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JDS Energy & Mining Inc.

Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada

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Prepared for:

KAMINAK GOLD CORPORATION

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NOTICE

This report was prepared as a 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 are 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.

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Kaminak Gold Corporation (Kaminak) to carry out a preliminary economic assessment (PEA) of the Coffee Project (Coffee project or Coffee) which is a resource development gold exploration project owned by Kaminak Gold Corp. (Kaminak) and located in the White Gold district of west-central Yukon, approximately 130 km south of Dawson City. The project encloses several gold occurrences within an exploration concession covering an area of more than 600 square kilometres.

Four previous technical reports were prepared for the Coffee project pursuant to Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 (collectively, "NI 43-101") and documenting exploration work completed by Kaminak on the Coffee project in 2010, 2011, 2012 and 2013. All technical reports were filed on SEDAR.

This technical report summarizes the results of the Preliminary Economic Assessment (PEA) study and was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

1.2 Property Description and Ownership

The property comprises 3,021 contiguous Yukon quartz mining claims covering an aggregate area of 60,230 hectares (ha) owned by Kaminak. Kaminak has acquired 100% interest in the property, subject to a 2% net smelter royalty to Mr. Shawn Ryan (prospector from Dawson City, YT) half of which can be repurchased by Kaminak.

1.3 Geology and Mineralization

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. The YTT also hosts volcanogenic massive sulphide (VMS) and Mississippi Valley-type (MVT) deposits.

The Coffee project area is underlain by a package of metamorphosed Paleozoic rocks of the Yukon-Tanana terrane 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, 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 (dacite and andesite).

Exploration drilling has led to the discovery of gold mineralization in 19 separate areas of the Coffee project: Supremo T1, Supremo T2, Supremo T3, Supremo T4, Supremo T5, Supremo T7, Sumatra, Latte, Latte North, Latte Extension, Double Double, Arabica, Americano West, Americano, Espresso, Kona, Macchiato, Cappuccino, and Sugar. Gold mineralization is commonly found in narrow to broad gold-bearing locally brecciated structures or dacite dykes 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-As-Sb-S-Au fluid resulting in the formation of arsenian pyrite bearing gold. 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.

1.4 History, Exploration and Drilling

In 2013, the exploration work completed on the Coffee project included:

- Soil geochemical sampling
- Bedrock mapping and sampling
- 62 core boreholes (12,273 m)
- 240 reverse circulation boreholes (43,205 m)

The purpose of the 2013 drilling program was to expand upon previous results, focusing on the Supremo (142 reverse circulation boreholes, 26,339 m; 30 core boreholes, 5,953 m) and Latte and Latte North (60 reverse circulation boreholes, 10,125 m; 19 core boreholes, 4,225 m) zones. In addition, limited programs designed to test the Sumatra and Arabica soil anomalies were completed (38 reverse circulation boreholes, 6,742 m; 13 core boreholes, 2,094 m).

Borehole locations were planned and marked by Kaminak geologists using a handheld GPS device. A compass was used to determine borehole azimuth and inclination. Downhole surveys were completed for all core boreholes using a Reflex EZ-Shot electronic single shot (magnetic) device. All reverse circulation boreholes were surveyed with an Icefield Tools Gyro Shot device.

Collar locations were surveyed following completion by Challenger Geomatics Ltd. of Whitehorse, YT using Real Time Kinematic (RTK) GPS using five control points established and set by Challenger Geomatics.

Samples were placed in sealed bags and shipped by charter plane or barge and then trucked by Kaminak, Small's Expediting or ALS staff to the ALS Minerals preparatory laboratory in Whitehorse prior to analysis in the ALS laboratory in North Vancouver. Each sample was assayed for gold using conventional fire assay procedures on 30-gram charges, and analysed for a suite of trace elements using aqua regia digestion. Samples reporting greater than 0.3 grams per tonne (g/t) Au were subsequently analysed by cold cyanide shake test to measure recoverability by cyanide solution.

1.5 Mineral Processing and Metallurgical Testing

In 2011 and 2012, Inspectorate Exploration & Mining Services Ltd. ("Inspectorate") of Richmond, BC conducted preliminary cyanide leaching tests including bottle roll, carbon in leach and carbon in pulp, and column leach test work.

In 2013, Kaminak engaged Kappes, Cassiday, and Associates (KCA) to undertake comprehensive metallurgical testing on both core and bulk samples from the Coffee deposit.

Core Composite Testing

A total of seven core composites were generated and were identified as Supremo Oxide, Supremo, Upper Transition, Supremo Lower Transition, Latte Oxide, Latte Upper Transition, Latte Lower Transition, and Latte Sulphide. Head analyses for gold for these samples varied from 1.23 g/t to 2.47 g/t.

Portions of material from the Latte Sulphide composite were used to conduct preliminary flotation tests which indicated that between 58% and 72% of the gold was concentrated into a rougher concentrate of 9.8% to 10.5% of the sample weight.

The Latte Sulphide composite was submitted for comminution testing and the results of the comminution test work were a Rod Mill Work Index of 12.73 kWh/tonne and a Ball Mill Work Index of 15.06 kWh/tonne.

Bottle roll leach testing was conducted on a portion of material from each composite and a summary of the gold extractions from the bottle roll leach test work is presented in Table 1-1.

Table 1-1: Summary of Bottle Roll Leach Test Work - Core Composites

Description	Head Average Au (g/t)	Calculated Head Au (g/t)	Extracted Au (%)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
Supremo, Oxide	1.461	1.436	94	1.29	1.50
Supremo, Upper Transition	1.227	1.447	78	2.12	1.00
Supremo, Lower Transition	1.569	1.639	53	1.45	1.00
Latte, Oxide	1.488	1.570	92	1.27	1.50
Latte, Upper Transition	1.479	1.369	51	1.15	1.50
Latte, Lower Transition	1.656	1.462	38	1.57	1.50
Latte, Sulphide	2.469	2.460	13	1.35	1.50

Preliminary agglomeration test work was conducted on portions of crushed material from each composite except the Latte Sulphide composite. All agglomeration tests passed the criteria put forth by KCA. It was determined from this test work that the column leaching would be undertaken without the use of agglomeration.

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 in an enclosed refrigeration unit at a target 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 1-2.

Table 1-2: Summary of Column Leach Test Work – Core Composites

Description	Target p80 Size (mm)	Target Temp. (°C)	Calculated Head Au (g/t)	Extracted Au (g/t)	Extracted Au (%)	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)
Supremo, Oxide	25	4	1.573	1.455	92	0.17	1.51
Supremo, Oxide	12.5	4	1.435	1.343	94	0.28	1.5
Supremo, Oxide	12.5	22	1.547	1.471	95	0.52	1.57
Supremo, Upper Transition	12.5	4	1.488	1.081	73	0.31	1
Supremo, Lower Transition	12.5	4	1.674	0.797	48	0.38	1
Latte, Oxide	25	4	1.622	1.462	90	0.19	1.51
Latte, Oxide	12.5	4	1.54	1.382	90	0.27	1.51
Latte, Upper Transition	12.5	4	1.535	0.717	47	0.46	2.01
Latte, Lower Transition	12.5	4	1.416	0.411	29	0.64	1.51
Latte, Sulphide	12.5	4	2.365	0.126	5	0.46	1.51

For the Supremo Oxide material, three column leach tests were conducted to compare the extraction values of material leached at two particle sizes (80% passing 25 and 12.5 mm) and leach temperatures (4°C and 22°C). A comparison of gold extractions from the 25 and 12.5 mm leached material showed an increase from 92 to 94%, with respect to particle size reduction. A similar comparison of the material leached at 4°C and 22°C, showed a gold extraction increase from 94 to 95%. For the Latte Oxide material, two column leach tests were conducted to compare the extraction values of material leached at two different particle sizes (80% passing 25 and 12.5 mm). A comparison of gold extractions from the 25 and 12.5 mm leached material did not show any obvious gold extraction increase with respect to particle size reduction. Gold extraction percentages were at 90% for both particle sizes.

Categorization of oxidation has in the past been undertaken via visual estimation of the proportion of Oxide and Sulphide. Presently, the Transitional is simply divided into an ‘Upper’ and ‘Lower’ zone based on ≥50% oxidized material and ≤50% oxidized material, respectively. Although the visual estimate of the amount 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 found to be much higher. The poorer recoveries from the Latte Transitional material are felt 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 initial drilling year of 2010 up to and including 2013, have been subjected to a cyanide soluble assay. The difference between the cyanide soluble assay and the original fire assay may be utilized to provide an indication of the gold within the sample that is amenable to cyanide leach. By extension, it also indicates the amount 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.

There is a strong correlation of the cyanide soluble recovery and the actual column leach test recovery which 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.

Bulk Sample Testing

The bulk Supremo Oxide sample was collected from a surface trench across the T3 mineralized structure at 6974250mN. The bulk Latte Oxide sample was collected from a surface trench across the Latte mineralized structure at 583250mE.

Each bulk composite sample was assigned a KCA sample ID and was subsequently utilized for head analyses; head screen analyses with assays by size fraction, bottle roll leach test work, agglomeration test work and column leach test work.

Head analyses for the Latte Oxide varied from 1.61 g/t to 1.5 g/t and the Supremo Oxide varied from 4.08 g/t to 4.37 g/t.

Bottle roll leach testing was conducted on a portion of material from each composite. A summary of the gold extractions from the bottle roll leach test work is presented in Table 1-3.

Table 1-3: Summary of Bottle Roll Leach Test Work – Bulk Samples

KCA Sample No.	Description	Calculated Head Au (g/t)	Extracted Au (%)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
69580	Latte Oxide	1.474	94	0.06	2.5
69581	Supremo Oxide	3.586	96	1.62	2.5

Preliminary agglomeration test work was conducted on portions of crushed material from each sample. All agglomeration tests passed the criteria put forth by KCA.

A total of four column leach tests were conducted utilizing composite sample material crushed to target sizes of 100% passing 175 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 in a walk-in freezer maintained at a target temperature of 4°C. A summary of the column leach test extractions for gold are presented in Table 1-4.

Table 1-4: Summary of Column Leach Tests – Bulk Samples

KCA Sample No.	Description	Crush Size p100 (mm)	Calculated Head Au (g/t)	Extracted Au (%)	Days of Leach	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
69580	Latte Oxide	175	1.28	88	100	0.56	1.01
69580	Latte Oxide	31.5	1.122	92	100	1.08	1.00
69581	Supremo Oxide	175	4.313	85	152	0.91	1.52
69581	Supremo Oxide	31.5	3.438	92	100	0.93	1.51

For the column leach test work, gold extractions ranged from 85% to 92%.

For each sample, a comparison of the column leach test results showed a higher gold extraction percentage for the column leach test conducted utilizing material leached at a finer crush size.

From the head screen versus tail screen analyses, comparisons of the approximate gold extractions from similar size fractions showed minor variations. The small variations indicate that there are no obvious differences in leaching kinetics between tests conducted at 4°C versus those conducted at ambient temperature (22°C).

1.6 Mineral Resource Estimate

The mineral resource estimates for the Coffee project were updated during the period from December 2013 through the end of January 2014 using a geostatistical block modelling approach constrained by gold mineralization wireframes. The model considers information from 961 core and reverse circulation boreholes drilled by Kaminak from 2010 to 2013 (185,000 m). Four individual block models were constructed using MineSight® (v8.20) with limits determined based on the local UTM coordinated system (Nad83 datum, zone 7). Block size was set at 5 by 5 by 2 m at Kona and Double Double and 10 by 5 by 3 m at Latte and Supremo, with block long axes aligned parallel to the strike of the gold mineralization.

The boundaries of the gold mineralization were interpreted by Kaminak from drilling data on vertical sections spaced at 25 to 50 m intervals. These were linked into a series of 3D domains that form the basis in controlling the distribution of gold mineralization in the resource models. Borehole assay 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 estimation. A gold grade was estimated for each model block using ordinary kriging and estimation parameters derived from variography.

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 which, in general, represent decreasing intensity of oxidation with depth below surface.

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

After validation through a combination of visual inspection and statistical evaluations, the block models were classified on the basis of confidence in the geological continuity and distance from informing data. Block model quantities and grade estimates were classified according to the CIM Definition Standards on Mineral Resources and Mineral Reserves (November 2010). Blocks in the Indicated category form relatively continuous zones of mineralization delineated by three or more drill holes on a nominal 25 m pattern. In the main part of the Latte zone, gold mineralization is thicker and more consistent in nature and, as a result, resources in this area can be included in the Indicated category based on drilling on 35 m spacing. Resources are included in the Inferred 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 deposits form relatively continuous, sub-vertical zones of gold mineralization extending from the surface to a depth of several hundred metres. The deposits are amenable to open pit (OP) or underground extraction (or a combination of both). The "reasonable prospects for economic extraction" were tested using floating cone pit shells based on reasonable technical and economic assumptions (for example, site operating costs of C$20 per tonne mined, a pit slope of 45° and gold prices ranging from $1,300/oz to $1,700/oz. These initial evaluations assume 100% mining and metallurgical recoveries). These pit optimization evaluations are used solely for the purpose of testing the "reasonable prospects for economic extraction," and do not represent an attempt to estimate mineral reserves. There are no mineral reserves at the Coffee project. The optimization results are used to assist with the preparation of a Mineral Resource Statement and to select appropriate reporting assumptions.

Analyses of the floating cones show that the majority of the Oxide and Transitional gold mineralization could potentially be amenable to open pit extraction methods as these shells extend to depths of over 200 m below surface in many areas. Studies show that 80% of the oxide and transitional mineral resource is located within 150 m of surface and 94% of these resources are within a maximum depth of 200 m below surface.

Although these studies suggest that some mineralized areas may not be economically viable, this represents a relatively small proportion of the resource. Mineralization at Coffee showing reasonable prospects for economic viability has been included in the estimation of mineral resources.

The Mineral Resource Statement for the Coffee project is presented in Table 1-5. The Mineral Resource Statement is reported at two cut-off grades. Oxide and Transition Mineral Resources are reported at a cut-off grade of 0.5 g/t gold while Sulphide Mineral Resources are reported at a cut-off grade of 1.0 g/t gold reflecting the generally greater depths and differing metallurgical characteristics of this material. The updated mineral resource estimate consists of an Indicated Resource of 14 million tonnes (Mt) grading 1.56 g/t Au for 719,000 oz Au, and an Inferred Resource of 79 Mt at 1.36 g/t Au for 3,455,000 ounces of gold.

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 1-5: Estimate of Mineral Resources for the Coffee Project*

Area	Oxide			Upper Transition			Lower Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Indicated															
Supremo	2,967	2.13	203	847	1.62	44	183	1.78	11	3,997	2.01	258	0	0	0
Latte	5,588	1.54	277	2,773	1.22	109	1,958	1.16	73	10,319	1.38	459	42	1.52	2
Combined	**8,555**	**1.75**	**480**	**3,619**	**1.32**	**153**	**2,141**	**1.21**	**83**	**14,316**	**1.56**	**717**	**42**	**1.52**	**2**
Inferred															
Supremo	42,003	1.21	1,636	9,001	1.3	377	2,579	1.41	117	53,583	1.24	2,129	564	1.47	27
Latte	5,673	1.23	224	3,518	1.46	166	3,878	1.43	179	13,070	1.35	569	4,529	1.95	284
Dbl. Dbl.	1,772	2.99	170	1,974	1.81	115	206	1.49	10	3,951	2.32	295	189	2.21	13
Kona	989	1.48	47	1,473	1.2	57	0	0	0	2,462	1.32	104	244	1.57	12
Combined	**50,437**	**1.28**	**2,078**	**15,967**	**1.39**	**714**	**6,662**	**1.43**	**306**	**73,066**	**1.32**	**3,098**	**5,525**	**1.89**	**336**

*Oxide and Transition mineral resources reported at a cut-off grade of 0.5 g/t gold. Sulphide mineral resources reported at a cut-off grade of 1.0 g/t gold. Cut-off grades based on a gold price of US$1,300 per ounce, site operation costs of US$20.00 per tonne mined and assume 100 percent mining and metallurgical recovery. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have a demonstrated economic viability.

1.7 Mineral Reserve Estimate

Indicated and Inferred resources were used in the life-of-mine (LOM) plan and Inferred material represents 80% of the material planned for processing. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study (PFS) or a feasibility study (FS) of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Coffee project.

1.8 Mining

The Coffee deposit is amenable to development as an open pit mine. Mining of the deposit is planned to produce a total of 53.4 Mt of heap leach feed and 212.4 Mt of waste (4.0:1 overall strip ratio) over a 12-year mine production life (includes two years of pre-production). The current LOM plan focuses on achieving consistent heap leach production rates, and mining of higher grade material early in the production schedule, as well as balancing grade and strip ratios.

The projected mining method, potential production profile and plan, and mine plan are conceptual in nature and additional technical studies will need to be completed in order to fully assess their viability. There is no certainty that a potential mine will be realized or that a production decision will be made. A mine production decision that is made without a FS carries additional potential risks, which include, but are not limited to, the inclusion of Inferred mineral resources. Inferred resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mine design and mining schedules, metallurgical flow sheets, and process plant designs may require additional detailed work and economic analysis, and internal studies to ensure satisfactory operational conditions and decisions regarding future targeted production.

1.8.1 Open Pit Mine Plan and Phasing

Figure 1-1 illustrates the proposed overall site layout for the Coffee project, including the OP, waste rock facilities, heap leach facilities, and proposed plant site locations.

The mine design process for the deposit commenced with the development of OP 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 Table1-6).

DWG. NO. | REFERENCE DRAWINGS

PROJECT | PROCESS | CIVIL | MECH. | STRUCT. | PIPING | SERVICES | ELECT. | INSTR. | NO | DESCRIPTION | BY | DATE
ISSUE

PROJECT | PROCESS | CIVIL | MECH. | STRUCT. | PIPING | SERVICES | ELECT. | INSTR. | NO | DESCRIPTION | BY | DATE
REVISIONS

SECTION:
SCALE:
DESIGN. BY J.D.S. May 2 2014
DRAWN BY: B. Wong May 2 2014
CHECK. BY
APP. BY:

JDS Energy & Mining Inc.

Kaminak Gold Corp.

COFFEE GOLD PROJECT
PROJECT LOCATION MAP
PLAN

FILENAME: | PROJECT NUMBER | DRAWING NUMBER 100-10-001 | REV.

Table 1-6: Mine Planning Optimization Input Parameters

Parameter	Unit	Value
Revenue, Smelting & Refining		
Gold price	US$/oz Au	1,250
Exchange Rate	US$:C$	0.95
Payable metal	%	100
TC/RC/Transport	C$/oz Au	7.50
Royalty @ 1% (assumed after capital buyout of $2M before Pre-production)	C$/oz Au	12.50
Net gold value per ounce	C$/oz	1,296
Net gold value per gram	C$/g	41.66
OPEX Estimates		
OP Waste Mining Cost	C$/t mined	2.25
OP MHL Feed Mining Cost	C$/t mined	2.25
Processing Cost for oxide in all deposits	C$/t milled	7.00
Processing Cost for transition in Supremo, DD & Kona	C$/t milled	7.25
Processing Cost for transition in Latte	C$/t milled	7.50
G&A	C$/t milled	4.00
Total OPEX (ex. Mining) –for oxide in all deposits	C$/t milled	11.00
Total OPEX (ex. Mining) – for transition in Supremo, DD & Kona	C$/t milled	11.25
Total OPEX (ex. Mining) – for transition in Latte	C$/t milled	11.50
Recovery and Dilution		
Gold Recovery		
Leach Recovery in oxide for Supremo, DD & Kona	%	90
Leach Recovery in oxide for Latte	%	87
Leach Recovery in upper transition for Supremo, DD & Kona	%	70
Leach Recovery in upper transition for Latte	%	44
Leach Recovery in lower transition for Supremo & DD	%	45
Leach Recovery in lower transition for Latte	%	26
External Mining Dilution	%	5
Mining Recovery	%	98
Other		
Overall Pit Slope Angles = interramp slopes flattened for ramp allowance		
Supremo	degrees	45
Latte	degrees	40 - 44
Double Double	degrees	37
Kona	degrees	40
Leach Production Rate (250 operating days per year x 20,000 tpd)	Mtpa	5.0

CAE NPV Scheduler (NPVS) software, along with the mineral inventory block models produced by SIM Geological Inc., were used to determine the optimal mining shells with the assumed overall slope angles presented above. Preliminary phases for the Supremo, Latte, Double Double and Kona deposits were selected and preliminary mine planning and scheduling was then conducted on these selected optimal shells. The PEA life of mine (LOM) proposed mineable resources for the deposit are presented in Table1-7 (for both Oxide and Upper Transition material).

Both Indicated and Inferred resources were used in the LOM plan of which Inferred resources represent 80% of the material planned to be processed. Mineral resources that are not mineral reserves have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves.

Table 1-7: Proposed Mining Plan

Description	Unit	Value
Mine production life	yr	12
Heap Leach diluted feed	Mt	53.4
Diluted gold grade (head grade)	g/t	1.23
Contained gold	koz	2,111
Waste	Mt	212.4
Total material	**Mt**	**265.8**
Strip ratio	t:t	4

1.8.2 Mine Schedule

The various pit shells for the Coffee 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 Coffee deposits are most economical when the OP phases are mined concurrently. The OP mining is envisioned to be undertaken by Kaminak, the project owner.

The heap leach throughput was planned at a net yearly production of approximately 5.0 Mt. Pre-production stripping and heap leach pad construction was planned to occur in Years -2 and -1, with the commencement of leach processing occurring late in Year -1. Table 1-8 shows a summary of LOM total material movement by year for both the mine and the heap leach facility.

Table 1-8: LOM Production Schedule

				Year												
Area	**Description**	**Units**	**TOTAL**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Supremo	Oxide HL Feed	ktonnes	**39,000**	-	-	-	763	5,000	5,000	5,000	5,000	5,000	5,000	5,000	4,000	-
	Oxide Mined Gold Grade	g/t	**1.13**	-	-	-	0.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.37	-
	Transition HL Feed	ktonnes	**1,000**	-	-	-	-	42	19	-	32	332	96	95	574	-
	Transition Mined Gold Grade	g/t	**1.64**	-	-	-	-	1.01	1.25	-	1.34	1.04	2.61	3.44	1.60	-
	Total HL Feed	ktonnes	**40,000**	-	-	-	763	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	-
	Total Mined Gold Grade	g/t	**1.14**	-	-	-	0.69	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.40	-
	Waste Mined	ktonnes	**171,000**	-	-	-	5,000	19,000	25,000	22,000	22,000	20,000	20,000	21,000	17,000	-
	Strip Ratio (t:t)	t:t	**4.24**	-	-	-	6.71	3.88	5.00	4.40	4.36	3.90	4.02	4.27	3.68	-
Latte	Oxide HL Feed	ktonnes	**11,000**	852	2,000	5,000	3,000	-	-	-	-	-	-	-	-	-
	Oxide Mined Gold Grade	g/t	**1.28**	1.09	1.13	1.42	1.23	-	-	-	-	-	-	-	-	-
	Transition HL Feed	ktonnes	**725**	-	17	259	450	-	-	-	-	-	-	-	-	-
	Transition Mined Gold Grade	g/t	**1.83**	-	1.24	1.72	1.92	-	-	-	-	-	-	-	-	-
	Total HL Feed	ktonnes	**11,000**	852	2,400	5,000	3,000	-	-	-	-	-	-	-	-	-
	Total Mined Gold Grade	g/t	**1.32**	1.09	1.13	1.44	1.33	-	-	-	-	-	-	-	-	-
	Waste Mined	ktonnes	**24,000**	4,000	7,000	9,000	5,000	-	-	-	-	-	-	-	-	-
	Strip Ratio (t:t)	t:t	**2.16**	4.40	3.03	1.74	1.53	-	-	-	-	-	-	-	-	-
Double Double	Oxide HL Feed	ktonnes	**1,000**	-	-	11	1,200	-	-	-	-	-	-	-	-	-
	Oxide Mined Gold Grade	g/t	**3.32**	-	-	5.05	3.30	-	-	-	-	-	-	-	-	-
	Transition HL Feed	ktonnes	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Transition Mined Gold Grade	g/t	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total HL Feed	ktonnes	**1,000**	-	-	11	1,200	-	-	-	-	-	-	-	-	-
	Total Mined Gold Grade	g/t	**3.32**	-	-	5.05	3.30	-	-	-	-	-	-	-	-	-
	Waste Mined	ktonnes	**14,000**	-	-	2,000	11,000	-	-	-	-	-	-	-	-	-
	Strip Ratio (t:t)	t:t	**11.27**	-	-	211.67	9.49	-	-	-	-	-	-	-	-	-
Kona	Oxide HL Feed	ktonnes	**730**	-	-	-	-	-	-	-	-	-	-	-	730	-
	Oxide Mined Gold Grade	g/t	**1.22**	-	-	-	-	-	-	-	-	-	-	-	1.22	-
	Transition HL Feed	ktonnes	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Transition Mined Gold Grade	g/t	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total HL Feed	ktonnes	**730**	-	-	-	-	-	-	-	-	-	-	-	730	-
	Total Mined Gold Grade	g/t	**1.22**	-	-	-	-	-	-	-	-	-	-	-	1.22	-
	Waste Mined	ktonnes	**3,700**	-	-	-	-	-	-	-	-	-	-	-	4,000	-
	Strip Ratio (t:t)	t:t	**5.07**	-	-	-	-	-	-	-	-	-	-	-	5.07	-
Total Mine	Oxide HL Feed	ktonnes	**52,000**	852	2,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	-
	Oxide Mined Gold Grade	g/t	**1.21**	1.09	1.13	1.43	1.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.34	-
	Transition HL Feed	ktonnes	**2,000**	-	17	259	450	42	19	-	32	332	96	95	574	-
	Transition Mined Gold Grade	g/t	**1.71**	-	1.24	1.72	1.92	1.01	1.25	-	1.34	1.04	2.61	3.44	1.60	-
	Total HL Feed	ktonnes	**53,000**	852	2,400	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	-
	Total Mined Gold Grade	g/t	**1.23**	1.09	1.13	1.44	1.71	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.37	-
	Waste Mined	ktonnes	**212,000**	4,000	7,000	11,000	21,000	19,000	25,000	22,000	22,000	19,000	20,000	21,000	20,000	-
	Strip Ratio (t:t)	t:t	**3.97**	4.40	3.03	2.19	4.24	3.88	5.00	4.40	4.36	3.90	4.02	4.27	3.87	-
	Total Material Mined	ktonnes	**266,000**	5,000	10.000	16,000	26,000	24,000	30,000	27,000	27,000	24,000	25,000	26,000	26,000	-

1.9 Waste Management

Waste rock from the open pits at the Coffee project is planned to be deposited in various engineered waste dumps adjacent to the proposed open pits. In addition, the pre-stripping waste material from Year -2 of the LOM schedule will be used to develop the heap leach embankment requirements.

1.10 Recovery Methods

The process flowsheet includes a three-stage crushing plant followed by a heap leach operation. Gold is extracted by an Adsorption-Desorption-Recovery (ADR) carbon plant. The process flowsheet is based on an HL processing rate of 5.0 million dry tonnes per year. The process plant will be located near the HLF to minimize the pumping and pipeline requirements for pregnant and barren solutions.

Run-of-mine mineralized material is trucked from the mine and normally dumped directly into primary jaw crusher. A small stockpile of ROM HL feed will be available to allow mining operations to continue if the primary jaw crusher is not operating. Two additional stages of crushing, secondary and tertiary in closed-circuit, are used to reduce the product size to nominal 80% passing 12.5 mm.

The design particle crush size is minus 16 mm (80% passing 12.5 mm), however, mineralized material types found to leach adequately at coarser sizes could bypass the tertiary crushing circuit. Based on experience gained during actual operations, the crush size for each material type may be modified as conditions permit.

A crushed HL feed stockpile will hold approximately 1,500,000 tonnes and will use grasshopper-type portable conveyors and a self-propelled radial stacker equipped with a stinger conveyor to build the stockpile. The stockpile is required because the design criteria only allows the heap leach facility (HLF) to be loaded for 250 days per year, hereby avoiding heap loading during the coldest period of the year.

Final crushed mineralized material is reclaimed by a front-end loader and portable conveyors and fed onto a series of grasshopper-type portable conveyors for heap construction.

The HLF contains the following facilities:

- Valley fill heap leach pad complete with liner system
- In-heap solution pond for storage of pregnant solution
- Pregnant solution recovery wells and vertical turbine pumps for collection and transport of pregnant leach solution
- Piping systems for the collection and distribution of both barren and pregnant solution
- Heating of a portion of barren solution to maintain thermal balance

The liner system of the in-heap solution storage area will have a double liner of linear low-density polyethylene (LLDPE), separated by a synthetic geonet mesh over a compacted sub-base of local material. The liner system of the HLF not used for solution storage will be a single LLDPE liner over an engineered compacted sub-base from local material.

Barren leach solution will be applied to the HL by a drip emitter irrigation system. The pregnant solution will flow to the in-heap solution pond and will then be transferred to the ADR recovery plant using wells and vertical turbine pumps.

A system of diversion ditches collects the drainage runoff from uphill catchment areas around the HLF. An emergency storm water pond located below the HLF is available in case of an extreme storm event. If this occurs, the excess solution will be routed to the storm water pond and later used to supplement the supply make-up water.

The pregnant solution will be pumped to the carbon adsorption circuit. The carbon adsorption circuit consists of a series of five cascading carbon columns. The barren solution that will discharge from the final carbon column drains to barren solution tanks. Cyanide solution, liquid caustic and antiscalent are added to the barren solution as needed. Barren solution is then pumped back to the leach pad. The loaded carbon from the first carbon column is advanced to the desorption circuit. The loaded carbon will be acid washed and sent to the carbon strip vessel.

The pregnant solution that flows out of the top of the strip vessel will flow to an electrowinning cell. At the conclusion of the strip cycle, the stripped carbon will be thermally regenerated in the carbon reactivation kiln to maintain its activity.

Gold will be plated onto knitted-mesh steel wool cathodes in the electrowinning cell. The gold-bearing sludge and steel wool will be retorted to remove any mercury. The retort residue will be mixed with fluxes and then smelted to produce gold doré and slag. The doré will be transported to a refiner for further purification. Slag is processed to remove prills for re-melting in the furnace.

Several considerations to adequately mitigate the Yukon climate have been included in the general design criteria:

- All conveyors will be fitted with covers
- A valley fill heap configuration with an in-heap solution pond and temperature monitoring for pregnant leach solution
- Burying the drip emitter lines
- Ability to heat a portion of the barren solution
- Heat tracing of and insulation of the barren solution tank and pipelines
- Dedicated stand-by generators for backup power supply to pregnant and barren solution pumps
- All crushing and process buildings will be pre-engineered steel structures with insulated steel roofs and walls.

All power for the project will be supplied by diesel generators. Estimated power required for the process plant is 3.9 megawatts (MW). Water will be supplied from wells developed near the plant facilities and will be adequate to supply all process, camp and mine facilities.

An assay and metallurgical laboratory facility is 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.

The estimates for ultimate gold recovery for various mineralized material types were based on the results of laboratory column leach tests conducted by KCA. For reference purposes the distribution of mined material the final mine plan is presented in Table 1-9.

Table 1-9: Distribution of Material Types in Final Mine Plan

Material Type	Tonnage Mined	Distribution (%)
Supremo Oxide	39,100,000	73
Supremo Upper Transition	1,200,000	2
Latte Oxide	10,500,000	20
Latte Upper Transition	700,000	1
Double Double Oxide	1,200,000	2
Kona Oxide	700,000	1
Total	**53,400,000**	**100.0**

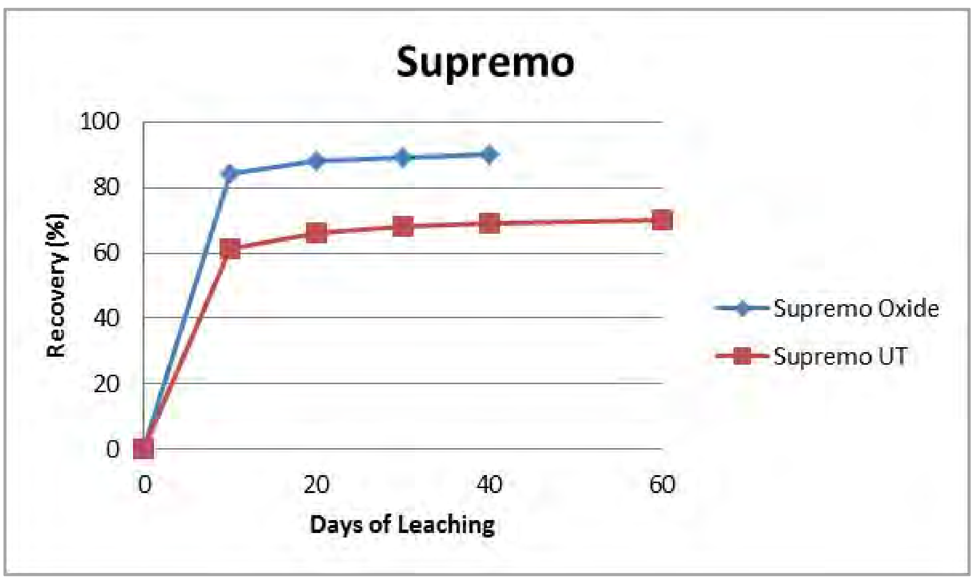
A summary of the ultimate recoveries and reagent consumptions used for each area is shown in Table 1-10. The recoveries are based on actual test results reduced by 3%.

Table 1-10: Ultimate Gold Recovery and Reagent Consumption Used in the Production Model for Each Area

Sample	Ultimate Au Recovery (%)	Reagent Consumptions	
		NaCN (kg/t)	CaO (kg/t)
Supremo Oxide	90	0.20	1.50
Supremo Upper Transition	70	0.20	1.50
Latte Oxide	88	0.20	1.50
Latte Upper Transition	44	0.20	1.50
Double Double Oxide	90	0.20	1.50
Kona Oxide	90	0.20	1.50

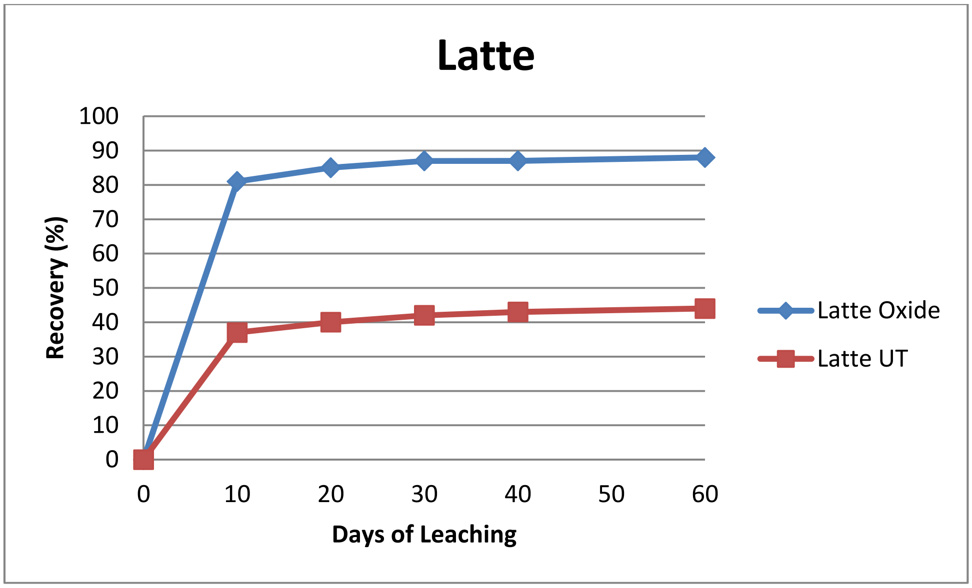
Actual laboratory data of gold recovery and leach time was used to generate the leach profiles. The leach time actually experienced in the laboratory was doubled to generate the leach profiles for both the Supremo and Latte mine areas presented below in Figures 1-2 and 1-3.

Figure 1-2: Supremo Leaching Profiles



As an example in Figure 1-2 above a leach time of 40 days indicates an ultimate 90% recovery for Supremo Oxide. The actual laboratory leach time to achieve this 90% recovery was only 20 days. A safety factor of 100% has been added to the laboratory leach time.

Figure 1-3: **Latte Leaching Profiles**



All Coffee project mine areas reached ultimate extraction in 60 days or less. This rapid leach cycle will help to reduce the inventory of ultimate recoverable gold contained in the heap leach pad during normal operations, but an inventory of recoverable gold will be present. The inventory of recoverable gold in the process circuit will mainly be attributed to the unleached material placed on the leach pad as cover for the drip emitter system, estimated be about 550,000 t of HL feed and contain 22,000 oz of gold.

Although the leach cycle should minimize the amount of gold from partially leached material an allowance of 5,000 oz has been made. Gold inventories contained in process solutions and on carbon are estimated to contain 3,000 oz. A process inventory of 30,000 oz of gold is expected and will build over a three year period at 10,000 oz per year and will be recovered at the end of the mine life.

Based on the final mine plan, a gold production model was developed. The gold production model considers the different mining areas and their ultimate gold extractions for each area. Since the HLF will not be loaded for 100 days per year, a stockpile of crushed ore will be required. Usually the stockpile will be full at the start of leaching in the spring. Since the heap loading equipment has a higher capacity than the crushing circuit, the stockpile will be depleted by the time heap loading operations are curtailed in the latter part of the year. The stockpile will then be filled over the winter months.

A mixture of HL feed material at varying recoveries and grades will be contained in the stockpile. The stockpile inventory is calculated by keeping a running total of tonnage and recoverable ounces of gold. As the stockpile is reclaimed these tonnes and recoverable ounces are then loaded onto the HLF.

Table 1-11 presents the annual gold production. In Year -1, the calculated gold production of 63,000 oz will be reduced by 10,000 oz of process inventory. Of the produced 53,000 oz, only 50% or 26,000 oz are credited as sold. The remaining 27,000 ounces are carried over to Year 1 production to allow for plant start-up.

Table 1-11: Annual Gold Production

	Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	TOTAL
Total Mined and Crushed To Crushed HL feed Stockpile	Oxide HL Feed	852,000	2,383,000	4,741,000	4,538,000	4,958,000	4,981,000	5,001,000	4,969,000	4,660,000	4,878,000	4,906,000	4,668,000	-	**51,534,000**
	Oxide Grade	1.09	1.13	1.43	1.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.34	-	**1.21**
	Oxide Oz	30,000	86,000	218,000	247,000	130,000	211,000	189,000	173,000	149,000	166,000	205,000	202,000	-	**2,006,000**
	Oxide Recovered Oz	26,000	76,000	192,000	220,000	117,000	190,000	170,000	155,000	134,000	150,000	185,000	181,000	-	**1,797,000**
	Transition HL Feed	-	17,000	259,000	450,000	42,000	19,000	-	32,000	332,000	96,000	95,000	574,000	-	**1,914,000**
	Transition Grade	-	1.24	1.72	1.92	1.01	1.25	-	1.34	1.04	2.61	3.44	1.60	-	**1.71**
	Transition Oz	-	664	14,000	28,000	1,371	774	-	1,000	11,000	8,000	10,000	30,000	-	**105,000**
	Transition Recovered Oz	-	292	6,300	12,000	960	542	-	951	8,000	6,000	7,000	21,000	-	**63,00**
	Total Tonnes	852,000	2,400,000	4,999,000	4,987,000	5,587,000	5,587,000	5,001,000	5,001,000	4,992,000	4,973,000	5,000,000	5,242,000	-	**53,448,000**
	Total Grade	1.09	1.13	1.44	1.71	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.37	-	**1.23**
	Total Recovered Oz	26,000	76,000	198,000	232,000	118,000	191,000	170,000	156,000	142,000	155,000	192,000	202,000	-	**1,859,000**
Beginning Stockpile Inventory	Total Tonnes	852,000	3,252,000	6,251,000	5,587,000	5,587,000	5,600,000	5,588,000	5,589,000	5,581,000	5,554,000	5,554,000	5,797,000	797,000	**-**
	Total Recovered Oz	26,000	103,000	237,000	255,000	145,000	206,000	192,000	177,000	161,000	172,000	209,000	223,000	31,000	**-**
From Crusher Stockpile to Heap Leach	Total Tonnes	-	2,000,000	5,651,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	797,000	**53,448,000**
	Total Recovered Oz	-	63,000	215,000	228,000	130,000	184,000	172,000	158,000	144,000	155,000	188,000	192,000	31,000	**1,859,000**
Closing Stockpile Inventory	Total Tonnes	852,000	1,252,000	600,000	587,000	587,000	587,000	588,000	589,000	581,000	554,000	554,000	797,000	-	**7,276,000**
	Total Recoverable Oz	26,000	39,000	23,000	27,000	15,000	22,000	20,000	19,000	17,000	17,000	21,000	31,000	-	**250,000**
Gold Production	Gold in Heap Inventory		10,000	10,000	10,000	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	(30,000)	
	Recovered Oz	-	53,000	205,000	218,000	130,000	184,000	172,000	158,000	144,000	155,000	188,000	192,000	61,000	**1,859,000**
Plant Start Up Allowances	Ounces		27,000	(27,000)											
Gold Sales	Ounces		**27,000**	**231,000**	**218,000**	**130,000**	**184,000**	**172,000**	**158,000**	**143,853**	**155,000**	**188,000**	**192,000**	**61,000**	**1,859,000**

1.11 Project Infrastructure

The project envisions the construction of the followings key infrastructure items:

- Approximately 250 km all-season access road from Carmacks to the project site
- Approximately 7 km of new on-site access roads for light vehicles to by-pass the active mining areas
- New airstrip
- Primary, secondary and tertiary crushing systems
- 1.5 M t crushed HL feed stockpile
- Carbon adsorption plant and gold refinery
- Truck shop, warehouse and camp
- Fresh water supply developed from groundwater
- Bulk explosives storage and magazines
- Power plant and bulk fuel storage
- Potable, fire and sewage water systems

1.12 Environment Assessment and Permitting

The Project will be subject to an environmental assessment (EA) under the Yukon Environmental and Socio-economic Assessment Act (YESAA), administered by the Yukon Environmental and Socio-economic Assessment Board (YESAB). The Project will require an Executive Committee screening.

Baseline environmental studies were initiated in 2010. In 2014, a comprehensive baseline program is underway.

1.12.1 Social Considerations

The Project is located within the traditional territory of the Tr’ondëk Hwëch’in First Nation. A portion of the claim block is located within the overlap area with Selkirk First Nation. The proposed road alignment is located within the traditional territories of Little Salmon Carmacks First Nation, Selkirk First Nation and Tr’ondëk Hwëch’in First Nation. The Project and proposed road alignment are located within White River First Nation’s asserted traditional territory.

Kaminak have an Exploration Communication and Cooperation Agreement with Tr'ondëk Hwëch'in. Kaminak is developing a community and Aboriginal engagement program for 2014.

1.13 Capital Costs

The capital cost estimate (CAPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a preliminary study.

The CAPEX estimate includes the costs required to develop, sustain, and close the operation for the planned 11-year mine life, which includes a 2 year construction period. Closure and reclamation takes place in Years 11-15.

The CAPEX estimate is shown in Table 1-12. The sustaining capital is carried over operating years 1 through 11, and closure costs are projected over Years 11 to 15.

Table 1-12: LOM Capital Costs

Capital Cost	Pre-Production (C$M)	Sustaining/ Closure (C$M)	Total (C$M)
Capitalized Mining	50.3	0.0	50.3
Pre-Production Operating Costs	16.7	0.0	16.7
Site	57.1	4.8	61.9
Mining Equipment	46.3	64.5	110.8
Leach Facility	56.1	17.2	73.3
Camp	10.3	0.0	10.3
Indirects	37.0	0.0	37.0
Closure	0.0	40.0	40.0
Subtotal	**273.8**	**126.5**	**400.3**
Contingency	31.0	19.0	50.0
Total Capital Costs	**304.8**	**145.5**	**450.3**

1.14 Operating Costs

The operating cost estimate (OPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a preliminary study.

The operating cost estimate in this study includes the costs to mine, handle and transport HL feed to the mill, mill and process the material to doré and the associated on-site general and administrative expenses (G&A). These items total the mine operating costs. The total life-of-mine costs are summarized in Table 1-13.

Table 1-13: LOM Operating Costs (excluding costs capitalized in pre-production)

Operating Cost	$/tonne HL	LOM C$M
Mining & Rehandle ($2.29/t mined)	11.86	610.0
Processing	6.67	343.2
G&A	4.00	205.8
Total Operating Costs	**22.53**	**1,159.0**

1.15 Economic Analysis

An economic model was developed to estimate annual cash flows and sensitivities of the project. All costs, metal prices and economic results are reported in Canadian dollars (CDN, 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 are no reliable long-term predictive tools.

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 $46.0M which includes a 15% contingency
- Nominal 2014 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.)
- 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 1-14.

Sensitivities to metal prices, head grade, OPEX and CAPEX were conducted by adjusting each variable up and down 15% independently of each other. 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 1-15.

Table 1-14: Economic Results

Parameter	Unit	Value
Au Price	US $/oz	1,250
Exchange Rate	US:CDN	0.95
Production		
Mine Life	Yrs	11.1
Au Produced	LOM k oz	1,859
	Avg k oz/yr	167
LOM NSR (after royalties)	$M	2,405.3
Operating Costs	LOM $M	1,246.3
	$/t milled	22.53
Capital Costs		
Pre-Production	$M	273.8
Sustaining & Closure	$M	126.5
Subtotal	$M	400.3
Contingency 15%	$M	50.0
Total Capital Costs	**$M**	**450.3**
Operating Cash Flow	$M	1,246.3
	$M/yr	112.8
Cash Cost	$/oz	645.43
Economic Results		
After-Tax Free Cash Flow	$M	515.2
	Avg $M/yr	46.6
Discount Rate	%	5
Pre-Tax NPV$_{5\%}$	**$M**	**522.4**
Pre-Tax IRR	**%**	**32.8**
Pre-Tax Payback	**Yrs**	**1.8**
After-Tax NPV$_{5\%}$	**$M**	**330.4**
After-Tax IRR	**%**	**26.2**
After-Tax Payback	**Yrs**	**2.0**

Table 1-15: Economic Sensitivities

After-Tax NPV$_{5\%}$ ($M)			
Variable	**-15%**	**100%**	**+15%**
Metal Prices	158.9	330.4	497.8
Head Grade	158.4	330.4	498.1
Operating Costs	410.2	330.4	249.6
Capital Costs	391.1	330.4	269.8

1.16 Conclusions

It is the conclusion of the QPs that the PEA 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 in the PEA.

The Coffee 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 pre-feasibility or feasibility stage.

There is also a likelihood 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 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.

To date, the QPs are not aware of any fatal flaws for the Project.

1.17 Recommendations

It is recommended that the project proceed to the feasibility study stage in line with Kaminak's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Kaminak's project development plans.

It is estimated that a feasibility study and supporting field work would cost approximately $25.5 million. A breakdown of the key components of the next study phase is as follows in Table 1-16.

