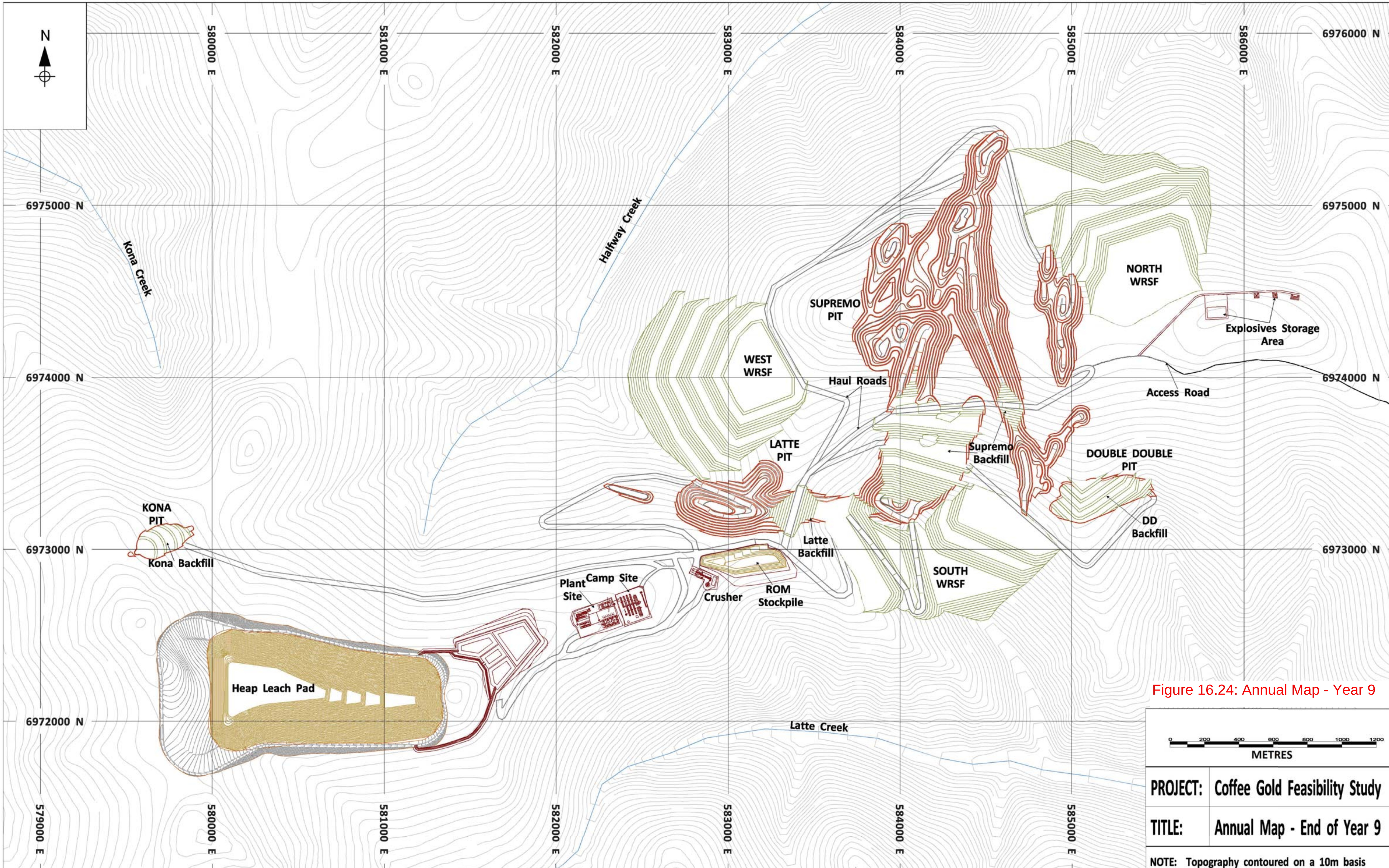


Figure 16.24: Annual Map - Year 9

16.2.6.6 Waste Rock Scheduling and Storage Area Design

16.2.6.6.1 Waste Rock Storage Facility Geotechnical Aspects

Foundation Conditions

Geotechnical investigations have been conducted to characterize the subsoil conditions within the WRSF foundation footprints including sonic core drilling of soils, test pit excavation, diamond core drilling and laboratory test programs (SRK 2016a). The results of these investigations and subsequent analyses were used to provide the basis for geotechnical design recommendations.

The three WRSFs will be constructed in mid to upper slope areas which are generally dominated by in-situ residual soils and colluvium derived from weathering of bedrock. The colluvial material is variable and typically contains mixtures of gravels, sands and silts with a thin veneer of organics in the upper approximately 20 to 30 cm. The depth of soil beneath is typically between 0.5 m and 2 m on the slopes with depths of up to approximately 10 m deep in the valley bottoms. Foundation soils are typically ice-poor. The thickness of the strongly weather bedrock is variable but is generally less than a metre. In general the foundation conditions are considered conducive to construction of the WRSFs as proposed.

Permafrost is deep beneath the North and West WRSFs and non-existent to shallow beneath the South WRSF. The active zone is generally shallow across the site (less than 2 m), except in areas where vegetation has been stripped (i.e. road cuts) (Lorax 2016). Additional details regarding permafrost conditions at Coffee Gold can be found in Section 18.

Geotechnical Design

The North and South WRSFs will be constructed from the top (ultimate crest) downward as bottom-up construction (i.e. hauling to the base of the dump and constructing the dump upwards in lifts) is not considered practical due to the large elevation difference and steep terrain between the pits and the ultimate toe of the WRSFs. The West WRSF will be constructed in a bottom-up sequence.

The WRSFs will be constructed on permafrost soils. Permafrost soils are expected to provide suitable foundation conditions for WRSFs, provided the foundation soils remains frozen. To ensure the foundation remains frozen, the first lift (approximately 5 m) of all new WRSF areas should be placed during the winter season.

In the event that the first lift of waste rock is constructed during the summer months, the WRSFs could be subject to elevated pore water pressures within the active layer resulting in an increased risk of instability.

Prior to construction, vegetation and topsoil (assumed to be 30 cm) within the footprint should be stripped and stockpiled for reclamation purposes. Stripping of materials will be most productive during the fall season after the seasonal thaw has occurred in the active zone. The first lift should then be placed in the coming winter, once the upper soils have refrozen.

All thaw-unstable soils are to be removed from areas of the dump footprint within 15 m of all final and interim dump toes. Thaw-unstable soils are not anticipated to be extensive; however, additional investigations are required at the detailed design level to delineate thaw unstable soils within these areas. It has been assumed that soils and weathered bedrock to a 1 m depth will be removed within 15 m of the final and interim phase toes.

Individual lifts of waste rock should not exceed 50 m in height except for the first lifts of the North and South WRSFs where thicker bottom lifts will be required due to the steep terrain and valley inverts. Approximately 56 m wide berms should be left between each lift resulting in an overall slope angle of 2.5:1 (H:V).

Flow-through rock drains are planned beneath the WRSFs. Additional details regarding rock drains and water management can be found in Section 18.4.

16.2.6.7 Waste Rock Scheduling

Most waste rock from the open pits is planned to be deposited in various engineered WRSF adjacent and near to the pits from which the waste is sourced. Some waste rock will be backfilled into mined out pits at Latte, Supremo and Double Double in order to create causeways. These shorten ore haulage routes to the crusher compared to having to haul material around the pits.

Three external WRSFs will be constructed; West, North and South. Total waste material removed from the pits, including that used for construction and placed as backfill totals 265 Mt. Suitable NPAG waste rock will be used for road construction, lay down areas, as well as the base of the heap leach pad

Geochemical characterization indicates that the exposed pit walls at the Kona pit are potentially acid generating. As such, the waste rock from the Kona pit will be stored in a temporary waste rock facility adjacent to the pit during mining and then backfilled into the mined out pit.

Each WRSF is planned to be constructed by placing material at its natural angle of repose (approximately 1.5H:1V) with 50 m wide safety berms spaced approximately every 50 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 WRSF designs.

The topsoil and subsoil that will be removed from the opens pit, heap leach pads and WRSF footprints will be placed in a temporary stockpile located immediately north of the heap leach pad.. Based on field recommendations from SRK, the top 0.3 m of material will be removed. A waste rock berm will be constructed on the downhill slope of this stockpile.

The West WRSF is planned to have an ultimate crest elevation of 1,210 masl (maximum 250 m dump height) and will be located to the north of the Latte pit and west of the Supremo pit. The WRSF is designed to be built from the bottom up by placing material at its natural angle of repose in 50 m lifts. The toe of each subsequent lift would be set back 50 m from the crest of the previous lift, resulting in an overall angle of 2.5:1V. This WRSF has a planned design capacity of 69 Mt.

The South WRSF is planned to be located southeast of the proposed Latte pit and south of Supremo pit and will have an ultimate crest elevation of 1,050 masl (approximately a 200 m maximum dump height) and a design capacity of 50 Mt.

Given the location of the WRSF it will need to be built in a series of 40 to 60 m lifts in a top-down approach. An access ramp along the western limit is planned to be incorporated into the design as the WRSF advances. The toe of each bench is planned to be set back 50 m from the crest of the previous bench, resulting in an overall angle of 2.5H:1V.

The North WRSF is immediately east of the Supremo final pit limits. This WRSF would be built in a series of 50 m wrap around benches in a similar fashion to the South WRSF. The final crest elevation for the WRSF is designed to be 1,250 masl with a design capacity of 97 Mt and a final overall face angle of 2.5H:1V.

The temporary waste storage facility for Kona pit is designed to be built in a similar fashion to the above WRSFs, but once mining is complete the waste material placed in this temporary storage facility would be backfilled into the Kona pit.

Table 16.12 summarizes waste tonnages allocated to the various WRSFs by year for the LOM production schedule. Figure 16.25 illustrates the location and ultimate designs of these proposed waste rock facilities.

Table 16.12: Annual Waste Allocations by Destination (in Mt)

	Year									
Destination	**Y- 1**	**Y1**	**Y2**	**Y3**	**Y4**	**Y5**	**Y6**	**Y7**	**Y8**	**Y9**
North WRSF	6.4	11.3		6.1	9.1	14.4	30.7	27.8	8.2	0.5
South WRSF	8.7	6.1	3.1	4.8	13.7	9.0	1.6			
West WRSF		0.2	18.2	16.6	12.4	7.5				
Kona WRSF			2.8	0.2						
In-pit Backfill	**15.1**	**17.6**	3.9	3.1		4.1	1.6	6.9	21.5	4.8
TOTAL	6.4	11.3	**28.0**	**30.8**	**35.2**	**35.0**	**33.9**	**34.7**	**29.7**	**5.3**

Source: JDS 2015

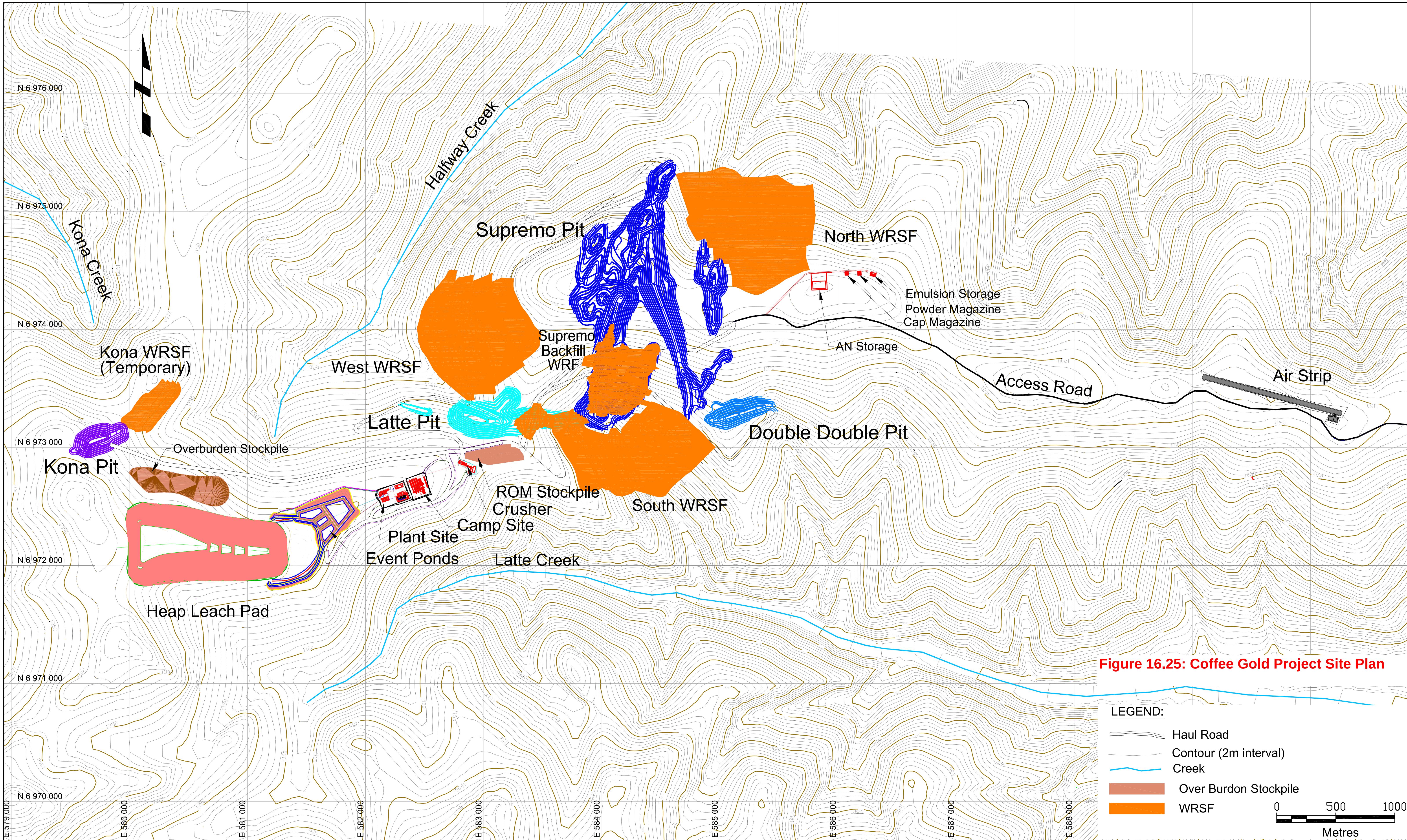


Figure 16.25: Coffee Gold Project Site Plan

16.2.7 Mine Equipment Selection

16.2.7.1 Introduction

The open pit mining activities for the Coffee Gold Project were assumed to be undertaken by an owner-operated fleet. The equipment was selected for a standard open pit mining operation with conventional drill, blast, load and haul, considering bulk excavation of waste using hydraulic excavators, and bulk-selective loading of ore using a front-end loader or smaller hydraulic back hoe. 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.

Mining equipment types mentioned in the following chapters are for reference purposes only. The equipment referred to was used for estimation purposes, and was not based on a competitive selection process.

16.2.7.2 General Operating Parameters

The open pits are designed with 5 m benches in both waste and ore headings with adequate phase geometry to achieve a maximum production rate of 39 Mt/year. The design calls for annual heap leach ore loading rate of 5.0 Mt. Mining is scheduled to advance sequentially through the pits, and in the early years of the mine life, up to two pits active at any time. Given the required production rate and pit geometries, vertical advance rates average three benches per quarter, with frequent requirement for ramp development and opening of new benches.

Time definitions, work regime structure, and standard standby and delay parameters were applied to the mine equipment selection.

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, 59% for the hauling equipment, and 65% for support and auxiliary equipment.

16.2.7.3 Blasthole Drilling and Blasting

Based on the selected bench height (drilling will occur on 10 m high benches) and the production schedule requirements, a 229 mm hole diameter production drill was selected for waste and a 152 mm hole diameter production drill was selected for ore. Drill pattern details are shown in Table 16.13.

Table 16.13: Drilling Parameters

Item	Unit	Waste	Ore
Diameter	mm	229	152
Dry density (in-situ)	t/m^3	2.6	2.5
Drill bench height	m	10	10
Burden	m	5.3	3.5
Spacing	m	6.6	4.4
Sub-drill	m	1.6	1.1
Total hole length	m	11.6	11.1
Stemming	m	3.7	2.5
Tonnes/hole	t	895	381
Drilling factor	t/m	77	35
Penetration rate	m/hr	14	24

Source: JDS 2015

To achieve the recommended bench face angles (BFA) and inter-ramp angles (IRA) within the stronger rock mass units, 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 (76 mm) drill.

Based on these parameters, annual production capacity was estimated for each type of drill for each period of the mine plan. The blast design assumes the use of ANFO as the main explosive and, given the relatively dry climatic conditions of the project area, 15% emulsion usage was contemplated. Table 16.14 shows the planned blasting parameters for both ore and waste (assuming dry loading conditions).

Table 16.14: Blasting Parameters

Item	Unit	Waste	Ore
Column Charge	m	7.9	8.6
Column Charge	kg	288	139
Powder factor	kg/t	0.32	0.37

Source: JDS 2015

16.2.7.4 Loading

Diesel hydraulic excavators were selected as the primary loading equipment, supported by front-end loaders (FEL) and smaller hydraulic backhoes. The main criterion for loading equipment selection is the ability to load trucks with payloads of 144 t, while allowing for selective mining. The hydraulic front shovels will primarily undertake the mining of ore and waste material, while the FELs and smaller excavators will complement the main shovel fleet (e.g. lower, confined benches of the open pits). The performance of the primary loading units was calculated on the basis of the operational equipment productivities and the truck loading times for both ore and waste material.

The number of passes and fill factors are summarized in Table 16.15. In addition to the loading time, the loading unit productivities include waiting, maneuver and unproductive time estimates. Based on these parameters, annual production capacity was estimated for each type of loading unit for each period of the mine plan.

Table 16.15: Loading Parameters

Item	Unit	Value
Dry density (in-situ)	t/m^3	2.6
Material Swell Factor	%	40
Hydraulic Excavator		
Bucket Size	m^3	15
Bucket Fill Factor	%	95
Size of truck to load	t	144
Avg. buckets to load	#	5.5
Avg. bucket cycle time	sec	40
Avg. spot time	sec	30
Total time to load	min	3.7
Front-end Wheel Loader		
Bucket Size	m^3	11.5
Bucket Fill Factor	%	90
Size of truck to load	t	144
Avg. buckets to load	#	7.6
Avg. bucket cycle time	sec	60
Avg. spot time	sec	30
Total Time to Load	min	7.3
Hydraulic Excavator		
Bucket Size	m^3	4.5
Bucket Fill Factor	%	95
Size of Truck to Load	t	144
Avg. Buckets to Load	#	18
Avg. Bucket Cycle Time	sec	40
Avg. Spot Time	sec	30
Total Time to Load	**min**	**12.1**

Source: JDS 2015

16.2.7.5 Hauling

The truck fleet for the Project was selected to match the loading fleet selected, 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). Separate values were calculated for haulage within the pits (from the bench to the pit exits) and outside of the pit (between the pit exit and the final destination – primary crusher/stockpile or waste dumps). Requirements for haulage of crushed ore to the heap leach pads were also accounted for. The distances were differentiated on the basis of ramp and horizontal haulage.

Runge Talpac software was used to determine truck requirements and productivities. Table 16.16 summarizes the haul cycle parameters used in calculating truck productivities. Truck performance was calculated for every loading unit and period of the mine plan. It reflects travel time and other fixed times of the load/haul/dump cycle.

Table 16.16: Haulage Cycle Parameters

Description	Unit	Value
Rated payload	tonnes	144
Dump time at crusher/stockpile	min/load	1.5
Dump time at waste dump	min/load	1
Stopped time (non-hauling)	% of Net operating hour	10
Overall Effective utilization	%	59

Source: JDS 2015

16.2.7.6 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 waste dumps 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 one track dozer (Komatsu D275-class), one dozer (Komatsu WD500-class), one grader (Komatsu GD825-class) and one water truck (75 m^3).

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

- One 76 mm drill for secondary blasting and pre-split drilling;
- 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;
- Tire manipulator; and
- Mobile lights for lighting of pits, waste dumps and construction areas.

16.2.8 Use-of-Time Definitions and Work Schedules

16.2.8.1 Time Model

The time model used for calculating open pit equipment hours is shown in Table 16.17.

Table 16.17: Time Model Structure

<table>
<tr><td colspan="4">Total Available Hours</td></tr>
<tr><td colspan="3">Available Hours</td><td>Maintenance</td></tr>
<tr><td colspan="2">Operational Hours</td><td>Standby</td><td></td></tr>
<tr><td>Effective Hours</td><td>Operational Losses</td><td colspan="2"></td></tr>
</table>

16.2.8.2 Definitions

The definitions used in the time model are:

- Total available hours;
    - Hours in a calendar year. At 24 h/d and 365 d/a, the total available hours is 8,760 /a;
- Available hours;
    - Total available hours less maintenance hours per piece of equipment;
- Maintenance hours;
    - Includes waiting for maintenance personnel, waiting for maintenance equipment or spare parts, travel time to and from the shop, actual maintenance time, movements and waiting time within the shop;
- Standby hour
    - The unit is mechanically operable but is not manned or used (e.g. schedule loss, safety meetings, meals, breaks, blasting, shift change, weather outages, refueling, and training);
- Operational hours
    - Available hours less standby time; used for costing purposes;
- Operational loss hours
    - The equipment is operating but not performing its specific production duty (drill rig setup, shovel/drill moves, cleaning of work faces);
- Effective hours
    - Operational hours less standby time;

- The criteria for lost time have been applied through the following factors:
  - Mechanical availability - (measure of maintenance down time), is expressed as available hours divided by total available hours. For Year 5 and beyond a reduction in mechanical availability of 5% has been applied.
  - Use of availability – operational hours divided by available hours;
  - Operating efficiency – effective hours divided by operational hours; and
  - Overall effective utilization – product of mechanical availability, utilization, operator efficiency and operational losses.

On this basis, the target equipment availability and utilization were defined for each of the major equipment units shown in Table 16.18.

Table 16.18: Availability, Target Use of Availability and Effective Utilization of Major Equipment

Open Pit Equipment	Mechanical Availability (>Year 5) (%)	Use of Availability (%)	Effective Utilization (%)
229 mm dia. Rotary, Crawler Drill	85	90	65
Diesel, 15 m^3 Front Shovel	85	96	57
Diesel, 11.5 m^3 Wheel Loader	85	96	57
144-tonne Haul Truck	85	96	59

Source:

16.2.8.3 Work Regime Detail

Senior supervisory personnel are planned to be on a 2-week in/2-week out shift rotation. Production, maintenance and technical personnel (comprising a total of four crews based on a 12-hour shift schedule) will be on a 2-week in/2-week out rotation. The standard shift is 12 hours.

16.2.9 Mine Equipment Requirements

16.2.9.1 Summary

Information in this study is based on material and data provided by a number of equipment manufacturers. These models referred to are for estimation purposes only and do not represent a competitive selection process.

An annual summary of the open pit fleet requirement is shown in Table 16.19.

Table 16.19: Open Pit Mine Primary Equipment Requirements

Type	TOTAL UNITS	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
D50KS Drill (152 – 229 mm)	7	4	5	7	7	7	7	7	7	7	3
DX800 Drill (76 mm)	1	1	1	1	1	1	1	1	1	1	1
Komatsu HD1500 Truck (144t)	20	10	12	18	16	15	20	18	20	19	10
Komatsu PC3000 Shovel (15m3)	3	2	2	3	3	3	3	3	3	3	3
Komatsu PC800 Excavators (4.5m3)	2	1	2	2	2	2	2	2	2	2	2
Komatsu WA900 Wheel Loader (11.5m3)	3	2	2	3	3	3	3	3	3	3	3
Komatsu D275 Track Dozers	5	4	4	5	5	5	5	5	5	5	3
Komatsu WD500 Wheel Dozer	2	1	2	2	2	2	2	2	2	2	1
Komatsu GD825 Grader	3	2	2	3	3	3	3	3	3	3	1
Water Truck (75m3)	2	1	1	2	2	2	2	2	2	2	1
Total Primary Equipment	48	28	33	46	44	43	48	46	48	47	28

Source: JDS 2016

16.2.9.2 Drilling

Two types of blast hole drilling are envisioned for the open pit mines:

- Blast pattern drilling to allow for fragmenting of rock for mining; and
- Wall stability control and secondary drilling, as required.

Production drilling is planned to be performed on 10 m benches (selective mining on 5 m benches is planned) with drill holes of 229 mm diameter. A typical pattern is described in Section 16.2.7.3 along with an explanation of the use of pre-shear or wall control drilling (79 mm diameter).

Penetration rates were calculated using industry-standard formulas based on estimated rock hardness (ore and waste material) and drill hole diameter. The average penetration rates applied for the 229 mm diameter drill are 14 m per operating hour.

The required operating hours for the drills were calculated from the scheduled mining volumes (taking into account material splits), drill pattern size and penetration rates. The annual drill requirements are shown in Table 16.20.

16.2.9.2.1 Grade Control

Consideration has been given to what type of grade control procedures would be effective for successfully defining ore feed for the open pit operation that is proposed at Coffee Gold to supply 5.0 Mt/a to the heap leach facility.

Gold mineralization at Coffee Gold is located within a series of steeply dipping structures that cross-cut all rock units on the property. A series of “mineralized” or “mineral” domains in each resource area have been interpreted using a combination of surface mapping, geologic core (and reverse circulation chips) logging, and the distribution of gold grades in drill hole sample data.

These domains encompass rocks that exhibit geologic conditions with the potential to host gold mineralization and, in most cases, contain elevated gold grades. Each deposit area comprises a series of sub-parallel, often braided mineralized zones that coalesce and bifurcate along the general strike-orientation of the mineral domain. The Coffee Gold deposits form relatively continuous, sub-vertical zones of gold mineralization extending from the surface to depths (locally) of more than 200 m.

Given the orientation and distribution of the gold mineralization above defined gold COGs it may prove difficult to define ore/waste boundaries with the typical blast pattern that is planned. Conventional vertical blast hole drilling and sampling methods may miss many of the narrow, strongly mineralized zones or be biased if drilled down a mineralized zone.

As such, to better define the location and grade of the narrow mineralized zones, angled RC drilling is proposed. An angle RC grade control program (assumed to be undertaken by a drilling contractor) would feature drilling rows or fences of angled drill holes oriented perpendicular to the strike of the gold system. The spacing of the drill fences and holes along each fence should initially be designed to intersect the mineralized zones in order to confidently be able to confirm the location of the ore zones. The depth of the angle RC holes should be designed to test three 10-metre benches (e.g. 50° inclined angle holes would need to be drilled approximately 60 m long).

An effective RC grade control program would need to be designed to achieve different coverage for various locations within the various deposits. Over the life of the Project, it is estimated that approximately 400,000 drill metres in total would be required.

Table 16.20: Annual Drilling Requirements

Description	Units	Year									
		Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine Production Rate, OP Waste Mined	‘000s t/year	15,125	17,601	28,036	30,800	35,163	35,015	33,828	34,730	29,715	5,349
Mine Production Rate, OP Ore Mined	‘000s t/year	3,579	4,923	6,563	4,463	4,356	4,647	5,232	5,056	6,230	1,308
Mine Production Rate, Total OP Material Mined	‘000s t/year	18,705	22,524	34,598	35,263	39,519	39,662	39,060	39,785	35,945	6,656
Crawler-Mounted, Rotary Tri-Cone, 229 mm Dia.											
Ore Productivity required	metres/year	96,445	134,252	178,989	121,714	118,794	126,730	142,704	137,883	169,924	35,662
Waste Productivity required	metres/year	181,556	214,005	340,876	374,488	427,540	425,735	411,298	422,265	361,293	65,033
Drill Mechanical Avail:	%	90%	90%	89%	88%	86%	85%	85%	85%	85%	85%
Drill Utilization:	%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Operator Efficiency:	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Drill Fleet Required	#	4	5	7	7	7	7	7	7	7	3
Crawler-Mounted, Rotary Tri-Cone, 79 mm Dia.											
Ore Productivity required	metres/year	6,304	7,066	9,420	6,406	6,252	6,670	7,511	7,257	8,943	1,877
Waste Productivity required	metres/year	15,414	15,340	24,203	26,092	29,584	29,585	28,717	29,425	25,521	4,627
Drill Mechanical Avail:	%	90%	90%	89%	88%	86%	85%	85%	85%	85%	85%
Drill Utilization:	%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Operator Efficiency:	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Drill Fleet Required	#	1	1	1	1	1	1	1	1	1	1

Source: JDS 2016

16.2.9.3 Blasting

Based on the assumption that most explosive loading would be in dry blast holes, 85% ANFO is planned to be used for blasting and a total of 15% emulsion to account for wet loading conditions. In order to limit the use of emulsion and where possible in wet conditions (assumed to be limited given climatic conditions at site), dewatering of the blast holes would be undertaken and/or bag liners used. 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. The owner would provide fuel oil, explosives magazines and delivery trucks. The owner's personnel are assumed to be responsible for loading and pattern tie-ins.

16.2.9.4 Loading

Primary loading is planned to be performed by diesel front shovels with a 15 m^3 bucket. A front-end wheel loader with an 11.5 m^3 bucket and a 4.5 m^3 excavator would be used for secondary loading, rehandle and shovel support.

The productivities of the excavators and wheel loader were calculated with respect to the selected haul trucks and are summarized in Table 16.21.

Dig rates reflect the selective nature of the mining operation with respect to the narrow, steeply dipping ore veins at Coffee Gold.

Table 16.21: Loading Unit Productivity

Item	Units	Ore		Waste	
		15 m^3 Excavator	11.5 m^3 Wheel Loader	15 m3 Excavator	11.5 m^3 Wheel Loader
In-situ density	t/m^3	2.5	2.5	2.6	2.6
Swell factor	Loose:bank	1.4	1.4	1.4	1.4
Loose material density	t/ m^3	1.8	1.8	1.9	1.9
Bucket size	m^3	15	11.5	15	11.5
Bucket fill factor	%	95	90	95	90
Tonnes per bucket	t	25.2	18.3	26.4	19.2
Size of truck to load	t	144	144	144	144
Theoretical buckets to load	#	5.8	7.9	5.5	7.6
Average loading cycle time	sec	40	60	40	60
Average spot time between loads	sec	30	30	30	30
First bucket time	sec	10	10	10	10
Total time to load truck	min	3.9	7.6	3.7	7.3
Theoretical avg. truck loads per day	#	261	133	275	139
Truck load factor	%	95	95	95	95
Loading productivity	t/d	37,600	19,200	39,700	20,000
Loading productivity	t/op hr	1,570	800	1,650	835

Source: JDS 2015

Operating hours for the loading fleet were estimated by the amount of material to be moved within a specified period and the associated productivities. Fleet size was then calculated using total operating hours for the period and the operating hours per unit within the period. Operating hours and fleet size are shown in Table 16.22.

16.2.9.5 Hauling

Primary haulage is planned to be done with 144 t payload haul trucks.

In order to calculate required production haul truck hours, haulage profiles were calculated by bench and period, for the various open pit deposits taking into account material destination and type. Runge Talpac software was used to determine truck requirements and productivities. All hours were then summarized by period for the determination of truck fleet size and truck additions.

The annual haul truck requirements are shown in Table 16.23.

Table 16.22: Annual Loading Equipment Requirements

Description	Units	Year									
		Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine Production Rate, OP Waste Mined	'000s t/year	15,125	17,601	28,036	30,800	35,163	35,015	33,828	34,730	29,715	5,349
Mine Production Rate, Waste Rehandle	'000s t/year	0	0	0	0	0	0	0	0	0	3,237
Mine Production Rate, Ore Mined from OP	'000s t/year	3,579	4,923	6,563	4,463	4,356	4,647	5,232	5,056	6,230	1,308
Mine Production Rate, Ore Rehandle	'000s t/year	4,385	6,314	5,098	6,692	6,714	6,439	6,056	6,300	5,368	5,063
Mine Production Rate, Total Material Moved	'000s t/year	23,090	28,837	39,696	41,954	46,233	46,100	45,116	46,086	41,313	14,956
Diesel, 15 m^3 Front Shovel											
Material Productivity required	'000s t/year	8,429	9,443	9,322	9,933	11,065	10,850	10,813	10,431	9,298	6,122
Shovel Mechanical Avail:	%	90	90	89	88	86	85	85	85	85	85
Shovel Utilization:	%	96	96	96	96	96	96	96	96	96	96
Operator Efficiency:	%	84	84	84	84	84	84	84	84	84	84
Shovel Fleet Required	#	2	2	3	3	3	3	3	3	3	3
Diesel, 11.5 m^3 Wheel Loader											
Material Productivity required	'000s t/year	4,528	4,502	4,323	3,456	3,692	3,895	3,562	4,055	3,774	2,661
Loader Mechanical Avail:	%	90	90	89	88	86	85	85	85	85	85
Loader Utilization:	%	96	96	96	96	96	96	96	96	96	96
Operator Efficiency:	%	84	84	84	84	84	84	84	84	84	84
Loader Fleet Required	#	2	2	3	3	3	3	3	3	3	3
Diesel, 4.5 m^3 Excavator											
Material Productivity required	'000s t/year	1,238	949	2,172	1,786	1,964	1,867	1,990	2,625	2,096	600
Loader Mechanical Avail:	%	90	90	89	88	86	85	85	85	85	85
Loader Utilization:	%	96	96	96	96	96	96	96	96	96	96
Operator Efficiency:	%	84	84	84	84	84	84	84	84	84	84
Loader Fleet Required	#	1	2	2	2	2	2	2	2	2	2

Source: JDS 2015

Table 16.23: Annual Haulage Equipment Requirements

Description	Units	Year									
		Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine Production Rate, OP Waste Mined	'000s t/year	15,125	17,601	28,036	30,800	35,163	35,015	33,828	34,730	29,715	5,349
Mine Production Rate, Waste Rehandle	'000s t/year	0	0	0	0	0	0	0	0	0	3,237
Mine Production Rate, Ore Mined from OP	'000s t/year	3,579	4,923	6,563	4,463	4,356	4,647	5,232	5,056	6,230	1,308
Mine Production Rate, Ore Rehandle	'000s t/year	4,385	6,314	5,098	6,692	6,714	6,439	6,056	6,300	5,368	5,063
Mine Production Rate, Total Material Moved	'000s t/year	23,090	28,837	39,696	41,954	46,233	46,100	45,116	46,086	41,313	14,956
144 t Haul Truck											
Total Net Operating Hours	NOH/year	39,646	57,006	89,565	74,623	79,725	89,418	95,460	104,264	94,267	32,781
Truck Mechanical Avail:	%	90	90	89	88	86	85	85	85	85	85
Truck Utilization:	%	96	96	96	96	96	96	96	96	96	96
Operator Efficiency:	%	80	80	80	80	80	80	80	80	80	80
Truck Fleet Required	#	10	12	18	16	15	20	18	20	19	10

16.2.9.6 Ancillary and Support Equipment

The main function of the ancillary and support equipment functions are described below.

- Komatsu D275 track dozer: to be primarily used for shovel support and clean up, dump maintenance, road construction, high-wall cleaning and other activities as needed;
- Komatsu WD500 wheel Dozer: to be used to support waste dump maintenance, drill pattern cleanup, and shovel floor maintenance; and
- Komatsu GD825 grader: to be used primarily for road maintenance and pit and dump floor maintenance, road construction.

The selected ancillary equipment is shown in Table 16.24.

Table 16.24: Planned Open Pit Mine Ancillary Equipment

Description	Number
Komatsu D275 - class track dozer	5
Komatsu WD500 - class wheel dozer	2
Komatsu GD825 - class grader	3

Source: JDS 2015

16.2.9.7 Equipment Replacement Criteria

Equipment suppliers provided estimates for equipment life, and where information was lacking, industry standards and JDS experience were used. The estimated useful equipment lives are shown in Table 16.25.

Given the estimated 10-year life of the Coffee Gold open pits, a number of equipment replacements are envisioned

Table 16.25: Equipment Life Cycle

Equipment Unit	Operating Life (hours)
Major equipment	
Blasthole drill (229 mm)	50,000
Blasthole drill (79 mm)	45,000
Diesel hydraulic front shovel	70,000
Front-end Wheel loader	49,000
Haul truck	70,000
Support equipment	
Track dozer	40,000
Rubber tired dozer	56,000
Motor grader	56,000

Source: JDS 2015

16.2.9.7.1 Schedule of Purchases

The schedule of equipment purchases (including replacement units) is shown in Table 16.26. The years shown are the first year of use for each unit.

16.2.10 Mine Maintenance

The key elements provided by equipment maintenance are equipment safety, availability, reliability, and operability.

The strategy for repair and maintenance of the open pit mobile equipment fleets for the project is planned to be a balance of minimizing risk and cost to Kaminak. All maintenance on site will be carried out with Kaminak 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 are anticipated. However, should they be required it is anticipated that they would be performed on site by contractors.

Table 16.26: Planned Equipment Purchase Schedule

	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Type										
D50KS drills (152 – 229 mm)	4	1	2				2			
DX800 drill (76 mm)	1									
Komatsu HD1500-7 trucks (144 t)	10	2	6			2				
Komatsu PC3000 shovels (15 m3)	2		1							
Komatsu PC800 excavators (4.5 m3)	1	1								
Komatsu WA900 wheel loaders (11.5 m3)	2		1				1			
Komatsu D275 track dozers	4		1			3	2			
Komatsu WD500 wheel dozers (4.2 m blade)	1	1								
Komatsu GD825 graders	2		1							

Source: JDS 2015

16.2.11 Mine Services

16.2.11.1 Emissions Control

Dust monitoring will be conducted routinely during operations and effective dust suppression will be essential during mine operation. The most likely areas of dust generation would be from road traffic in the pit during excavation and blasting, ore placement and spreading on the heap leach pad, the crushing facilities, waste rock and ore stockpiling activities.

16.2.12 Mine Personnel and Organization Structure

16.2.12.1 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 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.

16.2.12.2 Personnel Activities

The mining operation will be headed by the Mine Manager, who will report to the General Manager. Four mining departments are planned;

- Mine operations;
- Mine maintenance;
- Mining engineering, and
- Geology.

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, waste rock storage facility 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 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.

Staff and labour requirements over the LOM plan are summarized in Table 16.27.

Table 16.27: Annual Personnel Requirements

Description	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine Operations										
Drillers	17	22	27	27	27	27	27	27	27	9
Blasters	2	2	2	2	2	2	2	2	2	2
Blasting helpers	4	4	4	4	4	4	4	4	4	4
Bulk explosives truck drivers	2	2	2	2	2	2	2	2	2	2
Shovel/loader operators	16	20	25	24	24	24	24	24	24	15
Haul truck drivers	36	42	61	54	54	68	61	68	62	34
Track dozer operators	12	12	14	14	15	17	14	14	14	8
R.T. Dozer Operator	4	8	8	8	8	8	8	8	8	4
Grader operators	8	8	9	8	8	8	8	8	8	4
Water/ancillary truck drivers	2	2	3	3	3	3	3	3	3	2
Labourer/trainees	5	5	6	10	8	8	11	13	7	4
Subtotal Mine Operations	**104**	**123**	**161**	**156**	**155**	**171**	**164**	**173**	**161**	**87**
Total crew	26	31	41	39	39	43	41	44	41	22
Mine Maintenance										
Heavy equipment mechanics	8	8	12	12	12	12	12	12	12	8
Welders/mechanics	8	8	12	12	12	12	12	12	12	8
Electricians/instruments	4	6	6	6	6	6	6	6	6	4
Lube/PM mechanics/light duty mech.	8	8	12	12	12	12	12	12	12	8
Tiremen	4	4	4	4	4	4	4	4	4	4
Labourers/trainees	3	5	5	5	5	5	6	6	4	5
Subtotal Mine Maintenance	**35**	**39**	**51**	**50**	**51**	**51**	**52**	**52**	**50**	**37**
Total/crew	9	10	13	13	13	13	13	13	13	10
Technical/Supervisory										
Mine operations manager	1	1	1	1	1	1	1	1	1	1
Mine superintendent	1	1	1	1	1	1	1	1	1	1
Maintenance superintendent	1	1	1	1	1	1	1	1	1	1
Mine shift foremen	6	6	6	6	6	6	6	6	6	6
Maintenance planner	1	1	1	1	1	1	1	1	1	1
Maintenance shift foremen	4	4	4	4	4	4	4	4	4	4
Chief mining engineer	1	1	1	1	1	1	1	1	1	1
Senior mine engineer	1	1	1	1	1	1	1	1	1	1
Mine engineers	2	2	2	2	2	2	2	2	2	2
Ore control engineers	2	2	2	2	2	2	2	2	2	2
Mine technicians	1	2	2	2	2	2	2	2	2	-
Surveyors	2	2	2	2	2	2	2	2	2	2
Survey assistants	4	4	4	4	4	4	4	4	4	2

Description	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine clerk	1	1	1	1	1	1	1	1	1	1
Chief geologist	1	1	1	1	1	1	1	1	1	1
Mine geologists	2	2	2	2	2	2	2	2	2	1
Technicians/ore control	2	2	2	2	2	2	2	2	2	2
Subtotal Technical/Supervisory	**33**	**34**	**34**	**34**	**34**	**34**	**34**	**34**	**34**	**29**

Source: JDS 2015

16.2.12.3 Labour Build-up and Initial Training Schedule

Key operation's personnel are envisioned to be recruited during the construction phase of the Project. The management team, including managers, human resources, health and safety, would be in place for construction. Staffing levels would then progressively increase during the construction phase of the Project.

17 Recovery Methods

17.1 Summary

This section describes the recovery methods used for the Kaminak 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 5.0 Mt of ore per year with an average gold head grade of 1.45 g/t at an overall gold recovery of 86.3%.

The two-stage crushing plant will operate at a nominal 18,182 t/d throughput, 275 days per year. During the coldest part of the year (January through March) crushing and heap leach pad loading activities will be suspended. The process plant, located near to and down-gradient from the heap leach facility HLF to minimize the pumping and pipeline requirements for pregnant and barren solutions, will operate 365 days per year. The pregnant solution will flow to the plant at a nominal rate of 455 m^3/h and a design flowrate of 600 m^3/h. The plant is designed to process 5 t of carbon per day using an absorption, desorption and refining process to extract gold from the pregnant solution to produce the gold doré.

The gold ore processing facilities will include the following unit operations:

Crushing and Ore Handling

- Primary crusher: a vibrating grizzly screen and jaw crusher in open circuit producing a final product P_{80} of approximately 190 mm;
- Secondary crusher: a vibrating screen and cone crusher operating in reverse closed circuit producing a final product P_{80} of 50 mm, and;
- Heap placement: crushed ore stacked to a 3,000 t - capacity stockpile, reclaimed by a front-end loader and hauled to the heap leach pad by 144 t trucks.

Heap Leach Pad

- Crushed ore stacking and spreading by dozers
- 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, and thermal regeneration of carbon to remove organic foulants, carbon stripping to recover gold into solution.
- Gold refining: gold electrowinning (sludge production), filtration, drying, mercury retorting, and smelting to produce gold doré.

A process flowsheet, crushing plant layout and process plant layout are presented in Figures 17.1, 17.2 and 17.3.

Figure 17.1: Process Flowsheet

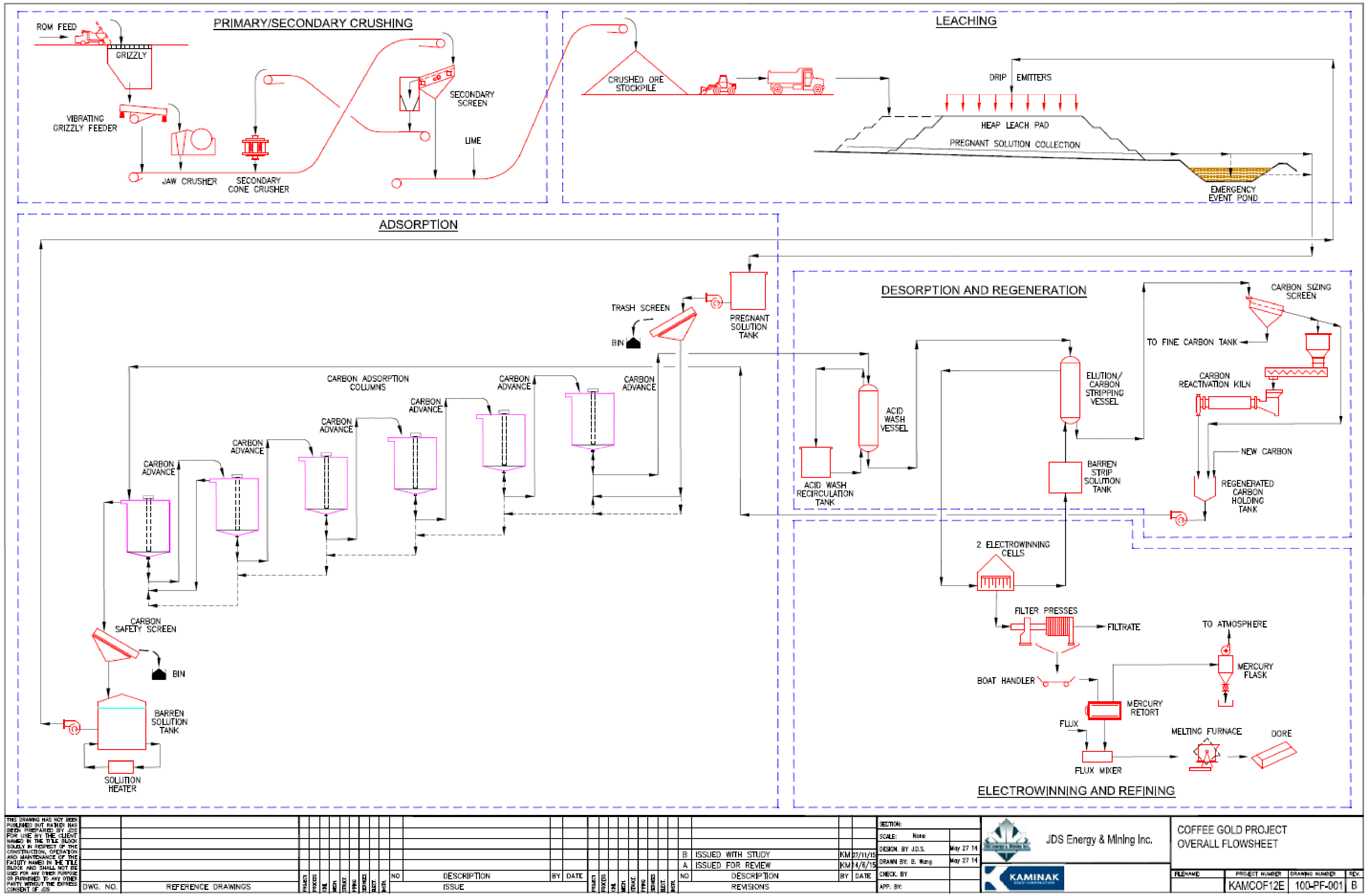


Figure 17.2: Crushing Plant Layout

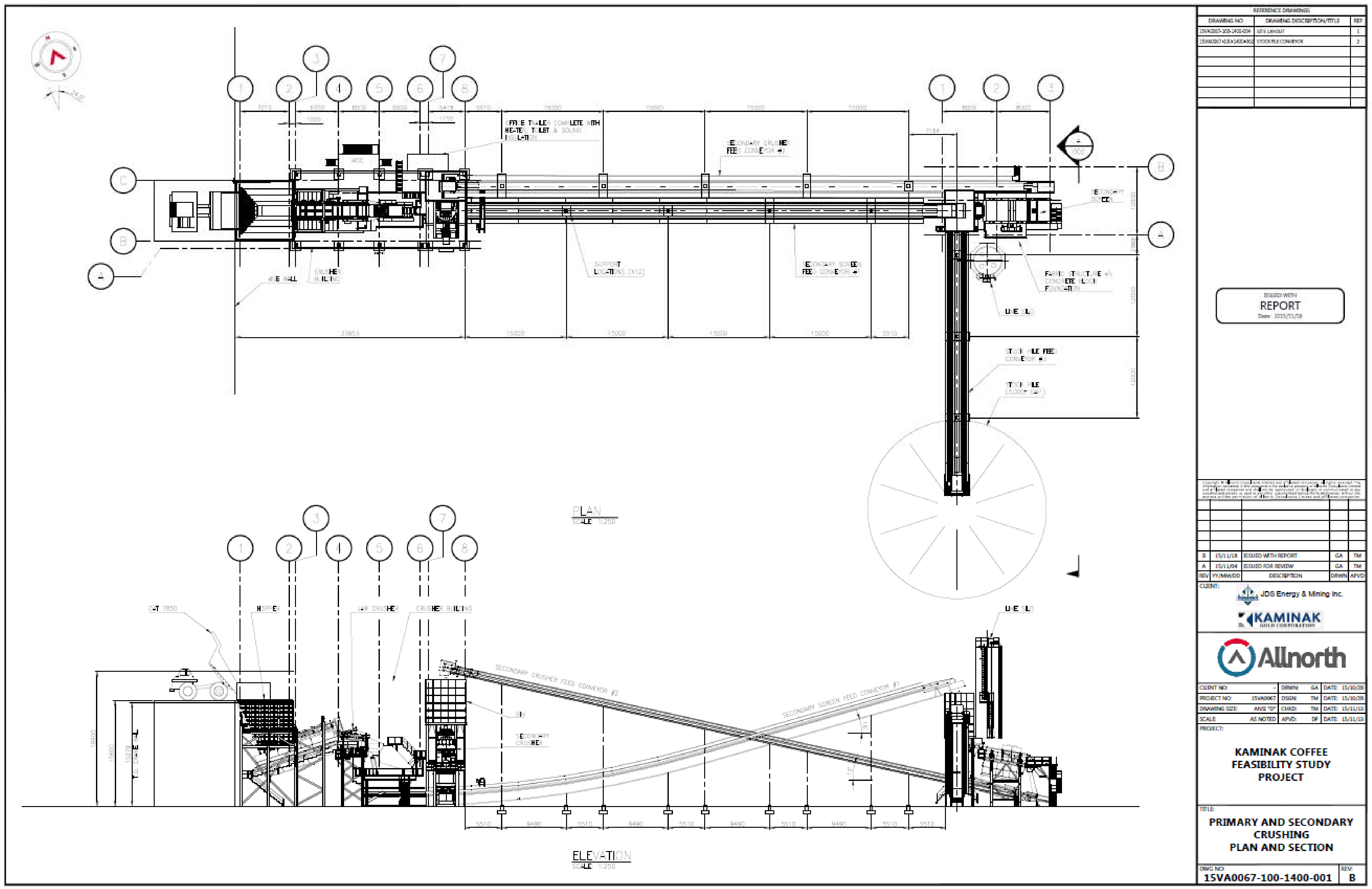
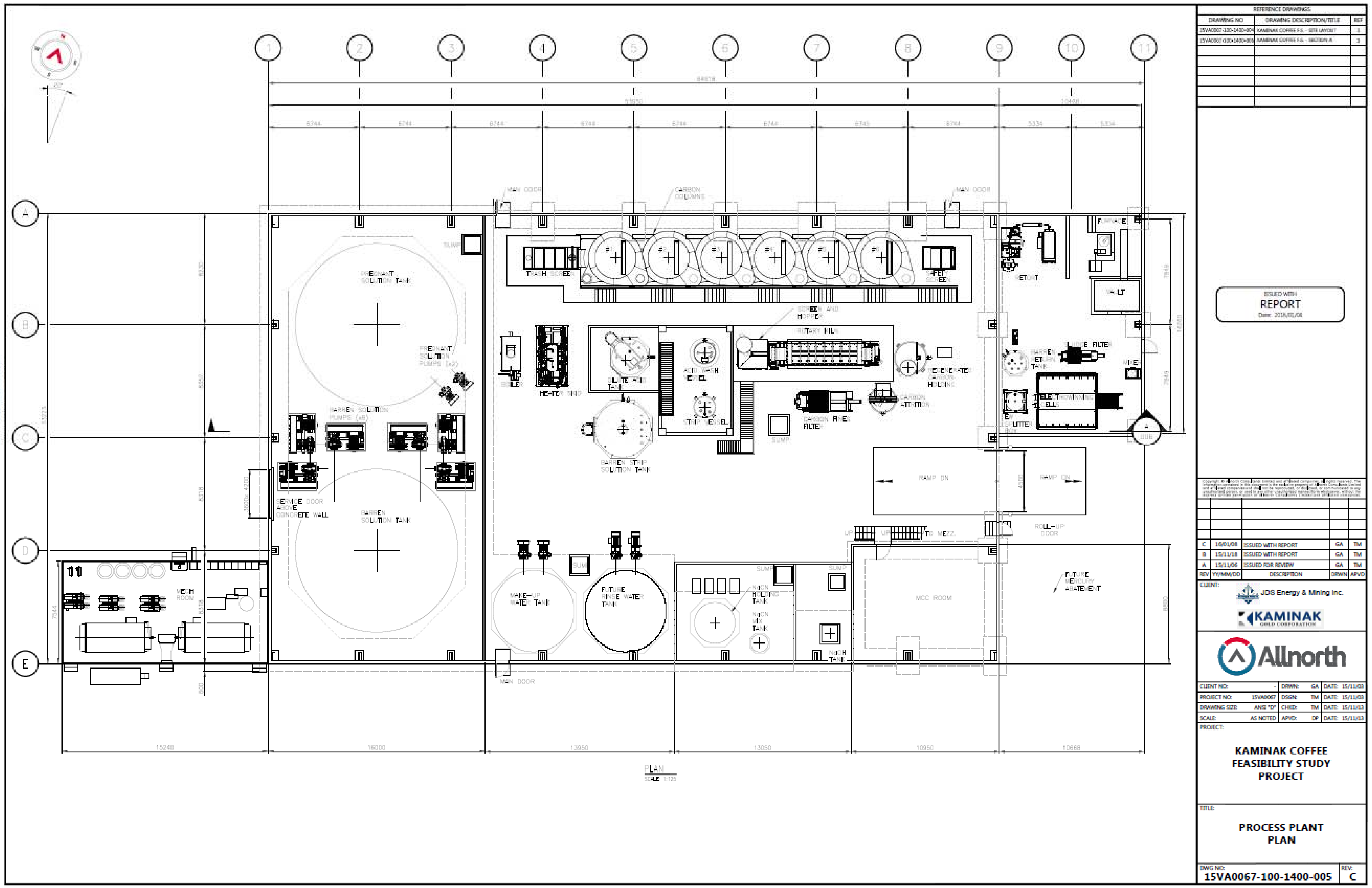


Figure 17.3: Process Plant Layout



17.2 Process Design Criteria

The process design criteria and mass balance detail the annual ore production, major flows, and plant availability. Several considerations to mitigate the Yukon climate, especially with respect to the severe winter conditions, have been included in the general design criteria:

- Pregnant solution will be heated as required to maintain a minimum discharge temperature at the heap leach pad of 6ºC;
- The drip emitter lines will be buried in the winter to provide a degree of insulation;
- Barren and pregnant solution–feed pipelines will be buried a minimum depth of 1.5 m;
- Dedicated standby generators will be installed for backup power supply to the barren solution pumps;
- The crushing and process plant buildings will be pre-engineered steel structures with insulated steel roofs and walls.

The key process design criteria are summarized in the Table 17.1.

Table 17.1: Process Design Criteria

General	Unit	Value
Annual Treatment Rate	t/y	5,000,000
Crushing Plant Operation	d/y	275
Crushing Plant Operation	t/d	18,182
Crushing Plant Operation	h/d	18
Design Rate Crushing Plant Operation	t/h	1,010
Heap Loading and Spreading Method	-	Truck and dozer
Heap Loading Operation	d/y	275
Heap Loading Operation	t/d	18,182
Heap Loading Operation	h/d	18
Design Rate Heap Loading Operation	t/h	1,010
Average LOM Feed Grade	g /t Au	1.45
Overall Ultimate LOM Recovery	%	86.3
Ore Characteristics		
Specific Gravity (Average)	t/m^3	2.52
Dry Crushed HL feed Bulk Density	t/m^3	1.6
Run-of-Mine Moisture	%	4
Bond Crusher Work Index (Oxides)	kWh/t (Latte)	11.5
Abrasion Index (Oxide)	g (Supremo)	0.078
Crushing		
Days per Week	d	7
Days per Year	d	275
Shifts per Day	shifts	2
Shift Length	h	12
Crusher Availability	%	75
Hours per Day	h	18
Primary Crusher Feed		
Type		Apron Feeder
Size	mm	1,800 x 11,200
Motor	kW	90
Type		Vibrating Grizzly Screen
Size	mm	3,900 x 2,100
Motor	kW	2 x11
Primary Crusher		
Type		Jaw
Size	mm	1,300 x 1,500
Closed Side Setting	mm	165
Motor	kW	220
Product Size, 80 % Passing	mm	190

General	Unit	Value
Secondary Screen Feed Conveyor		
Type		Belt (Covered)
Width	mm	1,800
Motor	kW	80
Secondary Crusher Screen		
Type		Vibrating, Single Deck
Size	mm	3,000 x 6,000
Screen Deck Aperture	mm	65
Motor	kW	2 x 30
Secondary Crusher Feed Conveyor		
Type		Belt (Covered)
Width	mm	1,200
Motor	kW	60
Secondary Crusher		
Type		Standard Cone Crusher
Size		CH890 or equivalent
Motor	kW	750
Closed Side Setting	mm	46
Stockpile Feed Conveyor		
Type		Belt (Covered)
Width	mm	1,400
Motor	kW	37
Stockpile		
Feed Size, 80 % Passing	mm	50
Stockpile Capacity, Live	t	3,000
Leach Pad (Mines Group)		
Ultimate Design	mt	47,000,000 (nominal)
		61,500,000 (with expansion)
Slope Stability, factor of safety		1.3 (static)
		1.0 pseudo-static
Ultimate Height	m	80
Lift Height	m	10 (nominal)
Heap Slope, Overall	h:v	2.5:1
Leach Cycle, Primary	d	40
Solution Application Rate	$l/hr/m^2$	10
Solution Flow Rate	m^3/h	455 (nominal)/600 (maximum)
Area Under Leach	m^2	45,500
Ponds and Diversion Channels for Heap		
Design storm event for peak flow rate, 100-yr, 24-hr	mm	79
Design storm event for pond storage, 24-hr PMP	mm	280

General	Unit	Value
Heap drainage	m^3	32,760
Freeboard in ponds	mm	500
Pregnant Solution	-	Piping system – Gravity to Process Plant
Tank Dimensions	m x m	12 x 12
Pumping		
Number of Units Installed		2
Number of Units Operating		1
Type		Krebs millMAX-e™ Model 10x8-21.3 HH MMC or equiv.
Motor	kW	45
Barren Solution		
Tank Dimensions	m x m	12 x 12
Pumping		
Number of Units Installed		4 (6 future)
Number of Units Operating		2 (3 future)
Type		Krebs millMAX-e™ Model 8x6-27.5 HH MMC or equiv.
Motor	kW	298
Capacity per Pump	m^3/h	600
Barren Solution Heating		
Type	-	Boiler (designed with 50% excess capacity)
Months	-	November through March
Temperature Heating	°C above Pregnant. Temperature	8
Minimum Discharge Temperature at HLF	°C	6
Carbon-in-Columns		
Quantity per Train	-	6
Number of Trains	-	1
Tank Dimension	m x m	3.2 x 3.2
Power - Pumps	kW	21
Capacity	m^3/h	455 nominal, 600 design
Carbon Acid Wash		
Power - Pumps	kW	8
Carbon Capacity	mt	5
Dilute Acid Tank Capacity	bV	3
Dilute Acid Tank Dimensions	m x m	3.5 x 3.5
Carbon Stripping (Elution)		
Power - Pumps	kW	6
Carbon Capacity	mt	5
Barren Strip Tank Capacity	bV	4
Barren Strip Tank Dimensions	m x m	4.0 x 4.0

General	Unit	Value
Heat Skid – Strip Solution Heating		
Power	kW	15
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)	5 (11,500 x 2,500 x 2,500)
Drying Oven/Mercury Retort		
Power	kW	35
Doré Furnace		
Type	-	Induction
Power	kW	125

Source: JDS 2015, All equipment sizes and power requirements are approximate

17.3 Process Description

17.3.1 Primary Crushing

Run-of-mine ore (ROM) will be trucked from the open pits and dumped directly into a primary feed hopper. Primary crusher feed will be drawn from the feed hopper by an apron feeder discharging onto a vibrating grizzly screen. The grizzly screen oversize will feed the primary jaw crusher. The grizzly undersize and jaw crusher product are to be transported to the secondary screen by a secondary screen feed conveyor, which is equipped with a metal detector and magnet.

The crushing plant will operate 275 days per year. 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.

17.3.2 Secondary Crushing and Screening

Ore from the secondary screen feed conveyor will be transported to the secondary vibrating screen. Screen undersize material (final product with a P_{80} of 50 mm) will be conveyed to the 3,000 t heap leach feed stockpile. 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 1.5 kg/t. Screen oversize material will be conveyed to the secondary cone crusher. The secondary cone crusher discharge and jaw crusher product combine on the secondary screen feed conveyor back to the secondary screen.

17.3.3 Crushed Heap Leach Feed Stockpile

The crushing plant product, with a final crushed ore P_{80} of 50 mm, will be reclaimed and loaded at a nominal rate of 18,182 t/d by a front-end loader and transported to the heap leach pad by a fleet of 144 t haul trucks. The haul trucks will be of the same type as those used for the mining operation, and will be fully integrated into the mining operation in order to optimize operating efficiencies.

When leach pad loading operations are suspended during January, February and March, the crushing plant will be shut down.

17.3.4 Heap Leach Facility

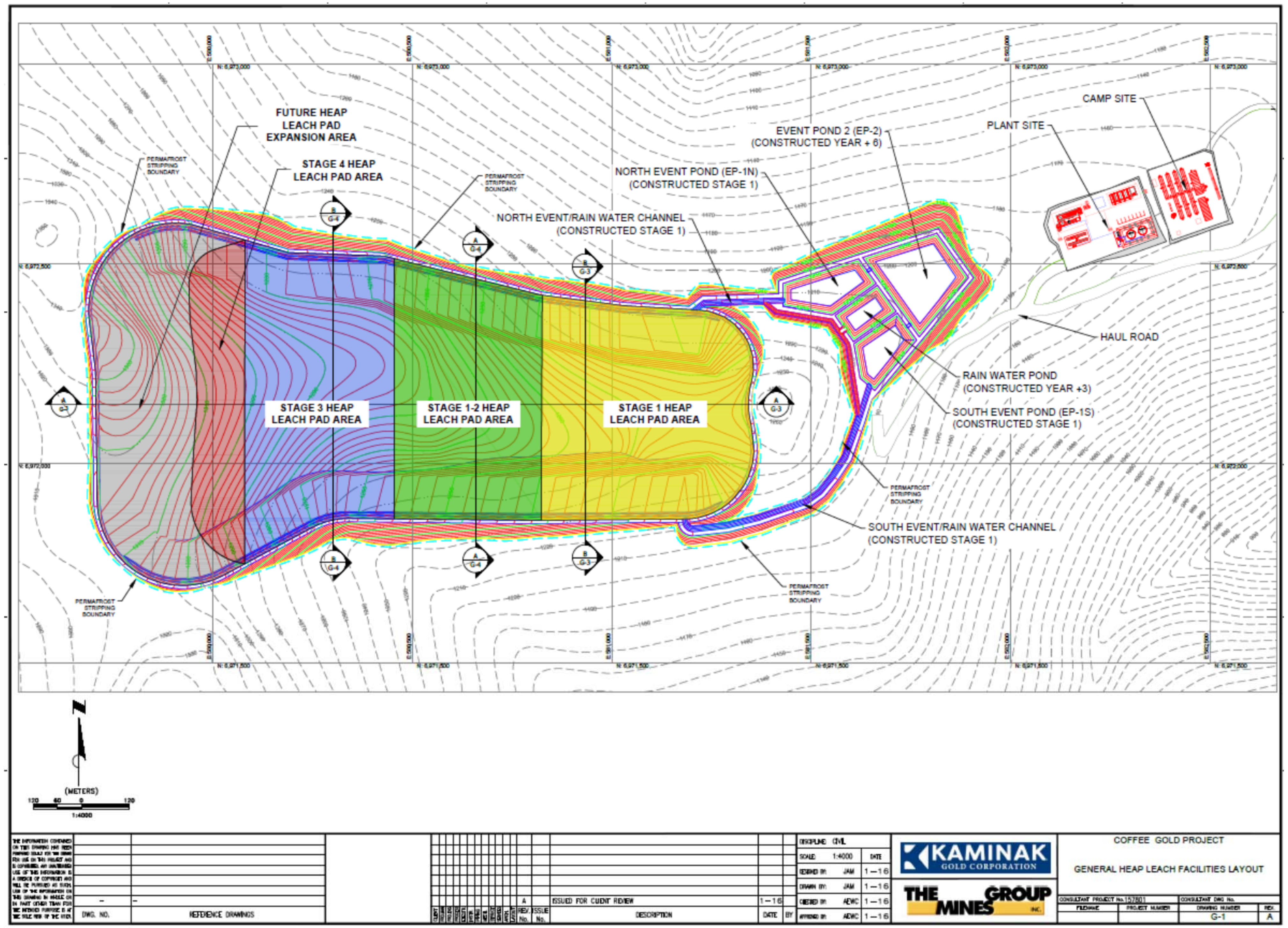
Kaminak engaged The MINES Group (MINES) to develop Feasibility Study-level plans for a heap leach facility (HLF), including the supporting geotechnical, seismicity, hydrogeological, and water balance analyses. Additionally, Geo-Logic Associates (GLA) was engaged to provide geotechnical testing of fresh and leached ore samples, and perform thermal modeling of the heap. The MINES work, summarized below, is drawn from the MINES Group report entitled: "Feasibility Design Report for Coffee Heap Leach Facility, Yukon Territory, Canada, January 2016". GLA's geotechnical testing and modeling is summarized within the full report and associated documents are attached as appendices in the MINES report.

The HLF is designed to allow crushed ore stacking to a maximum height of approximately 80 m (measured vertically over the liner system), which results in a design capacity of 47 Mt. There is the opportunity to expand the pad to at least 61.5 Mt. The HLF comprises the following:

- Conventional, multi-lift (nominally 10 m per lift), free-draining heap over a gently sloping heap leach pad (HLP) along the axis of the ridgeline west of the ADR plant as shown in Figure 17.4;
- The leach pad will be graded and constructed in a nominally balanced cut-and-fill manner using locally borrowed (within the heap boundary) rock for structural fill, supplemented as needed by mine waste including waste rock and, if available, thaw-stable soil for lining the pad subgrade before placement of the liner system.;
- Permanent and interim perimeter diversion channels and berms to manage surface water flows;
- Perimeter access and ore haulage roads;
- Leach pad liner system will be constructed in the steps described below:
    - Graded subgrade to provide a non-puncturing surface for the geosynthetic liner;
    - Leak detection using horizontal wick drains to operate as large-scale lysimeters;
    - Reinforced geosynthetic clay liner (GCL);
    - Primary geomembrane liner, 2.0 mm thick linear low density polyethylene (LLDPE), bottom side aggressively textured, and
    - Overliner gravel (crushed ore), 500 mm thick, with drainage pipes, to protect the liner from ripping during initial ore stacking and to minimize the hydraulic head directly over the geomembrane;

- Gravity drainage from the leach pad to the pregnant tank at the ADR plant or (in the case of an upset) the events ponds in double-contained and buried pipes;
- Raincoats placed over portions of the heap starting in Year 3 to reduce the volume of precipitation entering the process circuit and to increase heat retention in the winter; and
- Conventional external ponds for events and clean water storage from precipitation run-off. There will be three events ponds and one rainwater pond. The primary determiner to the required storage volume is leach pad area, and thus as the pad is expanded in stages (1 through 4) the pond capacity requirements increase. Thus, the four ponds will be built in different years as listed in Table 17.2. Events ponds EP-1S and EP-1N will have three synthetic liner layers: a double HDPE geomembrane, separated by a leak detection and collection layer, and underlain by a GCL. Events pond EP-2 and the rainwater pond will be lined with two geosynthetic liners: an HDPE liner over a GCL. The difference in liner systems between the various ponds is due to the differences in solution chemistry (the rainwater pond receives only clean water and EP-2 is designed for the PMP event, which would be highly diluted solution), and frequency and duration of storage (EP-2 may never be used and then only for a very short period).

Figure 17.4: Site Heap Leach Facility and Plant Layout



Source: The Mines Group 2015

Event ponds are sized to contain the following cumulative volume: (a) all contact water produced by a PMP event; (b) full heap drainage during a power outage or pump failure; (c) seasonal water accumulation assuming a wet cycle. The PMP storage volume is sufficient to contain the runoff from multiple back-to-back 200-year precipitation events and may be compared to a 10,000 year event. Full heap drainage has been estimated by applying the full operating pregnant flow rate of 455 m^3/hr for a 72-hr period without attenuation. If power or pumping capacity is lost and the irrigation flow ceases, the rate of flow out of the heap will begin to decline rapidly and will approach negligible rates within a few days..

The pond sizes, design criteria and year to be placed in service are summarized in Table 17.2.

Table 17.2: Event Pond Design Criteria and Containment Capacities

Pond	Design Criteria (m^3)[1]	Actual Capacity to Free Board (m^3)	Actual Capacity to Pond Crest (m^3)	Year Placed in Service
EP-1N	191 360[2]	112,349	122,184	Year -1
EP-1S		89,777	97,810	Year -1
EP-2	210,000	222,874	240,468	Year 6
Raincoat	47,000	51,925	57,074	Year 3

Note:

1. The events ponds capacities include seasonal water accumulation, full heap drainage, the 24-hour PMP storm event, and seasonal solution accumulation.

2. Combined containment capacity required through year 6.

Source: Mines Group 2016

The heap leach pad will be constructed in stages over several years and Table 17.3 summarizes the size and year of construction for each stage. The ultimate heap leach pad will accommodate 47 Mt and will be constructed in four stages, with the first stage constructed in Years -2 and -1 and sized for a minimum capacity of 6 Mt of ore. Subsequent stages will each be constructed with 18 months to three years of ore capacity. Ore will be truck and dozer stacked in 10 m lifts to a total average height of about 50 m and a maximum of 80 m (measured vertically over the liner system). Stage 1 is sized for an ore depth of 30 m to create a sufficiently large thermal mass to allow winter irrigation. Each pad stage will be separated from the adjacent stage by a ditch or berm and drainage pipe, providing hydraulic (solution) isolation between stages. In addition, cells will be created within each stage by constructing a drainage ditch or berm with a drainage pipe every 100 m. The berms and ditches will allow high-resolution tracking of solution chemistry (especially gold tenor) and will aid in progressive closure by allowing rinsing of the older portions of the heap beginning by Year 4.

Table 17.3: Summary of Stages of Leach Pad Construction

Stage	Elevation of Crest of Heap (masl)*	Stage Area (m^2)	Cumulative Pad Area (m^2)	Cumulative Capacity (Mt)	Year for Construction
1	1270	271,289	271,289	7.2	-1
2	1300	226,827	498,116	19.2	1
3	1340	256,281	754,397	39.4	3
4	1350 (Final)	64,653	819,050	47.3	7

Source: Mines Group 2016

* Elevation when next stage needs to be constructed (stacking will continue on previous stage)

A Komatsu D2750-class dozer will be used to spread and construct the lifts of ore on the heap. The toe of each lift will be set back from the crest of the lower lift nominally 10 m to create an overall slope of 2.5H: 1V. The benches provide heap slope stability, protect the liner system from damage due to sloughing, and provide access to the heap for operators. The overall slope of 2.5H : 1V was also selected to facilitate closure.

17.3.4.1 Heap Leach Liner System

The leach pad liner system was designed to collect process and rinse solutions and protect surface and groundwater quality through the operating life and active closure of the heap. The system consists of two layers of synthetic liners with leak detection below and a hydraulic head control or drainage layer on top of the liners. Each component is discussed below, from the bottom up.

Graded subgrade

The subgrade will be constructed using competent, thaw stable rock borrowed from within the HLP area, supplemented with mine waste rock or thaw-stable soi from mine pre-stripping l. Because the bottom liner is a heavily reinforced GCL, and GCLs are self-sealing and provide excellent protection for geomembranes, the system will be very tolerant of subgrade conditions. Nevertheless, finer material from the rock or thaw-stable soil will be used in the top lift of fill, and as a thin plating layer for areas of exposed rock cut, to create a smooth, non-puncturing surface.

Leak detection system

Leak detection will be accomplished by three separate systems: monitoring wells installed away from the pad; unsaturated or vadose zone monitoring under the leach pad using an adaptation of lysimeter technology; and electrical leak location surveys performed after construction of each stage of the leach pad.

Vadose zone monitoring: In the upper zone of the fill and in the subgrade of exposed cut, horizontal wick drains will be installed under each collection ditch or berm to operate as large-scale lysimeters. The collection ditches and berms create cells within each stage of the leach pad to allow solution control during operations and rinsing, and these will be spaced every 100 m.

These locations were selected for the wick drains as these are where solution will be concentrated. This represents a considerable level of protection above industry standards.

Electrical leak location (ELL) surveys: This is an adaptation of geophysical methods to liner systems, and uses electrical resistance to locate holes by finding areas of low resistance or high conductivity. ELL surveys have been used for over 20 years but are relatively new to mining applications. These surveys dramatically reduce the likelihood of having holes post-construction and reduce the size for any holes which may be left behind. Several authors have postulated that 75% of all geomembrane damage and nearly all of the significant damage (holes large enough to allow leakage of any environmental consequence) happens during placement of the overliner layer. Thus, the net result of using ELL surveys is a 10- to 100-fold reduction in risk of consequential leakage. Thus, by using ELL surveys it is possible to come as close to a perfect or “leak free” liner as the best available/applicable technology (BAT) allows. ELL surveys can be done over bare liner, filled ponds, and over the top of the overliner. For Coffee, the plan is to do the surveys only on the leach pad after the overliner is placed, which is the most critical stage.

Reinforced geosynthetic clay liner (GCL)

The synthetic liner system consists of two components: a geomembrane liner over, and in direct contact with, a low permeability soil liner. The Coffee area does not contain clay or clay-like sources, and thus a GCL liner was selected. GCLs offer some significant advantages over compacted clay liners (CCLs), and they are often used as system upgrades even with local clay sources are available. The principal advantages can be summarized as follows;

- GCLs are manufactured products with tight production tolerances and robust quality control procedures, which results in products that are more reliable than field-constructed systems;
- The bentonite used in GCLs has a permeability several orders of magnitude lower than CCLs;
- The bentonite is layered between geotextile fabrics so that the geotextiles provide significant additional puncture protection for the geomembrane. They also provide confinement for the bentonite (any soil liner’s permeability decreases with better confinement);
- GCLs are not subject to construction defects common for CCLs such as inhomogeneity, cracking, and inclusion of high permeability materials such as stones and roots;
- The quality of contact between the geomembrane and low permeability soil layer directly affects the leakage through any defects, and geomembrane/GCL contacts are better than most geomembrane/CCL contacts; and,
- GCLs are more resistant than CCLs to the effects of cyclic freeze-thaw and wetting-drying.

The Coffee design uses the thickest and the most heavily reinforced GCL on the market. This produces an installed product that is most likely to be defect free; that is, the installation process is where most of the damage occurs, and the reinforced products tolerate installation stresses best. The reinforcement also makes the heap more stable by increasing the shear strength of the geomembrane/GCL system. The selected product (CETCO DN or DN9) also has the highest swell factor of any of the GCLs on the market, which improves its as-installed performance significantly.

Primary geomembrane liner

The selected geomembrane is a 2.0 mm thick linear low density polyethylene (LLDPE) with the bottom side aggressively textured. The texturing is to provide a close bond between the geomembrane and GCL.

The two most common materials used for leach pad liners, as well as liners for a wide range of other industrial containment systems, are high density polyethylene (HDPE) and LLDPE. HDPE has been in use since the 1970s and has been the dominate liner material for most applications, including heap leach pads, since the 1980s. It is a highly reliable material with a good performance history. However, HDPE is stiffer, less puncture resistant and produces lower interface shear strengths than LLDPE. Stiffer materials present installation and performance problems at very low temperatures. Puncture resistance is the most critical parameter for a leach pad liner, and shear strength is important for heap slope stability.

For heaps of maximum ore depth of up to about 120 m, the industry standard liner is 1.5 mm thick HDPE. For Coffee, a 2.0 mm thick liner has been selected to provide additional robustness. To have consequential leakage through a liner the system must first have defects, and the risk of defects occurring is related to the puncture resistance of the liner. This is, in turn, related to the liner thickness and the resin type. Increasing the thickness improves puncture resistance in a way that is greater than proportional. The geomembrane specification also calls for tighter tolerance on the quality of the resin than industry standards, and tighter control of the as-manufactured liner thickness. Industry norms and the standard specifications issued by industry organizations allow the thickness to vary by plus or minus 10%. However, current technology allows closer tolerances and the Coffee specifications will require that (a) the minimum thickness of any roll of geomembrane to be not thinner than minus 5% of nominal, and (b) the average of all rolls to be at least the nominal (i.e. minus zero percent). This increases the possible minimum average thickness from an industry standard of 1.50 minus 10%, or 1.35 mm, to 2.00 mm, or an increase of 48%. Puncturing is related to thickness in a greater than proportional manner, and increasing thickness by 48% probably reduces puncturing by about 70% (this is a generalization, as the precise increase depends on a wide variety of factors including the specific geometry of each stone in contact with the liner).

LLDPE has been in common use for leach pad liners for about 20 years, though it is not the most common liner for the simple reasons that it is more expensive and less available. It is, in other words, a premium product. For the Coffee installation, the singular disadvantages of LLDPE is that is has a lower resistance to prolonged and direct exposure to sunlight. However, its resistance is on the order of years and no leach pad liner material should be uncovered and exposed for longer than a few months (the summer construction season).

Overliner gravel (crushed ore) and drainage pipe

The overliner system serves two purposes: protecting the geomembrane liner from damage during ore stacking and operations, and to drain process and rinse waters out of the system in a manner that minimizes hydraulic head over the liner.

To design the overliner system for puncture protection, a series of laboratory tests were performed to verify that the liner and the selected gravel are compatible under the expected loads. The depth of ore over the liner will average less than 50 m, and the maximum will typically be 60 m with a small area receiving 80 m of ore near the end of the operational life. The laboratory tested the coarsest samples of durable ore from the metallurgical testing program of both Supremo and Latte. These had been crushed by the metallurgical laboratory to a P_{80} of 50 mm. For the puncture testing they were further screened to a P_{100} of 39 mm then tested in contact with the selected geomembrane (2.0 mm LLDPE). The subgrade for the puncture test was a GCL over an open graded minus 20 mm gravel, which is more aggressive than the actual conditions for construction.

17.3.4.2 Heap Leach Siting Trade-Off Study

At the beginning of the FS a trade-off analysis considering three sites for the leach pad was performed:

- The original impounding valley leach pad (VLP) used for the July 2014 PEA (hereinafter called the KP VLP);
- A newly-considered, non-impounding VLP (called Valley C) located in the valley west of and adjacent to the KP VLP site. A free draining configuration was selected as there is ample room for conventional ponds at this location, and external ponds are considered to be safer and lower risk than internal ponds (Breitenbach and Smith 2012). External ponds cost less to construct than internal ponds and require less waste rock (Smith and Parra 2014); and
- A conventional, free-draining flat pad located on the ridgeline to the west of the Latte pit. In the trade-off analysis, two separate pads were considered (called Pad A and Pad B) to accommodate the entire ore reserve considered in the Feasibility Study. Once the rest of the facilities were sited these two were merged into a single, larger pad (called the Ridgetop Pad).

The trade-off study concluded that the optimum site for and style of leach pad will be a conventional, non-impounding pad with a free-draining heap located on the ridge-top to the west of the ADR plant. This location and configuration provides the lowest initial capital cost, the lowest life of mine total capital cost, and the lowest risk. The ridge-top option reduces initial and LOM capital costs by $52 M and $26 M, respectively, as compared to the KP VLP option. The ridge-top option also reduces the construction schedule by one year because of the reduction in earth works and the elimination of a need for early waste rock to build a dam, which also affects the mine pre-stripping schedule and costs. Conventional free-draining heaps are also lower risk than impounding valley fill facilities that have steep construction slopes, the impoundment of significant quantities of process solutions behind a large dam, the potential for large stresses and the resulting large deformations in the liner system as the ore settles down slope against the fixed liner system, and the problems which can arise when liners leak (Breitenbach and Smith 2012). Therefore the selected leach pad configuration was a conventional, flat leach pad with a free-draining heap.

17.3.4.3 Raincoat Technology

For the initial years of operation, Years -1 to Year 3, and based on the synthetic climate model for precipitation developed by Lorax, it is anticipated that the heap leach facility has a neutral or slight negative (water shortage) water balance. As the leach pad increases in size, the amount of precipitation entering the system increases. Beginning at the end of Year 3 the water balance would become surplus (more water collected than used). To avoid a chronic surplus of water and the associated need to treat for discharge, temporary exposed geomembrane (EGC) covers or raincoats will be used over the heap. Raincoats divert meteorological water from the process circuit to a pond for storage for later use as make-up water or to be released from the site as non-contact water (Breitenbach and Smith 2007).

For the purposes of this study, the material used for cost estimating was 1.0 mm thick HDPE geomembrane. A conservative allowance for annual replacement of 25% of the installed area was used for cost estimating.

Raincoats provide the following operational advantages over other water management techniques:

- Minimize dilution of the pregnant solution, the flow volume to the ADR plant, and reagent consumption;
- Reduce the required capacity of the events ponds by reducing the area of the heap exposed to storm events, which reduces the likelihood of an unwanted environmental discharge and the volume of such a discharge should it occur;
- Reduce the required backup pump and application pipeline system capacity for recirculation of excess flows back to the heap surface as make-up water (i.e. rainwater is stored for use as make-up water in a pond adjacent to the heap);
- Reduce, avoid or postpone the cost and risk of treatment and release of surplus waters; and
- Protect weak ore from degradation and reduce fines migration within the heap due to peak flows, along with the corresponding loss of permeability. Most of the Coffee ore is highly resistant to degradation and thus this may not be a factor for the Coffee heap.

In addition, experience at pilot and demonstration plants for nickel and copper heap leaching have shown that using raincoats can increase the temperature of the ore and solution. This has been confirmed by site-specific thermodynamic modelling for Coffee Gold, and the increased ore and pregnant solution temperature in the critical winter season is about 4^{o}C.

17.3.4.4 Cold Weather Considerations

Heap leaching in cold climates requires special considerations to prevent freezing of the heap and the leach solution. Factors that impact the solution and in-heap ore temperatures include ambient temperature at the time of ore stacking, covering or burial of drip irrigation lines, use of heap covers such as raincoat or thermal covers, the temperature of the barren solution as applied to the heap, and the thermal mass of the heap (tonnes under leach, width and depth of heap). Thermodynamic modelling was performed in two stages to simulate expected ore and solution temperatures under different operating strategies. The first stage was used to determine the general sensitivity of the operating temperatures to a range of operating criteria, and the second stage of modelling to optimize the specific design parameters. Mean monthly temperatures at the site ranged from a low of -22.5ºC to a high of 13.6ºC, while in-situ ore temperatures (in the mine) range from around zero to about -1.5ºC.

The thermodynamic computer model used for Coffee Gold was originally developed by Geo-Logic Associates to simulate changes of in-heap ore temperatures over time for a demonstration copper sulphide heap (Schrauf et al., 2014). The model was shown to closely match measured temperature profiles in the heap over a 30 month period, and the model accurately simulated the impact of adding heat to the system and the placement of a geomembrane thermal cover over the heap.

The key difference between the Coffee Gold model and the original model was that the original modeling considered exothermic heat input from sulfide oxidation, which does not occur during cyanide leaching. There is a small tonnage of ore (approximately 1 – 2% of the total HLF) from the Kona pit that contains sufficient sulfide minerals to generate additional heat in the heap during oxidation, but this was not accounted for due to the small volume of sulphide-bearing ore, and because the ore will be encapsulated in an isolated cell to improve closure performance of the heap. The results of this modeling are discussed in this section and presented in more detail in the report entitled "Thermodynamic Modeling of Proposed Heap Leach, Coffee Gold Project" prepared by Geo-Logic Associates, August, 2015, which is included as Appendix D of The MINES Group 2016 design report (MINES 2016). The Coffee Gold thermal model used the following key input variables:

- Meteorological data (ambient air temperature, wind speed, solar radiation, ratio of measured vs potential maximum solar radiation, and relative humidity) specified time step used in the model (the model used two week time steps);
- Surface cover characteristics (solar absorptivity and thermal resistivity) of either exposed ore or the cover material was specified for each time step. Cover materials of both ore and geomembranes were considered in the model;
- Ore properties (heat capacity, lift thickness, lift area, and ore temperature when stacked) for each ore lift; and
- Solution properties (flow rate, heating of barren solution, total dissolved solids) specified for each time step.

Several inherently conservative assumptions were used in the model, which suggests that the forecast operating temperatures may be lower than will actually be realized. This matches the experience at Fort Knox (Alaska) and Veladero (Argentine Andes) in that both of these sites have similar climatic conditions to Coffee and those heaps are operating at warmer temperatures than the modelling forecasts for Coffee. The assumptions and simplifications included:

- The temperature of the ore stacked on the heap equals the average daily ambient temperature. This is conservative because stacking will generally not occur during the coldest hours of the day, operators can elect to not stack ore during the coldest days if solution temperatures are lower than desired, and the model ignored the solar heat gained by the ore during handling;
- The on-site weather station data suggests that during the critical winter months the site is slightly warmer than the data used for the thermodynamic modelling;

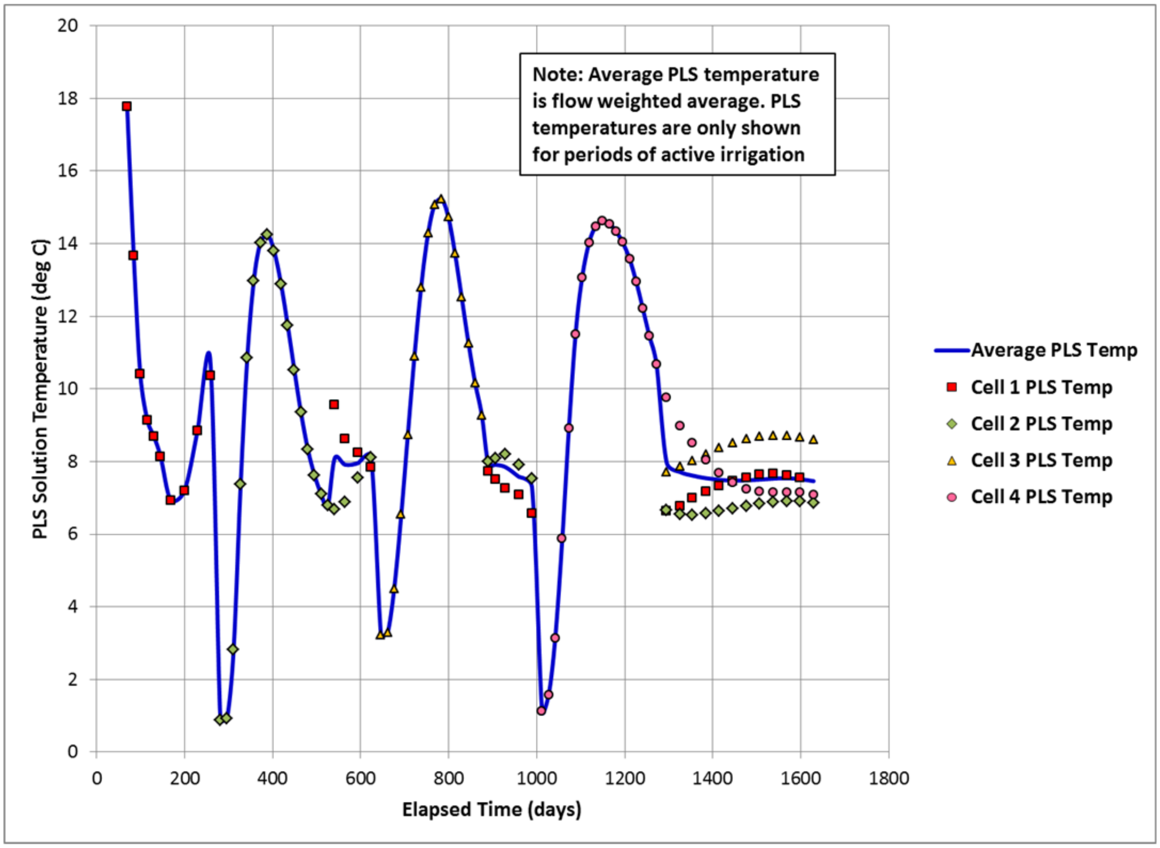
- Waste heat from the electrical generation plants will be available from April 1 to December 31 of each year. In the winter the power demand will be much lower since the crushing circuit will not be operational and most of the available waste heat will be dedicated to heating buildings. It was estimated that from April 1 to December 31, 4 to 6ºC of heating (before considering line losses) can be applied to the barren solution using power plant fugitive heat. This was ignored in the thermodynamic modelling; and,
- The exothermic acid-base reactions and sulphide oxidation were ignored in the model.

The modelling results were used to develop design and operational criteria. They indicated that a combination of seasonal ore stacking, drip line burial and heating of the leach solution will be required to maintain solution temperatures above freezing during the initial years of stacking. As the size of the heap increases and thermal covers are added in Year 3, solution heating will no longer be necessary at average annual temperature profiles. The following key design parameters were determined from the thermal modelling:

- Drip lines will be buried (or covered with a layer of crushed ore) at least 1 m deep during the period November through March each year;
- Based on experienced at other sites, and especially the recommendations from personnel at Veladero, a spare irrigation area equal to about 100% of the primary irrigation area will be provided in every winter starting in November of Year 1 (sufficient room cannot be provided in Year -1 because of the limited area of the Stage 1 pad and the trade-off between area and ore depth). This will allow irrigation to continue even in the event that the primary irrigation network freezes;
- Ore crushing and stacking will occur for 275 days per year, from April to December only;
- Solution heating November through March of each year,-1 to Year 3, by increasing the barren solution temperature by 8ºC above the pregnant solution temperature, to provide a 6^{o}C increase at the drip lines after heat loss in the piping; and
- Beginning in Year 3, raincoats will be used during the spring snowmelt and summer rainy seasons to reduce the meteoric water entering the process circuit. Modelling of the Coffee heap as well as the prior modelling and operational experience at other operations determined that geomembrane covers also act as thermal insulators. In the specific case of Coffee they increase the pregnant solution temperatures by about 4ºC in the winter. Therefore, it was decided to leave the raincoats in place in the winter to take advantage of this benefit, beginning in Year 3.

Figure 17.5 shows the modelled pregnant solution temperature by month for the first 54 months of operations. This modelling period was selected because it takes the heap through leaching and into rinsing of the first two stages of the pad, and it shows warming temperatures at the end of the modelling period. Further, Year 3 (about day 1,000 on the figure) is when raincoats will start to be used. Importantly, by this time the thermal mass of the heap is significant and the resulting internal heap temperatures less directly affected by ambient conditions.

Figure 17.5: Forecast Pregnant Solution Temperatures through Year 3



Source: Mines Group 2016

In parallel with the thermodynamic modelling, an industry practices study was also completed (Sinha and Smith, 2015). This study found that there are or have been about 70 heap leach operations in artic and subarctic climates. The authors used data from 28 of these projects to compile operating statistics for some 210 Mt per annum (Mt/a) of heap leach production, with leaching rates ranging from under 1 Mt/a to 30 Mt/a. Of the sites surveyed, the coldest months averaged –18°C and ranged from –6ºC to –31ºC; the length of the cold season ranged from four to ten months. Conventional leach pads are used at the majority of sites (57%). Dynamic heaps (on/off leach pads) and impounding VLP are used by 28% and 18% of the sites, respectively (the total exceeds 100% because one site used both conventional dynamic heaps). Table 17.4 summarizes the statistics from the mines studied.

Table 17.4: Summary of Cold-Climate Heap Leach Operations

Description	Number of Operations	Percentage of Total Number %
Total number of operations included in the study	28	100
Heap leach pad type:		
Conventional pad (*)	16	57
Dynamic heap (on/off pad)	8	28
Impounding valley leach pad	5	18
Metal mined:		
Gold	24	86
Copper	3	11
Nickel	1	4
Year-round stacking	12	43
Year-round irrigation	16	57

* One mine in the study uses both conventional and dynamic pads, hence it is included in both categories.

Source: Mines Group 2016

Key findings from this study, which influenced the design of the Coffee Gold HLP, can be summarized as:

- Winter ore stacking strongly affects the operating temperatures and most sites elect to not stack in the coldest months;
- Operational problems during the first one or two winters after commissioning are relatively common, suggesting that additional measures should be implemented such as having a larger thermal mass and a heat plant;
- Many sites with similar winter temperatures have operated successfully with solution heating only from waste electrical generation heat;
- Exposed ponds can freeze in the winter and special provisions are required to protect liners and avoid water shortages during stepped-up demand when ore stacking resumes in spring;
- Average temperature-based planning (which is generally the case) can require operational adjustments such as additional solution heating, deeper dripper burial, and a shorter stacking season, to accommodate colder-than-average winters;
- Although there are successful impounding VLPs and dynamic heaps in cold climates, the industry seems to prefer non VLPs (85% of surveyed installations). This may be partly for economic reasons as conventional pads are less expensive than VLPs (Smith and Parra, 2014), and party due to the increased risk of liner leakage and compliance issues inherent with in-heap solution storage (Breitenbach and Smith, 2012);

- For year-round irrigation, insulating and heat tracing or burying all piping including drip lines may be necessary. Providing spare irrigation area(s) for the coldest sites in case primary drip areas freeze is also a practice recommended by some successful operators. Seasonal irrigation requires pond capacity for the seasonal drainage of the heap and the Coffee modelling indicated no thermodynamic benefit to cessation of irrigation in the winter.

17.3.5 Solution Management

17.3.5.1 Barren Solution

Piping:

The barren solution will be pumped from the barren tank located in the plant to the HLP in a double-walled pipeline. Heated barren solution will be pumped at a nominal rate of 455 m^3/h (design capacity 600 m^3/h) from the plant to the northeast corner of the HLP, where it will connect into the pad distribution system. The pipeline consists of a 1,025 m 12” standard weight carbon steel carrier pipe in a 16” fiberglass reinforced plastic containment pipe buried to a minimum of 1.5 m below grade to prevent solution freezing. A leak detection system, including moisture sensing cable and associated programmable logic controller (PLC) will be installed and tied into the plant master PLC. On the pad, the barren solution will be transferred to 340 m of 18” HDPE pipe that will distribute the solution 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 for winter operations to reduce the likelihood of freezing. Beginning at the end of Year 1, the installed drip area will be doubled before the onset of winter each year to allow for backup piping in the event that the primary area freezes. In the first year of operation the starter-heap geometry does not allow for redundant piping.

Solution Heating:

Barren solution will be heated by a diesel-fired boiler located adjacent to the plant building during the colder months, November through March, to maintain the thermal balance in the HL. The boiler is designed to heat the solution by 8°C to provide a minimum discharge temperature at the drip emitters of 6°C. Further, the plant is sized to provide 50% more heat than required (i.e., 8°C at the drip emmiters). Excess heat from the generators will be used to heat the solution when available.

17.3.5.2 Pregnant Solution

The pregnant solution will flow from the pregnant solution collection area at the toe of the HLP by gravity to the plant in an 18” standard dimension ratio 7.3 carrier pipe in a 24” SDR 17 containment pipe. The pipes have been sized for a nominal flowrate of 455 m^3/h (design capacity 600 m^3/h). The pregnant solution pipes will be buried at a minimum depth of 1.5 m and run for approximately 1 km from the northern and southern sides of the HLP to the plant. These lines will be installed with a leak detection system which monitors air pressure in the annular void between the two pipes.

17.3.5.3 Rinse Solution

Rinse pipelines will be installed to handle a flow rate of 230 m^3/h. In Year 3 the rinse pipelines will be utilized to transfer rinse solution between the plant and pad. The pipelines will be installed in the same trench as the barren and pregnant solution pipelines. The rinse pipelines will provide standby piping for the barren and pregnant solutions.

17.3.6 Carbon Adsorption

The carbon adsorption circuit consists of a train of six 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 heap leach pad. The solution will flow counter- current to the movement of carbon from column 1 to column 6. 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 will be pumped to the barren tank. Cyanide solution, caustic solution, antiscalant and make-up water are added to the barren tank as needed. Barren solution will be heated to increase solution temperature by 8^{o}C before being pumped back to the leach pad in order to maintain the thermal integrity of the heap leach pad. On average, 5 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 sixth column will advance to the fifth column and then freshly reactivated carbon will be added.

17.3.7 Desorption and Gold Refining

17.3.7.1 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

1.1.1.1 Carbon Stripping (Elution)

After acid washing, the loaded carbon will be stripped of the adsorbed gold using a modified ZADRA process. The strip vessel holds approximately 5.0 t of carbon. During elution containing approximately 1 % sodium hydroxide and 0.1 % sodium cyanide, at a temperature of 140 °C and 450 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.

17.3.7.2 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 7.5 t 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 5.0 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.

17.3.7.3 Refining

Pregnant solution will flow by gravity to a secure gold room. The solution will flow through one of two 3.54 m^3 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 retorted to remove any mercury. The retort residue 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 prills for re-melting in the furnace. The gold bars will be stored in a vault located in the gold room prior to secure off-site transportation by aircraft.

17.3.8 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 0.2 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 metred directly from the totes for distribution in the plant.

Hydrated lime will be delivered to the site in bulk by trucks and stored in a 200 t lime silo. The lime will be delivered at a rate of 1.5 kg/t of ore by screw feeder onto the heap leach feed conveyor during heap loading operations.

17.3.9 Laboratory

An assay and metallurgical laboratory will be equipped to perform sample preparation and assays by atomic absorption, fire assay, and 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, 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.

17.4 Gold Production Model

17.4.1 Introduction

The estimates for ultimate gold recovery and reagent consumptions for the various mineralized material types were presented in Section 13. For reference, a summary of the distribution of the different material types in the mine plan are presented in Table 17.5.

Table 17.5: Distribution of Material Types in Mine Plan

	Ore	Au Oz. (000s)		% Tonnage and Au Distribution		
	(000s)	Contained	Recoverable	Ore Tonnes	Contained Au	Recoverable Au
Oxide	38,105	1,731	1,569	82.2	80.3	84.3
Upper Transition	5,326	269	215	11.5	12.5	11.5
Middle Transition	2,185	112	66	4.7	5.2	3.5
Lower Transition	740	45	12	1.6	2.1	0.6
Total Ore	46,356	2,157	1,862	100	100	100

Source: JDS 2015

The gold production model is developed from a combination of metallurgical testing data, the mine production schedule, the heap leach facility construction sequence (or stacking plan), and the leaching plan for the application of barren solution to the heaps.

17.4.2 General Methodology

Along with the life of mine production schedule, a life of mine heap leach stacking schedule has been generated. This details the tonnages, grades, and the associated contained and recoverable gold over the life of the mine. The steps to generate the gold production model were:

- Tracking of ore stacking on the heap leach pad by lift, contained gold, and ultimate recoverable gold on a quarterly basis.
- Solution addition per lift was used to calculate the solution: solids ratio (tonnes of solution: tonnes of ore). Based on the solution to solids ratio, the percentage of the ultimate gold recovery at that point in time was determined. (see Section 13);
    - o At the beginning of each year there will be an in-process inventory of recoverable gold remaining from two general areas:

  - The recoverable gold in the ore on the leach pad that has not been leached to completion, and
  - The recoverable gold contained in solution, carbon, and the gold refining area that has not yet been processed into doré.
- All of the in-process inventory will only be recovered after all mining and additions to the heap leach pad have ceased. At that time leaching and the process plant will continue to recover the in-process inventory.

17.4.3 Gold Leaching and Processing: Description and Timing

According to the production schedule, ore will be loaded on the heap starting in July of Year -1. The cover layer for the leach pad liner system will be produced from the crushing plant in the latter part of June of Year -1. The cover layer will comprise ore crushed to a top size, P_{100}, of 37.5 mm. The leach pad will be loaded in 10 m lifts in stages. The initial configuration of the ridge-top leach pad constructs the south and north side of the leach pad separately. Initial operations will allow lifts to be built on one side while leaching takes place on the other side. By alternating the building and leaching of the North and South sides, the practice of leaching the newest and freshest ore will be followed. To maximize solution distribution to the ore, dripper-emitter systems will not only be placed on the tops of the heaps, but also on the slopes wherever possible. Each lift will, on completion, have a new emitter system installed. Previous systems will be left in place and used for as long as they have a functional life and starting in Year 3 for rinsing the first stages of the HLP.

In Year-1, to meet thermal balance requirements, a total of 3.5 Mt of ore are needed to be in place by the end of the year. The process plant will not be operational until September of Year -1 at which time the application of leach solution and gold recovery will commence. The detailed heap loading and construction schedule (stacking plan) and the leaching plan are on a quarterly basis from Year -1 through Year 3, while Years 4 through 10 are on an annual basis. For heap loading, the values of the contained and recoverable gold have been distributed on a weighted average basis to the tonnages contained in each individual lift of the leach pad.

A summary of the heap loading schedule is presented in Table 17.6.

Table 17.6: Heap Loading Schedule

Year	Stage	Lift	Elevation (masl)	Max Height (m)	Top Area (m^2)	Volume (m^3)	Tonnage (Mt)	Au Contained (Oz)	Au to Recover (Oz)
-1	1	1-1S	1230	10	22,630	103,243	164,736	7,113	6,217
	1	1-2S	1240	10	51,927	319,956	510,528	22,043	19,267
	1	1-3S	1250	10	54,361	487,235	777,441	39,483	33,994
	1	1-1N	1230	10	31,256	212,459	339,003	14,637	12,794
	1	1-2N	1240	10	41,260	369,374	589,380	25,448	22,242
	1	1-3N	1251.6	11.6	72,507	701,240	1,118,911	60,395	51,734
Total Year -1							**3,500,000**	**169,119**	**146,248**
1	1	1-4S	1260	10	44,850	459,573	732,213	41,010	33,923
	1	1-5S	1270	10	45,948	532,017	847,634	40,289	33,093
	1	1-4N	1260	8.4	90,688	752,142	1,198,348	64,880	53,595
	1	1-5N	1270	10	54,590	569,644	907,583	44,455	36,314
	2	2-1S	1250	10	19,956	114,195	181,941	9,328	7,558
	2	2-2S	1260	10	33,485	293,865	468,199	24,003	19,450
	2	2-1N	1250	10	16,584	118,231	188,371	9,657	7,825
	2	2-2N	1260	10	43,263	298,579	475,710	24,389	19,762
Total Year 1							**5,000,000**	**258,011**	**211,520**
2	2	2-3S	1270	10	66,856	590,695	888,149	38,601	31,962
	2	2-4S	1260	10	53,131	480,145	721,930	32,336	26,181
	1	1-6S	1280	10	54,996	569,230	855,875	38,857	31,218
	2	2-3N	1270	10	64,360	507,070	762,413	33,136	27,437
	2	2-4N	1280	10	68,415	626,856	942,520	42,246	34,187
	1	1-6N	1280	10	46,561	551,431	829,113	37,653	30,246
Total Year 2							**5,000,000**	**222,829**	**181,230**
3	1+2	1290	1290	10	177,829	1,931,463	2,867,181	138,892	119,531
	1+2	1300	1300	10	129,856	1,436,763	2,132,818	105,025	90,378
Total Year 3							5,000,000	243,917	209,909
4	1+2						5,000,000	265,527	228,916
5	3						5,000,000	252,242	218,953
6	3						5,000,000	205,890	180,651
7	4						5,000,000	208,558	186,783
8	4						5,000,000	209,407	186,393
9	4						2,855,965	121,341	111,161
Total Life of Mine							**46,355,964**	**2,156,841**	**1,861,764**

Source: JDS 2016

The leaching calculations show the dates, amount of solution added to each lift, solution to solids ratio, percentage of ultimate gold recovery, and recoverable gold inventory.

At the end of Year 3 the calculations indicate that the ounces of recoverable gold added to the heap and the gold actually produced have reached equilibrium and the amount of in-process inventory has stabilized. From Year 4 to Year 8, the gold produced is taken as the amount of recoverable gold added to the heap with the inventory of gold to be recovered remaining constant. In the third quarter of Year 9, the mined ore and ROM stockpile are depleted. At that time the recovery of the in-process inventory will begin and will continue through to Q2 Year 10. It is assumed that 50% of the in-process inventory will be recovered in Years 9 and 10.

The following tables provide the details of the gold production model.