Table 1-16 Cost Estimate to Advance Project to Feasibility Stage

Component	Estimated Cost (M$)	Comment
Infill drilling	14.0	Conversion of inferred resources to indicated within and immediately adjacent to the pit shells. Drilling will include holes for combined resource, geotech and hydrogeology purposes plus additional 4 geotechnical holes
Metallurgical testing	0.5	Variability test work and crush size trade off study
Condemnation drilling	3.0	Drilling under waste dumps, HLF & infrastructure to ensure no sterilization of resources
Geochemistry	0.5	ABA accounting tests and humidity cell testing to determine acid generating potential of all rock units and mitigation plans
Geotechnical/ Hydrology/Hydrogeology	1.0	Mine and surface facilities geotechnical investigations (logging, test pitting, sampling, lab tests, etc.)
Engineering	3.5	FS-level mine, infrastructure & process design, cost estimation, scheduling & economic analysis
Environment	3.0	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**25.5**	Excludes corporate overheads and future permitting activities

2.0 INTRODUCTION

2.1 Basis of Technical Report

This Technical Report was compiled by JDS Energy & Mining Inc. (JDS) for Kaminak Gold Corporation (Kaminak). This technical report summarizes the results of the preliminary economic assessment (PEA) study and was prepared following the guidelines of 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 the Consultants, some of which are associated or affiliated with Kaminak. The scope of work for each company is listed below, and combined, makes up the total Project scope.

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

- Compile the technical report which includes the data and information provided by other consulting companies
- Mine planning
- Conduct optimal pit design and production schedule
- Select mining equipment
- Establish potentially mineable resources
- Design required site infrastructure, identify proper sites, plant facilities and other ancillary facilities
- Estimate OPEX and CAPEX for the Project
- Prepare a financial model and conduct an economic evaluation including sensitivity and Project risk analysis
- Interpret the results and make conclusions that lead to recommendations to improve value, reduce risks.

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

- PEA-level geotechnical assessment and estimate of appropriate overall pit slope angles.

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

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

Fred Lightner, P.E., Director of Mine Development (Kaminak) scope of work included:

- Implement and supervise the metallurgical testing program
- Develop a conceptual flowsheet, specifications and selection of leach process equipment
- Establish recovery values based on metallurgical testing results
- Design heap leach processing to realize the predicted recoveries.

Allison Rippin Armstrong, Director of Lands and Environment (Kaminak) scope of work included:

- Review environmental and other permit requirements
- Summarize environmental results and concerns.

2.3 Qualified Person Responsibilities and Site Inspections

The Qualified Persons (QPs) preparing this technical 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 with the exception of Fred Lightner and Allison Rippin Armstrong, who are subject to management or consulting contracts with Kaminak. The QPs are not insiders, associates, or affiliates of Kaminak with the exception of Mr. Lightner and Ms. Rippin Armstrong, who are subject to the terms of a consulting contract and an employment agreement, respectively, with Kaminak. The results of this technical 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

QP	Company	Report Section(s) of Responsibility
Mike Makarenko, P.Eng.	JDS	1, 2, 3, 15, 21, 24, 28
Gord Doerksen, P.Eng.	JDS	19, 22, 25, 26, 27
Scot Klingmann, P. Eng.	JDS	18
Dino Pilotto, P.Eng.	JDS	16 (except 16.3)
Mike Levy, P.E.	SRK	16.3
Robert Sim, P. Geo.	SIM Geological Inc.	4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23
Fred Lightner, P.E.	Kaminak	13, 17, 20

QP site visits were conducted as follows:

- Mike Makarenko visited the project site September 29-October 1, 2013
- Gord Doerksen has not visited the project site
- Scot Klingmann has not visited the project site
- Dino Pilotto has not visited the project site
- Mike Levy has not visited the project site but relied on observations and core photos from Cody Bramwell of SRK; Cody visited the site from July 29 until August 1, 2013.
- Robert Sim visited the site several times including September 12-14, 2011, August 28-29, 2012 and May 15-16, 2013
- Fred Lightner visited the project site August 27 – 28, 2013 and September 29-October 1, 2013

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 directly taken from the most recent historical technical report titled “Mineral Resource Evaluation, Coffee Gold Project, Yukon Territory, Canada” dated March 12, 2014 prepared by SIM Geological Inc. and Kappes, Cassiday & Associates.

All tables and figures are sourced from JDS, unless otherwise indicated.

2.5 Units, Currency and Rounding

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

3.0 RELIANCE ON OTHER EXPERTS

The QP's 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:

- Wentworth Taylor, an Independent CA, for taxation information
- Allison Rippin Armstrong – Kaminak's Vice President Lands and Development, for environmental, permitting and First Nation information

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.

4.0 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 City (Figure 4-1). The project comprises 3,021 contiguous claims covering an aggregate area of approximately 60,230 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 0).

4.2 Underlying Agreements

Kaminak's rights to the Coffee claims were acquired from prospector Mr. Shawn Ryan of Dawson City, 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 C$400,000; issuing 2,000,000 shares; and fulfilling a C$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 million, with annual advance royalty payments of C$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

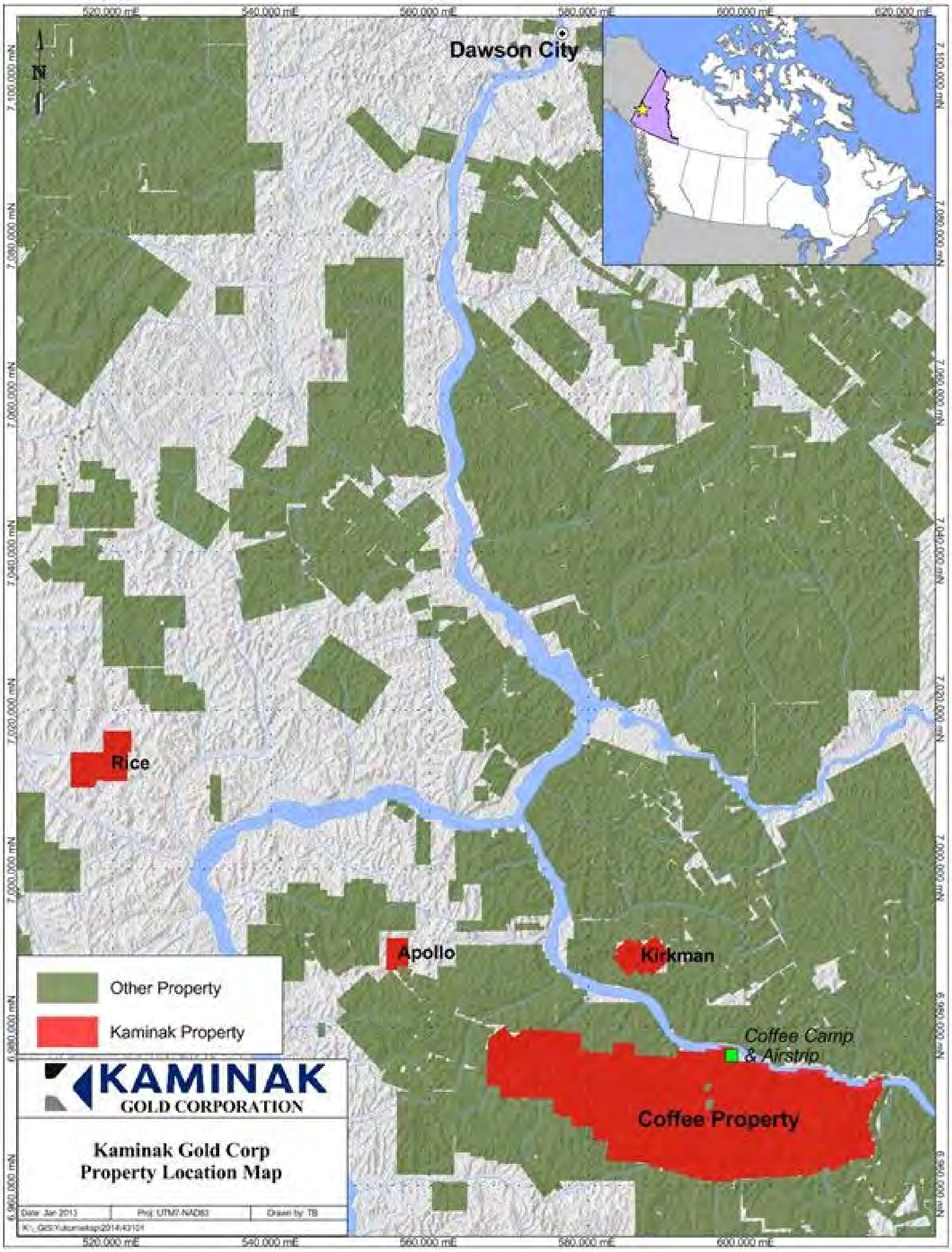


Figure 4-2: Mineral Tenure Map

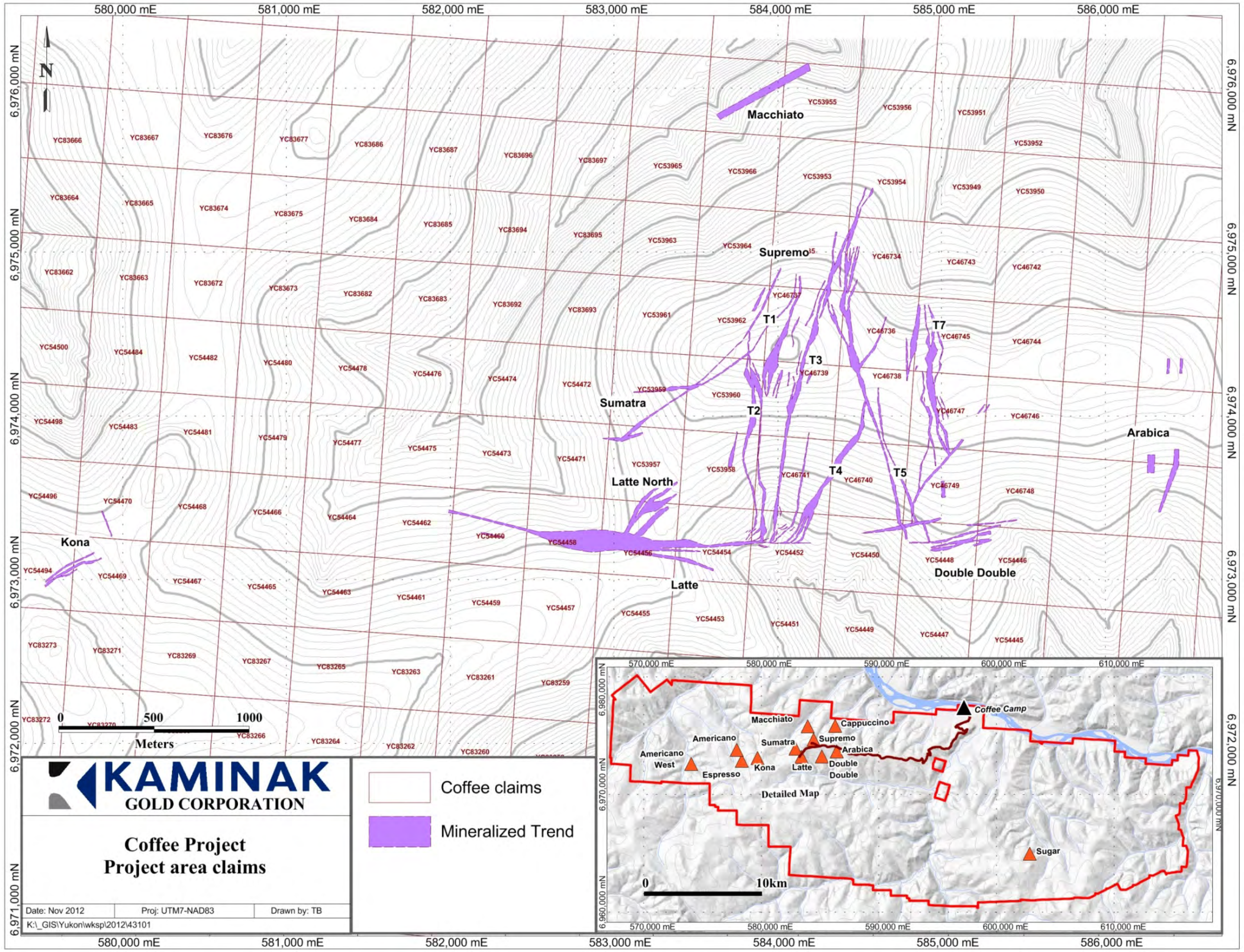


Table 4-1: Kaminak Coffee Property Claims

Property	Claim#	Grant Numbers	Expiry Date	NTS
Coffee	1-16	YC46734-YC46749	15-Dec-2032	115J14
Coffee	17-36	YC53949-YC53968	15-Dec-2028	115J14
Coffee	37-54	YC54445-YC54462	15-Dec-2028	115J14
Coffee	55-62	YC54463-YC54470	15-Dec-2027	115J14
Coffee	63-68	YC54471-YC54476	15-Dec-2028	115J14
Coffee	69-92	YC54477-YC54500	15-Dec-2027	115J14
Coffee	105-112	YC60176-YC60183	15-Dec-2028	115J14
Coffee	113-226	YC83190-YC83303	15-Dec-2026	115J14
Coffee	227-276	YC83652-YC83701	15-Dec-2026	115J14
Coffee	277-344	YC89405-YC89472	15-Dec-2026	115J14
Coffee	345-404	YC93441-YC93500	15-Dec-2023	115J13-14
Coffee	405-410	YC97368-YC97373	15-Dec-2023	115J14
Coffee	411-578	YC92601-YC92768	15-Dec-2023	115J13-14
Coffee	587-610	YC92777-YC92800	15-Dec-2023	115J13-14
Coffee	611-625	YC93351-YC93365	15-Dec-2023	115J13-14
Coffee	627-726	YC96801-YC96900	15-Dec-2023	115J13-14
Coffee	727-792	YC92535-YC92600	15-Dec-2023	115J14
Coffee	793-865	YC92818-YC92890	15-Dec-2023	115J14
Coffee	866-894	YC93271-YC93299	15-Dec-2023	115J14
Coffee	895-910	YC92801-YC92816	15-Dec-2023	115J14
Coffee	911-960	YD12701-YD12750	15-Dec-2024	115J14
Coffee	961-969	YD13231-YD13239	15-Dec-2024	115J14
Coffee	970-1416	YD13241-YD13687	15-Dec-2024	115J14
Coffee	1421-1429	YD13692-YD13700	15-Dec-2024	115J14
Coffee	1430	YD42501	15-Dec-2024	115J14
Coffee	1435-1496	YD42506-YD42567	15-Dec-2024	115J14
Coffee	1497-1714	YD42701-YD42918	15-Dec-2024	115J14-15
Coffee	1715-1718	YD43085-YD43088	15-Dec-2024	115J14
Coffee	1719-1781	YD43929-YD43991	15-Dec-2024	115J13-14
Coffee	1782-1954	YD43992-YD44164	15-Dec-2023	115J13-14
Coffee	1955-2124	YD16283-YD16452	15-Dec-2024	115J14
Coffee	2125-2346	YD89255-YD89476	15-Dec-2024	115J14-15
Coffee	2347-2596	YD91501-YD91750	29-Sep-2022	115J15
Coffee	2597-2724	YD91751-YD91878	29-Sep-2021	115J14-15
Coffee	2725-2740	YD91879-YD91894	29-Sep-2022	115J15
Coffee	2741-2812	YD91895-YD91966	29-Sep-2021	115J15
Coffee	2813-2846	YD91967-YD92000	29-Sep-2022	115J15
Coffee	2847-2936	YD90101-YD90190	29-Sep-2022	115J15
Coffee	93-104	YC60164-YC60175	15-Dec-2028	115J14
Coffee	579-586	YC92769-YC92776	15-Dec-2023	115J14
Cream	1-22	YC60088-YC60109	15-Dec-2024	115J13
Cream	23-68	YC83144-YC83189	15-Dec-2023	115J13
Lion	1-16	YC83761-YC83776	15-Dec-2024	115J14
Sugar	1-10	YC95568-YC95577	15-Dec-2025	115J15

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.

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.

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 Environmental and Heritage Considerations

The Coffee project is an undeveloped exploration project. Surface disturbances resulting from work completed by Kaminak include building of an access road, temporary trails for drill access, drill pad clearing, trenching, diamond core drilling, reverse circulation drilling, and other low impact geological activities.

In 2010, Kaminak developed an environmental procedures document that provides guidelines for minimizing environmental impact from exploration activities while following best practices and complying with relevant legislation and regulations. Included in this document are procedures for reclamation and rehabilitation, including an action plan for spill response, and for the initiation of a wildlife monitoring log. The environmental procedures document has been updated each year to reflect new information.

In September 2010, Kaminak retained Access Consulting Group (Access) of Whitehorse, Yukon to initiate an environmental base line survey. Works initiated immediately included commencement in October 2010 of monthly water quality sampling of the main streams and creeks draining from the area of exploration activity at the Coffee project. Monthly water quality sampling has been ongoing since October 2010.

Access has collected 42 monthly datasets, from October 2010 to May 2014 of sample data from locations on and tributaries to Coffee, Halfway and Independence Creeks, and from the Yukon River upstream and downstream of the Project. Samples have been analysed by Maxxam Analytics of Burnaby, BC and evaluated against the full suite of parameters specified by the Canadian Council of the Ministers of the Environment *Water Quality Guidelines for the Protection of Aquatic Life* (“CCME-PAL”).

On February 24, 2014, Access concluded that, “The water quality data to date on the Coffee Project area show system conditions which are generally consistent with unimpacted small creek systems monitored at other projects in the central Yukon. This water quality baseline program provides a good characterization of actual water quality conditions prior to mining and naturally occurring physiochemical conditions and temporal variability observed for the Coffee Project area. The three years of data collected shows seasonal variation in baseline water quality, with similar trends observed between seasons, with a number of parameters naturally exceeding the CCME PAL guidelines.” (Keesey, 2014)

In 2013, Access added two additional hydrology stations to the baseline survey and initiated a geochemical baseline program. Kaminak also retained Tetra Tech EBA of Whitehorse start a hydrogeology baseline survey. Four stations were established.

During August 2010, in collaboration with the Tr’ondëk Hwëch’in, Kaminak retained Matrix Research Limited (Matrix Research) of Whitehorse, Yukon, to conduct a heritage resources overview assessment and preliminary field reconnaissance over the Coffee property. One historical site and three pre-contact First Nations sites were identified within the property.

Buffers were set up around the sites. However, none occur in proximity to the established zones of gold mineralization and there was no impact on exploration programs during the remainder of the 2010 season or preceding years.

In 2011, as follow-up to the overview assessment, a more detailed heritage assessment was undertaken including archaeological fieldwork, which was completed in June 2011. Matrix Research communicated results to both Kaminak and Tr'ondëk Hwëch'in during the fieldwork. The survey confirmed that no heritage sites were located within, or overlapping with, zones of established gold mineralization or planned exploration, nor areas of ancillary infrastructure including the Coffee project camp area and the proposed access road. Buffer zones around heritage sites were established at the minimum required distance of 30 m for protection of the sites. Kaminak received the final report detailing the findings and recommendations of the 2011 Heritage Assessment survey in July 2012 (Matrix Research, 2012).

In 2012, an Oral History project of the Coffee project area was undertaken by the Tr'ondëk Hwëch'in Heritage Department and supported by Kaminak. The project included a site visit in July 2012 by the project participants including Tr'ondëk Hwëch'in First Nation Heritage Officers and the lead researcher/author of the Oral History project.

4.5 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.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Coffee project is located within the Dawson Range, approximately 130 km south of Dawson City and approximately 160 km northwest of Carmacks. The claims form an irregular rectangular block situated parallel to and south of the Yukon River (Figure 5-1). The Casino copper-gold porphyry deposit (Western Copper Corporation) is located approximately 30 km southeast of the main drilled zones on the Coffee project.

Access to the property is by airplane or helicopter from Whitehorse and/or Dawson and by barge via the Yukon River.. In 2011, Kaminak constructed a 23 km road from the barge landing at the Coffee project camp to the Supremo and Latte drilling areas. This road was the main access to the exploration activities during 2012 and 2013.

5.2 Local Resources and Infrastructure

There are currently no all-weather or winter roads connecting the Coffee project to any of the major communities in the Yukon, although Kaminak is permitted for a winter road from Carmacks. An airstrip is located at the Coffee project camp approximately 10 km from the areas of gold mineralization.

River transport along the Yukon River, with multiple barge access points to the Coffee project camp is available for five months during the summer period when the river is free of ice. The barge landing is located at the camp.

The proposed project infrastructure details are contained in Section 18.

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 City, the nearest access point, has a daily temperature average above freezing for 180 days per 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 m to 1,500 m above sea level. The majority of the property is above tree line and contains short shrubby vegetation. The Coffee 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. Overview of the Coffee project camp looking south
B. Core yard at Coffee 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 overview looking northwest of the Espresso, Kona, Latte, Supremo and Double Double mineralized zones.

6.0 HISTORY

The Coffee project area has limited hard rock exploration history and only minor placer activity. Coffee project has experienced sporadic exploration for placer gold from the turn of the last century to 1982. Hard rock exploration in the area before 1981 was limited to reconnaissance work in the 1960s and 1970s for porphyry copper.

C.D.N. Taylor, P.Eng., (Atlantic Energy Limited, August 1981) 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 (cited in Jaworski and Meyer, 2000).

Deltango Gold Ltd. conducted silt and soil sampling in 1999 in the area of the Coffee project claims and recommended further work, based on anomalous results (Jilson, 2000). In 1999-2000, a brief Coffee project area exploration program was conducted by consultants for 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, and minor fluid inclusion work. The 1999 work identified an open-ended reconnaissance soil gold anomaly. The 2000 work further delineated this anomaly to be approximately 400 by 900 m and further soil sampling in addition to mechanized trenching were recommended (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 project claims (Ryan, 2007; Ryan, 2008).

In June 2009, Kaminak executed an option agreement with Mr. Shawn Ryan to acquire the Coffee 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 in 2010, 2011, 2012, and 2013 on Supremo, Latte, Double Double, Kona, Espresso, Americano, and Sugar.

The exploration and drilling activities completed by Kaminak from 2009 to 2013 are discussed in Sections 9 and 10.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

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 Yukon-Tanana terrane is the pre-Late Devonian Snowcap assemblage, 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., 2006). 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., 2006).

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 Yukon-Tanana terrane (Allan et al., 2013; Colpron et al., 2006). Closure of the Slide Mountain Ocean by the Latest Permian 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 Yukon-Tanana terrane (S2 of Berman et al., 2007).

East-dipping subduction along the now docked YTT caused intra-arc shortening and contractional 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 within the Dawson Range (McCausland et al., 2006; Dusel-Bacon et al., 2002; Gabrielese and Yorath, 1991). Extension and unroofing 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

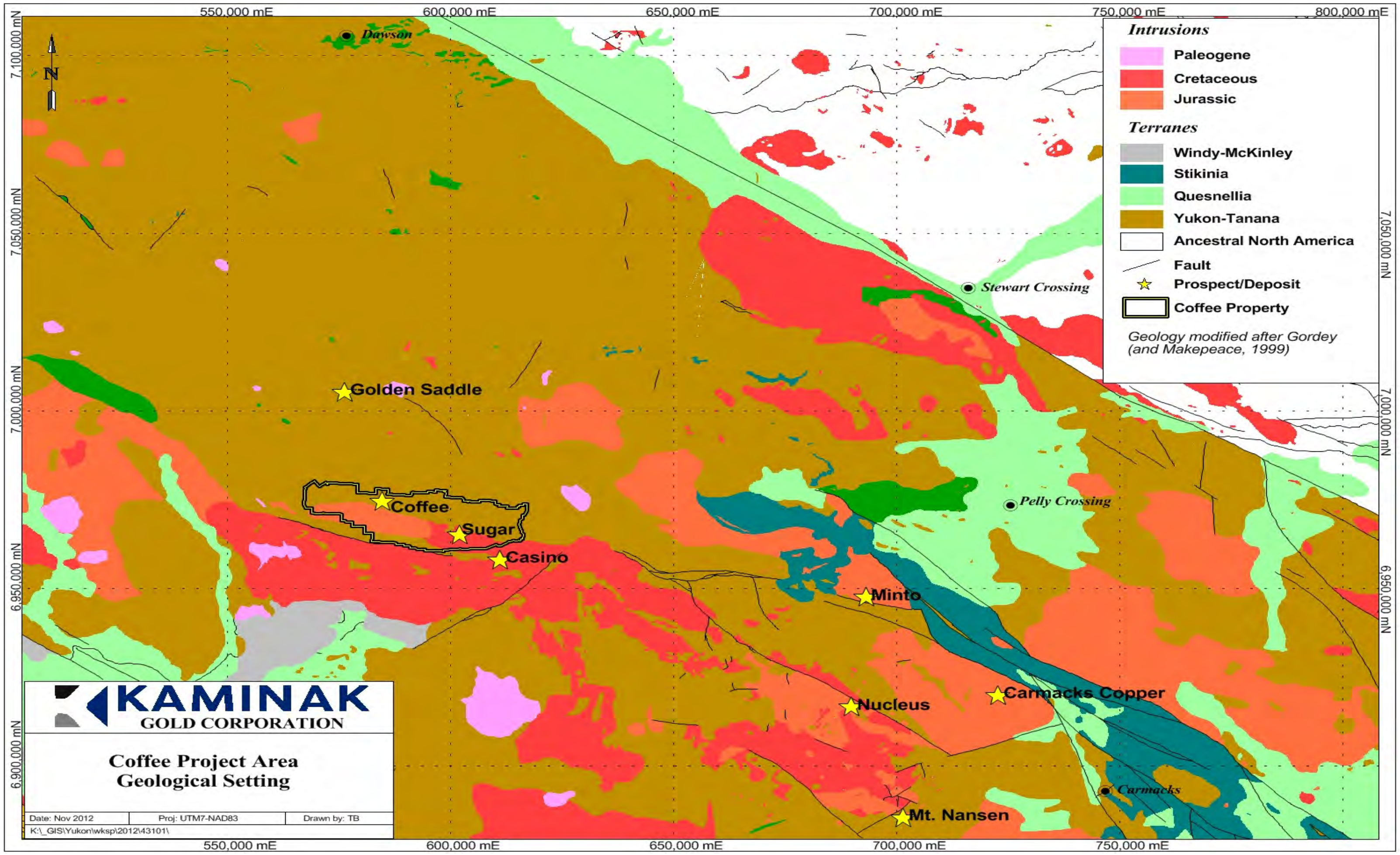
The Coffee project area is underlain by a package of metamorphosed Paleozoic rocks of the YTT that was intruded by a large granitic body in the Late Cretaceous. The Paleozoic rock package consists of a mafic schistose to gneissic panel which overlies the Sulphur Creek orthogneiss. Both packages form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

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, 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 (dacite and andesite).

Due to the rare outcrop exposure on the property (< 5%), the geological map (Figure 7-1 and Figure 7-2) 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 Project Area



(Modified after Gordey and Makepeace, 1999)

Table 7-1: Main Rock Units in the Coffee 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.

Figure 7-2: Geological Map of the Main Drilled Zones in the Coffee Project Area

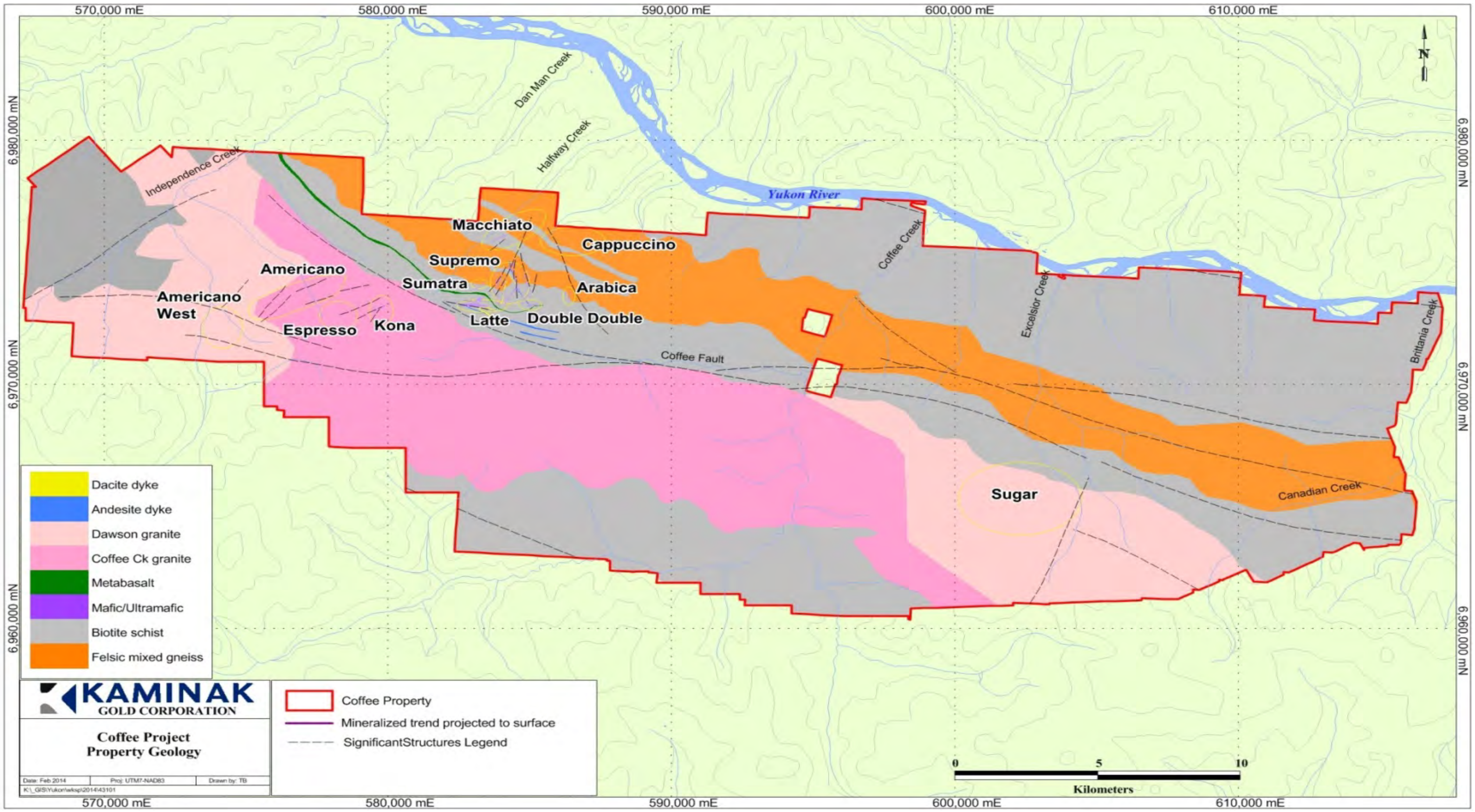
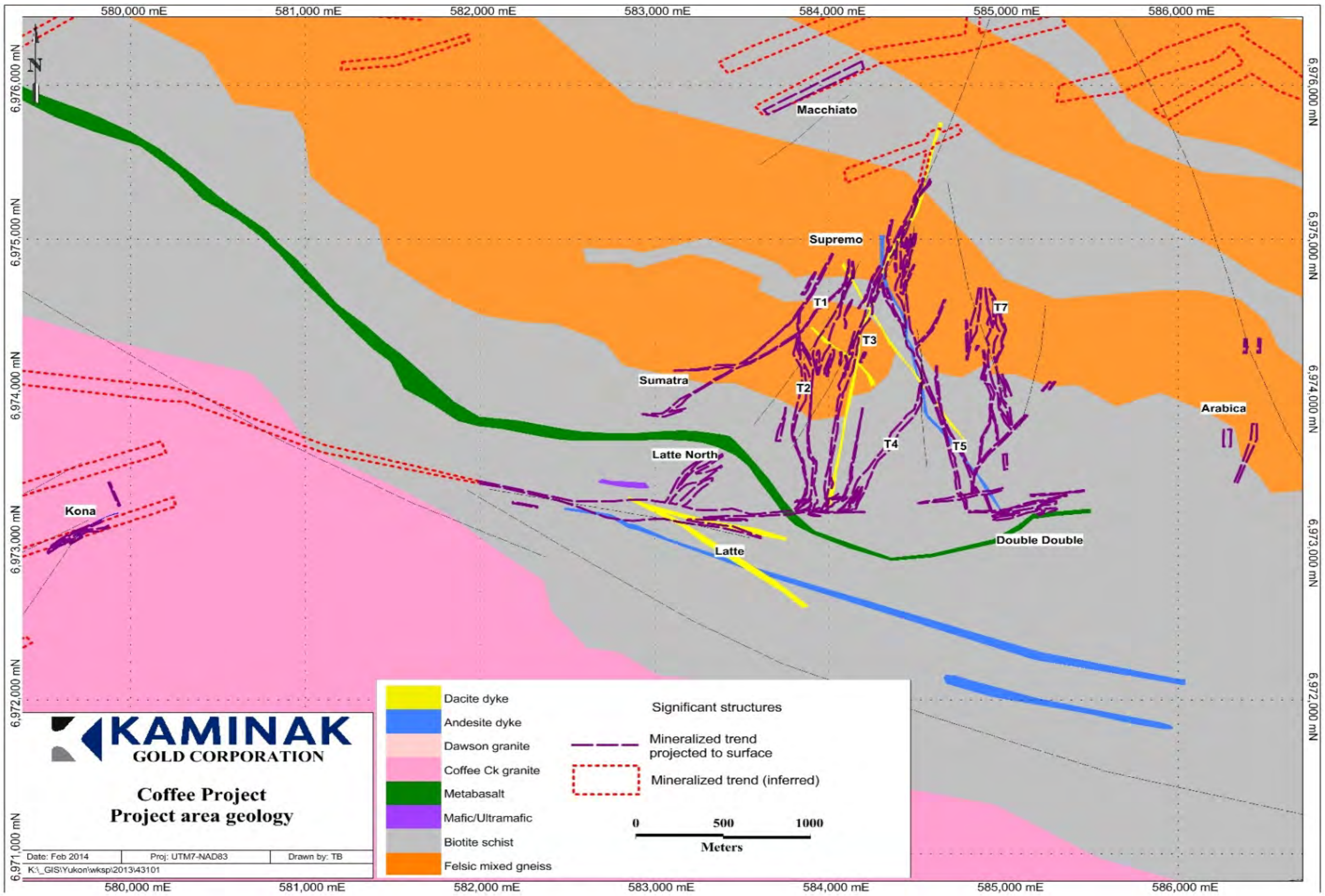


Figure 7-3: Geology in the Supremo, Latte, Double Double, and Kona Areas

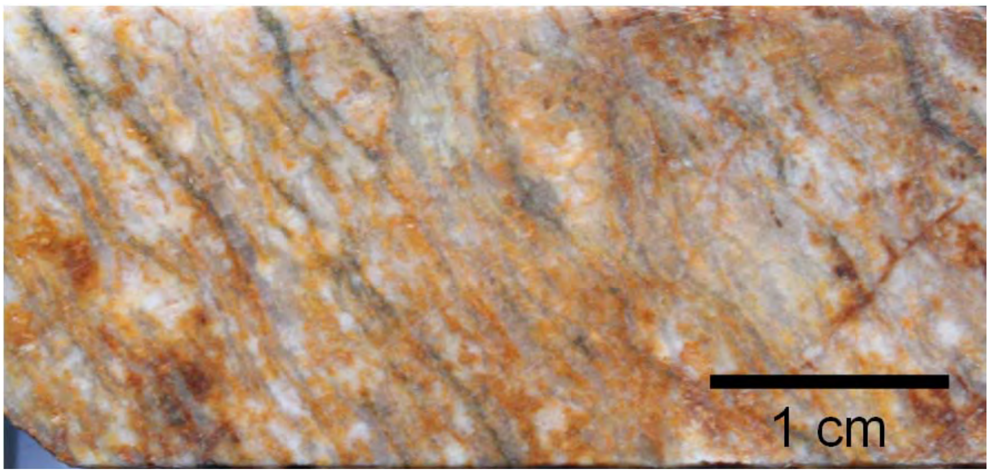


7.2.1 Lithologies

7.2.1.1 Felsic Gneiss

Felsic gneiss comprises variable quartz, feldspar augen, biotite and muscovite (Figure 7-4). The felsic gneiss is intercalated with volumetrically minor biotite-feldspar (± quartz ± muscovite ± amphibole) schist. Typical drill core intervals of biotite-feldspar schist within the dominant augen gneiss package vary in thickness from 0.3 to 10 m. They represent approximately 30% of the rock volume. Felsic gneiss hosts gold mineralization in the Supremo area.

Figure 7-4: Quartzo-feldspathic Augen Bearing Gneiss from Supremo Area, Borehole CFD0002 at 144m



7.2.1.2 Biotite Schist

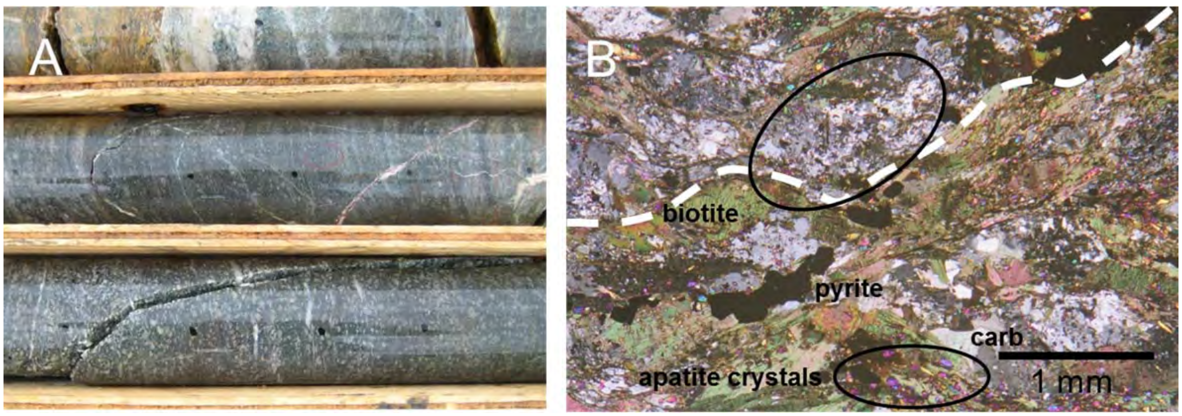
Biotite (± feldspar ± quartz ± muscovite ± carbonate) schist exhibits variable mineralogy and schistose to mylonitic textures (Figure 7-5). The rocks are variably retrogressed depending on their location relative to areas of metamorphic strain or mineralized intervals.

The schistose texture is defined by interconnected biotite laths up to 1.5 mm in size which wrap around relict and replaced feldspar porphyroblasts. The feldspars vary in size but are generally < 2 mm, and laths of phengitic muscovite occur in close association with biotite. Retrogression of the biotite schist produced chlorite replacement of biotite and exsolution of ilmenite within individual biotite laths. Feldspars are replaced by illite/sericite, carbonate, quartz, and epidote or zoisite.

The biotite schists are locally intercalated with marble bands that range from 0.3 m to over 5.0 m in width. The marble bands increase in volumetric importance toward the top of the sequence, and typically occur in localized groupings where band frequency increases to multiple thin (10 – 30 cm) bands per metre.

Carbonate stringers occur throughout the biotite schist package and feldspars are for the most part entirely replaced by carbonate, sericite, and quartz. The biotite schist is the primary host for gold mineralization at Latte and Double Double.

Figure 7-5: Biotite Schist in Drill Core and Thin Section



A. Biotite Schist in core from Latte. Borehole CFD0012 at 147 m

B. Polished thin section of (A) showing pyrite along the rock fabric and replacement of feldspar by quartz, sericite and carbonate (cross polarized light). White dashed line indicates rock fabric. Circled area in top centre of image highlights quartz-sericite alteration after plagioclase.

7.2.1.3 Muscovite Schist

Muscovite schist is mainly composed of quartz, muscovite, sericite/illite, and relict feldspar with a schistose texture which can locally grade to mylonitic. The schistose texture is defined by muscovite up to 2 mm in size which wraps around feldspar porphyroblasts replaced by sericite/illite and quartz, although up to 10 % biotite may be present (Figure 7-6). Minor cubic brassy pyrite is present as a foliation-concordant feature as seen in the biotite schist. Rare fine-grained ilmenite (< 0.1mm) is present along mica foliation, and the minor biotite present is readily replaced by chlorite.

The muscovite schist unit may have schistose or mylonitic texture locally. In contrast to biotite schist, it is rarely laterally traceable across drill sections, suggesting it is the product of a different, less mafic protolith which was sporadically deposited in the pre-metamorphic environment. It occurs at Latte and Double Double within the schistose and gneissic panels.

Figure 7-6: Felsic Schist in Drill Core and Thin Section

A. Felsic schist in drill core. Borehole CFD0013 at 161 m

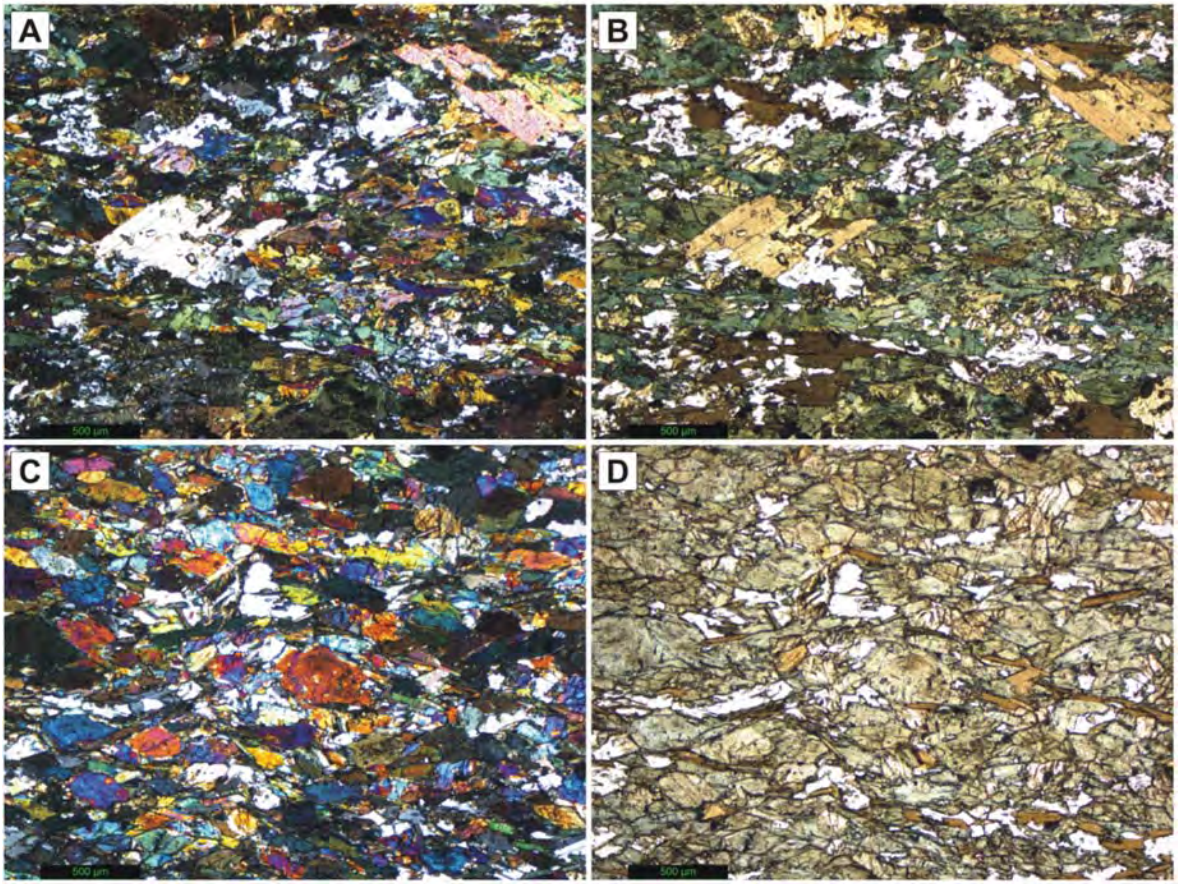
B. Polished thin section of (A) showing biotite and pyrite in a ground mass (gm) of quartz and sericite (plane polarized light).

7.2.1.4 Biotite Amphibolite

Thin intervals of amphibolite are present in the upper stratigraphy of the mafic schistose and gneissic package, comprising < 20% of the sequence. Massive amphibolite forms dark green-black units within the biotite schist but has not been observed in close association with muscovite schist. Amphibolite intervals generally contain 80-90% hornblende, 10-20% biotite, and rare plagioclase and quartz. Weak to moderate foliation is defined by alignment of amphibole (Figure 7-7).

Amphibole is fine-grained (< 0.5 mm) while biotite laths are generally larger in size (1 mm). Greenschist-facies retrogression is manifested by coarse epidote after amphibole and weak to moderate chlorite replacement of biotite. Rare leucoxene is observed as an alteration product after very minor ilmenite.

Figure 7-7: Thin Section Images of Biotite Amphibolite and Massive Amphibolite



A. Coarse brown biotite laths and fine-grained green hornblende define foliation through the sample. CFD0082 at 170.6 m, XPL
B. Same image as (A), PPL
C. Fine-grained hornblende comprising >95% of the sample is intermixed with fine laths of biotite. Sample is interlayered with coarse metagabbro. CFD0114 at 296 m, XPL.
D. Same as (C), PPL.

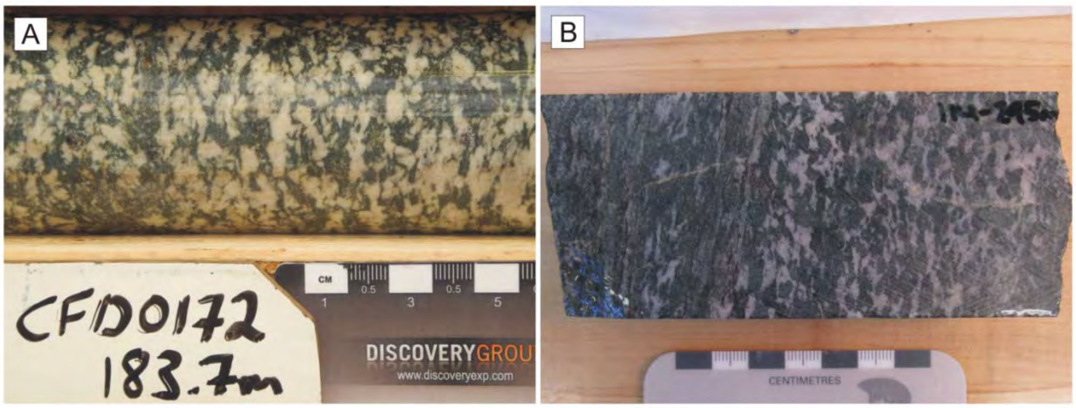
7.2.1.5 Amphibolite

Amphibolite within the deeper, mafic footwall of the schistose and gneissic package is composed of fine-grained hornblende with varying (≤15%) biotite content. While visually similar to biotite amphibolite found within the upper portions of the schistose panel, biotite content is marginally lower and the rocks are more Mg-rich. Biotite laths within the amphibolite are up to 0.25 mm in size, and minor quartz and fine-grained feldspar is present throughout the rock. In areas of strong retrogression, up to 30% of the biotite is replaced by chlorite, while up to 60% of amphibole is replaced by epidote.

7.2.1.6 Interleaved Metagabbro and Amphibolite

Metagabbro intervals are composed of coarse grained hornblende forming as radiating laths or coarse (3-4 mm) subhedral crystals. These units are variably foliated, with weak to nearly mylonitic fabrics (Figure 7-8). The metagabbros are very Mg-rich, and are interleaved on the metre scale with a geochemically identical amphibolite different to the two previously described.