Table 17.7: Heap Construction and Leaching Schedule: Year -1

Stacking			Leaching							
Lift	Tonnage	Days	Days	Soln Tonnes	Soln/Ore Ratio	Cum Soln Ratio	Ultimate Recovery %	Au to Recovery (oz)	Au to Solution (oz)	Inventory Au in Ore (oz)
Quarter 3 Year - 1										
1N	339,003	17	12	152,460	0.45	0.45	63.0%	12,794	8,055	4,739
2N	589,380	31	12	137,437	0.23	0.23	32.6%	22,242	7,261	14,981
1S	164,736	8	18	249,316	1.51	1.51	95.1%	6,217	5,912	305
2S	510,528	26	18	228,568	0.45	0.45	62.7%	19,267	12,076	7,190
3S	146,352	8								
Quarter 4 Year - 1										
1N	339,003		33	475,200	1.40	1.85	97.2%	12,794	4,376	363
2N	589,380		33	475,200	0.81	1.04	90.6%	22,242	12,888	2,093
1N	339,003		29	417,600	1.23	3.08	100.0%	12,794	363	0
2N	589,380		29	417,600	0.71	1.75	96.7%	22,242	1,350	743
3N	1,118,911	59	29	350,465	0.31	0.31	43.9%	51,734	22,686	29,048
1S	164,736		30	432,000	2.62	4.14	100.0%	6,217	305	0
2S	510,528		30	432,000	0.85	1.29	93.6%	19,267	5,963	1,227
3S	146,352									
3S	631,089	33								
3S	777,441		30	385,354	0.50	0.50	69.4%	33,994	23,590	10,404
Summary				**Tonnes**						
Total Au to Heaps				**169,119**						
Total Recoverable Au to Heaps				**146,248**						
Total Au to Solution				**104,825**						
Total Recoverable Au in Ore Inventory				**41,423**						
Total Recoverable Au in Circuit Inventory				**10,000**						
Total In-Process Inventory				**51,423**						

Table 17.8: Heap Construction and Leaching Schedule: Year 1

Stacking			Leaching							
Lift	Tonnage	Days	Days	Soln Tonnes	Soln/Ore Ratio	Cum Soln Ratio	Ultimate Recovery %	Au to Recovery (oz)	Au to Solution (oz)	Inventory Au in Ore (oz)
Quarter 1 Year 1										
2N	589,380		90	1,296,000	2.20	3.95	100.0%	22,242	743	0
3N	1,118,911		90	1,296,000	1.16	1.47	94.8%	51,734	26,366	2,683
2S	510,528					1.29	93.6%	19,267		1,227
3S	777,441					0.50	69.4%	33,994		10,404
Quarter 2 Year 1										
3N	1,118,911		41	590,400	0.53	2.00	97.7%	51,734	1,470	1,212
4N	1,198,348	66	41	518,499	0.43	0.43	60.6%	53,595	32,465	21,130
2S	510,528		50	720,000	1.41	2.70	99.5%	19,267	1,134	93
3S	777,441		50	720,000	0.93	1.42	94.5%	33,994	8,529	1,875
4S	468,319	25								
Quarter 3 Year 1										
3N	1,118,911		15	216,000	0.19	2.19	98.3%	51,734	333	879
4N	1,198,348		15	216,000	0.18	0.61	84.2%	53,595	12,659	8,471
3N	1,118,911		37	532,800	0.48	2.67	99.5%	51,734	598	281
4N	1,198,348		37	532,800	0.44	1.06	90.9%	53,595	3,574	4,897
5N	907,583	50	37	478,345	0.53	0.53	73.8%	36,314	26,795	9,519
2S	510,528		40	576,000	1.13	3.83	100.0%	19,267	93	0
3S	777,441		40	576,000	0.74	2.16	98.2%	33,994	1,264	611
4S	468,319									
4S	263,894	15								
4S	732,213		40	532,067	0.73	0.73	85.9%	33,923	29,140	4,783
5S	495,189	27								
Quarter 4 Year 1										
3N	1,118,911		20	288,000	0.26	2.93	99.9%	51,734	222	59
4N	1,198,348		20	288,000	0.24	1.30	93.7%	53,595	1,497	3,400
5N	907,583		20	288,000	0.32	0.84	87.7%	36,314	5,040	4,479
2-1N	188,371	11	37	521,498	2.77	2.77	99.6%	7,825	7,796	29
2-2N	475,710	26	37	504,257	1.06	1.06	90.9%	19,762	17,963	1,798
3S	777,441		35	504,000	0.65	2.81	99.7%	33,994	507	104
4S	732,213		35	504,000	0.69	1.41	94.4%	33,923	2,896	1,887
5S	495,189									
5S	352,445	19								
5S	847,634		35	453,142	0.53	0.53	74.8%	33,093	24,768	8,325
2-1S	181,941	10						7,558		7,558
2-2S	468,199	26						19,450		19,450
Summary				**Tonnes**						
Total Au to Heaps				**258,011**						
Total Recoverable Au to Heaps				**211,520**						
Total Au to Solution				**205,853**						
Total Recoverable Au in Ore Inventory				**47,089**						
Total Recoverable Au in Circuit Inventory				**10,000**						
Total In-Process Inventory				**57,089**						
Au to Doré				**205,853**						

Table 17.9: Heap Construction and Leaching Schedule: Year 2

Stacking			Leaching							
Lift	Tonnage	Days	Days	Soln Tonnes	Soln/Ore Ratio	Cum Soln Ratio	Ultimate Recovery %	Au to Recovery (oz)	Au to Soln (oz)	Inventory Au in Ore (oz)
Quarter 1 Year 2										
3N	1,118,911					2.93	99.9%	51,734		59
4N	1,198,348					1.30	93.7%	53,595		3,400
5N	907,583					0.84	87.7%	36,314		4,479
2-1N	188,371					2.77	99.6%	7,825		29
2-2N	475,710					1.06	90.9%	19,762		1,798
3S	777,441					2.81	99.7%	33,994		104
4S	732,213					1.41	94.4%	33,923		1,887
5S	847,634					0.53	74.8%	33,093		8,325
2-1S	181,941		90	1,285,084	7.06	7.06	100.0%	7,558	7,558	0
2-2S	468,199		90	1,267,908	2.71	2.71	99.5%	19,450	19,357	93
Quarter 2 Year 2										
3N	1,118,911		15	216,000	0.19	3.12	100.0%	51,734	59	0
4N	1,198,348		15	216,000	0.18	1.48	94.9%	53,595	644	2,755
5N	907,583		15	216,000	0.24	1.08	91.2%	36,314	1,296	3,183
2-1N	188,371		15	216,000	1.15	3.92	100.0%	7,825	29	0
2-2N	475,710		15	216,000	0.45	1.51	95.1%	19,762	830	969
2-3N	762,413	42						27,437		27,437
3S	777,441		11	158,400	0.20	3.01	100.0%	33,994	104	0
4S	732,213		11	158,400	0.22	1.63	95.9%	33,923	489	1,397
5S	847,634		11	158,400	0.19	0.72	85.8%	33,093	3,633	4,692
2-2S	468,199		50	720,000	1.54	4.25	100.0%	19,450	93	0
2-3S	888,149	49	50	666,711	0.75	0.75	86.3%	31,962	27,570	4,392
2-4S	16,104	1								
Quarter 3 Year 2										
4N	1,198,348		10	144,000	0.12	1.60	95.7%	53,595	430	2,326
5N	907,583		10	144,000	0.16	1.24	93.3%	36,314	742	2,441
2-2N	475,710		35	504,000	1.06	2.57	99.3%	19,762	830	139
2-3N	762,413		35	458,255	0.60	0.60	84.0%	27,437	23,051	4,386
2-2N	475,710		12	172,800	0.36	2.94	99.9%	19,762	120	19
2-3N	762,413		12	172,800	0.23	0.83	87.4%	27,437	933	3,453
2-4N	942,520	51	12	116,249	0.12	0.12	17.3%	34,187	5,903	28,283
4S	732,213					1.63	95.9%	33,923		1,397
5S	847,634					0.72	85.8%	33,093		4,692
2-3S	888,149		35	504,000	0.57	1.32	93.8%	31,962	2,407	1,984
2-4S	16,104									
2-4S	705,826	39								
2-4S	721,930		35	460,684	0.64	0.64	84.6%	26,181	22,142	4,039
6S	18,321	1								
Quarter 4 Year 2										
4N	1,198,348		8	115,200	0.10	1.69	96.3%	53,595	344	1,982
5N	907,583		8	115,200	0.13	1.37	94.1%	36,314	307	2,134
2-2N	475,710		28	403,200	0.85	3.78	100.0%	19,762	19	0
2-3N	762,413		28	403,200	0.53	1.36	94.0%	27,437	1,820	1,633
2-4N	942,520		28	403,200	0.43	0.55	77.2%	34,187	20,475	7,809
6N	829,113	46	16	180,653	0.22	0.22	30.5%	30,246	9,226	21,020
4S	732,213		20	288,000	0.39	2.02	97.7%	33,923	631	766
5S	847,634		20	288,000	0.34	1.06	90.9%	33,093	1,687	3,005
2-3S	888,149		20	288,000	0.32	1.64	96.0%	31,962	691	1,293
2-4S	721,929		20	288,000	0.40	1.04	90.6%	26,181	1,567	2,473
6S	18,321									
6S	837,554	46								
6S	855,875		40	524,647	0.61	0.61	84.2%	31,218	26,284	4,934
Summary				**Tonnes**						
Total Au to Heaps				**222,829**						
Total Recoverable Au to Heaps				**181,230**						
Total Au to Solution				**181,271**						
Total Recoverable Au in Ore Inventory				**47,048**						
Total Recoverable Au in Circuit Inventory				**10,000**						
Total In-Process Inventory				**57,048**						
Au to Doré				**181,271**						

Table 17.10: **Heap Construction and Leaching Schedule: Year 3**

Stacking			Leaching							
Lift	Tonnage	Days	Days	Solution Tonnes	Soln/Ore Ratio	Cum Soln Ratio	Ultimate Recovery %	Au to Recovery (oz)	Au to Solution (oz)	Inventory Au in Ore (oz)
Quarter 1 Year 3										
4N	1,198,348		45	648,000	0.54	2.24	98.4%	53,595	1,148	834
5N	907,583		45	648,000	0.71	2.08	97.9%	36,314	1,383	751
2-3N	762,413		45	648,000	0.85	2.21	98.3%	27,437	1,180	453
2-4N	942,520		45	648,000	0.69	1.24	93.3%	34,187	5,505	2,304
6N	829,113		90	1,296,000	1.56	1.78	96.9%	30,246	20,076	944
4S	732,213					2.02	97.7%	33,923	0	766
5S	847,634					1.06	90.9%	33,093	0	3,005
2-3S	888,149					1.64	96.0%	31,962	0	1,293
2-4S	721,929					1.04	90.6%	26,181	0	2,473
6S	855,875					0.61	84.2%	31,218	0	4,934
Quarter 2 Year 3										
4N	1,198,348		12	172,800	0.14	2.38	98.9%	53,595	257	577
5N	907,583		12	172,800	0.19	2.27	98.6%	36,314	230	520
2-3N	762,413		24	345,600	0.45	2.66	99.4%	27,437	300	153
2-4N	942,520		24	345,600	0.37	1.61	95.7%	34,187	836	1,467
6N	829,113		36	518,400	0.63	2.41	99.0%	30,246	647	297
1290N.1	833,334	46	36	468,400	0.56	0.56	78.7%	34,741	27,338	7,403
4S	732,213		23	331,200	0.45	2.48	99.1%	33,923	473	293
5S	847,634		23	331,200	0.39	1.45	94.7%	33,093	1,246	1,759
2-3S	888,149		23	331,200	0.37	2.02	97.7%	31,962	561	732
2-4S	721,929		23	331,200	0.46	1.50	95.0%	26,181	1,158	1,315
6S	855,875		46	662,400	0.77	1.39	94.3%	31,218	3,139	1,795
1290S.1	833,333	45	10	94,000	0.11	0.11	15.8%	34,741	5,486	29,255
Quarter 3 Year 3										
4N	907,583		5	72,000	0.08	2.46	99.1%	53,595	98	479
4N	762,413		12	172,800	0.23	2.69	99.5%	53,595	203	276
5N	762,413		5	72,000	0.09	2.37	98.9%	36,314	114	406
5N	942,520		12	172,800	0.18	2.55	99.3%	36,314	137	269
2-3N	762,413		20	288,000	0.38	3.04	100.0%	27,437	153	0
2-4N	942,520		20	288,000	0.31	1.91	97.4%	34,187	566	901
6N	829,113		25	360,000	0.43	2.84	99.7%	30,246	219	78
6N	829,113		12	172,800	0.21	3.05	100.0%	30,246	78	0
1290N.1	833,334		12	172,800	0.21	0.77	86.5%	34,741	2,727	4,676
1290N.2	600,258	33	25	323,985	0.54	0.54	75.6%	25,024	18,909	6,115
1300N.1	233,076	13	12	158,815	0.68	0.68	85.2%	9,877	8,417	1,460
4S	732,213		25	316,067	0.43	2.91	99.9%	33,923	245	48
5S	847,634		25	360,000	0.42	1.88	97.2%	33,093	849	910
2-3S	888,149		30	432,000	0.49	2.50	99.2%	31,962	469	263
2-4S	721,929		30	432,000	0.60	2.09	98.0%	26,181	785	530
6S	855,875		25	360,000	0.42	1.81	97.0%	31,218	864	931
6S	855,875		30	432,000	0.50	2.31	98.7%	31,218	525	406
1290S.1	833,333		25	360,000	0.43	0.54	76.3%	34,741	21,011	8,243
1290S.2	600,257	33	30	395,985	0.66	0.66	84.9%	25,024	21,244	3,780
1300S.1	233,075	13								
Quarter 4 Year 3										
4N	907,583		19	273,600	0.30	2.99	100.0%	53,595	270	6
5N	762,413		19	273,600	0.36	2.91	99.9%	36,314	218	52
2-4N	942,520		27	388,800	0.41	2.32	98.7%	34,187	470	432
1290N.1	833,334		19	273,600	0.33	1.10	91.5%	34,741	1,711	2,965
1290N.2	600,258		27	388,800	0.65	1.19	92.8%	25,024	4,316	1,799
1300N.1	233,075		19	273,600	1.17	1.86	97.2%	9,877	1,181	279
1300N.2	833,333	46	27	338,800	0.41	0.41	56.9%	35,312	20,099	15,213
4S	732,213		19	273,600	0.37	3.28	100.0%	33,923	48	0
5S	847,634		19	273,600	0.32	2.20	98.3%	33,093	356	555
2-3S	888,149		27	388,800	0.44	2.94	99.9%	31,962	234	29
2-4S	721,929		27	388,800	0.54	2.63	99.4%	26,181	373	158
6S	855,875		19	273,600	0.32	2.63	99.4%	31,218	217	189
1290S.1	833,333		19	273,600	0.33	0.87	88.1%	34,741	4,108	4,135
1290S.2	600,257		27	388,800	0.65	1.31	93.7%	25,024	2,208	1,571
1300S.1	233,075		19	259,616	1.11	1.11	91.7%	9,877	9,058	819
1300S.2	833,334	46	27	338,800	0.41	0.41	56.9%	35,312	20,099	15,213

Summary	Tonnes
Total Au to Heaps	**243,917**
Total Recoverable Au to Heaps	**209,909**
Total Au to Solution	**213,542**
Total Recoverable Au in Ore Inventory	**43,414**
Total Recoverable Au in Circuit Inventory	**10,000**
Total In-Process Inventory	**53,414**
Au to Doré	**213,542**

Table 17.11:Summary of Annual Gold Production

	Yr -1		Yr 1				Yr 2				Yr 3				Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4							
Oz. Au Produced	23,305	71,520	27,109	43,599	74,456	60,690	26,915	34,749	56,557	63,051	29,291	41,673	77,613	64,965							
Yearly Au Produced	94,825		205,853				181,271				213,542				228,916	218,953	180,651	186,783	186,393	137,868	26,707
Summary of Au Distribution																					
Total Oz to Heaps	169,119		258,011				222,829				243,917				265,528	252,242	205,890	208,558	209,408	121,341	
Recoverable Oz to Heaps	146,248		211,520				181,230				209,909				228,916	218,953	180,651	186,783	186,393	111,161	
Oz Recovered	94,825		205,853				181,271				213,542				228,916	218,953	180,651	186,783	186,393	137,869	26,708
Recovery of Total Oz	56.1%		79.8%				81.3%				87.5%				86.2%	86.8%	87.7%	89.6%	89.0%	113.6%	
Recovery of Recoverable Oz	64.8%		97.3%				100.0%				101.7%				100.0%	100.0%	100.0%	100.0%	100.0%	124.0%	
Total Oz to Heaps	169,119		427,131				649,959				893,876				1,159,404	1,411,645	1,617,535	1,826,093	2,035,500	2,156,842	
Recoverable Oz to Heaps	146,248		357,768				538,998				748,907				977,823	1,196,776	1,377,427	1,564,210	1,750,603	1,861,764	
Oz Recovered	94,825		300,678				481,949				695,492				924,408	1,143,361	1,324,012	1,510,795	1,697,188	1,835,056	1,861,764
Recovery of Total Oz	56.1%		70.4%				74.2%				77.8%				79.7%	81.0%	81.9%	82.7%	83.4%	85.1%	86.3%
Recovery of Recoverable Oz	64.8%		84.0%				89.4%				92.9%				94.5%	95.5%	96.1%	96.6%	96.9%	98.6%	100%
Oz In-Process Inventory	51,423		57,090				57,049				53,415				53,415	53,415	53,415	53,415	53,415	26,708	0

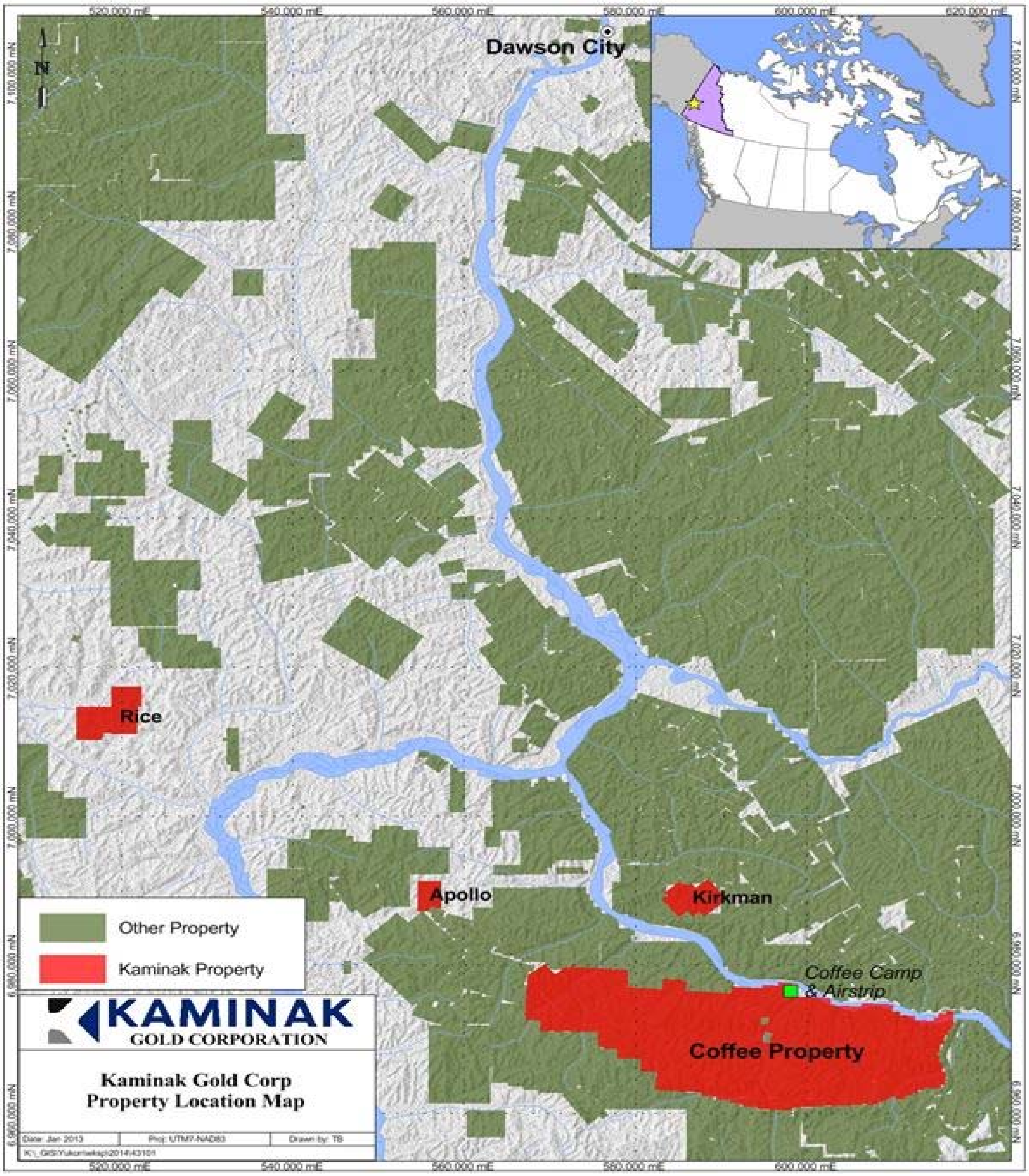
18 Project Infrastructure and Services

18.1 Overview and Design Criteria

The Coffee Gold Project is located in west-central Yukon, within the Whitehorse Mining District, Canada, 130 km south of Dawson. The location of the Coffee Gold site is shown in Figure 18.1.

The Coffee site will be accessible by an all-weather access road originating in Dawson for approximately 295 days per year and by air on a year-round basis. Due to the remote nature of the site, additional infrastructure is required for access, power generation, consumable storage and accommodations. The Coffee site layout is shown in Figure 18.2.

Figure 18.1: Location Map of the Coffee Gold Property



Source: Kaminak 2016

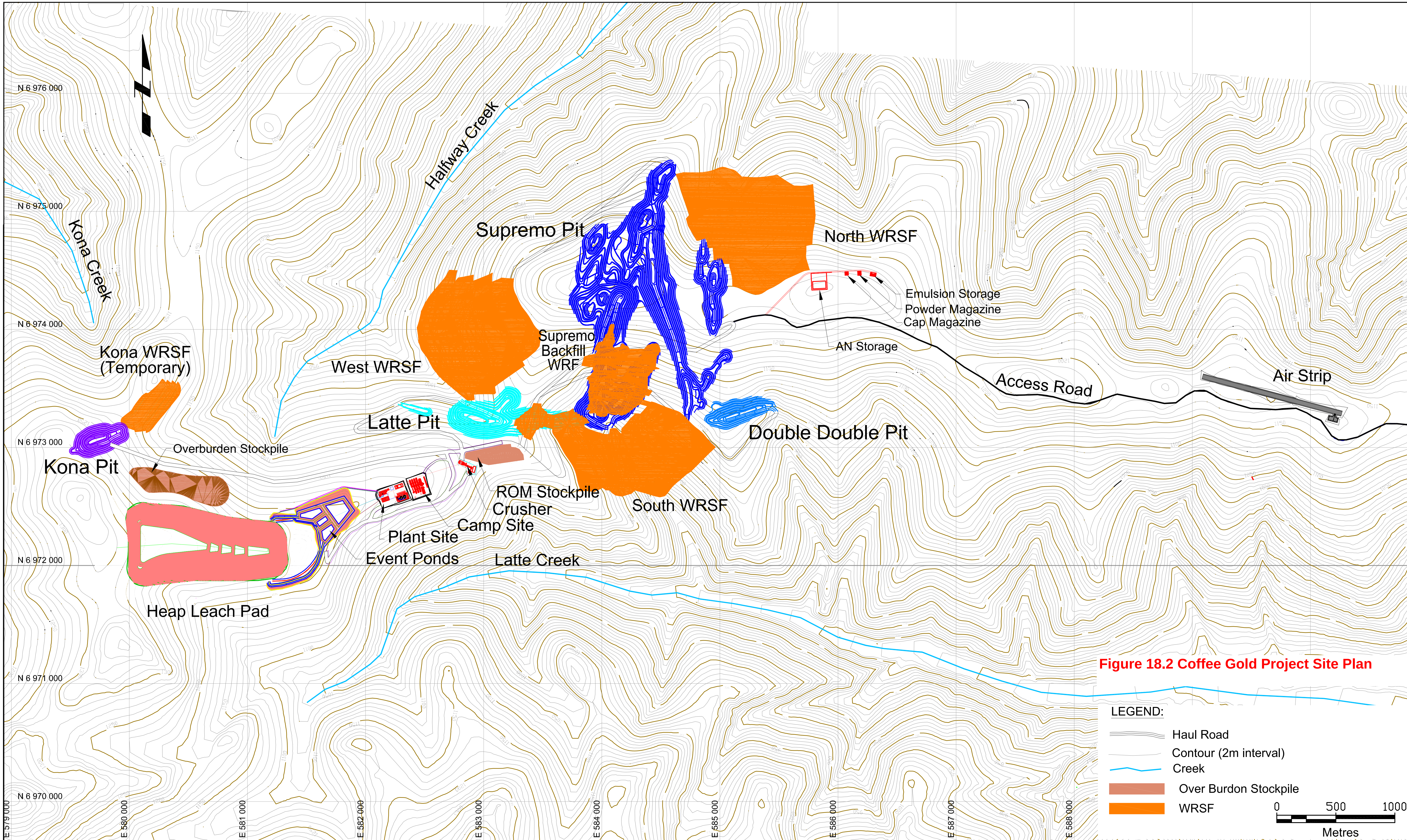


Figure 18.2 Coffee Gold Project Site Plan

18.1.1 General Infrastructure Design Criteria

The design of the facilities was dictated by the northern location of the site and the 10-year mine life. Accordingly, the facilities will be of an appropriate quality to ensure safe and reliable operation, while being fit-for-purpose to minimize cost. Some examples of resultant design elements that derived from this philosophy included:

- Use of pre-fabricated structures wherever appropriate to reduce onsite direct and indirect costs;
- Use of fabric-covered buildings wherever practical;
- Use of compacted-fill floors where appropriate;
- Minimize Project and building footprint to optimize heating, heat recovery and piping/power distribution;
- Minimize the difference in elevation and the horizontal distances between the open pits, mill site, crushing plant, airstrip, and heap leach facility, with the intent of minimizing the capital and operating costs for truck haulage, roads, earthworks, and pipelines between these elements;
- Structural painting only where needed for protection or safety (e.g. fuel tanks not painted);
- Respect environmental design requirements such as set back from water courses;
- Consider site water management requirements;
- Consider traffic management and safety;
- Consider climatic conditions; and
- Consider local wind patterns with respect to noise, dust, and other atmospheric emissions.

18.1.2 Foundation Soil and Permafrost Conditions

Geotechnical investigations conducted to characterize the subsoil conditions at the primary infrastructure sites included sonic core drilling of soils, test pit excavation, diamond core drilling, and laboratory test programs (SRK 2016a). The results of these investigations and subsequent analyses were used to provide the basis for geotechnical design recommendations.

The Project site is located in the northern Dawson Range of the Yukon-Tanana Terrane in an area that did not experience widespread glaciation. The landscape evolved through erosional and periglacial processes. The topography generally consists of rounded ridges with incised v-shaped valleys (AECOM, 2012).

The site infrastructure will be located on the ridge-top east of the heap leach facility (Figure 18.2). The ridgetops and upper slopes are generally dominated by in-situ residual soils and colluvium derived from weathering of bedrock. The colluvial material is variable and typically contains mixtures of gravels, sands and silts with organic materials in the upper 0.1 to 0.2 m layer. The ridgetop soils are up to approximately 1.8 m deep and generally ice-poor. The thickness of the strongly weathered bedrock is variable but is generally less than a metre.

The Coffee site is located in an area of discontinuous permafrost. The permafrost is deepest on the ridgetops and north facing slopes at the site. Permafrost depth beneath the infrastructure site area is estimated at approximately 70 to 80 m below ground surface (Lorax 2016). One shallow thermistor string installed near the crusher pad indicates a permafrost temperature of approximately - 0.3 °C with an approximately 2 m active zone.

18.1.3 Infrastructure Foundation Preparation Recommendations

The camp and plant sites will be constructed on cut-and-fill pads as shown in Figure 18.3. The plant site will have cuts of up to approximately 6 m at the uphill (northwest) corner and fills of up to approximately 12 m on the downhill (southeast) corner. The camp site will have cuts of up to 3.5 m at the uphill (northwest) corner and fills of up to 10 m on the downhill (southeast) corner.

Critical structures that cannot tolerate differential settlements such as the process plant will be founded on competent bedrock. The currently planned cut depths are anticipated to reach fresh, competent bedrock. However, this will be confirmed for those critical facility locations during detailed design studies. Drilling and blasting will be required for the fresh bedrock and possibly for the upper weathered bedrock and soils if excavated in winter. Frozen soils and/or weathered rock will not be suitable for reuse as structural fill and will be placed in thin lifts in one of the waste rock storage facilities.

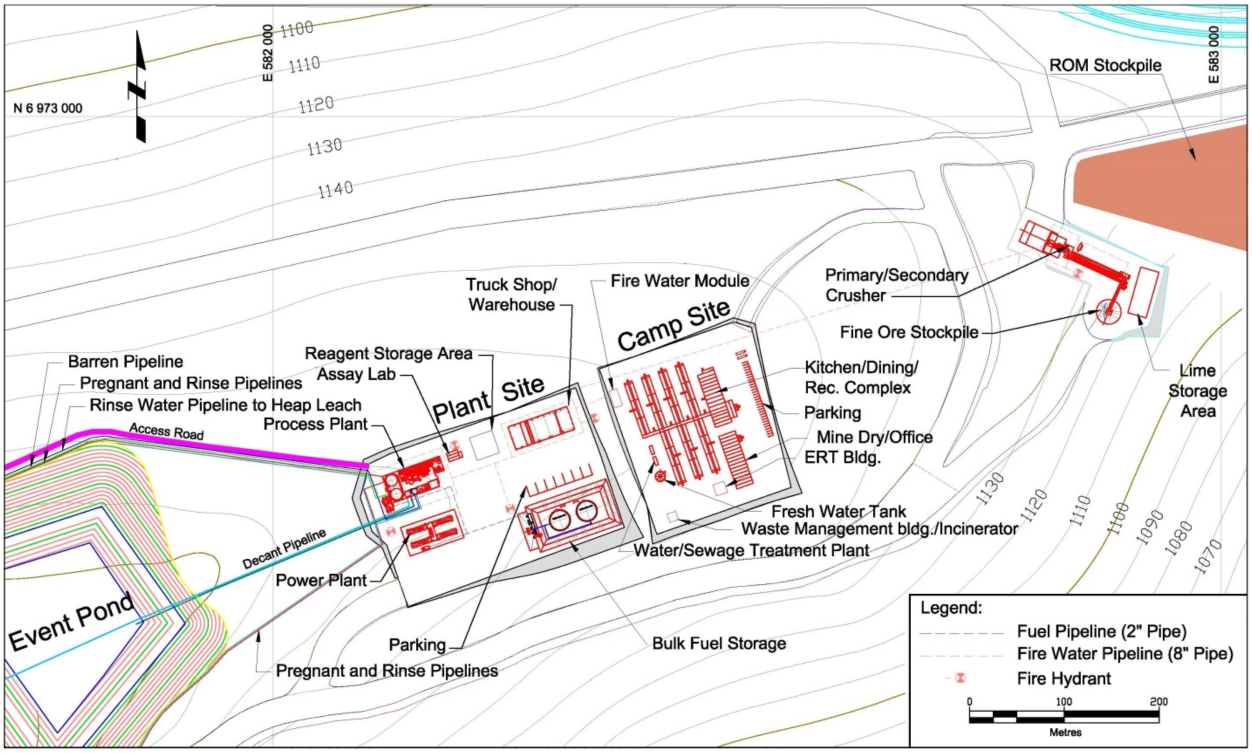
Non-critical structures that are able to tolerate minor differential settlements will be designed on fill sections of the pads. Fill will consist of free-draining, coarse, granular materials, and preferably angular durable rock-fill to prevent build-up of excess pore pressures. Where structural fill is to be placed on an existing natural slope, the fill will be keyed into the natural slope by excavating steps into the slope at the edge of successive lifts of structural fill. Rock-fill pads will be constructed in lifts no greater than 1.5 m with the maximum rock size limited to 0.9 m. Engineered slopes constructed of structural or rock-fill will be made at a gradient of 2H:1V or flatter. Buildings will be set back a minimum of 10 m from the crest of fill slopes.

18.2 On-Site Infrastructure

As much infrastructure as possible is located close to the process plant in order to make the operation of the site efficient, especially during the northern Canadian winters. The major infrastructure at the Coffee site is:

- Crusher system;
- Heap leach facilities and event ponds;
- Process-related facilities;
- Truck shop and warehouse;
- Mine dry and office complex;
- Power plant;
- Utilities;
- Permanent camp (established for the construction phase);
- Storage areas;
- Airstrip;
- Waste rock storage facilities;
- Fuel storage farm and containment;
- Industrial waste management facilities such as the incinerator; and
- Site water management facilities.

Figure 18.3: Coffee Gold Site Infrastructure

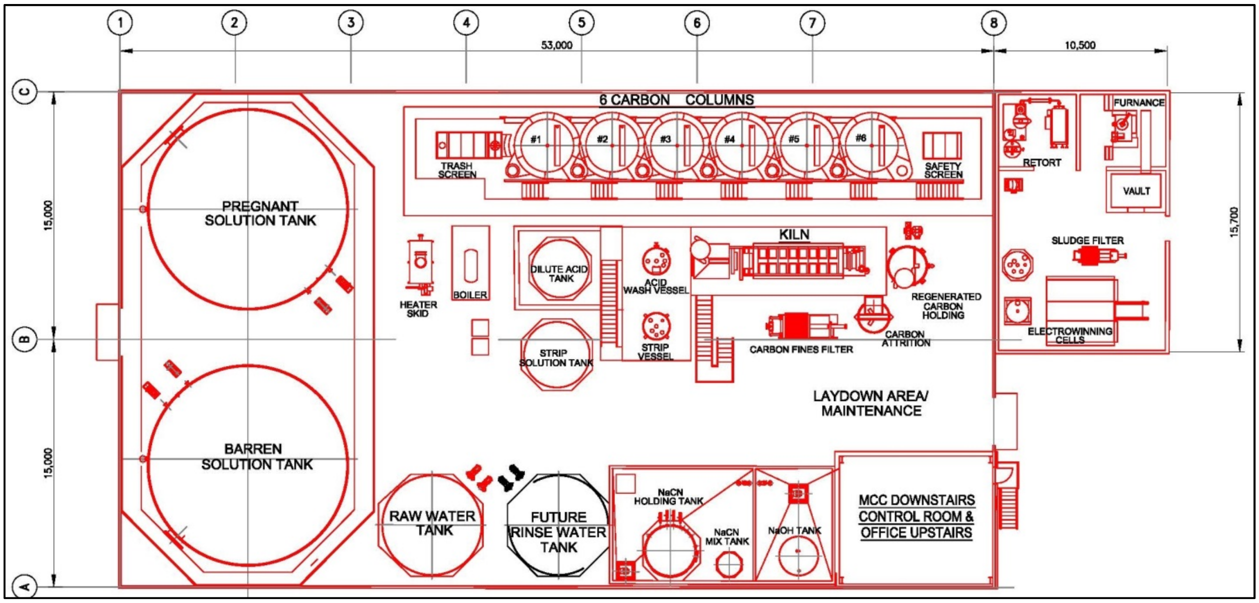


Source: JDS 2016

18.2.1 Process Building

The process plant is located in a pre-engineered building. The ADR building will measure 33 m wide by 54 m long by 17 m tall and come equipped with a 10 t overhead crane for equipment maintenance. The refinery building will measure 11 m wide by 16.5 m long by 7 m tall. Figure 18.4 shows the layout of the process building. The building is heated to 10°C by glycol air handlers and unit heaters.

Figure 18.4: Process Building Layout



Source: JDS 2016

18.2.2 Truck Shop and Warehouse Building

The truck shop complex at the Coffee site will consist of a 65 m long by 21 m wide by 12 m tall structural steel, pre-engineered building designed to accommodate facilities for repair and maintenance of mining equipment and light vehicles. The building will also house warehouse storage space for spare parts, consumables and other materials and equipment. The truck shop area breakdown is provided in Table 18.1.

Table 18.1: Truck Shop/Warehouse Floor Areas

Description	Area (m^2)	Comments
Service Bays	1,248	3 truck bays, + 1 wash bay each 13 m wide x 24 m deep
Warehouse	312	13 m wide x 24 m deep

Source: JDS 2016

The service bays are designated for the service and repair of the major mining equipment which includes 144 t haul trucks and 11.5 m^3 front-end loaders. The facilities will include automatic hose reels in one bay for dispensing engine oil, transmission fluid, hydraulic oil, air, solvent, diluted coolant, and grease. The truck shop will be equipped with a 10 t overhead crane that will be accessible by all service bays. The building is heated to 10°C by glycol air handlers and unit heaters.

Tire repair will be done outside, weather permitting. In poor weather, tire repair will be done in the shop with the appropriate safety measures, such as personnel access control and clearances.

18.2.2.1 Laydown Area

Laydown areas for major process plant consumables are located to the west of the truckshop building and at the crusher area. Spare parts that do not require protection from the elements can be stored in the area south of the reagent storage area.

A separate construction laydown area has not been designated but the plant area pad was developed to allow for sufficient space around the infrastructure to store materials and equipment. Should additional storage area for construction materials be required, the area east of the camp site may be utilized.

18.2.3 Mine Dry and Office Complex

The 1,070 m^2 mine dry and office complex will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will rest on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will be provided to connect the mine dry and office complex with the camp core facilities and rooms.

The mine dry facility will service construction and operations staff during the life of the Project. It will be capable of servicing 85 workers during shift change and contain the following:

- Male and female clean and dirty lockers; and
- Showers and washroom facilities with separate male and female sections.

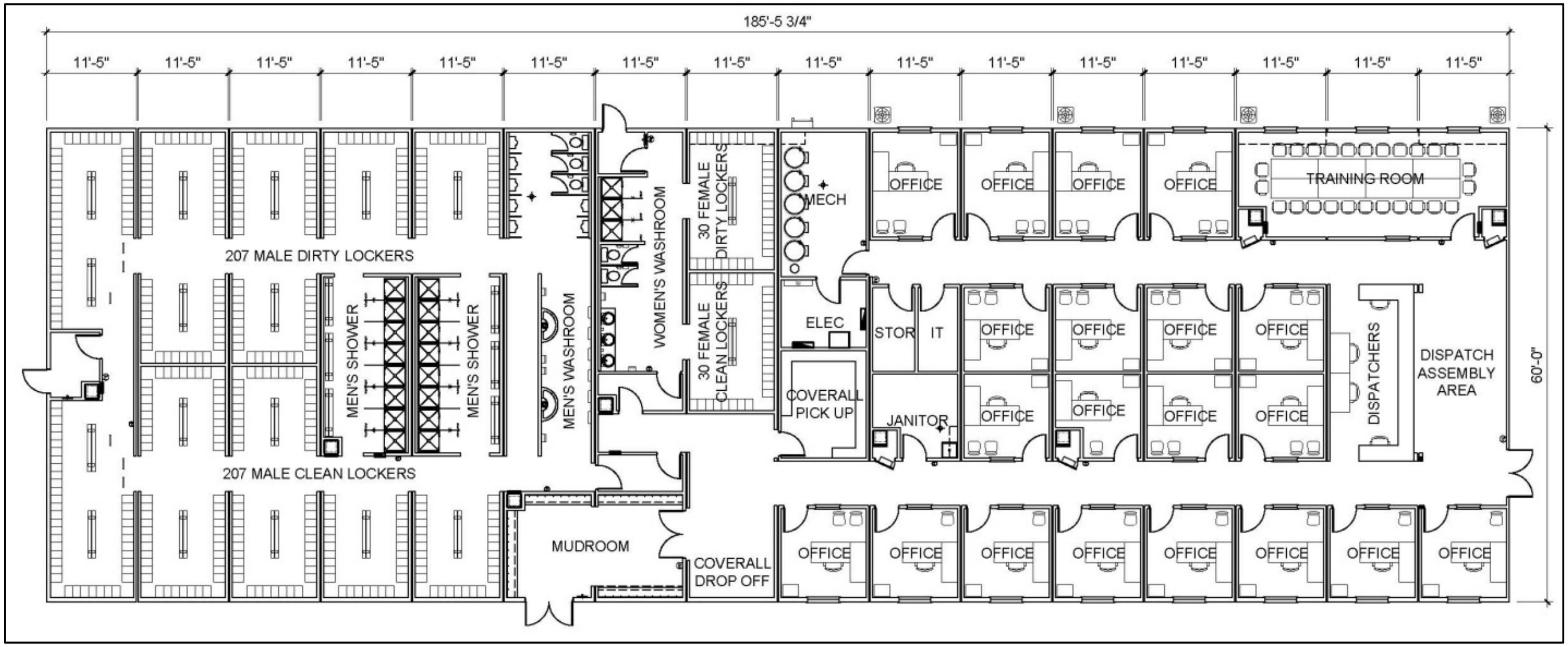
A male : female ration of ~6:1 was assumed.

The site office facility will contain the following items:

- Private offices;
- Main boardroom; and
- Mine operations line-up area.

A layout of the mine dry/office complex can be found in Figure 18.5.

Figure 18.5: Mine Dry and Office Complex Layout



Source: JDS 2016

18.2.4 Camp

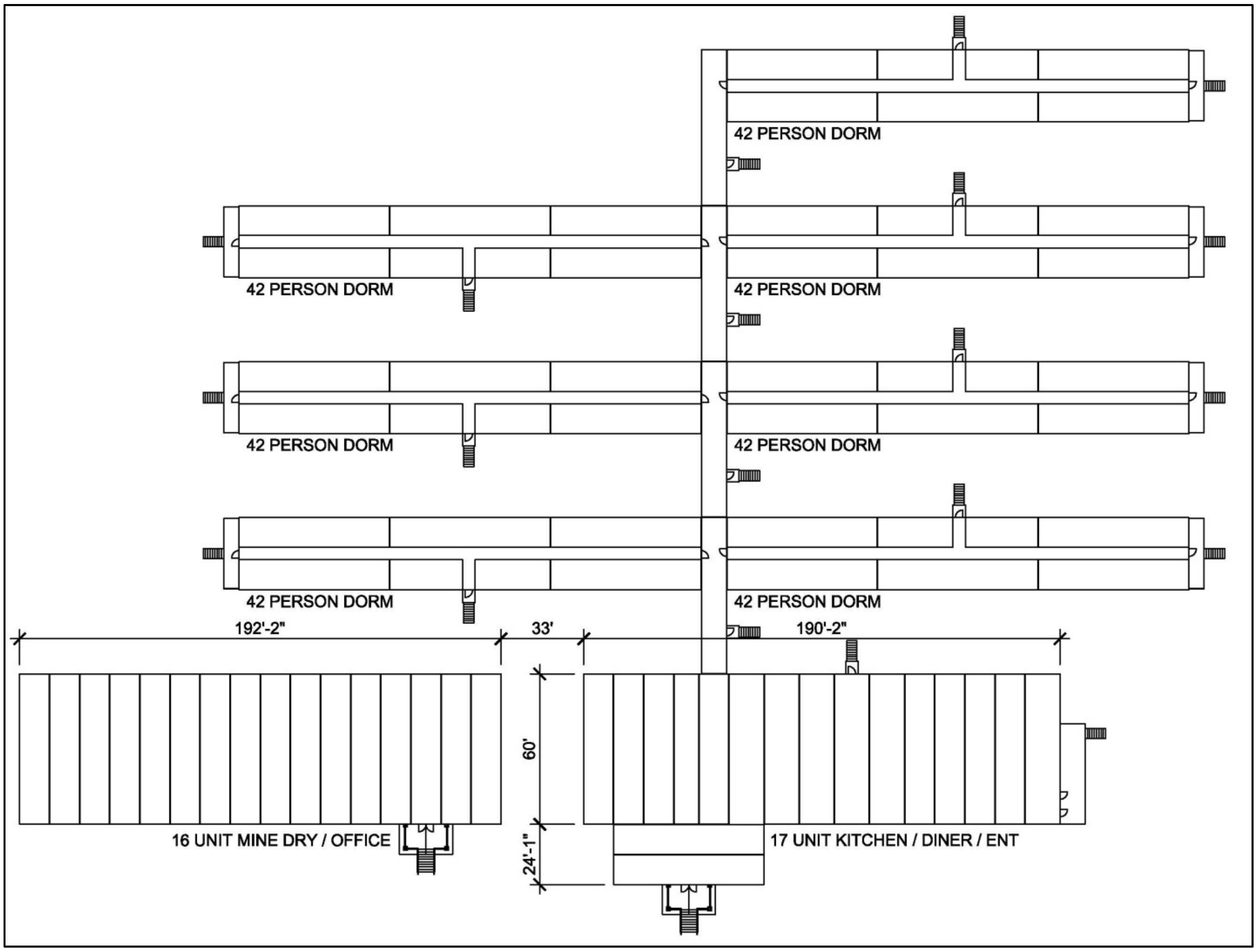
The camp will comprise single-occupancy rooms with central washrooms. It will be used during the construction phase and throughout the operations phase. There will be seven dormitory wings, each capable of housing 42 people for a total of 294 beds.

The kitchen/dining/recreation complex will include the following:

- Kitchen complete with cooking, preparation and baking areas, dry food storage and walk-in freezer/cooler. The kitchen will be provided with appropriate specialized fire detection and suppression systems;
- Dining room with serving and lunch preparation areas;
- First aid room;
- Mudroom complete with coat and boot racks, benches and male-female washrooms;
- Housekeeping facilities;
- Reception desk and lobby; and
- Recreation area.

The general camp layout can be found in Figure 18.6.

Figure 18.6: General Camp Layout



Source: JDS 2016

The camp will be constructed from modular units manufactured off-site in compliance with highway transportation size restrictions. Camp modules will rest on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will connect the main camp complex and dormitory wings.

18.2.5 Fuel Storage

Fuel storage capacity has been designed for a ten week period of diesel consumption at full production to supply mining and ancillary equipment, and power generation. This capacity accommodates the periods when no fuel deliveries by road to site are possible during the winter freeze-up of the Stuart and Yukon Rivers (about six weeks) and the spring thawing of the rivers to allow barge crossings to commence. Table 18.2 provides the annual fuel consumption for the Coffee site.

Table 18.2: Coffee Site Fuel Usage

Area	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
	'000 L									
Process & Infrastructure (including power generation)	7,721	7,721	7,721	6,663	5,103	5,103	5,103	5,103	3,051	489
Mining Operations	22,212	25,244	22,996	24,614	25,849	26,483	27,745	25,918	8,720	0
Surface & Infrastructure	652	651	652	651	652	651	652	651	643	133
Earthworks	83	42	123	16	11	27	15	20	0	0
Total Fuel	**30,667**	**33,657**	**31,492**	**31,945**	**31,616**	**32,265**	**33,516**	**31,692**	**12,414**	**621**

Source: JDS 2016

Year 2 activities consume the maximum fuel, a 10-week supply during this year results in an 8 ML total diesel fuel storage requirement. A detailed fuel consumption breakdown for a 10-week period during winter freeze-up for Year 2 is provided in Table 18.3.

Table 18.3: Year 2 Diesel Storage Requirements

Area	'000 L
Process & Infrastructure	2,286
Main Power Plant Average Power Consumption	1,012
Process Plant - Diesel (Boiler and Kiln)	57
HL Barren Solution Heating	1,213
Incinerator Fuel Consumption	4
OP Mining - Operations	4,855
Site Support	125
Total Diesel Storage (10 weeks):	7,266

Source: JDS 2016

Two 4 ML field-erected fuel tanks will be constructed at the Coffee Site in Year -2. The fuel tank farm bund will be lined with HDPE for spill containment. Fuel dispensing equipment for mining, plant services, and freight vehicles will be located adjacent to the fuel tank bund and the fueling area will drain into the bund. A fuel transfer module will provide fuel to the power plant day tank and diesel consumers in the Process Plant.

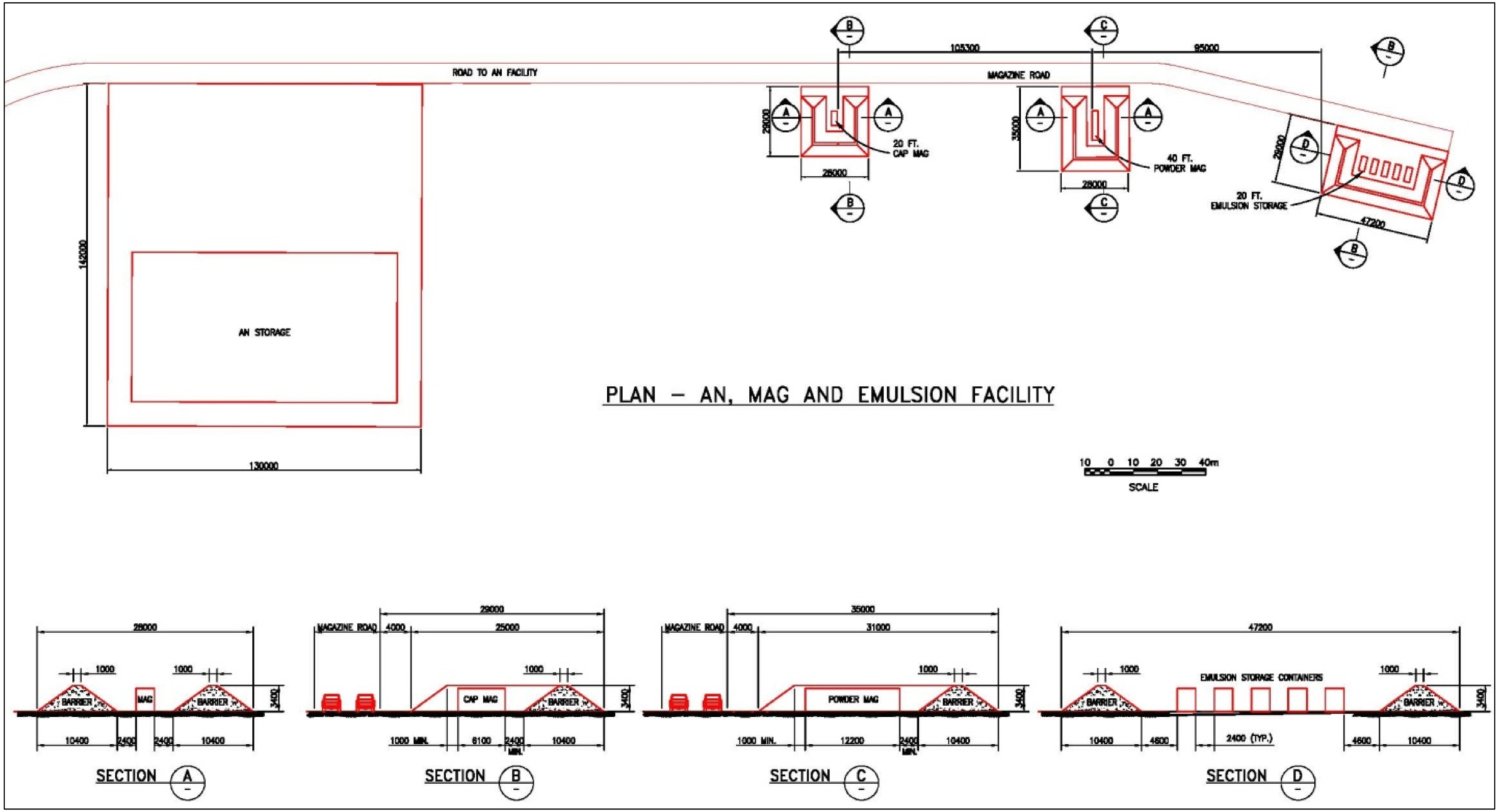
18.2.6 Explosives Storage and Preparation

Explosive storage at the Coffee site consists of three main components:

- Bulk ammonium nitrate (AN) storage;
- Bulk emulsion storage; and
- Explosive storage magazines.

The layout of the explosive storage area is shown in Figure 18.7.

Figure 18.7: Explosive Storage Layout



The design of all storage facilities will meet government regulations and will be located according to required separation distances as regulated by the Explosives Regulatory Division (ERD) of Natural Resources Canada (NRCan). The quantity-distances table provided in ERD's "Q-D Principal Manual", along with the maximum storage capacity, were used to determine the required minimum separation distances that are outlined in Table 18.4.

Bulk AN prill will be shipped to site either in 25 t bulk transport trailers or in 1 t tote bags. During the periods when the all-weather access road is available, AN prill will be shipped in trailers and stored in the AN silo at the AN facility. Sufficient quantities of AN prill in tote bags will be stockpiled at the AN facility prior to the close of the access road during freeze-up and break-up to allow for ongoing blasting operations during these periods. AN tote bags will be moved from the AN storage area and transferred into the AN silo via an auger system as required.

The AN storage area is sized to allow for ten weeks of storage or a maximum of 2,050 t of AN prill. The storage area will be lined with a HDPE liner to provide spill containment.

Bulk emulsion product required for open pit blasting will be shipped to site in 20 t tanker trailers and offloaded either into the 60 t emulsion silo located at the AN storage facility or into 20 ft ISO containers located at the emulsion storage area.

Packaged explosives and explosive detonators will be stored in approved explosive magazines located on separate pads. The powder magazine will be a 40 ft container magazine capable of holding 32 t of explosives while the cap magazine will be a 20 ft container magazine capable of holding approximately 600 cases of detonators. Each magazine will be surrounded on three sides with earthen berms to prevent propagation and significantly reduce the separation distances from other areas of the operation.

Table 18.4: Explosive Quantity - Distance Requirements

	Maximum Quantity (t)	Q-D Classification	Separation Distance (m)
AN Storage to Inhabited Building	2,050	Scale Factor = 9.6	968
AN Facility to Inhabited Building	60	D7	610
AN Facility to Powder Magazine	32	D4	265
Magazine to Inhabited Building	32	D7	730
Magazine to Blasting Operations	32	D5	485
Inter-Magazine Distance	100	D2	115
Emulsion Storage to Inhabited Building	100	D7	1,040
Emulsion Storage to Blasting Operations	100	D5	690
Emulsion Storage to AN Facility	100	D2	375

Source: JDS 2016

18.2.7 Waste Management

A 9 m wide by 9 m long waste management building will be provided to house the site incinerator and for sorting waste and recyclable materials. Food waste from the kitchen facilities will be segregated and burned daily in the incinerator to limit wildlife attraction associated with the disposal of food waste.

The incinerator is dual-chamber and diesel-fired capable of burning the daily site waste. Design capacity is based on a maximum of 294 people, each generating 3 kg of waste per day, or a total of 880 kg/day.

All hazardous waste will be back-hauled to Whitehorse via the all-weather access road for disposal at an approved facility. There will be sufficient 'backhaul' space available on transport trucks returning to Whitehorse.

Recyclable waste is also expected to be back-hauled by transport trucks to suitable off-site recycling facilities for proper disposal.

Non-hazardous, non-leaching, inorganic garbage will be collected and disposed of within an on-site landfill, to be located in the North WRSF.

A land farm for storing and treating hydrocarbon-contaminated soil will also be located in the North WRSF.

18.2.8 Ancillary Structures

An assay laboratory will be located adjacent to the process plant. This facility will serve the plant assay, environmental, and metallurgical requirements. The laboratory consists of pre-fabricated modules and ancillary equipment, such as drying ovens, dust and fume control, and heating equipment.

A 9 m wide by 9 m long Emergency Response Team building will be located adjacent to the mine dry/office complex to house the site's ambulance and fire truck.

18.2.8.1 Utilities and Services

18.2.8.1.1 Sewage Treatment Plant (STP)

Sewage will be treated by a membrane bioreactor (MBR) plant that will be constructed, assembled and tested prior to shipment to site. A sludge drying system will also be provided in a separate 40 ft container. The dewatered sludge will be disposed of in the incinerator.

The treatment plant will include influent screening, an equalization/bioreactor tank (to handle the daily peaks in flow), a membrane system, a treated effluent storage tank and UV disinfection. The treated effluent will be regularly tested prior to being discharged to the surrounding environment.

18.2.8.1.2 Fresh/Fire Water

Fresh water is planned to be transported by water truck from the Yukon River to the fresh and fire water tank at the camp site. Fire water will be stored at the bottom of the tank while the fresh water will be drawn off the top portion of the tank.

18.2.8.1.3 Potable Water

Water will be pumped from the fresh and fire water tank to the potable water treatment plant. The plant will be contained in a 20 ft shipping container assembled prior to shipment to site. It will contain the complete treatment system including filtration, UV disinfection and chlorine disinfection.

Treated water from the potable water plant will be stored in an insulated and heated potable water storage tank which accommodates the potable water demand variances and then distributed to the camp and mine dry facilities. The plant has a capacity of 90,000 L/day and is sized for 300 people based on a consumption of 275 L/person/day.

18.2.8.1.4 Heating, Ventilation, Dust Control, and Fume Extraction

The heating plant planned for the Coffee Project is an amalgamation of three systems. During normal operation the building heat load is provided through genset waste-heat recovery units. This genset waste-heat recovery accounts for one of the systems. When two gensets are operational there is 4,000 kW of recoverable heat available. In the winter when the crushing plant is not operating there will be only one genset in operation and it will provide 2,000 kW of recoverable heat. Typical heat recovery efficiency is 80%, meaning that the minimum recoverable heat is 1600 kW.

Using the design criteria it was calculated that the maximum building heat load is approximately 3,500 kW at a temperature of -49ºC. Consequently there will be a heat demand shortfall under most winter conditions. The breakeven ambient temperature where no additional building heat is required is -32ºC with both gensets running.

To make up the additional heat demand two additional heating systems will be employed. For the heat demand in the camp a modular boiler room will be installed in parallel with the camp heating system. This boiler is sized for 1,800 kW and will be primarily used to boost the heat to the camp but will be able to provide full backup if the gensets are not able to operate. Also, given the modular design of the camp boiler plant it may be repurposed in the future.

The third portion of the heating system is the heat captured from the barren solution boiler. The barren solution boiler is sized so that it can heat the barren solution and also provide heat to the process plant building. Redundancy for the process plant heating comes from a truck shop bypass line that allows the truck shop to be isolated from the genset waste-heat recovery loop. By isolating the truck shop the heat can be directed to the process plant if the barren solution boiler is taken off line.

Continuous ventilation will be provided for all personnel-occupied spaces, as well as select unoccupied spaces. Ventilation rates will vary depending on the level of occupancy and the intended use of the space, in accordance with applicable codes and standards. Ventilation systems will include make-up air units for continuous supply of tempered air, exhaust fans to provide the required number of air changes per hour, and localized exhaust fans to remove fumes, where required. The process plant includes dust control and fume extraction systems.

18.2.8.1.5 Site Communications

Site-wide communication design will incorporate reliable communications systems to ensure that personnel at the Project site have adequate voice, data, and other communication channels available.

Communications will be facilitated by satellite internet connectivity, Voice Over Internet Protocol (VoIP) phones, UHF radio and a wireless network. A trunked radio system consisting of handheld, mobile and base digital radios will provide wide-area communications coverage.

18.2.8.1.6 Fire Protection System

The Coffee site facilities will be protected, at a minimum, from fire in accordance with applicable codes and standards. The fire alarm system will consist of manual pull stations at building exits and audible and visual notification devices throughout the work areas.

All surface mobile equipment will be fitted with fire extinguishers. The fleet of open pit mining equipment will also contain fire suppression systems.

The firewater main, hydrant, and standpipe system will service the Coffee site facilities by a fire water tank and modularized pump unit. The fire water pump system will include a main pump and jockey pump which are electrically powered and a diesel-driven standby pump. The fire water pumping system will be housed in a modular building. A fire water truck will provide supplemental protection. All buildings and conveyors will have fire extinguishers and some will have standpipe systems and fire truck connections. There are no sprinkler systems inside the truckshop or the process plant.

The power generators, because they are large diesel engines, will each have a Liquid Vehicle System (LVS) fitted. The camp, Administration Offices and Mine Dry will be fitted with sprinklers connected to the firewater main.

18.2.8.2 Power Supply

18.2.8.2.1 Power Generation

A single captive power plant will be used to meet the electrical power demand necessary to support the Coffee Gold operation. The power plant comprises four diesel-fired reciprocating engine generator sets (gensets) in a N+2 (2+2) arrangement. Each generator will be prime rated for 2.250 MW running at 1,800 rpm and generating power of 4,160 V. The peak gross power is 4.5 MW (two operating gensets @ 100%; 2 x 2.25 MW). The estimated electrical loads are shown in Table 18.5.

Table 18.5: Coffee Site Electrical Load

	Connected (kW)	Peak (kW)	Average Annual (kW)	Average Summer (kW)	Average Winter (kW)	Energy Annual (kWh)
Ore Crushing And Handling	1,578	1,219	669	886	8	5,860,723
Process Plant	2,603	1,227	825	805	886	7,226,565
On-Site Infrastructure	1,344	833	684	681	692	5,987,931
Grand Total	5,525	3,278	2,178	2,371	1,585	19,075,219

Source: Allnorth 2016

The power plant will be modular with all gensets interconnected. Each genset will be packaged in a walk-in, sound-attenuated enclosure that is constructed, assembled and tested prior to shipment to site. Each genset package will include:

- One generator providing 2,250 kW @ 1,800 rpm;
- One plate and frame heat exchanger with associated controls;
- One exhaust gas waste-heat boiler;
- Local engine control;
- Local motor control centre for module load;
- One engine module complete with ventilation, exhaust system and blanketing; and
- Fire suppression.

The power plant will also include one electrical building, complete with the following:

- Four generator breakers with protective relays;
- One grounding package with neutral grounding;
- One station service feeder and transformation; and
- MCC power distribution.

Given the modular and expandable nature of the power plant, two gensets will be mobilized in Year -2 to provide construction power. An additional genset will be added in Year -1 in advance of plant start-up and the fourth genset will be added in Year 1 to achieve the N+2 configuration.

18.2.8.2.2 Power Distribution

The power plant includes all switchgear and control equipment to link the generators. This equipment includes 4,160 V switchgear for the generators and process plant feeders, load-sharing systems, neutral grounding equipment, surge suppression, local and master control systems, and all necessary low-voltage distribution equipment for power plant ancillaries.

Power will be distributed throughout the plant site at 4,160 V. There will be five distribution feeders, including two spare positions.

The electrical loads at the primary crusher area will be fed with a 4,160 V overhead power line. Seasonal open pit dewatering will be handled by diesel pumps eliminating the need for long-distance, high-voltage transmission lines.

18.2.8.2.3 Power Plant Opportunities

The diesel engines will be equipped with a natural gas conversion kit that will allow diesel fuel to be substituted with up to 70% liquid natural gas (LNG). Due to the low cost of diesel fuel at the time the Feasibility Study was completed, the power generating cost difference between natural gas and diesel fuel was not significant enough to justify the additional $1.5M capital expense associated with LNG storage and vapourization. If in the future diesel fuel costs increase, significant power generation costs savings may be realized by substituting LNG for diesel.

18.2.8.3 Site Roads

The road network for the Coffee site will consist of haul roads and service roads.

In general, site roads will be constructed with embankment fills. The embankment material will be rock material sourced from infrastructure earthwork activities or from open pit waste material. Appropriate thicknesses of road bed material will be utilized according to the existing ground conditions.

The Coffee site will have 25 km of external pit haul roads and 3 km of service roads (not including the all-weather site access road). Service roads will be used for smaller vehicles (i.e. light trucks) to access ancillary infrastructure such as the airstrip, AN storage facility, and camp site.

The haul road design criteria are described in Section 16 while service road design criteria are as follows:

- Design vehicle: light/medium truck;
- Minimum width of travelling surface: 8 m;
- Design speed: 50 km/h;
- Side slopes: 2H:1V;
- Maximum grade: 8%;
- Safety berms (fills > 3 m in height): 0.5 m.

18.2.8.4 Airstrip

The airstrip is a vital component of the Coffee site infrastructure as air transportation is the primary means for mine personnel and incidental freight transportation.

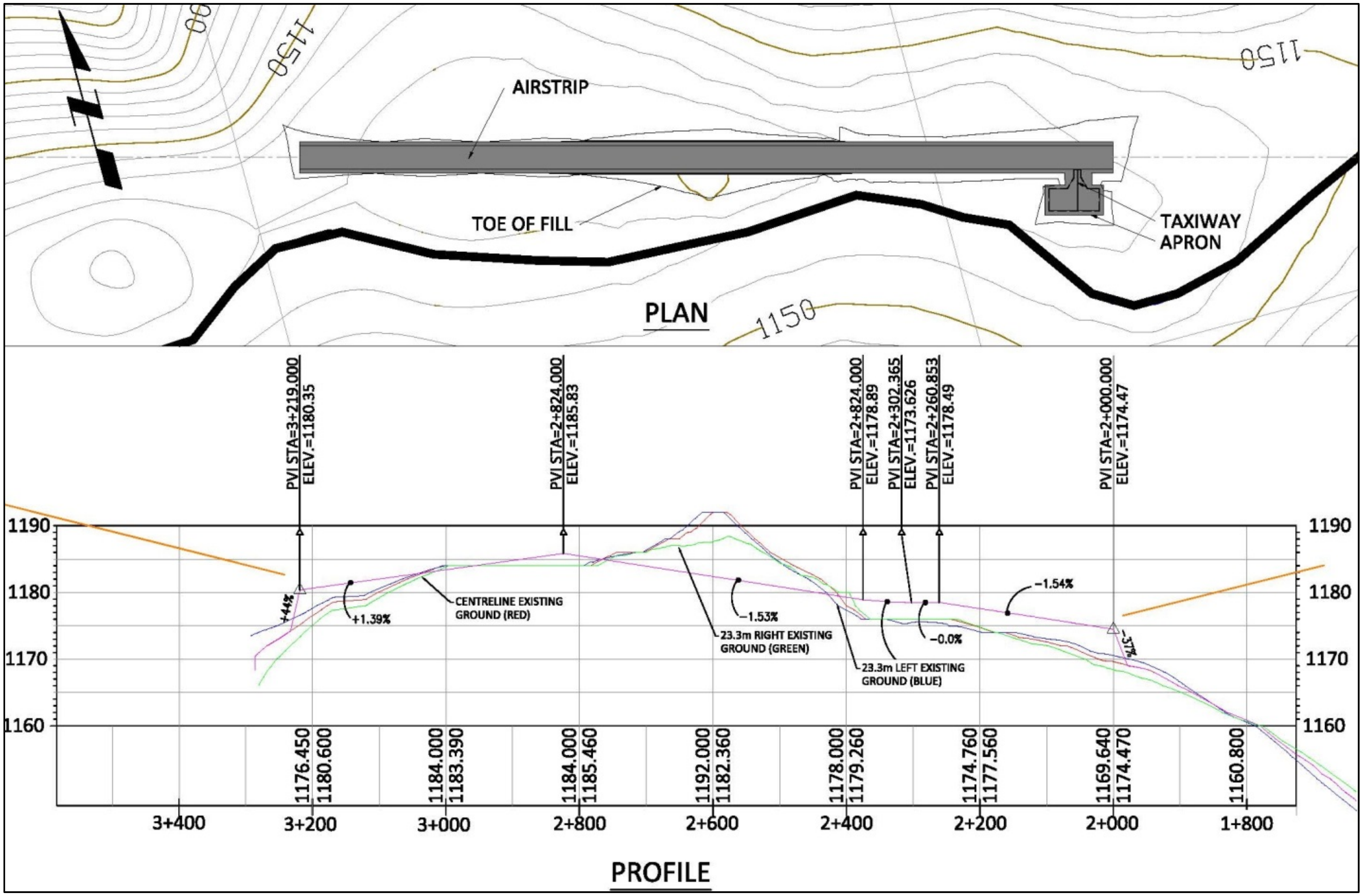
18.2.8.4.1 Airstrip Design and Construction

The Coffee airstrip is designed to handle turboprop passenger aircraft similar in size to a Hawker Siddeley 748. The airstrip is also sufficiently sized to handle cargo aircraft up to a de Havilland DHC-5A Buffalo. The airstrip will be designated as “registered”, thereby allowing for the use of approved charter aircraft without having to comply fully with Transport Canada’s standards as set out in “TP312E Aerodrome Standards and Recommended Practices”.

The Coffee airstrip will be constructed to a 1,220 m (4,000 ft) length and 35 m width. The taxiway to the apron area is 10 m wide with 6 m graded areas along each side. The apron is suitably sized for two aircraft to maneuver and park. The airstrip is to be constructed from cut-and-fill material local to the airstrip. Fill material will be placed in 300 mm lifts and compacted. The airstrip graded areas will be capped with a minimum layer of 300 mm of 19 mm minus granular crushed rock.

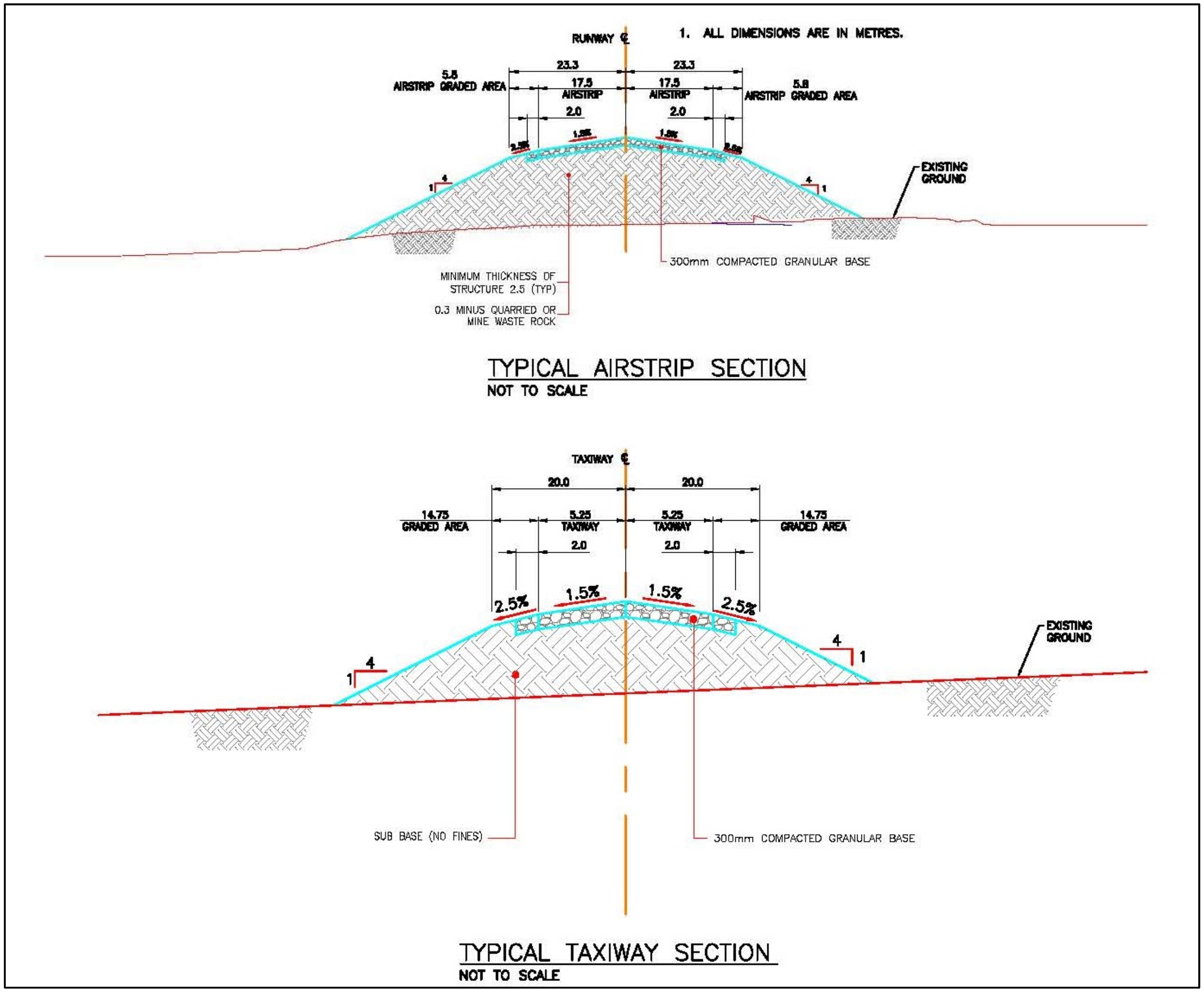
The airstrip plan and profile are shown in Figure 18.8. Figure 18.9 provides typical cross sections of the airstrip and taxiway.

Figure 18.8: Airstrip Plan and Profile



Source: JDS 2016

Figure 18.9: Airstrip and Taxiway Typical Sections



Source: JDS 2016

The obstacle limitation surfaces (OLS) for the Coffee airstrip are based on the use of turboprop aircraft.

18.2.8.4.2 Airstrip Navigation Equipment

The Coffee airstrip will be equipped with a GPS Instrument Approach system allowing for instrument flight rules (IFR) approaches and departures under suitable weather conditions. The airstrip lighting package will include:

- Runway edge lighting;
- Taxiway edge lighting;
- Precision approach path indicators (PAPI); and
- Omni-directional approach lighting system (ODALS).

The lighting controls will allow pilots to control the lighting remotely via radio (ARCAL) and also allow operation by ground personnel.

A pre-fabricated modular operations centre will be located on the airstrip apron and will contain all electrical services and controls for the airstrip. The operations centre will contain radio equipment for ground-to-air communications. A four element automated weather observation station (AWOS) will be located alongside the operations centre and provide wind direction and speed, visibility and ceiling readings to pilots during landing and take-off.

The primary power supply for the airstrip will be from a dedicated diesel generator located at the flight operations centre.

18.2.8.4.3 Airstrip Ancillary Equipment

Site support equipment such as a grader, dump trucks, loaders and a compactor will be used for airstrip surface maintenance and snow removal. The mine's water truck will be utilized to spread chemical dust control product during the summer (drier) months. A pick-up truck within the site support equipment fleet will be utilized for measuring the runway surface friction index.

Passengers arriving at the Coffee site will be transported directly from the aircraft to the camp facilities using the site's crew buses. Departing passenger will then be returned from the camp to the aircraft. During the period when passengers are being transferred the support crew will unload any baggage and freight and transfer it to the camp or warehouse facilities.

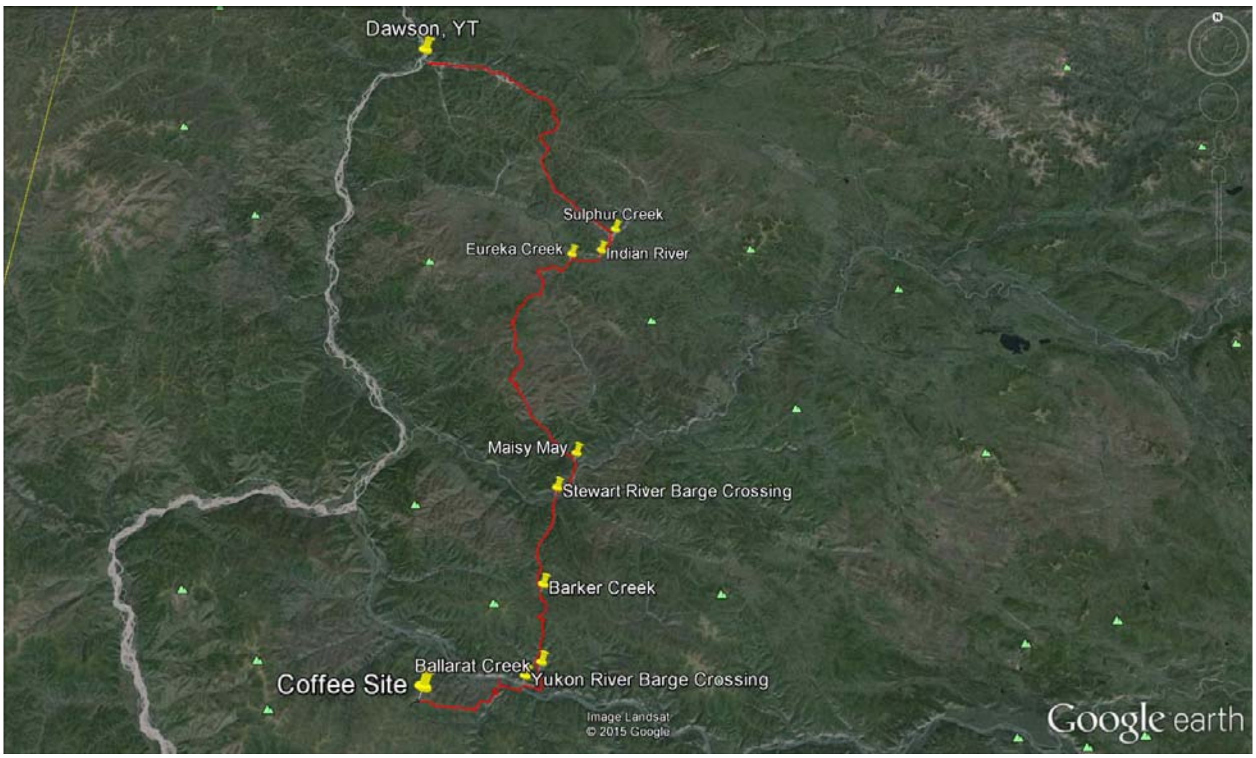
A licensed radio operator will be made available prior to any flight leaving its origin and will remain in contact with the aircraft until it has landed and departed. This will allow the operator to relay weather and friction index information to the pilots.

18.3 Access Road

18.3.1 Introduction

The all-weather access road between Dawson and the mine site will form a critical component of the Project infrastructure that will be used to provide equipment, fuel, and other supplies during construction and operations. Forestry-road type construction will commence in Year -3 with substantial completion prior to the start of site construction in Year -2. Figure 18.10 shows the intended route of the road.

Figure 18.10: All-weather Access Road Alignment



Source:

The access road to the site will originate 16 km outside of Dawson, YT at the junction of Hunker Creek Road. The road will initially follow existing government maintained roads up to Sulphur Creek. Once past Sulphur Creek the road will generally following existing roads utilized by placer miners with some upgrades and realignments required to meeting design criteria and make the road suitable for year-round access.

The access road will cross two major rivers; the Stewart River and the Yukon River. During periods of open flow barges will be utilized to move transport trucks across each river. When frozen, ice roads will be constructed to allow access across the rivers. No river access will be possible during the spring thaw and fall freeze-up periods each year. Logistics and storage of fuel and consumable materials during these periods has been considered with respect to storage and laydown areas.

The winter ice road construction of river crossings will begin in early December of each year. Once the road is constructed and deemed suitable for hauling, the road will be monitored and maintained to ensure safe and continuous operation until the end of April.

18.3.2 Road Alignment

Table 18.6 provides a description and distance of each section of the access road.

Table 18.6: Access Road Section Distances

Road Section	Distance (km)	Description of Work
Dawson to Sulphur (Gov't maintained road)	62.7	Improve drainage, raise subgrade in areas, rebuild soft spots.
Indian River	12.9	Follows existing placer roads, minor re-alignment to maintain design criteria.
Eureka Creek	6.4	Follows existing placer roads, 2 km re-alignment Sta 16+800 to 18+800 to improve gradient.
Henderson Ridgetop	33.5	Follows existing placer roads, 11km re-alignment Sta 27+300 to 38+300 and 2 km re-alignment Sta 42+200 to 43+400 to maintain design criteria.
Maisy May	22.5	Follows existing placer roads to Sta 73+800, then new road to 75+200.
North Bank Stewart	8	New road: entire section from Maisy May to North Bank of Stewart crossing.
Barker	18.9	Follows existing placer roads, minor re-alignment to maintain design criteria.
Barker-Ballarat	16.7	New road entire section.
Ballarat	7.8	Follows existing placer roads to Sta 124+000, New road to North Bank of Yukon crossing.
Winter road	4.1	New road construction from the south shore of the Yukon River to be used annually during the winter road season.
Yukon River to Coffee Site	19.8	Existing access road from south bank of Yukon crossing to site – no significant construction required.
Total Distance (Dawson to Site)	**213.3**	

Source: Onsite 2016The access road will cross two major rivers: the Stewart River and the Yukon River. Each river crossing will have a barge to ferry transport trucks across the river during the open water season. The access road intersects several major streams and creeks which will be crossed using either clear span bridges or culverts. Table 18.7 provides a list of the major stream crossing locations and the type of structure to be installed.

Table 18.7: Major Stream Crossing Structures

Crossing	Distance from Dawson (Km)	Structure Type
Sulphur Creek	68	12.2 m Long Bridge
Indian River	69.9	27.4 m Long Bridge
Eureka Creek	76.7	3,600 mm x 18m Long CSP
Maisy May Trib 1	120	1,800 mm x 17m Long CSP
Maisy May Trib 2	121.5	1,500 mm x 14m Long CSP
Maisy May Trib 3	121.1	1,500 mm x 17m Long CSP
Maisy May Trib 4	127.9	12.2m Long Bridge
Maisy May Proper	137.8	18.2m Long Bridge
Lower Barker	147	21.3m Long Bridge
Mid-Barker Camp	157.3	9.1m Long Bridge
Mid-Barker Tailings 2	158.7	12.2m Long Bridge
Upper Barker Trib 1	162	2,400 mm x 10m Long CSP
Upper Barker Trib 2	163.3	2,400 mm x 14m Long CSP
Upper Barker	165.1	12.2m Long Bridge
Upper Ballarat	176.1	2,400 mm x 10m Long CSP
Mid-Ballarat	181.4	9.1 m Long Bridge
Lower Ballarat Trib	182.7	9.1 m Long Bridge
Ballarat Proper	182.9	15.2 m Long Bridge

Source: Onsite 2016

18.3.3 Design Criteria

Table 18.8 provides the design criteria used for the land portion of the access road.

Table 18.8: Access Road Design Criteria – Land Portions

	Unit	Value
Design Speed (Valley Bottoms)	km/hr	50
Design Speed (Mountainous Terrain)	km/hr	30
Finished Road Width	m	5
Surfacing Depth	mm	300
Minimum Curve Radius	m	35
Minimum Stopping Sight Distance	m	65
Maximum Favourable Road Grade (Sustained/Short Pitch)	%	8/10
Maximum Adverse Road Grade (Sustained/Short Pitch)	%	8/10
Minimum Switchback Radius	m	18
Maximum Switchback Grade	%	8
Design with Intervisible Turnouts		Yes

Source: Onsite 2016

Table 18.9 provides the design criteria used for the ice crossings on the Stewart and Yukon Rivers.

Table 18.9: Access Road Design Criteria - Ice Crossings

Operating Speeds	Unit	Value
Loaded trucks on ice	km/hr	25
Loaded trucks on land	km/hr	30
Loaded trucks on/off portages	km/hr	10
Empty trucks on ice	km/hr	35
Empty trucks on land	km/hr	55
Road Width on ice	m	10

Source: JDS 2016

Table 18.10 provides minimum ice thickness and total allowable weight for various vehicle configurations.

Table 18.10: Load Limits at 100% of Highway Legal Gross Vehicle Weight

Vehicle Configuration	Minimum Ice Thickness (m)	Total Allowable Weight (kg)
2-Axle Hotshot	0.66	14,600
3-Axle Hotshot	0.73	22,500
6-Axle Tractor Trailer	0.89	46,500
7-Axle B-Train	0.96	56,500

Source: JDS 2016

18.3.4 Access Road Operations

The access road will be operated on a year-round basis with the exception of periods when the Stewart and Yukon Rivers are either freezing-up in the fall through early winter or breaking-up in the spring. There are two distinct operating seasons for the road; one during periods of open water flow in the two major rivers and the other during the winter months when the rivers are frozen. During the open water period barges will be utilized to ferry transport trucks delivering fuel and dry freight across the Stewart and Yukon Rivers. During the winter months, when the rivers are frozen, ice crossings will be constructed to allow transport trucks to drive across the rivers.

The access road is expected to be open an average of 295 days per year with barge service beginning each year in late May and being suspended at the beginning of November. A six week period is expected for the river to freeze up. Natural freezing of the river will be facilitated or augmented by the pumping of water on the surface ice to enhance ice thickening rates. This is anticipated to allow hauling on the ice crossings to commence in mid-December of each year. The ice crossings are expected to be in operation until late April when ice thickness will no longer be sufficient to support heavy equipment. Break-up is anticipated be about four weeks from the last use of the ice road before barge services will be able to resume. Operating periods are based on historical averages from barge and ice crossings over various rivers in the Yukon (Dawson Ferry and the Minto Mine barge operation) as well as data obtained from ferry and ice crossings in the Northwest Territories.

Freight and fuel will be transported to the Coffee site annually on the access road beginning in Year -2. Freight and fuel quantities have been estimated for the life of the mine and are shown in Table 18.11.

Table 18.11: Annual Access Road Freight and Fuel Quantities

	Freight (t)	Fuel ('000 L)
Year -2	7,336	3,289
Year -1	17,836	14,340
Year 1	21,644	30,667
Year 2	21,791	33,657
Year 3	22,615	31,492
Year 4	21,597	31,945
Year 5	24,010	31,616
Year 6	21,584	32,265
Year 7	24,469	33,516
Year 8	20,756	31,692
Year 9	5,946	12,414
Year 10	221	621
Total LOM	**209,806**	**287,514**

Source: JDS 2016

Freight and fuel will be transported to the Coffee site by trucking contractors using standard on-highway trucks. Freight hauling will primarily utilize tridem flat deck trailers for annual consumable transportation and a variety of trailer configurations for oversize equipment. Freight hauling is expected to average 25 t per load. Fuel hauling will utilize standard tridem tankers with a 46,500 L capacity. Annual freight loads have been estimated based on the above load factors and are listed in Table 18.12.

Table 18.12: Annual Truck Loads

	Freight Loads	Fuel loads	Total Loads
Year -2	294	71	365
Year -1	714	309	1,023
Year 1	866	660	1,526
Year 2	872	724	1,596
Year 3	905	678	1,583
Year 4	864	687	1,551
Year 5	961	680	1,641
Year 6	864	694	1,558
Year 7	979	721	1,700
Year 8	831	682	1,513
Year 9	238	267	505
Year 10	9	14	23
Total LOM	**8,397**	**6,187**	**14,584**

The barges are estimated to operate an average of 158 days per year, except during the estimated 6-week (42 days) road closure awaiting freeze-up. With the estimated six week (42 days) road closure during freeze-up, approximatley 200 days or 55% of the annual loads will be transported to site during the open water season.Barges will operate on a Monday-to-Friday, day shift only basis. Each barge will be operated by a certified captain and have a labourer assistant. Table 18.13 provides the annual loads to be transported during the open water season along with the average weekly and daily load requirements. Barges will be self-propelled deck barges with a minimum net tonnage of 40 t. The deck length will be a minimum of 27 m with a maximum draft of 1.2 m.

Table 18.13: Open Water Season Truck Loads to Site

	Total Loads	Weekly Average	Daily Average
Year -2	200	8.9	1.8
Year -1	561	24.9	5.0
Year 1	836	37.0	7.4
Year 2	875	38.8	7.8
Year 3	867	38.4	7.7
Year 4	850	37.7	7.5
Year 5	899	39.8	8.0
Year 6	854	37.8	7.6
Year 7	932	41.3	8.3
Year 8	829	36.7	7.3
Year 9	277	12.3	2.5
Year 10	0	0.0	0

Source: JDS 2016

The ice crossings are estimated to be open for haulage an average of 137 days per year. With the estimated 4-week (28 days) road closure during break-up, approximately 165 days or 45 % of the annual loads will be transported to site during the winter season. The access road will be open to traffic on a Monday-to-Friday, day shift only basis.

Table 18.14 provides the annual loads to be transported during the winter season along with the average weekly and daily load requirements.

Table 18.14: Winter Season Truck Loads

	Total Loads	Weekly Average	Daily Average
Year -2	165	8.4	1.7
Year -1	462	23.6	4.7
Year 1	690	35.3	7.1
Year 2	721	36.8	7.4
Year 3	716	36.6	7.3
Year 4	701	35.8	7.2
Year 5	742	37.9	7.6
Year 6	704	36.0	7.2
Year 7	768	39.2	7.8
Year 8	684	34.9	7.0
Year 9	228	11.6	2.3
Year 10	23	1.2	0.2

Source: JDS 2016

Fuel off-loading at the Coffee site will be performed by the haul truck drivers utilizing pumping systems at the bulk fuel storage facilities. Freight off-loading will be performed by the site support crew's heavy equipment operators with assistance from the haul truck drivers and labourers. Existing site support equipment located at site will be utilized to off-load freight trucks. Any freight that has been identified for removal from site will be loaded onto empty freight trucks.

18.3.5 Access Road Maintenance

The majority of the day-to-day maintenance of the access road will be performed by the site support crew. A dedicated motor grader will operate on the road on a Monday-to-Friday, day shift only basis. A third party contractor will supply and deliver aggregate material for re-surfacing and sanding of passes during the winter. During periods of extended adverse weather, road maintenance activities may need to be increased. Additional equipment from the site support and mining fleets may be utilized during these periods.

Construction of the ice crossings over the Stewart and Yukon Rivers are planned to begin construction in mid-November starting in Year -2. At this time it is expected that the rivers will be frozen and able to support light tracked-equipment.

It has been assumed that ice crossings will be constructed by an experienced contractor. The contractor will provide all labour and equipment to construct the ice crossings and open the winter road section of the access road from the south bank of the Yukon River to intersect the existing access road between the Yukon River and the Coffee site.

Prior to construction, ice profiles will be reviewed and permissible equipment weights will be approved. Snowcats, with a low bearing-pressure, will work behind the profiling crew, clearing the snow along the ice alignments. Once the snow is cleared from the ice, flooding will commence in order to increase ice thickness as quickly as possible. Portages will be roughed in on the initial pass and a final grooming will take place once the crews have the entire route opened to design widths.

River crossings constructed on ice will adhere to Fisheries and Oceans Canada requirements for ice bridges and snow fills as well as DFO "Under-Ice Water Withdrawal Protocols".

The road will be open to haul vehicles once all the portages are constructed. An independent ice engineer will verify ice thicknesses and provide loading charts for the various vehicle configurations.

Road maintenance during the winter season will focus on the following tasks:

- Maintaining road widths and repairing damage to the ice sheets as required;
- Sanding of the portages as and when required;
- Ice profiling every second day until the road reaches 100% capacity, then weekly; and
- Rescue and recovery work as required.

Once the winter season is complete each year, maintenance crews will perform the following tasks to decommission the road:

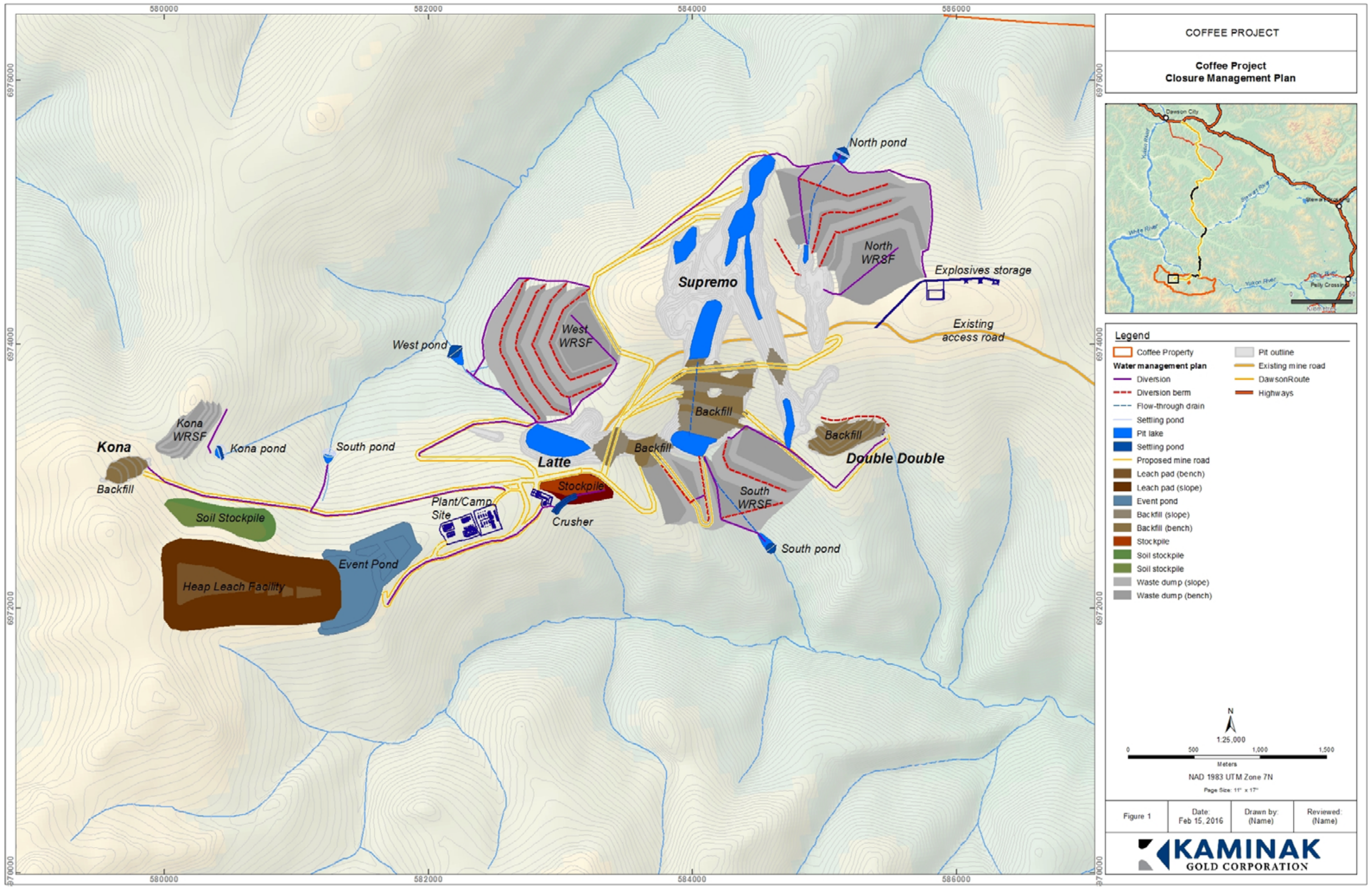
- Remove all gravel on the ice surfaces that may have ended up on the ice as a result of sanding the portages;
- Gather all road signs and properly store them for future use;
- Remove any garbage that is found along the route; and
- Remove any hydrocarbon spills that are found along the route.

18.4 Surface Water Management

A hydrology and hydraulic analysis was prepared for the maximum extent of the Project at the end of mine life. The analysis used the FS mine plan including the layout of the pits, waste dumps, heap leach pad, crusher and process plants, roads and other support facilities. The design uses hydrologic information to estimate runoff volumes, peak flow rates and catchments. The design is not intended to be the final configuration for drainage collection, but simulates a probable configuration relevant to the design scenario presented for the mine site. The calculations outlined below are conservative in terms of total anticipated runoff, pond storage, and conveyance capacity.

Figure 18.11 presents the site layout at maximum extent at the end of Year 10. Components of the water management system include diversion channels, diversion berms, flow-through rock drains in the WRSF, and sediment collection ponds.

Figure 18.11. Water Management Plan



Source SRK 2016

Sedimentation ponds will be located at the pour point of catchments receiving contact water from mine workings. Changes to the location and/or footprint of each WRSF and their sequence of development may affect the volume of runoff collected. The conveyance system capacity is conservatively designed to accommodate minor variations from the current mine plan that could affect estimates of water volume storage requirements or peak flows.

18.4.1 Design Criteria and Parameters

Design criteria for the Project provide adequate measures to reduce erosion caused by concentrated flows, capture sediment, and provide a stable conveyance system. Design criteria and assumptions for routing stormwater and attenuation are as follows:

- Rainfall depths are based on meteorological information provided by Lorax (2015a);
- Rainfall runoff modelling is based on the Soil Conservation Service (SCS) Type I storm hydrograph;
- Collection and routing of impacted runoff through sedimentation ponds (excluding the landing strip area, and mine access road) to settle total suspended solids (TSS);
- Channels and berms convey the 100-year, 24-hour storm event including average daily maximum snowmelt;
- Surface grading will be performed during operations to accommodate drainage into design shown;
- Provide retention volume equal to the estimated runoff from the 10-year, 24-hour storm event within the sedimentation ponds, and release rate over a period of 48 hours or longer;
- Provide detention volume equal to approximately half the estimated maximum 24-hour snow melt volume (Lorax 2015a);
- Pond embankments include 1 m of freeboard volume;
- Dam design for sedimentation pond freeboard does not include estimations for wind setup and wave run-up;
- Pond embankment crest widths will have a minimum width of 14 m to provide access for maintenance traffic;
- Pond outlet structures will accommodate flows up to the 100-year, 24-hour storm event;
- Each pond will have emergency spillways to convey the 200-year, 24-hour storm event (a larger event may be considered upon further evaluation of the ponds);
- TSS concentrations in runoff sediment loads are uncertain. However, storage capacity within the ponds is expected to provide retention time to settle the majority of TSS. The 48-hour retention time should facilitate the settling of TSS particle sizes greater than 5μm;
- Flocculation may be required to manage TSS, especially during spring freshet and large storm events due to the increased quantity of runoff and reduced water quality;
- Hydrologic parameters for approximate SCS hydrologic soils groups are based on AECOM (2012);

- Conveyance channels are currently designed to a 1 m depth. The final design depth may be reduced following further refinement of the design; and
- Maintain a minimum of 0.3 m of dry freeboard in the channel (i.e. between the normal water surface elevation and the top of channel) for the design storm and to account for debris and ice build-up.

18.4.2 Hydrologic Analysis

The hydrologic modelling program HEC-HMS version 4.1, U.S. Army Corps of Engineers (USACE 2013), was used to perform the hydrologic calculations. HEC-HMS model parameter include the following:

- Catchment delineation;
- Precipitation;
- Snowmelt;
- Precipitation losses;
- Transformation;
- Channel routing; and
- Runoff calculations.

18.4.3 Hydraulic Analysis

Manning's equation was used to estimate the flow depth using a trapezoidal channel geometry, assuming flow depth equals normal depth. Channel geometry and Manning's 'n' value (coefficient of roughness which varies with riprap size) were varied to determine an appropriate channel configuration to convey the 100-year, 24-hour peak flows calculated in the HEC-HMS model. The channels are designed with a minimum of 0.3 m of dry freeboard for the peak flow event to allow for possible ice build-up, sedimentation, and climatic and natural uncertainties.

To protect channels and diversion berms from erosion, rock armour (riprap) size was determined using NRCS Practice Standard 486 as developed for the United States Federal Highway Administration (Robinson 1998). The Practice Standard uses Manning's equation and an iterative calculation to determine the depth and velocity of flow. The "n" value is based on depth of flow over a rough surface and varies depending on riprap size and slope derived from the expressions of Strickler, Anderson and Abt et al (1988). The practice calculates a stable median riprap size (D_{50}) to resist the tractive force of design peak flow in the channel.

18.4.3.1 Conveyance

The conveyance system comprises of channels, diversion berms, haul road drainage ditches, and surface drainage (both natural and constructed) directing runoff to a conveyance route. Runoff is conveyed from the working mine area to sedimentation ponds located in key locations immediately downstream of WRSFs or other impacted areas. Stormwater is routed through the ponds, where sediment settles before discharging to the environment.

18.4.3.2 Rock Drains

Flow-through drains are proposed as a method to drain pit lakes which may form at the end of mine life or develop during operation. The proposed locations of the drains are depicted in Figure 18.11. The drains will be constructed of coarse waste rock selected and stockpiled during mining operations. Geotextile fabric or a filter (sorted rock layer) may be used to prevent migration of fines into the flow-through drain from above. Waste rock size characterization is needed before determining the need for a filter layer. Alternatively a thicker layer of drain rock could be deposited to reduce the influx of fines into the drainage pathway. The flow-through drains will accommodate the 100-year, 24-hour storm event, including snowmelt with capacity of up to four times the calculated runoff peak quantity. Table 18.15 summarizes runoff volume anticipated to be conveyed through each flow-through drain, as well as calculated dimensions.

Table 18.15: Flow-through Drain Summary

Name	Storm "Q" (m^3/s)	Minimum Bottom Width (m)	Minimum Height (m)	Drain Rock Diameter (m)	Approximate Volume (m^3)
North Dump Drain	3.33	10	3.4	0.75	40,000
South Dump Drain 1	1.6	8	2.5	0.75	24,000
South Dump Drain 2	6.16	15	4.8	0.75	81,000

Source: SRK 2016

The flow-through drains for the North and South dumps are constructed in areas where permafrost may be present. Ice formation within the flow-through drains is a concern. However, they are designed to accommodate up to four times the 100-year 24-hour flow and also due to the perviousness of the waste rock dump, it is unlikely that ice could form and block flow entirely. Channels are located near the spill point of the pit lakes (near the connection with the flow-through drains) to accommodate overflow should the flow-through drains clog or freeze. The channels are designed for the 100-year event; if the drains were to freeze, the channels have the capacity to handle the average yearly runoff or spring freshet. An alternate flow path could be sized to provide conveyance of the 100-year, 24-hour storm event or greater.

The preferred alternative is to provide conveyance through the WRSFs as opposed to over their surface. However, capacity for overland flow could be provided at a modest cost to the Project.

18.4.4 Attenuation of Runoff

Runoff from the mine site will be routed to five retention ponds located downstream of proposed mining areas; North Pond, South Pond, West Pond, Kona Pond, Heap Pond and Ore Pond. The ponds serve two purposes. Firstly, they settle the TSS load prior to discharge, and secondly they reduce the peak discharge rate of a storm by attenuating (storing and releasing) runoff and discharging it at a lower peak rate. Each pond is sized to store and release a volume of water equal to or less than the 10-year, 24-hour storm event runoff volume.

Runoff volumes greater than the 10-year event, but less than or equal to the 100-year event, will be routed through the pond riser. Storms up to the 200-year event will be routed through the emergency spillway. Peak discharge (m^3/s) from the ponds will be at a rate less than or equal to the pre-development rates for storms less than or equal to the10-year, 24-hour event.

18.4.4.1 Sedimentation Ponds

The ponds will be constructed by building a berm at the downstream end of the pond. All pond embankments are greater than 8 m in height. However, none will impound more than 60,000 m^3 of water. Table 18.16 summarizes designed capacity and dimensions for each pond.

Table 18.16. Pond Designed Capacity and Dimensions

Description	Drainage Area (km^2)	Crest Elev. (m)	Embank. Fill Volume (m^3)	Dam Height (m)	Dam Crest Width (m)	Dam Crest Length (m)	10yr 24hr Runoff Volume (m^3)	Approx. 24-Hour Snowmelt Volume (m^3)
North Pond Dam	2.2	962	57,000	14	14	120	26,000	57,000
West Pond Dam	1.96	921	59,000	13	14	140	23,000	51,000
South Pond Dam	2.97	827	31,500	12	14	90	26,000	77,000
Heap Pond Dam	1.13	1,035	21,500	10	14	100	14,000	29,000
Kona Pond Dam	0.18	1,160	20,000	9	14	115	3,500	4,700
Ore Pond Dam	0.37	1,103	10,500	4	6	200	4,000	9,700

Source: SRK 2016

The maximum average daily snow melt during spring freshet (Lorax, 2015a) will generate a runoff volume approximately two to three times that of the 10-year, 24-hour storm event. Sizing the ponds to accommodate the maximum snow melt water volume over a 24-hour period is costly and the consequent increase in impoundment size could change the classification of the impoundment structures. The TSS load during larger runoff events would be better managed by adding flocculent at the pond inlet to enhance settling. By sizing the ponds to accommodate a volume greater than the average storm event each year, but less than the maximum runoff (snow melt), the retention time within the pond will increase and resulting more efficient collection of sediment for these smaller events. The spring freshet will likely generate a large quantity of runoff.

However, peak discharge rates will be much less than a 100-year, 24-hour event. This is due in part to the slow and constant rate of snow melt, versus the peak discharge of a rainstorm. Due to the large influx of freshet runoff, the ponds will be filled to capacity with overflow discharging through the riser structure.

18.4.4.2 Total Suspended Solid Settling

The sedimentation ponds reduce TSS by allowing settling of the sediment to occur and thereby increase water quality at the discharge point. Reduction of TSS is accomplished by gravity settlement during extended retention and/or by the addition of flocculant. The ponds are sized to accommodate a volume of water equal to the volume of the10-year, 24-hour storm event runoff.

During large storm events or spring freshet it is unlikely that water quality can be met by gravity settling alone; flocculent will be used to enhance TSS settling. Flocculation stations will be located at the inlet end of the ponds and will treat inflow prior to entering the ponds. Sediment accumulated within the ponds will be removed when permanent pool storage depth is less than 1 m at the riser, or at a minimum prior to winter freeze-up.

18.4.5 Site Grading

The drainage pattern shown in Figure 18.11 assumes drainage structures will be maintained throughout operations and allow contact water to report to a conveyance structure and then ultimately to a sediment pond or pit lake. The pit lakes will act as primary settling ponds before discharging to the sediment ponds below.

The benches on the waste rock dumps will be designed to slope (inwards) away from the dump crest. Runoff is concentrated along toe of each bench and prevented from running over the dump face by a series of diversion berms (originally safety berms).Runoff is diverted to the perimeter of the waste rock dumps and collected in channels at the dump perimeter. Maintaining this drainage pattern during operations may be challenging, and a variation of channels and berm cuts directing runoff down the dump face may be considered for minor runoff volumes. Directing large quantities of concentrated runoff down the face of the dump should be avoided. Large quantities of runoff may pose problems such as erosion of the dump face, or the construction of a suitable armoured conveyance channel (maximum slope for a riprap channel is 2.5:1).

Once a bench is developed to full build-out, runoff will be diverted to the perimeter channels. The channels located at the perimeter the dumps will be sized to accommodate either scenario (down the face, or to the perimeter), as the total drainage area is the same, and expected runoff quantities intercepted will be similar. Diverting concentrated flow water down the dump face will be avoided to prevent erosion on the slope. The water management system (berms, diversion, channels and ponds) will be constructed according to the final build-out of the dumps and pits. Interim water management structures will be built as required.

At closure, TSS will be monitored and will likely decrease after mining. This is especially true for the North and South WRSFs where the majority of runoff will be diverted to pit lakes for primary settlement. In areas where WRSFs remain uncovered, it is unlikely that the sediment basins can be removed unless the site is stabilized with vegetation or rock armouring to reduce erosion and sediment migration from the dump benches and faces.

18.5 Mobile Equipment

Mobile site support equipment provides support to operations at the Coffee site. A list of site support equipment is provided in Table 18.17. The support equipment fleet is based on similar equipment utilized at other northern Canadian mining operations.

Table 18.17: Site Support Equipment

Equipment Description	Quantity
1 T Diesel Crew Cab Pick-up - Ford F350	6
2 T Diesel Pick-up c/w Heated Van - Ford F550	1
5 T Flat Deck Truck	1
20T Picker Truck - Western Star 4900 XD	1
Water Truck	1
Vacuum Truck	1
Tractor w/ Deck Trailer	1
44 Passenger Bus - Freightliner	2
Ambulance/Rescue - Ford F450	1
Tool Carrier - Cat 966K (c/w Attachments)	1
Skid Steer Loader (1Cu.M)	2
Excavator (~1.0 CU.M) CAT 320DL	1
3 T Forklift - Warehouse - CAT 2DP6000	1
65ft Man-Lift - Genie S-65	1
Portable Diesel Heaters	4
Pipe Fuser - McElroy T900	1
Cat 140 Grader	1
River Barge	2

Source: JDS 2016

18.6 Manpower

The site support manpower crew 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 transport between the camp and aircraft;
- Loading and unloading of aircraft;
- Airstrip operations and maintenance;
- Waste management duties (incineration, water treatment, hazardous waste handling);
- Plant site snow removal;
- Site water management;

- Access road maintenance; and
- Barge operations.

Table 18.18 provides the composition of the site support crew.