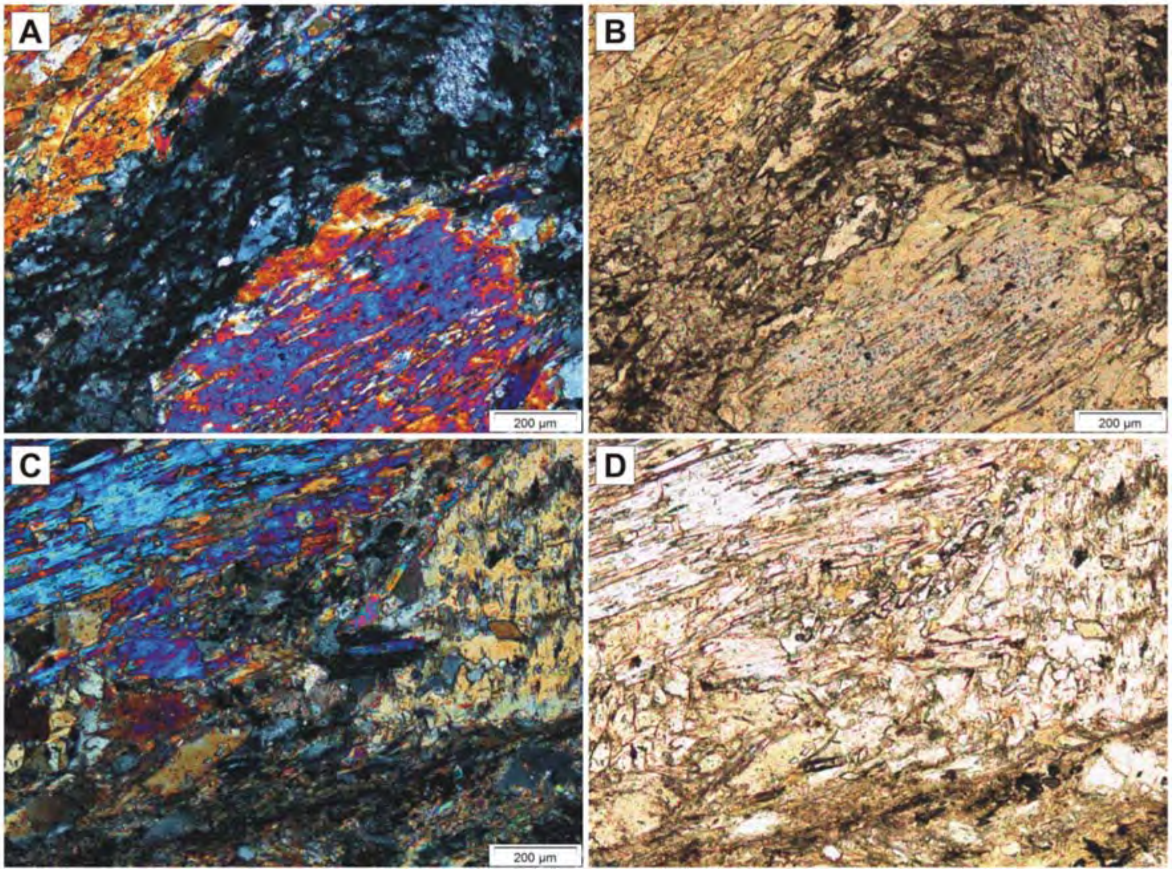
Figure 7-8: Drill Core Samples of Metagabbro



A. Unfoliated metagabbro from CFD0172 at 183.7 m
B. Variably foliated metagabbro with stronger foliation on left of sample, and coarse domain on right. CFD0114 at 295 m.

Where the unit displays metagabbroic textures, strong greenschist facies retrogression has completely replaced feldspar and amphibole crystals are retrogressively zoned from magnesiohornblende core compositions to actinolitic compositions at grain margins (Figure 7-9). Where the unit is composed of fine-grained, massive amphibolite, similar zonation is observed, although on a lesser scale. The massive amphibolite is composed of >95% magnesiohornblende to actinolite, with very minor biotite (Figure 7-9 C, D). Trace element geochemical analysis of both units demonstrates that both rocks are from the same source.

Figure 7-9: Thin Section Images of Metagabbro



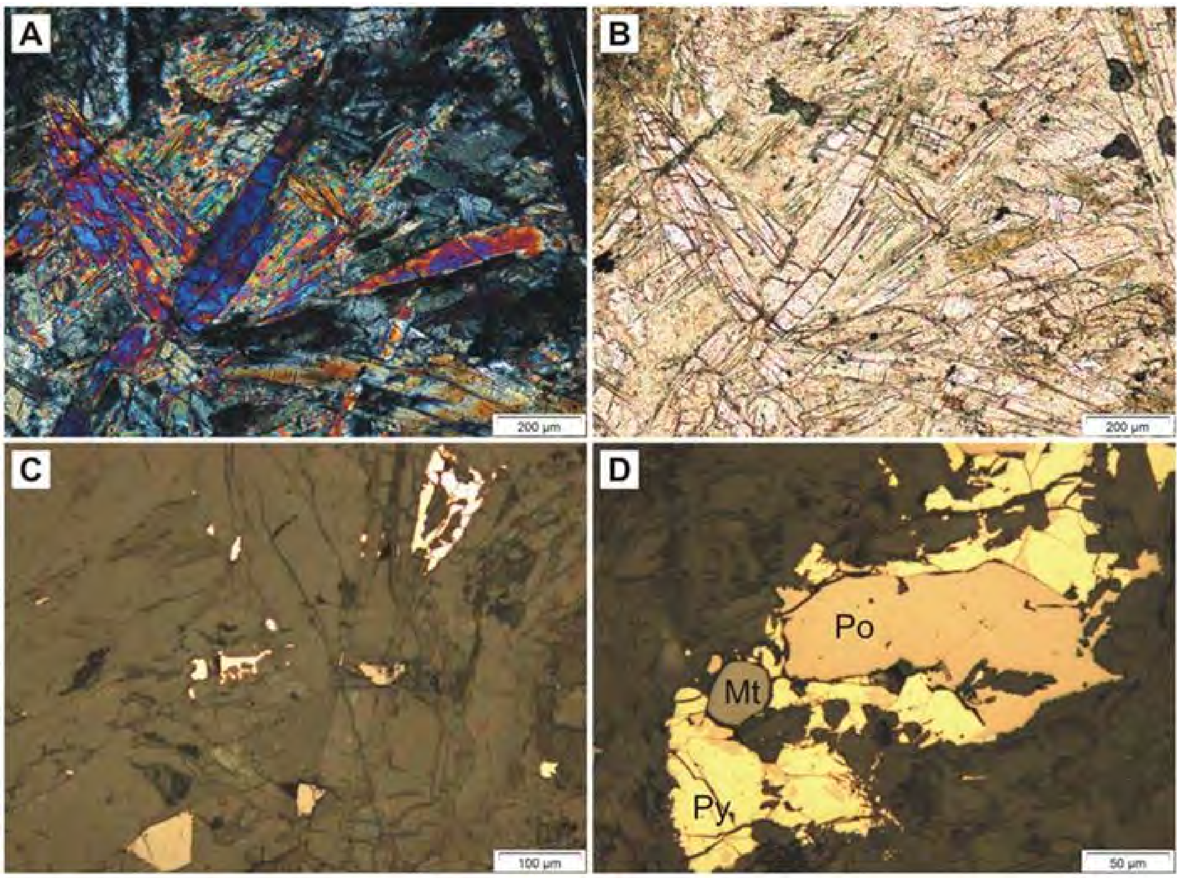
A. Coarse hornblende crystals with moderate retrogression to actinolite surrounded by chlorite-carbonate. CFD0114 at 295 m, XPL.
B. Same as (A), PPL.
C. Shredded amphibole crystal with coarse actinolite. CFD0010 at 180 m, XPL.
D. Same as (C), PPL.

7.2.1.7 Ultramafics

Ultramafic rocks are found at the Coffee project as both thin, 1-2 m, highly-deformed talc schists, and as an approximately 20 m thick panel within the Snowcap assemblage schistose rocks at the Latte zone. The thin and highly strained ultramafics are found throughout the schistose-gneissic panel at Coffee and are altered to talc, magnesite, and serpentine. They commonly contain high-chromium magnetite, and sulphides including pyrite and pyrrhotite (Figure 7-10, Figure 7-11).

The 20 m thick panel of ultramafic is found within the west-central region of the Latte zone. This panel is strongly serpentinized and contains coarse high-chromium magnetite, Mg-chlorite, talc, and serpentine (Figure 7-12). The unit is interpreted as a slice of Slide Mountain assemblage serpentinite which was tectonically emplaced during low-angle Jurassic thrust faulting. No mineralization is observed within this unit, although significant mineralization is intersected at the exterior margins of the panel. It is currently thought that the ultramafic panel may act as an aquitard to mineralizing fluid at Coffee.

Figure 7-10: Thin Section Images of Ultramafic Slivers from CFD0035 at 253 m



A. Laths of actinolite in close association with talc and chlorite, XPL
B. Same as (A), PPL
C. Reflected light image of fine sulphides within the ultramafics: visible pyrite and pyrrhotite, RL
D. Zoom of intergrown pyrite and pyrrhotite with accessory magnetite, RL.

Figure 7-11: Core Box Photo of a Thin Ultramafic Sliver in CFD0164 from 31.8 - 36.14 m

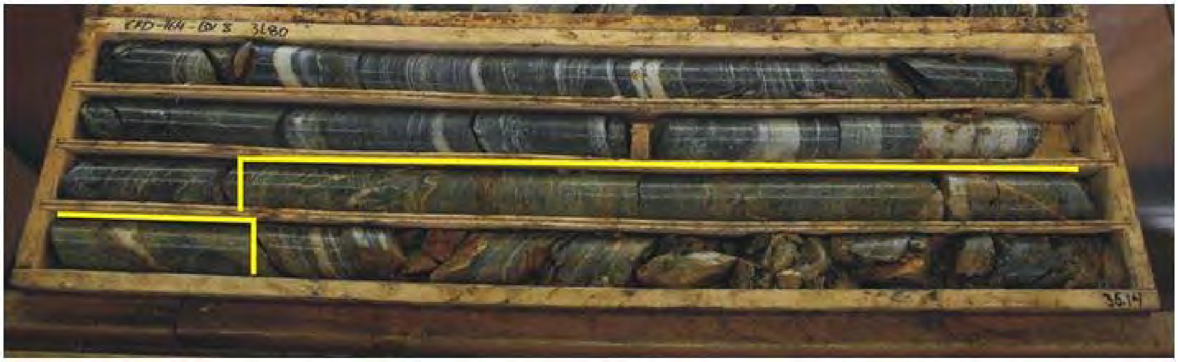
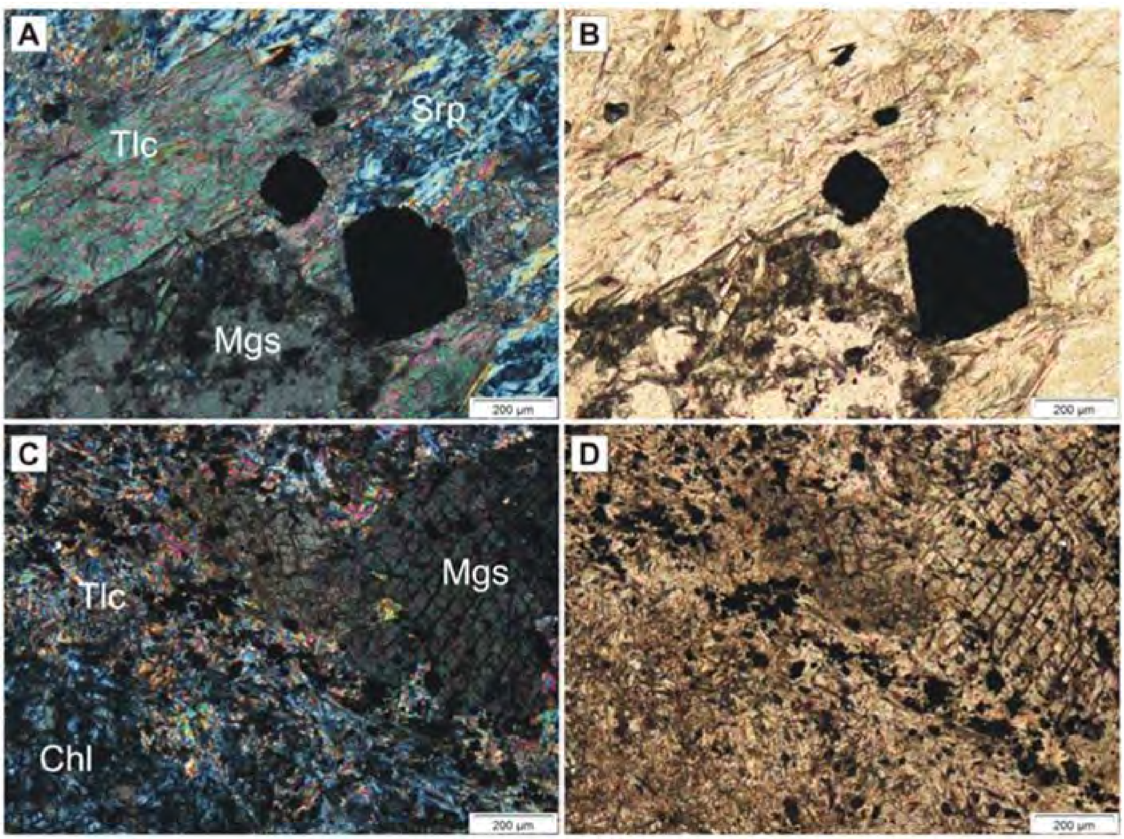


Figure 7-12: Thin Section Images of Serpentinized Ultramafics

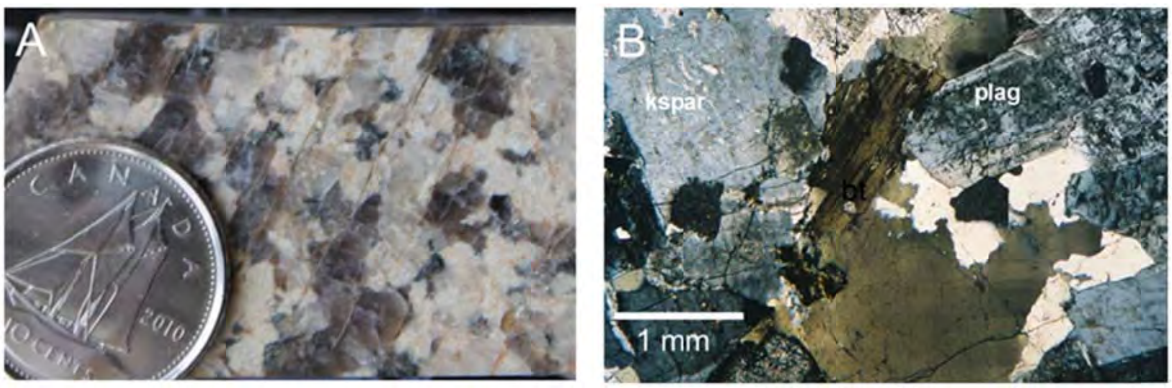


A. Coarse crystals of magnesite rimmed by talc within a serpentine matrix. CFD0082 at 135 m, XPL
B. Same as (A), PPL
C. Coarse magnesite crystals with fine sulphide, talc, and chlorite in the groundmass. CFD0113 at 120.4 m, XPL
D. Same as (C), PPL.

7.2.1.8 Granite

Equigranular granite underlies the southern third of the Coffee project area. This rock consists of coarse plagioclase, potassium feldspar, quartz, biotite, and hornblende (Figure 7-13). The contact between the schistose and gneissic rock panel and the granite itself occurs along the northern margin of the granite on the Coffee Creek fault in the deposit area. Only minor hornfelsing of the schistose and gneissic panel has been observed.

Figure 7-13: Coffee Creek Granite in Drill Core and Thin Section



A. Fresh granite exhibiting a weak foliation. Borehole CFD0053 at 128 m
B. Polished thin section of (A), showing biotite, plagioclase, potassium feldspar, and quartz, XPL.

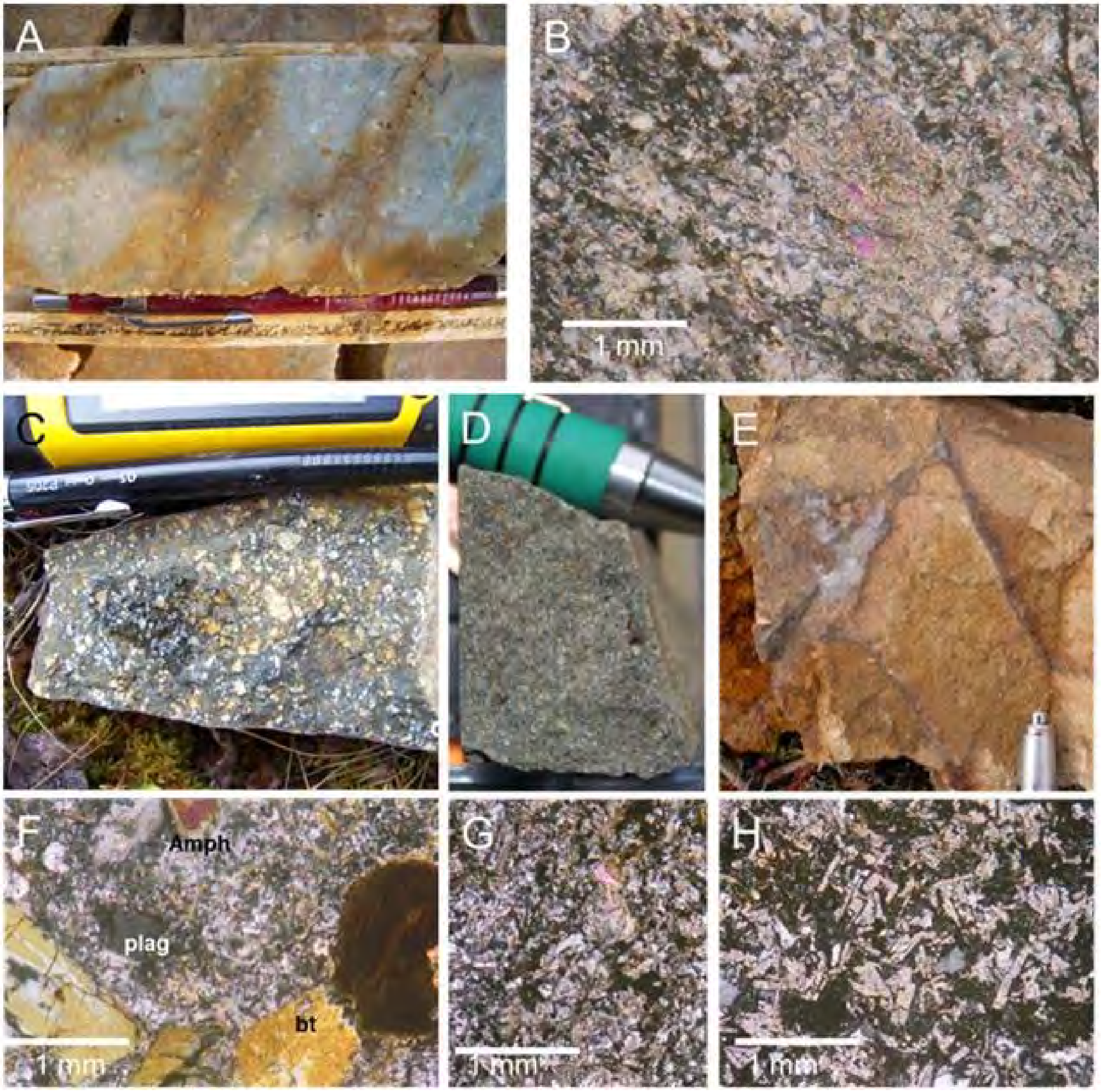
Limited geochemical study of the granite indicates that it is sourced from the same parent melt as a suite of dacitic dykes at Coffee. The granite itself is geochemically constrained as a ferroan, calc-alkalic or alkali-calcic, peraluminous A-type granite.

7.2.1.9 Dykes

Dykes of both andesitic and dacitic compositions are found at all gold prospects at Coffee. Andesite dykes are typically aphanitic with feldspar phenocrysts generally 0.3 – 1 mm in size (Figure 7-14). Local and rare andesite porphyries are intersected with euhedral feldspar phenocrysts up to 3 mm in width which are easily visible in hand sample. Preserved feldspar crystals occasionally exhibit polysynthetic twinning. The groundmass of the dykes is extremely fine-grained and contains fine feldspar crystals and a large clay component, dominated by kaolinite.

Dacite dykes are almost exclusively aphanitic. Feldspar phenocrysts are rarely visible in hand sample, occasionally reaching 1mm in size (Figure 7-14). The dykes are a light grey colour and are usually bleached white by pervasive clay alteration, sometimes obscuring primary igneous textures. Oxidized dacite dykes are easily distinguished by a liesegang-banded oxidation pattern.

Figure 7-14: Dykes in Grab Samples, Drill Core, and Thin Section



A. Dacite dyke in drill core. CFD0006 at 68.8 m
B. Polished thin section of dacite showing fine-grained quartz and sericite. CFD0009 at 27 m, PPL
C. Feldspar (andesite) porphyry. Collected from a trench in the Supremo area
D. Diorite grab sample. Collected at 584325mE, 6974769mN
E. Fluid altered diorite. Collected at 584218mE, 6974489mN
F. Polished shin section of (C), PPL
G. Polished thin section of (D), and likely a fine-grained equivalent of (C), PPL
H. Polished thin section of (E), and likely a fluid altered equivalent of (C), PPL.

7.2.2 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 was deposited during the latest Cretaceous event.

Table 7-2: Tectonic Events at Coffee

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

(Berman et al., 2007; Buitenhuis, 2014; Mackenzie and Craw, 2013)

<u>Metamorphic Foliation</u>

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 (S_2 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° to the south-southwest (190-230°)
- Latte: 35-55° to the south-southwest (180-210°)
- Double Double: 35-65 degrees to the south-southwest (170-200°)

<u>Jurassic Shearing</u>

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. The 20 m thick lens of ultramafic rock north of Latte was emplaced during this thrusting.

Brittle Fracturing and Faulting

Following post-collision uplift and erosion in the YTT (Berman et al., 2007) 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 so are syn-to-post granite emplacement (±98Ma). Dacite and andesite dykes follow these brittle fractures.

Gold-mineralized structures comprise strike-extensive planar zones with a range 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. However, both in cross section and along strike, gold mineralization, elevated arsenic and antimony, alteration, oxidation, deformation intensity and the presence of sub parallel pre-mineralization dykes display continuity.

Felsic (dacitic) to intermediate (andesitic) dykes crosscut all other lithologies and generally strike in either a northerly direction (northwest to northeast striking in the Supremo, Double Double, and eastern Latte zones) or an easterly direction (east to southwest striking in the Western Latte and Kona zones). The dykes commonly occur within the mineralized structural deformation zones. The dykes are commonly deformed and carry gold mineralization, especially along their margins, clearly indicating they pre-date mineralization. These observations suggest that the dykes exploited (intruded along) older pre-existing structures and in turn were deformed by subsequent reactivations of those structures during later deformation events. Furthermore, the dykes may have provided additional rheological contrast which focused strain, fluid flow, and sulphide and gold mineralization.

Structural measurements of vein orientations and deformation fabrics from oriented drill core provide hard data on the structural geometries, but are often not available in the mineralized zones due the incoherent nature of fractured and often clay-altered core. Where measured, key structural orientations from within mineralized zones, including dominant fracture orientation, internal fracture or shear fabric, breccia margin, vein or dyke margin orientation are used to interpret mineralization geometry.

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. Where not yet drilled, mineralized structures can be traced by: 1) the orientation of linear gold-in-soil anomalism (surface expression of ‘day-lighting’ bedrock structures), 2) the orientation of linear magnetic lows, and rarely by 3) linear topographic structures including erosion-resistive (T3) and erosion-recessive (Latte and Double Double).

The Latte zone comprises an east-west trending steeply to moderately south-dipping (65° in the west to 85°in the east) shear zone characterized by brittle deformation that overprints older ductile strain fabrics, consistent with a multiply-reactivated shear zone environment. The Latte structure can be traced over 8 km to the east and west of the Latte gold zone, and appears to mark the southern boundary of gold mineralization in the Latte-Supremo-Double Double area (although exploration south of the Latte structure has not been exhaustive).

The Latte North zone comprises a northeast-striking shear zone dipping about 60° to the southeast. It appears to be a splay from the Latte zone.

The Supremo zone comprises a corridor of steeply dipping (70 to 90°) north-south trending structures, the "T structures", which crosscut the augen gneiss host rock. T3 is the dominant structure in this orientation. Step out drilling completed in 2013 indicates that T2, T3 and T4 all curve to the west as they approach Latte from the north and merge with the Latte zone to the south. T5 appears to merge with the Double Double Zone to the south. The T7 structure has not been connected to other zones via drilling but soil geochemistry and geophysics suggests that it connects with the Double Double Zone to the south.

The Double Double zone is hosted within a steeply north-dipping (80 to 90°) east-northeast–west-northwest trending structure located 600 metres along strike from the eastern margin of the Latte Zone. The Double Double structure is characterized by brittle deformation of the host metasedimentary/ metavolcanic stratigraphy, without the pre-existing ductile fabric seen in the Latte Zone.

The Sumatra zone consists of two structures: a southwest-striking main structure dipping about 75° northwest and a northern, east-striking structure dipping about 60° south. The northern structure merges into the main structure. The northeast end of Sumatra appears to curve north and merge into the Supremo T2 zone.

Granite-hosted gold prospects located west-southwest of Supremo (Kona, Espresso, and Americano) are hosted both along strike to the west of the Latte structure, and in related splays and cross-structures. These zones exhibit similar steeply-dipping planar structures with overlying linear gold-in-soil anomalies. Unlike Supremo, Latte, and Double Double, these western zones are hosted within the Coffee Creek Granite. These zones may represent an array of faults connected by linking structures; further work is needed to better define these structures.

7.3 Mineralization

Exploration drilling completed from 2010-2013 has led to the discovery of significant gold mineralization in over 19 separate areas of the Coffee project: Supremo T1, Supremo T2, Supremo T3, Supremo T4, Supremo T5, Supremo T7, Latte, Latte North, Sumatra, Arabica, Latte Extension, Double Double, Americano West, Americano, Espresso, Kona, Macchiato, Cappuccino, and Sugar (Table 7-3).

Table 7-3: Main Mineralized Zones Investigated by Drilling on the Coffee Project Area

Zone	Host Rocks	Summary Description
Supremo	Augen Gneiss	Narrow gold-bearing brittle structures with gold commonly hosted in matrix-supported breccia and dacite dykes. Gold associated with quartz-sericite-pyrite alteration.
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 biotite laths. Potential remobilization of gold to other structures.
Double Double	Augen Gneiss	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 strong disseminated sulphide mineralization.
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.
Americano, Americano West and Espresso	Granite	Zones of fracture-controlled and disseminated pyrite. Gold hosted in quartz-sericite altered granite similar to Kona. Stibnite noted at Americano West.
Macchiato and Cappuccino	Augen Gneiss	Strong oxidation, silica flooding, abundant limonite and brecciation noted at Macchiato.
Sugar	Granodiorite	Sooty pyrite with arsenopyrite, pyrrhotite and/or stibnite in subvertical quartz-carbonate veins with silica-sericite alteration as halos. Gold hosted in granodiorites to quartz monzodiorites.
Sumatra	Biotite-feldspar Schist, Augen Gneiss	Gold hosted within two structures dipping in opposite orientations which intersect in the eastern portion of the zone. Strong oxidation after both disseminated, foliation controlled mineralization as well as common polyphase brecciation.
Arabica	Augen Gneiss	Strong oxidation, silicification within moderately-to-steeply east dipping structural corridors. Mineralization characterized by strong limonite-hematite.

7.3.1 Supremo

The Supremo Zone is hosted in the northern augen gneiss package and consists of a number of discrete north-to-northeast trending, steeply-dipping structures (T1 to T8), and spaced 50 to 100 m apart, based on linear gold-in-soil anomalies and extensive drilling.

Core drilling from 2010–2013 and reverse circulation drilling from 2011–2013 focused on significant high-grade gold mineralization identified in the north-northeast–trending steeply east-dipping T-structures associated with breccias and dykes. From east to west the main drill-tested T-structures are: T1- T2 (1,100 m strike length, open north and south), T3 (2,250 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 clay 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 7-15a)

a). This style of gold mineralization generally yields grades between 5 and 60 g of gold per tonne (g/t gold).

Breccia textures range from mature matrix-dominant phases with rounded fragments to wall rock crackle breccias, and 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.

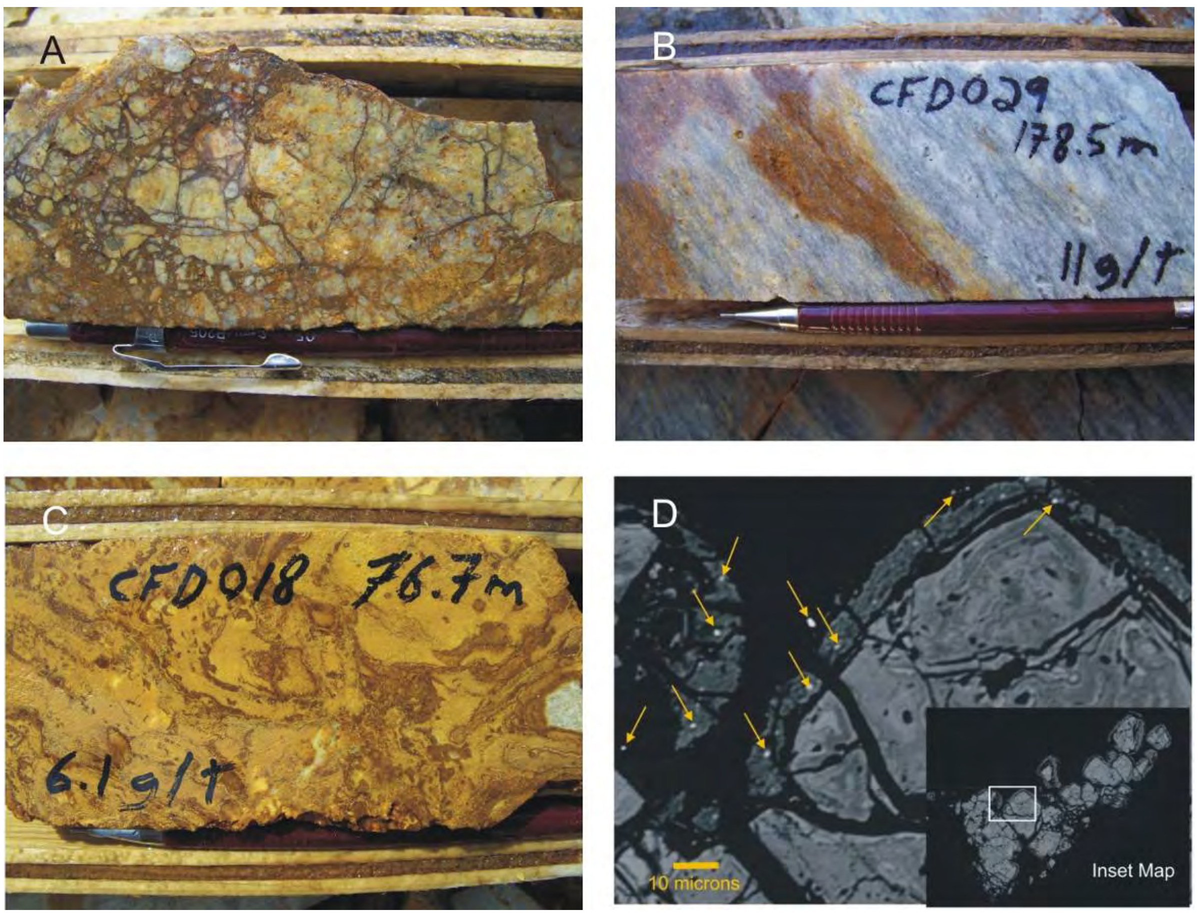
The lower grade gold mineralization is associated with pervasive hydrothermal alteration of non-brecciated gneissic host rock replacing biotite with pyrite and yields grades ranging between 2 and 10 g/t gold (Figure 7.15b). The hydrothermal alteration is characterized by an overall removal of potassium and aluminum with the addition of sulphide and silica.

Andesite and dacite dykes appear to have utilized the same structures as mineralizing fluids, but they are themselves altered and locally auriferous (Figure 7-15c). 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.

Preliminary portable infrared mineral analyser (PIMA), ASD TerraSpec portable infrared mineral spectroscope, and electron microprobe work indicate that illite and iron-carbonate compose part of the alteration mineral assemblage associated with gold at Supremo. Micron-scale gold is 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 7-15dd).

The microscopy and microprobe work 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 7-15: 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.

7.3.2 Latte

Drilling across an east-west trend of gold-in-soil anomalies at Latte has intersected gold mineralization beginning at surface (Figure 7-16). This linear trend overlies the Latte and Latte North structures. Latte consists of a stacked set of moderately-to-steeply south-southwest dipping, east-southeast striking brittle-ductile structures, while Latte North splays off from the main Latte structure, dipping moderately to steeply to the southeast and striking to the northeast. No shear fabric or observable high strain is visible in association with the steep and mineralized Latte structures. All structures remain open at depth, and step-back drilling during the 2011 season intersected mineralization at depths of up to 450 metres-below-surface.

Figure 7-16: Expression of the Latte Structure at Surface Looking East (Section 583250mE)

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 section. To the east these structures separate, with a splay striking approximately 85˚ which dips near vertically to the south. The structures continue to the east and eventually merge into the complex Connector zone, where the Supremo north-south structures and the east-west Latte and Double Double structures converge (Figure 7-3 and Figure 10-1). Total traceable length of the mineralized Latte structure is in excess of 2,100 m.

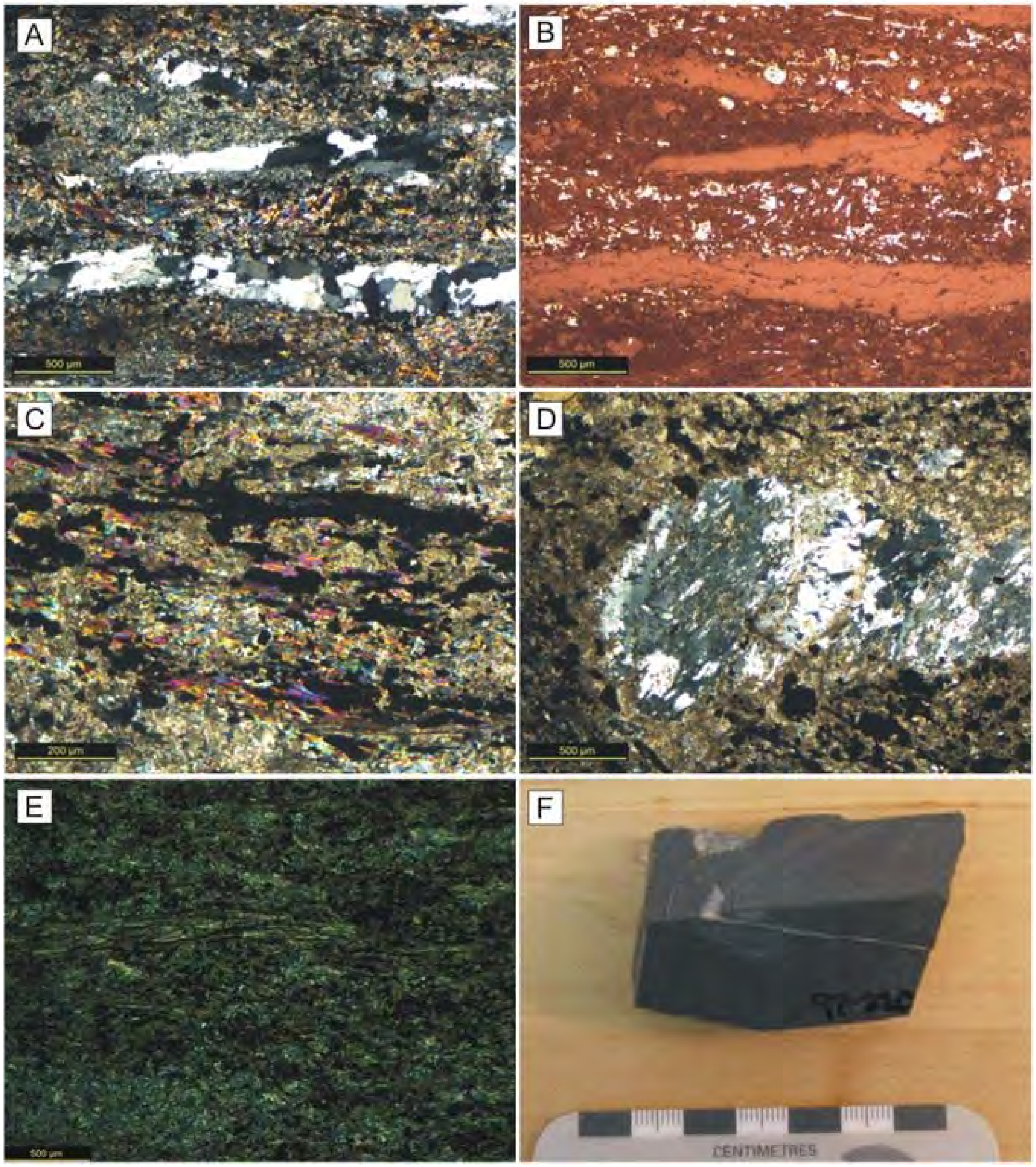
Latte North displays identical mineralization textures as Latte Main; however the structures strike at approximately 45° and dip approximately 60° to the southeast. Latte North currently splays away from the main Latte corridor for a strike length of 275 m.

Mineralization at Latte consists of multiple distinct styles of mineralization beginning with disseminated gold-bearing arsenian pyrite, overprinted by later brecciation and late fluid ingress (Figure 7-17). 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 reacts strongly with Fe-bearing biotite within the biotite schist at Latte. A sulphidation reaction proceeds, where Fe within the biotite is leached to form fine-grained arsenian pyrite, illite, and dolomite which pseudomorphously replace the parent biotite lath (Figure 7-17: Reaction 1). Titanium within the parent biotite is removed and incorporated in the hydrothermal illite, as well as hydrothermal rutile. A third white mica phase grows out of solution as fine laths which rim some mineralization-phase white mica.

Figure 7-17: Reaction 1: Simplified Mica Sulphidation Reaction

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

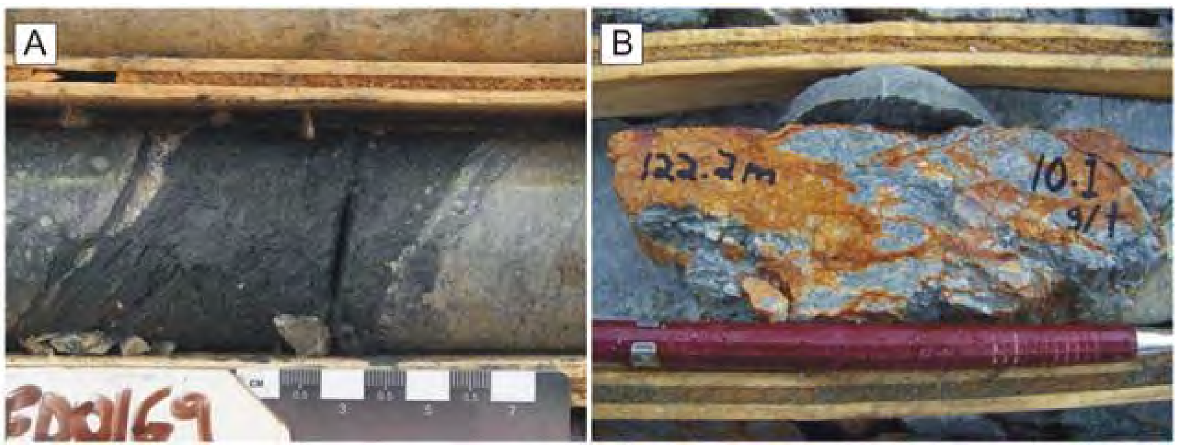
Figure 7-18: Disseminated Mineralization within the Latte Zone



A. Finely disseminated arsenian pyrite along relict micaceous foliation. CFD0164 at 464 m, XPL
B. Same as A, RL
C. Detail of sulphidized mica with fine arsenian pyrite along relict foliation planes. CFD0164 at 469 m, XPL
D. Relatively fresh feldspar porphyroblast surrounded by sulphidized mica and dolomite. CFD0164 at 469 m, XPL
E. Strong disseminations of fine-grained arsenian pyrite. CFD0097 at 220 m, XPL
F. Hand sample image of E.

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 7-18). 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 which precipitated extremely fine gold-bearing arsenian pyrite along their margins are interpreted to be of the same phase as the sulphide-matrix breccias. There is potential for significant remobilization of gold from disseminated intervals due to continued and intense fluid-rock interaction.

Figure 7-19: Core photographs of Latte Zone Pyritic Faults and Sulphide-matrix Breccia

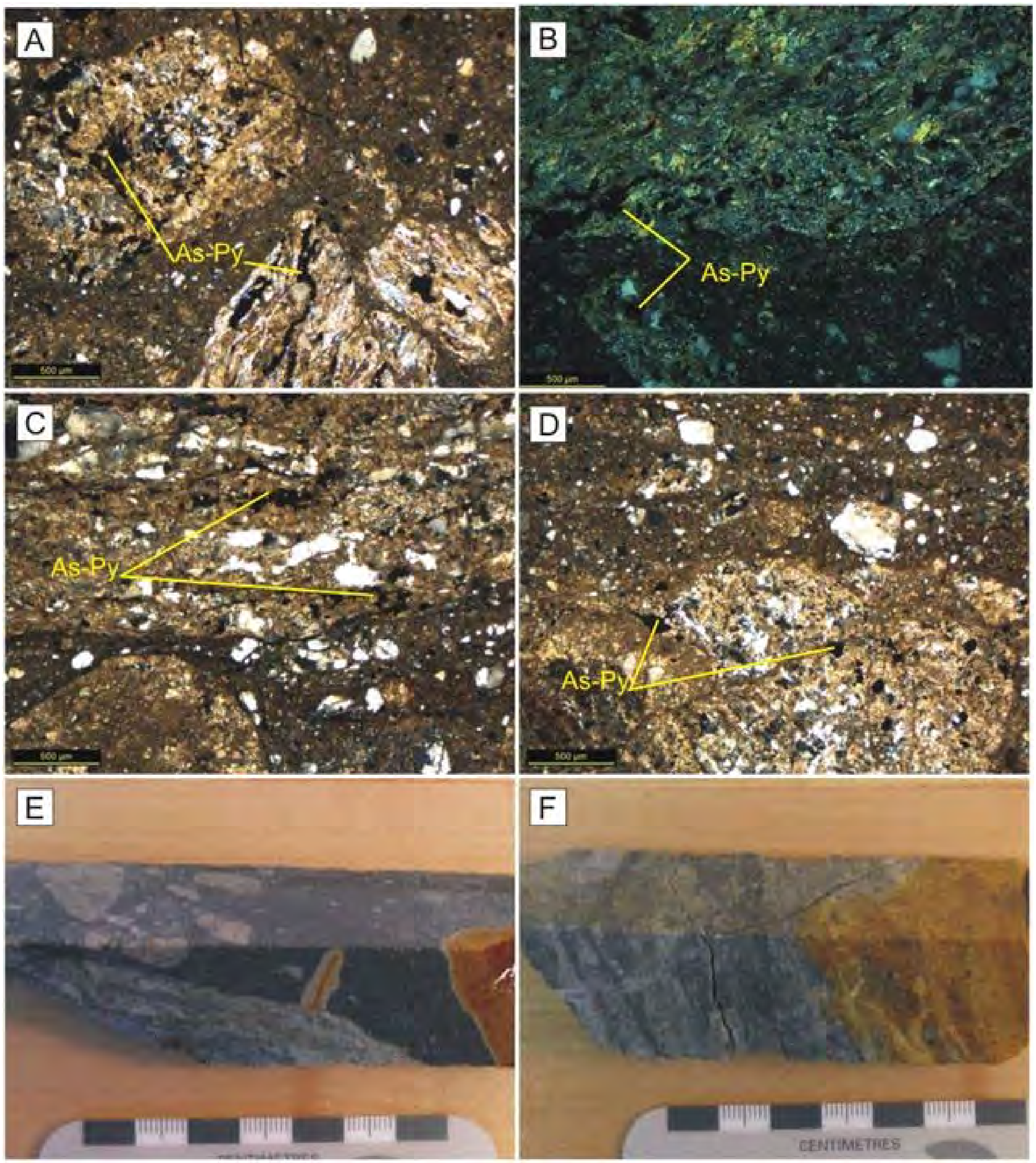


A. Pyritic fault in CFD0169 at 237.8 m
B. Sulphide-matrix breccia from CFD0010 at 122.2 m.

Late breccias are also common, with angular-to-subrounded clasts of mineralized wall rock set in a matrix of rock flour and silica (Figure 7-19). These breccias exhibit both tectonic and fluid textures and can be greatly comminuted, and locally polyphase. No additional mineralization is observed within these breccias.

Mineralization at Latte is hosted exclusively by schistose rocks or breccias. 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 7-20: 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.

7.3.3 Double Double

The Double Double Zone trends east-northeast with a known strike length of 600 m, 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 appears to be structurally controlled and 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 7-20a).

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 7-20b).

Similar to the Supremo Zone, gold at Double Double is micron-scale (Figure 7-20c), and illite has been detected by infrared spectroscopy within the mineralized intervals. Other alteration minerals observed at Double Double include sericite, epidote, leucoxene, hematite, and carbonate.

7.3.4 Kona

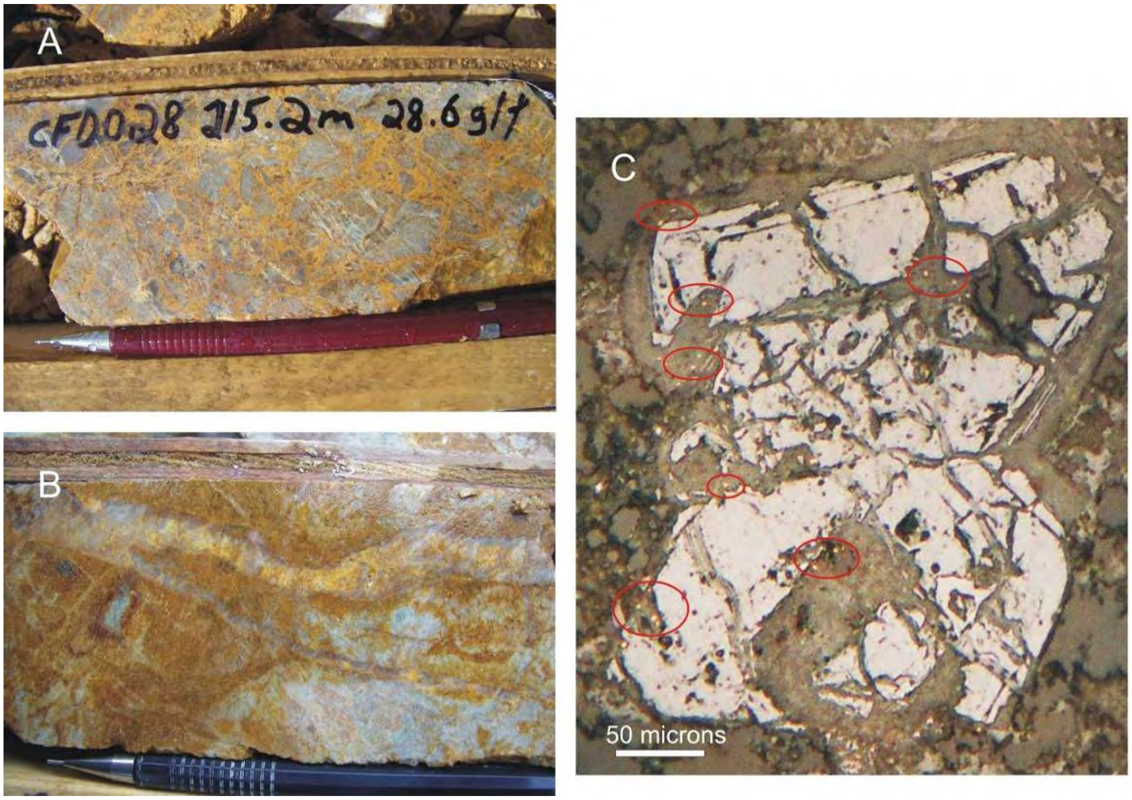
Drilling in the Kona Zone was designed to investigate gold-in-soil anomalies and encountered a different style of mineralization hosted in granitic rocks. The gold mineralization is hosted in near-vertical brittle structural zones directly underlying gold-in-soil anomalies.

The Kona Zone is hosted in equigranular granite and consists of east-northeast trending, steeply south-dipping stacked 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 pyrite, which typically replaces ferromagnesian minerals (Figure 7-21a), and also occurs as veins/veinlets or fracture fill, and in sulphide-matrix fault breccias (Figure 7-21b).

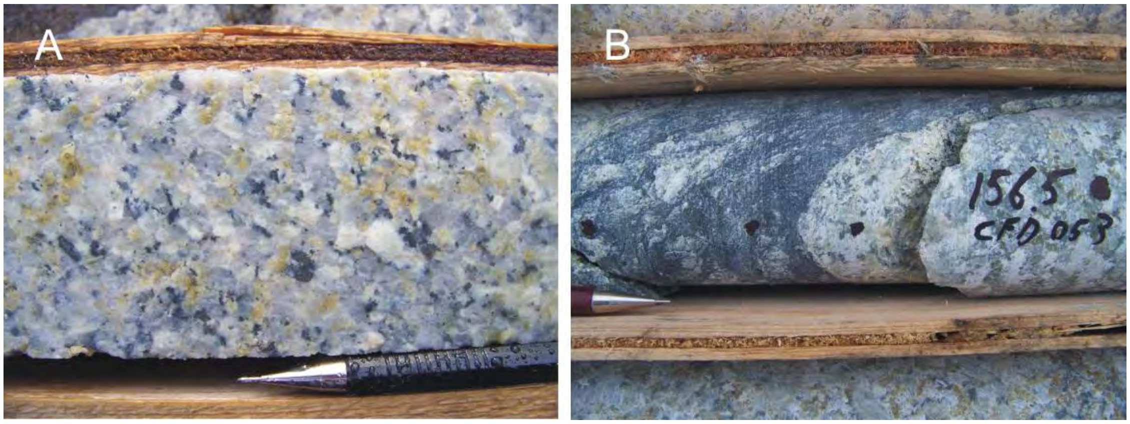
Minor realgar and orpiment have both been observed in reverse circulation cuttings from Kona during the 2011 drill program.

Figure 7-21: 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.

Figure 7-22: 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.

7.3.5 Americano, Americano West and Espresso

The Americano area is underlain by granite and comprises two parallel northeast trending linear gold-in-soil trends totalling more than 4 km in length. These two trends become linked to the east by a north by northeast trending gold-in-soil anomaly informally known as the Americano “link” structure.

Widely-spaced boreholes were drilled at Americano in 2010–2011 in order to test for the presence of steeply-dipping gold-bearing brittle structures analogous to the nearby Kona gold zone. The Espresso zone is located between Kona and Americano, associated with a large gold-in-soil anomaly. This area was tested with limited drilling in 2010.

Gold zones drilled at Americano and Espresso are hosted in sulphidic and clay-altered brittle fault zones crosscutting granite, similar to the Kona zone. Limited reconnaissance drill testing beneath gold-in-soil anomalies at Americano West in 2011, to the southwest of the Americano “link” structure, yielded several narrow gold intervals. The Americano West area is underlain by equigranular granite and the gold-bearing intervals are characterized by silica-sericite-clay alteration and fine-grained pyrite replacing mafic minerals. Minor pyrite stringers, sulphide-matrix fault breccia, and clots/dissemination and veins of stibnite were also noted at Americano West.

7.3.6 Macchiato and Cappuccino

The Macchiato and Cappuccino zones, located north and northeast of the Supremo Zone, respectively, are hosted by the augen gneiss host rock package with significant gold intervals intersected at Macchiato and minor gold encountered at Cappuccino during preliminary diamond drilling in 2011. Significant gold intervals at Macchiato are characterized by strong oxidation and silica flooding associated with pervasive limonite and hematite. Crackle breccias with silica-limonite or clay cement were observed in addition to silica-limonite vein and veinlet networks cutting strongly altered host wall rock. The steeply-dipping gold zone appears to trend northeast. The mineralization style encountered at Macchiato is very similar to that observed in the Supremo Zone.

7.3.7 Sugar

The Sugar area is located in the southeastern area of the Coffee project, 22 km southeast of Supremo and 12.5 km south-southeast of the Coffee project camp. Drilling tested the five largest soil anomalies greater than 100 parts of gold per billion over an area roughly 3.5 by 1.5 km.

The geology of the Sugar area is comprised of intermediate intrusions of various affinities. The most prevalent unit is a multi-phase, equigranular, medium grained granodiorite to quartz monzodiorite. This unit is intruded by two porphyritic units: a hornblende-phyric medium-grained granodiorite, and a plagioclase-hornblende porphyry. These units are in turn locally intruded by aphanitic mafic dykes. Additional volumetrically minor units include small diorite and tonalitic dykes and rafts of metasedimentary rocks. The host granodiorites abut the biotite schist package to the north of the Sugar area; although preliminary evidence suggest a fault contact (observed in SGD0010), the nature of the contact remains unresolved.

Mineralization at Sugar is comprised of sooty pyrite ± arsenopyrite ± pyrrhotite ± stibnite in quartz-carbonate veins, silica sericite minor chlorite and clay salvages on the contacts of some of the veins. Further infrared spectroscopy work has since determined the clays are predominantly kaolinite and illite. These veins or vein sets are subvertical and east-west oriented.

7.3.8 Sumatra

The Sumatra zone is located to the north of the Latte zone along the contact between the augen gneiss and biotite-feldspar schist. Mineralization occurs within two separate structures which underlie a broad ENE-trending soil anomaly. The first structure dips steeply (near-vertical) to the northwest, gradually reclining to an approximate dip of 60° through the middle portion of the corridor, and finally steepens in the eastern portion of the structure. The second structure strikes approximately E-W and dips roughly 70° to the south. These structures intersect at 583350mE, forming an hourglass-like structure.

Mineralization consists of strong disseminations of arsenian pyrite along relict schistose fabric in addition to clay-altered and heavily oxidized breccias. Multiple phases of brecciation are preserved in some intervals. Some mineralized intervals preserve schistose fabric, while others are heavily altered to the point of fabric destruction.

7.3.9 Arabica

Arabica is located to the east of the Supremo T-structures and is hosted within the augen gneiss panel. Mineralized structures run N-S and dip steeply (approximately 70°) to the east, with a current defined strike length of approximately 800 m. Mineralization is hosted within felsic gneiss and local, thin intervals of biotite schist. RC drilling during the 2013 field season identified strong limonite-hematite oxidation in addition to silicification within the mineralized intercepts.

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) from surface. As a result of this preferential weathering, oxidation is channelized along the structural corridors.

Intense oxidation is commonly observed at depths of more than 200 m below surface. The majority of the Supremo and Double Double deposits are strongly oxidized; however, at Latte oxidation is less pervasive, extending to about 125 m below surface in some areas. As a result, transitional facies material forms a larger proportion of the Latte deposit than in Supremo and Double Double.

Oxidation state of both mineralization and country rock has important implications for mineral resource evaluation, the quality of the rock mass and metallurgical processing. Thus systematic classification and re-logging of oxidation facies was undertaken in 2012 in order to ensure consistency and accuracy of drilling information. Re-logging utilized the existing logs, core photographs, and archived reverse circulation chips.

7.4.1 Cyanide Solubility Analyses

In 2013, a comprehensive sampling program was implemented in order to systematically quantify the cyanide solubility of gold on a sample-by-sample basis. In order to quantify the degree of oxidation within mineralized rocks at Coffee, a cyanide shake test was employed to measure the amount of cyanide soluble gold within a sample. The ratio of cyanide soluble (AuCN) to total gold (from fire assay) provides information regarding the degree of oxidation and, in general, an indication of the potential cyanide gold leach characteristics of the rocks.

Cyanide shake tests have been 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. In new drill holes, cyanide shake testing is performed on a sample-by-sample basis, providing a resolution of 1m in diamond drill core and of 1½ 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.

7.4.2 Oxide Categorization

Four oxide types or domains have been interpreted for the Coffee deposit as described below. The oxide zone is relatively consistent and supported but a large proportion of the data. The degree of oxidation is often highly variable in the two (upper and lower) transition zones as reflected by the oxidation percentage ranges listed below.

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

7.4.3 Three Dimensional Modelling of Oxidation Surfaces

The ratios of AuCN/total Au have been interpolated in the block model and are utilized in combination with qualitative (visual) estimates of the intensity of oxidation, to provide information regarding the depth and intensity of oxidation in the structural domains. The location and extent of the four oxide domains, described above, are interpreted in cross-section and linked together into a series of 3D domains. These domains are then used to define zones of similar oxidation properties.

7.5 Three Dimensional Modelling of Structure and Gold Mineralization

Interpreted gold mineralized structures and structural corridors were digitized using Micromine. Polygons outlining the mineralized trends were digitized on cross section, and then connected along strike between sections to create solid wireframe models. Two models were created based on different parameters as outlined below. The wireframe models were utilized during resource estimation as discussed in Section 13.

7.5.1 Structural Domain Wireframes

Broad structural domain wireframes were constructed on the basis of geological criteria: lithology, alteration, and arsenic and gold abundance. Rock codes indicative of structural deformation include fracture zones, breccia, hydrothermal alteration products, and the presence of intermediate-felsic dykes. The structural domains were interpreted on a section-by-section basis using Micromine.

The domains were extended beyond drilling information along strike by half of the drill spacing. For example, at Latte where spacing is typically 100 m, the wireframes were extended 50 m both east and west beyond the extent of drilling. Domains were extended vertically to a distance of half of the vertical separation of boreholes on section.

7.5.2 Gold Mineralization Wireframes

After modelling the structural domains, gold mineralization wireframes were created within the structural domain wireframes. Gold wireframes were modelled using a threshold of 0.3 g/t gold, and polylines were snapped to the drilling assay data.

In all gold zones except for Latte, the gold mineralization is continuous within the zone, resulting in negligible internal dilution included within the gold wireframes. However, for the Latte gold wireframe that is characterized by thicker zones of mineralization, 1 to 5 m of internal dilution (less than 0.5 grams per tonne gold) was allowed.

As for the structural domain wireframes, gold mineralization wireframes were extended halfway the drilling spacing beyond drilling information. Gold mineralization wireframes were not used for direct construction of the block model, but were used to help guide the 3D planes used to describe the overall trend of the gold mineralization. These planes were then used in turn to help orient search directions to relate samples during grade interpolation of the block model (see Section 14.3).

8.0 DEPOSIT TYPES

The Coffee project lies within an east-southeast–west-northwest trending tectono-magmatic domain that consists of a series of Cretaceous-age granitoid intrusions (Dawson Batholith and Coffee Creek Granite) as well as Yukon-Tanana terrane metamorphic rocks. This domain extends almost 200 km between Freegold Mountain to the southeast and Coffee to the northwest, and is sub-parallel to the larger Tintina gold belt in which it lies. The domain is host to a number of significant mineral deposits including porphyry systems (e.g., Casino and Nucleus), epithermal (e.g., Mt. Nansen), and orogenic gold systems (e.g., Golden Saddle, Boulevard, and Longline).

The gold mineralization found to date on the Coffee project is hydrothermal in origin and both structurally and lithologically controlled. Structures which cut schistose host rocks promote formation of disseminated Au-bearing arsenian pyrite through mica sulphidation, with associated strong dolomite-illite-clay alteration. Hydrothermal breccias host mineralization in arsenian pyrite with associated silica-sericite-clay alteration. Quartz veining is rare, although pre-mineralization quartz veins of probable Jurassic age are locally associated with zones of higher grade mineralization. Thin quartz+carbonate+Au-bearing arsenian pyrite veins are observed within the Latte zone. The gold mineralization is characterized by elevated arsenic and antimony. At Supremo, there is a weak positive correlation between gold and silver. At Latte, there is a weak positive correlation between gold-silver and gold-calcium. At Double Double, bismuth and antimony show a weak positive correlation with gold. At Kona, a weak positive correlation between gold and uranium, mercury and barium is observed.

Reduced intrusion-related gold deposits are a common style of gold system within the Tintina gold belt. These systems typically display anomalous Bi, W, and Te in addition to low salinity and CO_2-rich ore fluids, and are located proximal to reduced, alkaline, volatile-rich plutons (Hart and Goldfarb, 2005; Lang and Baker, 2001). Other salient features include low-grade auriferous sheeted vein systems within the pluton cupolas, and general low sulphide abundance within ore. Coffee does not appear to be a reduced intrusion-related gold deposit due to the lack of Bi, W, and Te anomalies, the lack of a pluton associated within mineralization, and generally high sulphide abundance.

At each of the deposits at Coffee, fine-grained arsenian pyrite is the dominant host for micron-scale gold mineralization. The Coffee system displays significant arsenic-antimony geochemical anomalies with local realgar and orpiment; however, the lack of mercury or thallium anomalies as well as the absence of carbonate host rocks in the Coffee tectonic stratigraphy eliminates the possibility of Coffee being a Carlin-type gold system (Emsbo et al., 2006).

Mineralization at the Supremo and Latte zones is accompanied by a weak gold-silver correlation; however any other significant gold-base metal associations are absent. This generally rules out epithermal gold systems although Coffee does exhibit some features such as chalcedonic quartz and open space fracture-fill in some mineralized intervals.

Similarities to epithermal mineralization are also exhibited in the gold-to-silver ratios observed at Coffee. These ratios range from 1:5 in the Latte Zone to 1:13 in the Supremo Zone, all with the notable absence of base metals. These ranges are consistent with those exhibited by both low sulphidation (base metal poor, 10:1 to 1:10 gold:silver; Sillitoe, 1993) and high sulphidation (1:2 to 1:10 gold:silver; White and Hedenquist, 1995) epithermal systems. High sulphide abundance is commonly observed on the Coffee project, most notably at the Latte zone. High sulphidation deposits are known to commonly be genetically related to and occur above porphyry copper or copper-gold systems (Arribas, 1995; Arribas et al., 1995; Sillitoe, 1973). The nearest known copper-gold porphyry system is Casino, located 30 kilometres from Coffee. However, due to the significant textural and metal association differences between the Coffee gold mineralization and typical epithermal mineralization, no association between Coffee and porphyry mineralization is implied and no geochemical correlation to assert this relationship has been identified.

Recent work in the Dawson Range has demonstrated that both the Boulevard prospect and the Golden Saddle deposit are orogenic gold systems (Allan et al., 2013; Bailey, 2013; and McKenzie et al., 2013). The Golden Saddle deposit is located approximately 40 km to the north of the Coffee property and contains a resource of 1.4 Moz Au (Weiershäuser et al., 2010). The deposit is hosted within metamorphic rocks of the Yukon-Tanana terrane similar to the Coffee project, with mineralization focused at the intersection of a north-striking thrust fault and east-trending Jurassic sinistral transpressional faults (Bailey, 2013). Alteration which accompanies gold-bearing pyrite mineralization consists of a quartz-carbonate-illite assemblage. Trace elemental associations include Au-Ag-Pb-S-Te. Mineralization was constrained by 187Re/187Os ages of 163-155 Ma for molybdenite which occurs in gold-bearing veins. This age is consistent with Jurassic regional uplift and exhumation, and is suggested to be representative of a post-peak orogenic gold event (Bailey, 2013).

The Boulevard prospect is located approximately 10 km to the southwest of the Coffee project. It consists of a sheeted vein system of gold-rich quartz-sulphide-carbonate veins and fault breccia hosted within mafic schist (McKenzie et al., 2013). Mineralization is constrained to 95.0 ± 0.4 Ma by 40Ar/39Ar cooling ages for hydrothermal sericite directly linked to gold mineralization. The deposit is modelled as a mid-Cretaceous orogenic gold system related to rapid exhumation of the Dawson Range following the intrusion of the Dawson Range batholith and Coffee Creek granite (McKenzie et al., 2013). Adjacent to Boulevard and just south of Coffee, the Toni Tiger molybdenum showing hosts quartz-molybdenite veins dated to 96-95 Ma. Fluid inclusion study of both Boulevard and Toni Tiger suggests that the two prospects are genetically related and formed from H_2O-CO_2-NaCl orogenic fluids of low salinity, between 279 and 310˚C and > 1kbar (McKenzie et al.,2013). Boulevard itself exhibits a distinctly mesothermal-orogenic signature due to the presence of sheeted quartz veins, adjacent molybdenite mineralization, and Au-As-Sb-(Pb-Zn-Cu) metal associations.

To the northwest of the Coffee project, the 93-92 Ma Longline deposit in the Moosehorn Range is structurally controlled and hosts high-grade (~30 g/t) gold mineralization within sheeted quartz veins (Joyce, 2002). The mineralization is controlled by shallowly east-northeast dipping, north-northwest striking brittle reverse fault structures which cut the Dawson Range batholith. Alteration associated with mineralization includes muscovite, sericite, Fe-carbonate, pyrite, arsenopyrite, clay, quartz, and tourmaline (Joyce, 2002). Gold-bearing quartz veins precipitated from a H2O-CO_2-CFl_4-NaCl + N_2 fluid of moderate salinity between temperatures of 260 and 300˚C, at a pressure of approximately 1.3-1.9 kbar and a depth of 5-7 km (Joyce, 2002).

Detailed study of the Latte gold zone suggests that Coffee is an epizonal, early-brittle stage orogenic gold deposit (Buitenhuis, 2014; Allan et al., 2013). The fluid responsible for mineralization at Latte is likely the cool (220-250˚C), shallow extension of the mineralizing fluid responsible for gold mineralization at Boulevard and molybdenite mineralization at Toni Tiger. A CO_2-rich fluid flowed through the region, powered by the anomalous geothermal gradient caused by the rapid unroofing of the Dawson Range in the mid-Cretaceous. This fluid formed sheeted quartz veins within the mesozonal domain represented by Boulevard and Toni Tiger, where the base metal and silica content of the fluid was depleted during vein formation. The fluid, now mostly depleted in base metals, travelled upwards in the system into the epizonal domain of the Coffee project, where it was controlled by the structural framework of the Coffee project fault system and reacted with favorable host rocks. The fluid travelled along brittle structures and deposited gold-rich arsenian pyrite within schistose rocks through sulphidation, and in high-energy pulses, formed gold-rich hydrothermal breccias (Buitenhuis, 2014). The timing of gold mineralization at Coffee is post-emplacement of the Coffee Creek granite, and is most likely, based on field, geochemical, and petrographic observations, to be syn-to-post mineralization at Boulevard (approximately 95 Ma) in age. The possibility exists, however, that Coffee could be related to a younger, unidentified mineralizing event. Geochronological confirmation is pending.

9.0 EXPLORATION

Kaminak has carried out exploration on the Coffee project over the course of five separate field seasons: 2009, 2010, 2011, 2012, and 2013. Exploration in 2009 consisted of soil sampling, trenching, mapping, prospecting, and a ground magnetic survey.

Exploration in 2010 followed up with the same activities in support of a 16,104 m core drilling program. The 2011 exploration campaign consisted of reverse circulation and diamond core drilling, trenching, geophysics, mapping, and soil sampling (Table 9-1).

Exploration work carried out from 2009 to 2012 is summarized in detail in the previous technical reports (2011, 2012, and 2013). Work completed during the 2013 field season is described below.

Table 9-1: Exploration Work Completed by Kaminak

Coffee Exploration Summary by Year					
Year	**Soil Samples**	**Trenching (m)**	**Mapping and Sampling (days)**	**Geophysics**	**Geomorphology**
2009	6,000	4,000	10	261 line-km ground magnetic survey	N/A
2010	9,473	4,000	10	579 line-km ground magnetic survey	N/A
2011	10,958	3,824	15	4,842 line-km airborne magnetic and gamma-ray spectrometric; 15.9 line-km HLEM and Ohm mapper surveys	Yes
2012	4,603	N/A	40	N/A	N/A
2013	5,047	169	2	18 days of Induced Polarization	N/A
Totals	**36,081**	**11,993**	**77**	N/A	N/A

9.1 Exploration Work by Kaminak in 2013

9.1.1 Soil Sampling

5,047 grid geochemical samples were collected in 2013 by Ground Truth Exploration Inc. of Dawson, YT to both follow up on previously sampled ridge and spur gold-in-soil anomalies, and provide greater detail to previous sample grids.

A total of nine new soil grids were sampled with 100 m sample spacing to cover anomalous ridge and spur anomalies. An infill sampling program was undertaken on the main contiguous soil grid in order to provide more detail to previously outlined anomalies. The main resource area was sampled between previous soil grid lines to decrease the sample spacing to 50 m. Macchiato and Cappuccino were also resampled to decrease the sample spacing to 25 m (Figure 9-1).

9.1.2 Trenching

Four trenches were dug in July, 2013: three at Supremo, dug east-west across T3 at 6974250mN, 6974375mN and 6974400mN, and one at Latte, dug north-south across the Latte structure at 583250mE (Figure 9-2). Trenches were dug to test geology and continuity of mineralization at surface and to provide sampling material for metallurgical testing. All trenches were laid out by Kaminak geologists and excavated by JDS' 320 Caterpillar excavator.

Trench 6974250mN

Trench 6974250mN was excavated to a length of 40 m. From 15 m to 25 m, the trench intersected 10 m at 4.08g/t Au in strongly to intensely silica+sericite±clay-altered felsic gneiss and dacite dyke.

Trench 6974375m

Trench 6974375mN was excavated to a length of 30 m. From 6 m to 17 m, the trench intersected 11 m at 20.05g/t Au in strongly to intensely silica+sericite±clay-altered silicified clast breccia, felsic gneiss and dacite dyke.

Trench 6974400m

Trench 6974400mN was excavated to a length of 21 m. From 9 m to 21 m, the trench intersected 12 m at 10.93g/t Au in moderately to intensely silica+sericite±clay-altered felsic gneiss and dacite dyke.

Trench 583250mE

Trench 6974400mN was excavated to a length of 78 m. From 10 m to 66 m, the trench intersected 56 m at 1.31g/t Au in moderately to locally-intensely silica+sericite-altered biotite schist.

The results from these trenches confirm that gold mineralization does extend to surface. The gold grades encountered in the Supremo T3 trenches tend to be higher in comparison to proximal drill holes. The trench at Latte produced similar thickness and grades in comparison to the resource model in this area. 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.

9.1.3 Mapping and Prospecting

Independent consultants Doug Mackenzie and Dave Craw conducted two days of geological mapping on the property.

9.1.4 Geophysical Surveys

Ground Truth Exploration Inc. of Dawson, YT conducted induced polarization surveys for 18 days.

Figure 9-1: 2013 Soil Geochemistry Program

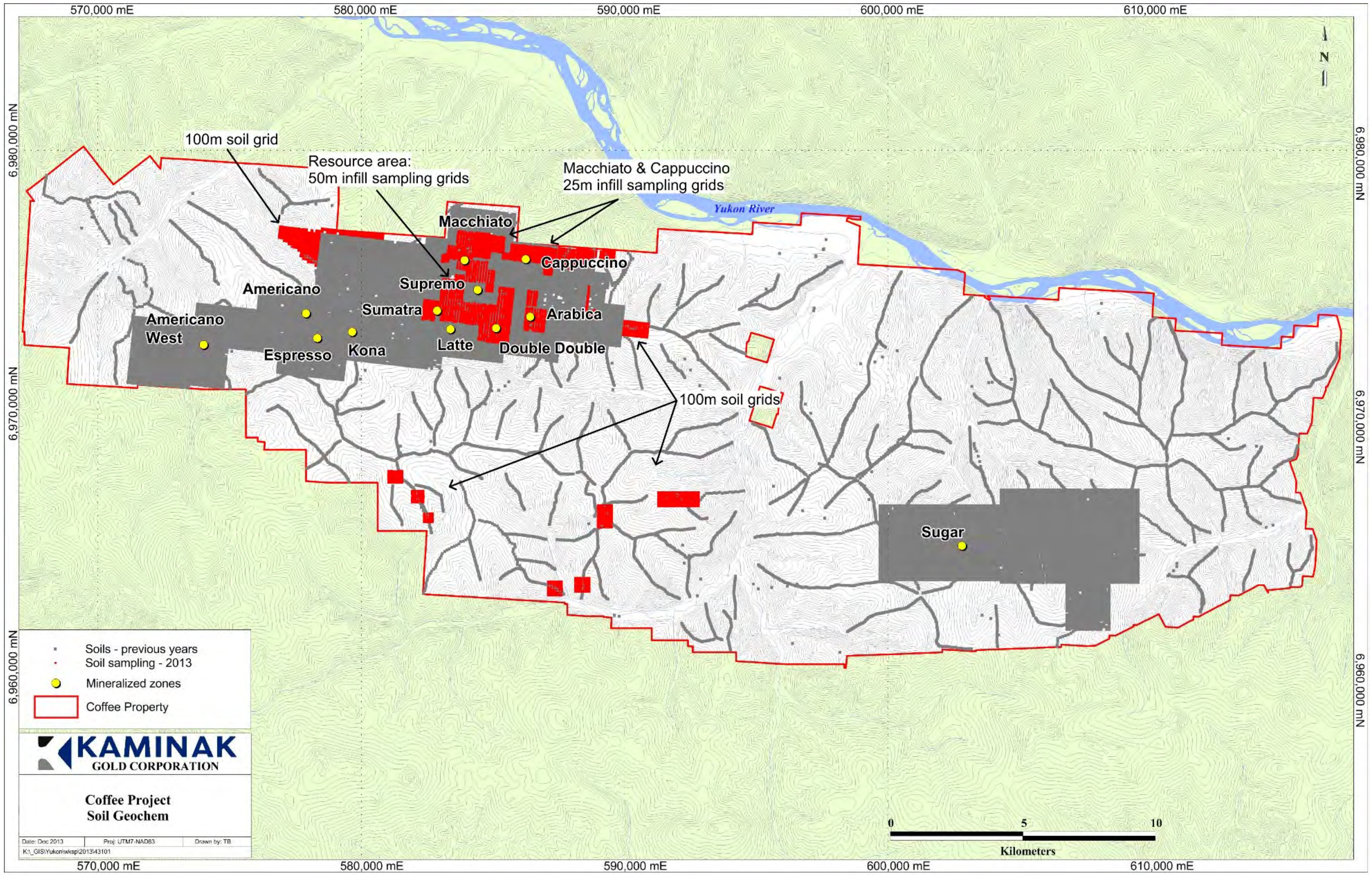
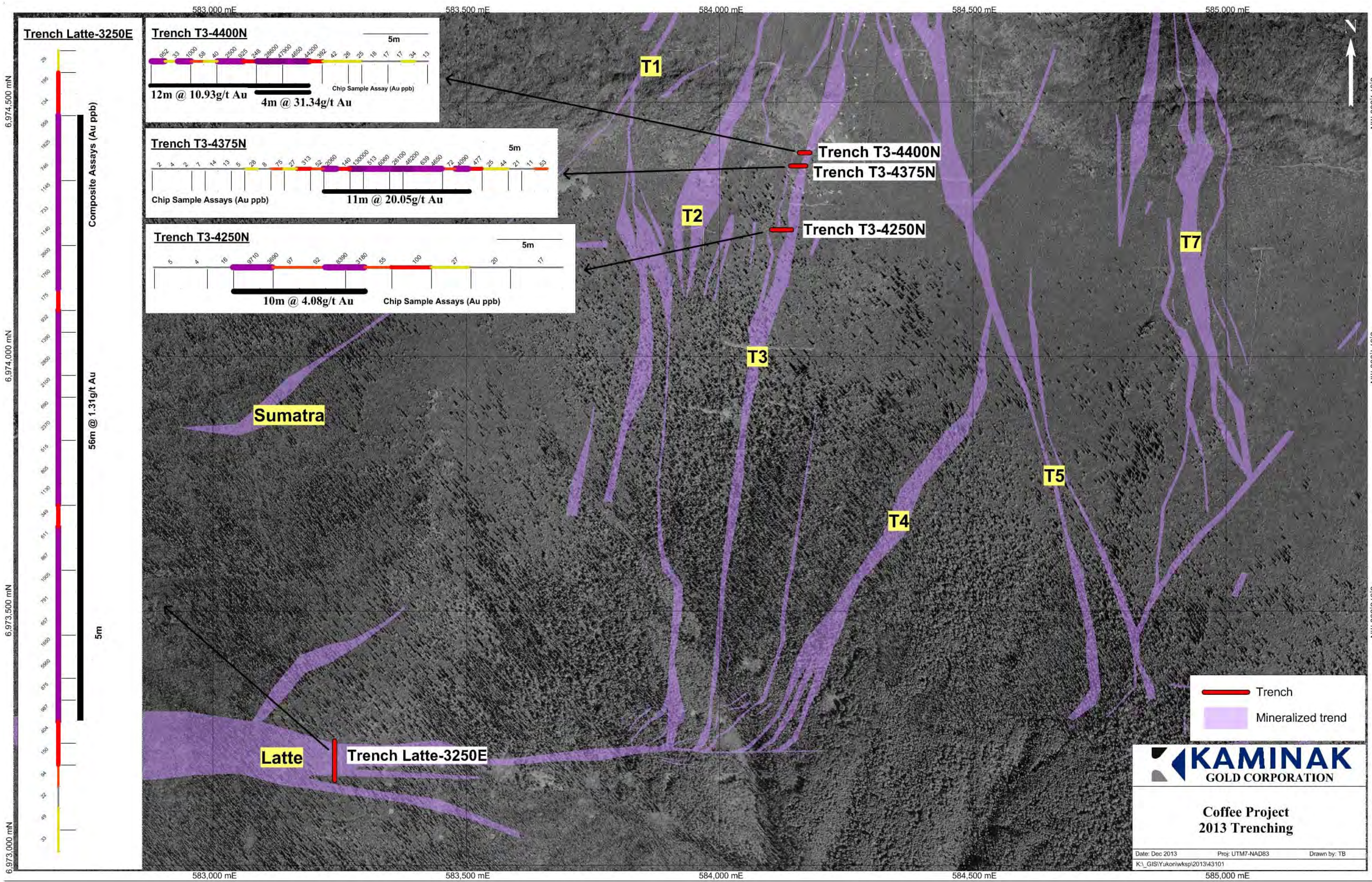


Figure 9-2: 2013 Trenching Program



9.2 Sampling Method and Approach

Sampling of geological materials completed by Kaminak during 2009 through 2013 was performed by experienced geological technicians under the supervision of appropriately qualified geologists. The following paragraphs summarize the sampling methodology and approach for the soil and rock chip samples.

9.2.1 Soil Sampling

The purpose of the soil sampling was to map the distribution of gold and associated metals in the soils with the hypothesis that gold (and other metals) in soil bears direct relationship with gold mineralization in bedrock that outcrops poorly over the project area.

Soil sampling was carried out by Ground Truth Exploration from Dawson City, Yukon. Soil samples were collected over a grid pattern of northerly directed lines spaced by 100 m with sampling stations spaced by 50 m. The exception to this orientation is the 2011 Sugar area sampling, in which the grid was rotated to easterly directed lines spaced by 100 m with sampling stations spaced by 50 m. In 2012, the Sugar area sampling returned to a pattern of northerly directed lines following the better understanding of mineralization distribution gained from soil sampling and trenching completed in 2011.

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

Samples were submitted by the contractor to Acme Analytical Laboratories in Vancouver, BC. The sample information was downloaded from the handheld computers into spreadsheets, and subsequently integrated into Kaminak's Coffee project database.

9.2.2 Rock Chip Sampling

In 2009 to 2012 trenches, rock samples were taken in trenches over 5 m horizontal intervals. Samples were collected by chipping sub cropping rock with a rock hammer on the wall or base of the trench over the desired interval taking care to collect a representative sample of the interval. Inherently, this selective sampling approach can introduce sampling bias, but the purpose of this sampling was to link gold-in-soil anomalous areas to outcropping or sub cropping bedrock and to define worthy drilling targets. In such circumstances, a positive sampling bias is generally desirable.

For 2013 trenching, heavy equipment was used to excavate trenches to the bedrock interface, where they were then ripped down into bedrock up to 0.5 m below the bedrock surface. Trenches were marked following preliminary excavation with metre marks, and the trench was subsequently re-entered to rip clean bedrock material on 2 m intervals which was then placed on the side of the trench for later metallurgical sampling.

Trenches 6974375mN and 6974400mN were chip sampled every metre from the bottom of each trench using a continuous chip sample along the trench base. Trench 6974250mN was too deep for the samplers to safely enter, and so was sampled from the clean metallurgical samples placed at the side of the trench. Trench 583250mE quickly filled with water following completion, and thus also had to be sampled from the metallurgical samples on the side of the trench.

The location of the centre of each sample was recorded using a handheld GPS unit. Other descriptive attributes and geological information about the sample were recorded into logging software on a daily basis and incorporated into the project database.

10.0 DRILLING

10.1 Sampling Method and Approach

Sampling of geological materials completed by Kaminak during 2009 through 2013 was performed by experienced geological technicians under the supervision of appropriately qualified geologists. The following paragraphs summarize the sampling methodology and approach for core and reverse circulation boreholes.

10.1.1 Drill Core Sampling

The drilling approach was to target the structural trends with fences of one or more core boreholes drilled perpendicular to the strike of the interpreted structures on variably spaced sections. Most sections received two to five boreholes per structure, designed to sample the targeted structures at different depths (typically to maximum depths of 200 m below surface). This strategy was used to provide maximum geologic information about each target. The approach was adjusted during drilling to allow for the testing of extensions of interesting geology, or assay results on adjacent sections. Rather than “fan” a series of holes from one set-up, Kaminak typically drills each hole from a unique set-up, resulting in a series of sub-parallel holes that often intersect at close to right angles with the target horizon. The resulting intersections generally represent “true” thickness through the zone.

Drill core was transported daily by truck or helicopter to the logging facility at the Coffee 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 at nominal 1 m intervals, as close to the metre mark as possible. Core was then logged by a geologist, recording lithology, alteration, structure, and mineralogy, directly into a data shed using a laptop computer. Core photographs were then 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. The remaining half was returned to the core boxes. Commercial blank and control (standard reference) samples were inserted at a rate of one 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 project camp for future reference and testing. Sample books provided by ALS Minerals were used to record borehole number, location, sampling interval, and date of sampling. All sample books are organized and archived at the Kaminak Vancouver office for future reference.

10.1.2 Reverse Circulation Chip Sampling

Reverse circulation drilling was completed on the Coffee project in 2010 through 2013. The drilling approach was similar to that employed for diamond core drilling. The drill holes were designed to target structural trends with structural pierce points designed to have a nominal vertical spacing of 25 m, and with individual fences spaced between 12.5 and 50 m apart, depending upon the level of geological confidence of the structural trend.

The reverse circulation drill works by driving compressed air through a pneumatic hammer attached to a semi-permeable bit, which acts like a jack-hammer. 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 5gallon pail. Each sample comprises one 5-foot run, with the borehole and rods being blown out (cleaned) between each run. The total volume of cutting is split through a 1:7 riffle splitter, into a sample typically measuring 2 kg in size, and the larger volume of reject material that is retained at the drill site in a series of individual retention bags. Sample chips are sieved from a spear sample of the retention bag and logged by the geologist on-site directly into a field laptop, which is in turn backed up digitally each night. Sample bags collected for analysis are transported daily by truck or helicopter to the processing facility at the Coffee project camp. Each sample is then analysed on the XRF instrument before being shipped to ALS Minerals for analysis.

10.2 2013 Drilling

During 2013, 302 boreholes (55,478 m) were drilled: 62 core boreholes (12,273 m) and 240 reverse circulation boreholes (43,205 m) at Supremo, Latte, Arabica and Sumatra (Table 10.1, Figure 10-1). Representative cross sections with interpreted structural zones and Au composites for Supremo and Latte are shown in Figure 10-2 and Figure 10-3.

10.2.1 Core Drilling

Core drilling took place between May and October 2013 and was contracted to Cyr Drilling International Ltd. of Winnipeg, Manitoba. One Boyles 37 drill rig was dragged between drill sites by excavator or bulldozer. All core drilled was NQ2 diameter (50.5 mm).

Borehole locations were planned and marked by Kaminak geologists using a handheld GPS. A compass was used to determine borehole azimuth and inclination. Boreholes were drilled at an angle of between 70 and 45° from the horizontal, depending upon the target. Downhole surveys were completed for all boreholes using a Reflex EZ-Shot® electronic single shot (magnetic) device. Downhole deviation of boreholes was measured using these tools at nominal 30 m intervals. Collar locations were surveyed following completion by Challenger Geomatics Ltd. of Whitehorse, YT with a Real Time Kinematic (RTK) GPS using five established control points.

Core retrieved from boreholes was moved from the drilling sites to the base camp by truck or occasionally by helicopter. At the camp, core was examined for consistency, re-assembled, and marked for orientation. RQD was measured by a trained technician. Core pieces were then selected for portable XRF analysis on 1 m intervals in mineralized rocks and on 2 m intervals in non-mineralized zones. Core was then described (logged) and photographed by a geologist and marked for sampling. Finally, SG measurements were recorded for each major lithology and for each potentially mineralized interval. All descriptive information was captured digitally on-site using a Microsoft Access database. Core samples were cut in half lengthwise and half of the core was sent for analyses at ALS Minerals in Whitehorse.

Diamond core recovery data is available for nearly all drill holes on the Coffee property. Global average core recovery is 95%, with 94% of all sample intervals demonstrating recoveries greater than 80%. Approximately 1% of sample intervals have recoveries less than 50%. There is no apparent relationship between drill core recovery and gold content at Coffee.

10.2.2 Reverse Circulation Drilling

RC drilling took place between March and October 2013 and was contracted to Northspan Explorations Ltd. of Kelowna, British Columbia. Two skid-mounted “Super Hornet” drill rigs were dragged between drilling sites by excavator or bulldozer. All RC boreholes were 92 mm diameter.

Reverse circulation chips were logged on-site by Kaminak geologists, prior to being transported back to the Coffee project camp by truck. At the camp, the sample bags were analysed by portable XRF prior to being shipped to the analytical laboratory for preparation.

No recovery data is available for reverse circulation drilling; however, personal site inspection indicates that recoveries are very good. While fine dust is lost to the air during drilling, this represents a very small amount of sample material and is not believed to affect sample integrity to a measurable degree. Sample retention bags were observed to show consistent sample size, which is indicative of constant sample recovery throughout the drilling process.

Figure 10-1: Drill Hole Plan

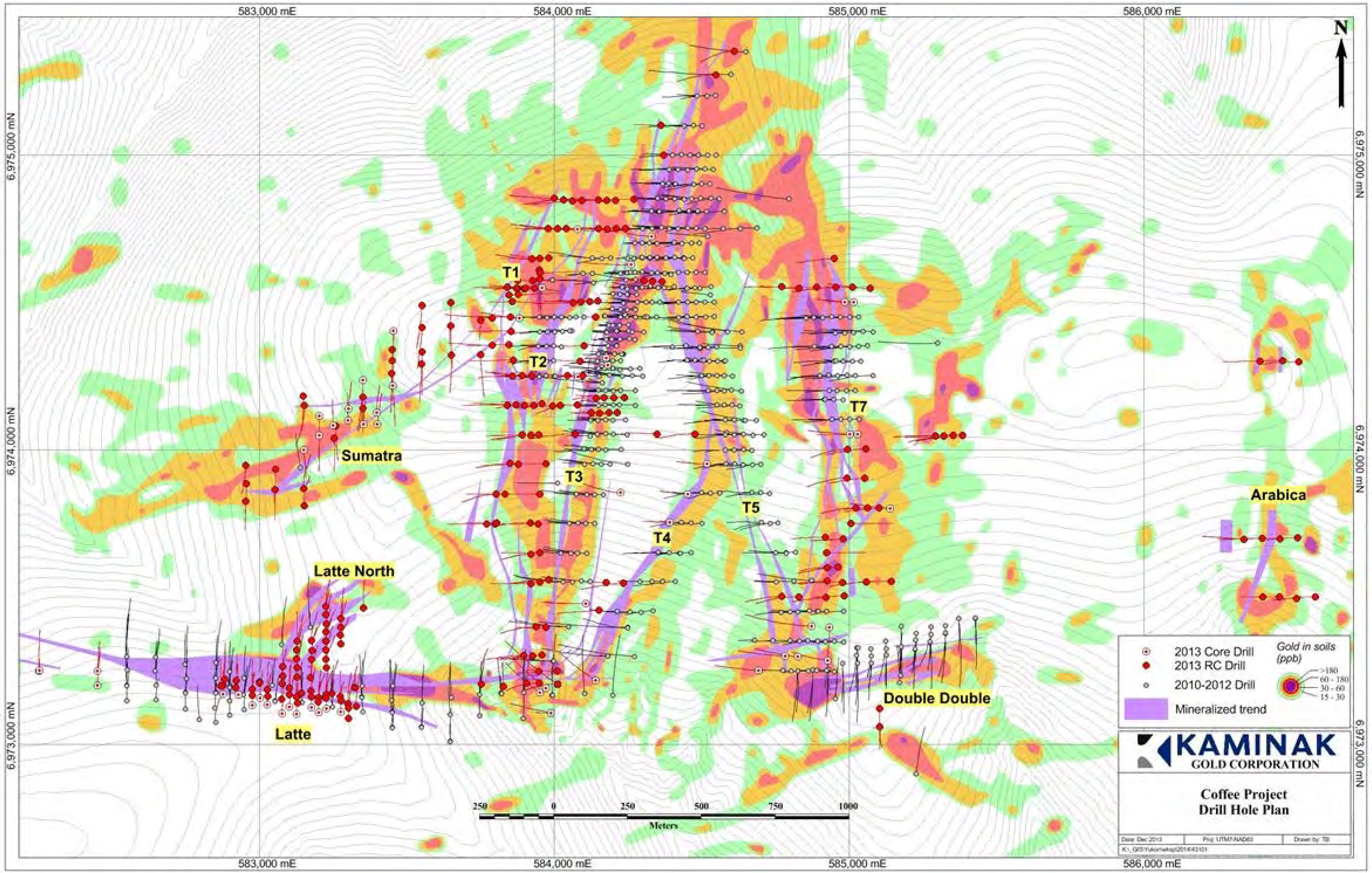


Figure 10-2: Drill Section through Supremo, 6974300N, Looking North

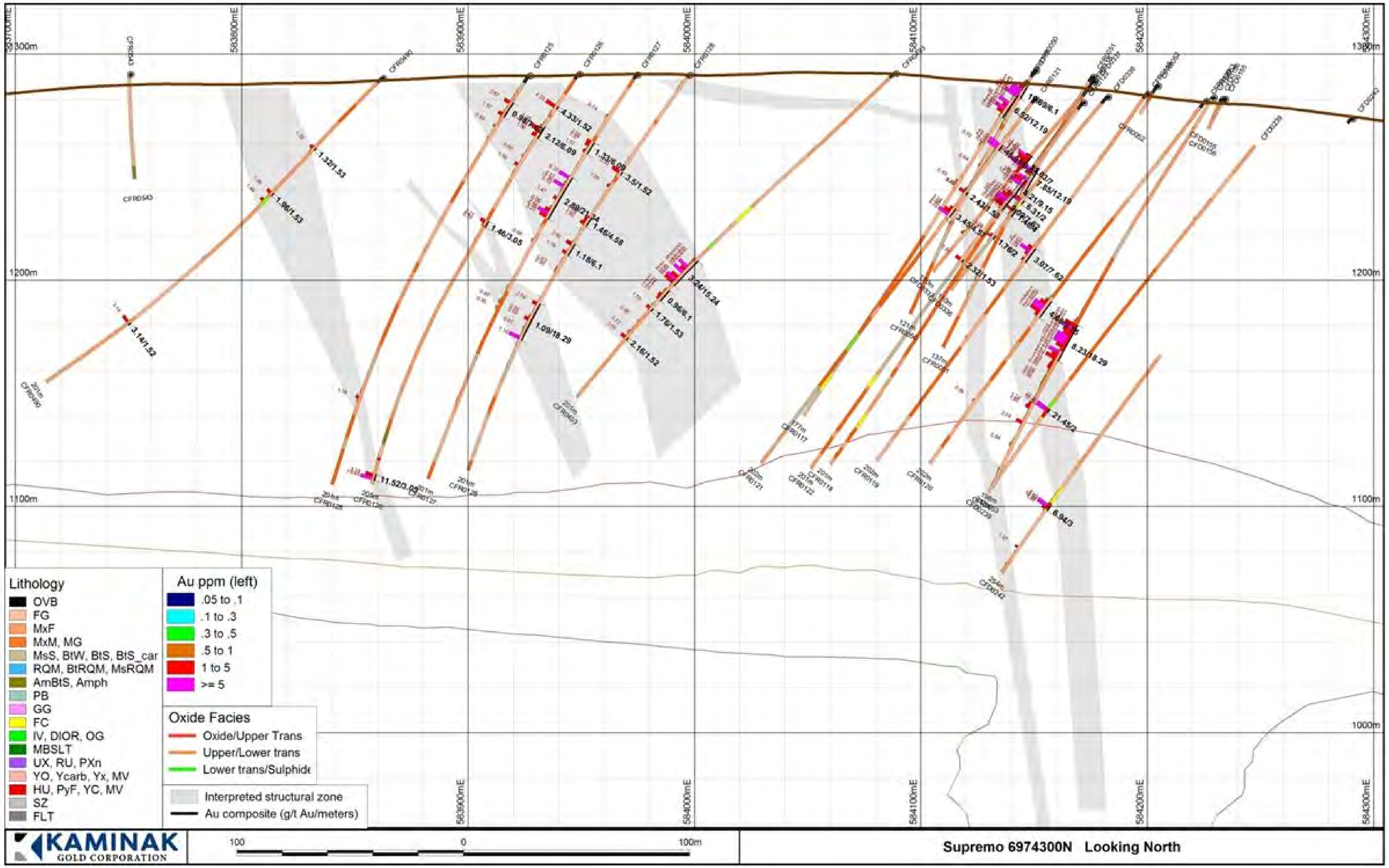
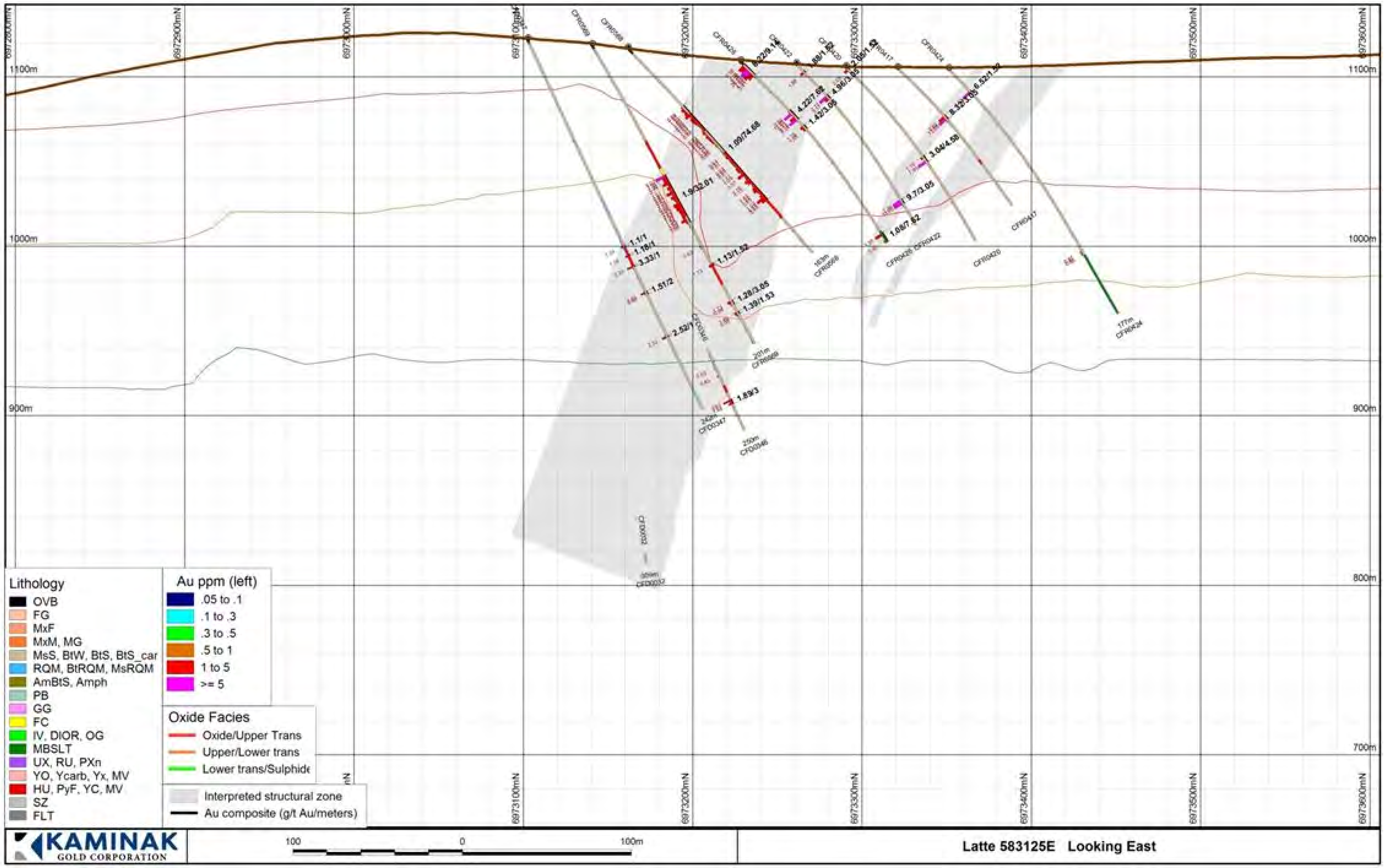


Figure 10-3: Drill Section through Latte, 583125E, Looking West



10.3 Drilling Summary

As of the end of the 2013 field season, a total of 961 drill holes for approximately 185,171 m of cumulative drilling have been completed at the Coffee project. A complete drilling summary by year, drilling method, and zone is provided in Table 10-2. In addition, tables detailing the top 25 gold intersections by diamond and reverse circulation drilling are provided (Table 10-2, Table 10-3).