Table 18.18: Site Support Manpower Crew

Labour Position	On-site Labour
Surface foreman	1
Electrician	1
Facilities maintenance - tradesman	1
Mobile equipment operator	3
Labourers/apprentices	2
Mobile equipment operator – road maintenance	1
Barge captain	2
Labourers – barge operations	2
Total support crew (on-site)	**13**

Source: JDS 2016

19 Market Studies and Contracts

Detailed market studies on the potential sale of gold from the Coffee 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 Gordon 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

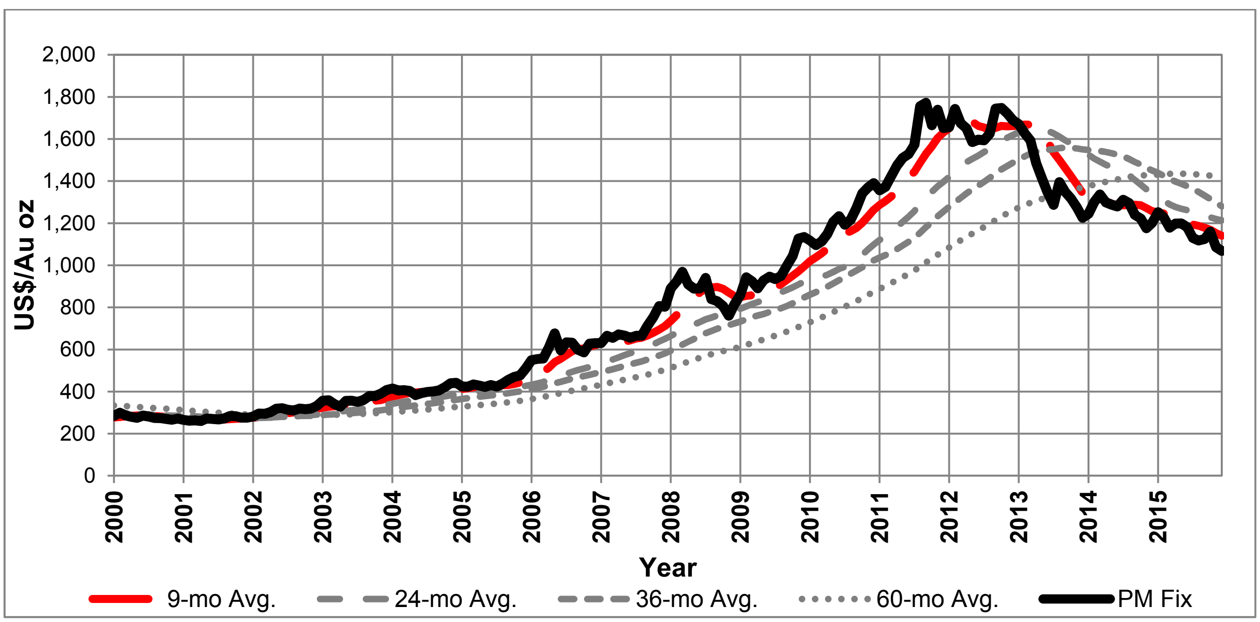
Assumptions	Unit	Value
Au Payable	%	99.8
Au Refining Charge	US$/oz	1

Source: JDS 2016

19.1 Metal Prices

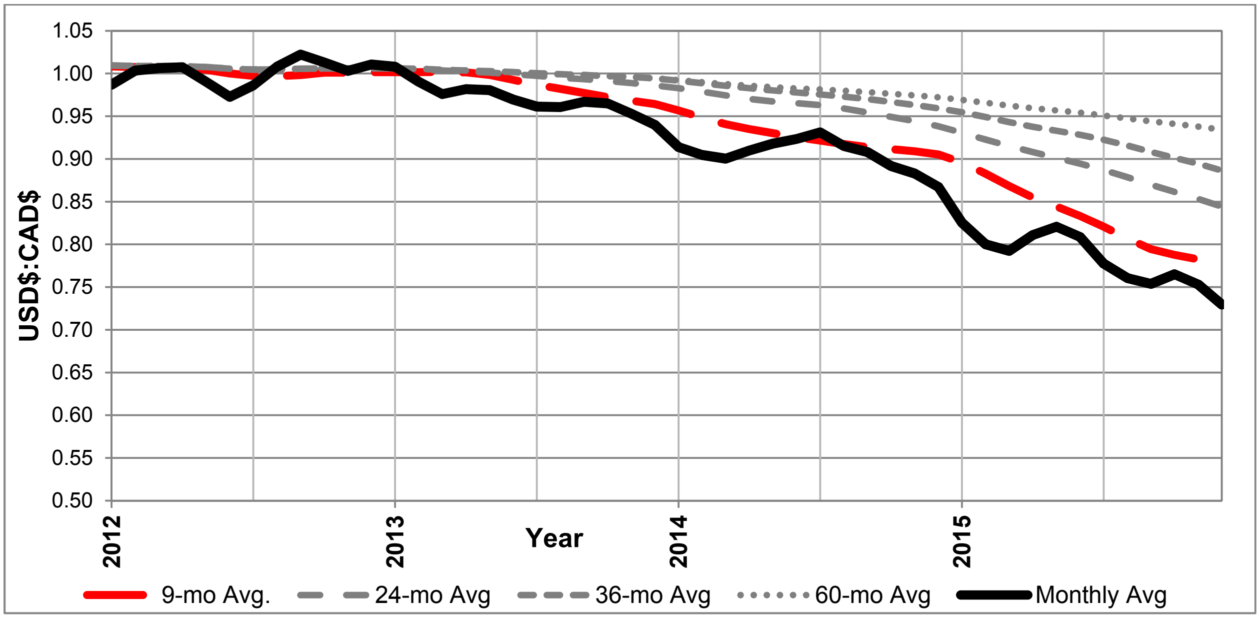
The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical gold prices are shown in Figure 19.1 and demonstrate the change in metal prices from 1998 to 2015. Historical average US$:C$ exchange rates are shown in Figure 19.2.

Figure 19.1: Gold Price History (Kitco Spot)



Source: JDS 2015

Figure 19.2: Monthly Average US$:C$: Foreign Exchange Rate – Bank of Canada



Source: JDS 2015

The gold price used in the economic analysis is based on the 9-month trailing average spot rate during November 2015 sourced from Kitco Metals Inc. The US$:C$ exchange rate used in the economic analysis is based on the 9-month trailing average spot rate during November 2015. 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
Au Price	US$/oz	1,150
F/X Rate	US$:C$	0.78

Source: JDS 2015

19.2 Contracts

19.2.1 Royalties

There is a 2% net smelter return royalty (NSR) on the property, payable to prospector Mr. Shawn Ryan of Dawson, YT. The NSR is subject at any time to a 1% buy-back for $2M, with annual advance royalty payments of $20,000 commencing December 31, 2013.

Total third party royalties for the project amount to C$29.4 M over the life of mine should the buy-back be exercised.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Assessment and Permitting for the Coffee Gold Mine

20.1.1 Overview

Kaminak wishes to construct, operate and eventually close the Coffee Gold Project mine. To achieve this, Kaminak will require an assessment under the Yukon Environmental and Socio-economic Assessment Act (YESAA) prior to licensing and permitting.

In 2010, Kaminak initiated the collection of baseline environmental data, including social and heritage information. The collection of baseline information, which is ongoing, has centred on those elements which are foci for assessment and regulatory processes and include soils, surficial geology, geochemistry, wildlife, water, climate, vegetation, fish, aquatic resources, heritage resources, and socio-economic setting. The data collected to date is to inform both assessment and regulatory processes as well as establish baseline conditions for future monitoring.

Kaminak is committed to supporting the direct and meaningful involvement of local Yukon First Nations in the environmental and socio-economic assessment process. The proposed Project area, including the access road from Dawson, is in the traditional territories of the Tr'ondëk Hwëch'in, Selkirk First Nation, First Nation of Nacho Nyak Dun and the asserted area of White River First Nation. First Nation input and participation has been central to the planning and implementation of baseline data collection, re-vegetation trials, and closure and reclamation planning. Through ongoing dialogue, First Nations environmental and socio-economic values are being identified and will be incorporated into and reflected in the assessment.

20.1.2 Project Proposal

Kaminak is developing a project proposal that will include the information specified by the Yukon Environmental and socio-Economic Assessment Board (YESAB) Executive Committee for consideration during their ESA. The proposal will describe the Project, the environmental and socio-economic conditions, the potential effects of the Project on these conditions, mitigations for adverse effects, and monitoring proposed to inform environmental management decisions. The proposal will contain a record of consultation with potentially affected communities and First Nations.

Work has commenced in preparing the project proposal that describes the potential environmental and socio-economic effects of the Project. Technical consultants with the expertise in relevant disciplines have been retained to develop the project proposal. It is anticipated that the project proposal will be submitted to YESAB in July 2016.

20.2 Regulatory Licences, Permits and/or Authorizations

20.2.1 Overview

Concurrent with the assessment of the project proposal, Kaminak will be applying for a water licence from the Yukon Water Board, a quartz mining licence from the Government of Yukon, Department of Energy, Mines and Resources, and other authorizations. Although the environmental assessment process and permitting processes can run concurrently, authorizations cannot be issued in advance of the completion of the ESA process and the issuance of the Decision Document. To facilitate applications for the authorizations, Kaminak will submit regulatory-ready information for assessment to YESAB. This information will be used to apply for regulatory authorizations required to construct, operate and eventually decommission and reclaim the mine site. The intent is to enable assessment and permitting processes to occur in parallel and shorten the total time for assessment and permitting processes.

Table 16.1 provides, but is not limited to, a listing of the federal and territorial acts, regulations, guidelines applicable to the Coffee Project. Three of the major authorizations are briefly described below.

Table 20.1: Applicable Acts, Regulations and Guidelines relevant to the Coffee Project

Acts	Regulations	Guidelines/Permits/Licences
Federal		
Aeronautics Act	Canadian Aviation Regulations, General Operating and Flight Rules	
Canadian Environmental Protection Act (1999 c.33)	Storage Tank Systems for Petroleum Products and Allied Petroleum Products Regulations	Canadian Council of Ministers for the Environment (CCME) - Environmental Code of Practice for Above-ground and Underground Storage Tank Systems Containing Petroleum and Allied Petroleum Products
		Notice with respect to substances in the National Pollutant Release Inventory
	Environmental Emergency Regulations [SOR/2003-307]	
	Interprovincial Movement of Hazardous Waste and Hazardous Recyclable Material Regulations	Environment Canada Technical Document of Batch Waste Incineration (January 2010)
	Pollutant Discharge Reporting Regulations, SOR-95-351	
Canada Water Act (1985 c.11)		
Canada Wildlife Act (1985 w9)		
Species at Risk Act (2002 c.29)		
Migratory Birds Convention Act (1994 c.22)	Migratory Birds Regulations (C.R.C., c. 1035)	
Fisheries Act [R.S.C. c. F-14]	Metal Mining Effluent Regulations [SOR/ 2002-2222]	
		Fisheries Productivity Investment Policy: A Proponent's Guide to Offsetting
		Freshwater Intake End-of-Pipe Fish Screen Guideline
Explosives Act (1985 c.E-17)	Ammonium Nitrate and Fuel Oil Order	Blasting Permit, Purchase and Possession Permit, Permit to Transport Explosives
	Explosives Regulations	
Navigation Protection Act (R.S. 1985 c. N-22)	Navigable Waters Works Regulations	Section 5(2) approval
National Fire Code of Canada		
National Building Code of Canada		
Transport of Dangerous Goods Act [1992, c.34]	Transportation of Dangerous Goods Regulations [SOR/2001-286]	
Yukon First Nations Land Claims Settlement Act		
Yukon Environmental and Socio-economic Assessment Act	Assessable Activities, Exceptions and Executive Committee Projects Regulations	Decision Document
	Decision Body Time Periods and Consultation	
	Regulations	
The Tr'ondëk Hwëch'in Final Agreement	Heritage	
The Selkirk First Nation Final Agreement	Heritage	
First Nation of Nacho Nyak Dun Final Agreement	Heritage	

Acts	Regulations	Guidelines/Permits/Licences
Territorial - Yukon		
Boiler and Pressure Vessel Act		Pressure Vessel Boiler Permit
Building Standards Act		Building Permit, Plumbing Permit
Environment Act	Air Emissions Regulations	Air Emission Permit
	Contaminated Sites Regulation	Land Treatment Facility Permit
	Storage Tank Regulations	Above-Ground Storage Tank Permit
	Solid Waste Regulations	Waste Management Permit
	Special Waste Regulations	Special Waste Permit
	Spills Regulations	Contaminated Material Relocation Permit
Forest Protection Act	Forest Protection Regulation	Open Burn Permit
Gas Burning Devices Act		Gas Installation Permit
		Gas Burning Devices Permit
Highways Act	Bulk Commodity Haul Regulations	Access Permit
	Highways Regulations	Work in Highway Right-of-way Permit
Occupational Health and Safety Act	Blasting Regulation	Blasters Permit
	Occupational Health and Safety Regulations	
	Workplace Hazardous Materials Regulation	
Public Health and Safety Act	Public Health Regulations	Sewage Disposal System Permit
	Camp Sanitation Regulations	
	Sewage Disposal Systems Regulation	
	Drinking Water Regulation	
Quartz Mining Act	Quartz Mining Land Use Regulation	Quartz Mine Licence
	Security Regulation	
	Quartz Mining Fees and Forms Regulation	
Territorial Lands (Yukon) Act	Territorial Lands Regulation	Commercial Timber Permit
	Land Use Regulation	Land Use Permit
		Quarry Permit
		Aerodrome Licence
Waters Act	Waters Regulation	Type A Water License, Type B Water License
Wildlife Act	Wildlife Regulation	
Workers Compensation Act	Multiple Requirements	
Yukon Land Claim Final Agreements, An Act Approving		
Yukon Historic Resources Act	Archaeological Sites Regulations	Archaeological Sites Permit

Source:

20.2.2 Water Licence

The Yukon Water Board regulates the use of water and/or the discharge of waste to water. It issues water licences that specify in part the quantity of water that can be used, conditions specific to the discharge of waste to water, monitoring requirements, and environmental management plans to be prepared and implemented. Kaminak's Coffee Gold Project will require a Type "A" Water Licence.

20.2.3 Quartz Mining Licence

The Yukon Quartz Mining Act, administered by the Government of Yukon's Department of Energy, Mines and Resources, regulates hard rock mineral exploration and mine development in the Yukon. A quartz licence serves as a regulatory and decision-making framework that delineates how a company will develop and manage the mine over the life of the Project[1].

20.2.4 Environmental and Mine Operation Plans

Environmental management plans will be assembled under an Environmental Management Plan, which provides overarching direction for environmental and development management at the Coffee Gold Project. It is supported by a suite of project-specific mitigation, monitoring and/or management plans that set out the Project's standards and requirements under the Quartz Mining Licence and/or Water Licence for particular areas of environmental management. These plans are currently in preparation.

20.3 Environmental Studies

Baseline environmental studies were initiated in 2010. A weather station was established in 2012. Baseline studies include but are not limited to water quality, water quantity, climate, geochemistry, metal leaching/acid rock drainage, wildlife, and vegetation. The scope of baseline studies is in part informed by regular consultation with First Nations and engagement with regulators.

Baseline and associated research studies are currently ongoing. All the information collected will be used in preparing an ESA, and applying for permits, authorizations and/or licences required to construct, operate, and ultimately close the mine.

20.4 Mine Reclamation and Closure Plan

20.4.1 Overview

A reclamation and closure plan (RCP) will be submitted with the project proposal. The RCP will include a liability estimate for reclaiming and closing the mine. Consultation has been initiated with First Nations, communities, regulatory agencies and public organizations to gain an understanding of their expectations and concerns with regard to reclamation and closure. The RCP will demonstrate how Kaminak considered and addressed the expectations and concerns throughout the mine planning process.

The Yukon Water Board has stated that it seeks to "…issue licences only when there is a reasonable certainty that an acceptable level of reclamation of the site can be achieved during mining and/or following cessation of mining[2]". The mine plan was developed with closure in mind and ensures the integration of closure considerations into the mine's planning and operational processes. In taking this approach, reclamation and closure will be assisted by timeous planning, which at a later date will lead to an effective closure while also reducing the overall reclamation and closure cost.

[1] Source http://www.emr.gov.yk.ca/mining/pdf/qm_license_application_guide.pdf

[2] Source: http://www.emr.gov.yk.ca/mining/pdf/mml_reclamation_closure_planning_quartz_mining_projects_aug2013.pdf

Over the life of the mine successive iterations of the plan may be expected every two years, each iteration providing more detail and greater certainty regarding the sequence of events to occur during reclamation and closure.

While the mine is in operation, progressive reclamation will be employed to reclaim areas no longer needed for mining. This approach will employ best practices and will ultimately advance the return of disturbed areas to self-sustaining biological ecosystems, while at the same time reducing the overall final cost of reclamation.

20.4.2 Reclamation Bond Requirements

Financial liability is to be considered and addressed within the overall RCP for the Coffee Gold Project. This liability is the cost to implement the approved reclamation and closure plan to reclaim and close the mine were it to close prematurely during its operational life. Kaminak will provide the required security to government, to cover the financial liability for reclamation and closure. Over the life of the mine, the security held by government will remain commensurate with the outstanding reclamation and closure liability at that time. As the mine plan is subject to change, the financial liability will be reviewed every two years, which parallels updates to the RCP. [3]

The current anticipated closure costs for the Project are estimated to be $60M including contingency.

Adjustments to the security are expected to be made to account for progressive reclamation or failure to meet reclamation objectives, changes in liabilities, knowledge, technology and risk, to account for costs associated with a temporary closure, to account for changes to the net present value of security; or any material change.

The RECLAIM model, or equivalent, will be used in calculating financial reclamation and closure liability for the mine. This calculation will result in a projected liability determination following completion of construction and will be used by government in setting the security value.

20.5 Community and Government Engagement and Consultation

There is an expectation by all parties consulted that mine development will be undertaken in a responsible manner and that measures will be put in place for the protection of the environment and public health and safety. Additionally, the parties look to socio-economic benefits from the mine flowing to them, and that business opportunities accrue to the Yukon.

Engagement and consultation is well underway; it is a process through which Kaminak is building relationships into meaningful partnerships between itself and various stakeholders. Kaminak views consultation not as a single conversation but a series of conversations allowing the creation of capacity and understanding of the Coffee Project among First Nations and communities who may potentially be impacted. Consultation entails an implicit commitment to include First Nations and stakeholders in the decision-making process. This applies to activities or developments that remain open to modification based on their input.

[3] Adapted from Yukon Mine Site and Reclamation Closure Policy Financial and Technical Guidelines September 2013

Consultation is providing Kaminak with an opportunity to provide Project information, learn of issues and concerns, ask questions, and help shape elements of the Project by suggesting improvements for Kaminak to consider. Consultation has also for a platform to investigate future collaboration and partnerships.

Parties consulted to date include communities, Yukon and federal agencies, First Nations, business associations and the public. Based on Kaminak's consultation and engagement principles and its existing communications policies and commitments, the scope and frequency of consultation activities are expected to increase as the assessment and regulatory documents are developed. This will ensure that views expressed by the parties consulted are given full and fair consideration in the planning and development of the mine, and in its operation and decommissioning.

Kaminak recognizes the importance of engaging and consulting First Nations, on whose traditional territory the mine is to be located, and in establishing long term, good-neighbour relationships with them. Kaminak will respond to First Nations' concerns and accommodate their interests, concerns and priorities before making decisions. Successful engagement and consultation is expected to lead to First Nations understanding the Project, and sharing in the benefits and economic opportunities it will provide. It will allow Kaminak to have first-hand knowledge of the concerns and priorities First Nations have with the Project.

The use of local and traditional knowledge provided by the First Nations will factor into the proposed mine's policies and design of monitoring programs. It will allow Kaminak to avoid culturally and/or ecological significant and sensitive areas. The company has made a commitment to provide support to First Nations for their involvement in planning and traditional use studies/oral history projects.

21 Capital Cost Estimate

Preparation of the capital cost estimate (CAPEX) for the Coffee Gold Project has been developed by JDS for Kaminak from input by various stakeholders including The Mines Group (MINES), SRK, and Onsite Engineering Ltd. (Onsite).

The CAPEX is based on the JDS philosophy that emphasizes accuracy over contingency, and uses defined and proven Project execution strategies. The estimates were developed using first principles, applying directly-related Project experience, and the use of general industry factors. Almost all of the estimates used in this Project were obtained from engineers, estimators, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

The following cost estimates are described in this section:

- Initial Capital Cost – includes all costs incurred to develop the Project to a state of nameplate production (18,182 t/d); and
- Sustaining Capital Cost – includes all costs incurred during production for initial and ongoing open pit installations and development, life of mine equipment acquisitions and replacements, and annual heap leach expansions.

Sunk costs and owner's reserve are not considered in this section.

All cost estimates are based on the specific scope and execution plans described in this study. Deviations from these plans will affect the capital costs.

A Work Breakdown Structure (WBS) was established for the initial capital cost estimate. Costs have been classified into the various WBS areas to ensure that the entire Project scope has been captured.

Table 21.1 summarizes the capital cost estimate by area and activity. Table 21.2 shows the capital cost distribution.

Table 21.1: Summary of Capital Costs by Category

Capital Cost	Initial $M	Sustaining $M	LOM $M
Mining	85.4	47.5	132.9
On-Site Development	7.7	0.9	8.7
Ore Crushing & Handling	16.4	0.0	16.4
Heap Leach	28.2	34.5	62.7
Process Plant	27.6	1.0	28.6
On-Site Infrastructure	43.1	2.8	46.0
Off-Site Infrastructure	24.3	0.0	24.3
Indirects	31.7	4.7	36.4
EPCM	18.9	1.5	20.4
Owner's Costs	7.9	0.0	7.9
Closure	0.0	60.5	60.5
Subtotal	**291.4**	**153.4**	**444.8**
Contingency	26.1	7.2	33.3
Total Capital Costs	**317.4**	**160.6**	**478.1**

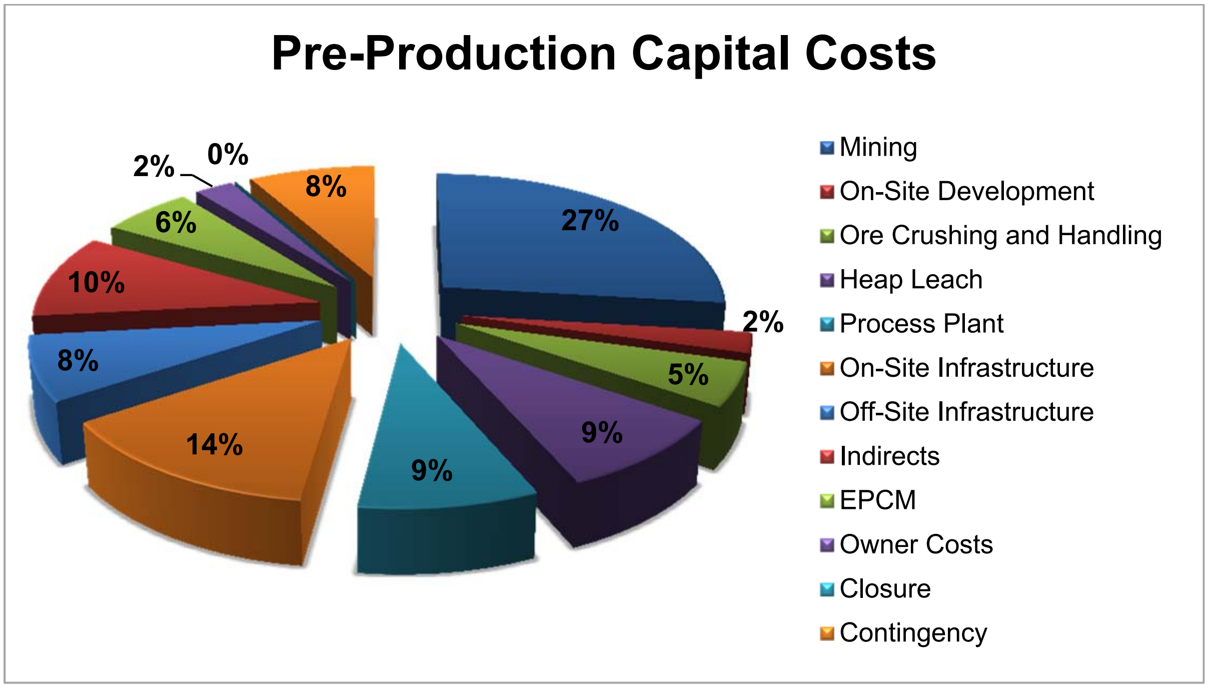
Source: JDS 2015

Table 21.2: Summary of Capital Cost Distribution

Capital Cost	Initial Capital Distribution (%)	Sustaining Capital Distribution (%)
Mining	26.9	29.6
On-Site Development	2.4	0.6
Ore Crushing & Handling	5.2	0.0
Heap Leach	8.9	21.5
Process Plant	8.7	0.6
On-Site Infrastructure	13.6	1.8
Off-Site Infrastructure	7.7	0.0
Indirects	10.0	2.9
EPCM	6.0	0.9
Owner's Costs	2.5	0.0
Closure	0.0	37.7
Subtotal	91.8	95.5
Contingency	8.2	4.5
Total Initial Capital Cost Distribution	**100.0**	**100.0**

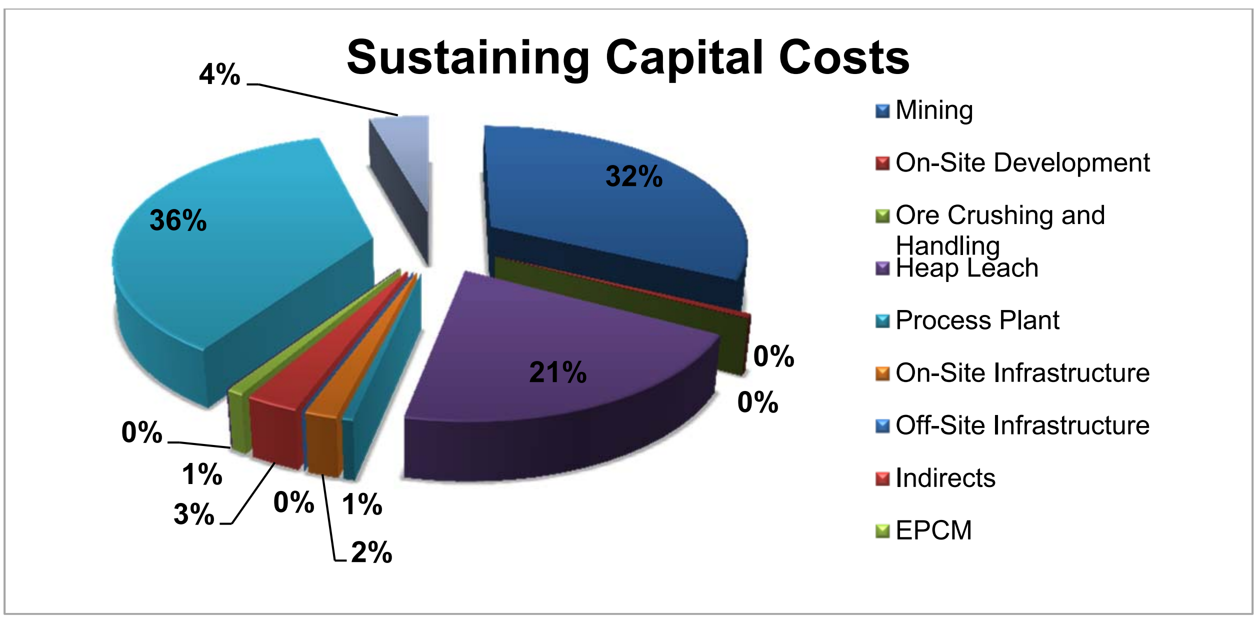
Source: JDS 2015

Figure 21.1: Breakdown of Pre-Production Capital Costs



Source: JDS 2015

Figure 21.2: Breakdown of Capital Expenditures during Production



Source: JDS 2015

21.1 Basis for Capital Cost Estimates

The accuracy of the capital cost estimate is in the range of +/-15%, which represents a JDS Feasibility Study Budget/AACE Class 2 Estimate.

This estimate was prepared with a base date of Q4-2015 and does not include any escalation beyond this date. The quotations used for this study were obtained in Q4-2015 and are valid for a period of 90 calendar days.

The capital cost estimate uses Canadian dollars as the base currency. When required, quotations received from vendors were converted to Canadian dollars using a currency exchange rate of US$:C$ 0.78. Duties and taxes are not included in the capital estimate.

21.1.1 Responsibility Matrix

This capital cost estimate was developed by a team of engineers, procurement specialists and cost estimators. JDS is responsible for the development and assembly of the overall capital cost estimate with input from companies shown in Table 21.3.

Table 21.3: Scope of Work

Description	Responsibility	Scope
Open Pit Mining (WBS 1000)	JDS	OP Mine Development & Production OP Mining Equipment OP Mine Services & Equipment Haul Roads
On-site Development (WBS 2000)	JDS	Bulk Earthworks Site Water Management Airstrip Roads & Utilities
	SRK	Site Water Management
Ore Crushing & Handling (WBS 3000)	JDS & Allnorth	Detailed Civil Works Concrete (quantities by Allnorth) Internal Steel Buildings Mechanical Equipment Mechanical Bulks/Platework Piping Electrical Equipment & Bulks (quantities by Allnorth) Instrumentation Equipment & Bulks (quantities by Allnorth) E-Houses and Plant Lighting (quantities by Allnorth)
Heap Leach (WBS 4000)	MINES	Leach Pads Earthworks Leach Pads Liner Leach Pads Piping Ponds Earthworks Ponds Liner Ponds Piping
	JDS	Detailed Civil Works Mechanical Equipment Pipelines
Process Plant (WBS 5000)	JDS & Allnorth	Detailed Civil Works Concrete (quantities by Allnorth) Internal Steel Buildings Mechanical Equipment Mechanical Bulks/Platework Piping Electrical Equipment & Bulks (quantities by Allnorth) Instrumentation Equipment & Bulks (quantities by Allnorth) E-Houses and Plant Lighting (quantities by Allnorth)
On-Site Infrastructure (WBS 6000)	JDS & Allnorth	Detailed Civil Works Concrete (quantities by Allnorth) Internal Steel Buildings/Facilities Mechanical Equipment Mechanical Bulks/Platework Glycol Piping (quantities by Allnorth) Piping Electrical Equipment & Bulks (quantities by Allnorth) Instrumentation Equipment & Bulks (quantities by Allnorth) Electrical Supply/Distribution (quantities by Allnorth) Mobile Equipment

Description	Responsibility	Scope
Off-Site Infrastructure (WBS 7000)	On-Site	Access Roads Access Road Maintenance Major Crossings Barge Landings
Indirects (WBS 9000)	JDS	Camp & Catering Field Indirects Freight & Logistics Vendor Reps Start-Up & Commissioning Capital Spares First Fills
EPCM (WBS 10000)	JDS	Mining Process Plant Infrastructure Water Management Construction Management
	MINES	Heap Leach
	On-Site	Access Road
Owner's Costs (WBS 11000)	JDS	G&A (labour, offices, freight, misc. items)
Contingency (WBS 12000)	JDS	Mining, Process, On-Infrastructure, Off-Infrastructure Indirects, EPCM, and Owner's costs (labour, materials, equipment, subcontract)

21.2 Basis of Cost Estimate for the Ore Handling, Process Plant, Infrastructure and Heap Leach

The basis of the cost estimate describes the methods, organization, assumptions and exclusions used to develop the Capital Cost Estimate the Project. The cost estimate includes the following elements:

Quantity Development

- CAPEX was developed largely from engineering quantities obtained from material takeoffs. In-house benchmarks were used where the engineering information were not sufficiently developed to prepare accurate quantities.

Direct Field Labour

- Direct field labour is the skilled and unskilled labour generally supplied by the contractor to install the permanent equipment and bulk materials at the Project site. Direct field installation man-hours were developed using estimated unit man-hours for each commodity multiplied by the quantity. Adjustments to standard man-hours were made to each commodity using a productivity factor (PF) to reflect the specific conditions at the Project site. These conditions include climate, physical extent of the site, working schedule, industrial environment, labour availability, etc.

Labour Rate

- A set of 'All-In labour Rates' was developed for each commodity, each based on a specific crew mix and proposed work cycle, and applied against direct field man-hours to generate direct field labour costs. Rates are based on an agreement between Ledcor Industrial Alberta and CLAC Local 63 valid to July 2012, and have been adjusted for inflation at 2% annually to reflect 2015 costs. The rates have been 'built up' to include all wages, benefits, government assessments, incentive pay, overtime costs, contractor indirects and contractor profit.

Productivity Factors for Labour

- A productivity factor has been applied to the standard base hours where the basis for the estimated work hours differs from the actual work environment. Factors for each commodity have been applied to reflect northern work environments. The factors apply to productive labour only and do not affect non-productive labour (travel to the workface, tool box meetings, safety inductions etc.).

Equipment Costs

- Estimates for major mechanical equipment are based on budget quotations. Major equipment is loosely defined as equipment costing greater than a million dollars and a delivery time greater than ten months. For minor equipment, prices were obtained from budget quotations or from similar recent equipment quotes. Miscellaneous and or undefined equipment has been factored based on historical data where time and cost efficiencies can be achieved without significant impact on the estimate accuracy.

Bulk Material Costs

- Bulk material costs have been calculated as either part of the built-up rates applied to engineering MTO's or factored costs or allowances. Built-up unit rates are based on project specific supply costs. Waste factors applied to bulk materials are shown in Table 21.4.

Table 21.4: **Waste Factors**

Commodity	Waste Factor
Civil & Earthworks	5%
Concrete	2%
Steel	1%
Piping	4%
Electrical Bulks	10%
Instrumentation Bulks	10%

Source: JDS 2015

Bulk material costs that were incorporated into the estimate include the following components:

- Site development and bulk earthworks;
- Concrete;
- Steel work;
- Mechanical bulks;
- Architectural;
- Piping;
- Electrical and instrumentation bulks; and
- Facilities.

The costs developed for facilities are a combination of unit rates, allowances and budget quotes. These costs were assessed based on specifications and requirements outlined by engineering design. The methodologies for costing of the major facilities are set out in Table 21.5.

Table 21.5: Facility Cost Basis

Facility	Cost Basis
Construction & Permanent Camp	Budget quotes have been obtained based on the estimated camp sizes
Ancillary Buildings	Budget quotes have been obtained based on design requirements and building sizes
Power Plant	Budget quotes have been obtained based on the estimated design electrical load
Incinerator Equipment & Building	Budget quotes have been obtained based on design requirements and building sizes
Truck Shop & Wash Bay	Building sizes have been determined by project requirements. The 'Pre-Engineered' truck shop has been quoted and the internal finishing costs have been included based on unit costs from JDS's internal data
Fresh, Fire, Process and Potable Water	Major holding tanks and pipelines have been quantified and priced based on project commodity costs. A budget quote has been obtained for the Portable Treatment plant has been based on budget quotes
Sewage Treatment	A budget quote has been obtained for the Sewage Treatment plant has been based on budget quotes

Source: JDS 2015

21.3 Mining

The mining is described in Section 16 of the report and contains detailed descriptions of the development methodology and equipment. A summary of the estimated costs for mining development and equipment are shown in Table 21.6.

Table 21.6: On-Site Development Cost Estimate (WBS 1000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
1000	**Mining**			
1100	Mine Development & Production	24.6	0	24.6
1600	Mining Equipment	60.0	45.3	105.2
1700	Haul Roads & Infrastructure	0.9	2.3	3.1
	Total Mining Costs	**85.4**	**47.5**	**132.9**

Source: JDS 2015

21.4 On-Site Development

The on-site development is described in Section 18 of the report and contains detailed descriptions of the site earthworks, drainage, and internal roads. A summary of the estimated costs for on-site development are shown in Table 21.7.

Table 21.7: On-Site Development Cost Estimate (WBS 2000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
2000	**On-Site Development**			
2100	Bulk Earthworks	1.8	0	1.8
2200	Site Water Management (includes discharge WTP)	2.9	0.9	3.9
2300	Airstrip	2.8	0	2.8
2400	Infrastructure	0.2	0	0.2
	Total On-Site Development Costs	**7.7**	**0.9**	**8.7**

Source: JDS 2015

21.5 Ore Crushing and Handling and Process Plant

The ore crushing and handling facilities and process plant are described in Section 17 of the report. Section 17 also contains images from the 3D model that are used to derive quantities. A summary of the estimated costs for ore handling and process plant are shown in Table 21.8.

Table 21.8: Ore Crushing & Handling and Process Plant Cost Estimate (WBS 3000 & 5000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
3000	**Ore Crushing & Handling**	**16.4**	**0**	**16.4**
3100	Primary Crushing	7.7	0	7.7
3200	Secondary Screening & Crushing	7.3	0	7.3
3500	Ore Crushing & Handling	0.6	0	0.6
3600	Lime	0.6	0	0.6

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
	Process Plant	**27.6**	**1.0**	**28.6**
5100	Process Plant Building	6.5	0	6.5
5200	Carbon Adsorption	3.0	0	3.0
5300	Acid Wash & Elution	1.6	0	1.6
5400	Carbon Regeneration	1.6	0	1.6
5500	Electrowinning & Refining	2.2	0	2.2
5600	CN Detox	0	1.0	1.0
5700	Reagents	0.8	0	0.8
5800	Process Utilities	4.3	0	4.3
5900	Pre-Production OPEX - Process	7.6	0	7.6
	Total Crushing & Process Plant Costs	**55.8**	**1.0**	**56.8**

Source: JDS 2015

The ore handling and process plant sections of the estimate include the following scope:

- Detailed earthworks;
- Concrete;
- Internal steel (equipment supports and access platforms);
- Mechanical equipment;
- Platework;
- Piping;
- Electrical;
- Instrumentation and process control; and
- Buildings (as a separate WBS section, including process plant piping).

21.5.1.1 Heap Leach

Heap Leach (HL) comprises of the pads, solution distribution & collection system, and ponds. The HL is described in Section 17 of the report. A summary of the HL costs are shown in Table 21.9.

Table 21.9: Heap Leach Capital Cost Estimate (WBS 4000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
4000	**Heap Leach**			
4100	Leach Pads	17.7	29.1	46.8
4200	Solution Distribution & Collection System	5.0	2.0	7.0
4300	Ponds	5.5	3.3	8.9
	Total Heap Leach Costs	**28.2**	**34.5**	**62.7**

Source: JDS 2015

21.6 Infrastructure

21.6.1.1 On-Site Infrastructure

The on-site infrastructure is described in Section 17 of the report. A summary of the on-site infrastructure costs are shown in Table 21.10.

Table 21.10: On-Site Infrastructure Capital Cost Estimate (WBS 6000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
6000	**On-Site Infrastructure**			
6100	Electrical Supply & Distribution	12.4	2.7	15.2
6200	Water Supply & Distribution	2.0	0	2.0
6300	Assay Laboratory	1.4	0	1.4
6400	Construction Camp/Permanent Camp	6.8	0	6.8
6500	Waste Management & Removal	0.4	0	0.4
6600	Ancillary Facilities	12.5	0.1	12.6
6700	Bulk Fuel Storage & Distribution	3.5	0	3.5
6800	IT & Communications	0.9	0	0.9
6900	Site Mobile Fleet	3.2	0	3.2
	Total On-Site Infrastructure Costs	**43.1**	**2.8**	**46.0**

Source: JDS 2015

21.6.1.2 Off-Site Infrastructure

The off-site infrastructure relates to the main access road and river crossings described in Section 18 of the report. A summary of the off-site infrastructure costs are shown in Table 21.11.

Table 21.11: Off-Site Infrastructure Capital Cost Estimate (WBS 7000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
7000	**Off-Site Infrastructure**			
7100	Access Roads	21.1	0	21.1
7200	River Crossings	3.2	0	3.2
	Total Off-Site Infrastructure Costs	**24.3**	**0**	**24.3**

Source: JDS 2015

The access road from Dawson to the Coffee site was designed on the basis of criteria provided in Section 18. The road was divided up into ten sections and material quantities required to upgrade, re-align or construction new sections of road were determined. Unit costs were established for all material quantities based on Yukon contractor equipment and labour rates. Major stream crossings and barge landings were estimated separately.

The cost estimate for the access road includes the following items:

- Camp and catering;
- Mobilization;
- Stripping and clearing;
- Rock and earthworks;
- Surfacing;
- Culvert and geotextile; and
- Engineering QA/QC.

21.7 Indirect Costs

Indirect costs include items that are necessary for the completion of the Project but are not part of the direct costs. They are considered Project indirects and are in addition to contractor indirects. The indirect costs are shown in Table 21.12.

Table 21.12: Indirect Capital Cost Estimate (WBS 9000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
9000	**Indirects**			
9100	Camp & Catering	7.1	0	7.1
9300	Construction Field Indirects	12.8	0	12.8
9400	Freight & Logistics	6.9	4.7	11.5
9500	Vendors Reps.	0.7	0	0.7
9600	Start-up & Commissioning	1.5	0	1.5
9700	Spares	2.1	0	2.1
9800	First Fills	0.7	0	0.7
	Total Indirect Costs	**31.7**	**4.7**	**36.4**

Source: JDS 2015

21.7.1.1 Camp & Catering

Camp and catering costs have been estimated based on the approximate camp size and construction schedule. Database pricing has been carried based on similar projects per man-day, catering prices are based on recent quotations.

21.7.1.2 Construction Field Indirects

Construction field indirect costs include the following items:

- Construction support;
- Equipment rentals and purchases;
- Water management equipment;
- Temporary construction facilities;
- First aid & medical;
- Waste management;
- Mobilization/demobilization; and
- Contractor supervision.

21.7.1.3 Freight/Logistics

Freight costs include the following items:

- Ground freight;
- Air freight costs; and
- Backhaul costs.

21.7.1.4 Vendor Representatives

Vendor representatives will be required at the project site during construction to verify that the installation of the main equipment has been performed in compliance with technical specifications. Representatives will also be required during the pre-commissioning stage. Vendor representative costs are 2% of the total equipment and materials less mining fleet and mobile equipment.

21.7.1.5 Commissioning and Start-up

Commissioning and start-up costs were based on supervision required for the plant and major equipment. Commissioning costs are 4% of the total equipment and materials less mining fleet and mobile equipment. In addition, the power plant commissioning costs were broken out by the supplier.

21.7.1.6 Spare Parts

Spare parts have been considered for start-up, one year of operations and capital. Spare parts costs are 6% of the total equipment and materials less mining fleet and mobile equipment. In addition, the power plant spare part costs were broken out by the supplier.

21.7.1.7 First Fills

First fills are required for start-up and include the following:

- Lime and reagents;
- Lubricants;
- Glycol for district heating; and
- Other fills for initial setup.

First fills costs are 2% of the total equipment and materials less mining fleet and mobile equipment.

21.8 Engineering, Procurement, and Construction Management (EPCM)

The EPCM estimate uses a first principles approach based on man-hours and consultant rates. EP costs are based on 15 months of detailed engineering and the CM costs are based on 18 months of construction. The EPCM costs are summarized in Table 21.13.

Table 21.13: EPCM Capital Cost Estimate (WBS 10000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
10110	Engineering & Procurement – EP	5.0	0.7	5.8
10120	Construction Management – CM	13.9	0.7	14.6
	Total EPCM Costs	**18.9**	**1.5**	**20.4**

Source: JDS 2015

Associated services include the following:

- Detailed engineering;
- Procurement;
- Contract management;
- Construction management and supervision;
- Administration and document control;
- Field engineering;
- Quality assurance/quality control (QA/QC);
- Health and safety;
- Surveying; and
- Commissioning.

21.9 Owner's Costs

Owner's costs that are included in the cost estimate are based on the following:

- Owner's team and consultants during the implementation phase. This includes owner's labour, offices, owner's consultants, and head office overhead and costs during detailed engineering and construction period;
- Third party costs, such as environmental studies, permits and geotechnical studies;
- Insurances and fees;
- Owner's start-up and commissioning crew;
- Recruitment and training of operation and maintenance staff;
- Community associated costs;

- Corporate affairs and administration; and
- All internal Kaminak fees and costs.

A summary of the Owner's costs are shown in Table 21.14.

Table 21.14: Owner's Cost Estimate (WBS 11000)

WBS	Description	Initial ($M)	Sustaining ($M)	LOM Total ($M)
11000	G&A labour	4.7	0	4.7
11000	Health, Safety, Medical & First Aid	0.2	0	0.2
11000	Environmental	0.6	0	0.6
11000	Human Resources	0.1	0	0.1
11000	Insurance & Legal	1.6	0	1.6
11000	External Consulting	0.1	0	0.1
11000	IT & Communications	0.3	0	0.3
11000	Office & Miscellaneous Costs	0.1	0	0.1
11000	Satellite Offices	0.1	0	0.1
	Total Owner's Cost	**7.9**	**0**	**7.9**

Source: JDS 2015

21.10 Contingency

Contingency is a provision of funds for unforeseen or inestimable costs within the defined Project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy. The contingency is meant to cover events or incidents that occur during the course of the Project, which cannot be quantified during the estimate preparation and do not include any allowance for Project risk. No provision is made, or contingency allowed, for design changes or changes to the scope of work.

It is important to note that contingency does not cover force majeure, adverse weather conditions, government policy changes, currency fluctuations, escalation and other Project risks. As well, the contingency will be based solely on the capital estimate and no other Project risks, such as schedule delays or HAZOP assessments.

The contingency factors take into account that most mobile and equipment cost estimates were based on quotations and, therefore, attracted a lower quantum of contingency. Similarly, usage was generally built up from first principles. Subcontract or "Other" costs generally included vendor packages with a mix of equipment and labour costs and, consequently, a slightly higher contingency factor was applied. Table 21.14 shows the contingency factors used in the "Initial" and Sustaining Capital "S/C" estimate.

Table 21.15: Contingency Factors (WBS 12000)

Capital Cost Category	Labour (%)	Materials (%)	Equip (%)	Other (%)	Initial CAPEX (%)	Sust. CAPEX (%)	LOM Total (%)
Mining	0	0	0.5	8	2.1	2.1	2.1
On-Site Development	0	0	12	15	14.0	14.0	14.0
Ore Crushing & Handling	14	12	12	12	12.5	12.5	12.5
Heap Leach	14	12	0	15	14.6	14.6	14.6
Process Plant	14	12	12	12	12.4	12.4	12.4
On-Site Infrastructure	14	12	12	12	12.3	12.3	12.3
Off-Site Infrastructure	0	0	0	10	10.0	10.0	10.0
Indirects	10	10	10	10	10.0	10.0	10.0
EPCM	10	0	0	10	10.0	10.0	10.0
Owner's Costs	0	0	0	10	10.0	10.0	10.0
Closure*	0	0	0	0	0	0	0
Total Contingency %					10.4	11.4	8.3
Total Contingency ($M)					26.0	7.3	33.4

Source: JDS 2015

*A contingency allowance has been carried in the closure costs estimate; contingency has not been included in the above table.

A contingency of 0.5% was applied to mining equipment due to the assumption of the entire fleet provided by one vendor and misc. mining equipment was estimated based on historical data. Mine development received an 8.0% contingency, which was lower than the average contingency due to costs built up from first principles and recent quotations.

Site development earthworks and heap leach costs are primarily made up of sub-contracts (i.e. all-in rates). Earthwork quantities typically vary once construction commences; therefore, a larger contingency of 15.0% was applied.

Crushing, process equipment, and infrastructure items received the same contingency of 12.0% for materials, equipment and subcontract as majority of the equipment materials were quoted at a feasibility level of 10-15%. Labour was based on historical data; therefore, more risk is associated with the costs and required a larger contingency of 14.0%.

Indirects and EPCM received a 10.0% contingency due to costs built up from first principles.

21.11 Sustaining Capital

The main sustaining capital cost comprises of the heap leach and open pit mining during the operations phase.

The following sustaining capital items will be required for the site:

- Open pit sustaining capital is used for the replacement of equipment over the mine life;
- On-site development sustaining capital is used for required drainage and water management on site;
- Heap leach sustaining capital is used for running additional pipelines, pad expansion, and larger ponds. Work includes additional earthworks, liners, and piping;
- Process plant sustaining capital is used for adding the CN Detox system;
- On-site infrastructure sustaining capital is used for additional gensets and AN storage capabilities; and
- Indirect & EPCM sustaining capital is used for additional freight of materials and equipment and engineering/construction supervision during the expansion of the heap leach.

21.12 Reclamation and Closure Cost Estimate

Progressive reclamation and closure activities will begin as soon as mining at the Double Double pit has been completed in Year 2 and will continue through the rest of the 10-year mine life.

Mine closure occurs in four well-defined phases:

- Operational closure – Years 2 to 10: as and when each pit, WRSF and stages of the HLF are decommissioned they will be closed;
- Post-mining closure – Years 11 to 15: closure activities relating to terminating the mining operation, dismantling infrastructure and reclamation;
- Active closure – Years 16 to 20: maintenance, monitoring and closure of remaining facilities; and
- Post-closure – Year 21 onwards: monitoring only.

Mine closure and reclamation activities include the following:

- Heap leach rinsing and re-sloping and capping;
- Managing hazardous waste;
- Dismantling and disposing of all structures and equipment;
- Landfilling all inert waste, including equipment drained of all oils and hazardous materials;
- Transporting all hazardous waste from the Project sites;

- Disposing all liners and pipelines;
- Collecting and treating all contaminated soils;
- Re-contouring the site areas to provide positive drainage; and
- Scarifying, placement of re-vegetation layer and seeding of disturbed surfaces.

The following assumptions were used to build up the closure cost estimate and are summarized in Table 21.16:

- Mobile equipment required for closure was assumed to be rental or third party equipment;
- Labour was assumed to be provided by the owner using fully burdened labour rates;
- Unit cost estimates were based on the rental equipment fleet; and
- No salvage recovery was included in the closure cost estimate.

Table 21.16: Basis of Closure and Reclamation Estimate Summary

Category	Estimate Basis
Open pit	Open pit closure costs were estimated by applying unit costs from first principles and quantities based on current designs.
Heap leach closure	Heap leach closure costs were estimated by applying unit costs from first principles and estimated quantities based on current designs. Liner placement was based on vendor quotes.
Water management structures	Water management structure closure costs were estimated by applying unit costs to material quantities estimated on current designs.
Buildings and equipment	Buildings and equipment closure costs were estimated using previous project-closure production data which was scaled by area and material quantities.
Site pads, roads and airstrip	Infrastructure pads, roads and airstrip closure costs were estimated by applying unit costs from first principles and quantities based on the current design for restoring drainage, scarifying and re-vegetation layer. Seeding and fertilizing units costs were based on vendor quote from previous projects.
Access road	Access road closure costs were estimated by applying unit costs from first principles and quantities based on the current design for restoring drainage, scarifying and re-vegetation layer.
Water treatment	Water treatment plant capital costs were based on previous projects. Treatment costs were based on reagent costs for similarly sized equipment. Labour costs were based on owner provided labour.
Contaminated Soil	Soil investigations were based on unit costs per metre drilled at the required intervals over the testing footprint.
Post mining and active closure Monitoring	Cost allowances were based on similar projects.
Indirects	Indirect costs were based on the required man-days to complete post mining and active closure activities and associated accommodation, fuel, site services, tool, transportation and supply costs. Indirect costs also include allowances for project management, general and administrative, helicopter support and a 12% contingency.

Source: JDS 2015

The estimated closure costs by period are shown in Table 21.17.

Table 21.17: Closure Costs by Period

Cost Category	Total Cost (M$)	%
Operational closure	15.9	26
Post-mining closure	29.5	49
Active closure	15.1	25
Total	**60.5**	**100**

Source: JDS 2015

21.13 Capital Cost Exclusions

The following items have been excluded from this capital cost estimate:

- Working or deferred capital;
- Financing costs;
- Refundable duties;
- Currency fluctuations;
- Lost time due to severe weather conditions;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resulting from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any Project sunk costs (studies, exploration programs, etc.);
- Escalation cost;
- Depreciation and depletion allowances;
- Environmental permits;
- Performance bond;
- Builders risk insurance;
- Surface land rights, including water and wildlife compensation;
- Water rights acquisition, and
- Hiring and relocation.

22 Operating Cost Estimate

22.1 Introduction & Summary

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 JDS Feasibility Study Budget Class 3 Estimate.

The operating cost estimate is broken into seven major sections:

- Open Pit Mining;
- Crushing, Heap Leach and Processing;
- Site Services; and
- General & Administrative.

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 Q3 Year -1 are capitalized; subsequent periods commencing in Q4 Year -4 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 $24.10 per tonne processed. Average annual operating costs and total unit costs are summarized in Table 22.1.

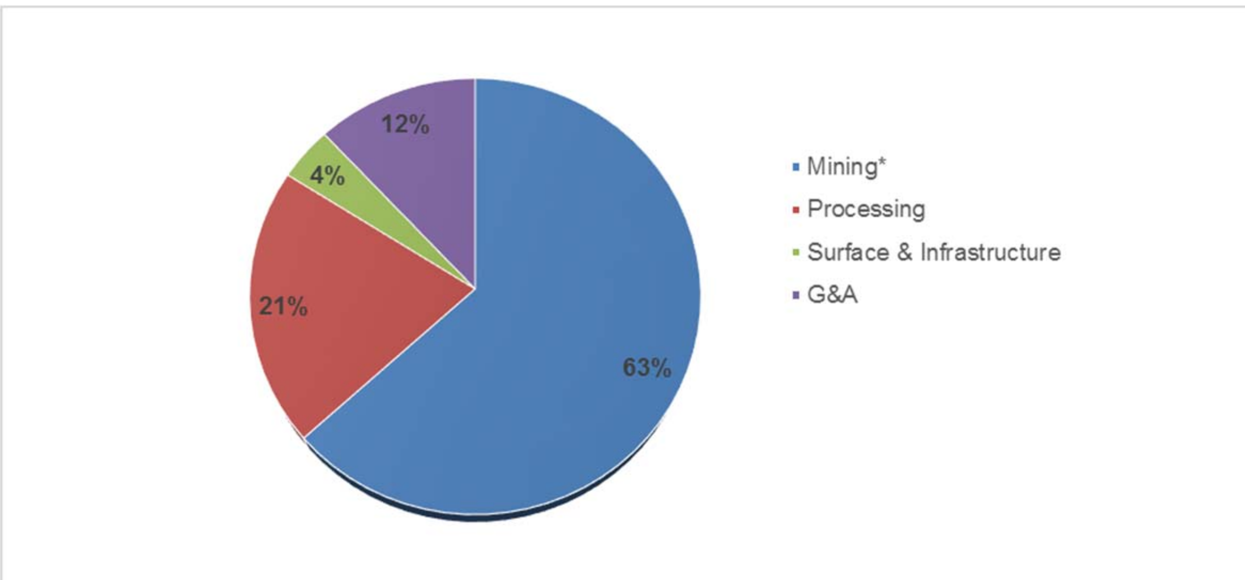
Figure 22.1 and Figure 22.2 illustrate the operating cost distribution. Annual operating costs by area are outlined in Table 22.2.

Table 22.1: Estimated Average Operating Cost by Area

Operating Cost†	Average $M/a	LOM $M	Unit Costs $/t Processed¤
Open Pit Mining*	77.0	707.4	15.26
Processing	25.1	230.3	4.97
Site Services	4.9	45.2	0.97
General & Administrative	14.6	134.2	2.89
Total Operating Costs	**121.7**	**1,117.1**	**24.10**

(†): Operating Costs include the working capital during the pre-production period
(*): Average LOM Open Pit Mining Cost amounts to $2.27/t mined at a 5.7 strip ratio
(¤): $/t leached calculated based on tonnes leached once gold production begins (Y-1Q3). A total of 46.4 Mt are leached during this period.
Source: JDS 2015

Figure 22.1: Total Operating Cost Distribution by Area



Source: JDS 2015

Table 22.2: Annual Operating Cost by Area

	Unit	LOM	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Annual Operating Cost													
Open Pit Mining	$M	707.4	21.6	58.8	83.2	83.6	86.0	89.0	89.9	93.3	80.1	21.8	0.0
Processing	$M	230.3	11.1	26.1	26.1	27.1	26.1	24.4	24.2	24.7	24.4	14.3	1.7
Site Services	$M	45.2	1.9	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.7	4.0	0.9
General & Administrative	$M	134.2	3.2	14.0	14.6	14.8	14.8	14.8	14.9	14.9	14.7	11.7	1.8
Total Operating Costs*	$M	1,117.1	37.8	103.7	128.8	130.3	131.7	133.1	133.9	137.7	123.9	51.9	4.4
Total Ore Processed	M tonnes	44.1	1.4	4.9	6.6	4.5	4.4	4.6	5.2	5.1	6.2	1.3	0.0
Unit Operating Cost by Year													
Open Pit Mining	$/t leached	$15.26	10.98	11.94	12.68	18.74	19.75	19.16	17.19	18.45	12.85	16.70	0.00
Processing	$/t leached	$4.97	4.43	5.30	3.98	6.08	5.98	5.26	4.63	4.89	3.92	10.97	0.00
Site Services	$/t leached	$0.97	0.95	0.98	0.73	1.08	1.10	1.04	0.92	0.96	0.75	3.06	0.00
General & Administrative	$/t leached	$2.89	2.37	2.85	2.23	3.31	3.39	3.19	2.85	2.95	2.36	8.98	0.00
Unit Operating Costs	$/t leached	$24.10	18.73	21.07	19.62	29.20	30.23	28.64	25.59	27.25	19.88	39.70	0.00

Note: Total Operating Costs includes working capital which is equivalent to three months of operating costs (approx. $23M) incurred during Year -1.

Source: JDS 2015

22.2 Operations Labour

This section provides an overview of total workforce and the methods used to build the labour rates.

Table 22.3 summarizes the total planned workforce during project operations.

Table 22.3: Summary of Peak Employment by Area

Department	Total Persons Employed (Peak)
Open Pit Mining	248
Processing	73
Site Services	117
Total	438

Source: JDS 2015

Labour base rates were determined by reference to other northern Canadian operations and benchmarked against Costmine (Canadian Mine Salaries, Wages, Benefits 2014 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;
- Travel pay of eight hours per rotation 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 6% of scheduled hours; and
- Insurance allowance of 8% of base pay.

22.3 Open Pit Mine Operating Costs

Open pit mining activities were assumed to be undertaken by the owner, except for the grade control RC drilling program. Costs are presented in 2015 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 gold-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 and WRSF operations, road maintenance, mine supervision, technical services and a grade control program.

The average open pit operating costs for the LOM plan are presented in Table 22.4 and Table 22.5, 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).

The distribution of operating costs by activity and category are illustrated in Figure 22.2 and Figure 22.3, respectively.

Table 22.4: Open Pit Operating Cost Estimate – by Activity

Activity	$/t mined*
Drill and Blast**	0.83
Load and Haul	1.14
Mine General	0.07
Mine Maintenance	0.16
Technical Services	0.07
Total Open Pit Operating Cost	2.27

Note:
*Excludes pre-production period
**Includes contractor RC grade control drill program
Source: JDS 2015

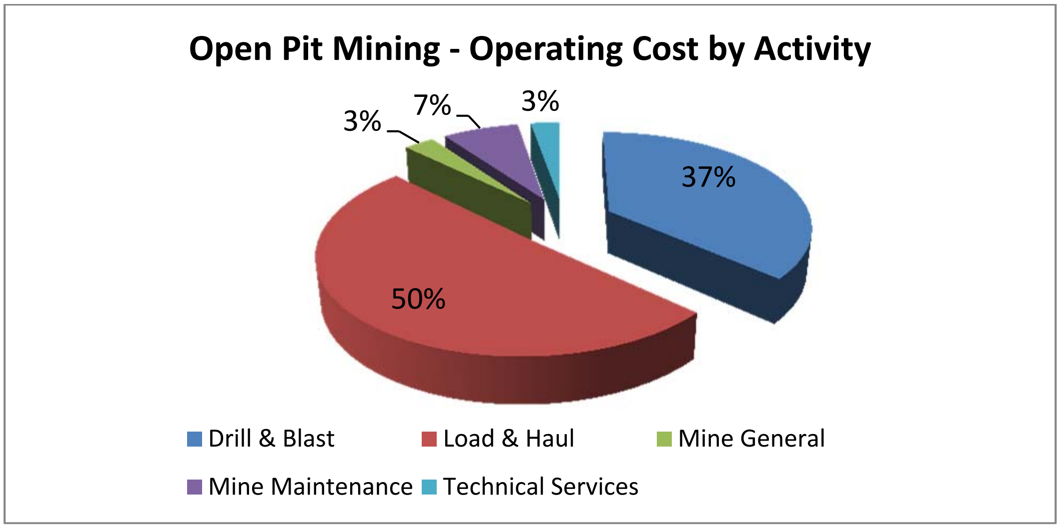
Table 22.5: Open Pit Operating Cost Estimate – by Category

Category	$/t mined*
Labour	0.59
Power and Fuel	0.59
Parts & Repair	0.54
Lubrication	0.04
Wear Items	0.03
Tires	0.07
Explosives	0.34
Services	0.07
Total Open Pit Operating Cost	**2.27**

*Note: Excludes pre-production period

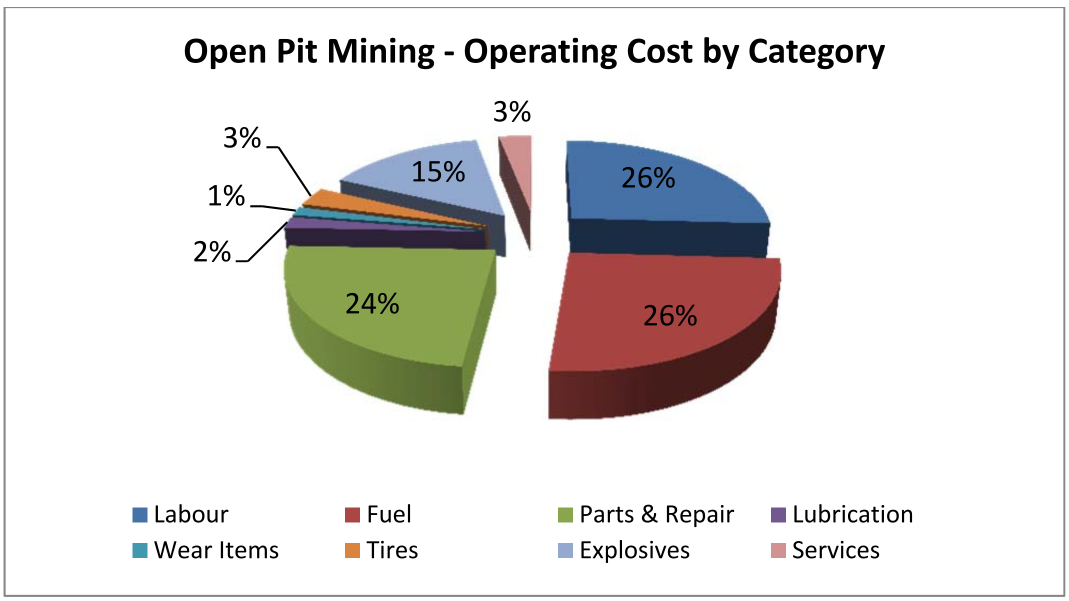
Source: JDS 2015

Figure 22.2: Distribution of Open Pit Mine Operating Costs by Activity



Source: JDS 2015

Figure 22.3: Distribution of Mine Operating Costs by Category



Source: JDS 2015

22.3.1 Basis of Estimate

22.3.1.1 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.

22.3.1.2 Open Pit Labour

Mining labour was 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 were benchmarked against similar operations. Table 22.6 summarizes the direct open pit mining workforce labour quantities and rates.

Table 22.6: Open Pit Labour Complement and Rates

Position	Manpower Complement	Manpower On Site	Shift Rotation	Salaried/Hourly	Loaded Salary per Year Hourly Rate
Supervision					
Operations Manager	1	1	2&2	Salaried	$258,400
Mining Superintendent	1	1	2&2	Salaried	$152,400
Mine Shift Foreman	6	3	2&2	Salaried	$125,900
Maintenance Superintendent	1	1	2&2	Salaried	$152,400
Maintenance Shift Foreman	4	2	2&2	Salaried	$104,700
Subtotal - Supervision	13	8			
Drill & Blast					
Driller, Blasthole	27	14	2&2	Hourly	$52
Blaster	2	1	2&2	Hourly	$52
Blasting Helper	4	2	2&2	Hourly	$42
Bulk Explosives Truck Driver	2	1	2&2	Hourly	$49
Subtotal - Drill & Blast	35	18			
Load & Haul					
Shovel/Loader Operator	25	13	2&2	Hourly	$56
Truck Driver	68	34	2&2	Hourly	$49
Track Dozer Operator	17	9	2&2	Hourly	$52
R.T. Dozer Operator	8	4	2&2	Hourly	$52
Grader Operator	9	5	2&2	Hourly	$52
Water/Ancillary Truck Diver	3	2	2&2	Hourly	$49
Labourer/Trainee	13	7	2&2	Hourly	$42
Subtotal - Load & Haul	**127***	**74**			
Maintenance					
Heavy Equipment Mechanic	12	6	2&2	Hourly	$63
Welder/Mechanic	12	6	2&2	Hourly	$63
Electrician/Instrument	6	3	2&2	Hourly	$63
Lube/PM Mechanic/Light Duty Mechanic	12	6	2&2	Hourly	$63
Tireman	4	2	2&2	Hourly	$42
Labourer/Trainee	6	3	2&2	Hourly	$42
Subtoal - Maintenance	**52**	**26**			
Technical Services					
Maintenance Planner	1	1	2&2	Salaried	$104,700
Chief Mining Engineer	1	1	2&2	Salaried	$184,200
Senior Mine Engineer	1	1	2&2	Salaried	$147,000
Mine Engineer	2	1	2&2	Salaried	$104,700
Ore Control Engineer	2	1	2&2	Salaried	$104,700

Position	Manpower Complement	Manpower On Site	Shift Rotation	Salaried/Hourly	Loaded Salary per Year Hourly Rate
Mine Technician	2	1	2&2	Salaried	$88,400
Surveyor	2	1	2&2	Salaried	$88,400
Surveyor Assistant	4	2	2&2	Salaried	$88,400
Clerk	1	1	2&2	Salaried	$77,500
Chief Geologist	1	1	2&2	Salaried	$104,700
Mine Geologist	2	1	2&2	Salaried	$88,400
Technician/Ore Control	2	1	2&2	Salaried	$88,400
Subtotal - Technical Services	21	13			
Total Open Pit Mining Operations	259*	139			

(*) Totals may not match due to timing of maximum annual quantity

Source: JDS 2015

22.3.1.3 Mine Consumable Requirements

Consumable cost estimates were assembled from equipment suppliers, cost services, factors and JDS experience.

The diesel fuel price of $0.90/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 22.7 with unit costs for tires, lubes and explosives. Maintenance parts and repair vary by equipment fleet and period, however, average unit costs over the life of each piece of major and support equipment are shown in Table 22.8 (excluding all labour and fuel costs).

Table 22.7: Open Pit Consumable Cost Detail

Item	Unit	Average Cost
Diesel fuel	$/litre	0.90
Lube cost	$/litre	6
Tires - haul trucks	$/ea	22,300
Blasting Supplies		
AN cost	$/kg	0.99
Emulsion cost	$/kg	1.68
Handidets_12m	$/ea	10.08
Handidets_15m	$ ea	12.24
Trunk_Line_4m	$/ea	5.04
Trunk_Line_6m	$/ea	6.05
Boosters	$/ea	6.39

Source: JDS 2015

Table 22.8: Average Maintenance and Repair Cost on Open Pit Equipment

Open Pit Equipment	Average Cost ($/hr)
Drill #1 (229 mm dia.)	170
Drill #3 (76 mm dia.)	33
Shovel #1 (15 m^3)	151
Shovel #3 (4.5 m^3)	30
Loader #1 (11.5 m^3)	138
144-t class haul truck	121
Track dozer (D275)	72
Wheel dozer (WD500)	49
Grader (GD825)	68
Water truck (75 m^3)	76

Source: JDS 2015

22.3.2 Drill and Blast Operating Cost

The average LOM drilling and blasting operating cost is $0.83 t/mined for a total of $259M. The drilling operation includes the owner-operated drill fleet and the contract RC grade control drill program.

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;
- Contract services

For blasting agents, an average mixture of 85% ANFO (a mixture of 94% ammonium nitrate prill and 6% diesel fuel) to 15% emulsion (to account for wet loading conditions) was assumed. Annual explosive consumption was calculated based on mine scheduling and drill productivity.

Annual bulk explosive requirement by year is shown in Table 22.9 (based on material and drill type). The overall powder factor is estimated at 0.32 kg/t. Table 22.10 summarizes LOM drill and blasting costs for the Coffee Gold Project.

Table 22.9: Annual Open Pit Explosive Requirements for Blasting

Year	ANFO (t)	EMULSION (t)	Total Material Blasted (kt)
Y -1	2,038	360	7,474
Y 1	6,172	1,089	22,524
Y 2	9,445	1,667	34,598
Y 3	9,546	1,685	35,263
Y 4	10,674	1,884	39,519
Y 5	10,723	1,892	39,662
Y 6	10,584	1,868	39,060
Y 7	10,771	1,901	39,785
Y 8	9,791	1,728	35,945
Y 9	1,819	321	6,656
Total	**881,563**	**14,394**	**300,487**

Source: JDS 2015

Table 22.10: Drill and Blast Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost ($M)*
Labour	0.08	23
Fuel (drill and blasting requirements)	0.14	45
Maintenance & Operating Consumables		
Parts & Repair	0.11	32
Lubrication	0.01	3
Wear Items (bits, undercarriage, etc.)	0.09	26
Explosives and Accessories	0.36	112
Services	0.06	18
Total	**0.83**	**259**

*Note: Excludes pre-production period

Source: JDS 2015

22.3.3 Load and Haul Operating Cost

The average LOM loading and hauling operating cost is $1.14 t/mined for a total of $358M and represents 50% of the total mine operating cost. The load and haul cost includes delivery of the ore to the crusher and the heap leach pad as well as rehandling costs associated with the ROM stockpile.

Loading and hauling costs include:

- Labour;
- Diesel fuel;
- Tires and rims;
- Oils and lubricants;
- Repair and maintenance parts; and
- Wear items (buckets, teeth, undercarriage structures).

The estimated cost of loading and hauling over the LOM is shown in Table 22.11.

Table 22.11: Load and Haul Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost ($M)*
Labour	0.26	80
Fuel	0.43	132
Maintenance &Operating Consumables		
Parts & Repair	0.33	105
Lubrication	0.03	10
Wear Items (buckets, teeth, undercarriage, etc.)	0.03	10
Tires	0.07	22
Total	**1.14**	**358**

*Note: Excludes pre-production period

Source: JDS 2015

22.3.4 Mine General Operating Cost

The average LOM mine general operating cost is $0.07 t/mined for a total of $20M. 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 22.12.

Table 22.12: Mine General Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost ($M)*
Labour	0.03	10
Support Equipment	0.03	9
Tools, Supplies	0.01	1
Total	0.07	20

*Note: Excludes pre-production period

Source: JDS 2015

22.3.5 Mine Maintenance Operating Cost

The average LOM mine maintenance cost is $0.16 t/mined for a total of $51M. 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 Kaminak personnel using the company's own installations.

The LOM mine maintenance cost is summarized in Table 22.13.

Table 22.13: Mine Maintenance Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost ($M)*
Labour	0.15	49
Tools, Supplies	0.01	1
Total	**0.16**	**51**

*Note: Excludes pre-production period

Source: JDS 2015

22.3.6 Technical Services Operating Cost

The average LOM technical services cost is $0.07 t/mined for a total of $19M. This encompasses labour costs required for the technical services group, including all mine engineering staff and the mine geology group.

Table 22.14 summarizes the LOM technical serves operating cost.

Table 22.14: Technical Services Cost

Category	Average LOM cost ($/t mined)*	Total LOM Cost ($M)*
Labour	0.06	17
Tools, Supplies	0.01	2
Total	0.07	19

*Note: Excludes pre-production period

Source: JDS 2015

22.4 Process Operating Costs

The processing operating cost estimate includes operating and maintenance costs for:

- Crushing;
- ADR process plant;
- HLP piping and drip emitter installation and maintenance; and
- Barren and pregnant solution handling between the HLP and plant.

Although the power plant provides electricity across all infrastructure facilities, a large component of the power consumed is for the process plant (particularly crushing). Consequently, the power cost is included in the process operating cost section as a kWhr unit cost. Labour assigned to the power generating sets is included as a plant operating cost.

Crushed-ore handling including loading, hauling and spreading are included in the mining operating costs.

Mobile equipment costs are included in infrastructure costs described in Section 18.

The process operating costs take into account the cessation of crushing and reduced gold recovery during Q1 of each year.

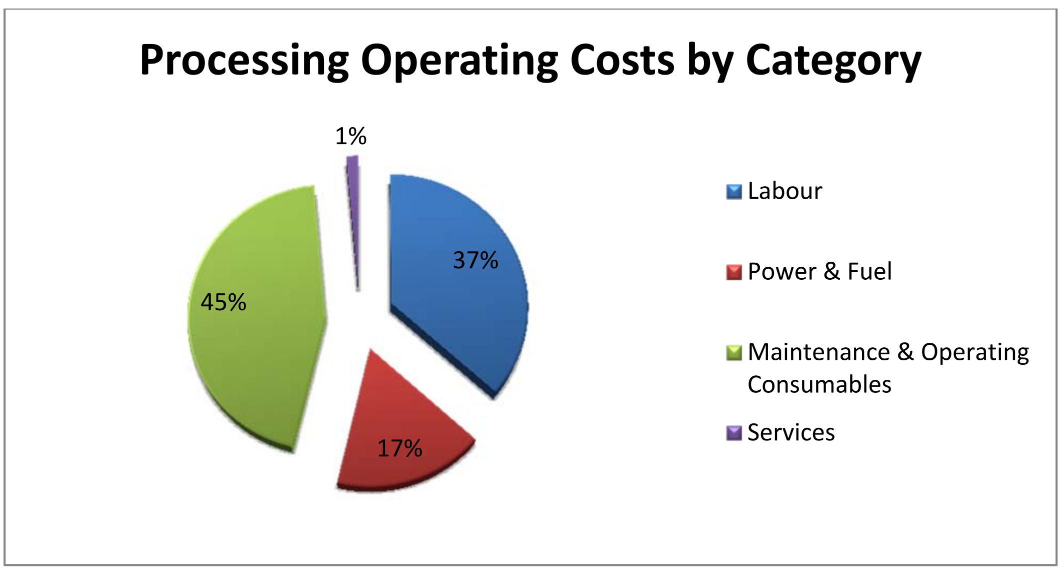
A summary of the process plant operating cost is presented in Table 22.15. A graphical representation of the process operating cost breakdown is depicted in Figure 22.4.

Table 22.15: LOM Processing Operating Costs

Cost Category	$/t processed
Labour	1.81
Power & Fuel	0.85
Maintenance & Operating Consumables	2.23
Services	0.08
Total Process Operating Costs	4.97

Source: JDS 2016

Figure 22.4: Process Operating Cost Distribution by Category



Source: JDS 2016

22.4.1 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 22.16. The number of personnel required for the process plant was developed from similar projects and operating mines. Labour costs are considered fixed operating costs and do not vary with annual throughput. Total labour costs are $1.81/t of ore processed and account for 37% of the overall process operating costs.

Table 22.16: Process Labour Complement and Rates

Position	Manpower Complement	Manpower On Site	Shift Rotation	Salaried/ Hourly	Fully Loaded Rate $/a
Mill Staff					
Mill Superintendent	1	1	4X3	Salaried	152,359
Operations Shift Foreman	2	1	2&2	Salaried	125,859
Plant Metallurgist	2	1	2&2	Salaried	104,659
Metallurgical Technician	2	1	2&2	Salaried	88,414
Mill Operations					
Control Room Operator	4	2	2&2	Hourly	121,756
Crusher Operator	4	2	2&2	Hourly	101,074
Carbon Plant Operator	4	4	2&2	Hourly	121,756
EW/Gold Room Operator	4	2	2&2	Hourly	121,756
Helpers	4	2	2&2	Hourly	75,196
Helpers (crushing)	4	2	2&2	Hourly	62,423
HL Operations					
HL PAD Operators (two for drip lines and two for welding pipe)	8	4	2&2	Hourly	91,734
Laboratory					
Lab Manager	2	1	2&2		99,359
Assayer	6	3	2&2	Hourly	106,360
Sample Prep/Trainees	4	2	2&2	Hourly	90,963
Power Plant					
Power Plant Operator	4	2	2&2		121,756
Millwright	2	1	2&2		137,147
Mill Maintenance					
Maintenance Superintendent	1	.5	2&2		152,359
Maintenance Planner	1	.5	2&2		104,659
Millwrights/Welders	8	4	2&2		137,147
Electricians	2	1	2&2		137,147
Instrumentation	2	1	2&2		137,147
Apprentice	8	4	2&2		106,360
Total - Labour	**79**	**42**			

Source: JDS 2016

22.4.2 Power & Fuel

The power and fuel costs for the crushing and process plants and solution heating account for approximately $0.8/t or 17% of the total process operating cost. Power for the process plant will be supplied at $0.22/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 $0.86/L. Diesel costs are for the operation of the kiln and boiler in the plant as well as the barren solution boiler. The majority of diesel consumed will be to heat the barren solution recirculating through the HLP from November to April to maintain a minimum temperature at the drip emitters of 6^{o}C

22.4.3 Maintenance and Operating Consumables

Maintenance and consumables account for 45% of the process operating costs. The maintenance and consumable costs are summarized in Table 22.17.

Table 22.17: Maintenance and Operating Consumables Cost Summary

Consumables	$/t processed
HL Operations – Piping & Drip Emitters/Liners	0.26
Liners	0.12
Reagents	1.70
Maintenance	0.15
Total	2.23

Source: JDS 2016

22.4.3.1 Heap Leach Operations

The cost to provide piping and drip emitters on each lift of the HLP and raincoats are accounted for under heap leach operations. The piping and drip emitter costs were based on a cost per square metre of lift area. In Y-1 and on the last lifts place on the HLP each year the piping and drip emitters will be duplicated and buried to reduce the possibility of production losses due to freezing. The cost for HL operations is $0.26/t of ore processed.

Commencing in Q2 Year 3 and in subsequent years an allowance for raincoats, to reduce dilution of the HLP, are included in the operating costs.

22.4.3.2 Crusher Liners

Jaw and cone crusher liner costs were based on a combination of vendor data and experience. Crusher liner costs were provided by vendors. An additional $150,000/y was allotted for miscellaneous spares. A total cost of $0.12/t was included for crusher liners and miscellaneous spares.

22.4.3.3 Reagents

The reagent consumption and cost summary is presented in Table 22.18. The quantity of reagents required for the operation is based on testwork, 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 22.18: Reagent Consumption Costs

Reagents	Annual Usage (t/a)	Cost per tonne Including Freight ($)	Processing Cost ($/t)
Lime	7,500	490	0.69
NaCN	1,005	4,231	0.81
HCL	201	560	0.02
NaOH	181	1,346	0.05
Carbon	55	4,103	0.05
Antiscalant	39	4,936	0.04
–			0.04
Total			**1.70**

Source: JDS 2016

22.4.3.4 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 $0.15/t of ore processed.

22.4.4 Services

The cost of operating the assay laboratory was provided by vendors. The cost per year was estimated to be $373k or approximately $0.08/t.

22.5 Power Costs

A single captive power plant will be used to meet the electrical power demand necessary to support the complete Coffee Site operation. The Power Plant design is based on four diesel-fired reciprocating engine generator sets (gensets) in an N+2 (2+2) arrangement.

Power costs have been calculated based on fuel consumption rates specified by the manufacturer based on the calculated plant load factor. The average fuel consumption rate for the power plant is 0.251 L/kWh. The fuel consumption rate was multiplied by the delivered diesel price of $0.856/L to obtain an operating cost of $0.215/kWh. A $0.09/kWh repair and maintenance cost was added to obtain a total power cost for the Project of $0.224/kWh.

Power costs excluded plant operations and maintenance labour as these are included in process plant labour costs.

22.6 Infrastructure & Site Services Operating Costs

Infrastructure and site services operating costs account for the costs such as site services support, access road operations and maintenance, annual winter road construction and infrastructure operations and maintenance.

Table 22.19 summarizes the infrastructure and site services operating costs.

Table 22.19: Summary of Site Services & Infrastructure Costs

Site Services Costs	$/t processed
Infrastructure Operations	0.30
Site Services Support	0.48
Access Road Operations & Maintenance	0.14
Winter Road Construction	0.06
Total Site Services Costs	0.97

Source: JDS 2016

22.6.1 Infrastructure Operating Costs

Infrastructure operating costs include all costs associated with infrastructure activities for a large-scale mining operation in a remote northern Canadian location. The scope includes:

- Power for all surface infrastructure facilities including parking areas, laydown areas, fuel and lube supply and distribution, maintenance and fleet facilities, ANFO mixing, permanent camp, water supply and distribution, and truck shop and maintenance;
- Consumables; and
- Maintenance.

The estimated LOM site services cost is $0.30/t of ore processed or $1.5M per year. The cost estimates are shown in Table 22.20.

Table 22.20: Infrastructure Operations Costs

Infrastructure Operations	LOM $M	$M/a	$/t Processed
Power	12.1	1.3	0.26
Consumables	0.2	0.02	0.004
Maintenance	1.5	0.2	0.04
Total	13.8	1.5	0.30

Source: JDS 2016

22.6.1.1 Power

Power for the surface infrastructure at Coffee is supplied at $0.224/kWh as described in section 22.5. Annual surface infrastructure power consumption is estimated to be 5,988 MWh/yr.

22.6.1.2 Consumables

Consumables required for site services includes diesel for the waste incinerator estimated to be 21,900 L/a. The diesel cost is $0.856/L delivered to site.

22.6.1.3 Maintenance

The cost of maintenance parts is estimated at 1% of purchased mechanical and electrical equipment costs labour costs are included in the site infrastructure labour cost.

These maintenance costs are assumed to remain constant throughout the life of the mine.

Table 22.21: Site Infrastructure Maintenance Costs

Description	% Cost of Capital/yr	Mech/Elec Equipment Capital Cost ($M)	Total Annual Maintenance Cost ($000's)
Airstrip Equipment	1	1.3	13
Water Supply & Distribution	1	1	10
Permanent Camp	1	5.1	51
Waste Management	1	0.4	4
Ancillary Facilities	1	5.6	56
Bulk Fuel Storage & Distribution	1	2.6	26
Total Maintenance Cost			**160**

Source: JDS 2016

22.6.2 Site Services Support

Site services costs include support services for the Coffee site:

- Labour costs for surface services and maintenance of infrastructure facilities;
- Surface mobile equipment operations and maintenance;

The manpower required to perform site services work is listed in Table 22.22.

.

Table 22.22: Site Services Support Rates & Quantities

Position	Manpower Complement	Manpower Onsite	Shift Rotation	Salaried/Hourly	Loaded Annual Salary $
Surface foreman	2	1	2&2	Salaried	104,659
Electrician	2	1	2&2	Hourly	137,147
Facilities maintenance - tradesman	2	1	2&2	Hourly	137,147
Mobile equipment operator	6	3	2&2	Hourly	114,058
Labourers/apprentices	4	2	2&2	Hourly	90,963
Total	**16**	**8**			

Source: JDS 2016

Surface mobile equipment operations and maintenance costs include fuel and maintenance for each piece of support equipment shown in Table 22.23. Costs are based on an allowance for operating hours per year.

Table 22.23: Support Equipment Quantities

Equipment Description	Equipment Quantity
1 T Diesel Crew Cab Pick-up - Ford F350	6
2 T Diesel Pick-up c/w Heated Van - Ford F550	1
5 T Flat Deck Truck	1
20T Picker Truck - Western Star 4900 XD	1
Water Truck	1
Vacuum Truck	1
Tractor with Deck Trailer	1
44 Passenger Bus - Freightliner	2
Ambulance/Rescue - Ford F450	1
Tool Carrier - Cat 966K (c/w Attachments)	1
Skid Steer Loader (1.0 m^3)	2
Excavator (~1.0 m^3) CAT 320DL	1
3 T Forklift - Warehouse - CAT 2DP6000	1
65ft Man-Lift - Genie S-65	1
Portable Diesel Heaters	4
Pipe Fuser - McElroy T900	1

Source: JDS 2016

The estimated LOM site services cost is $0.48/t processed or $2.3M per year. The cost estimates are shown in Table 22.24.

Table 22.24: Site Services Costs

Site Services	LOM $M	$M/yr	$/t processed
Equipment Maintenance	2.0	0.2	0.04
Fuel	2.9	0.3	0.07
Labour	17.2	1.8	0.39
Total	22.1	2.3	0.50

Source: JDS 2016

22.6.3 Access Road Operations & Maintenance

Access road operations and maintenance includes labour and equipment costs associated with:

- Barge operations and maintenance for open water period crossings on the Yukon and Stewart Rivers; and
- Maintenance and upkeep of the access road.

The manpower required to perform access road operations and maintenance work is listed in Table 22.25.

Table 22.25: Access Road Operations & Maintenance Labour Rates & Quantities

Position	Manpower Complement	Manpower Onsite	Shift Rotation	Salaried/Hourly	Loaded Annual Salary $
Mobile equipment operator - grader	1	1	5&2	Hourly	87,794
Barge captain	2	2	5&2	Hourly	100,391
Barge labourers	2	2	5&2	Hourly	75,001
Total	**5**	**5**			

Source: JDS 2016

Access road operations and maintenance costs include fuel and maintenance for each piece of support equipment shown in Table 22.26. Costs are based on estimated operating hours per year.

Table 22.26: Access Road Equipment Quantities

Equipment Description	Equipment Quantity
Cat 140 Grader	1
River Barge	2

Source: JDS 2016

The estimated LOM access road operations and maintenance cost is $0.14/t of ore processed or $0.7M per year. The cost estimates are shown in Table 22.27.

Table 22.27: Access Road Operations and Maintenance Costs

Access Road Operations & Maintenance	LOM $M	$M/a	$/t processed
Equipment Maintenance	2.6	0.3	0.06
Fuel	1.3	0.1	0.03
Labour	2.4	0.3	0.05
Total	6.3	0.7	0.14

Source: JDS 2016

22.6.4 Winter Ice Road (WIR) Construction

Winter ice road construction will occur annually on the Stewart and Yukon Rivers and through Coffee Creek. WIR construction costs include contractor labour and equipment costs required to construct the winter road annually.

The manpower required to perform WIR construction is listed in Table 22.28.

Table 22.28: Winter Ice Road Construction Labour Rates & Quantities

Position	Manpower Complement	Manpower Onsite	Shift Rotation	Salaried/Hourly	Loaded Annual Salary $ *
Supervisor	2	1	2&2	Hourly	103
HEO - skilled	2	1	2&2	Hourly	90
HEO - semi skilled	2	1	2&2	Hourly	75
Labourer	6	3	2&2	Hourly	59
Total	**12**	**6**			

(*) Contract labour rates

Source: JDS 2016

Mobile equipment costs are determined from hourly rates for contractor equipment. Equipment rates are inclusive of repairs, maintenance and fuel. Costs are based on an allowance for operating hours per year. Table 22.29 lists the WIR construction equipment.

Table 22.29: Winter Ice Road Equipment Quantities

Equipment Description	Equipment Quantity
1 T Diesel Crew Cab Pick-up - Ford F350	1
Dump Truck 10m3 capacity- Western Star 4900	1
Grade - Cat 140M	1
Snow Cat - Prinoth BR350	1
Water Truck c/w Plow	2
Ice Auger	2
Flood Pumps	2

Source: JDS 2016

The estimated LOM winter ice road construction cost is \$0.06/t processed or \$0.3M per year. The cost estimates are shown in Table 22.30.

Table 22.30: Winter Road Construction Costs

	LOM \$M	\$M/yr	\$/t milled
Equipment Maintenance	0.8	0.08	0.02
Fuel	0.1	0.01	0.003
Labour	1.6	0.18	0.04
Total	2.5	0.28	0.06

Source: JDS 2016

22.7 General & Administrative

General and administrative costs comprise the following categories:

- Labour;
- On-site items as such camp catering, health and safety, environmental, human resources, legal, external consulting, communications and office supplies;
- Satellite office and off-site warehousing; and
- Employee travel (to and from site).

The total G&A unit operating cost is estimated at \$2.89 per tonne processed. Table 22.31 summarizes the annual G&A operating costs.

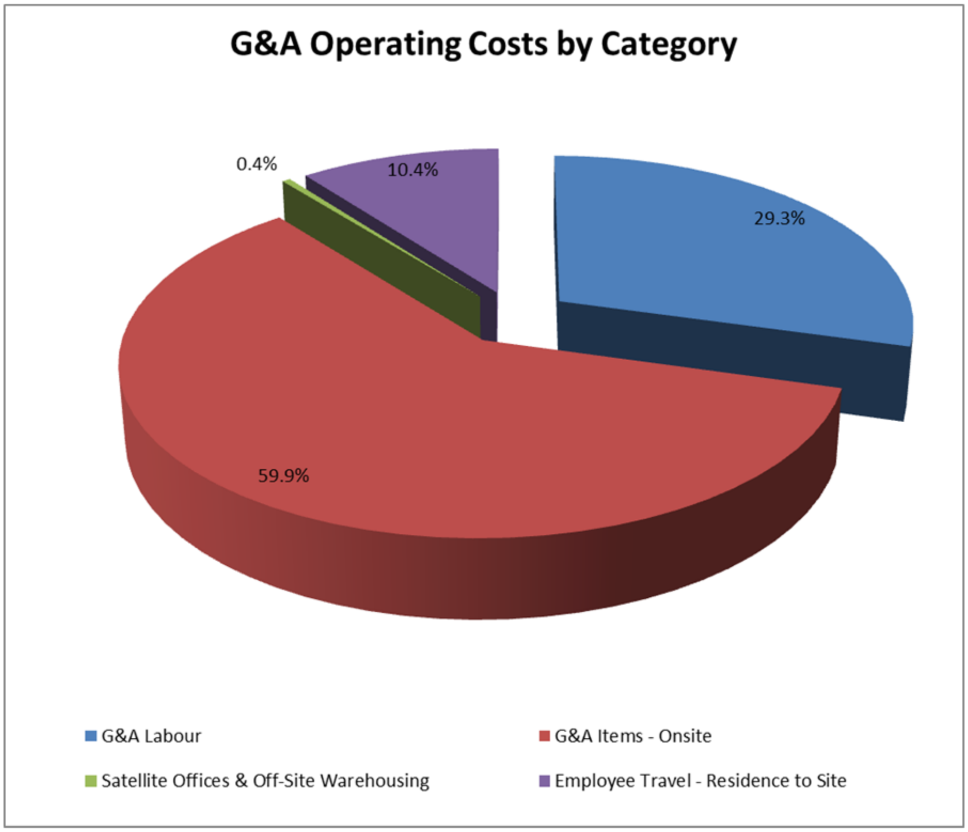
Figure 22.5 illustrates the breakdown of the G&A operating costs.

Table 22.31: Summary of G&A Costs

Cost Category	\$/t Processed
Labour	0.85
Items – on site	1.73
Satellite offices & off-site warehousing	0.01
Employee travel	0.30
Total G&A Costs	2.89

Source: JDS 2016

Figure 22.5: Distribution of G&A Costs by Category



Source: JDS 2016

Table 22.32: General and Administrative Detailed Costs

Area	LOM $M	$/t processed
G&A Labour		
General Management	4.2	0.09
Human Resources	3.0	0.06
IT/OT Support	1.6	0.03
Administration	10.5	0.23
Health & Safety	3.0	0.06
Environmental	5.3	0.11
Security	4.5	0.10
Camp Support	7.3	0.16
Subtotal G&A Labour	**39.3**	**0.85**
G&A On-Site Items		
Camp Catering & Cleaning	55.6	1.20
Health & Safety, Medical, & First Aid	2.2	0.05
Environmental	7.0	0.15
Human Resources	0.8	0.02
Insurance & Legal	8.1	0.17
External Consulting	1.7	0.04
IT & Communications	3.6	0.08
Office & Miscellaneous Costs	1.4	0.03
Subtotal G&A On-Site Items	**80.3**	**1.73**
Satellite Office & Off-Site Warehousing	**0.6**	**0.01**
Employee Travel	**13.9**	**0.30**
Total G&A Costs	**134.1**	**2.89**

Source: JDS 2016

22.7.1 Labour

General and administrative labour includes all on-site and off-site positions. Costs were estimated from first principles using fully burdened labour rates that were benchmarked against other similar operations.

Table 22.33 summarizes the G&A workforce labour rates and quantities.

Table 22.33: G&A Labour Complement and Rates

Position	Manpower Complement	Manpower On Site	Shift Rotation	Salaried/ Hourly	Loaded Annual Salary $
General Management					
General Manager	1	1	4&3	Salaried	353,759
Secretary	1	1	4&3	Salaried	77,462
Human Resources					
HR Superintendent	1	1	4&3	Salaried	125,859
Human Resources Coordinator	1	1	4&3	Salaried	99,359
Community Relations Coordinator	1	0	5&2	Salaried	99,359
IT/OT Support					
Technicians	2	1	2&2	Salaried	88,413
Administration					
Controller/Accountant	1	0	5&2	Salaried	125,859
Payroll Supervisor	1	0	5&2	Salaried	99,359
Payroll Clerk	2	0	5&2	Salaried	77,462
Sr. Purchasing/Contracts	1	1	4&3	Salaried	104,659
Purchasing Agent	2	2	4&3	Salaried	88,413
Warehouse Supervisor	1	1	4&3	Salaried	104,659
Warehouse Clerk	4	2	2&2	Hourly	90,963
Health and Safety					
Safety & Training Superintendent	1	1	4&3	Salaried	131,159
Health and Safety Officer	2	1	2&2	Salaried	99,359
Trainer	2	1	2&2	Salaried	99,359
Environmental					
Environmental Superintendent	1	1	4&3	Salaried	147,059
Road Monitor	1	0	4&3	Salaried	77,462
Environmental Technician	2	1	2&2	Salaried	88,413
Environmental Monitors	2	1	2&2	Salaried	77,462
Security					
Protective Services Supervisor	1	1	4&3	Salaried	131,159
Protective Services Officers	4	2	2&2	Salaried	88,413
Camp Support Services					
Camp Manager	2	1	2&2	Salaried	99,359
Camp Clerk	2	1	2&2	Hourly	67,119
Catering	22	11	2&2	Hourly	Incl. in Camp
Housekeepers	20	10	2&2	Hourly	Incl. in Camp
Janitors	4	2	2&2	Hourly	67,119
Nurse/Paramedic	2	1	2&2	Salaried	88,413
Total	**87**	**46**			

(*) Total may not match due to timing of maximum quantity by year
Source: JDS 2016

22.7.2 G&A On-Site Items

General and administrative on-site items include the following:

- Camp catering and cleaning based on vendor quoted unit costs and estimated camp occupancy levels;
- Health and safety, medical and first air based on annual allowances for health and safety supplies, training and medivac support;
- Environmental costs based on annual allowances for monitoring programs, helicopter support and training;
- Human resources costs associated recruiting, medicals and community relations;
- Insurance and legal costs based on insurance quotes and estimates for annual legal services;
- External consulting costs based on annual estimates;
- IT and communications based on a vendor provided quote for services; and
- Office and miscellaneous costs based on annual estimates for office supplies and bank charges.

The estimated LOM G&A on-site item cost is $1.82/t of ore processed or $8.9M per year. The cost estimates are shown in **Error! Not a valid bookmark self-reference.**.

Table 22.34: G&A On-site Item Costs

G&A On-site Items	LOM $M	$M/a	$/t Processed
Camp Catering & Cleaning	55.6	6.2	1.20
Health & Safety, Medical, & First Aid	2.2	0.2	0.05
Environmental	7.0	0.7	0.15
Human Resources	0.8	0.1	0.02
Insurance & Legal	8.1	0.9	0.17
External Consulting	1.7	0.2	0.04
IT & Communications	3.6	0.4	0.08
Office & Miscellaneous Costs	1.4	0.2	0.03
Total	**80.3**	**8.9**	**1.82**

Source: JDS 2016

22.7.3 Satellite Office and Off-site Warehousing

Satellite office and off-site warehousing includes monthly rental of office space in Whitehorse for off-site personnel. The estimated LOM satellite office and off-site warehousing cost is $0.01/t of ore processed or $60K per year.

22.7.4 Employee Travel

All employees will be transported to and from site by charter aircraft from Whitehorse, YT. Charter flight costs were based on a contractor quotation of $9,000 per return trip. The required charter flight schedule was calculated based on shift rotations. A 90% capacity factor was included to allow for cancellations and visitors. The estimated LOM employee travel cost is $0.32/t of ore processed or $1.5M per year.

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 2015 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: Life of Mine Plan Summary

Parameter	Unit	Value
Mine Life	Years	10
Total Ore	M t	46.4*
Strip Ratio	w:o	5.7
Processing Rate	Kt/d	14
Average Au Head Grade	g/t	1.45
Au Payable	LOM k oz	1,858
	Average koz/a	202

(*) includes all ore processed: 44.1 Mt of ore are processed during the production period.

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);
- Closure cost of $60.5M (including 12% contingency);
- Nominal 2015 dollars;
- No inflation;
- No taxes or duties;
- Numbers are presented on a 100% ownership basis and do not include management fees or financing costs; 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 in the tax calculations.

23.2 Timing of Revenues and Working Capital

23.2.1 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 operating costs were scheduled to occur in the economic analysis:

Mining Operating Costs

- 100% of consumables, material and fuel required for mine operations are assumed to be purchased one calendar quarter (three months) prior to the actual consumption. This models the incurrence of the costs for the consumables, materials, and fuel before the actual use;
- Labour costs are assumed to be incurred as they are paid; and
- A total of $13.4 M is assumed to occur in the pre-production period of cash flows (Year -1) to account for working capital.

Processing Operating Costs

- 100% of consumables and fuel are assumed to be incurred one calendar quarter (three months) prior to the actual use based on the proposed processing schedule;
- Labour costs are assumed to be incurred as they are paid; and
- A total of $8.6M of processing operating costs is calculated to occur in the pre-production period of cash flows.

Surface & Infrastructure Operating Costs

100% of consumables and fuel are assumed to be incurred one calendar quarter (three months) prior to the actual occurrence/requirement based on the proposed processing schedule;

- Labour costs are assumed to be incurred as they are paid; and
- A total of $1.0M of site surface operating costs is calculated to occur in the pre-production period (Year -1).

A total of $23.0M has been considered in the pre-production period as working capital.

23.2.2 Revenues & NSR Parameters

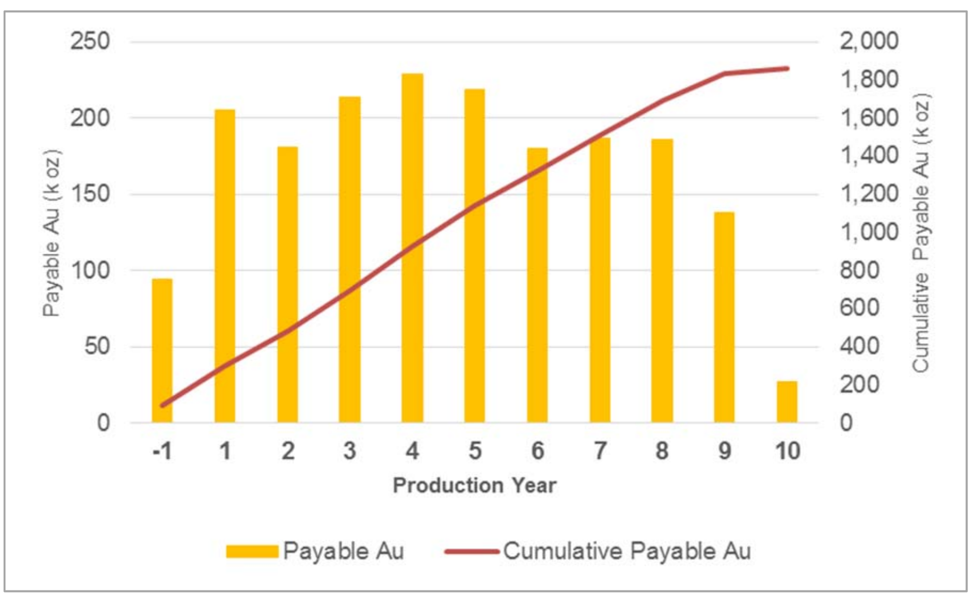
Mine revenue is derived from the sale of gold doré into the international marketplace. No contractual arrangements for refining exist at this time. However, the parameters used in the economic analysis were confirmed by a leading industry entity. These details can be found in Section 19 (Market Studies) of this report. Gold production and sale is assumed to begin in Q4 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	%	99.8
Au Refining Charge	US$/oz	1

Source: JDS 2016

Figure 23.1: Annual and Cumulative Payable Gold Production



Source: JDS 2016

23.3 Summary of Capital Costs

The capital costs were used for the economic analysis are set out below. The pre-production period spans 33 months including EPCM and access road construction. On-site construction, included in this period, has a duration of 15 months (commencing in Q2 Year -2). Table 23.3 summarizes the capital costs used in the economic analysis. Detailed information can be found in Section 21 of this report.

Table 23.3: Summary of Capital Costs

Capital Cost	Pre-Production $M	Production/Sustaining $M	LOM $M
Mining	85.4	47.5	132.9
On-Site Development	7.7	0.9	8.7
Ore Crushing and Handling	16.4	0	16.4
Heap Leach	28.2	34.5	62.7
Process Plant	27.6	1	28.6
On-Site Infrastructure	43.1	2.8	46
Off-Site Infrastructure	24.3	0	24.3
Indirects	31.7	4.7	36.4
EPCM	18.9	1.5	20.4
Owner Costs	7.9	0	7.9
Reclamation/Closure	0	60.5	60.5
Subtotal	**291.4**	**153.4**	**444.8**
Contingency	26.1	7.2	33.3
Total Capital	**317.4**	**160.6**	**478.1**

Source: JDS 2016

23.4 Summary of Operating Costs

Total LOM operating costs amount to $1,117 M. This translates into an average cost of $24.10/t ore processed over the life of mine. These costs are shown in **Error! Not a valid bookmark self-reference.**. A detailed analysis of the operating costs can be found in Section 22 of this report.

Table 23.4: Summary of Operating Costs

Operating Cost†	$/t processed	LOM $M
Mining*	15.26	707.4
Processing	4.97	230.3
Surface & Infrastructure	0.97	45.2
G&A	2.89	134.2
Total Operating Costs	**24.10**	**1,117.1**

(†): Operating costs include the working capital during the pre-production period
(*): Average LOM open pit mining cost amounts to $2.35/t mined at a 5.7:1 strip ratio
Source: JDS 2016

23.5 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 Ernst & Young Vancouver in order to derive the after-tax valuation of the Project.

A detailed tax analysis was completed specifically for the purpose of evaluating the Coffee Gold 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 referenced to the value of the output from the mine on an escalating 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.
- The royalty rate ranges from 0% to 12% based on the taxable revenue from saleable gold less 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.

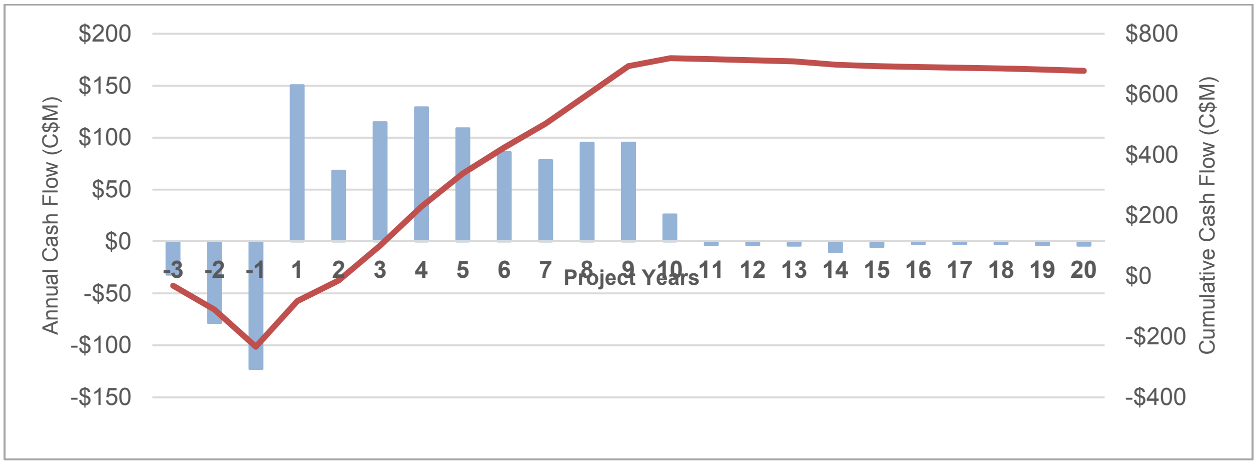
23.6 Third Party Royalties

Third party royalties have been considered in the economic analysis. A total of $29.4 M of third party royalties are payable over the life of mine. Details related to the third party royalties are outlined in Section 19 of this report.

23.7 Economic Analysis

The Project is economically viable with an after-tax internal rate of return (IRR) of 36.8% and a net present value at 5% ($NPV_{5\%)}$ of $455.3M. Figure 23.2 shows the projected cash flows used in the economic analysis. Table 23.5 shows the detailed results of this evaluation.

Figure 23.2: Annual and Cumulative After-Tax Cash Flows



Source: JDS 2015

Table 23.5: Summary of Economic Results

Category	Unit	Value
Net Revenues	$M	2,708
Operating Costs	$M	1,094
Cash Flow from Operations	$M	1,614
Capital Costs*	$M	478
Cash Cost‡	US$/oz	482
Cash Cost (Incl. Sustaining Capital)º	US$/oz	550
Net Pre-Tax Cash Flow	$M	1,113
Pre-Tax NPV5%	$M	**762.3**
Pre-Tax IRR	%	**50.4**
Pre-Tax Payback (from start of commercial gold production)	Years	**1.5**
Break-Even Pre-Tax Gold Price	US$/oz	**707**
Total Taxes	$M	430
Net After-Tax NPV5%	$M	**455.3**
After-Tax IRR	%	**36.8**
After-Tax Payback (from start of commercial gold production)	Years	**2.0**
Break-Even After-Tax Gold Price	US$/oz	**732**

(*): Includes pre-production, sustaining, closure and reclamation capital costs
(‡): (Refining Costs + Third Party Royalties + Operating Costs)/Payable Au oz
(º): (Refining Costs + Third Party Royalties + Operating Costs + Sustaining Capital Costs) /Payable Au oz
Source: JDS 2016

23.8 Sensitivity

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 23.6 and Table 23.7 and illustrated in Figure 23.3 and Figure 23.4.The Project proved to be most sensitive to changes in foreign exchange rate followed by metal price, head grade and capital costs. The Project showed least sensitivity to operating and capital costs.