Table 10-1: Coffee 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
2013 Summary		**Totals**	**302**	**55,478**
Coffee Project		**Property Totals**	**961**	**185,171**

Table 10-2: Selected Diamond Borehole Assays

Coffee Top Assay Results - Diamond Core					
Borehole ID	From (m)	To (m)	(m)	Gold (g/t)	Prospect
CFD0090	105	109	4	74.9	Double Double
CFD0001	15	30.5	15.5	17.07	Supremo T3
CFD0027	139	174	35	6.3	Double Double
CFD0228	129	135	6	36.55	Supremo T3
CFD0016	53	67	14	12.43	Supremo T3
CFD0210	31	42	11	15.52	Supremo T3
CFD0082	109	126	17	9.61	Latte
CFD0199	158	171	13	12.53	Supremo T3
CFD0215	64	68.5	4.5	34.95	Double Double
CFD0221	212	225	13	10.48	Supremo T3
CFD0342	120	202	82	1.65	Latte
CFD0205	51.5	55	3.5	36.29	Double Double
CFD0053	3.25	60	56.75	2.21	Kona
CFD0115A	142	201	59	2.11	Latte
CFD0340	89	124	35	3.52	Latte
CFD0080	60	67	7	17.37	Latte
CFD0298	6	50	44	2.69	Supremo T1-T2
CFD0234	226	240	14	8.26	Supremo T3
CFD0183	164	170	6	19.14	Supremo T3
CFD0261	304.5	328.5	24	4.33	Double Double
CFD0011	33	76	43	2.31	Latte
CFD0270	219	228	9	10.21	Supremo T4-5
CFD0303	95	111	16	5.67	Supremo T1-2
CFD0223	86	89	3	29.23	Supremo T3
CFD0044	98	137	39	2.23	Latte

Table 10-3: Selected Reverse Circulation Borehole Assays

Coffee Top Assay Results - Reverse Circulation					
Borehole ID	**From (m)**	**To (m)**	**(m)**	**Gold (g/t)**	**Prospect**
CFR0252	182.88	201.17	18.29	14.51	Supremo T4
CFR0124	120.7	131.37	10.67	19.56	Supremo T3
CFR0035	30.48	45.72	15.24	13.5	Supremo T3
CFR0276	24.38	32	7.62	21.9	Supremo T3
CFR0567	50.29	135.64-+	85.35	1.92	Latte
CFR0254	1.53	53.34	51.81	3.09	Supremo T4-5
CFR0053	117.04	135.33	18.29	8.23	Supremo T3
CFR0121	24.99	28.04	3.05	45.91	Supremo T3
CFR0563	38.1	86.87	48.77	2.85	Latte
CFR0565	67.06	121.92	54.86	2.44	Latte
CFR0050	8.23	14.33	6.1	19.89	Supremo T3
CFR0321	3.05	16.76	13.71	8.49	Supremo T5
CFR0144	82.3	114.3	32	3.62	Supremo T4-5
CFR0557	92.96	163.07	70.11	1.64	Latte
CFR0433	57.91	71.63	13.72	8.31	Latte North
CFR0576	6.1	70.1	64	1.71	Latte
CFR0027	18.29	27.43	9.14	11.61	Supremo T3
CFR0283	9.75	29.57	19.82	5.27	Supremo T5
CFR0286	102.11	114.3	12.19	8.33	Supremo T3
CFR0342	36.58	94.49	57.91	1.72	Supremo T5
CFR0051	46.94	59.13	12.19	7.85	Supremo T3
CFR0566	44.2	83.82	39.62	2.38	Latte
CFR0200	80.77	112.78	32.01	2.94	Supremo T4
CFR0408	102.11	114.3	12.19	7.23	Supremo T1-2
CFR0556	80.77	103.63	22.86	3.8	Latte

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Historical Sampling

Soil samples collected by Mr. Shawn Ryan in 2007 were analysed by Acme Analytical Laboratories (Acme) in Vancouver, BC. The management system of the Acme laboratory is accredited ISO 9001:2008 by BSI America Inc. Acme implements a quality system compliant with the International Standards Organization (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 round robin proficiency tests.

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

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

11.2 Sampling by Kaminak

Kaminak used two primary laboratories for assaying samples collected during the 2009 through 2013 programs.

Soil samples collected in 2009 through 2013 were submitted to the accredited Acme laboratory. The samples were prepared and assayed using the same methodology used to assay samples submitted by Mr. Ryan in 2007. Soil samples were prepared using standard preparation procedures and analysed for a suite of 36 elements using aqua regia digestion followed by ICP-AES on 15-gram subsamples (method code 1DX2).

All drill core, reverse circulation, trench, and grab samples collected in 2010 through 2013 were submitted to ALS Minerals for preparation and assaying. The management system of the ALS Group of laboratories is accredited ISO 9001:2000 by QMI Management Systems Registration. Samples were crushed and pulverised by the Whitehorse preparation facility and shipped to ALS North Vancouver for assaying. The North Vancouver laboratory is accredited ISO/IEC 17025:2005 by the Standards Council of Canada for certain testing procedures, including those used to assay samples submitted by Kaminak. ALS Minerals laboratories also participate in international proficiency tests such as those managed by CANMET and Geostats Pty Ltd.

All 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 City, then via road transport by expeditor or Kaminak personnel directly to ALS Minerals' preparation facility in Whitehorse. Samples were conveyed within rice sacks sealed by uniquely numbered security tags to minimize voluntary or inadvertent tampering. Security tags were tracked through the transport until receipt by ALS Minerals. No rice sacs were reported tampered with during 2010, 2011, 2012, or 2013.

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, pulverised 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-gram subsamples (50-gram samples were used in 2010). In 2010 and 2011 all samples were also submitted for a suite of 35 elements using an aqua regia digestion and ICP-AES finish on 5-gram subsamples. In 2012, samples from only select boreholes (54 boreholes in total) were submitted for the 35-element analysis. In 2013, 87 boreholes were submitted for the 35-element analysis.

In 2013 samples grading greater than 0.3 g/t gold were submitted for cyanide leach analysis. For this analysis, a 30-gram subsample is weighed into a closed 100 mL plastic vessel. 60 mL of sodium cyanide solution (0.25% NaCN, 0.05% NaOH) is then added and the sample is shaken until homogenized. Following homogenization, the solution is rolled for an hour before an aliquot is taken and centrifuged. Finally, the sample is analysed by atomic absorption spectrometry. 8,016 samples from all drilling programs (2010 through 2013 inclusive) were analysed by cyanide leach analysis during 2013.

Samples grading in excess of 10 g/t gold were re-assayed from a second 30-gram split (50-gram split in 2010) using a fire assay procedure and a gravimetric finish. In 2012 and 2013, samples grading in excess of 20 g/t gold was submitted for screened fire assay from a 1,000 gram coarse reject split. The screened fire assay was passed through a 100-micron mesh, with the oversize fraction (roughly four weight percent on Kaminak samples in 2012) undergoing gravimetric analysis following fusion, whereas the undersize fraction was split into two 50-gram samples and finished using atomic absorption. The average between the two minus fractions was then combined together with the plus fraction to give the total weighted average gold.

In 2010, samples assaying more than 100 g/t silver (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-gram charges. Two samples from 2011 reported more than 100 g/t silver, but were not re-assayed. No samples from 2012 or 2013 drilling returned greater than 100 g/t silver.

Approximately one in 100 master pulps from core and reverse circulation samples submitted to ALS Minerals in 2010, 2011, 2012, and 2013 were submitted annually at the conclusion of each exploration season to Acme Labs for umpire check assaying.

All zones drilled in a given year were represented in the check assay samples, and 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 ran greater than 0.3 g/t gold in order to provide an accurate test of lab performance and avoid running a large number of near-detection samples. Kaminak did not use an umpire laboratory to verify the assay results for soil samples delivered by Acme in 2009 through 2013.

In 2010, two composite core samples were submitted to the Inspectorate Exploration & Mining Services Ltd (Inspectorate) in Burnaby, British Columbia for preliminary metallurgical testing. In 2011, Kaminak submitted one additional composite core sample for follow-up heap leach column testing to the Inspectorate laboratory. The Inspectorate laboratory is part of the Veritas Bureau Group, which provides a wide range of testing services to the mineral industry. The Inspectorate laboratories are accredited to relevant national and international standards including ISO 17025. In 2012, Kaminak submitted additional core samples for further metallurgical testing by the Inspectorate laboratory.

In 2013, seven metallurgical composite samples were assembled from drill core and submitted to Kappes, Cassiday & Associates (KCA) in Reno, Nevada. Each composite was utilized for head analyses; head screen analyses with assays by size fraction, bottle roll leach test work, agglomeration test work, and column leach test work. In addition, some material from one composite was used for comminution and flotation test work, all at KCA. KCA has completed extensive metallurgical test work for northern projects including Kinross' Fort Knox project in Alaska and Victoria Gold's Eagle project in the Yukon.

11.3 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. In 2012 and 2013, 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.

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, and 26 samples in 2013 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.

Field specific gravity measurements indicate a mean of 2.61 from 5,307 samples representing all deposit areas. The standard deviation of the sample population is 0.160.

11.4 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of the exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management, and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation, and assaying processes. They are also important to prevent sample mix-up and monitor the voluntary or inadvertent contamination of samples. Assaying protocols typically involve regular duplicate and replicate assays and insertion of quality control samples. Check assaying is typically performed as an additional reliability test of assaying results. This typically involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

The exploration work conducted by Kaminak was carried out using a quality assurance and quality control program meeting industry best practices for early stage exploration properties. Standardized procedures are 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.

With the beginning 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. In 2013, Kaminak used eight standards, with certified assay values ranging from 0.268 to 9.38 g/t gold and one blank with a certified assay value of less than 0.01 g/t gold (Table 11-1). For 2011, 2012, and 2013 drill core samples, reverse circulation chip samples, and 2011 and 2013 trench samples, blanks and certified reference materials were alternated and inserted at a rate of one every ten samples. For 2010 rock samples, certified reference materials were inserted approximately at a rate of one every 30 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 and assigning a separate sample number out of sequence from the original samples. Reverse circulation field 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 analysed samples.

Table 11-1: Specifications of the Certified Control Samples Used by Kaminak in 2013

Reference Material	Gold (g/t)	Standard Deviation (g/t)	Number of Samples
CDN-BL-10	<0.01	-	2,178
CDN-GS-P3C	0.268	0.01	46
CDN-GS-P7H	0.791	0.03	415
CDN-GS-1J	0.968	0.06	62
CDN-GS-1L	1.18	0.05	390
CDN-GS-2K	1.97	0.10	384
CDN-GS-3L	3.15	0.25	29
CDN-GS-6D	6.08	0.22	408
CDN-GS-9A	9.38	0.32	442

11.5 Comments

The Qualified Persons reviewed the field procedures and analytical quality control measures used by Kaminak. The analysis of the analytical quality control data is presented in Section 12. 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 Persons, 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.

12.0 DATA VERIFICATION

12.1 Verification by Kaminak

The exploration work carried out on the Coffee 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 geologists. In the opinion of the Qualified Persons, the field exploration procedures used at Coffee generally meet industry practices.

The quality assurance and quality control program implemented by Kaminak is comprehensive and supervised by adequately 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 that season were managed by Kaminak personnel using Maxwell data management applications. In early 2011, the 2010 data were migrated to the 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). Re-assayed batches passed the quality control failure thresholds and were accepted. 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

12.2 Verifications by the Authors of this Technical Report

12.2.1 Site Visit

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 three separate occasions; September 12-14, 2011, August 28-29, 2012 and May 15-16, 2013. Each visit was similar in process and Mr. Sim was given full access to all aspects of the project and all questions were addressed in an open and professional manner.

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.

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 project in a very organized and professional manner that follows accepted industry standards.

12.2.2 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 2013 was discussed in previous technical reports (Couture and Siddorn, 2011, Couture and Chartier, 2012, and Couture et al., 2013) and is not reproduced here.

The authors of this report 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 analysed using bias charts, quantile-quantile and relative precision plots. The analytical quality control data produced by Kaminak in 2013 are summarized in Table 12-2). The external quality control data produced on this project represents 13.2% of the total number of samples (Table 12-2) submitted for assaying in 2013.

Table 12-2: Summary of Analytical Quality Control Data Produced by Kaminak in 2013

	Reverse Circulation Samples	(%)	Core Samples	(%)	Total	(%)	Comment
Sample Count	28,056		10,411		**38,467**		
Blanks		5.66%		5.69%		**5.66%**	
CDN-BL-10	1,586		592		**2,178**		<0.01 g/t Au
Reference Material	1,585	5.65%	591	5.68%	**2,176**	**5.66%**	
CDN-GS-P3C	46		0		**46**		
CDN-GS-P7H	297		118		**415**		
CDN-GS-1J	62		0		**62**		
CDN-GS-1L	285		105		**390**		
CDN-GS-2K	280		104		**384**		
CDN-GS-3L	14		15		**29**		
CDN-GS-6D	293		115		**408**		
CDN-GS-9A	308		134		**442**		
Field Duplicates	487	1.74%	243	2.33%	**730**	**1.89%**	
Total QC Samples	**3,658**	**13.04%**	**1,426**	**13.70%**	**5,084**	**13.21%**	
Check Assays							
Acme Labs	249	0.89%	153	1.45%	**402**	**1.04%**	Umpire Lab Testing

In general, the performance of the control samples (certified reference materials including both blanks and reference material) inserted with samples submitted for assaying used by Kaminak is acceptable (example presented in Figure 12-1a). ALS Minerals delivered assay results for the certified reference materials within two standard deviations of the mean for all eight reference material tested and less than the recommended value for the one blank, with few exceptions.

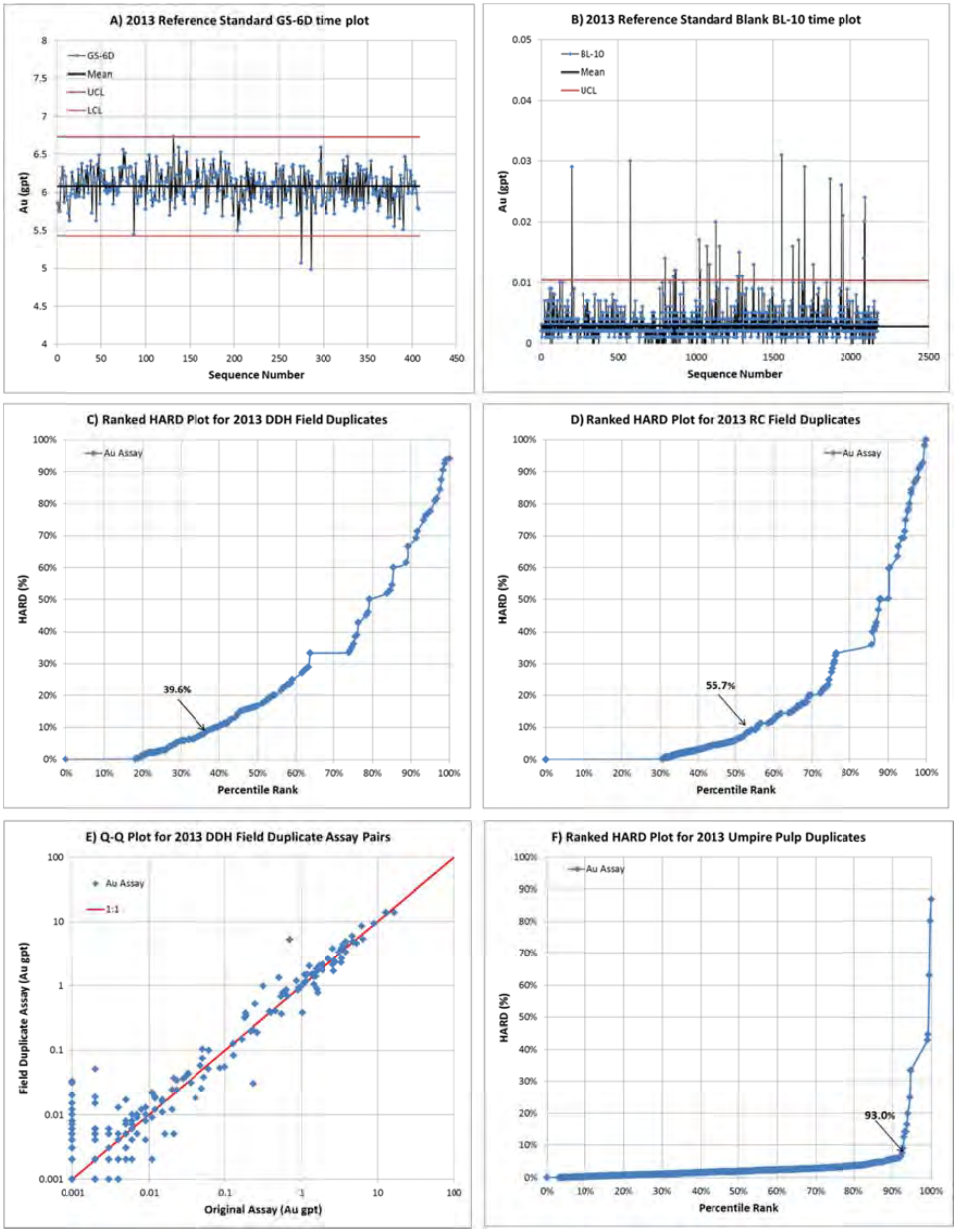
Assay results delivered by ALS Minerals of reference material reported less than 6% of the results falling outside of two standard deviations of the certified values for each standard. Few other potential failures identified in the data examined by the authors of this report can be related to sample mislabelling. Approximately only 1% of blanks returned assay values above 0.01 g/t gold (Figure 12-1b).

Paired assay data for field duplicates produced by ALS Minerals and examined by the authors of this report suggest that gold grades are difficult to reproduce. Rank half absolute difference (HARD) plots suggest that only 39.6% of the core field duplicate sample pairs and 55.7% of the reverse circulation field duplicate sample pairs have HARD below 10% (Figure 12-1c and Figure 12-1d). The poor reproducibility of field duplicate results is common in structurally hosted gold deposits. In general, however, the reproducibility is worse nearing the detection limits, as expected (Figure 12-1e).

Results from umpire laboratory testing in 2013 indicate good reproducibility and no significant deviation or bias in results between labs. HARD plots suggest 93.0% of the check assay sample pairs have HARD below 10% (Figure 12-1f).

In the opinion of the Qualified Persons, the analytical results delivered by ALS Minerals are sufficiently reliable to support mineral resource evaluation.

Figure 12-1: 2013 Selected QAQC Plots



12.2.3 Database Verification

Following the completion of the mineral resource models, the sample data from 15 randomly selected drill holes from the 2013 program, representing approximately 5% of the data, was exported from the MineSight® for validation purposes. The gold grades were manually compared to the values listed in certified assay certificates provided from the lab. Of the 1,964 samples checked, five samples had values that differ from the certified value. None of these errors are of any significance and are likely the results of multiple analysis of the same sample. This is well within the “acceptable” error rate of 1%. The results of the manual database verification indicate that the database is sound and sufficiently reliable to support the estimation of mineral resources.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testing 2011 - 2012

In 2011 and 2012 Kaminak commissioned SRK Consulting Inc. (SRK) to supervise preliminary metallurgical testing on core sample rejects collected on the Coffee Project. The metallurgical testing work was conducted by Inspectorate Exploration & Mining Services Ltd. ("Inspectorate") of Richmond, British Columbia. Inspectorate conducted preliminary cyanide leaching tests including bottle roll, carbon in leach and carbon in pulp, and column leach. John Starkey, P.Eng. of Starkey & Associates Inc., an SRK associate metallurgist, supervised the testing program. Metallurgical test work carried out from 2010 to 2012 is summarized in detail in the previous technical reports (2011, 2012, and 2013). Work completed in 2013 is described below.

In 2013 Kaminak engaged Kappes, Cassiday, and Associates (KCA) to undertake comprehensive metallurgical testing on core and bulk samples from the Coffee deposit. KCA has issued two (2) reports which present the results of their testing in detail. Much of the information presented below is summarized from the KCA reports. KCA has reviewed this report and agrees it presents an accurate summary of the test work.

13.2 Core Drillhole Composite Metallurgical Testing 2013 by Kappes, Cassiday and Associates

13.2.1 Summary of Metallurgical Test Work

In June 2013, the laboratory facility of Kappes, Cassiday & Associates (KCA) in Reno, Nevada received sample material from the Coffee Project of Kaminak Gold Corporation. A total of seven composites were generated and assigned unique sample numbers (KCA Sample Nos. 68151 through 68157). Each composite underwent head analyses; head screen analyses with assays by size fraction, bottle roll leach test work, agglomeration test work and column leach test work. Additionally, portions of material from KCA Sample No. 68157 (Latte, Sulphide composite) were utilized for comminution test work and flotation test work.

All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures.

13.2.2 Sample Receipt and Preparation

A summary of the samples utilized for each composite is presented in Table 13-1.

Table 13-1: Summary of Received Core Composite Samples

KCA Sample No.	Sample Description	Number of Individual Samples within Composite	Number of Holes Sampled	Received Weight, kg
68151	Supremo, Oxide	150	16	310.8
68152	Supremo, Upper Transition	130	12	127.7
68153	Supremo, Lower Transition	112	6	123.5
68154	Latte, Oxide	128	13	324.3
68155	Latte, Upper Transition	99	8	137.4
68156	Latte, Lower Transition	96	7	129.1
68157	Latte, Sulphide	73	1	99.5
Total:		**788**		**1252.4**

The samples are believed to adequately represent the major known mineralization areas currently defined.

13.2.3 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. 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 select crush sizes were utilized for head screen analyses with assays by size fraction.

For the Latte, Sulphide composite (KCA Sample No. 68157), 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 13-2.

Table 13-2: Coffee Project Summary of Head Analyses – Core Composites

KCA Sample No.	Description	Average Assay, Au (g/t)	Weighted Avg. Head Assay, Au (g/t)
68151	Supremo, Oxide	1.461	1.500
68152	Supremo, Upper Transition	1.227	1.482
68153	Supremo, Lower Transition	1.569	1.597
68154	Latte, Oxide	1.488	1.555
68155	Latte, Upper Transition	1.479	1.453
68156	Latte, Lower Transition	1.656	1.309
68157	Latte, Sulphide	2.469	2.333

Note (1): Weighted average assay value is the average of two (2) head screen analyses.

13.2.4 Flotation Test Work

Portions of material from the Latte, Sulphide composite (KCA Sample No. 68157) were utilized for a two-phase kinetic flotation test program. The test work consisted of reagent scoping test work (Phase 1), followed by grind optimization test work (Phase 2) utilizing the results from the reagent scoping tests. Flotation tests were conducted in a laboratory-scale Denver flotation apparatus utilizing Reno municipal tap water. The products from each flotation test were individually assayed for gold, silver, copper, lead and total sulphur.

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. Utilizing the various reagent schemes, the reagent scoping tests showed that between 62% and 69% of the gold was concentrated into between 8.0% and 8.9% of the sample weight.

Utilizing the results from the reagent scoping test work, a total of 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. At the various grind sizes, the grind size optimization tests showed that between 58% and 72% of the gold was concentrated into between 9.8% and 10.5% of the sample weight.

13.2.5 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 was completed to provide Bond Rod Mill and Ball Mill Work indices for the sample. The results of the comminution test work were Rod Mill Work Index of 12.73 kwh/tonne and a Ball Mill Work Index of 15.06 kwh/tonne.

13.2.6 Bottle Roll Leach Test Work

Bottle roll leach testing was conducted on a portion of material from 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 utilized 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 extractions from the bottle roll leach test work is presented in Table 13-3.

Table 13-3: Summary of Bottle Roll Leach Test Work – Core Composites

Description	Head	Calculated	Extracted Au	Consumption	Addition
Supremo, Oxide	1.461	1.436	94	1.29	1.50
Supremo, Upper Transition	1.227	1.447	78	2.12	1.00
Supremo, Lower Transition	1.569	1.639	53	1.45	1.00
Latte, Oxide	1.488	1.570	92	1.27	1.50
Latte, Upper Transition	1.479	1.369	51	1.15	1.50
Latte, Lower Transition	1.656	1.462	38	1.57	1.50
Latte, Sulphide	2.469	2.460	13	1.35	1.50

13.2.7 Agglomeration Test Work

Preliminary agglomeration test work was conducted on portions of crushed material from each composite except the Latte, Sulphide composite (KCA Sample No. 68157). The purpose of the percolation tests was to examine the permeability of the material under various cement agglomeration levels. The percolation tests were conducted in small (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 or 8 kg cement per tonne of dry heap leach feed.

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 or 8 kg cement per tonne of dry heap leach feed.

All agglomeration tests passed the criteria put forth by KCA. It was determined from this test work that the column leaching would be undertaken without the use of agglomeration. The pH of the material was low in the tests which did not utilize cement.

Table 13-4 presents a summary of the agglomeration testing.

Table 13-4: Summary of Agglomeration Tests – Core Composites

Description	Target p80 Size, mm	Cement, kg/t dry HL feed	Initial Height, cm	Final Height, cm	Slump, %	Slump Result	Flow Result	Visual Estimate of % Pellet Breakdown	Pellet Result	Overall Test Result
Supremo, Oxide	25	0	27.94	27.94	0%	Pass	Pass	N/A	N/A	Pass
	25	2	29.21	29.21	0%	Pass	Pass	5	Pass	Pass
	25	4	28.58	28.58	0%	Pass	Pass	<3	Pass	Pass
	25	8	28.58	28.58	0%	Pass	Pass	<3	Pass	Pass
Supremo, Oxide	12.5	0	28.58	28.58	0%	Pass	Pass	N/A	N/A	Pass
	12.5	2	29.85	29.85	0%	Pass	Pass	<3	Pass	Pass
	12.5	4	29.21	29.21	0%	Pass	Pass	<3	Pass	Pass
	12.5	8	29.85	29.85	0%	Pass	Pass	<3	Pass	Pass
Supremo, Upper Transition	12.5	0	27.31	27.31	0%	Pass	Pass	N/A	N/A	Pass
	12.5	2	29.21	29.21	0%	Pass	Pass	5	Pass	Pass
	12.5	4	28.58	28.58	0%	Pass	Pass	<3	Pass	Pass
	12.5	8	29.21	29.21	0%	Pass	Pass	<3	Pass	Pass
Supremo, Lower Transition	12.5	0	26.67	26.67	0%	Pass	Pass	N/A	N/A	Pass
	12.5	2	27.94	27.94	0%	Pass	Pass	5	Pass	Pass
	12.5	4	29.21	29.21	0%	Pass	Pass	<3	Pass	Pass
	12.5	8	27.94	27.94	0%	Pass	Pass	<3	Pass	Pass
Latte, Oxide	25	0	26.04	26.04	0%	Pass	Pass	N/A	N/A	Pass
	25	2	27.31	27.31	0%	Pass	Pass	5	Pass	Pass
	25	4	29.21	29.21	0%	Pass	Pass	<3	Pass	Pass
	25	8	28.58	28.58	0%	Pass	Pass	<3	Pass	Pass
Latte, Oxide	12.5	0	24.77	24.77	0%	Pass	Pass	N/A	N/A	Pass
	12.5	2	27.31	27.31	0%	Pass	Pass	10	Pass	Pass
	12.5	4	28.58	28.58	0%	Pass	Pass	<3	Pass	Pass
	12.5	8	29.21	29.21	0%	Pass	Pass	<3	Pass	Pass
Latte, Upper Transition	12.5	0	26.67	26.67	0%	Pass	Pass	N/A	N/A	Pass
	12.5	2	26.04	26.04	0%	Pass	Pass	5	Pass	Pass
	12.5	4	26.67	26.67	0%	Pass	Pass	<3	Pass	Pass
	12.5	8	27.31	27.31	0%	Pass	Pass	<3	Pass	Pass
Latte, Lower Transition	12.5	0	24.13	24.13	0%	Pass	Pass	N/A	N/A	Pass
	12.5	2	24.13	24.13	0%	Pass	Pass	5	Pass	Pass
	12.5	4	24.77	24.77	0%	Pass	Pass	<3	Pass	Pass
	12.5	8	25.4	25.4	0%	Pass	Pass	<3	Pass	Pass

13.2.8 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 in an enclosed refrigeration unit at a target 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 13-5. For each group of samples (Supremo and Latte), a graphical presentation of gold extraction over time is presented in Figure 13-1 and Figure 13-2.

Table 13-5: Summary of Column Leach Test Work – Core Composites

Description	Target p80 Size (mm)	Target Temp. (°C)	Calculated Head Au (g/t)	Extracted Au (g/t)	Extracted Au (%)	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)
Supremo, Oxide	25	4	1.573	1.455	92	0.17	1.51
Supremo, Oxide	12.5	4	1.435	1.343	94	0.28	1.5
Supremo, Oxide	12.5	22	1.547	1.471	95	0.52	1.57
Supremo, Upper Transition	12.5	4	1.488	1.081	73	0.31	1
Supremo, Lower Transition	12.5	4	1.674	0.797	48	0.38	1
Latte, Oxide	25	4	1.622	1.462	90	0.19	1.51
Latte, Oxide	12.5	4	1.54	1.382	90	0.27	1.51
Latte, Upper Transition	12.5	4	1.535	0.717	47	0.46	2.01
Latte, Lower Transition	12.5	4	1.416	0.411	29	0.64	1.51
Latte, Sulphide	12.5	4	2.365	0.126	5	0.46	1.51

Figure 13-1: Supremo Core Composites – Column Leach Tests Gold Extractions

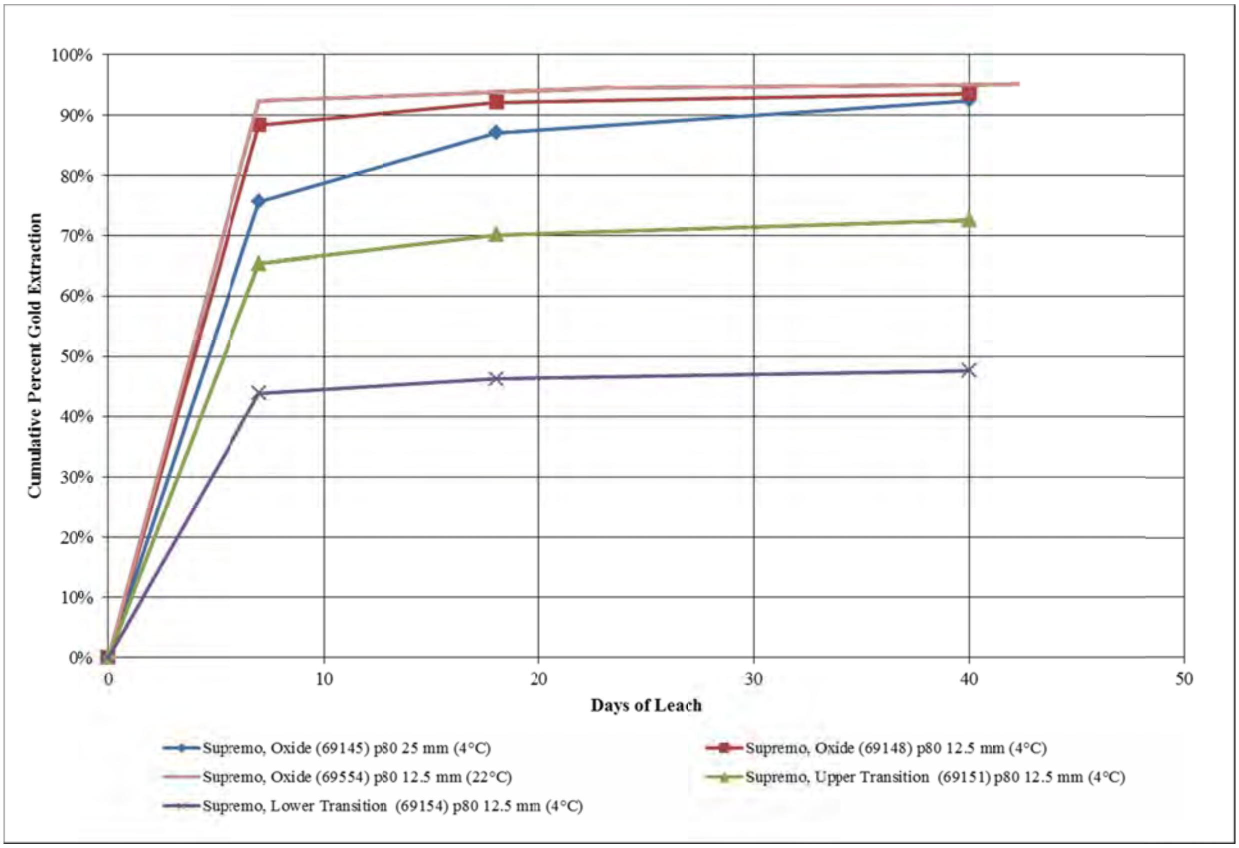
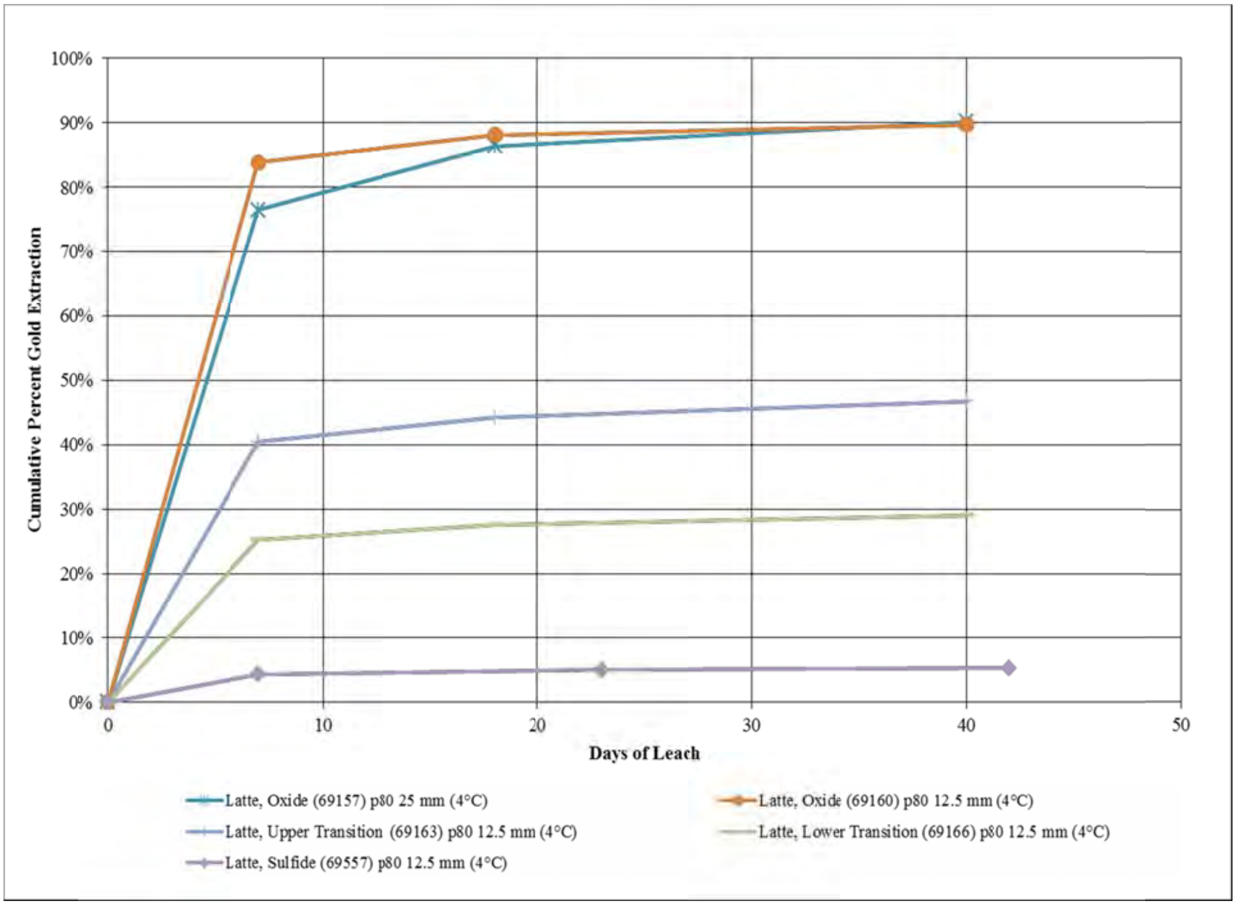


Figure 13-2: Latte Core Composites – Column Leach Tests Gold Extractions



13.2.9 Discussion

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

For the Supremo, Oxide material, three column leach tests were conducted to compare the extraction values of material leached at two particle sizes (80% passing 25 and 12.5 mm) and leach temperatures (4°C and 22°C). A comparison of gold extractions from the 25 and 12.5 mm leached material showed an increase from 92 to 94%, with respect to particle size reduction. A similar comparison of the material leached at 4°C and 22°C, showed a gold extraction increase from 94 to 95% and a sodium cyanide consumption increase from 0.28 to 0.52 kg per metric tonne of HL feed, with respect to increased temperature. Based on KCA's experience, sodium cyanide consumption is not directly related to test temperature and any observed increase in consumption is likely an abnormality.

For the Latte, Oxide material, two column leach tests were conducted to compare the extraction values of material leached at two different particle sizes (80% passing 25 and 12.5 mm). A comparison of gold extractions from the 25 and 12.5 mm leached material did not show any obvious gold extraction increase with respect to particle size reduction. Gold extraction percentages were at 90% for both particle sizes.

Column test extraction results were based upon carbon assays vs. the calculated head (carbon assays + tail assays). For the column leach tests conducted, the calculated heads based on carbon and solution assays compared well with each other.

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 feasibility study purposes, KCA normally discounts laboratory gold extractions by two to three percentage points when estimating field extractions. Based upon KCA's experience with mostly clean non-reactive ores, cyanide consumption in production heaps would be only 25 to 33% of the laboratory column test consumptions.

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 correlation of lower gold extractions with greater sulphide sulphur content. A summary comparing gold extraction and sulphur speciation is presented in Table 13-6. For each sample group, a graphical presentation of gold extraction versus sulphide content is presented in Figure 13-3 and Figure 13-4.

Table 13-6: Column Leach Test Gold Extraction vs. Sulphur Speciation

KCA Sample No.	Description	Column Extracted, % Au	Total Sulphur, %	Sulphide Sulphur, %	Sulphate Sulphur, %
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

Figure 13-3: Supremo Core Composites Gold Extraction vs. Sulphide Content

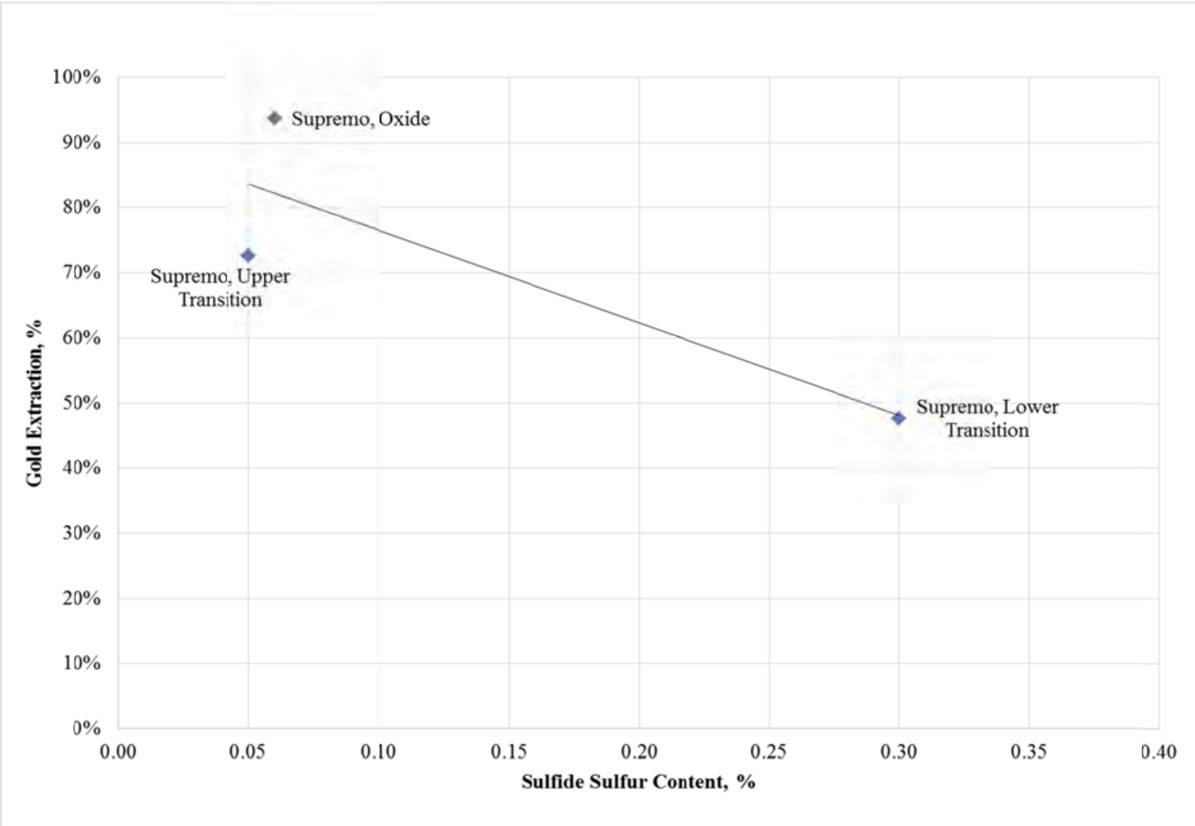
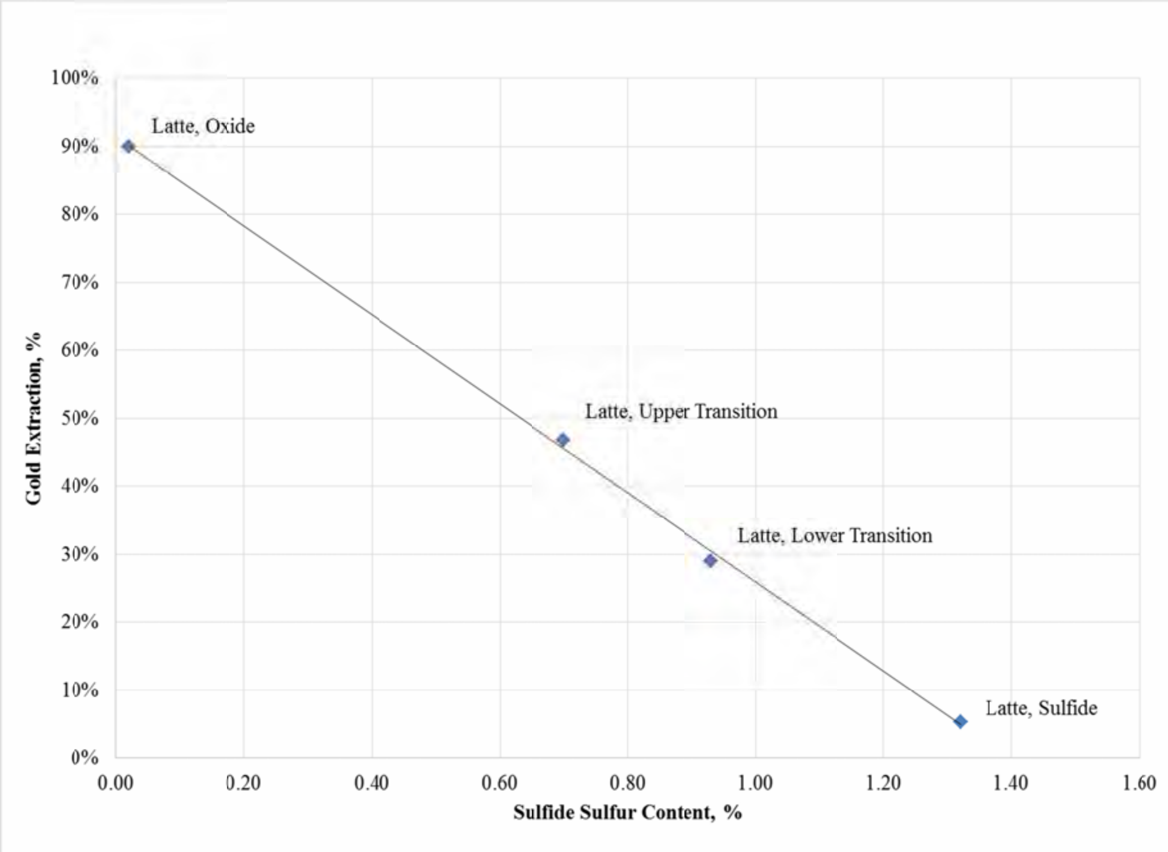


Figure 13-4: Latte Core Composites Gold Extraction vs. Sulphide Content



13.3 Cyanide Soluble Gold Test Work

Categorization of oxidation has in the past been undertaken via visual estimation of the proportion of Oxide and Sulphide. The oxidation profile at Coffee is variable from surface downwards and along the mineralized structures. Thus, the manual interpretation method is unlikely to be detailed enough to accurately assess the quantity and distribution of oxidation. 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 13-6 above although the visual estimate of the amount 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 poorer recoveries from the Latte Transitional material are felt 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 initial drilling year of 2010 up to and including 2013, have been subjected to a cyanide soluble assay. 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 utilized to provide an indication of the gold within the sample that is amenable to cyanide leach. By extension, it also indicates the amount 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 at the 0.5 inch crush size from the KCA testing program, and the cyanide soluble assays from the same samples as used in the testing composites, is presented in Table 13-7 below.