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 23.6.

Table 23.6: Pre-Tax $NPV_{5\%}$ Sensitivity Results

	Pre-Tax $NPV_{5\%}$ ($M)						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	465	564	663	762	861	960	1,059
F/X Rate	1,111	982	866	762	668	582	504
Head Grade	466	564	663	762	861	960	1,059
OPEX	881	842	802	762	723	683	644
CAPEX	823	802	782	762	742	722	702

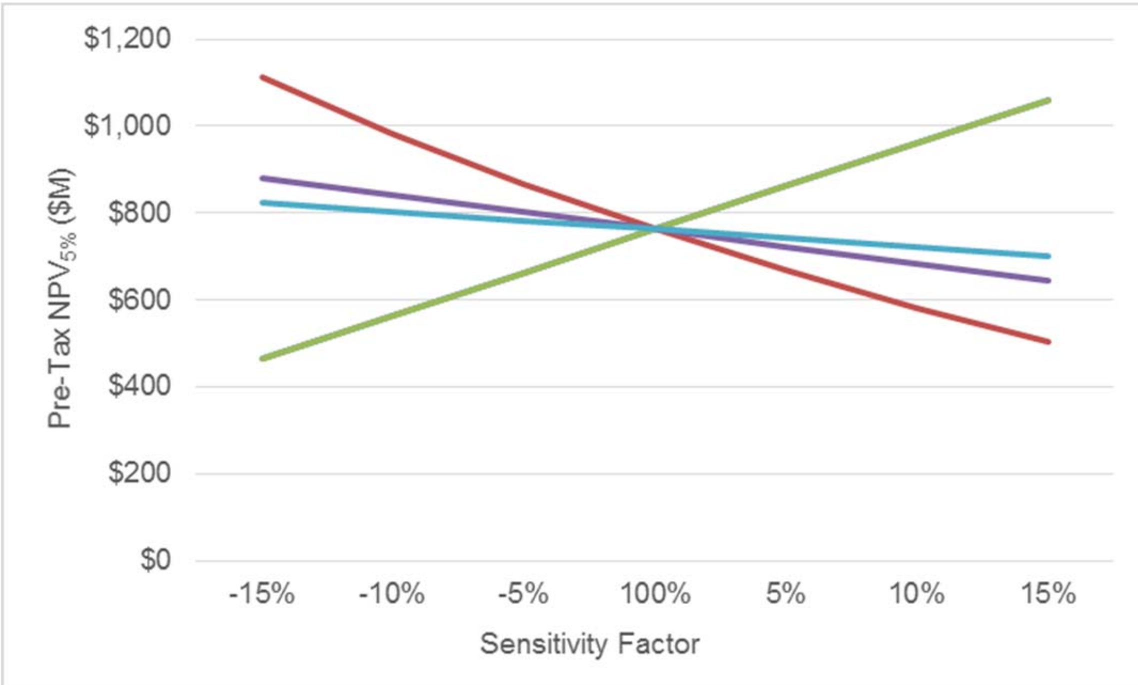
Source: JDS 2016

Table 23.7: After-Tax $NPV_{5\%}$ Sensitivity Results

	After-Tax $NPV_{5\%}$ ($M)						
	-15%	-10%	-5%	100%	5%	10%	15%
Metal Price	271	332	394	455	517	578	640
F/X Rate	672	592	520	455	397	344	295
Head Grade	271	333	394	455	517	578	639
OPEX	574	535	495	455	416	376	337
CAPEX	515	495	475	455	435	415	395

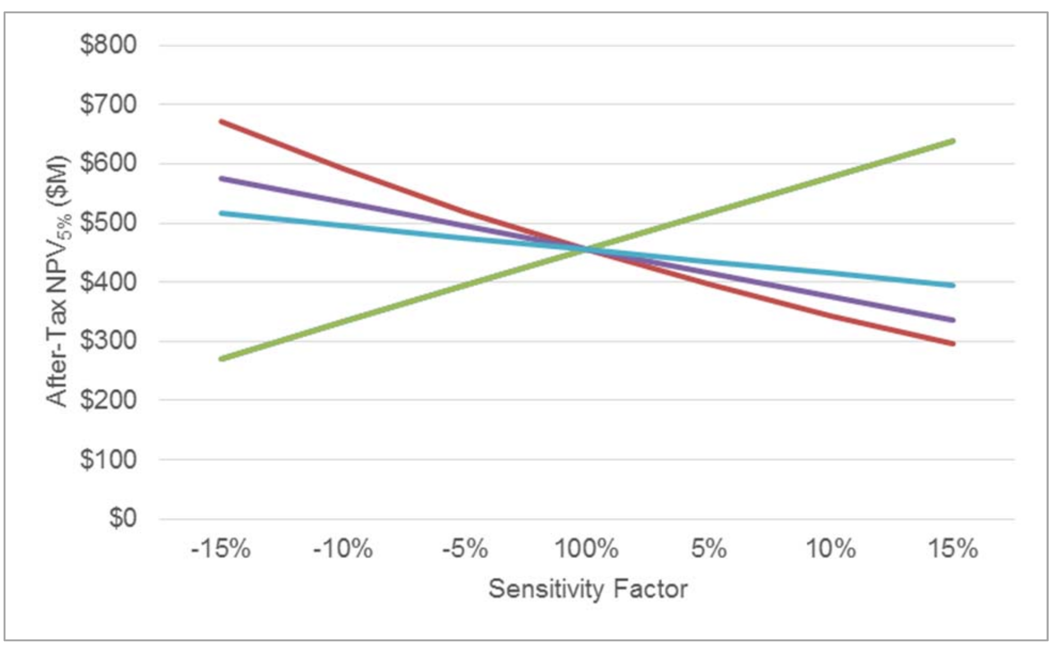
Source: JDS 2016

Figure 23.3: Pre-Tax NPV$_{5\%}$ Sensitivity



Source: JDS 2016

Figure 23.4: After-Tax NPV$_{5\%}$ Sensitivity



Source: JDS 2016

Table 23.8: Discount Rate Sensitivity Test Results on NPV

Discount Rate	Pre-Tax NPV ($M)	After-Tax NPV ($M)
0%	1,113	682
5%	762	455
7%	657	387
8%	611	356
10%	528	302

Source: JDS 2016

Kaminak - Coffee Creek FS Economic Model		-3 0	-2 1	-1 2	1 3	2 4	3 5	4 6	5 7	6 8	7 9	8 10	9 11	10 12	11 13	12 14	13 15	14 16	15 17	16 18	17 19	18 20	19 21	20 22	LOM Discount Period
Metal Price																									
Au	US $/oz	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	1,150	**1,150**
Exchange Rate	CAD:USD	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	**0.78**
MINE SCHEDULE																									
Supremo																									
Oxide Ore	M tonnes	-	-	-	-	2.3	3.5	3.7	3.8	4.6	4.9	5.8	1.2	-	-	-	-	-	-	-	-	-	-	-	**29.7**
Oxide Grade	g/t	-	-	-	-	1.57	1.59	1.73	1.52	1.24	1.28	1.30	1.38	-	-	-	-	-	-	-	-	-	-	-	**1.43**
Upper Transition Ore	M tonnes	-	-	-	-	0.0	0.1	0.4	0.7	0.5	0.1	0.4	0.1	-	-	-	-	-	-	-	-	-	-	-	**2.4**
Upper Transition Grade	g/t	-	-	-	-	1.81	1.46	1.52	1.88	1.58	1.28	1.18	1.58	-	-	-	-	-	-	-	-	-	-	-	**1.56**
Middle Transition Ore	M tonnes	-	-	-	-	0.0	0.0	0.1	0.2	0.2	0.0	0.1	0.0	-	-	-	-	-	-	-	-	-	-	-	**0.5**
Middle Transition Grade	g/t	-	-	-	-	3.08	2.49	1.41	1.58	1.79	1.07	1.25	2.58	-	-	-	-	-	-	-	-	-	-	-	**1.68**
Lower Transition Ore	M tonnes	-	-	-	-	0.0	0.0	0.1	0.1	0.0	0.0	0.0	0.0	-	-	-	-	-	-	-	-	-	-	-	**0.2**
Lower Transition Grade	g/t	-	-	-	-	3.47	4.11	1.68	1.70	1.75	2.05	1.32	3.83	-	-	-	-	-	-	-	-	-	-	-	**2.09**
Total Ore	**M tonnes**	**-**	**-**	**-**	**-**	**2.4**	**3.7**	**4.4**	**4.6**	**5.2**	**5.1**	**6.2**	**1.3**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**32.9**
Waste	M tonnes	-	-	-	-	19	30	35	35	34	35	30	5	-	-	-	-	-	-	-	-	-	-	-	**223.0**
Strip Ratio	w:o	-	-	-	-	8.2	8.1	8.1	7.5	6.5	6.9	4.8	4.1	-	-	-	-	-	-	-	-	-	-		**6.8**
Total Supremo Mined	**M tonnes**		-	-	-	22	34	40	40	39	40	36	7	-	-	-	-	-	-	-	-	-	-		**256**
Latte																									
Oxide Ore	M tonnes	-	-	2.7	2.7	1.5	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**7.2**
Oxide Grade	g/t	-	-	1.26	1.38	1.15	0.95	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.27**
Upper Transition Ore	M tonnes	-	-	0.4	1.0	0.9	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**2.5**
Upper Transition Grade	g/t	-	-	1.22	1.25	1.29	1.05	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.25**
Middle Transition Ore	M tonnes	-	-	0.1	0.5	0.7	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.4**
Middle Transition Grade	g/t	-	-	1.32	1.51	1.47	1.08	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.43**
Lower Transition Ore	M tonnes	-	-	0.0	0.1	0.3	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.5**
Lower Transition Grade	g/t	-	-	1.67	2.05	1.64	1.54	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.71**
Total Ore	**M tonnes**	**-**	**-**	**3.2**	**4.2**	**3.5**	**0.6**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**11.5**
Waste	M tonnes	-	-	8.7	12.9	5.9	0.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**28.2**
Strip Ratio	w:o	-	-	2.7	3.0	1.7	1.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		**2.44**
Total Latte Mined	**M tonnes**	-	-	11.9	17.2	9.4	1.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		**39.8**
Double Double																									
Oxide Ore	M tonnes	-	-	0.2	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.4**
Oxide Grade	g/t	-	-	3.62	2.66	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**3.06**
Upper Transition Ore	M tonnes	-	-	0.1	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.4**
Upper Transition Grade	g/t	-	-	3.76	3.89	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**3.84**
Middle Transition Ore	M tonnes	-	-	0.1	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.2**
Middle Transition Grade	g/t	-	-	1.87	3.11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**2.70**
Lower Transition Ore	M tonnes	-	-	0.0	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.0**
Lower Transition Grade	g/t	-	-	2.04	2.64	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**2.43**
Total Ore	**M tonnes**	**-**	**-**	**0.4**	**0.6**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**1.1**
Waste	M tonnes	-	-	6.4	4.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**10.9**
Strip Ratio	w:o	-	-	15.6	6.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		**10.30**
Total Double Double Mined	**M tonnes**	-	-	6.8	5.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		**11.9**
Kona																									
Oxide Ore	M tonnes	-	-	-	0.0	0.6	0.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.7**
Oxide Grade	g/t	-	-	-	0.63	1.13	1.84	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.20**
Upper Transition Ore	M tonnes	-	-	-	0.0	0.1	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.1**
Upper Transition Grade	g/t	-	-	-	0.63	1.05	1.86	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**1.22**
Middle Transition Ore	M tonnes	-	-	-	0.0	0.0	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.0**
Middle Transition Grade	g/t	-	-	-	0.57	0.85	0.66	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.84**
Lower Transition Ore	M tonnes	-	-	-	-	0.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**0.0**
Lower Transition Grade	g/t	-	-	-	-	2.22	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**2.22**
Total Ore	**M tonnes**	**-**	**-**	**-**	**0.0**	**0.7**	**0.1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**0.9**
Waste	M tonnes	-	-	-	0.2	2.8	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**3.2**
Strip Ratio	w:o	-	-	-	7.9	3.9	1.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		**3.73**
Total Kona Mined	**M tonnes**	-	-	-	0.2	3.6	0.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		**4.1**
TOTAL - All Zones																									
Oxide Ore	M tonnes	-	-	3	3	4	4	4	4	5	5	6	1	-	-	-	-	-	-	-	-	-	-		**38**
Oxide Grade	g/t	-	-	1.41	1.49	1.36	1.55	1.73	1.52	1.24	1.28	1.30	1.38	-	-	-	-	-	-	-	-	-	-		**1.41**
Upper Transition Ore	M tonnes	-	-	1	1	1	0	0	1	0	0	0	0	-	-	-	-	-	-	-	-	-	-		**5**
Upper Transition Grade	g/t	-	-	1.86	1.74	1.30	1.32	1.52	1.88	1.58	1.28	1.18	1.58	-	-	-	-	-	-	-	-	-	-		**1.57**
Middle Transition Ore	M tonnes	-	-	0	1	1	0	0	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-		**2**
Middle Transition Grade	g/t	-	-	1.59	1.84	1.49	1.25	1.41	1.58	1.79	1.07	1.25	2.58	-	-	-	-	-	-	-	-	-	-		**1.60**
Lower Transition Ore	M tonnes	-	-	0	0	0	0	0	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-		**1**
Lower Transition Grade	g/t	-	-	1.91	2.20	1.70	2.49	1.68	1.70	1.75	2.05	1.32	3.83	-	-	-	-	-	-	-	-	-	-		**1.88**
Total Ore	**M tonnes**	**-**	**-**	**4**	**5**	**7**	**4**	**4**	**5**	**5**	**5**	**6**	**1**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**		**46**
Waste	M tonnes		-	15	18	28	31	35	35	34	35	30	5	-	-	-	-	-	-	-	-	-	-		**265**
Strip Ratio	w:o	-	-	4.2	3.6	4.3	6.9	8.1	7.5	6.5	6.9	4.8	4.1	-	-	-	-	-	-	-	-	-	-		**5.72**
Throughput Rate	tpd			0	0	0	0	0	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-		**0**
Total Grade	g/t			1.50	1.61	1.38	1.54	1.70	1.57	1.29	1.28	1.30	1.42	-	-	-	-	-	-	-	-	-	-		**1.45**
Total Mined	**M tonnes**	-	-	19	23	35	35	40	40	39	40	36	7	-	-	-	-	-	-	-	-	-	-		**312**
Total Au to Heap	koz	-	-	169	258	223	244	266	252	206	209	209	121	-	-	-	-	-	-	-	-	-	-	-	**2,157**
Total Recoverable Au to Heap	koz	-	-	146	212	181	210	229	219	181	187	186	111	-	-	-	-	-	-	-	-	-	-	-	**1,862**
Au Recovery	%	0%	0.0%	86.5%	82.0%	81.3%	86.1%	86.2%	86.8%	87.7%	89.6%	89.0%	91.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%		**86.3%**
Au Recovered	koz	-	-	95	206	181	214	229	219	181	187	186	138	27	-	-	-	-	-	-	-	-	-	-	**1,862**
PAYABLE METALS																									
Total Recovered Au	koz	-	-	95	206	181	214	229	219	181	187	186	138	27	-	-	-	-	-	-	-	-	-		**1,862**
Payable Au	%		99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%		**99.8%**
	oz		-	95	205	181	213	228	219	180	186	186	138	27	-	-	-	-	-	-	-	-	-		**1,858**
	US$M	-	-	109	236	208	245	263	251	207	214	214	158	31	-	-	-	-	-	-	-	-	-	-	**2,137**
	C $M	-	-	140	303	267	314	337	322	266	275	274	203	39	-	-	-	-	-	-	-	-	-	-	**2,739**
Refining Costs	US$/payable oz		1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00		**1.00**
	C $M	-	-	0.1	0.3	0.2	0.3	0.3	0.3	0.2	0.2	0.2	0.2	0.0	-	-	-	-	-	-	-	-	-	-	**2.4**
Royalties	C $M		-	1	3	3	3	3	3	3	3	3	2	0	-	-	-	-	-	-	-	-	-		**27**
	C $M	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**2**
Net Smelter Return (NSR) after Royalties	**C $M**	**-**	**-**	**136**	**300**	**264**	**311**	**333**	**319**	**263**	**272**	**271**	**201**	**39**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**		**2,708**
OPEX																									
Mining	C $/tonne mined	-	-	0.44	2.61	2.41	2.37	2.18	2.24	2.30	2.35	2.23	3.28	-	-	-	-	-	-	-	-	-	-		**2.27**
	C $M	-	-	8	59	83	84	86	89	90	93	80	22	-	-	-	-	-	-	-	-	-	-		**694**
Processing	C $/tonne ore	-	-	0.70	5.30	3.98	6.08	5.98	5.26	4.63	4.89	3.92	10.97	-	-	-	-	-	-	-	-	-	-		**4.97**
	C $M	-	-	2	26	26	27	26	24	24	25	24	14	2	-	-	-	-	-	-	-	-	-	-	**222**
Surface and Infrastructure	C $/tonne ore	-	-	0.24	0.98	0.73	1.08	1.10	1.04	0.92	0.96	0.75	3.06	-	-	-	-	-	-	-	-	-	-		**0.97**
	C $M	-	-	1	5	5	5	5	5	5	5	5	4	1	-	-	-	-	-	-	-	-	-	-	**44**
G&A	C $/tonne ore	-	-	0.90	2.85	2.23	3.31	3.39	3.19	2.85	2.95	2.36	8.98	-	-	-	-	-	-	-	-	-	-		**2.89**
	C $M	-	-	3	14	15	15	15	15	15	15	15	12	2	-	-	-	-	-	-	-	-	-	-	**134**
Total Opex	**C $M**	**-**	**-**	**15**	**104**	**129**	**130**	**132**	**133**	**134**	**138**	**124**	**52**	**4**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**		**1,094**
	C $/payable oz	**-**	**-**	**155**	**505**	**712**	**611**	**576**	**609**	**743**	**739**	**666**	**377**	**165**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**589**
	C$/t ore	**-**	**-**	**4**	**21**	**20**	**29**	**30**	**29**	**26**	**27**	**20**	**40**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**		**24.10**
Net Operating Cashflow	**C $M**	**-**	**-**	**121**	**196**	**135**	**180**	**202**	**186**	**129**	**134**	**147**	**149**	**34**	**-**	**-**									**1,614**
CAPEX																									
Mining	C $M	-	0	85	24	0	10	0	6	5	2	0	-	-	-	-	-	-	-	-	-	-	-	-	**133**
On-Site Development	C $M	-	8	-	0	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**9**
Ore Crushing and Handling	C $M	-	6	11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**16**
Heap Leach	C $M	-	14	15	7	-	8	2	7	-	10	-	-	-	-	-	-	-	-	-	-	-	-	-	**63**
Process Plant	C $M	-	-	28	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**29**
On-Site Infrastructure	C $M	-	22	21	3	0	-	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**46**
Off-Site Infrastructure	C $M	24	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**24**
Indirects	C $M	-	12	20	1	0	1	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	**36**
EPCM	C $M	5	6	8	1	-	0	-	0	-	0	-	-	-	-	-	-	-	-	-	-	-	-	-	**20**
Owner Costs	C $M	-	4	4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**8**
Closure	C $M	-	-	-	-	0	0	1	1	1	1	3	1	8	5	5	4	10	5	3	2	2	4	4	**60**
Subtotal	**C $M**	**29**	**72**	**190**	**36**	**1**	**20**	**4**	**16**	**6**	**15**	**3**	**1**	**8**	**5**	**5**	**4**	**10**	**5**	**3**	**2**	**2**	**4**	**4**	**445**
Contingency	C $M	3	6	17	2	0	2	0	1	0	2	0	-	-	-	-	-	-	-	-	-	-	-	-	**33**
CAPEX incl. Contingency	**C $M**	**32**	**78**	**207**	**38**	**1**	**22**	**4**	**17**	**6**	**17**	**3**	**1**	**8**	**5**	**5**	**4**	**10**	**5**	**3**	**2**	**2**	**4**	**4**	**478**
Pre-Production	*C $M*	*32*	*78*	*207*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**317**
Sustaining & Closure	*C $M*	-	-	-	*38*	*1*	*22*	*4*	*17*	*6*	*17*	*3*	*1*	*8*	*5*	*5*	*4*	*10*	*5*	*3*	*2*	*2*	*4*	*4*	**161**
Working Capital	C $M	-	-	*23*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	***23***
Net Pre-Tax Cashflow	**C $M**	**(32)**	**(78)**	**(109)**	**158**	**134**	**159**	**197**	**169**	**123**	**117**	**144**	**147**	**27**	**(5)**	**(5)**	**(4)**	**(10)**	**(5)**	**(3)**	**(2)**	**(2)**	**(4)**	**(4)**	**1,113**
Cumulative Net Cashflow	C $M	(32)	(110)	(219)	(61)	73	232	429	598	721	839	983	1,130	1,157	1,152	1,147	1,143	1,133	1,128	1,125	1,123	1,120	1,117	1,113	
Pre-Tax Payback Period	**Years**																								**3.5**
Pre-Tax IRR	**%**																								**50.4%**
Pre-Tax NPV$_{5\%}$	**C $M**	$ (32)	$ (75)	$ (99)	$ 137	$ 110	$ 124	$ 147	$ 120	$ 83	$ 76	$ 89	$ 86	$ 15	$ (3)	$ (2)	$ (2)	$ (5)	$ (2)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	**762**
Income Taxes & Yukon Quartz Tax	**C $M**	**0**	**0**	**13**	**22**	**36**	**55**	**64**	**59**	**38**	**42**	**49**	**52**	**1**	**(2)**	**(1)**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**430**
Net After Tax Cash Flow	**C $M**	**-32**	**-78**	**-122**	**136**	**98**	**104**	**133**	**109**	**84**	**75**	**95**	**95**	**26**	**-3**	**-3**	**-4**	**-10**	**-5**	**-3**	**-2**	**-2**	**-4**	**-4**	**682**
Cumulative After-Tax Cash Flow	C $M	-32	-110	-232	-96	2	106	239	348	432	508	603	698	724	721	717	713	703	697	695	692	690	686	682	
After-Tax Payback Period	**Years**																								**4.0**
After-Tax IRR	**%**																								**36.8%**
After-Tax NPV$_{5\%}$	**C $M**	$ (32)	$ (75)	$ (110)	$ 117	$ 81	$ 81	$ 99	$ 78	$ 57	$ 49	$ 58	$ 56	$ 15	$ (2)	$ (2)	$ (2)	$ (5)	$ (2)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	**455**

24 Adjacent Properties

There are no adjacent properties considered relevant to this technical report.

25 Other Relevant Data and Information

25.1 Project Execution Plan

25.1.1 Introduction and Philosophy

The project execution plan (PEP) for the Coffee Gold Project Feasibility Study is based on principles tested and proven in the development of remote, logistically-challenged projects in northern Canada. These principles include:

- Safety in design, construction and operations is paramount to success;
- Simple, passive environmental solutions, minimizing disturbance footprint;
- Fit-for-purpose design, construction, and operation;
- Due to the high cost of transportation, consolidate construction and operational needs to the extent practical;
- Common equipment fleet purchased by owner at the outset and used for construction needs;
- Efficient operations; minimize site labour requirements;
- Negotiated contracts with suppliers, contractors, and engineers with proven track records in northern Canadian mine developments;
- Early completion of project components turned over to operations;
- Elimination of superfluous management organizations; and
- Same camp accommodation status applied to all site personnel (no management quarters).

25.1.2 Project Execution Plan Summary

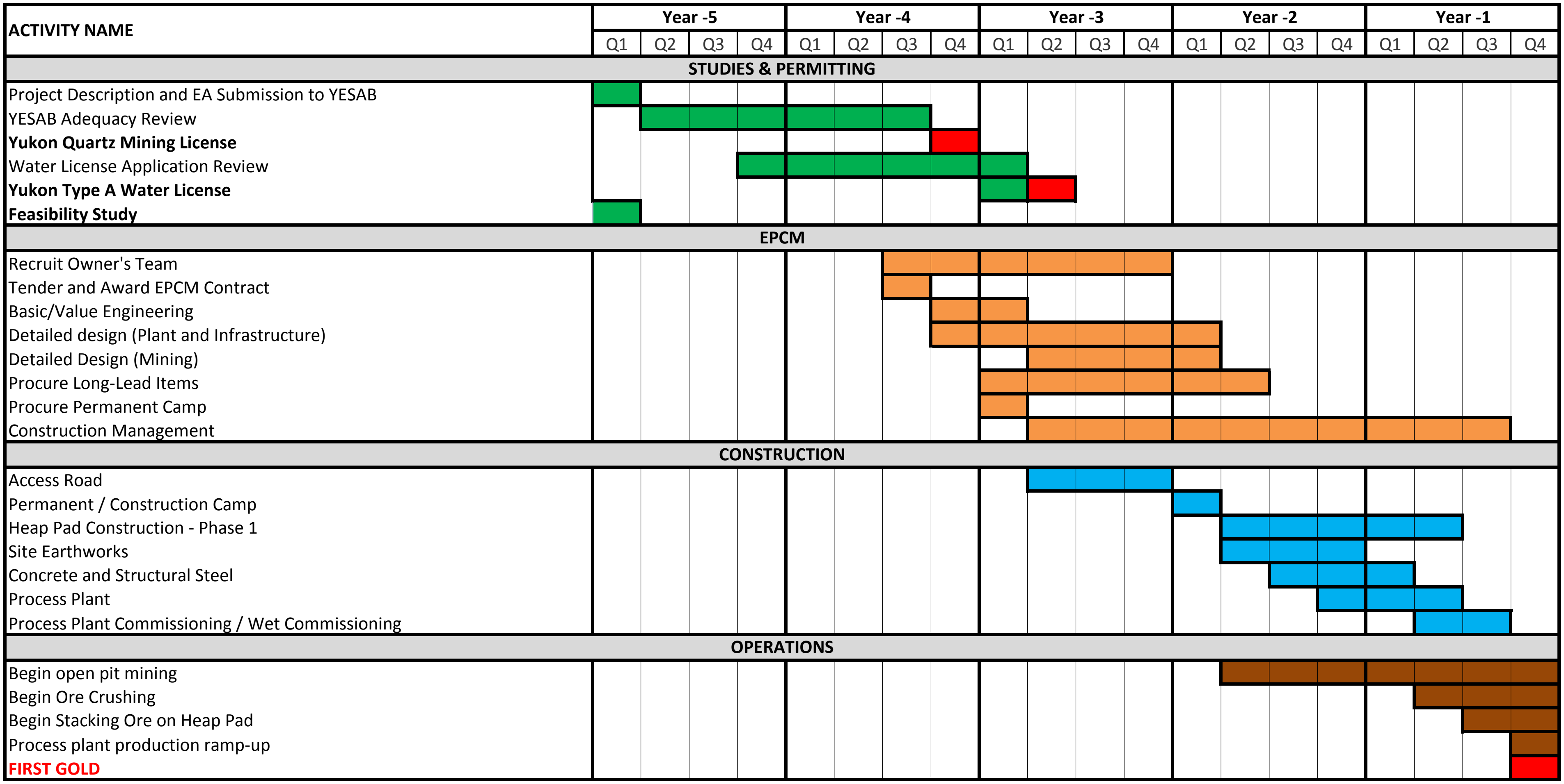
The PEP utilizes an all-weather access road from Dawson to the Project site as the primary delivery method for equipment and materials that are required for the construction of the Project. The access road is planned to be built prior to start of on-site construction activities in Q2 Year -2.

The majority of construction freight will be delivered on the all-weather access road. Passenger transportation and emergency freight delivers will be by air.

25.1.3 Project Construction Schedule

The Project construction schedule is driven by the completion of the access road by the end of Q4 Year -3 (construction commences in Q2 Year -3). A detailed schedule has been developed for the Project construction activities, utilizing the Feasibility Study cost estimate as the basis for determining the required man-hours. This scheduling exercise indicates that mechanical completion and wet commissioning can be accomplished within an 18-month construction period. The key schedule milestones are presented in Figure 25.1.

Coffee Gold Project - Project Execution Plan



25.1.4 Temporary Facilities for Construction

The Coffee Gold Project site currently has the following facilities:

- Exploration camp with approximately 80-person capacity and ancillary structures;
- 880 m gravel airstrip;
- Fuel storage for approximately 225,000 L of diesel fuel;
- Diesel generators (2 X~75 kW);
- Workshop and warehouse facilities; and
- Various pieces of heavy equipment to support exploration activities (specifically a D6, a 325C, and a 312C with a fuel truck and a light duty FEL).

In order to carry out construction at the Coffee site in preparation for operations, a significant increase in site facilities will be required. This includes:

- Expansion of site camp capacity to 296 people during construction and approximately 225 people during operations. The existing exploration camp will be used for initial earthwork activities and during the construction of the main camp facilities. The exploration camp will be utilized during construction as an overflow facility. Construction of the main camp will be the initial priority in Year -2.
- Construction of a new gravel airstrip closer to the project site. A 1,220 m airstrip will be constructed to accommodate propeller aircraft such as the Hawker Siddeley 748. Prior to the completion of the airstrip the Project will be supported by the existing exploration airstrip.
- Construction of additional diesel storage at the plant site area. A total of two 4 ML field-erected tanks will be constructed to store diesel required for operations. Both tanks will be constructed in Year -2.
- Installation of four 2.25 MW diesel generators (ultimately) to produce electrical power for the plant and infrastructure facilities during operations. Due to the modular nature of the generators, one unit will be mobilized in Year -2 to support the expanded camp and to provide construction power. Two additional units will be added in Year -1 prior to crusher and process plant commissioning. The fourth unit will be added in Year 1 to achieve an “N+2” configuration.
- Construction of ancillary structures: potable water treatment plant, sewage treatment plant, and laydown areas.

25.1.5 Construction Materials

All construction materials and equipment will be transported by off-highway trucks on the all-weather access road from Dawson. The exception is rock-fill and concrete aggregates which will be produced on site from quarries or as waste rock from Latte pit.

Annual material and equipment quantities (excluding catering freight) required during the pre-production period are listed in Table 25.1.

Table 25.1: Annual Material and Equipment Quantities Required Prior To Production

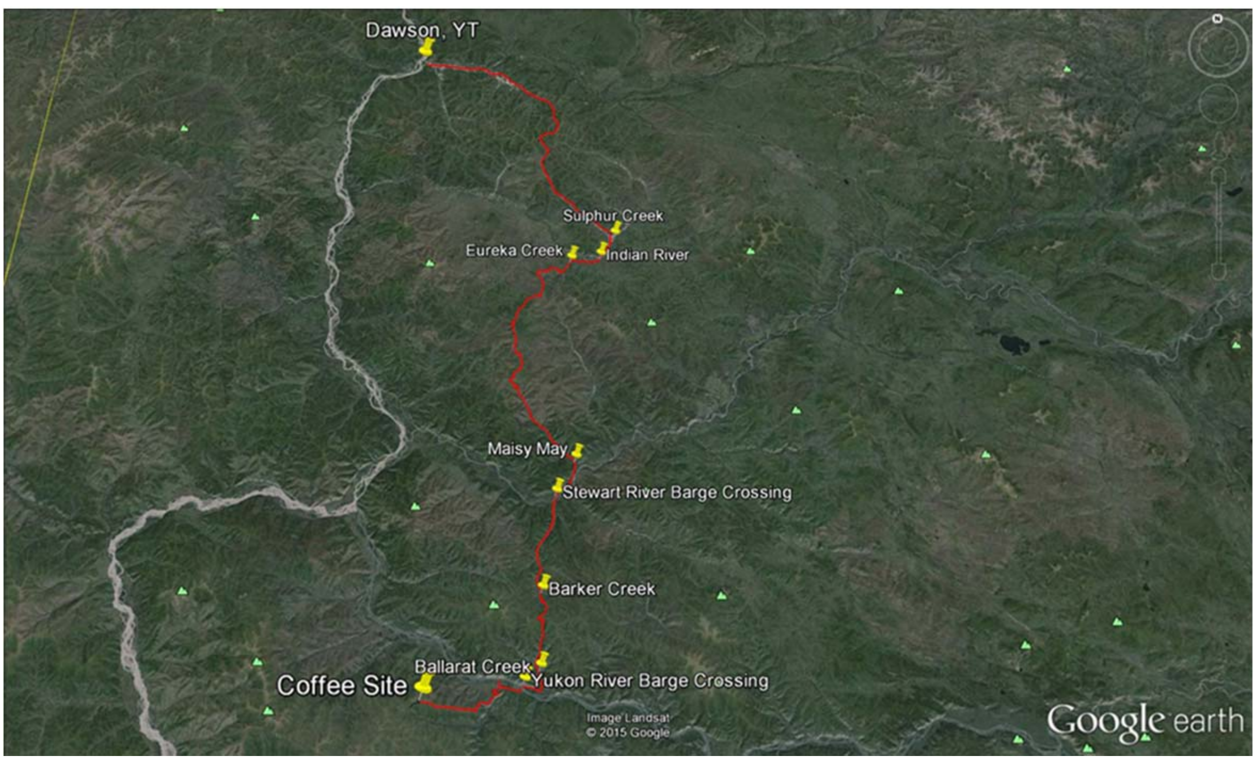
	Year -2 (tonnes)	Year -1 (tonnes)	Total (tonnes)
Mining and site support equipment	2,299	1,624	3,923
Construction equipment & material	4,724	5,965	10,689
Annual consumables	259	10,097	10,356
Total	**7,282**	**17,687**	**24,969**

Source: JDS 2016

25.1.6 Site Access and Mobilization

Site access will be provided by constructing an all-weather access road between Dawson and the Coffee site in Year -3, upon receipt of appropriate road construction approvals. The access road is design to be a forestry service-type road with a 5 m running surface and pull-outs to allow for opposing traffic to pass safety. Figure 25.2 shows the intended route of the road from Dawson to the Coffee site.

Figure 25.2. All-Weather Access Road Alignment



Source: JDS 2016

Access road construction will commence in Q2 Year -3 with substantial completion prior to the start of site construction beginning in Year -2. In order to meet this timeline, road construction will progress using multiple headings. It is envisioned that four crew headings will be required:

- Dawson heading south;
- Stewart River heading north;
- Stewart River heading south; and
- Yukon River heading north.

Table 25.2 provides a summary of the construction duration for each heading.

Table 25.2. Access Road Headings and Durations

Crew Heading	Start Location	End Location	Distance (km)	Construction Man-Days				Construction Duration (Days)
				Road	Stream Crossing	Barge Landing	Total	
Dawson South	Dawson	Henderson	110	3,177	113	0	3,290	224
Stewart North	Stewart	Henderson	36	2,010	109	53	2,172	150
Stewart South	Stewart	Ballarat	25	1,335	152	53	1,540	110
Yukon North	Yukon	Ballarat	22	1,365	90	100	1,555	110
Total			**193**	**7,887**	**464**	**206**	**8,557**	

Source: JDS 2016

Mobilization and ongoing construction support for the heading starting in Dawson can begin prior to the other three headings as year-round access to Dawson is available. The other three headings will be mobilized and supported via barge campaigns on the Stewart and Yukon Rivers and therefore will need to wait until the beginning of May when the rivers are free of ice.

Once the access road is completed, it will be operated on a year-round basis with the exception of periods when the Stewart and Yukon Rivers are either freezing-up in the fall through to early winter or breaking-up in the spring. There are two distinct operating seasons for the road; one during periods of open water flow in the two major rivers and the other during the winter months when the rivers are frozen. During the open water period, barges will be utilized to ferry transport trucks delivering fuel and dry freight across the Stewart and Yukon Rivers. During the winter months, when the rivers are frozen, ice crossings will be constructed to allow transport trucks to drive across the rivers.

The access road is expected to be open for an average of 295 days per year, with barge service beginning each year in late May and being suspended at the beginning of November. A six week period is expected to be required for the river to freeze up. Thickening of the ice will be accelerated by pumping water onto the surface ice. This is anticipated to allow hauling on the ice crossings to commence in mid-December of each year. The ice crossings are expected to be in operation until late April when ice thickness will no longer be sufficient to support heavy equipment. A four week period during break-up is anticipated before barge services will be able to resume.

Road operational periods are based on historical averages from road operations in the Yukon as well as JDS experience in the Northwest Territories.

25.1.7 Fuel Requirements

Fuel requirements for Coffee during construction are listed in Table 25.3.

Table 25.3: Coffee Pre-Production Fuel Requirements

	Year -2 (000 litres)	Year -1 (000 litres)	Total (000 litres)
Process & Infrastructure	2,214	5,513	7,727
Open Pit Mining - Operations	0	7,550	7,550
Surface & Infrastructure	346	487	833
Construction Equipment	421	463	884
Earthworks	308	326	634
Total Fuel Requirements	**3,289**	**14,339**	**17,628**

Source: JDS 2016

Fuel required for Year -2 and Year -1 will be transported to the Coffee site using 46,500 L triaxle fuel tanker trucks.

25.1.8 Other Temporary Facilities

Table 25.4 provides a list of other temporary equipment and facilities that will be utilized during construction.

Table 25.4: Other Temporary Equipment and Facilities

Equipment Description	Quantity
5 t fork Lift zoom-boom - Terex GTH-5519	1
65 ft man-lift - Genie S-65	1
85 ft man-lift - Genie S-85	2
125 ft man-lift straight boom - Genie S-125	1
125 ft man-lift articulating boom - Genie ZX-135/70	1
Mobile crane - RT160 - Tadano	1
Mobile crane - RT100 - Tadano GR1000-XL2	2
Tool carrier - Cat 966K (with Attachments)	1
1 t diesel crew cab pick-up - Ford F350	2
Tractor with deck trailer	1
Portable diesel light plants	4
Portable diesel heaters	4
Scissor lift - Genie GS4069RT	1
Portable office/wash module	1

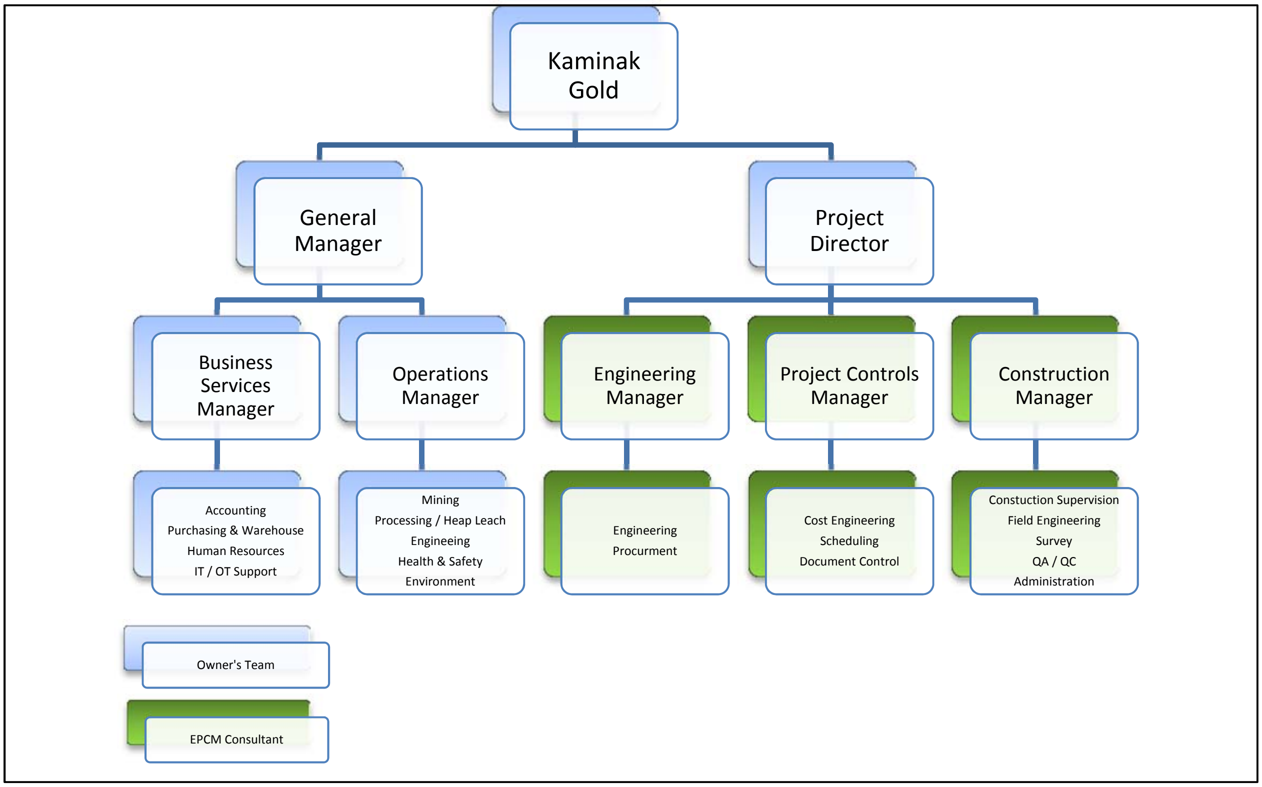
Source: JDS 2016

25.1.9 Engineering, Procurement and Construction Management

25.1.9.1 Project Development Organization

Project management will be by an integrated team comprising the owner's project management personnel and the project management consultant. The project management organizational structure is illustrated in Figure 25.3.

Figure 25.3: Project Management Organization Chart



Source: JDS 2016

The project management (PM) team will oversee the detailed engineering, procurement, and construction management activities for the Project. The PM team will also coordinate the work of the engineering consultants and other specialized consultants as required.

The PM team will be responsible for all project activities from detailed design through to commissioning and turnover to operations. The PM team will be available to backstop the operations teams with key supervision and management assistance when the operations personnel assume control of project components as they are completed.

25.1.9.2 Engineering

The engineering team will design each facility and component of the plant and infrastructure with estimated costs, procurement lead time and installation requirements. Each design package will then be turned over to the procurement team following a review by the project management team. The engineering team will also monitor construction and commissioning to ensure design compliance. This may include representation at vendor's factories, if required. Engineering tasks will include:

- Detailed design;
- Cost estimation;
- Drawings;
- Review of tenders for technical and design compliance;
- Field engineering; and
- QA/QC.

25.1.9.3 Procurement

In general, the project management team will oversee the selection and tendering of all equipment and bulk materials and commodities as a function of managing the engineering consultant. Tagged equipment is defined as uniquely designed and engineered equipment and assemblies required for the Project as documented in the project equipment lists. Bulk materials are not generally specifically engineered items and are not identified on the project equipment list. All bulk materials for the Project will be purchased, tracked and referenced to applicable specifications and standards.

The procurement team will utilize the design packages provided by engineering to obtain competitive tenders. The procurement team's tasks will include:

- Preparation of tender and contract documents;
- Pre-qualification of vendors;
- Review of tender submission for compliance and commercial terms;
- Contract award and execution;
- Logistics interface with owner's team;
- Payment of contractor and vendors;
- Contractor claims and dispute resolution; and
- Contract completion and close-out.

The following major procurement packages will be mobilized in Year -2 to support construction activities:

- Open pit mining equipment to site earthworks;
- Permanent camp (Including the sewage and water treatment plants, kitchen/diner, mine dry and offices);
- Power generation equipment package (1 unit);
- Temporary fuel storage tanks;
- Permanent diesel storage tank (two 4 ML tanks);
- Equipment and materials to support concrete installation;
- Equipment to support building construction;
- Process plant building; and
- Ancillary buildings (truck shop, waste management).

The following major procurement packages will be mobilized in Year -1 to support construction:

- Process plant equipment package;
- Crushing equipment package;
- Power generation equipment package (2 units); and
- Remainder of the open pit mining equipment to support pre-stripping.

25.1.9.4 Construction Management

The construction management team will be responsible for the following tasks:

- Project Controls;
- Schedule development, review and tracking;
- Warehousing;
- Contract administration;
- Purchasing;
- Cost tracking and forecasting;
- Scheduling and cost compliance;
- Constructability reviews of designs and contract documents;
- Coordination and management of site operations;
- Site health and safety;
- Camp management and maintenance;
- Site-wide labour relations;

- Site services;
- Document control;
- Reviewing contractor invoices for forwarding to procurement;
- Certification of contract completion for close-out;
- Input to contractor claims and dispute resolution;
- Survey control and as-builts;
- QA/QC; and
- Commissioning.

Table 25.5 lists the construction management team anticipated for the project. The table outlines the number of full time, on-site personnel required for the entire construction period.

Table 25.5: Construction Management Team Complement and Phasing

	Year -2			Year -1		
	Q2	Q3	Q4	Q1	Q2	Q3
Construction Manager	1	1	1	1	1	1
Construction Supervision - CSA	1	1	1	1		
Construction Supervision – Mech/Piping				1	1	
Construction Supervision - Electrical				1	1	
Construction Administrator/Document Controls	1	1	1	1	1	1
Commissioning						2
Engineering Manager	1	1	1	1	1	
Field Engineering/Project Coordinator	1	1	1	2	2	
QA/QC	1	1	1	1	1	
Project Controls Manager	1	1	1	1	1	
Cost Control/Estimating	1	1	1	1	1	
Scheduler	1	1	1	1	1	
Safety Officer (Construction)	1	1	1	2	2	1
Total Manpower	**11**	**11**	**11**	**15**	**14**	**5**

Source: JDS 2016

25.1.9.5 Commissioning

As each work package approaches completion, the contractors will work with the construction management team to attain substantial completion. The engineering and construction management teams will develop deficiency lists. Remedied deficiencies will be signed-off by the engineering and construction management teams to certify substantial completion.

As the various systems are certified, 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 construction management 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.

Senior operating management and operating staff will be recruited in Year -1 and will be responsible for commissioning the plant, with support from the existing engineering and construction management organizations, to enable production of the first gold late in Year -1. Construction staff will leave the site as each work package is completed. The procurement team will close out contracts when advised by the engineering and construction management teams. As commissioning is completed, Kaminak employees will take over the operation of the plant.

25.1.9.6 Construction Contract Strategy

The contracting strategy will be established by the project management team at the onset of the Project, which will address each contract battery limit, detailed scope of work and the cost structure of each. Contract work packages will be divided into manageable scopes, and awarded to contractors “best fit” for the work. Contractors will be pre-qualified by the procurement team based on their ability to execute the work in a safe and efficient manner, as demonstrated by past performance. Opportunities for qualified local and First Nations contractors will be given consideration when determining the work packages, providing that they can meet bid requirements and are available to provide value to the Project through competitive pricing.

An open shop labour strategy will be adopted for the Project, and the number of discipline contractors (such as concrete, structural steel and mechanical) will be minimized to mitigate the cost of separate administrations, duplication of temporary facilities and progress dictated by peer contractors. In the process plant and mechanical installations in particular, single general contractors will be selected and they will manage and coordinate the interfacing of the various trade disciplines, subcontractors and vendor representatives.

Contracts that extend into operations, such as camp catering, will be structured in conjunction with the owner’s operations personnel to ensure that operational needs are properly addressed.

25.1.10 Construction and Pre-Production Logistics

A critical aspect of the schedule revolves around the Project logistics plan. The logistics plan relies on the construction of the access road in Year -3 with substantial completion prior to the start of on-site construction in Q2 Year -2. The access road will be the primary means of delivering equipment and materials required for construction activities. A fixed wing aircraft campaign will support construction activities, with the delivery of select bulk materials and passengers through the duration of construction.

The plan addresses the requirements for dry freight, fuel and air freight. Personnel transport and catering freight to support the project schedule is also provided. The available source data and methods used to determine the feasible capacity of each delivery method are discussed within the logistics plan.

The subsections below describe the methodology used to determine the distribution of freight between the available delivery methods, as well as the available source data and methods used to determine the feasible capacity of each delivery method.

25.1.11 Load List & Methods of Delivery

There are an estimated 25,220 t of dry freight required to support construction and pre-production activities for the Project. A comprehensive load list was prepared to analyze the weights of all equipment and materials required on site. The 25,220 t of freight is made up of 24,969 t of material and equipment for construction along with 251 t of catering freight. Catering freight will be delivered either by aircraft or truck depending on whether or not the access road is open. During periods that the access road is closed, due to winter freeze-up or spring break-up, catering freight will be flown to site. The inbound freight breakdown by area is included in Table 25.6.

Table 25.6. Pre-production freight requirements

Freight Classification	Year -2 (tonnes)	Year -1 (tonnes)	Total (tonnes)
Ground Freight – Construction	7,282	17,687	24,969
Ground Freight – Catering	54	149	203
Airfreight - Catering	13	35	48
Total Dry Freight	7,349	17,871	25,220

Source: JDS 2016

In addition to the freight requirements, a total of 17.6 ML of fuel is required for the pre-production period. A comprehensive fuel demand list was prepared to analyze the volume of fuel required by area. All fuel will be transported to site by 46,500 L capacity triaxle fuel tankers.

25.1.11.1 Air Freight

25.1.11.1.1 Fixed Wing Aircraft

Fixed wing aircraft will deliver personnel and, when necessary, catering freight to the site throughout the construction period.

The Coffee Project site airstrip is designed to handle larger propeller passenger aircraft similar in size to a Hawker Siddeley 748. The airstrip is also sufficiently sized to handle cargo aircraft up to a De Havilland DHC-5D Buffalo, although it is not anticipated that the Buffalo will be utilized during the construction period. Aircraft and their associated capacities are listed in Table 25.7.

Table 25.7. Coffee Site aircraft and capacity

Aircraft	Primary Usage	Capacity
Hawker Siddeley 748	Supplies	4,080 kg
Hawker Siddeley 748	Passengers	40 person
De Havilland DHC-5D Buffalo	Material, equipment	8,164 kg

Source: JDS 2016

Flights will originate in Whitehorse, YT. The approximate straight-line distances between Whitehorse and the site is 340 km.

The sections below further describe the requirements for each type of cargo to be delivered by fixed wing aircraft.

Fixed Wing Aircraft Dry Freight

Aircraft will be chartered from Whitehorse for the delivery of bulk freight to the Coffee site on an as-needed basis. Typically, bulk freight will consist of catering supplies during the periods when the access road is closed. Table 25.8 provides expected number of flights using a Hawker Siddeley 748 to deliver dry freight during the pre-production period.

Table 25.8. Fixed Wing Aircraft Freight Requirements

	Year -2	Year -1	Total
Airfreight – tonnes	13	35	48
Number of flights	4	9	13

Source: JDS 2016

Fixed Wing Aircraft Passenger Transportation

For the purposes of the Feasibility Study it has been assumed that personnel will be mobilized to site from Whitehorse utilizing a Hawker Siddeley 748 aircraft. An average capacity factor of 90% has been used for load planning. Annual passenger delivery requirements are summarized in Table 25.9.

Table 25.9. Passenger Flight Requirements

	Year -2	Year -1	Total
Round-trip passenger numbers	2,276	6,328	8,604
Number of return flights	64	176	240

Source: JDS 20156Ground Freight

Freight and fuel will be transported to the Coffee Project site annually on the access road commencing in Year -2. Freight and fuel quantities have been estimated for the pre-production period and are shown in Table 25.10.

Table 25.10: Pre-Production Ground Freight and Fuel Quantities

	Freight (tonnes)	Fuel (000 litres)
Year -2	7,336	3,289
Year -1	17,836	14,340
Total LOM	25,172	17,629

Source: JDS 2016

Freight and fuel will be transported to the Coffee Project site by trucking contractors using off-highway trucks. Freight hauling will primarily utilize tridem flat deck trailers for standard loads and a variety of trailer configurations for oversize equipment. Freight hauling is expected to average 25 t per load. Fuel hauling will utilize standard tridem tankers with a 46,500 L capacity. Annual freight loads have been estimated based on the above load factors and are listed in Table 25.11.

Table 25.11: Pre-production Truck Loads

	Freight Loads	Fuel loads	Total Loads
Year -2	294	71	365
Year -1	714	309	1,023
Total LOM	1,008	380	1,388

Source: JDS 2016

The barges are estimated to operate an average of 158 days per year. With the estimated 6 week (42 days) road closure during freeze-up, approximately 200 days or 55% of the annual loads will be transported to site during the open-water season. Barges will operate on a Monday-to-Friday, day shift only basis. Each barge will be operated by a certified captain and have a labourer assistant. Table 25.12 provides the annual loads to be transported during the open water season along with the average weekly and daily load requirements.

Table 25.12: Open Water Season Pre-Production Truck Loads

	Freight (tonnes)	Fuel (000 litres)
Year -2	7,336	3,289
Year -1	17,836	14,340
Total LOM	25,172	17,629

Source: JDS 2016

The ice crossings are estimated to be open for haulage an average of 137 days per year, except during the estimated 4-week (28 days) road closure awaiting ice break-up. Approximately 45.2% (165 days) of the annual loads will be transported in total to site during the winter season. The access road will be open to traffic on a Monday-to-Friday day shift only basis. Table 25.13 provides the annual loads to be transported during the winter season along with the average weekly and daily load requirements.

Table 25.13: Winter Season Pre-Production Truck Loads

	Total Loads	Weekly Average	Daily Average
Year -2	165	8.4	1.7
Year -1	462	23.6	4.7

Source: JDS 2016

Fuel off-loading at the Coffee site will be performed by the haul truck drivers utilizing pumping systems at the bulk fuel storage facilities. Freight off-loading will be performed by site support crews with assistance from the haul truck drivers and labourers. Site support equipment will be utilized to off-load freight trucks.

25.2 Operational Logistics

25.2.1 Introduction

The primary focus of operational logistics revolves around the use of the all-weather access road from Dawson for the transportation of equipment, fuel and consumables to the Coffee site. Although the road will be used to deliver bulk consumables and equipment, fixed wing aircraft will deliver select materials, perishables (food) and passengers throughout the operational life of the mine.

The distribution of freight between the delivery methods, as well as the available source data and methods used to determine the feasible capacity of each delivery method is described.

25.2.2 Load List

An estimated 184,937 t of dry freight will be required to support operations over the mine life.. A comprehensive load list was prepared to analyze the weights of all equipment and materials required on site. The 184,937 t of freight is made up of 183,356 t of consumables material and equipment for operations and 1,580 t of catering freight. Catering freight will be delivered either by aircraft or truck depending on whether or not the access road is open.

During periods that the access road is closed, while waiting for winter freeze-up or spring break-up, catering freight will be flown to site. The inbound freight breakdown by year is included in Table 25.14.

Table 25.14: Operational Dry Freight Requirements

	Ground Freight (tonnes)		Airfreight Catering (tonnes)	Total Dry Freight (tonnes)
	Operations	Catering		
Year 1	21,513	131	31	21,675
Year 2	21,646	145	35	21,826
Year 3	22,467	148	35	22,650
Year 4	21,448	149	35	21,632
Year 5	23,861	149	36	24,046
Year 6	21,432	152	36	21,620
Year 7	24,317	152	36	24,505
Year 8	20,609	147	35	20,791
Year 9	5,855	91	22	5,968
Year 10	209	12	3	224
Total	**183,357**	**1,276**	**304**	**184,937**

Source: JDS 2016

In addition to the dry freight requirements, a total of 284 ML of fuel is required for the operational period. A comprehensive fuel demand list was prepared to determine the volume of fuel required by area. All fuel will be transported to site by 46,500 L capacity triaxle fuel tankers.

25.2.3 Methods of Delivery

25.2.3.1 Air Freight

25.2.3.1.1 Fixed Wing Aircraft

Fixed wing aircraft will deliver personnel and when necessary, catering freight, to the site throughout the production period.

The Coffee site airstrip is designed to handle propeller passenger aircraft similar in size to a Hawker Siddeley 748. The airstrip is also sufficiently sized to handle cargo aircraft up to a De Havilland DHC-5D Buffalo, although it is not anticipated that the Buffalo will be utilized during the production period. Aircraft and their associated capacities are listed in Table 25.15.

Table 25.15. Coffee Gold Project Site Aircraft and Capacity

Aircraft	Primary Usage	Capacity
Hawker Siddeley 748	Supplies	4,080 kg
Hawker Siddeley 748	Passengers	40 person
De Havilland DHC-5D Buffalo	Material, equipment	8,164 kg

Source: JDS 2016

Flights will originate in Whitehorse, YT. The approximate straight-line distances between Whitehorse and the site is 340 km.

The requirements for each type of cargo to be delivered by fixed wing aircraft are described below.

Fixed Wing Aircraft Dry Freight

Aircraft will be chartered from Whitehorse for the delivery of bulk freight to the Coffee site on an as-needed basis. Typically, bulk freight will consist of catering supplies during the periods when the access road is closed.

Table 25.16 provides expected number of flights using a Hawker Siddeley 748 to deliver dry freight during the operational period.

Table 25.16. Fixed Wing Aircraft Fright Requirements

	Airfreight (tonnes)	Number of Flights
Year 1	31	8
Year 2	35	9
Year 3	35	9
Year 4	35	9
Year 5	36	9
Year 6	36	9
Year 7	36	9
Year 8	35	9
Year 9	22	6
Year 10	3	1
Total	**304**	**78**

Source: JDS 2016

Fixed Wing Aircraft Passenger Transportation

Personnel will be transported to and from site from Whitehorse utilizing a Hawker Siddeley 748 aircraft. An average capacity factor of 90% has been used for load planning. Annual passenger delivery requirements are summarized in Table 25.17.

Table 25.17. Passenger Flight Requirements

	Round Trip Passenger Numbers	Number of Round Trip Flights
Year 1	5,599	157
Year 2	6,122	171
Year 3	6,254	175
Year 4	6,267	175
Year 5	6,296	176
Year 6	6,381	179
Year 7	6,374	178
Year 8	6,207	174
Year 9	3,961	111
Year 10	556	16
Total	**54,017**	**1,512**

Source: JDS 2016

25.2.3.2 Ground Freight

Freight and fuel will be transported to the Coffee site on the access road. Freight and fuel quantities have been estimated for the operational period and are shown in Table 25.18.

Table 25.18: Operational Ground Freight and Fuel Quantities

	Freight (tonnes)	Fuel (000 L)
Year 1	21,644	30,667
Year 2	21,791	33,657
Year 3	22,615	31,492
Year 4	21,597	31,945
Year 5	24,010	31,616
Year 6	21,584	32,265
Year 7	24,469	33,516
Year 8	20,756	31,692
Year 9	5,946	12,414
Year 10	221	621
Total	**183,357**	**269,885**

Source: JDS 2016

Freight and fuel will be transported to the Coffee Gold Project site by trucking contractors using off-highway trucks. Freight hauling will primarily utilize tridem flat deck trailers for standard loads and a variety of trailer configurations for oversize equipment. Freight hauling is expected to average 25 t per load. Fuel hauling will utilize standard tridem tankers with a 46,500 L capacity. Annual freight loads have been estimated based on the above load factors and are listed in Table 25.19.

Table 25.19: Operational Truck Loads

	Freight Loads	Fuel Loads	Total Loads
Year 1	866	660	1,526
Year 2	872	724	1,596
Year 3	905	678	1,583
Year 4	864	687	1,551
Year 5	961	680	1,641
Year 6	864	694	1,558
Year 7	979	721	1,700
Year 8	831	682	1,513
Year 9	238	267	505
Year 10	9	14	23
Total	**7,389**	**5,807**	**13,196**

Source: JDS 2016

The barges are estimated to operate an average of 158 days per year. With the estimated 6 week (42 days) road closure while awaiting freeze-up, approximately 200 days or 55% of the annual loads will be transported to site during the open-water season. Barges will operate on a Monday-to-Friday, day shift only basis. Each barge will be operated by a certified captain and have a labourer assistant. Table 25.20 provides the annual loads to be transported during the open water season along with the average weekly and daily load requirements.

Table 25.20: Open Water Season Operational Truck Loads

	Total Loads	Weekly Average	Daily Average
Year 1	836	37.0	7.4
Year 2	875	38.8	7.8
Year 3	867	38.4	7.7
Year 4	850	37.7	7.5
Year 5	899	39.8	8.0
Year 6	854	37.8	7.6
Year 7	932	41.3	8.3
Year 8	829	36.7	7.3
Year 9	277	12.3	2.5
Year 10	0	0.0	0
Total	**7,219**		

Source:

The ice crossings are estimated to be open for haulage an average of 137 days per year. With the estimated 4 week (28 days) road closure awaiting ice break-up, approximately 165 days or 45.2% of the annual loads will be transported to site during the winter season. The access road will be open to traffic on a Monday-to-Friday day shift only basis. Table 25.21 provides the annual loads to be transported during the winter season along with the average weekly and daily load requirements.

Table 25.21: Winter Season Operational Truck Loads

	Total Loads	Weekly Average	Daily Average
Year 1	690	35.3	7.1
Year 2	721	36.8	7.4
Year 3	716	36.6	7.3
Year 4	701	35.8	7.2
Year 5	742	37.9	7.6
Year 6	704	36.0	7.2
Year 7	768	39.2	7.8
Year 8	684	34.9	7.0
Year 9	228	11.6	2.3
Year 10	23	1.2	0.2
Total	**5,977**		

Source: JDS 2016

Fuel off-loading at the Coffee site will be performed by the tanker drivers utilizing pumping systems at the bulk fuel storage facilities. Freight off-loading will be performed by site support crews with assistance from the haul truck drivers and labourers. Site support equipment will be utilized to off-load freight trucks.

25.3 Reclamation and Mine Closure Plan

The reclamation and closure plan (RCP) will be submitted as part of the project proposal to YESAB. The RCP will include a liability estimate for reclaiming and closing the mine at the end of the mine life set out in this Feasibility Study. Consultation has been initiated with First Nations, affected communities, regulatory agencies and public organizations to gain an understanding of their expectations and concerns with regard to the reclamation and closure of the mine at the end of its operating life. The RCP will demonstrate how Kaminak considered and addressed the expectations and concerns throughout the mine planning process.

The RCP sets out the methodology and phases of reclamation and closure. Associated costs are set out in Section 21 (Capital Cost Estimation).

25.3.1 Closure Objectives

The RCP objectives are to return the mine site and affected areas to viable and, wherever practicable, self-sustaining ecosystems that are compatible with a healthy environment and human activities. To achieve this, the following objectives will be followed:

- Design the Mine for Closure: This involves identifying the processes and activities that occur during closure so that they may be incorporated as much as possible into operations. This includes progressive closure as allowed by and during active operations.
- Achieve Physical Stability: The objective of achieving sustainable physical stability after closure is to minimize risk to humans, wildlife, and the environment, and reduce long term closure maintenance costs. Mine components that are to remain after mine closure will be constructed or modified so that they are physically stable and not subject to undue erosion or subsidence.
- Achieve Chemical Stability: All mine components and waste materials remaining after mine closure will be chemically stable. Any chemical constituents that may be released from the mine area will be in such a form or concentration that they will not endanger humans, wildlife, or the environment.
- Consider Health and Safety: Eliminate or minimize hazards that may impact the health and safety of mine personnel, contractors, land-users, wildlife, and the environment.
- Consider Future Use and Aesthetics: Reclamation and re-vegetation studies were initiated in 2013. Local seeds are being collected and will be used to aid biological productivity of reclaimed and disturbed sites. Where practical, contouring, backfilling and re-vegetation will be undertaken to improve the aesthetics of the site.

25.3.2 Closure Criteria

The Yukon Water Board has stated that it seeks to "…issue licences only when there is a reasonable certainty that an acceptable level of reclamation of the site can be achieved during mining and/or following cessation of mining[4]". The mine plan is being developed with closure in mind such that closure considerations are integrated into the mine's planning and operational processes. In taking this approach, reclamation and closure will be assisted by early planning which, at a later date, will lead to an effective closure while also reducing the overall reclamation and closure cost. Over the life of the mine, successive iterations of the RCP will be required every two years, each iteration providing more detail and greater certainty regarding the sequence of events to occur during reclamation and closure.

The RCP for the Coffee Project will be developed in accordance with current best practices. With respect to final water quality standards and site-specific thresholds, the Metal Mining Effluent Regulations (MMER) and the Canadian Council of Ministers for the Environment (CCME) water quality guidelines will be adopted and applied.

25.3.3 Closure Costing

The reclamation and closure costs assume that all post-mining closure activities will be undertaken by contractors. The reclamation and closure costs are estimated at $60 million including contingency. This estimate will be further developed and optimized during the course of the mining operation and as progressive closure takes place.

25.3.4 Closure Schedule

Progressive reclamation activities will begin as soon as mining at the Double Double pit has been completed in Year 2 and continue through the rest of the 10-year operating mine life.

Mine closure occurs in four well-defined phases:

- Operational closure – Years 2 to 10: each pit, WRSF and stages of the HLF will be closed as they are decommissioned;
- Post-mining closure – Years 10 to 15: closure activities relating to terminating the mining operation, dismantling infrastructure and reclamation;
- Active closure – Years 16 to 20: maintenance, monitoring and closure of remaining facilities; and
- Post-closure – Year 21 onwards: monitoring only.

[4] Source: http://www.emr.gov.yk.ca/mining/pdf/mml_reclamation_closure_planning_quartz_mining_projects_aug2013.pdf

25.3.5 Temporary Closure

Should mining cease before completion of the mine plan, with the intention that the cessation is temporary, a care and maintenance program will be necessary. A Temporary Closure Plan will be developed to support the project proposal submission to YESAB. The Temporary Closure Plan will address, but not be limited to:

- Site security will be achieved by restricting access to authorized personnel only. Mine openings will be barricaded or guarded, warning signs will be placed around open pits and mine openings, hazardous materials and explosives will be safely stored, machinery and mobile equipment will be secured;
- Physical and chemical stabilization of mine structures;
- Inspection and monitoring activities will be initiated; and
- Temporary closure activities will continue until mining resumes or until the decision is made to permanently close the mine. Should the mine close permanently, a final Mine Closure Plan will be filed with the Yukon Environmental and Socio-economic Assessment Board (YESAB) and final closure activities will begin.

25.3.6 Closure Activities

25.3.6.1 Progressive Closure Activities

25.3.6.1.1 Open Pits

The sequence of the completion of each pit will be Double Double, Latte, Kona and then Supremo. Although production from the pits overlaps, there is opportunity to use Latte, Supremo and Double Double for the storage of waste rock as backfill. Supremo Pit will be mined last, and will consequently be used for non-hazardous waste disposal.

The waste rock placed in the mined out pits will be to a level that will allow it to be submerged a minimum of 2 m below the natural decant point of the pit.

Boulder fences will be placed around each open pit as it is mined out. Boulders will be minimum 1 m in diametre, placed 3 m from the final pit crest, and will be spaced no more than 3 m apart. The intention is not to prevent access, but to be a significant visual aid suggesting a change in landscape that will act as a warning sign to both humans and large mammals.

Certain areas of the North WRSF will be designated as landfills in order to accommodate the disposal of non-hazardous waste and equipment during and after the cessation of operations.

Pit sumps and associated pumps and pipelines are to be removed as each open pit is mined out. If the pumps are not to be reused, hazardous material will be removed from the pumps and disposed of at an off-site licensed facility. The pumps will be placed in the designated landfill. Associated pipelines, if not being reused, will be cleaned if necessary and also placed in the landfill.

Should pits fill to natural decant points, and once it is determined that water quality criteria have been met, natural drainage will be re-established. Appropriate erosion protection measures will be constructed at the overflow location to ensure management of suspended sediments.

Any mobile equipment used in the open pits that is past its service life will have all hazardous materials removed and disposed of at a licensed facility. The stripped equipment will be placed in a landfill on site.

Topsoil and subsoil from the pre-stripping of the surface of each pit, WRSF and facility footprint will be stored in a stockpile located immediately north of the heap leach facility. This material will be used for reclamation and re-vegetation at closure.

25.3.6.2 Heap Leach Facility

The heap leach facility will be progressively closed and reclaimed beginning in project Year 5. The slopes and crest of the heap will be regraded then capped with a geosynthetic clay liner (GCL) and 500 mm of overburden material. Final closure of the heap leach will begin when rinsing is complete in Year 12.

Two of the three event ponds will be reclaimed after rinsing is complete. The pond liners will be buried within the ponds and the ponds backfilled. The third pond will remain open to accept sludge from the water treatment plant until water treatment is complete. At that point the third pond will be reclaimed in similar fashion as the previous two ponds.

25.3.6.3 Waste Rock Storage Facilities and Non-Hazardous Landfills

WRSF will be constructed and maintained to meet closure criteria and will not require progressive re-sloping. At the end of active waste dumping, minor re-sloping may be carried out in order to ensure long term physical stability. The WRSF’s will not be covered or vegetated.

All landfill areas will be capped with a minimum of 5 m of waste rock.

25.3.6.4 Water Management Structures

All contact and non-contact water storage ponds will be pumped out and the containment walls breached. Contaminated sediments will be excavated and disposed of in the open pits. Any liners will be completely removed both in containment structures and diversions. Ponds and diversion structures will be backfilled where required to ensure proper drainage. The areas will then have a topsoil layer replaced.

25.3.6.5 Water Treatment

Water treatment of drain-down rinse water deriving from those parts of the heap leach that have been closed will commence in Year 9. The treatment plant will be constructed in Year 8 and will be housed in a repurposed infrastructure building. It is anticipated that the rinse water will exceed MMER water quality standards for discharges to the environment with respect to cyanide and arsenic, and possibly uranium and nitrates as well. The treatment plant will operate through closure to Year 20. It will operate for the approximately eight months of the year that flowing water is present.

The plant is designed to treat 34 m^3/hr (10 L/s). Influent for the plant will be drawn either via the events ponds or via the barren/pregnant solution tanks depending on operational requirements and the status of closure.

The first stage of the treatment process will be to add hydrogen peroxide to oxidize residual cyanide. The products of this process are cyanate and/or ammonia and carbon dioxide. This stage of the treatment system consists of a 1.4 m diameter X 1.7 m tall, carbon or stainless steel tank with an agitator. The agitator will have a 1 hp motor and a 0.7 m impellor. Wetted parts will be stainless steel. A dedicated metering pump will be used to dose the tank with hydrogen peroxide. The tank effluent will flow by gravity to the second stage of the process.

In the second stage of the process ferric sulphate is added in order to precipitate ferric hydroxide. First stage effluent will flow into a second 1.4 m diameter X 1.7 m tall, carbon or stainless steel tank with an agitator as per the first stage. The objective of the second stage is to remove arsenic and uranium. Arsenic and uranium will co-precipitate with the ferric hydroxide. A pH controller in the tank will be used to meter sulphuric acid dosing to the tank to optimize arsenic and uranium removal. The sulphuric acid metering pumps will draw sulphuric acid from a dedicated storage tank.

Flocculent will be added after the precipitation of ferric hydroxide to enhance settling of the precipitates. Precipitates will be separated from treated water in a 7 m diameter coated steel clarifier. The clarifier underflow will be pumped to a sludge disposal cell located in a spent portion of the HLF. Clarifier overflow will be pumped to a 0.6 m tall X 1.7 m diameter sand filter to remove residual suspended material. Treated water from the water treatment plant will either be discharged to the environment or used in additional rinsing of the heap. Water discharged to the environment will occur to either Halfway Creek or Latte Creek drainage via release to Latte pit where seepage via groundwater will occur.

When operation of the plant is no longer required, the plant will be decommissioned and reclaimed using measures similar to those described for the process plant. The cost of the plant is $2.0M and the annual operating and maintenance cost is estimated at $508K. Both these costs are captured in the mine closure costs.

25.3.6.6 Haul Roads and Service Roads

Haul roads and service roads that are no longer required once operations are complete will be decommissioned. This will entail removal of any culverts to re-establish pre-construction surface drainage, and general grading of the road surface to promote water runoff. Once grading is complete the areas will be scarified, top-soiled and seeded.

25.3.6.7 Post-Mining Closure Activities

Although reclamation continues progressively during the operational phase of the mine, closure activities are triggered by the completion of mining and ore processing. At this time the relevant infrastructure will be dismantled and final reclamation initiated.

25.3.6.8 General Dismantling and Disposal Procedures

Prior to dismantling (which may include demolition), all buildings and equipment will be inspected to ensure that potentially hazardous materials have been identified correctly and flagged for appropriate removal and handling. All equipment will be drained of fluids and cleaned to ensure that potentially hazardous materials are not placed within the landfill site.

25.3.6.9 Inert Solid Materials

Non-hazardous materials from site buildings, structures, and equipment will be dismantled and deposited in the landfill. Appropriate authorization for this non-hazardous waste disposal site will need to be obtained before starting closure and reclamation activities.

25.3.6.10 Hazardous & Materials

Potentially hazardous structures, equipment, materials and fluids will be transported from site.

Hazardous materials are generally expected to include waste oil, glycol, lubricants, solvents, paints, batteries, and miscellaneous chemicals. Some of these materials may be suitable for recycling if appropriate facilities are available in reasonable proximity (Alberta or British Columbia) at the time of mine closure and if recycling can be done at reasonable cost.

Hazardous materials will be stored in sealed containers and drums in a lined waste transfer area or temporary enclosure. The materials will be barged out to appropriate disposal, recycling, or salvage facilities during the next available summer season.

25.3.6.11 Process Facilities

The crusher facility will be decommissioned upon completion of mining activities in Year 10. The process plant will be decommissioned after the final rinsing of the heap leach is complete and the solution has been processed to extract the contained gold. Once all the ore and solution has been processed, the various circuits will be cleaned with water and the final washings will be treated prior to discharge. All potentially hazardous materials such as hydrocarbons, chemicals, and reagents will be removed and prepared for off-site disposal. Reagent tanks will be drained and cleaned prior to demolition. Any potentially hazardous materials will be drained from the process equipment. In addition, all utilities and services, including air, glycol, power, and water, will be shut off, de-energized, and drained as necessary to permit demolition to proceed safely and without compromising services to other areas in use.

All buildings and equipment will be dismantled and disposed of in the inert materials disposal site. Specific materials will be handled as follows:

- Concrete foundations and floor slabs will be broken down to original ground level and demolition rubble disposed in the landfill;
- Any buried piping will be removed to just below grade and ends will be capped. Any buried fuel and glycol lines will be flushed with water, removed, and buried in the landfill. Surface piping will be flushed, if necessary, removed, and buried in the inert disposal site.
- Any buried electrical cables will be cut approximately 1 m below grade at surface terminations and left intact. The remaining above-ground cable will be removed and disposed of in the inert disposal site.
- All other inert materials not suitable for reuse or salvage, such as metal cladding, wallboard, and insulation, will be disposed of in the inert disposal site.

25.3.6.12 Service Shop and Warehouse Complex

The shop and warehouse complex will be decommissioned, cleaned, and dismantled in a manner similar to the process plant. The building will be inspected and all potentially hazardous materials will be removed and packaged for off-site shipment and disposal. The remaining equipment and the building will be dismantled, demolished, and transported to the landfill site.

25.3.6.13 Power Plant

The power plant is modular with a total of four generator modules. Three of the four generators will be decommissioned, dismantled and reclaimed using measures similar to those described for the process plant. The fourth unit will be utilized until closure activities are complete.

25.3.6.14 Accommodations and Office Complex

A portion of the main camp complex will be utilized throughout active closure. Once active closure is complete, the remainder of the camp will be demolished and crews will utilize the pre-existing exploration camp adjacent to the Yukon River. The permanent facility will be decommissioned and reclaimed using measures similar to those described for the process plant.

25.3.6.15 Site Support Services

To support personnel during closure activities, site services, including potable water treatment, sewage treatment, and communications, will be maintained until they can be decommissioned, dismantled with the demolition debris, and disposed of on site in the inert disposal site. Smaller, temporary facilities at the exploration camp will support post-closure monitoring activities.

25.3.6.16 Access Road and Airstrip

The access road from Dawson to the Coffee site along with the barge crossing facilities on the Yukon and Stewart Rivers will be utilized for approximately one month per year throughout the active closure period. Once post-mining closure is complete, the new portions of the access road built for the Coffee Project, will be decommissioned. This will entail removal of any culverts and bridges to maintain pre-construction surface drainage, removal of the barge landings and general grading of the road surface to promote runoff shedding. Once grading is complete the road surface will be scarified. Once the access road has been decommissioned the annual resupply of consumables and materials required for active closure will be transported to site by barge campaigns on the Yukon River.

The airstrip will be reclaimed near the end of the project post-mining closure phase. Lighting, navigation equipment, and culverts will be removed, and contouring will be undertaken to eliminate potential hazards to wildlife. Reclamation will involve scarifying, top-soiling and seeding. Once the airstrip is reclaimed the existing airstrip at the exploration camp will be utilized.

All site roads not required for post-closure maintenance and monitoring will be decommissioned and reclaimed at the end of the active closure phase. The remaining roads will be reclaimed at the end of active closure monitoring. Post-closure access to the site will be primarily by aircraft.

25.3.6.17 Concrete Structures

All other above-grade concrete structures will be demolished, and any remaining below-ground footings/foundations will be covered with till or rock. Demolition concrete will be placed in the landfill site.

25.3.6.18 Fuel Storage Tanks

Before dismantling the permanent tanks, any remaining fuel will be withdrawn. Steel plate sections and distribution system components will be washed and disposed of in the landfill site, pursuant to regulatory approval. The containment berm and liner materials will be removed and the area regraded. Once the tanks are removed, appropriately sized portable fuel storage tanks will be used to store diesel for equipment and power generation.

25.3.6.19 Ammonium Nitrate and Emulsion Storage

The remaining inventory of explosives, comprising ammonium nitrate (AN) and emulsion product, at the end of mining will either be returned to the supplier or transferred to another licensed user. The pad areas will be regraded to blend with the surrounding topography then scarified, top-soiled and seeded.

The AN truck shop will be decommissioned, cleaned, and demolished in the same way as other site buildings.

25.3.6.20 Explosive Magazines

The remaining inventory of packaged explosives and detonators will be returned to the supplier or transferred to another licensed user. The cap magazines will be decommissioned, cleaned, and either removed from site for salvage or demolished in the same way as other site buildings.

25.3.6.21 Testing and Disposal of Contaminated Soil

The potential for ground contamination at the various facilities will be assessed. This will include the airstrip area, fuel tank farm, process plant, power plant, accommodations complex, service complex, waste management facilities, and storage facilities. Soils in these areas will be sampled during decommissioning and analyzed for contaminants such as hydrocarbons and glycol.

Any contaminated soils will be excavated and either treated on site to an acceptable standard or stored in appropriate sealed containers for off-site shipment and disposal.

A site investigation will be carried out to determine the volume of contaminated soil from hydrocarbon spills over the life of the mine. The investigation will be conducted using a direct push drill rig and drilling will occur down to a base of active layer (5 m). Drilling will be focused in all parking bays, fuel storage, wash bays, truck shops, maintenance areas, and generator areas, as well as along roads and in areas where spills have been known to occur.

If the volumes of contaminated soils are significant, the materials will be remediated with on-site land farms constructed specifically for this purpose. If the volumes are small, the material will be shipped off-site for disposal at a licensed facility.

25.3.6.22 Active Closure Activities

Most of the land disturbance associated with ore processing, infrastructure and mine disposal will have been stabilized and reclaimed as part of the progressive and active closure programs. Both of these phases of reclamation will be complete at the end of Year 15, at which point active closure will commence and last approximately five years.

Activities focus on providing support for water treatment and water quality monitoring. Water treatment will operate eight months per year, during the summer months. Sludge generated from the treatment of water will be disposed of in the heap leach event pond through year 19, until the final event pond is backfilled and reclaimed. Year 20 sludge will be transported off-site to an approved disposal facility.

Access to the site and required services will be during summer only and access to site will be by air only. Final demobilization will be serviced by a barge campaign on the Yukon River.

Facilities that will be maintained will be:

- The airstrip at the exploration camp;
- Some site roads;
- The exploration camp with sewage system;
- Potable water supply;
- Power generation; and
- The water treatment plant.

Fuel storage will also be retained as per the decommissioning plan. Services will require a small rental equipment fleet as follows:

- One excavator;
- Two pick-up trucks;
- One D6 dozer;
- One motor grader; and
- Two articulated dump trucks.

25.3.6.23 Monitoring

25.3.6.23.1 Geotechnical Monitoring

During operations, and until closure is complete, an annual geotechnical inspection must be carried out by a qualified geotechnical engineer licensed to practice in the Yukon. These inspections should be carried out during the summer months when there is no snow cover. Areas to be included in the inspection are the heap leach facility, all WRSFs, open pit high walls, all contact and non-contact water storage ponds, water diversion structures and any other surface infrastructure elements possibly affecting permafrost.

25.3.6.23.2 Water Quality Monitoring

Water quality monitoring will start at the beginning of construction and extend into operations. Monitoring at designated control points will be checked in accordance with the license criteria, both with respect to frequency and the necessary testing parameters. Following closure, water quality monitoring will systematically be scaled back, with ultimate cessation once there is at least five years of water quality monitoring that confirms that the final closure objectives have been met.

25.3.6.23.3 Terrestrial and Aquatic Effects Monitoring

Terrestrial and aquatic effects monitoring will be carried out in accordance with license criteria. Cessation of this monitoring will be when a 5-year period has elapsed and demonstrates that the system has achieved the stated closure objectives.

26 Interpretations and Conclusions

The economic results of this Feasibility Study demonstrate that the Project has positive economics and warrants development.

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.

23.1 Risk and Opportunity Management

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. Certain opportunities that can enhance the Project economics might also be identified during the early years of construction, particularly with respect to conservative engineering and design parameters applied during the engineering stage of Project development.

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 and opportunities for the Coffee Gold Project, including the potential impacts, and possible mitigations.

The project risk assessment was conducted in a series of workshops involving Kaminak personnel and consultant leads representing their respective disciplines. Emphasis was placed on the identification of currently significant Project risks and possible high level mitigation efforts. A formal review of the likelihood and consequence ratings and pre- and post-mitigation rankings was not conducted: this will be performed during the detailed engineering phase of development.

The remote location of the Coffee Gold site imposes an additional level of risk to the Project due to the large volume of equipment, fuel and materials that need to be transported to the mine site during construction and operations. Superimposed on the remoteness of the site are seasonal conditions relating to the crossing of the two major rivers that intersect the site access road. These have been carefully considered in the FS. However, seasonal weather variations are rarely predictable and appropriate mitigations will be required in the event of cooler summers or milder winters.

Permitting processes are known to be long and outcomes are sometimes unpredictable. The uncertainty of permitting timelines could have a major impact on the Project development schedule.

Extreme winter temperatures will impact personnel and equipment productivities during construction and operations.

The remote location of the site, shift rotations, climatic conditions and transportation requirements might make it difficult to attract and retain skilled personnel.

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 Feasibility Study, risk identification and review of mitigations will continue to be a priority during project development, construction and operations.

Table 26.1: Risks and Opportunities

Description	Aspect	Impact Description	Impact Risk	Impact Opportunity	*Risk Mitigation/Opportunity Steps*
Accessibility, Climate, Local Resources, And Physiography					
Climate	Change in annual precipitation	Higher precipitation	Exceedance of water conveyance/storage structures, including treatment	Reduction in make-up water requirements	*Continuous climatic monitoring and awareness of impact on operations. Emplacement of emergency response plans and resources. Adaptive capability in logistics and storage to respond to changes in transportation capacity. Timeous consultation with regulators with respect to changes in river crossing methods and structures. Adaptive management of contact water for storage and use during low-precipitation periods.*
			Flood conditions for barge/ferry crossing		
			Increased avalanche potential		
			Large volume of contact water	Increased assimilative capacity of surrounding watersheds	
		Lower precipitation	Inadequate make-up water available	Reduced water treatment requirements	
			Fire hazard (camp evacuation)		
			Reduced assimilative capacity of surrounding watersheds		
Geology And Exploration					
Geological Model	Accuracy of geological model	Location of the orebody	Pit unoptimized	Unidentified ore bodies mapped during mining by grade control operations	*Short-, medium- and long-term mine plan reviews and appropriate adaption. Ongoing review of grade control procedures Appropriate on-site technical skills base to recognize and respond to resource-based issues.*
			Unidentified zones treated as waste		
		Grade of the ore	HLF head grade lower than expected	HLF head grade higher than expected	
		Continuity of grade	Waste rock placed in HLF	Ore placed in WRSF	
		Classification of oxide facies	Lower recoveries than expected	Higher recoveries than expected	
Exploration	Identification of new orebody	Additional resources/reserves added to mine plan	Additional permitting required for expansion	Increase of LOM	
Mineral Processing And Metallurgical Testwork					
Metallurgy	Gold recovery test work	Different gold recoveries to predicted	Lower gold production	Higher gold production	*Ore sizing review. Heap leach solution management review.*
		Reagent consumption	Inadequate supply of and storage facilities for reagents.	Lower reagent consumption	*Increase leach pad stacking rate and or stacking area*
		Leach kinetics	Larger leach pad expansions	Smaller pad expansions	*Introduction of tertiary crushing stage*
			Lower near term gold production	Higher near term gold production	*Pre-crushing screening*
			Installed plant capacity insufficient		*Change in solution management.*
	Ore properties	Ore hardness	Reduced throughput	Increased throughput	*Blasting fragmentation control. Change to crush sizing and screening management. Procurement of temporary mobile crusher to increase throughput.*
			Inadequate crusher size/tertiary crushing		
		Ore sizing	Fines/clay (need for agglomeration)	Crusher bypass	
Mining					
Geotechnical Considerations	Pit wall stability	Slope interramp angles incorrect, unidentified structure/faults, change in rock strength characteristics	Wall failure - health and safety, disrupted production, clean up	Steeper slopes, lower strip ratio	*Appropriate geotechnical inspections and monitoring.*
			Flatter slopes, higher strip ratio, reduced reserves		
Mining Methodologies	Ore recovery or dilution	Operational results different than design criteria	Decreased reserves	Increased reserves	*Ongoing review of grade control procedures. Change pit limits to accommodate revised reserve base.*
Mine Design	Mine production capacity	Maintain design throughput	Failure to achieve ore production rates	Increased ore production rates	*Evaluation of haul routes, fleet sizes and equipment types.*
Waste Rock Storage Facilities	Waste Rock Storage Facility stability	Slope/foundation failure	Equipment damage and/or injury/loss of life		*Appropriate geotechnical inspections and monitoring*
			Production disruption		
	Waste rock volume	Different to mine plan	Greater dump footprint	Smaller dump footprint	*Review individual WRSF capacities and design wrt footprint area and height*
			Increased costs	Reduced costs	*Introduce water treatment earlier/later and change throughput capacity appropriately*
Processing					
Crushing	Crushing throughput	Crusher performance	Reduced production	Increased production	*Pre-crusher screening, introduction of additional module, reduced/increased running time*
Heap Leach Pad	Heap Stability	Slope failures	Reduced heap capacity		*Ensure slope design criteria is met through regular monitoring. Ongoing geotechnical monitoring. Shallower slopes required, necessitating expansion into phase 5. Increase bench width.*
	Construction	Quality control	Liner performance doesn't meet design		*Early detection by appropriate monitoring. Early start to mining and crushing to attenuate throughput rates.*
		Commissioning delay	1-year startup delay		
	Operation	Heap drainage/permeability	Permeability lower than expected; Reduced gold recovery		*Change ore crush size, leach rate and/or solution concentration.*
		Cold weather	Heap freezes	Less heat required	*Change barren solution heating temperature. Cover/insulate drip lines more effectively. Employ raincoats as thermal covers earlier in mine life.*
			Drip lines freeze	Stack ore more than 9 months/year	
	Seismic Activity	Damage to HLF	Loss of solution containment		*Ensure emergency response plans are in place regarding seismic-induced shut-down of barren solution pumping and automatic re-direct to event ponds and/or bypass to Latte pit depending on volume of solution containment compromised and integrity of event ponds following major seismic event.*
	Closure	Progressive and final closure	Proposed closure method unsuccessful	Demonstrate closure methodology to regulators; reduce bond requirements	*Review closure plan throughout mine life. Conduct pilot testing programs throughout mine life.*
			Higher bonding/closure costs		
Gold Recovery	Plant performance	Plant does not perform as designed	Deferring or reduced gold production	Additional capacity	*Review design and throughput rates and re-engineer appropriately*
			Capex for retrofit		
On-Site Infrastructure					
Fuel	Price	Increase/decrease in diesel price	Increased operating costs.	Reduced operating costs	*Flexibility to introduce lower cost fuels (LNG) as per genset design specs.*
	Spill	Fuel spill during transportation to site or at site storage facilities	Damage to the environment , reputational damage, fire		*Ensure emergency response plans in place for entire fuel supply and storage chain. Design adequate spill capacity around fuel tanks.*

Description	Aspect	Impact Description	Risk	Opportunity	Risk Mitigation/Opportunity Steps
Cyanide	Spill	Cyanide spill during transportation to site or at site storage facilities	Damage to the environment , reputational damage		*Ensure best practices are met; subscribe to cyanide management code and ensure compliance. Ensure emergency response plans in place for entire cyanide supply and storage chain. Design adequate spill capacity and suitable containment around storage facilities.*
Health, Safety, Environment	Accident	Significant incident involving personnel or contractors	Injury or death		*Health and Safety Management Plan will specify standard operating procedures as well as job hazard risk assessments. Regular safety meetings and education will be combined with safety audits to ensure site compliance with best practice. Adequate on-site emergency services with external backup resources available at short notice. Adequate emergency response plan in place.*
Off-Site Infrastructure					
Access Route	Access	Loss of route access	Inability to supply freight timeously during construction and operation		*Evaluate alternative supply routes and/or methodologies. Use air freight.*
		Longer shoulder season	Unable to resupply		*10 week on-site storage for all consumables. Airstrip provides air support back-up in emergency.*
	H&S incident	Significant incident involving truck/barge	Supply line interruptions		*Continuous monitoring and truck-to-base communications. Establish medical and environmental response plan and procedure.*
	Construction permit	Timing of permit approval	Delay in construction		*Evaluate alternative supply routes and/or methodologies*
	Communities	Dissatisfied stakeholders and road users	Supply line interruptions		*Continuous consultation and communication with stakeholders. Adequate storage facilities and inventories.*
Water Management					
Contact Water Collection and Storage	Diversion	Structures inadequate leading to loss of containment	Imposition of additional regulatory and sampling requirements leading to cost increases		*Continues monitoring and review of efficacy of water collection and storage structures. As mining operation matures identify potential in-pit storage strategies.*
			Temporary project shutdown		
			Clean up costs		
	Storage	Storage inadequate leading to loss of containment	Imposition of additional regulatory and sampling requirements leading to cost increases		
			Temporary project shutdown		
			Clean up costs		
	Discharge	Rock chemistry different from expected	Contact water not suitable for discharge to environment	Contact water is adequate for release without treatment	*Contact water may be diverted to supply make-up water. Storage capacity and ability to divert contact water to long-term storage in pits during operation. Closure mitigation may include partial cover of WRSF. Bring water treatment facility forward in mine plan.*
		Water quality prediction	Unable to discharge to environment	Contact water is adequate for release without treatment	
	HLF Water Balance	Make-up water requirments incorrect.	Inadequate make-up water available locally		*Water licenses in place to facilitate water import. Review annual water demand and storage. On-site availability of stand-by pumps and pipes to allow water import.*
	Surface water inflow to pit	Inflow exceeds designed capacity	Pits flooding		*Ongoing monitoring and management systems able to respond to changes in water management demands.*
Market Studies And Contracts					
Economic factors	Gold price and exchange rate	Changes in gold price and exchange rate impact economic viability of project.	Reduced economic viability and possible change to mine plan to increase overall gold cutoff grade.	Improved economic viability, possibility of incorporating lower grade reserves into mine plan.	*Short-, medium-, and long-term mine plans incorporate impact of changes in economic factors and mining/processing operations have ability to respond.*
Environmental Studies, Permitting, And Social Or Community Impact					
Permitting	Regulatory process	Permits awarded later than Q2 Year -2	Construction delayed by one year		*Continuous and appropriate communication with regulators and stakeholders during regulatory process. Robust pre-engagement strategy to identify risks, concerns, and opportunities in advance of entering formal regulatory process. Comprehensive submissions that address concerns and follow regulatory guidance documents.*
	Support for the Project	Lack of First Nation and/or Community Support	Permits not granted		*Ongoing engagement, adequately address issues and concerns, provide detailed information in a timely manner, negotiate agreements.*
Manpower					
Manpower	Workforce availability	Not able to adequately fill positions locally	Increased payroll cost and/or operational limitations		*Continuous labour marketing monitoring, attractive remuneration packages, incentives and workplace conditions.*
	Workforce skillset	Lack of adequate education and training programs locally	Unable to fill positions with local qualified individuals		*Consult to identify needs and interests. Develop and deliver training programs locally.*
Capital And Operating Costs					
Capital Cost	Capital cost estimation	Higher than estimated capital costs	Reduction of project valuation and increased payback period		*EPC process adheres strictly to "fit-for-purpose" ethos. Use of pre-used equipment and structures. Contract mining/equipment leasing.*
			Project financing and procurement execution jeopardized		
Operating Cost	Operating cost estimation	Higher than estimated operating costs	Project profitability reduced		*Appropriate frequency and detail of mine production chain planning to facilitate response to operating cost changes.*
			Increased payback period		
Procurement					
Procurement	Equipment lead time	Major mobile and fixed equipment lead times longer than anticipated	Delay in construction		*Early identification of equipment lead times and appropriate procurement strategy. Temporary use of leased equipment and/or contractor mining.*
Closure					
Closure	Permitting	Mine and closure execution plan does not meet regulatory requirements	Failure to obtain necessary permits to commence mining		*Continuous and appropriate communication with regulators during all phases of project planning, construction and operations. Continuous testing of closure methodologies during operations. Ensure operational decisions are aligned with closure strategy.*
	Progressive closure during operations	Failure to close completed heap leach cells	Permit noncompliance leading to additional costs and possible loss of operating permits		
	Closure cost	Closure execution plan incurs excessive costs	Closure costs reduce project valuation		

Source: JDS 2014

27 Recommendations

Based on the robust economic results of this Feasibility Study it is recommended that Kaminak progress the Coffee Gold Project to detailed engineering and construction. These costs are detailed in the capital section of this report; the initial EPCM costs leading to project construction are set out in Table 27.1.

Table 27.1: EPCM Costs for Project Construction.

Description	Initial ($M)
Engineering & Procurement – EP	5.0
Construction Management – CM	13.9
Total EPCM Costs	**18.9**

Source: JDS 2016

During the course of the Feasibility Study the following items were identified as having the potential to further improve the economics of the Project and should be pursued as part of the detailed engineering:

- Evaluate the benefits of utilizing leased mining equipment or a contract miner
- Further refinement of ore sizing evaluation:
    - ROM fragmentation and impact on crusher feed rate; and
    - Possibility of pre-crushing screening in order to feed ROM ore directly to the HLF.
- Crusher final product ore size and gold recoveries in order to optimize crusher throughput.
- Pilot testing of the various closure strategies examined during the Feasibility Study with the aim to reducing closure costs relating to heap leach pad capping, water management and general maintenance;
- Further refinement of the mine plan and sequence;
- Further refinement of WRSF construction and development through additional geotechnical investigation and permafrost evaluation.

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29 Units of Measure, Abbreviations and Acronyms

actual cubic feet per minute	Acfm
ampere	A
annum (year)	a
bed volumes per hour	BV/h
billion	B
billion tonnes	Bt
billion years ago	bya
billions of years	Ga
British thermal unit	BTU
centimetre	cm
centipoise	cP
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic metres per hour	m^3/h
cubic metres per second	m^3/s
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBa
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	US$
dollar (Canadian)	C$
dry metric tonne	dmt
foot	ft
gallon (US)	gal
gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a

hydraulic conductivity	K
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
litre	L
litres per minute	L/min
litres per second	L/s
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above mean sea level	mamsl
metres below-ground surface	mbgs
metres below sea level	mbsl
metres per minute	m/min
metres per second	m/s
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre m	L
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Normal cubic metres per hour	Nm^3/h
parts per billion	ppb
parts per million	ppm
pascal	Pa
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG

square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
standard cubic feet per minute	Scfm
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
Troy ounce	oz
volt	V
week	wk
weight/weight	w/w
wet metric tonne	wmt

29.1 Abbreviations and Acronyms

abrasion index	Ai
acid rock drainage	ARD
atomic absorption spectroscopy	AAS
Bench Face Angle	BFA
Bond Ball Mill work index	BMWi
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
capital cost allowance	CCA
capital expenditure	CAPEX
carbon dioxide	CO_2
carbon-in-leach	CIL
carbon-in-pulp	CIP
carbon monoxide	CO
Certified Reference Material	CRM
Coefficient of Variation	CV
opper sulphate	$CuSO_4$
crushing work index	CWi
cumulative net cash flow	CNCF
cut-off grade	COG
dead weight tonnage	DWT
drift and fill	DF
electrowinning	EW
engineering, procurement, and construction management	EPCM
fresh air raise	FAR
field electrical centre	FEC
Footwall	FW
Geological Strength Index	GSI
Global Positioning System	GPS
gold	Au
hanging wall	HW
hydrated lime	$Ca(OH)_2$
internal rate of return	IRR
International Standards Organization	ISO

internet protocol	IP
inter-ramp angle	IRA
Lerchs-Grossman	LG
life of mine	LOM
local area network	LAN
metal leaching	ML
Metal Mining Effluent Regulations	MMER
Mine Closure and Reclamation Plan	MCRP
net cash flow	NCF
net present value	NPV
net smelter return	NSR
neutralization potential/acid production	NP/AP
non-potentially acid generating	NPAG
overburden	OVB
oversize	O/S
post pillar cut-and-fill	PPCF
potentially acid generating	PAG
Prefeasibility Study	PFS
Preliminary Economic Assessment	PEA
Qualified Person	QP
quality assurance/quality control	QA/QC
Rock Mass Rating (1989 version)	RMR89
Rock Quality Designation	RQD
semi-autogenous grinding	SAG
sodium cyanide	NaCN
sodium hydroxide	NaOH
sodium metabisulphite	SMBS
specific gravity	SG
sulphur dioxide	SO_2
Tailings Storage Facility	TSF
three-dimensional	3D
total dissolved solids	TDS
total suspended solids	TSS
two dimensional	2D
unconfined compressive strength	UCS
uninterruptible power supply	UPS
variable frequency drive	VFD
Voice over Internet Protocol	VoIP
Volcanic-turbidite series	VTS
waste rock storage area	WRSA
wide-area network	WAN
weak acid dissoluble	WAD
weak acid dissoluble cyanide	CN_{WAD}
work breakdown structure	WBS
Workers Compensation Board	WCB

Appendix A

QP Certificates

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JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

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 Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, (the "Technical Report") prepared for Kaminak 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 Alaska, Wyoming and 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 Coffee Gold Project site on May 22, 2015;

5. I am responsible for Section numbers 1, 2, 3, 18 (except 18.1.2, 18.1.3, 18.1.4, 18.4), 19, 20, 21, 22, 23 (except 23.6.5), 25, 26, 27, 28, 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 prior involvement with the property that is the subject of the Technical Report and was QP for the technical report titled " Preliminary Economic Assessment for the Technical Report, Coffee Project, Yukon Territory, Canada" with an effective date of June 10, 2014 ;

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: January 6, 2016
Signing Date: February 18, 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 Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, (the "Technical Report") prepared for Kaminak 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 Alberta and Northwest Territories. 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 Coffee Gold Project site on May 22, 2015;

6. I am responsible for Section numbers 15 and 16 (except 16.2.4.1 and 16.2.6.6.1) 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 prior involvement with the property that is the subject of the Technical Report and was QP for the technical report titled " Preliminary Economic Assessment for the Technical Report, Coffee Project, Yukon Territory, Canada" with an effective date of June 10, 2014 ;

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: January 6, 2016
Signing Date: February 18, 2016

(original signed and sealed) "Dino Pilotto, P.Eng."

Dino Pilotto, P.Eng.

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CERTIFICATE OF AUTHOR

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 Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, (the "Technical Report") prepared for Kaminak 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 Coffee Gold Project site;
6. I am responsible for Section 17 (except 17.3.4 and 17.4) 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: January 6, 2016
Signing Date: February 18, 2016

Kelly S. McLeod, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Robert Sim, P.Geo, SIM Geological Inc.

I, Robert Sim, P.Geo, do hereby certify that:

1. I am an independent consultant of:

    SIM Geological Inc.
    6810 Cedarbrook Place
    Delta, British Columbia, Canada V4E 3C5

2. I graduated from Lakehead University with an Honours Bachelor of Science (Geology) in 1984.
3. I am a member, in good standing, of the Association of Professional Engineers and Geoscientists of British Columbia, License Number 24076.
4. I have practiced my profession continuously for 32 years and have been involved in mineral exploration, mine site geology and operations, mineral resource and reserve estimations and feasibility studies on numerous underground and open pit base metal and gold deposits in Canada, the United States, Central and South America, Europe, Asia, Africa and Australia.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am a co-author of the technical report titled "NI 43-101 Feasibility Study Technical Report for the Coffee Gold Project, Yukon Territory, Canada", dated February 18, 2016, with an effective date of January 6, 2016 (the "Technical Report") prepared for Kaminak Gold Corp. ("the issuer") and accept professional responsibility for sections 4 through 12, 14 and 24.
7. I visited the Coffee property on four occasions; September 12-14, 2011; on August 28-29, 2012; on May 15-16, 2013; and on September 24, 2014.
8. I have had prior involvement with the property that is the subject of the Technical Report. I was a co-author of previous Technical Reports dated January 10, 2013 and March 12, 2014.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer and related companies applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 18th day of February, 2016.

"original signed and sealed"

Robert Sim, P.Geo

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CERTIFICATE OF AUTHOR

I, Thomas Sharp, P.Eng., do hereby certify that:

1. I am currently employed as Principal Consultant with SRK Consulting (Canada) Inc. with an office at Suite 2200-1066 West Hastings Street, Vancouver, BC, V6E 3X2;
2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, (the "Technical Report") prepared for Kaminak Gold Corp. ("the Issuer");
3. I am a Professional Mining Engineer (P.Eng. #36988) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Mining Engineer in Nunavut and Montana. I am a Member of the Society for Mining, Metallurgy and Exploration.

   I am a graduate of Montana State University and Montana Tech with a B.Sc. and M.Sc. in Biological Sciences (1988 and 1993), M.Sc. in Environmental Engineering (1996) and a Ph.D. in Civil Engineering (1999). I have been involved in mining and have practiced my profession continuously since 1995. I conducted academic research on mine water chemistry and treatment for five years. I was water treatment manager for a major mining company and a member of their team developing life of mine water management strategies and estimating closure liability for operations in the US and South America. I have also worked as a consultant for over thirteen years and have experience preparing water management plans, site water and load balances, water treatment evaluations and water quality models for mine planning, operations and closure in the US, Canada, South America, and Europe.

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. I have not visited the Coffee Gold Project site;
5. I am responsible for Section numbers 18.4 and 25.3.6.5 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: January 6, 2016
Signing Date: February 17, 2016

ORIGINAL SIGNED AND SEALED BY

Thomas Sharp, P.Eng
Principal Consultant

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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 Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, prepared for Kaminak 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 Coffee Project site on May 22, 2015.
5. I am responsible for preparation of sections 16.2.4.1, 16.2.6.6.1, 18.1.2, 18.1.3, 18.1.4 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 prior involvement with the property that is the subject of the Technical Report and was QP for the technical report titled " Preliminary Economic Assessment for the Technical Report, Coffee Project, Yukon Territory, Canada" with an effective date of June 10, 2014;
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: January 6, 2016
Signing Date: February 18, 2016

"Original Signed and Sealed"

Michael E. Levy, P.E, P.G.

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Tucson 520.544.3688

CERTIFICATE OF AUTHOR

I, Mark E. Smith, P.E., do hereby certify that:

1. I am currently the owner and president of RRD International Corp with an office at 759 Eagle Drive, Incline Village, Nevada, 89451.

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, (the "Technical Report") prepared for Kaminak Gold Corp. ("the Issuer");

3. I am a registered civil and geotechnical engineer in California (#CE35469 and #G2082), a registered professional engineer and water rights surveyor in Nevada (#6546 and #701), a registered professional engineer in Idaho, Utah, Texas and South Dakota, and a registered structural engineer in Idaho and Utah. I am a Registered Member of the Society for Mining, Metallurgy & Exploration (#3005800). I hold a Diplomat in Geotechnical Engineering from AGP/ASCE.

4. I graduated with a Bachelor of Engineering (Civil Engineer) from the University of California (Davis) in 1979 and a Masters of Engineering (Civil Engineering) from the University of Nevada (Reno) in 1986.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I have visited the Coffee Gold Project site on May 22 and September 14, 2015;

7. I am responsible for Section numbers 17.3.4 and 17.4 of the Technical Report;

8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

9. I have had no prior involvement with the property that is the subject of the Technical Report.

10. I have read NI 43-101, and the Technical Report prepared in compliance with NI 43-101 and Form 43-101F1;

11. 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: January 6, 2016
Signing Date: February 17, 2016

Mark E. Smith, P.E.

PROFESSIONAL ENGINEER - STATE OF NEVADA
MARK E. SMITH
Exp. 31 Dec 16
CIVIL
No. 6546

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Website: www.kcareno.com e-mail dkappes@kcareno.com

CERTIFICATE OF AUTHOR

I, Daniel W. Kappes, P. Eng., do hereby certify that:

1. I am currently employed as President of Kappes, Cassiday & Associates with offices at 7950 Security Circle, Reno, NV 89506.

2. This certificate applies to the technical report titled "NI 43-101 Feasibility Study Technical Report for the Coffee Gold Project, Yukon Territory, Canada", with an effective date of January 6, 2016, (the "Technical Report") prepared for Kaminak Gold Corp. ("the Issuer");

3. I am a Professional Mining Engineer (Nevada P.Eng. #3223) registered with the State of Nevada, USA.

   I am a graduate of Colorado School of Mines (E.M., 1966) and of Mackay School of Mines (1972, M.S. Mining Engineering). I have been involved in mining since 1966 and have practiced my profession continuously since then. I founded and am President of Kappes, Cassiday & Associates, a 90-person consulting and construction management firm specializing in the precious metals mineral processing industry. As an individual and as a company, I have worked on more than 500 mining projects of which at least 50 have involved significant aspects of financial evaluation, technical design and construction management. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4. I have not visited the Coffee Gold Project site.

5. I am responsible for Section number 13 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 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: January 6, 2016
Signing Date: February 18, 2016

Daniel W. Kappes

Nevada Registered Professional Engineer Nevada #3223

PROFESSIONAL ENGINEER STATE OF NEVADA
DANIEL W. KAPPES
METALLURGICAL
No. 3223

Expires 30 June 2017