Table 13-7: Cyanide Soluble Recovery versus Column Leach Recovery

Sample Description	Gold Recovery %		Recovery Ratio
	Column Test	Cyanide Soluble	Column Test:
Supremo, Oxide	94%	98.4%	0.96
Supremo, Upper Transition	73%	78.2%	0.93
Supremo, Lower Transition	48%	51.2%	0.94
Latte, Oxide	90%	91.2%	0.99
Latte, Upper Transition	47%	46.5%	1.00
Latte, Lower Transition	29%	32.5%	0.89

The strong 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 Bulk Sample Metallurgical Testing 2013 by Kappes, Cassiday, and Associates

13.4.1 Sample Composite Selection for the Bulk Sample Metallurgical Test Program

The Supremo Oxide sample was collected from a surface trench across the T3 mineralized structure at 6974250mN. The Latte Oxide sample was collected from a surface trench across the Latte mineralized structure at 583250mE. Trenches were excavated to bedrock and sampling was completed on 2m intervals. The Supremo Oxide sample contained 59 individual sample intervals and the Latte Oxide sample contained 58 sample intervals. A total of approximately 1500 kg of material was collected from each trench and was sent to KCA where it was homogenized prior to splits being taken for the testing program.

13.4.2 Summary of Metallurgical Test Work

In September 2013, the laboratory facility of KCA in Reno, Nevada received sample material which comprised material from the 2m sample intervals collected from Supremo Oxide and Latte Oxide. For each sample group, the bulk interval material was combined into a composite sample. Each composite sample was assigned a unique sample number (KCA Sample Nos. 69580 (Latte) and 69581 (Supremo)) and prepared and utilized for head analyses, head screen analyses with assays by size fraction, bottle roll leach test work, agglomeration test work and column leach test work. All preparation, assaying and metallurgical studies were performed utilizing accepted industry standard procedures.

13.4.3 Head Analyses

For each sample, portions of the head material were ring and puck pulverized and analyzed for gold and silver 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. 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 select crush sizes were utilized for head screen analyses with assays by size fraction. A summary of the head analyses for gold is presented in Table 13-8.

Table 13-8: Summary of Head Analyses – Bulk Samples

KCA Sample No.	Description	Average Assay, Au (g/t)	Weighted Avg* Head Assay, Au (g/t)
69580	Latte Oxide	1.608	1.148
69581	Supremo Oxide	4.080	4.373

*: Values are the avg. of two (2) head screen analyses with assays by size fraction

13.4.4 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 gram portion of head material was ring and puck pulverized to a target size of 80% passing 0.075 millimetres. A 1,000 gram 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 grams sodium cyanide per litre of solution.

A summary of the gold extractions from the bottle roll leach test work is presented in Table 13-9.

Table 13-9: Summary of Bottle Roll Leach Test Work – Bulk Samples

KCA Sample No.	Description	Crush Size p100 (mm)	Calculated Head Au (g/t)	Extracted Au (%)	Days of Leach	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
69580	Latte Oxide	175	1.28	88	100	0.56	1.01
69580	Latte Oxide	31.5	1.122	92	100	1.08	1
69581	Supremo Oxide	175	4.313	85	152	0.91	1.52
69581	Supremo Oxide	31.5	3.438	92	100	0.93	1.51

13.4.5 Agglomeration Test Work

Preliminary agglomeration test work was conducted on portions of crushed material from each sample. The purpose of the percolation tests was to examine the permeability of the material under various cement agglomeration levels. The percolation tests were conducted at a range of cement levels with no compressive load applied utilizing 2 kilogram portions of material crushed to the target size of 100% passing 31.5 millimeters. All agglomeration tests passed the criteria put forth by KCA.

13.4.6 Column Leach Test Work

A total of four (4) column leach tests were conducted utilizing composite sample material crushed to target sizes of 100% passing 175 or 31.5 millimeters. The column tests utilizing material crushed to 100% passing 175 millimeters were conducted at ambient temperature (22°C). The column tests utilizing material crushed to 100% passing 31.5 millimeters were conducted in a walk-in freezer maintained at a target temperature of 4°C. A summary of the column leach test extractions for gold are presented in Table 13-10.

Table 13-10: Summary of Column Leach Tests – Bulk Samples

KCA Sample No.	Description	Crush Size, p100 mm	Calculated Head, Au (g/t)	Au Extracted (%)	Days of Leach	Consumption NaCN, (kg/t)	Addition $Ca(OH)_2$ (kg/t)
69580	Latte Oxide	175	1.280	88	100	0.56	1.01
69580	Latte Oxide	31.5	1.122	92	100	1.08	1.00
69581	Supremo Oxide	175	4.313	85	152	0.91	1.52
69581	Supremo Oxide	31.5	3.438	92	100	0.93	1.51

13.4.7 Discussion

A comparison of the direct and calculated head grades for gold from the various parts of this test program showed overall agreement with some variability. For the column leach test work, gold extractions ranged from 85% to 92% based on calculated heads which ranged from 1.122 to 4.313 grams per tonne. The sodium cyanide consumptions ranged from 0.56 to 1.08 kg/t. The material utilized in leaching was blended with 1.00 to 1.52 kg/t hydrated lime.

For each sample, a comparison of the column leach test results showed a higher gold extraction percentage for the column leach test conducted utilizing material leached at a finer crush size.

From the head screen versus tail screen analyses, comparisons of the approximate gold extractions from similar size fractions showed approximate variations of 1% (KCA Sample No. 69580) and 2% (KCA Sample No. 69581). The small variations indicate that there are no obvious differences in leaching kinetics between tests conducted at 4°C versus those conducted at ambient temperature (22°C).

The QP has deemed that the samples collected and the test work done are appropriate and sufficiently representative for this preliminary-level of study. There are no processing factors or deleterious elements that have been identified that could have a significant effect on the potential economic extraction. The use of heap leaching methods in a northern environment has been proven at several mines and has been planned and scheduled appropriately in this study.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The Mineral Resource Statement presented herein represents the second mineral resource evaluation prepared for the Coffee project in accordance with the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101).

The mineral resource estimation process was a collaborative effort between Kaminak Gold Inc. (Kaminak) and SIM Geological Inc. (SIM Geological). The interpretation of the geologic model was prepared by Kaminak personnel and was reviewed by SIM Geological and used as resource domains to constrain grade estimation. 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. Based on his education, work experience that is relevant to the style of mineralization and deposit type under consideration, and to the activity undertaken, and based on the membership to a recognized professional organization, Mr. Sim, is a Qualified Person pursuant to National Instrument 43-101, and independent from Kaminak. The effective date of the Mineral Resource Statement is January 28, 2014.

This section of the technical report describes the resource estimation methodology and summarizes the key assumptions considered by SIM Geological to prepare the resource model for the gold mineralization at the Coffee project. In the opinion of the Qualified Persons, the resource evaluation reported herein is a reasonable representation of the gold mineralization found in the Coffee 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* and is reported in accordance with the 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® v8.20). The project limits are based on the local UTM coordinate system (NAD83 Zone7). The block size varies between deposit areas: 5 x 5 x 2 m at Kona and Double Double, and increasing to 10 x 5 x 3 m at Latte and Supremo. The long axis of the blocks is aligned with the strike of the zone, and the shorter dimension is aligned across the strike direction. The database was developed by Kaminak during exploration programs conducted during the summer field seasons of 2010 through 2013.

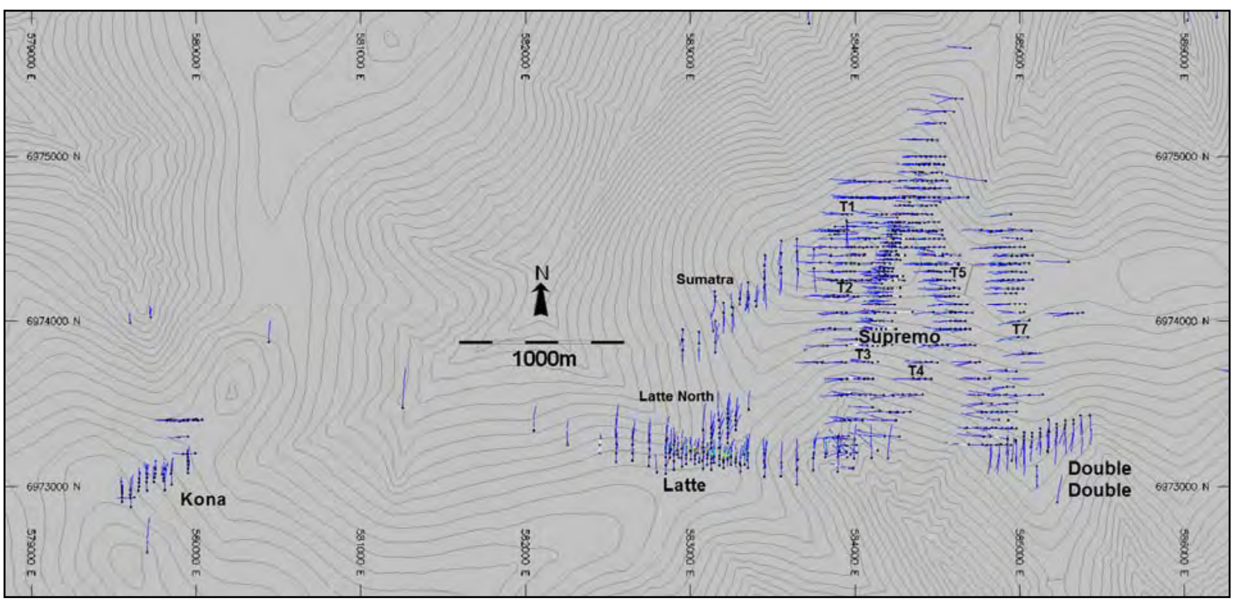
Mineral resource estimates are generated using drill hole sample assay results and the interpretation of a geologic model that relates to the spatial distribution of gold in the deposits. Interpolation characteristics were defined based on a combination of the geology, drill hole spacing, and geostatistical 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* (November 2010).

14.2 Available Data

There are a total of 961 individual drill holes in the project database with a total of 185,171 m of drilling; 352 holes (82,977 m) are diamond drill core holes and 609 holes (102,194 m) were drilled using reverse circulation drilling rigs.

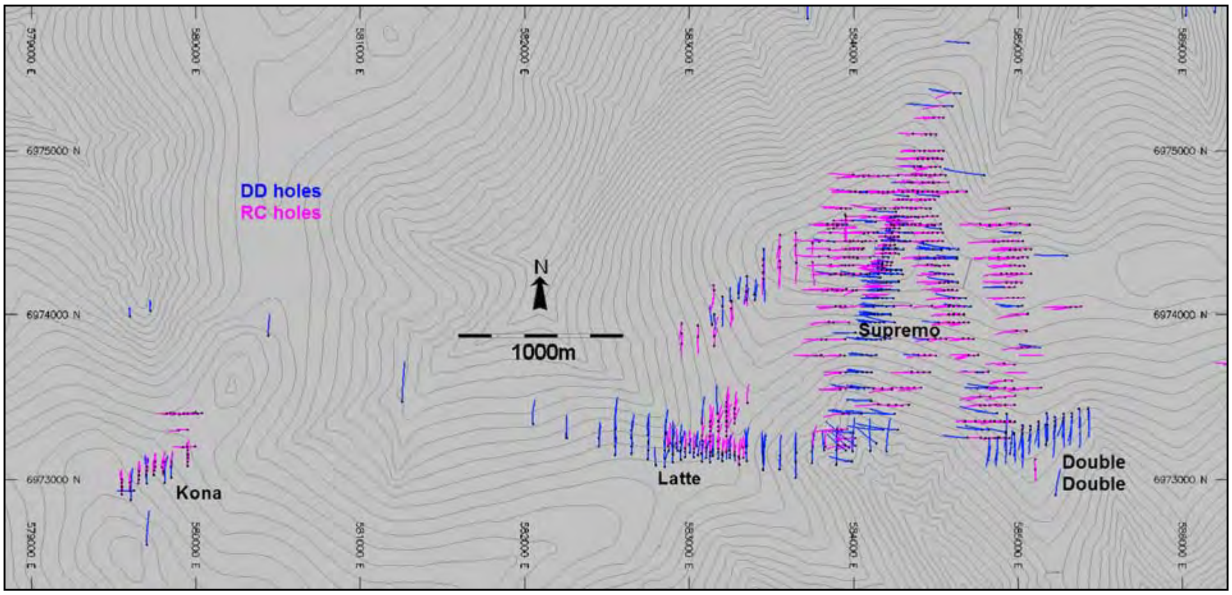
The majority of the drilling is conducted with holes located on north-south or east-west oriented cross sections and is designed to intersect the interpreted mineralized zones at right angles. Where holes are fanned from a single setup, the pierce angles between drill holes, and the typically steep-dipping target horizons, become smaller with depth. In such cases, Kaminak will often drill parallel holes on-section from individual setup locations. Overall, drilling has been 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



Analysis of gold assay data shows that there is no apparent grade distribution difference between diamond drill and reverse circulation samples. The distribution of diamond drill and reverse circulation holes is shown in Figure 14-2. There are areas where DD holes are more prevalent (Latte, Double Double and Supremo T3) and other areas the RC drilling has been more widely utilized.

Figure 14-2: Distribution of Diamond Drill (DD) and Reverse Circulation (RC) Drill Holes



The project database includes resource delineation drilling plus other drill holes that test surrounding exploration targets. Only drill holes that intersect the structural zone domains in each of the deposit areas have been considered for use in the resource models. A summary of the drill holes used in the resource models for the four deposit areas is listed in Table 14-1. Drilling at the south end of T7 during 2013 intersected a new zone that trends in an east-west direction and has been interpreted to represent a northern extension of the Double Double deposit.

Table 14-1: Summary of Drilling Used in Each Model Area to Estimate Mineral Resources

Deposit	**Number of Holes**	**Drilling (metres)**
Supremo	596	101,540
Latte	192	41,112
Double Double	52	13,327
Kona	39	6,273
Total	**581**	**116,329**

The majority of the drilling was conducted on cross sections oriented north-south or east-west and designed to intersect at approximately right angles to the strike orientation of the mineralized zones.

The majority of drilling at Double Double and Kona was conducted on north-south sections, spaced at 50 m intervals. The majority of on-section holes intersect the target horizon at 25 to 50 m intervals down the dip plane.

Drilling at Latte was initially conducted on north-south sections spaced at 100 m intervals, with pierce points at 50 m intervals along the dip plane. During the 2013 drilling program, the central portion of the Latte deposit was delineated with 50 m spaced drill holes on north-south sections spaced at 25 m intervals. This distribution of drilling increases the confidence in the resource estimate allowing for the designation of a portion of the Latte resource in the Indicated category.

Additional drilling identified two new southwest-northeast trending zones proximal to the Latte deposit; Latte North located immediately north of the main Latte zone and Sumatra, located approximately 700 m north of Latte.

Drilling at Supremo is conducted on east-west -oriented cross sections, typically spaced at 50 m intervals, with pierce points spaced at 25 to 50 m intervals on each section. For a strike distance of almost 550 m in the central part of the T3 zone, detailed drilling was conducted on sections spaced at 25 m with on-section holes at 25 m intervals. This density of drilling allows for a portion of the T3 resource to be included in the Indicated category.

The section spacing increases to 100 m, with on-section pierce points at 25 m intervals, at the north end of T3, and the southern ends of T3, T4, and T5. Rather than fan multiple holes from single setups, most drill holes at Supremo have unique setup locations that result in parallel holes that consistently intersect the target horizon at approximately right angles.

At the end of each drilling campaign, the drill hole collar locations are surveyed using a differential GPS. The collar location of each drill hole correlates very well with the local digital terrain (topographic) surface.

Although elevated arsenic values can often identify the structural zones in drilling, only the fire assay (total) gold data has been 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 sample data for each deposit area is presented in Table 14-2.

Table 14-2: Statistical Summary of Gold Assay Data

Element	Count	Total Length (metres)	Minimum (Au g/t)	Maximum (Au g/t)	Mean[(1)]	Std. Dev.
Supremo	78,735	104,683	0.001	86.800	0.214	1.501
Latte	35,131	40,018	0.001	48.700	0.253	1.264
Double Double	18,796	12,955	0.001	120.250	0.226	2.600
Kona	5,927	6,209	0.001	36.500	0.211	0.996

(1) Statistics are weighted by sample length.

During the 2013 field season, Kaminak added a series of 8,344 samples tested for cyanide soluble gold (AuCN) analysis using ALS method Au-AA13 (cold cyanide shake test). These samples were conducted on pulp rejects and target fire assay grades greater than 0.3g/t gold. The locally sparse distribution of these samples, coupled with the fact that they target only higher-grade material, does not support the ability to directly estimate AuCN grades in model blocks. However, ratios of AuCN/total Au were 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 includes lithologic designations obtained during geologic logging of the drill core and reverse circulation chips. Surface geologic mapping has provided the location of the structures on surface. Kaminak provided a topographic digital terrain surface as a gridded point file (x, y, z) that was originally produced using contour lines spaced at 10-metre intervals. This data was originally derived from a LiDAR survey of the property conducted by Eagle Mapping in 2010.

Individual sample intervals range from 0.1 to 7 m in length and average 1.23 metres. The standard sample interval for a diamond drill hole is 1 m, except at Double Double where 2012 drilling was sampled on 0.5 m intervals. Reverse circulation drilling is sampled on 1.52 m intervals.

Bulk density measurements were conducted for 5,328 samples in the database. Specific gravity measurements are typically made at 10 m intervals down most of the diamond drill holes. The frequency of specific gravity measurements may be increased within the structural zones.

Recovery data is available for essentially all diamond drill holes with an average of 95%. Ninety-four percent of the sample intervals show recoveries greater than 80%, and about 1% of sample intervals have recoveries below 50%. There is no apparent relationship between recovery and gold content. Recovery data is not available for reverse circulation drilling. Personal site inspection of the procedures indicates that recoveries are very good. There is a loss of very fine dust 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 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 recoveries.

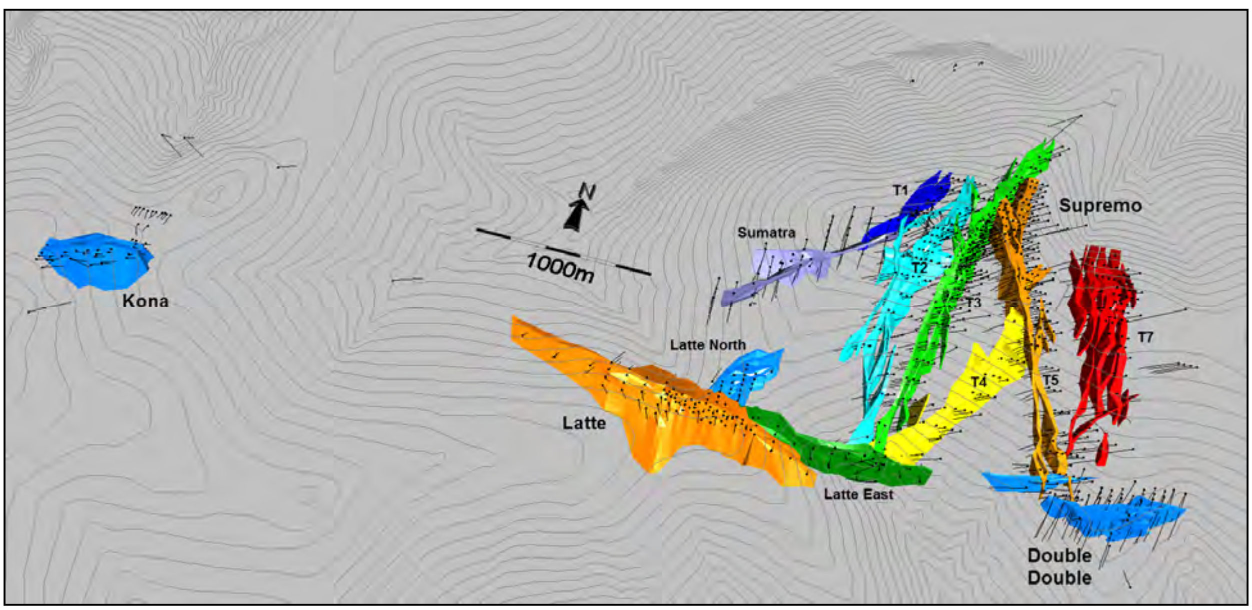
14.3 Geologic Model and Estimation Domains

Gold mineralization at Coffee 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 favourable horizons in many areas which have been 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 with regularity over strike lengths greater than 2 km.

A series of structural domains have been interpreted in each resource area using a combination of surface mapping, geologic core (and reverse circulation chip) logging, and the distribution of gold grades in drilling sample data. These structural domains represent the known geologic conditions that have the potential to host gold mineralization. In addition, Kaminak geologists have developed a more detailed interpretation within each structural domain that represents the interconnected nature of the (generally) higher-grade gold mineralization. Although it is believed that the gold mineralization is interconnected between drill holes, the detailed interpretation typically isolates only the higher-grade samples and represents a somewhat optimistic selection of the data between drill holes. Using the larger structural domains is a more appropriate approach to developing a resource model which includes some degree of internal dilution in the estimate. This may be considered a more conservative approach but, as the project evolves and the density of drilling data increases, the nature and continuity of gold mineralization is reflected in the block model.

The extent of these structural domains is shown in Figure 14-3. 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: Structural Domains at Supremo, Latte, Double Double, and Kona



Each deposit area is comprised of a series of sub-parallel, braided structural domains that coalesce and bifurcate along the general strike-orientation of the zone. Individual structural zones have been sub-divided for modeling purposes, and, within each zone, a three-dimensional plane was interpreted that represents the overall trend of the gold mineralization. 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 that reproduces the locally complex, undulating, and banded nature of the gold mineralization in the block model that would otherwise be impossible to achieve using traditionally-oriented search ellipses. The overall distribution of gold in the block model is similar to the detailed interpretation domains, but, as previously stated, some degree of internal dilution has been incorporated into the process.

Figure 14-4 shows the individual zones defined at Supremo, Latte, and Double Double. Figure 14-5 shows the trend planes defined for each individual structural zone at Supremo, Latte, and Double Double.

Figure 14-4: Individual Structural Zone Domains Defined at Supremo, Latte, and Double Double

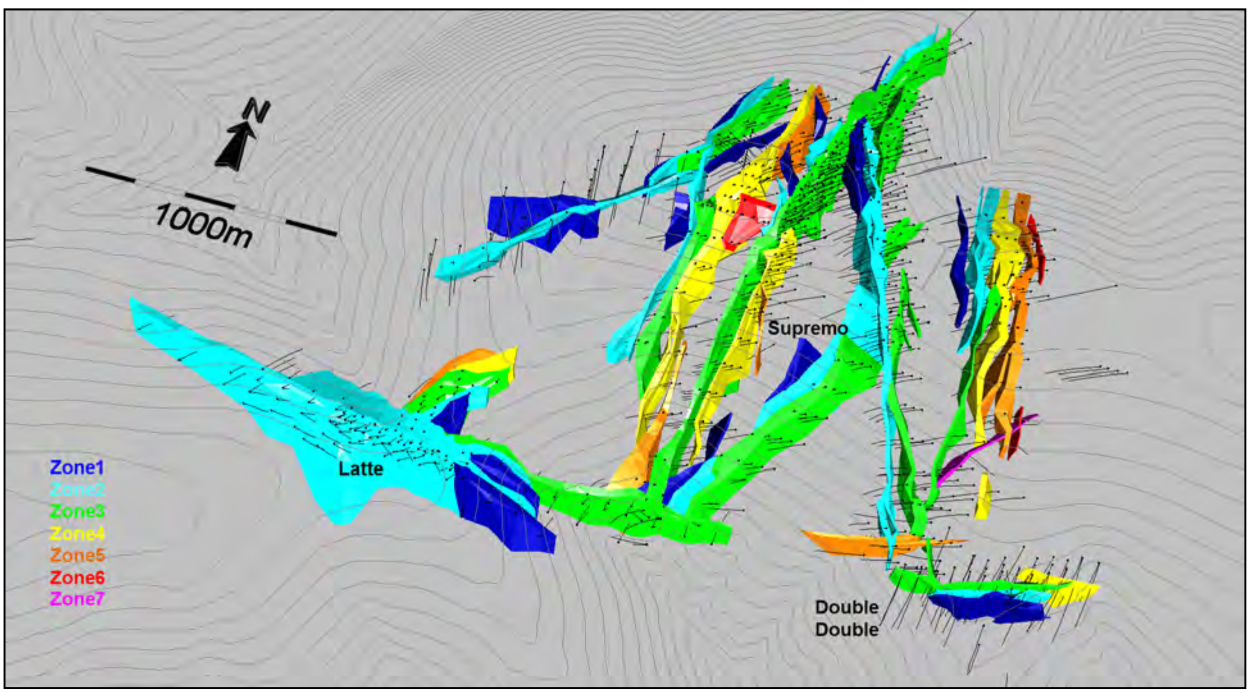
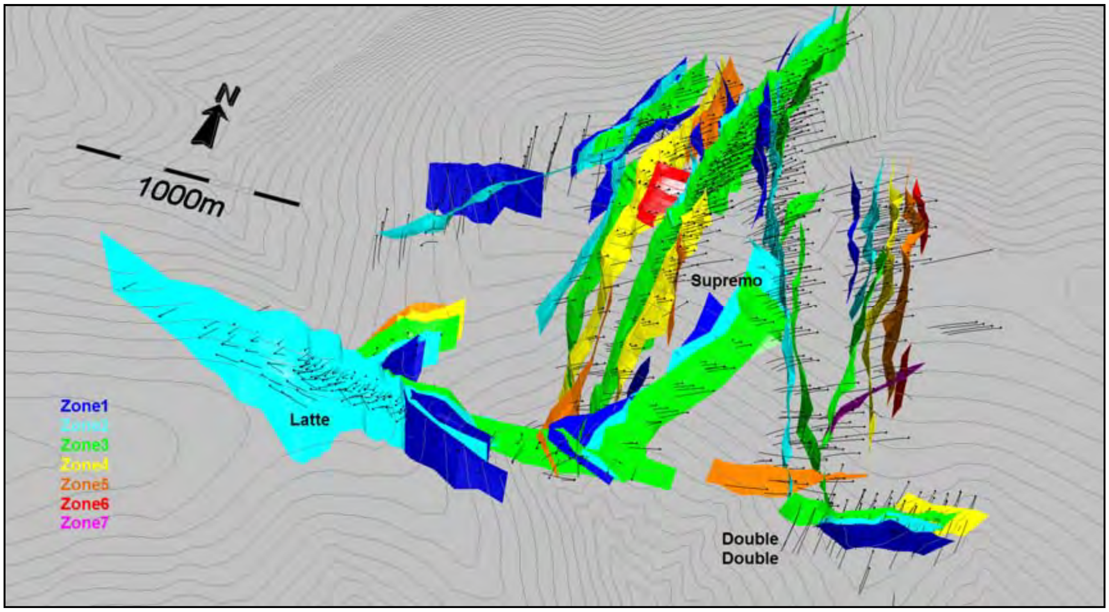


Figure 14-5: Planes Representing Trends of Mineralization in Each Structural Zone



During the 2013 field season, Kaminak undertook a sampling program testing for the cyanide soluble characteristics of the gold in the deposit. Samples selected for AuCN analysis were restricted to intervals where total gold grades are greater than 0.3g/t. This data is reasonably distributed but, because it excludes lower-grade sample intervals, is not sufficient to support direct estimation of AuCN estimates in model blocks. As an alternative, the ratio of AuCN/total Au has been calculated in samples where AuCN data is present. These ratios are then interpolated in the block model and are utilized in combination with qualitative (visual) estimates of the intensity of oxidation, to provide information regarding the depth and intensity of oxidation in the structural domains.

Oxidation appears to be channelled along the structural corridors that host the deposits. It is common to find intense oxidation at depths of over 200 m below surface. Strong oxidation is present over the majority of the Supremo and Double Double deposits. Oxidation is less pervasive at Latte, extending to about 125 m below surface in some areas. As a result, a higher proportion of the resource at Latte is characterised by moderate or transitional amounts of oxidation.

Four oxide types or domains have been interpreted for the Coffee deposit as described below. The oxide zone is relatively consistent and supported by a large proportion of the data. The degree of oxidation is often highly variable in the two (upper and lower) transition zones as reflected by the oxidation percentage ranges listed below.

- Oxide zone: intense to pervasive oxidation. (>90-95% oxidation)
- Upper Transition zone: moderate to intense oxidation (50-90% oxidation)
- Lower Transition zone: weak to moderately oxidized (10-50% oxidation)
- Sulphide zone: Fresh to weakly oxidized rocks (<10% oxidation)

A surface representing the base of colluvial overburden was also generated and all mineral zone domains have been truncated at this interface. 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: a too 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 1.53 m 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 hole and are generated at 1 m intervals down the length of the hole. Composites honour the structural domain contacts (in other words, individual composites begin and end at the point where a drill hole crosses the domain boundary). Several 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 grade. One of the main purposes of EDA is to determine if there is evidence of spatial distinctions in 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 is not statistically unique may impose a bias in the distribution of grades in the model.

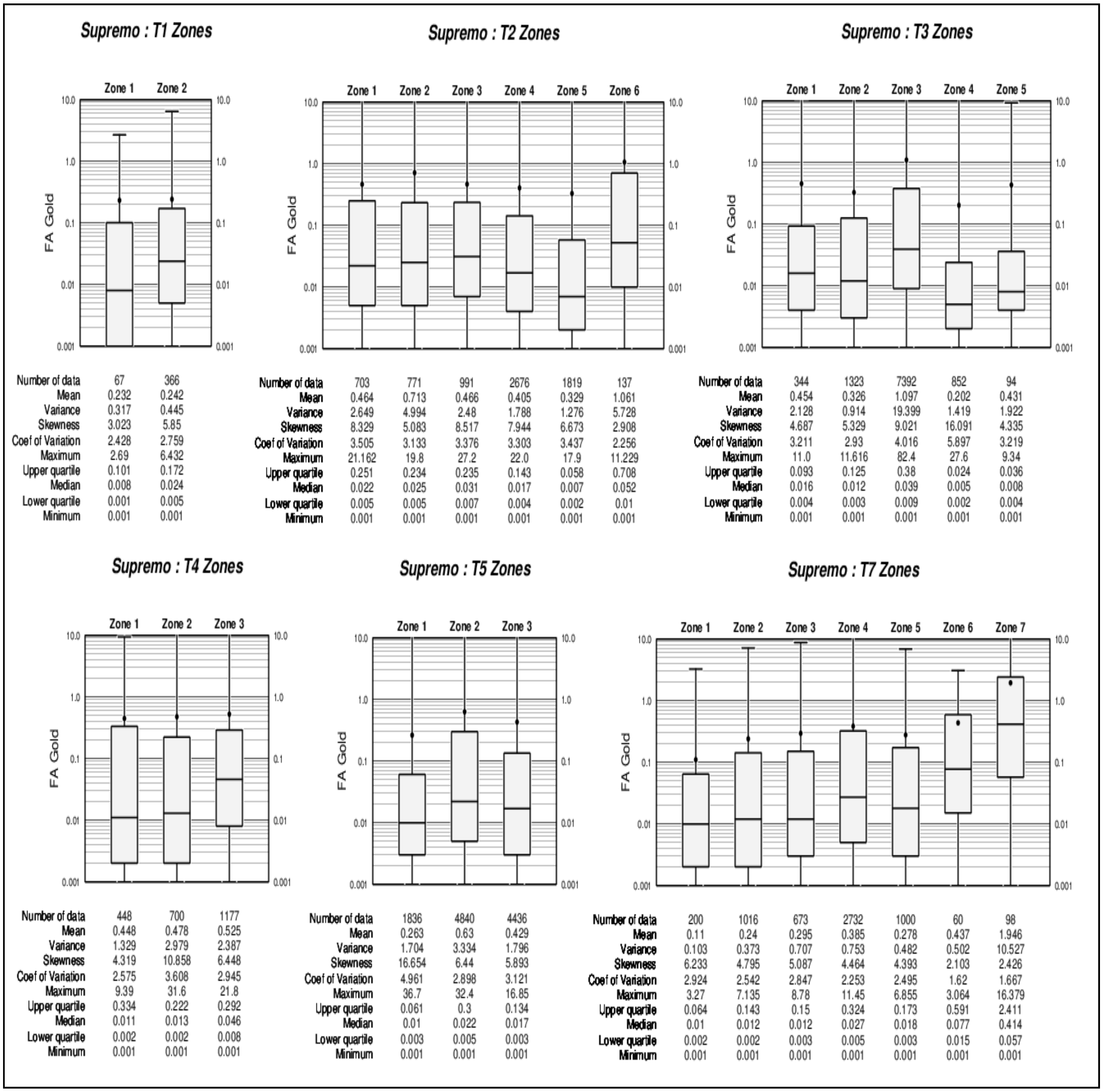
A domain boundary, which segregates the data during interpolation, is typically applied if the average grade in one domain is significantly different from another. A domain boundary may also be applied where a significant change in the grade distribution exists across the contact.

14.5.1 Basic Statistics by Domain

Summary statistics are evaluated using a series of boxplots; these boxplots compare the individual structural zone domains in each model area. Examples from the four deposit areas are shown in Figure 14-6, Figure 14-7, Figure 14-8, and Figure 14-9.

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 14-6: Boxplot for Gold in Structural Zone Domains at Supremo



Supremo : T1 Zones	Zone 1	Zone 2
Number of data	67	366
Mean	0.232	0.242
Variance	0.317	0.445
Skewness	3.023	5.85
Coef of Variation	2.428	2.759
Maximum	2.69	6.432
Upper quartile	0.101	0.172
Median	0.008	0.024
Lower quartile	0.001	0.005
Minimum	0.001	0.001

Supremo : T2 Zones	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	Zone 6
Number of data	703	771	991	2676	1819	137
Mean	0.464	0.713	0.466	0.405	0.329	1.061
Variance	2.649	4.994	2.48	1.788	1.276	5.728
Skewness	8.329	5.083	8.517	7.944	6.673	2.908
Coef of Variation	3.505	3.133	3.376	3.303	3.437	2.256
Maximum	21.162	19.8	27.2	22.0	17.9	11.229
Upper quartile	0.251	0.234	0.235	0.143	0.058	0.708
Median	0.022	0.025	0.031	0.017	0.007	0.052
Lower quartile	0.005	0.005	0.007	0.004	0.002	0.01
Minimum	0.001	0.001	0.001	0.001	0.001	0.001

Supremo : T3 Zones	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
Number of data	344	1323	7392	852	94
Mean	0.454	0.326	1.097	0.202	0.431
Variance	2.128	0.914	19.399	1.419	1.922
Skewness	4.687	5.329	9.021	16.091	4.335
Coef of Variation	3.211	2.93	4.016	5.897	3.219
Maximum	11.0	11.616	82.4	27.6	9.34
Upper quartile	0.093	0.125	0.38	0.024	0.036
Median	0.016	0.012	0.039	0.005	0.008
Lower quartile	0.004	0.003	0.009	0.002	0.004
Minimum	0.001	0.001	0.001	0.001	0.001

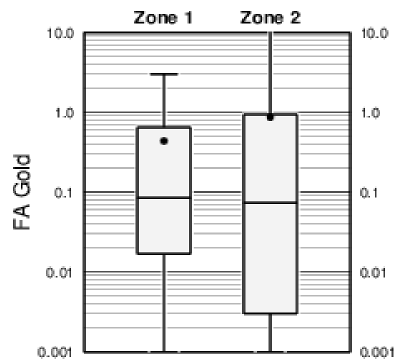
Supremo : T4 Zones	Zone 1	Zone 2	Zone 3
Number of data	448	700	1177
Mean	0.448	0.478	0.525
Variance	1.329	2.979	2.387
Skewness	4.319	10.858	6.448
Coef of Variation	2.575	3.608	2.945
Maximum	9.39	31.6	21.8
Upper quartile	0.334	0.222	0.292
Median	0.011	0.013	0.046
Lower quartile	0.002	0.002	0.008
Minimum	0.001	0.001	0.001

Supremo : T5 Zones	Zone 1	Zone 2	Zone 3
Number of data	1836	4840	4436
Mean	0.263	0.63	0.429
Variance	1.704	3.334	1.796
Skewness	16.654	6.44	5.893
Coef of Variation	4.961	2.898	3.121
Maximum	36.7	32.4	16.85
Upper quartile	0.061	0.3	0.134
Median	0.01	0.022	0.017
Lower quartile	0.003	0.005	0.003
Minimum	0.001	0.001	0.001

Supremo : T7 Zones	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	Zone 6	Zone 7
Number of data	200	1016	673	2732	1000	60	98
Mean	0.11	0.24	0.295	0.385	0.278	0.437	1.946
Variance	0.103	0.373	0.707	0.753	0.482	0.502	10.527
Skewness	6.233	4.795	5.087	4.464	4.393	2.103	2.426
Coef of Variation	2.924	2.542	2.847	2.253	2.495	1.62	1.667
Maximum	3.27	7.135	8.78	11.45	6.855	3.064	16.379
Upper quartile	0.064	0.143	0.15	0.324	0.173	0.591	2.411
Median	0.01	0.012	0.012	0.027	0.018	0.077	0.414
Lower quartile	0.002	0.002	0.003	0.005	0.003	0.015	0.057
Minimum	0.001	0.001	0.001	0.001	0.001	0.001	0.001

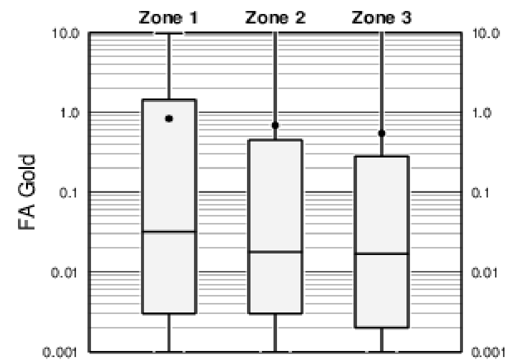
Figure 14-7: Boxplot for Gold in Structural Zone Domains at Latte

Latte : Main Zones



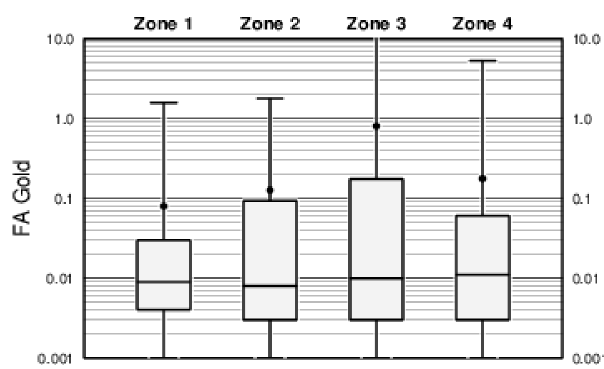
	Zone 1	Zone 2
Number of data	129	8380
Mean	0.44	0.871
Variance	0.452	4.59
Skewness	2.009	7.156
Coef of Variation	1.529	2.461
Maximum	3.01	48.7
Upper quartile	0.647	0.939
Median	0.085	0.074
Lower quartile	0.017	0.003
Minimum	0.001	0.001

Latte : East Zones



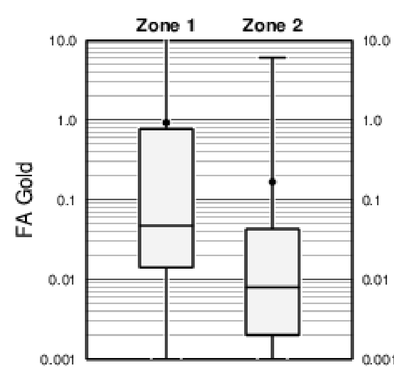
	Zone 1	Zone 2	Zone 3
Number of data	65	475	1499
Mean	0.835	0.69	0.548
Variance	3.064	5.356	3.428
Skewness	3.324	7.88	8.235
Coef of Variation	2.097	3.352	3.376
Maximum	9.71	30.2	32.5
Upper quartile	1.444	0.45	0.282
Median	0.032	0.018	0.017
Lower quartile	0.003	0.003	0.002
Minimum	0.001	0.001	0.001

Latte : North Zones



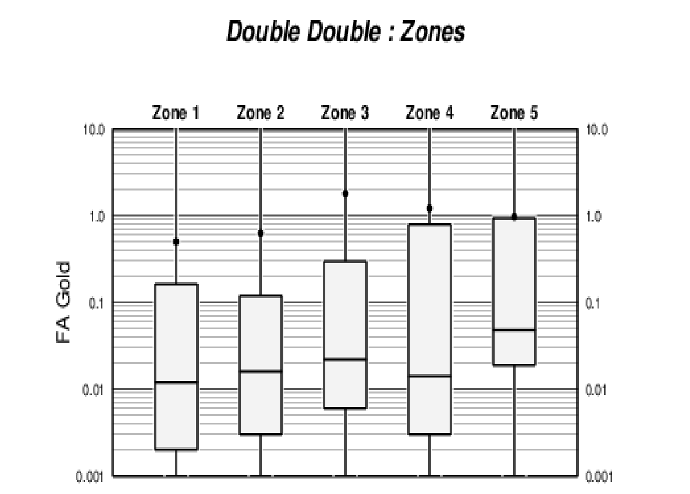
	Zone 1	Zone 2	Zone 3	Zone 4
Number of data	44	186	577	143
Mean	0.08	0.127	0.803	0.177
Variance	0.063	0.098	7.201	0.357
Skewness	5.087	3.673	5.67	5.988
Coef of Variation	3.148	2.452	3.344	3.384
Maximum	1.58	1.782	26.986	5.3
Upper quartile	0.03	0.093	0.175	0.061
Median	0.009	0.008	0.01	0.011
Lower quartile	0.004	0.003	0.003	0.003
Minimum	0.001	0.001	0.001	0.001

Latte : Sumatra Zones



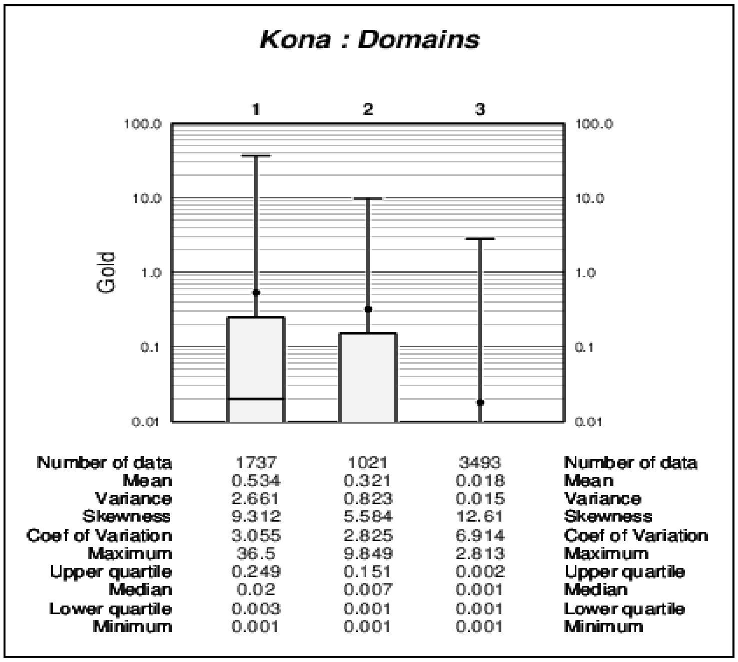
	Zone 1	Zone 2
Number of data	254	745
Mean	0.926	0.167
Variance	4.384	0.306
Skewness	3.657	6.059
Coef of Variation	2.261	3.322
Maximum	14.4	6.08
Upper quartile	0.775	0.043
Median	0.047	0.008
Lower quartile	0.014	0.002
Minimum	0.001	0.001

Figure 14-8: Boxplot for Gold in Structural Zone Domains at Double Double



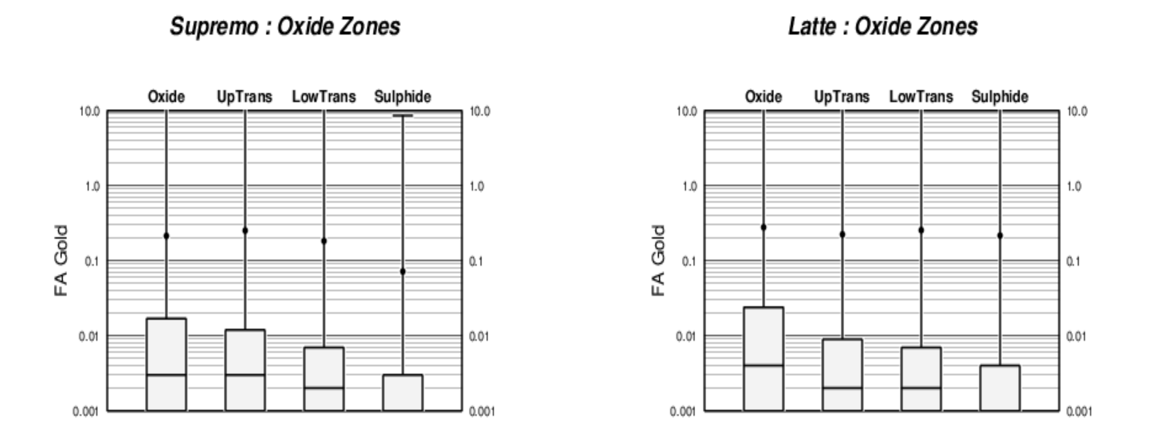
	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5	
Number of data	701	492	989	79	182	Number of data
Mean	0.502	0.628	1.798	1.217	0.975	Mean
Variance	3.053	10.365	67.419	6.44	3.961	Variance
Skewness	6.377	10.092	8.371	2.372	2.92	Skewness
Coef of Variation	3.48	5.127	4.568	2.084	2.041	Coef of Variation
Maximum	21.5	47.5	120.25	11.577	11.798	Maximum
Upper quartile	0.162	0.12	0.298	0.79	0.943	Upper quartile
Median	0.012	0.016	0.022	0.014	0.048	Median
Lower quartile	0.002	0.003	0.006	0.003	0.019	Lower quartile
Minimum	0.001	0.001	0.001	0.001	0.001	Minimum

Figure 14-9: Boxplot for Gold in Structural Zone Domains at Kona



The boxplot in Figure 14-10 shows the distribution of total gold in the oxide domains at Supremo and Latte. Gold grades tend to be higher in the oxide and decrease marginally with depth. There is significant overlap between domains.

Figure 14-10: Boxplot for (total) Gold in Oxide Domains at Supremo and Latte



Supremo : Oxide Zones

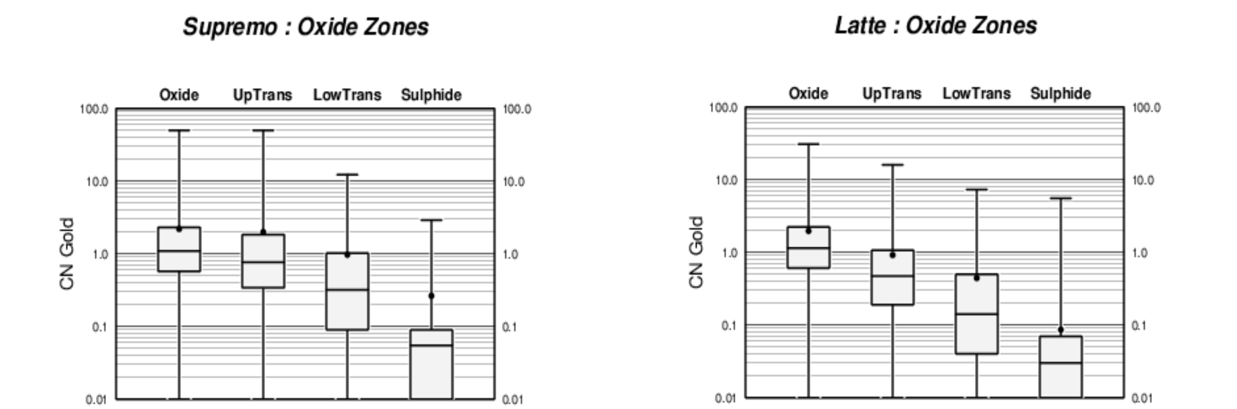
	Oxide	UpTrans	LowTrans	Sulphide
Number of data	91802	9468	2469	1240
Mean	0.214	0.252	0.183	0.072
Variance	1.978	3.412	1.275	0.253
Skewness	21.432	22.092	14.046	10.947
Coef of Variation	6.584	7.335	6.162	6.978
Maximum	82.4	78.7	27.7	8.56
Upper quartile	0.017	0.012	0.007	0.003
Median	0.003	0.003	0.002	0.001
Lower quartile	0.001	0.001	0.001	0.001
Minimum	0.001	0.001	0.001	0.001

Latte : Oxide Zones

	Oxide	UpTrans	LowTrans	Sulphide
Number of data	19373	10406	5657	5053
Mean	0.279	0.224	0.255	0.216
Variance	1.484	1.323	1.557	1.815
Skewness	10.687	15.198	11.197	15.087
Coef of Variation	4.366	5.13	4.888	6.247
Maximum	30.2	48.7	32.5	39.9
Upper quartile	0.024	0.009	0.007	0.004
Median	0.004	0.002	0.002	0.001
Lower quartile	0.001	0.001	0.001	0.001
Minimum	0.001	0.001	0.001	0.001

The distribution of AuCN data by Oxide domain is shown in Figure 14-11. The AuCN grades differ between domains but there is some local variability in the data which results in the relative spread in the data within each domain. It is important to note the decrease in data density with depth, especially at Supremo.

Figure 14-11: Boxplot for Cyanide Soluble Gold in Oxide Domains at Supremo and Latte

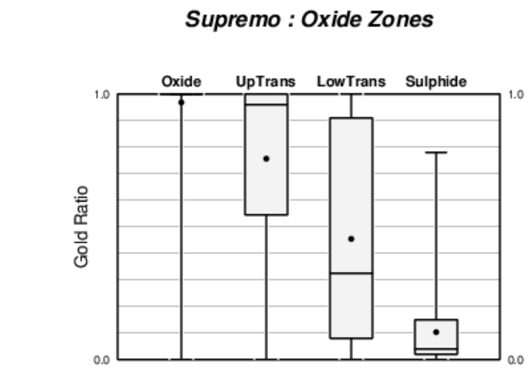


Supremo	Oxide	UpTrans	LowTrans	Sulphide
Number of data	7243	636	154	18
Mean	2.197	1.992	0.972	0.264
Variance	12.848	20.97	3.422	0.461
Skewness	5.859	6.89	3.819	3.331
Coef of Variation	1.632	2.299	1.903	2.566
Maximum	50.0	50.0	12.3	2.91
Upper quartile	2.32	1.839	1.02	0.09
Median	1.09	0.76	0.32	0.055
Lower quartile	0.57	0.34	0.09	0.01
Minimum	0.01	0.01	0.01	0.01

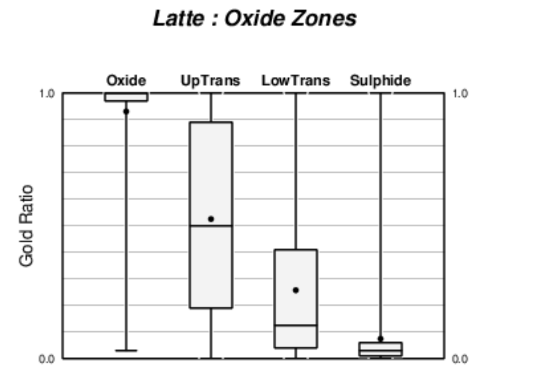
Latte	Oxide	UpTrans	LowTrans	Sulphide
Number of data	2201	855	406	366
Mean	1.969	0.92	0.443	0.086
Variance	7.539	2.012	0.58	0.097
Skewness	4.892	4.483	3.806	14.978
Coef of Variation	1.394	1.541	1.719	3.628
Maximum	30.8	16.05	7.27	5.53
Upper quartile	2.22	1.07	0.5	0.07
Median	1.14	0.47	0.14	0.03
Lower quartile	0.61	0.19	0.04	0.01
Minimum	0.01	0.01	0.01	0.01

Figure 14-12 shows the AuCN/Au ratios in the various oxide domains at Supremo and Latte. The interquartile range of data in the Oxide and Sulphide domains tends to be quite restricted suggesting these domains are well defined and supported by the sample data (although the amount of AuCN data in sulphide rocks is limited in most areas). The two transition zones show a much wider range of values suggesting a mix from fully oxidized to essentially fresh (sulphide) rocks.

Figure 14-12: Boxplot for AuCN/Au Ratios in Oxide Domains at Supremo and Latte



	Oxide	UpTrans	LowTrans	Sulphide	
Number of data	7243	636	154	18	Number of data
Mean	0.968	0.756	0.454	0.103	Mean
Variance	0.015	0.109	0.147	0.031	Variance
Skewness	-5.475	-1.084	0.326	3.138	Skewness
Coef of Variation	0.125	0.436	0.843	1.698	Coef of Variation
Maximum	1.0	1.0	1.0	0.78	Maximum
Upper quartile	1.0	1.0	0.91	0.15	Upper quartile
Median	1.0	0.96	0.325	0.04	Median
Lower quartile	1.0	0.544	0.08	0.02	Lower quartile
Minimum	0.0	0.0	0.0	0.0	Minimum



	Oxide	UpTrans	LowTrans	Sulphide	
Number of data	2201	855	406	366	Number of data
Mean	0.93	0.525	0.257	0.074	Mean
Variance	0.029	0.125	0.08	0.02	Variance
Skewness	-3.168	0.041	1.208	4.243	Skewness
Coef of Variation	0.183	0.671	1.098	1.919	Coef of Variation
Maximum	1.0	1.0	1.0	1.0	Maximum
Upper quartile	1.0	0.89	0.409	0.06	Upper quartile
Median	1.0	0.5	0.124	0.03	Median
Lower quartile	0.97	0.19	0.04	0.01	Lower quartile
Minimum	0.03	0.0	0.0	0.0	Minimum

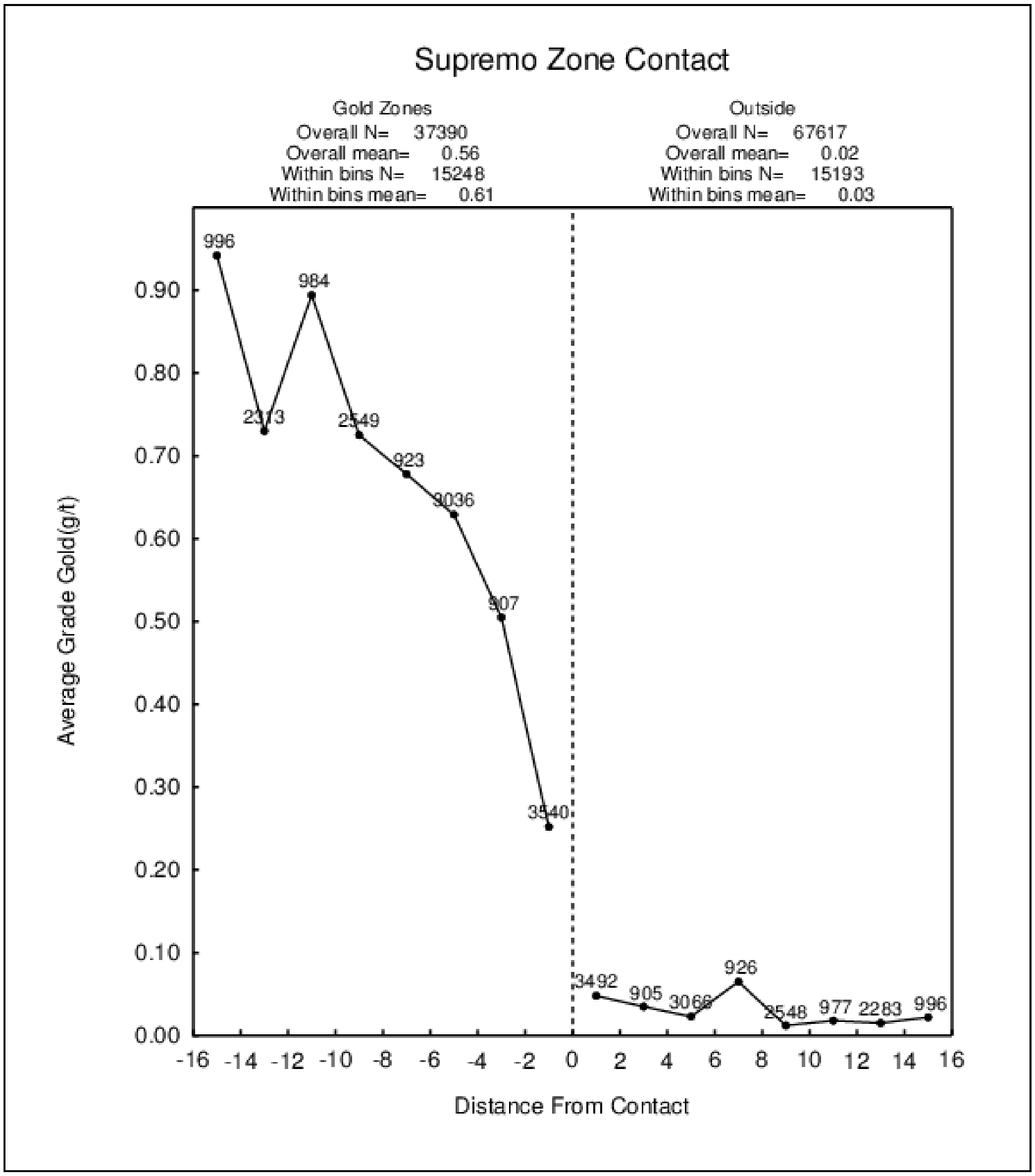
It is felt that the interpretation of the oxide domains is reasonable at this relatively early stage of the project evaluation. The nature of oxidation in these deposits is considered significant with respect to density factors and metallurgical properties. The AuCN data provides a sound basis for evaluating these properties. Limited AuCN data exists in some deposit areas (especially at depth) and additional sampling is recommended to support future studies.

14.5.2 Contact Profiles

Contact profiles evaluate the nature of 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 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 compared to sample data outside of the interpreted structural zone domains. Figure 14-13 shows an example from Supremo. There is a marked difference in gold grade between samples inside the structural zones compared to the surrounding data. This trend is similar for all deposit areas.

Figure 14-13: Contact Profile Comparing Samples Inside/Outside the Structural Domains at Supremo



14.5.3 Modeling Implications

Boxplots show that similarities and differences exist between the gold content of the individual structural zones in each of the deposit areas, but, overall, the individual structural zones all differ from samples located outside of the domains. This feature is also supported by the contact profiles that show the structural zone domains contain gold grades that exceed those in surrounding sample data. The author concludes that the interpreted structural zone domains contain data that is sufficiently different than surrounding sample data and these data should be segregated during model grade interpolations.

Although the results show that some differences exist between individual structural zones, they tend to be somewhat subtle. The individual structural 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.

Indications are that the degree of oxidation can be quite variable as it transitions from true oxide to sulphide material. Gold solubility ratios have been utilized to aid in the interpretation of these domains and they represent areas with similar oxidation properties. The generalization of the degree of oxidation through the transition zones is considered appropriate from a selectivity aspect as it is unlikely that different degrees of oxidation can be practically separated in these areas during mining. It is felt that these oxide domains are appropriate for initial (PEA –level) studies.

14.5.4 Conclusions

Each deposit area contains two or more individual structural zones that are used as hard boundary domains during the development of the resource model. This means that data is not mixed between zones during block grade interpolation. The resulting structural zone domains are summarized in Table 14-3. Note that the area outside of the structural zone is essentially barren and shows no potential for economic gold resources. No grade estimates were conducted outside of the structural zone domains.

Table 14-3: Summary of Estimation Domains

Area	Comments
Supremo	
T1 area	3 structural zones. Sub-vertical trending 30° azimuth.
T2 area	6 structural zones. The main zone has a strike length of over 1.5 km. All with approximately 15° azimuth and -70° dip to the east.
T3 area	5 structural zones. One zone extends over 2km with 20° azimuth and -80° dip to the east. The other 4 zones are less continuous but similar orientation. T3 contains some of the higher grade resources on the property.
T4 area	3 structural zones with 35° azimuth and -70° dip to the south-east. The larger of the 3 zones has strike length of about 1 km.
T5 area	3 structural zones. One of which has a strike length of over 1.6 km. In general, these have 345° azimuth and -80° dip to the east. The north end of T5 swings sub-parallel to T3.
T7 area	7 structural zones that trend north-south and are vertically oriented.
Latte	
Main area	2 structural zones. A thicker main zone with 110° azimuth and -65° dip to the south. A second zone is a "splay" extending from the east end of the Main zone.
East area	3 structural zones extending on the east end of the deposit area. Sub-vertical, east-west trending, zones with variable strike length up to 1.2km.
North area	5 structural zones interpreted as splays from the Main zone with 55° azimuth and -60° dip SE.
Sumatra	2 structural zones approximately 700m north of the Main Latte deposit. Trend 55° and dip -60° NW. Interpreted to merge to the east into T2 zone at Supremo.
Double Double	5 structural zone domains with 255° azimuth and -85° dip to the north.
Kona	2 structural zone domains with 70° azimuth and -85° dip to the south.

14.6 Specific Gravity Data

The methodology used to generate the specific gravity database is described in detail in Section 11.3 of this report.

There are a total of 5,328 samples tested for bulk density (SG). Approximately 40% of these samples occur inside of one of the interpreted structural domains and the remaining 60% of the data represents rocks outside of the mineralized domains. Although there is a relatively large specific gravity database, and the frequency of samples is generally quite good, the fact that these measurements have only been conducted on diamond drill core holes results in a lack of specific gravity data for both Kona and parts of Supremo and Latte. The distribution of specific gravity data is insufficient to interpolate individual values in model blocks. However, there is sufficient data present to generate average values that can be used to calculate resource tonnages.

In the previous resource estimate (November 2012), an average bulk density of 2.56 t/m^3 was applied to all blocks inside of the structural domains. The mean density of 2.56 t/m^3 remains the same following the addition of approximately 900 additional density samples collected in 2013. The distribution of density data has been analysed within the structural domains in each deposit area. A relationship exists between the density of rocks and the intensity of oxidation, a common feature in deposits of this type. There are differences between the deposit areas but these tend to be minor and may be influenced by the distribution of available SG data.

Table 14-4 lists the range, mean and standard deviation of densities by oxide domain. Although the average density of samples in the oxide domain average 2.51 t/m^3, it is felt that this is influenced by the distribution of available data and by several outliers. An average density of 2.54 t/m^3 is considered appropriate for oxide zone material. The average specific gravity values used to calculate tonnages in each of the oxide zones is considered reasonable based on the current data.

Table 14-4: Summary of Specific Gravity Data by Oxide Domain

Domain	Proportion of SG Data	Range min/max, (t/m^3)	Avg. Inside Structural domains, (t/m^3)	Standard Deviation	Avg. Assigned to model, (t/m^3)
Oxide	52%	1.74 – 3.79	2.51	0.17	2.54
Upper Transition	27%	1.54 – 3.08	2.58	0.16	2.58
Lower Transition	11%	1.37 – 2.94	2.63	0.17	2.65
Sulphide	10%	2.37 – 3.01	2.69	0.09	2.70

Near-surface blocks in overburden have been assigned a density of 1.90 t/m^3.

14.7 Evaluation of Outlier Grades

Histograms and probability plots were generated to show the distribution of gold in each structural zone. These were used to identify the existence of outlier grades in the composite database. The physical location of these potential outlier samples were reviewed in relation to the surrounding data. It was decided that, in most cases, potential outlier samples would be controlled through a combination of traditional top-cutting and the use of outlier limitations during block grade interpolation. An outlier limitation approach limits samples above a defined threshold to a maximum distance of influence during grade estimation. In most cases, a maximum range of 30 m was applied to outlier samples. A 50 m range was used in areas of Latte and Supremo where drill hole spacing increases to 100 m or more. The various thresholds and the resulting effects on the model areas are listed in Table 14-5.

The reduction in gold metal in all areas is considered reasonable for this deposit at this stage of evaluation. The relatively high reduction at Double Double is due to the small size of this deposit and the presence of a few very high-grade composites.

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	2.69	-	-	
T1 - Zone2	6.43	-	5	-8.60%
T1 - Zone3	13.55	-	6	
T2 - Zone1	21.16	-	10	
T2 - Zone2	19.8	-	15	
T2 - Zone3	27.2	-	10	
T2 – Zone4	22	-	15	-5.70%
T2 – Zone5	17.9	-	10	
T2 – Zone6	11.23	-	10	
T3 – Zone1	11	-	-	
T3 – Zone2	11.62	-	-	-6.30%
T3 – Zone3	82.4	70	50	
T3 – Zone4	27.6	15	10	
T3 – Zone5	9.34	-	7	
T4 – Zone1	9.39	-	7	
T4 – Zone2	31.6	20	10	-3.60%
T4 – Zone3	21.8	-	15	
T5 - Zone1	36.7	20	10	
T5 – Zone2	32.4	-	20	-3.10%
T5 – Zone3	16.85	-	15	
T7 - Zone1	3.27	-	-	
T7 – Zone2	7.14	-	5	
T7 – Zone3	8.78	-	5	
T7 – Zone4	11.45	-	-	-2.40%
T7 – Zone5	6.86	-	5	
T7 – Zone6	3.06	-	2	
T7 - Zone7	16.37	-	10	
All Supremo domains combined				-4.90%
Latte				
Main - Zone1	4.2	-	2	-2.10%
Main – Zone2	48.7	35	25	
East – Zone1	9.71	-	5	
East – Zone2	30.2	20	15	-13.00%
East - Zone3	32.5	20	15	
North - Zone1	1.58	-	-	
North - Zone2	1.78	-	-	
North - Zone3	26.99	-	15	-5.90%
North - Zone4	5.3	-	3	
North - Zone5	4.34	-	3	
Sumatra - Zone1	14.4	-	10	-5.10%
Sumatra – Zone2	6.08	-	5	
Latte Combined (Incl. Sumatra)				-5.10%
Double Double				
Zone1	21.5	-	12	
Zone2	47.5	25	15	-16.50%
Zone3	120.25	70	50	
Zone4	11.58	-	-	
Zone5	11.8	-	8	
Kona				
Zone1	36.5	15	10	-3.00%
Zone2	9.849	-	-	

(1) 1 m composites.
(2) Influence of composites above threshold limited to maximum 30 m during grade interpolation in zones where drilling is at 50 m intervals or less. Where drilling is spaced at 100 m or more, the outlier limit maximum distance is increased to 50 m (Latte East and North, Supremo T4 and parts of T3 and T7).
(3) Loss in metal in resource model limited to blocks within a maximum distance of 50 m from drilling.

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 amount of 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 Isaacs & Co. Due to a lack of available information in some areas, sample data from multiple structural zones was combined to generate correlograms. Multidirectional correlograms were generated from composited drill hole samples and the results are summarized in Table 14-6. 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 ones 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**
	0.300	0.500	0.200	36	0	41	44	0	25
Supremo T1			Spherical	22	180	49	33	180	65
				5	90	0	5	90	0
	0.250	0.650	0.100	13	0	77	5102	180	7
Supremo T2		Spherical		5	90	0	355	0	83
				3	180	13	5	90	0
	0.200	0.700	0.100	21	0	54	3848	0	8
Supremo T3		Spherical		10	90	0	257	180	82
				2	180	36	10	90	0
	0.300	0.622	0.078	69	0	0	462	180	89
Supremo T4		Spherical		10	0	90	14	0	1
				5	90	0	5	90	0
	0.250	0.600	0.150	11	0	86	900	0	45
Supremo T5		Spherical		5	90	0	342	180	45
				2	180	4	5	90	0
	0.250	0.650	0.100	17	0	3	800	0	-1
Supremo T7		Spherical		5	90	0	391	180	89
				4	180	87	5	90	0
	0.200	0.650	0.150	29	270	45	563	90	53
Latte Main		Spherical		6	0	0	68	90	-37
				3	270	-45	6	0	0
	0.250	0.400	0.350	27	270	-44	181	90	-6
Latte East			Spherical	6	0	0	6	0	0
				2	270	46	5	270	-84
	0.350	0.550	0.100	14	90	70	77	90	44
Latte North			Spherical	5	270	20	25	270	46
				5	0	0	5	0	0
	0.250	0.600	0.150	24	90	7	67	90	-29
Sumatra			Spherical	5	0	0	30	90	61
				4	270	83	5	0	0
	0.375	0.581	0.044	21	90	-82	250	90	-1
Double Double		Spherical (capped to 5g)		10	270	-8	185	270	-89
				5	0	0	5	0	0
	0.300	0.589	0.111	20	270	-4	4779	270	-2
Kona		Spherical		9	90	-86	439	90	-88
				5	0	0	5	0	0

Note: Correlograms modelled using sample data composited to 1 m intervals.

14.9 Model Setup and Limits

Four block models were initialized in MineSight® with the dimensions defined in Table 14-7. Two block sizes were selected considering the current drill hole spacing and the selective mining unit (SMU) size that is considered appropriate for deposits of this type and scale. In all cases, the short axis is oriented across the strike of the deposit. The models are not rotated.

In comparison to the previous model extents, the Supremo model has been expanded to the west to include the new T1 area. The Latte model has been expanded to the west to cover extensions to the Main zone and to the north to include the Sumatra zone. The Double Double model has been expanded to the west to include a new sub parallel zone near the south end of the Supremo T7 zone.

Table 14-7: Block Model Limits

Direction	Minimum(1) (m)	Maximum(1) (m)	Block Size (m)	Number of Blocks
Supremo				
East	583,450	585,160	3	570
North	6,973,100	6,975,500	10	240
Elevation	650	1320	5	134
Latte				
East	581,800	584,400	10	260
North	6,972,810	6,974,505	3	565
Elevation	500	1,300	5	160
Double Double				
East	584,400	585,500	5	200
North	6,973,030	6,973,500	2	235
Elevation	600	1,130	5	106
Kona				
East	579,450	580,050	5	120
North	6,972,850	6,973,300	2	225
Elevation	950	1,320	5	74

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

Using the domain wireframes, blocks in the model are assigned area and zone code values on a majority basis. Blocks with more than 50% of their volume inside a wireframe domain are assigned a zone code value of that domain.

The proportion of blocks within the structural zone domains is also calculated and stored within the model as a percentage. These values are used as a weighting factor to determine the volume and tonnage estimates.

Blocks are also assigned oxide codes on a majority basis. The portion of each block located below the topographic surface is stored as a percentage in each model block.

14.10 Interpolation Parameters

The block model grades for gold were estimated using ordinary kriging. Estimates 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 (*Her*mitian *co*rrection) grade distribution. This approach reduces the amount of smoothing (also known as averaging) in the model and, while there may be some uncertainty on a localized scale, this approach produces reliable estimates of the potentially recoverable grade and tonnage for the overall deposit. The interpolation parameters are summarized by domain in Table 14-8.

Table 14-8: Interpolation Parameters

Area/ Domain	Search Ellipse Range (metre) [1]			Number of Composites			Other
	X	Y	Z	Minimum	Maximum	Maximum Per Hole	
Supremo T1	2	200	200	1	9	3	
Supremo T2	3	200	200	1	20	5	1 hole per quadrant
Supremo T3	3	200	200	1	20	5	1 hole per quadrant
Supremo T4	2	200	200	1	9	3	1 hole per quadrant
Supremo T5	3	200	200	1	20	5	1 hole per octant
Supremo T7	3	200	200	1	15	5	1 hole per quadrant
Latte	150	3	150	1	15	5	1 hole per quadrant
Double Double	150	2	150	1	6	2	1 hole per quadrant
Kona	150	3	150	1	6	2	1 hole per quadrant

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

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.

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 through 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 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 included confirmation of the proper coding of blocks within the respective zone domains. The distribution of block grades was compared relative to the drill hole samples to ensure the proper representation in the model.

14.11.2 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, also known as the Discrete Gaussian Correction. With 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.

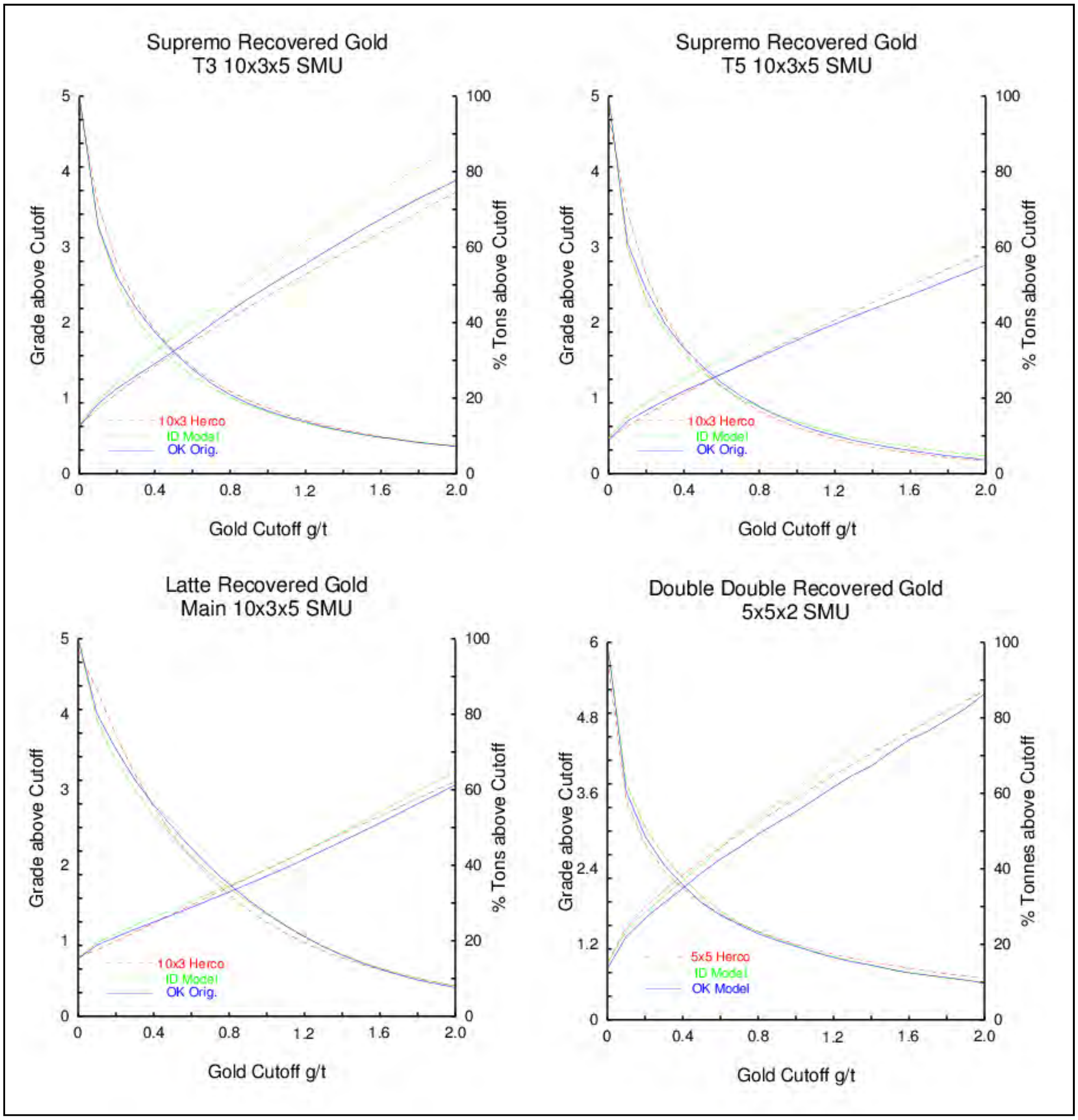
In general, the estimated model should be slightly higher in tonnage and slightly lower in grade when compared to the Herco distribution at the projected cut-off grade. These differences account for selectivity and other potential ore-handling issues which commonly occur during mining.

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 14-14.

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 14-14: Examples of Herco Plots

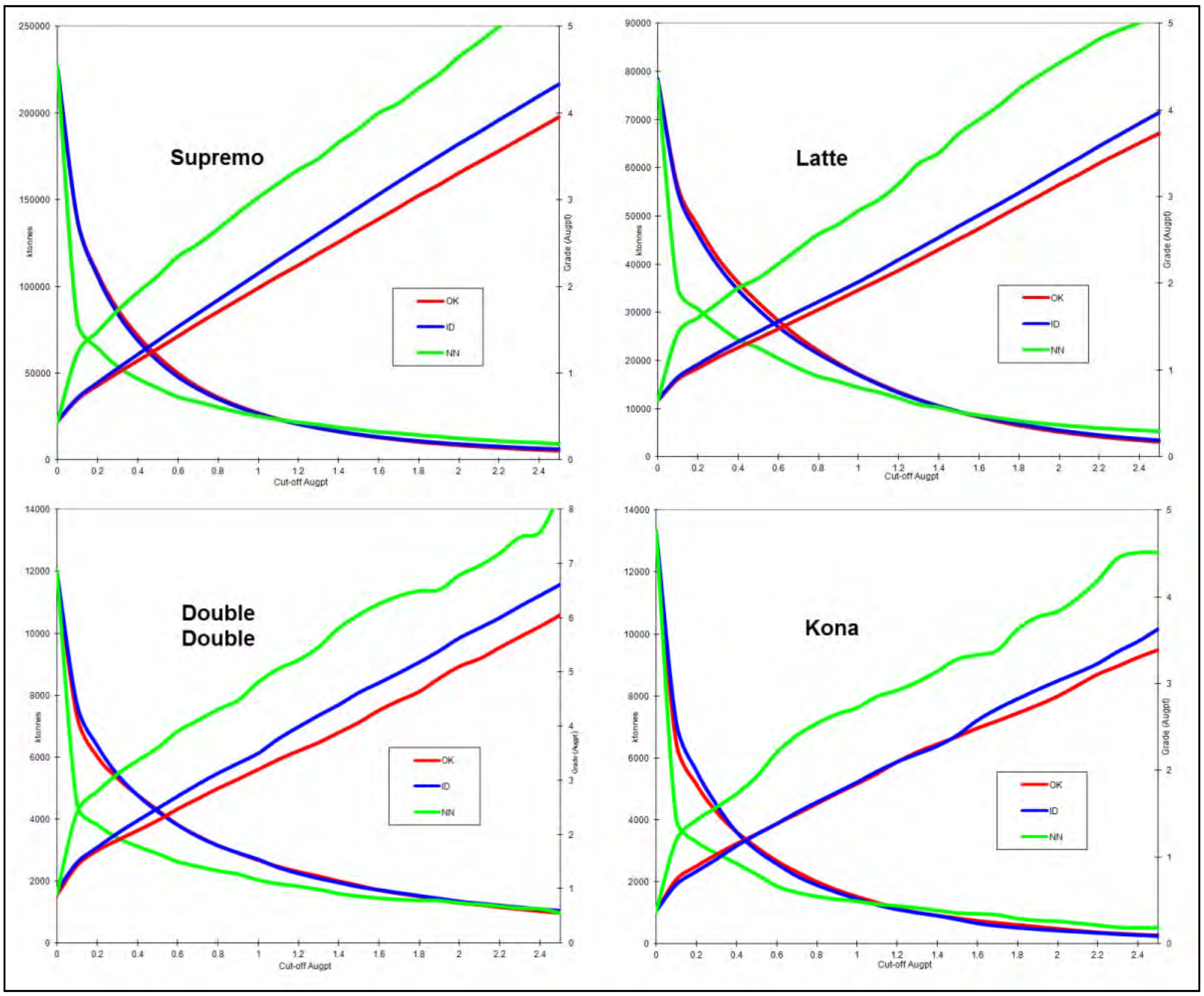


14.11.3 Comparison of Interpolation Methods

For comparison purposes, additional grade models were generated using the inverse distance weighted (ID^2) 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 14-15.

There is good correspondence between models at Supremo, Latte, and Kona. At Double Double, the results indicate that the ordinary kriging model is smoother than the inverse distance model.

Figure 14-15: Comparison of Ordinary Kriging (OK), Inverse Distance (ID^2) and Nearest Neighbour (NN) Models



14.11.4 Swath Plots (Drift Analysis)

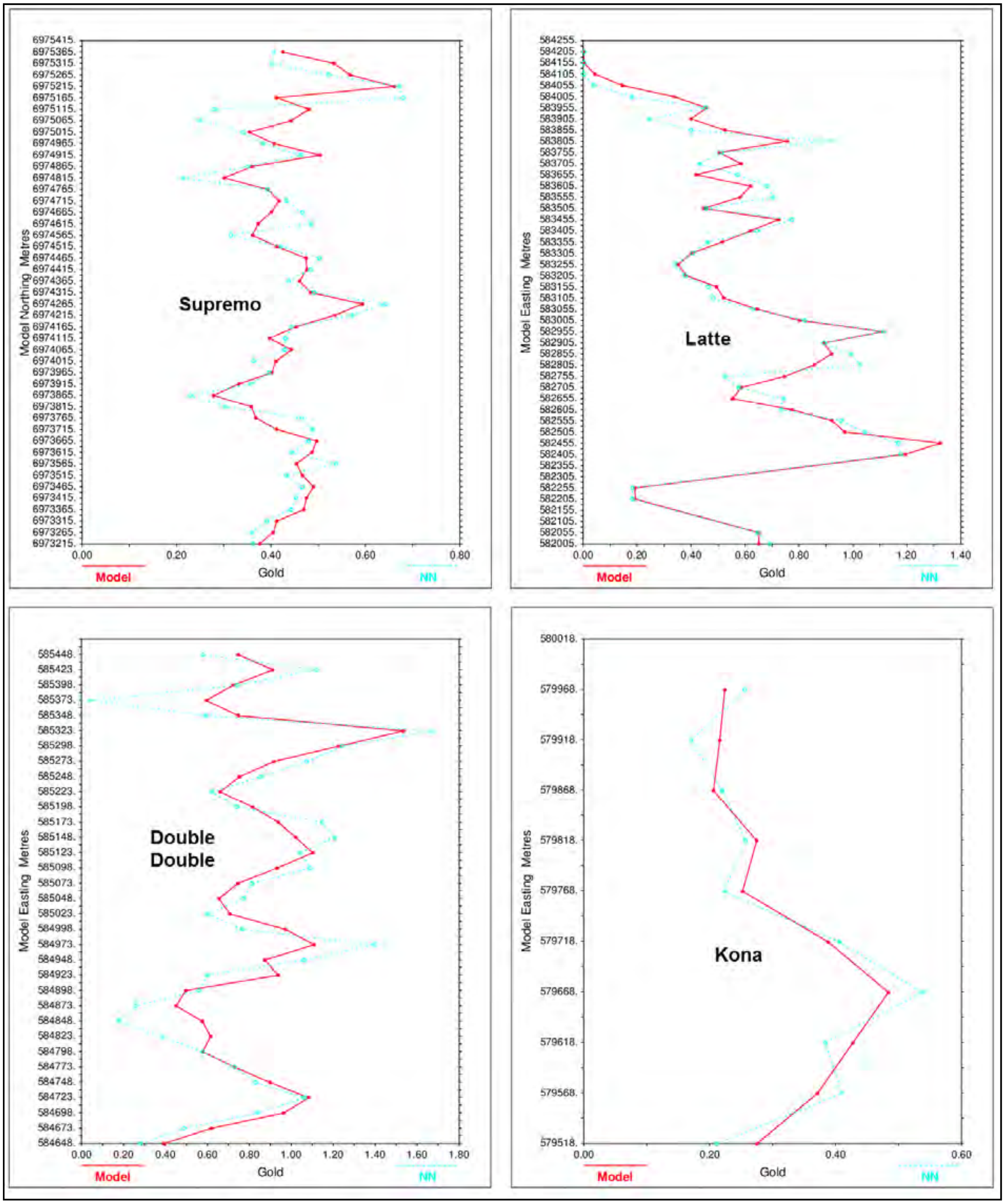
A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions throughout the deposit. Using the swath plot, grade variations from the ordinary kriging model are compared to the distribution derived from the declustered nearest 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. Some examples of swath plots at various orientations are shown in Figure 14-16.

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

Figure 14-16: Examples of Swath Plots



14.12 Resource Classification

The mineral resources were classified in accordance with the CIM *Definition Standards for Mineral Resources and Mineral Reserves* (November 2010). 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.

Almost all deposit areas have been tested with drill holes located at a maximum spacing of 100 m, with many areas delineated with drilling spaced at 50 m or less. Statistical and visual evaluation of the distribution of gold in the deposits suggests that relatively continuous zones of mineralization can be delineated with a reasonable degree of confidence when drill holes are spaced at up to 100-metre intervals. Based on this observation, model blocks have been considered for inclusion in the Inferred category if they occur within a maximum distance of 50 metres from a drill hole.

Statistical and visual evaluations suggest that zones of mineralization, delineated with drilling on a nominal 25 m grid pattern, exhibit a sufficient degree of confidence to be included in the Indicated resource category. One exception is in the Main part of the Latte deposit, where the mineralization tends to be much thicker and continuous from section to section. In this area, a 35 m pattern of holes is considered sufficient to provide the confidence level for resources in the Indicated category.

During the 2013 drilling program, parts of Supremo T3 and the Main Latte zone were delineated with drill holes that increase the confidence in the resource estimate allowing for the classification of resources in the Indicated category. Drill holes at Supremo T3 between 6974100N to 6974600N are spaced on a 25 m grid pattern to a distance of approximately 175 m below surface. The Main part of the Latte deposit, between 582850E to 583325E, has been tested with holes at 50m intervals on sections spaced at 25 m intervals resulting in a pattern of holes spaced at approximately 35 m intervals.

Some manual “smoothing” of these criteria was 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 confidence. This process resulted in a series of three-dimensional domains that were used to assign resource classification codes into model blocks.

The definitions of mineral resources categories are described as follows:

Indicated Mineral Resources – Resources are included in the Indicated category if they are located within a structural domain, form zones of relatively continuous mineralization and are delineated with three of more drill holes on a nominal 25 m pattern. In the Main part of the Latte deposit, where the mineralized zone tends to be thicker, and the minimum drill grid can be expanded to a 35 m pattern.

Inferred Mineral Resources – Resources are included in the Inferred category if they are located within a structural domain and within a maximum distance of 50 metres from a drill hole and exhibit a reasonable degree of geological continuity.

14.13 Mineral Resources

CIM *Definition Standards for Mineral Resources and Mineral Reserves* (November 2010) define a mineral resource as:

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

The "reasonable prospects for 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 taking into account extraction scenarios and processing recovery.

The Coffee gold deposits form relatively continuous, sub-vertical zones of gold mineralization extending from the surface to a depth of several hundred metres. The deposits are amenable to open pit or underground extraction (or a combination of both). The "reasonable prospects for economic extraction" were tested using floating cone pit shells based on reasonable technical and economic assumptions (for example, site operating costs of C$20 per tonne mined, a pit slope of 45° and gold prices ranging from $1,300/oz to $1,700/oz. These initial evaluations assume 100% mining and metallurgical recoveries). These pit optimization evaluations are used solely for the purpose of testing the "reasonable prospects for economic extraction," and do not represent an attempt to estimate mineral reserves. There are no mineral reserves at the Coffee project. The optimization results are used to assist with the preparation of a Mineral Resource Statement and to select and appropriate reporting assumptions.

Analyses of the floating cones show that the majority of the Oxide and Transitional gold mineralization could potentially be amenable to open pit extraction methods as these shells extend to depths of over 200 m below surface in many areas. Studies show that 80% of the oxide and transitional mineral resource is located within 150 m of surface and 94% of these resources are within a maximum depth of 200m below surface.

Although these studies suggest that some mineralized areas may not be economically viable, this represents a relatively small proportion of the resource.

Gold mineralization delineated by drilling at Coffee is primarily located within a maximum distance of 200m below surface. Condemning portions of the deposit from the mineral resource using assumed technical and economic factors would likely affect only a minor proportion of the resource at this stage of project evaluation. In the author's opinion, any or all of the mineralization at Coffee shows reasonable prospects for economic viability and, therefore, has been included in the estimation of mineral resources. Future detailed engineering studies are required to demonstrate the true economic viability of the resource.

The Mineral Resource Statement for the Coffee project is presented in Table 14-9. The Mineral Resource Statement is reported at two cut-off grades. Oxide and Transition Mineral Resources are reported at a cut-off grade of 0.5 g/t gold while Sulphide Mineral Resources are reported at a cut-off grade of 1.0 g/t gold reflecting the generally greater depths and differing metallurgical characteristics of this material. The distribution of base case resources at Supremo, Latte and Double Double is shown in Figure 14-17.

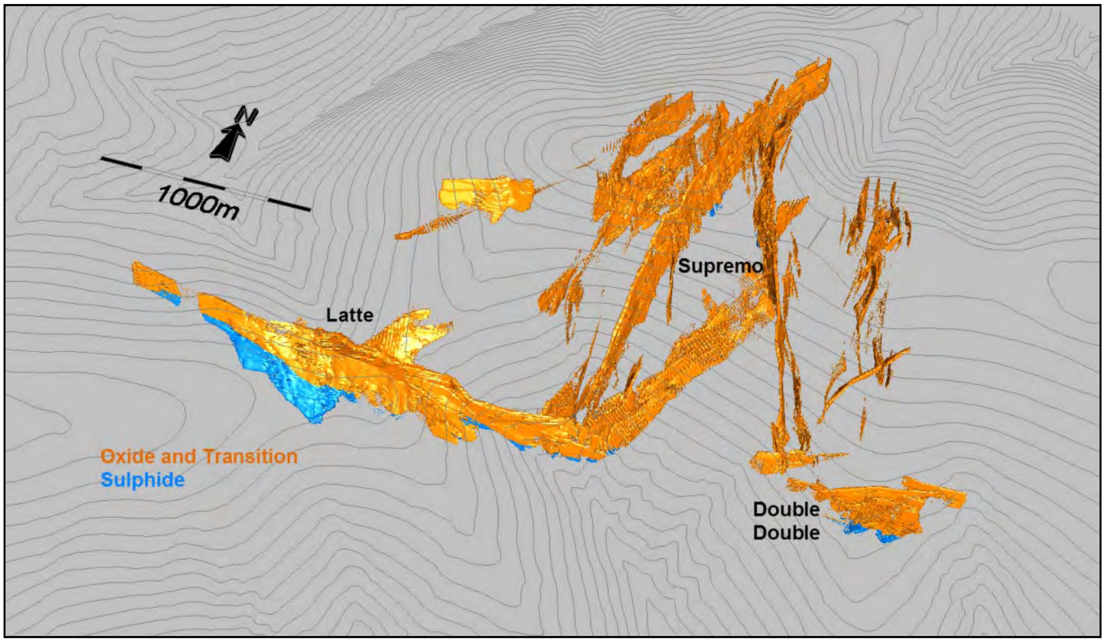
There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, or political issues which could materially affect the mineral resource.

Table 14-9: Estimate of Mineral Resources for the Coffee Project

Area	Oxide			Upper Transition			Lower Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Indicated															
Supremo	2,967	2.13	203	847	1.62	44	183	1.78	11	3,997	2.01	258	0	0	0
Latte	5,588	1.54	277	2,773	1.22	109	1,958	1.16	73	10,319	1.38	459	42	1.52	2
Combined	**8,555**	**1.75**	**480**	**3,619**	**1.32**	**153**	**2,141**	**1.21**	**83**	**14,316**	**1.56**	**717**	**42**	**1.52**	**2**
Inferred															
Supremo	42,003	1.21	1,636	9,001	1.3	377	2,579	1.41	117	53,583	1.24	2,129	564	1.47	27
Latte	5,673	1.23	224	3,518	1.46	166	3,878	1.43	179	13,070	1.35	569	4,529	1.95	284
Dbl. Dbl.	1,772	2.99	170	1,974	1.81	115	206	1.49	10	3,951	2.32	295	189	2.21	13
Kona	989	1.48	47	1,473	1.2	57	0	0	0	2,462	1.32	104	244	1.57	12
Combined	**50,437**	**1.28**	**2,078**	**15,967**	**1.39**	**714**	**6,662**	**1.43**	**306**	**73,066**	**1.32**	**3,098**	**5,525**	**1.89**	**336**

*Oxide and Transition mineral resources reported at a cut-off grade of 0.5 g/t gold. Sulphide mineral resources reported at a cut-off grade of 1.0 g/t gold. Cut-off grades based on a gold price of US$1,300 per ounce, site operation costs of US$20.00 per tonne mined and assume 100 percent mining and metallurgical recovery. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have a demonstrated economic viability.

Figure 14-17: Isometric View of the Distribution of Base Case Resources



14.14 Sensitivity of Mineral Resources

The sensitivity of mineral resources is demonstrated by listing resources at a series of cut-off thresholds in Table 14-10, Table 14-11 and Table 14-12.

14.15 Comparison with the Previous Estimate of Mineral Resources

Table 14-13 compares the new resource estimate with the previous resource released in November 2012. In general there has been an increase in Oxide and Transitional resources, at 0.5 g/t Au cut-off, of 28 Mt at an average grade of 0.99 g/t gold (936 koz contained gold). Note that approximately 400 koz gold is included in the Lower Transition domain.

Changes in resources are due to a combination of factors including:

- New drilling conducted in 2013 discovered additional resources in all areas. Includes new resource areas at T1 (+80 koz), Latte North (+60 koz) and Sumatra (+50 koz). Main additions occurred at T2 (+300 koz), T3 (+110 koz), T7 (+110 koz) and Latte Main and East (+100 koz).
- Reinterpretation of Oxide and Transition domains using combination of geology and AuCN data.
- Alterations in interpolation approach that result in better continuity of mineralization in the resource model (better application of dynamic anisotropy during interpolation).
- There is little change in the amount of Sulphide resources.

Table 14-10: Estimated Mineral Resource at 1.0 g/t Gold Cut-off

Area	Oxide			Upper Transition			Lower Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Indicated															
Supremo	1,734	3.15	175	467	2.37	35	110	2.48	9	2,311	2.96	220	0	0.00	0
Latte	3,573	1.99	229	1,540	1.62	80	839	1.74	47	5,952	1.86	356	42	1.52	2
Combined	**5,307**	**2.37**	**405**	**2,007**	**1.79**	**116**	**949**	**1.82**	**56**	**8,262**	**2.17**	**576**	**42**	**1.52**	**2**
Inferred															
Supremo	18,338	1.87	1,100	4,231	1.97	268	1,246	2.16	87	23,814	1.90	1,455	564	1.47	27
Latte	2,527	1.87	152	1,961	2.07	130	2,204	1.98	140	6,693	1.96	423	4,529	1.95	284
Dbl. Dbl.	1,228	4.00	158	1,135	2.62	96	119	2.05	8	2,482	3.28	262	189	2.21	13
Kona	565	2.06	37	687	1.76	39	0	0.00	0	1,252	1.90	76	244	1.57	12
Combined	**22,658**	**1.99**	**1,448**	**8,014**	**2.07**	**533**	**3,569**	**2.04**	**235**	**34,241**	**2.01**	**2,216**	**5,525**	**1.89**	**336**

Table 14-11: Estimated Mineral Resource at 1.5 g/t Gold Cut-off

Area	Oxide			Upper Transition			Lower Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Indicated															
Supremo	1,213	3.98	155	278	3.14	28	71	3.16	7	1,562	3.79	190	0	0.00	0
Latte	2,069	2.55	170	660	2.14	45	374	2.40	29	3,103	2.44	244	16	2.02	1
Combined	**3,281**	**3.08**	**325**	**939**	**2.44**	**74**	**445**	**2.52**	**36**	**4,665**	**2.89**	**434**	**16**	**2.02**	**1**
Inferred															
Supremo	9,533	2.47	756	2,217	2.65	189	702	2.90	65	12,452	2.52	1,011	173	2.05	11
Latte	1,371	2.42	107	1,259	2.54	103	1,315	2.48	105	3,945	2.48	314	2,301	2.66	197
Dbl. Dbl.	900	5.01	145	750	3.33	80	80	2.46	6	1,731	4.17	232	125	2.70	11
Kona	344	2.61	29	375	2.22	27	0	0.00	0	720	2.41	56	88	2.23	6
Combined	**12,149**	**2.66**	**1,037**	**4,602**	**2.70**	**399**	**2,097**	**2.62**	**177**	**18,848**	**2.66**	**1,612**	**2,686**	**2.61**	**225**

Table 14-12: Estimated Mineral Resource at 2.0 g/t Gold Cut-off

Area	Oxide			Upper Transition			Lower Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Indicated															
Supremo	919	4.69	139	186	3.85	23	59	3.45	7	1,164	4.50	168	0	0.00	0
Latte	1,178	3.17	120	260	2.80	23	191	3.05	19	1,629	3.10	162	8	2.24	1
Combined	**2,097**	**3.84**	**259**	**446**	**3.24**	**46**	**250**	**3.14**	**25**	**2,793**	**3.68**	**331**	**8**	**2.24**	**1**
Inferred															
Supremo	5,277	3.07	521	1,322	3.28	139	473	3.47	53	7,072	3.14	713	67	2.56	6
Latte	796	2.92	75	752	3.07	74	795	2.98	76	2,342	2.99	225	1,238	3.47	138
Dbl. Dbl.	666	6.17	132	518	4.05	67	39	3.19	4	1,223	5.18	204	63	3.63	7
Kona	232	3.03	23	188	2.70	16	0	0.00	0	420	2.88	39	51	2.60	4
Combined	**6,971**	**3.35**	**750**	**2,779**	**3.33**	**297**	**1,307**	**3.16**	**133**	**11,058**	**3.32**	**1,181**	**1,420**	**3.40**	**155**

Table 14-13 Comparison to Previous Resource Estimate

Jan 2014 Mineral Resource

Area	Oxide			Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Indicated												
Supremo	2,967	2.13	203	1,030	1.65	55	3,997	2.01	258	0	0.00	0
Latte	5,588	1.54	277	4,731	1.20	182	10,319	1.38	459	42	1.52	2
Combined	**8,555**	**1.75**	**480**	**5,761**	**1.28**	**237**	**14,316**	**1.56**	**717**	**42**	**1.52**	**2**
Inferred												
Supremo	42,003	1.21	1,636	11,580	1.33	494	53,583	1.24	2,129	564	1.47	27
Latte	5,673	1.23	224	7,396	1.45	345	13,070	1.35	569	4,529	1.95	284
Dbl. Dbl.	1,772	2.99	170	2,179	1.78	125	3,951	2.32	295	189	2.21	13
Kona	989	1.48	47	1,473	1.20	57	2,462	1.32	104	244	1.57	12
Combined	**50,437**	**1.28**	**2,078**	**22,629**	**1.40**	**1,020**	**73,066**	**1.32**	**3,098**	**5,525**	**1.89**	**336**

* Oxide and Transition mineral resources reported at a cut-off grade of 0.5 g/t gold. Sulphide mineral resources reported at a cut-off grade of 1.0 g/t gold.

Nov 2012 Inferred Mineral Resource

Area	Oxide			Transition			Oxide + Transition			Sulphide		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)	(ktonnes)	Au (g/t)	Au (koz)
Supremo	19,860	1.61	1,027	16,545	1.32	704	36,404	1.48	1,731	828	2.18	58
Latte	6,054	1.48	288	11,328	1.48	537	17,382	1.48	825	3,771	2.09	254
Dbl. Dbl.	1,175	3.16	120	1,966	1.9	120	3,141	2.37	240	188	2.11	13
Kona	989	1.48	47	1,473	1.2	57	2,462	1.32	104	244	1.57	12
Combined	**28,078**	**1.64**	**1,481**	**31,313**	**1.41**	**1,418**	**59,390**	**1.52**	**2,900**	**5,030**	**2.08**	**337**

15.0 MINERAL RESERVE ESTIMATES

Indicated and Inferred resources were used in the life-of-mine plan with Inferred resources representing 80% of the material planned for processing. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at the Coffee project.

16.0 MINING METHODS

16.1 Open Pit Optimization

Mine planning for the Coffee project was conducted using a combination of Mintec Inc., MineSight™ software and CAE NPV Scheduler (NPVS) software. The mineral inventory block models were produced by SIM Geological Inc. using MineSight™. NPVS was used for the open pit optimization, pit phase generation and life-of-mine scheduling.

Estimates were made for gold price, mining dilution, heap leach recovery, offsite refining costs, and royalties. Mining, processing, and general administration OPEX were also calculated based on leach processing throughput and, along with geotechnical parameters, formed the basis for open pit optimization. The mineral inventory block model for the Coffee project was used with NPVS software to determine optimal mining shells and pit phasing based on the design criteria summarized in Table 16-1. The OP mining costs assumed owner-operated mining and were estimated for both mineralized material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate.

Indicated and Inferred mineral resources were included in the pit optimization process (no mineral resources have been classified in the measured category). Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the Inferred resources would be upgraded to a higher resource category.

Table 16-1: Design Criteria

Parameter	Unit	Value
Revenue, Smelting & Refining		
Gold price	US$/oz Au	1,250
Exchange Rate	US$:C$	0.95
Payable metal	%	100
TC/RC/Transport	C$/oz Au	7.50
Royalty @ 1% (assumed after capital buyout of $2M before PP)	C$/oz Au	12.50
Net gold value per ounce	C$/oz	1,296
Net gold value per gram	C$/g	41.66
OPEX Estimates		
OP Waste Mining Cost	C$/t waste mined	2.25
OP HL feed Mining Cost	C$/t HL feed mined	2.25
Processing Cost for oxide in all deposits	C$/t milled	7.00
Processing Cost for transition in Supremo, DD & Kona	C$/t milled	7.25
Processing Cost for transition in Latte	C$/t milled	7.50
G&A	C$/t milled	4.00
Total OPEX (ex. Mining) –for oxide in all deposits	C$/t milled	11.00
Total OPEX (ex. Mining) – for transition in Supremo, DD & Kona	C$/t milled	11.25
Total OPEX (ex. Mining) – for transition in Latte	C$/t milled	11.50
Recovery and Dilution		
Gold Recovery		
Leach Recovery in oxide for Supremo, DD & Kona	%	90
Leach Recovery in oxide for Latte	%	87
Leach Recovery in upper transition for Supremo, DD & Kona	%	70
Leach Recovery in upper transition for Latte	%	44
Leach Recovery in lower transition for Supremo & DD	%	45
Leach Recovery in lower transition for Latte	%	26
External Mining Dilution	%	5
Mining Recovery	%	98
Other		
Overall Pit Slope Angles = interramp slopes flattened for ramp allowance	degrees	variable
Supremo	degrees	45
Latte	degrees	40 - 44
Double Double	degrees	37
Kona	degrees	40
Leach Production Rate (250 operating days per year x 20,000 tpd)	Mtpa	5.0

A series of optimized shells were generated for each of Supremo, Double Double, Latte and Kona deposits of the Coffee project based on varying revenue factors. The results were analyzed with shells selected as the basis for ultimate limits and preliminary phase selection.

16.1.1 Cut-off Grade

Table 16-2 summarizes the incremental (or process) cut-off grade calculations (based on input parameters and design criteria noted in Section 16.1). The incremental (or process) cut-off grade incorporates all OPEX except mining. This incremental 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 Whittle optimization. This process cut-off grade was applied to all of the estimates that follow.

Table 16-2: Gold Cut-off Grade Estimates

Parameter	Unit	Value
Cut-off Grade Calculations		
Supremo, Double Double and Kona Oxide Gold Cut-off Grade	g/t Au	0.31
Latte Oxide Gold Cut-off Grade	g/t Au	0.32
Supremo, Double Double and Kona Upper Transition Gold Cut-off Grade	g/t Au	0.41
Supremo, Double Double and Kona Lower Transition Gold Cut-off Grade	g/t Au	0.63
Latte Upper Transition Gold Cut-off Grade	g/t Au	0.66
Latte Lower Transition Gold Cut-off Grade	g/t Au	1.11

16.1.2 Pit Optimization Results

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 with the results shown.

Note that the NPV in this mine optimization summary does not take into account capital expenditures and is used only as a guide in shell selection and determination of the mining shapes.

As modelled, the block models for Supremo, Latte and Double Double overlap in small areas. To eliminate double counting waste tonnages in the areas of overlap the selected shell for Supremo (39(41)) was used as the topo surface for the optimizations for both Latte and Double Double.

The results of the pit optimization evaluation for each deposit for varying revenue factors are summarized in in Table 16-3 and Figures 16-1 to 16-8 for Indicated and Inferred Resources.

Table 16-3: Summary of Selected Optimized LG Pit Shells

Item	**Unit**	**Supremo**	**Latte**	**Double Double**	**Kona**	**Total**
Pit#		Pit 39 (41)	Pit 34 (37)	Pit 31 (32)	Pit 36 (41)	-
Revenue Factor		0.82	0.74	0.64	0.82	-
Oxide Diluted Heap Leach Feed	*(Mt)*	39.1	10.5	1.2	0.7	51.6
	Au (g/t)	1.13	1.28	3.32	1.22	1.21
	Au (k Oz.)	1,414	434	133	29	2,009
Transition Diluted Heap Leach Feed	*(Mt)*	1.2	0.7	-	-	1.9
	Au (g/t)	1.64	1.83	-	-	1.74
	Au (k Oz.)	63	43	-	-	106
Total Diluted Heap Leach Feed	*(Mt)*	40.2	11.3	1.2	0.7	53.4
	Au (g/t)	1.14	1.32	3.32	1.22	1.23
	Au (k Oz.)	1,476	476	133	29	2,115
Waste	*(Mt)*	171	24	14	4	213
Strip Ratio	*(w:o)*	4.2	2.2	11.2	4.3	4.0
Total Material	*(Mt)*	211	36	15	4	266

Figure 16-1: Supremo Pit Optimization Results - Overall Summary

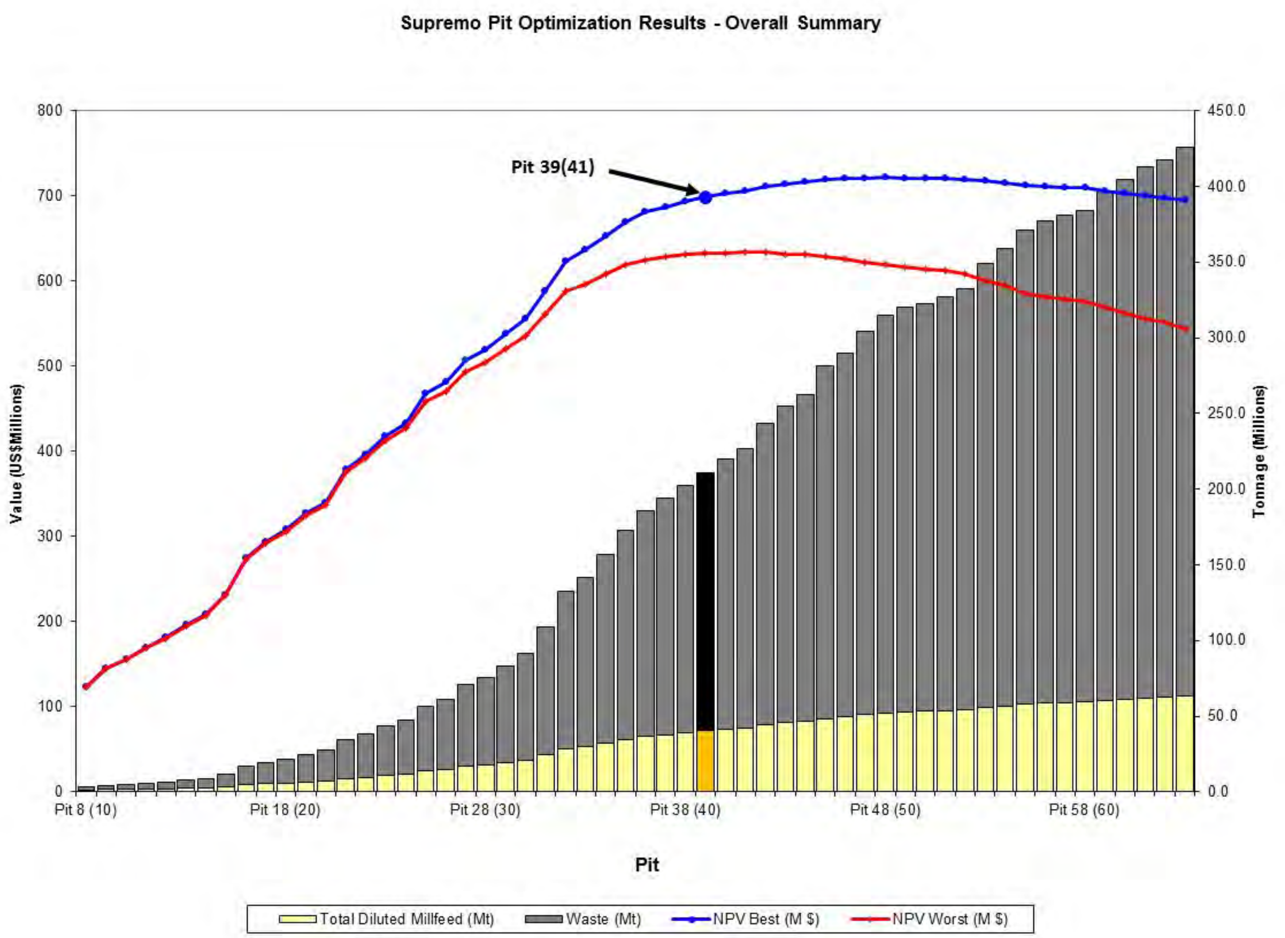
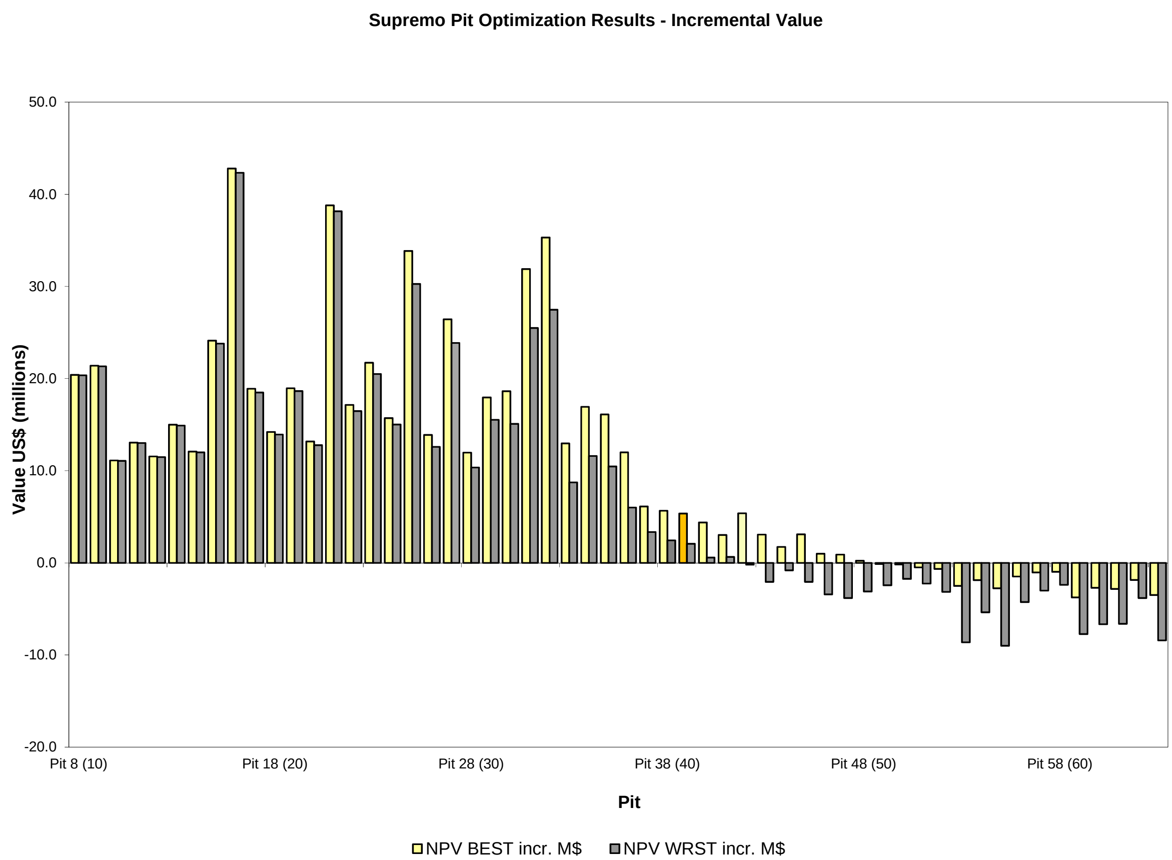


Figure 16-2: Supremo Pit Optimization Results – Incremental Value



For the Supremo deposit, shells beyond Pit Shell 39(41) add mineralized rock and waste tonnages to the ultimate pit, but have higher incremental strip ratios with minimal positive impact on the NPV. Compared to the revenue factor 1 pit shell, the selected shell mines 33% less rock and achieves 97% of the operating best case NPV, and actually 2% higher NPV for the worst case schedule. Based on the analysis of the shells, Pit Shell 39(41) was chosen as the base case ultimate shell for the deposit. In addition, four pit phases were selected based on access logistics and deposit geometry rather than simply on smaller nested shells.

Figure 16-3: Latte Pit Optimization Results - Overall Summary

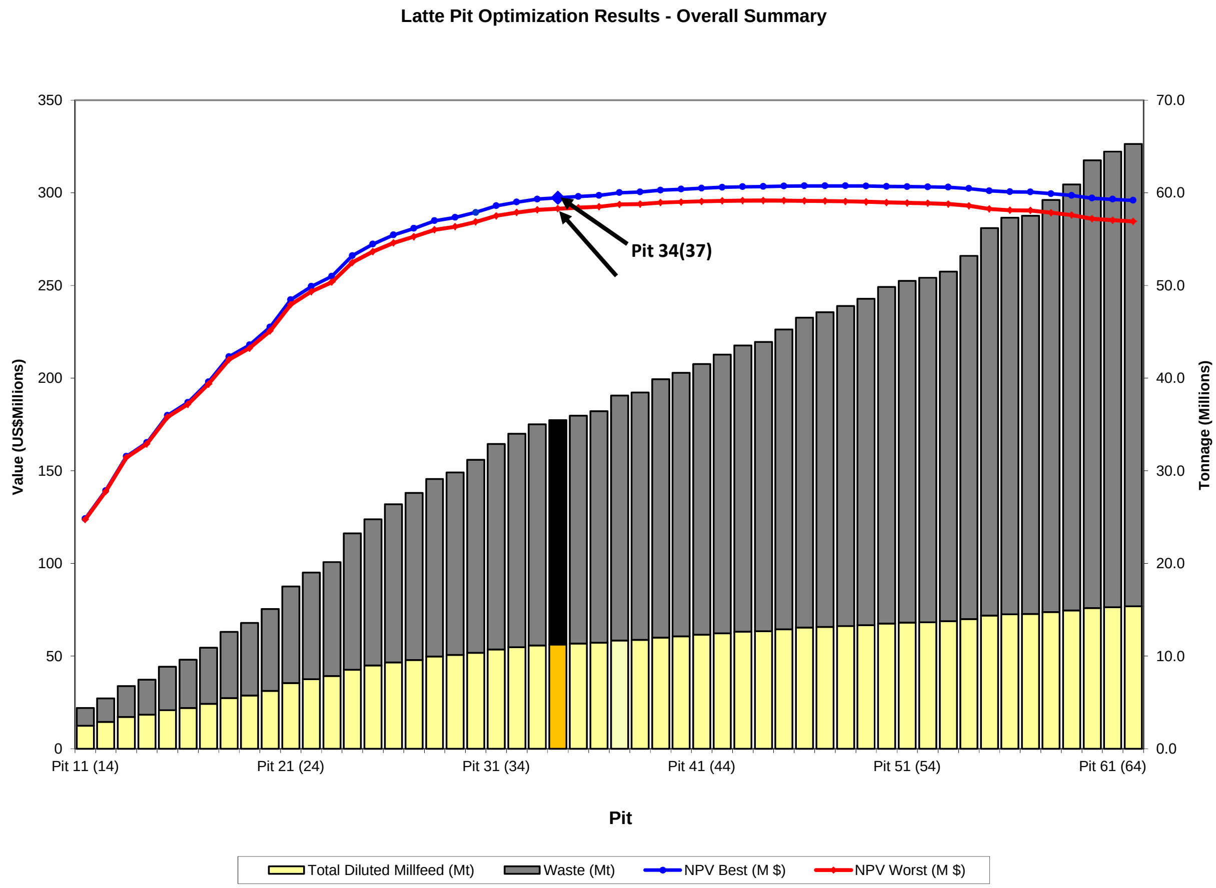
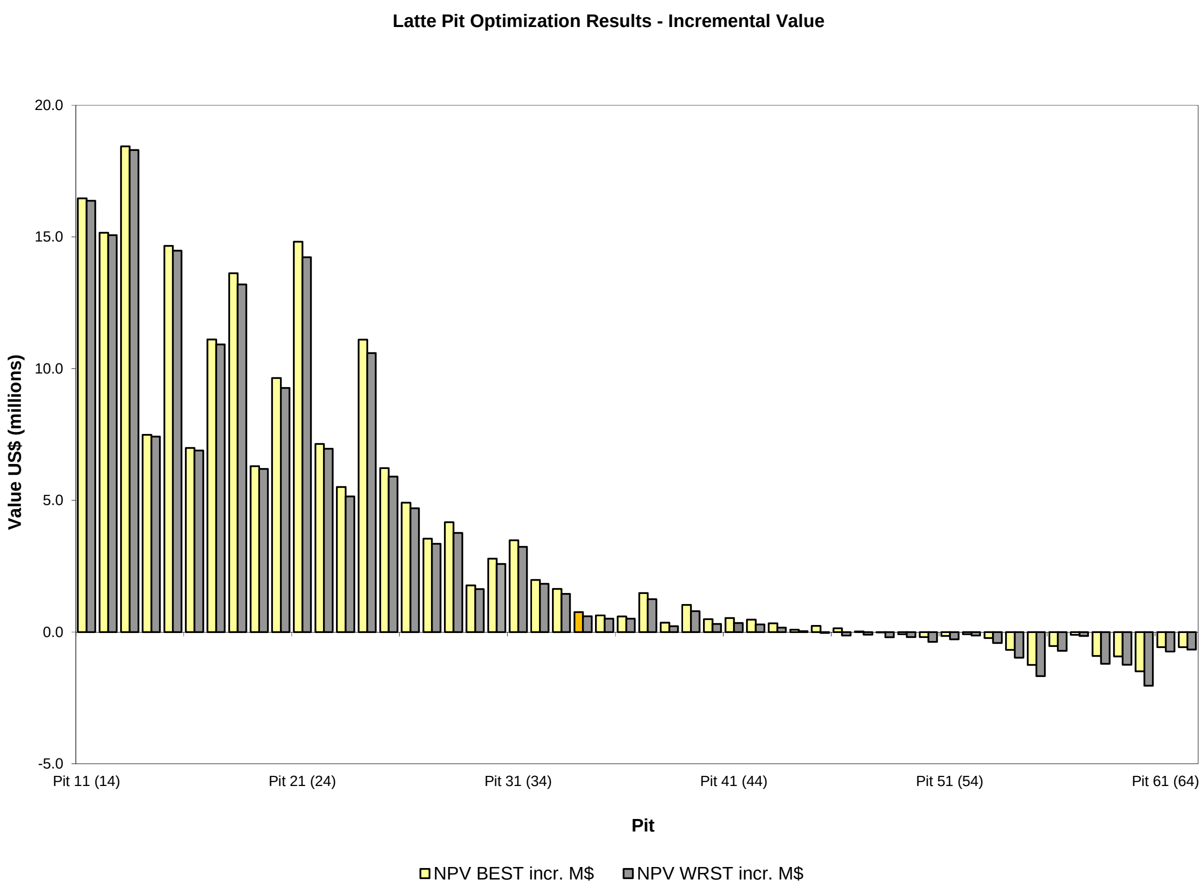
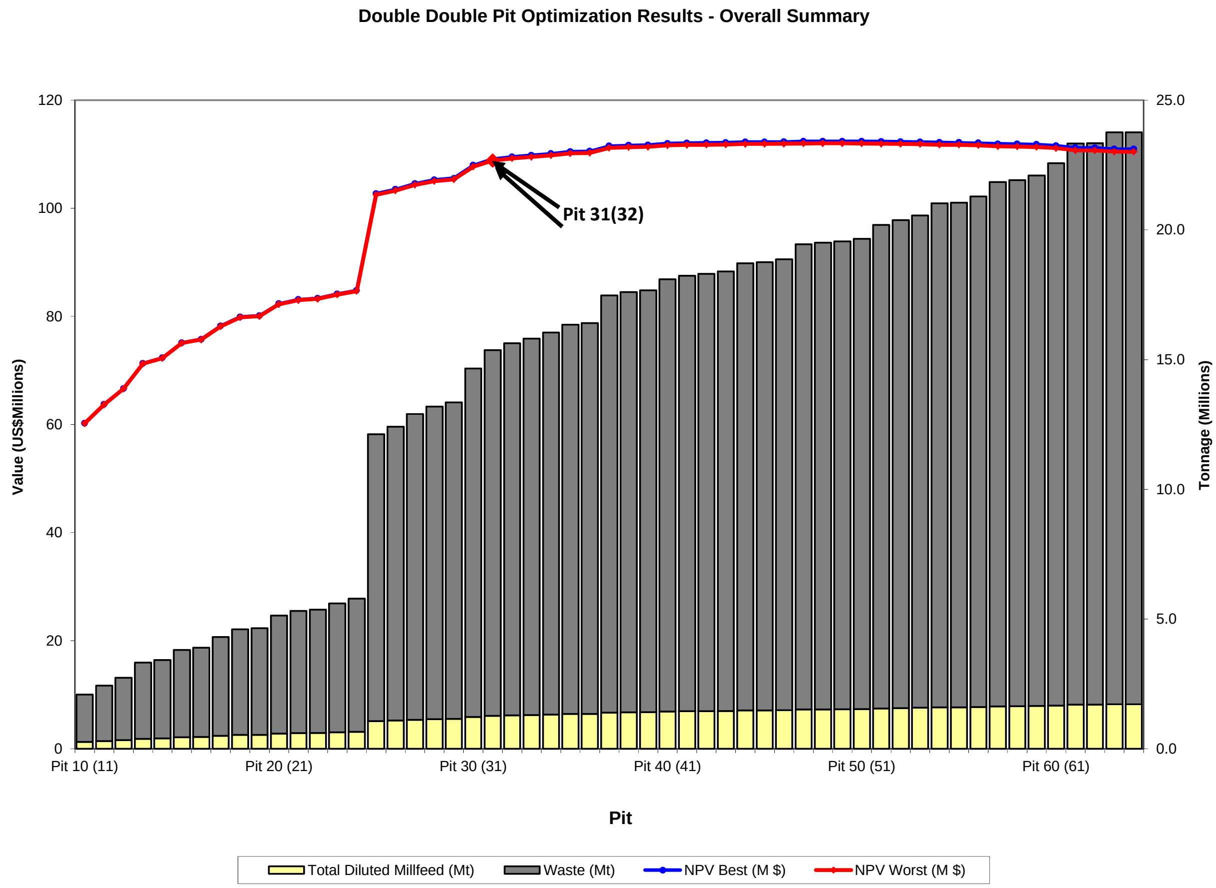


Figure 16-4: Latte Pit Optimization Results – Incremental Value



For the Latte deposit, shells beyond Pit Shell 34(37) add mineralized rock and waste tonnages to the ultimate pit, but have higher incremental strip ratios with minimal positive impact on the NPV. Compared to the revenue factor 1 pit shell, the selected shell mines 25% less rock and achieves 98% of the operating best case NPV, and 99% of operating NPV for the worst case schedule. Based on the analysis of the shells, Pit Shell 34(37) was chosen as the base case ultimate shell for the deposit. In addition, two pit phases were selected based on access logistics and deposit geometry rather than simply on smaller nested shells.

Figure 16-5: Double Double Pit Optimization Results - Overall Summary



For the Double Double deposit, shells beyond Pit Shell 31(32) add mineralized rock and waste tonnages to the ultimate pit, but have higher incremental strip ratios with minimal positive impact on the NPV. Compared to the revenue factor 1 pit shell, the selected shell mines 21% less rock and achieves 97% of the operating best case NPV, and 97% of operating NPV for the worst case schedule. Based on the analysis of the shells, Pit Shell 31(32) was chosen as the base case ultimate shell for the deposit. Due to the small size of the pit, it will be mined in a single phase.

Figure 16-6: Double Double Pit Optimization Results – Incremental Value

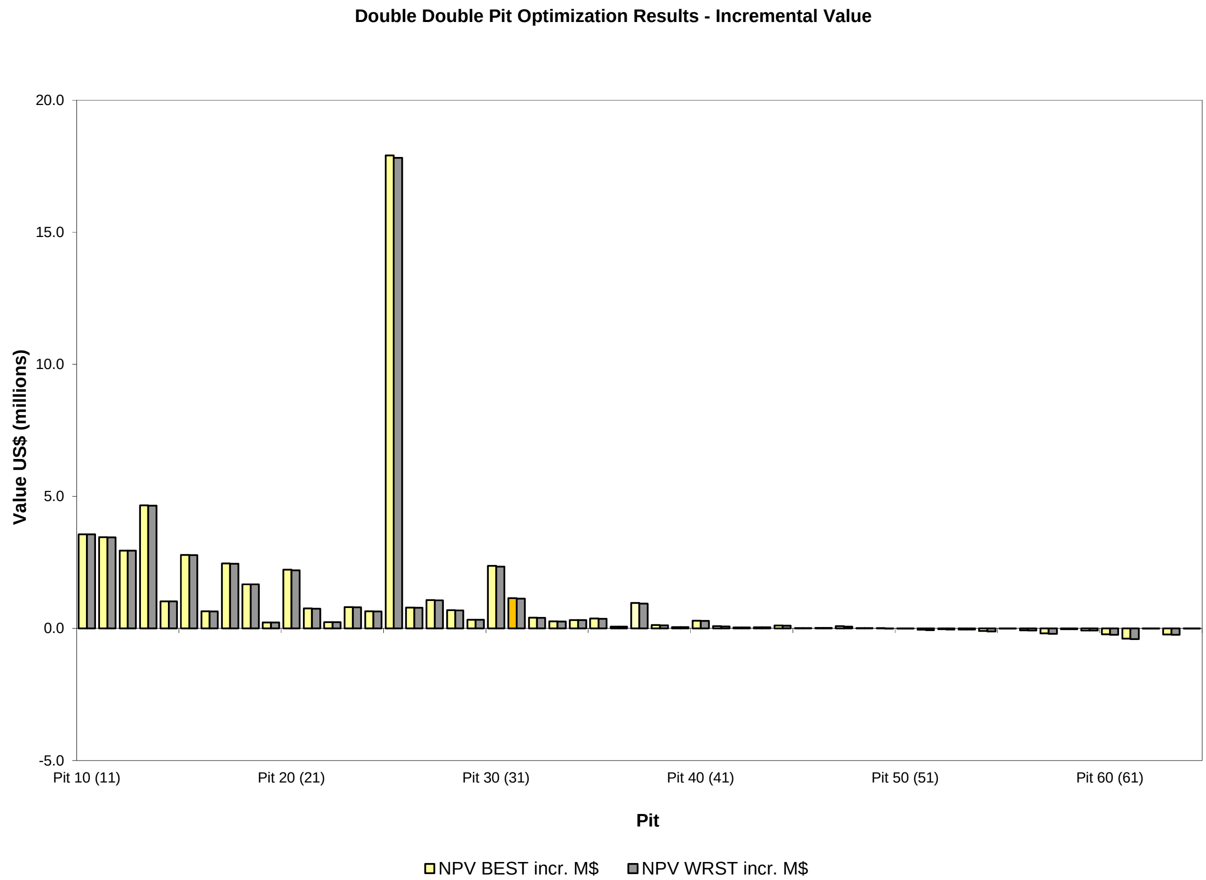
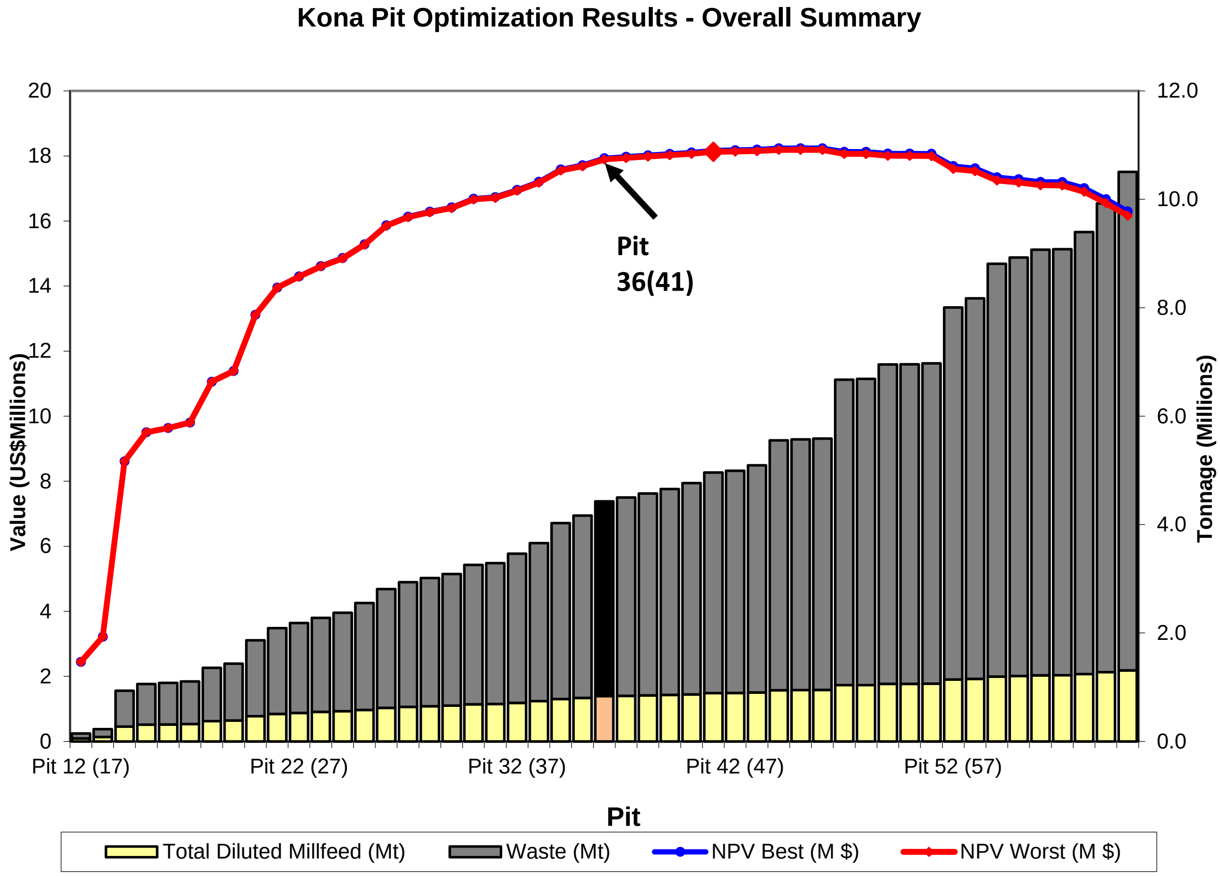
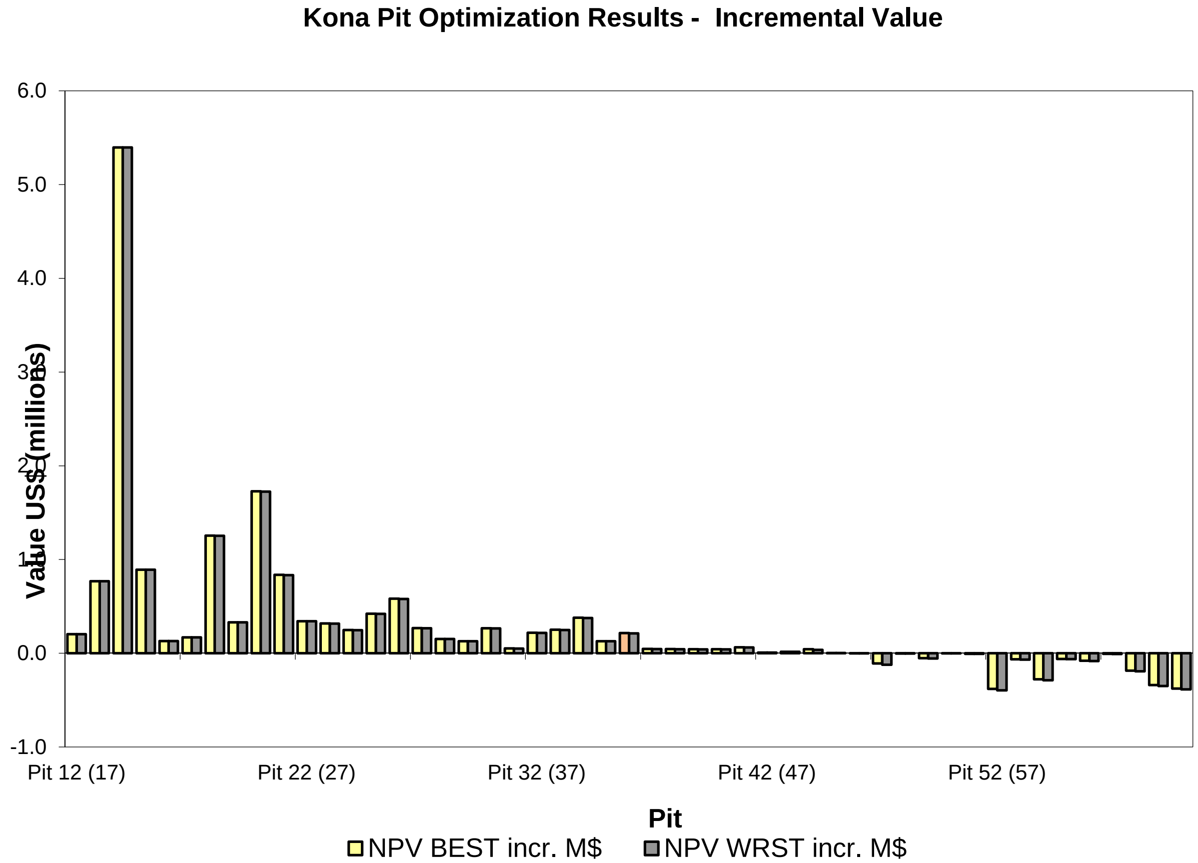


Figure 16-7: Kona Pit Optimization Results - Overall Summary



For the Kona deposit, shells beyond Pit Shell 36(41) add mineralized rock and waste tonnages to the ultimate pit, but have higher incremental strip ratios with minimal positive impact on the NPV. Compared to the revenue factor 1 pit shell, the selected shell mines 20% less rock and achieves 98% of the operating best case NPV, and 98% of operating NPV for the worst case schedule. Based on the analysis of the shells, Pit Shell 36(41) was chosen as the base case ultimate shell for the deposit. Due to the small size of the pit, it will be mined in a single phase.

Figure 16-8: Kona Pit Optimization Results – Incremental Value



16.2 Proposed Mining Method

The Coffee project deposits are amenable to be developed as an OP mine. Mining of the deposits is planned to produce a total of 53.4 Mt of heap leach feed and 212.4 Mt of waste (4.0:1 overall strip ratio) over a twelve-year mine production life (including 2 years of pre-production). The current LOM plan focuses on achieving consistent plant feed production rates, and mining of higher grade material early in the production schedule, as well as balancing grade and strip ratios.

16.3 Geotechnical Criteria

16.3.1 Geotechnical Characterization

A Preliminary Geotechnical Assessment was completed by SRK Consulting (U.S.), Inc. (SRK) to determine appropriate pit slope design parameters for the project. The assessment (SRK, 2013) was based on an on-site review of resource drill core and core logging procedures, review of resource core photographs and logging data, laboratory strength testing, and limited discontinuity orientation data. It was concluded from the assessment that the rock mass quality at the Coffee project will likely be governed by oxidation intensity and the potential for strength anisotropy along foliation. Most of the pit slopes will be comprised of the transition zone and sulphide materials which are generally competent and of good geomechanical quality.

Pervasive foliation such as that occurring within the schists and gneisses at Double Double, Latte and Supremo (and potentially Kona) provide planar surfaces of weakness which, when sufficiently steep and adversely oriented, can result in relatively ubiquitous modes of pit wall instability ranging from bench-scale to high interramp and, with extreme persistence, even to overall slopes.

At the Supremo deposit, the foliation generally dips parallel to the strike of the mineralization or long axis of the pit. This foliation orientation relative to the anticipated pit walls means that the foliation will not produce kinematically viable potential failure surfaces in the east, west or south walls of the proposed pit. On a strictly kinematic basis, the north pit wall will be highly subject to foliation instabilities; however, given the relatively tight radius of curvature expected for the north wall and with the relatively shallow foliation dip, relatively minimal impacts are expected on interramp or overall wall stability for the Supremo pit.

For the Latte and Double Double deposits, foliation dips are sufficiently steeper than at Supremo that shear strength along the foliation surfaces may not be adequate to resist failure development, if daylighted. Additionally, foliation surfaces at the Latte and Double Double deposits are adversely oriented perpendicular to the strike of the mineralization and dip into the pit (from the north wall), thereby leaving the long, continuous north pit walls susceptible to foliation fostered instabilities.

Overall slope stability or failure through the rock mass was verified empirically based on the Haines and Terbrugge (1991) method which indicates that maximum interramp slope angles of approximately 50 to 55° should have a factor of safety of approximately 1.3 for the transitional material based on rock mass failure. This estimate is based on slope heights in the range of 100 to 200 m, using a 10% reduction in rock mass rating (RMR) to estimate mining rock mass rating (MRMR), which is the basis of the Haines and Terbrugge method.

16.3.2 Pit Slope Design Parameters

For the Kona and Supremo deposits, 50° maximum interramp slope angles were used for the PEA mine planning, based on the overall rock mass quality and the lack of apparent adverse structural controls. Given the relatively shallow pit depths, steeper slopes are likely possible at later stages of development after geotechnical specific drilling and testing have been completed and more site specific data supported analyses have been conducted.

At the Latte and Double Double deposits, where foliation is anticipated to be dipping southward, into the pits at angles of 30° to 40°, shallower slope angles will likely be required to minimize the occurrence of multi-bench slope failures fostered by the foliation discontinuities. A 40° maximum interramp slope angle is used for the north wall (those walls with dip directions between 155° and 225° azimuth) of the Latte deposit and a 37° maximum interramp slope angle is used for the north wall (dip directions between 130 to 240° azimuth) of the Double Double deposit. An interramp slope angle of 50° is used for the remaining south, east and west wall slopes at Latte and Double Double.

Slope angles used for the PEA, based on dip direction of the pit wall, are summarized in Table 16-4. The recommendations are presented in terms of pit wall dip direction (e.g. for an east-west trending wall, facing south, the slope dip direction would be 180° azimuth).

Table 16-4: Summary of Pit Slope Angles

Deposit	Wall Dip Direction (azimuth)		Interramp Slope Angle (°)	Comments
	From (°)	To (°)		
Supremo	0	360	50	whole pit at same angle
Kona	0	360	50	whole pit at same angle
Latte	0	155	50	
	155	225	40	controlled by foliation dip
	225	360	50	
Double Double	0	130	50	
	130	240	37	controlled by foliation dip
	240	360	50	

The slope angles used for the PEA are controlled primarily by the necessity to retain suitable catch bench widths and to a lesser extent, overall or global slope stability slope stability. Where necessary, Whittle pit shells were reduced from the maximum interramp slope angles to account for haul roads.

16.4 Open Pit Mine Design

For the Coffee project deposit, the ultimate shell limits, along with the associated phasing, were based on the shell analysis described in this report. Preliminary waste dumps were then designed to account for the material produced in each mining phase and shell.

Table 16.5 through Table 16.6 represent plan and section views of the ultimate pit shapes for the various deposits.

The various pit shell dimensions are summarized in Table 16-4.

Table 16-5: Pit Shell Dimensions

Pit Shell	Length (m)	Width (m)	Depth (m)	Base elev (masl)
Supremo	2,250	800	200	1,025
Latte	1,600	300	110	950
Double Double	700	300	115	980
Kona	400	200	75	1,180

The preliminary pit shells were further analyzed and optimizations were conducted in order to better define the possible stage designs within the ultimate shell limit. The Supremo pit was divided into four phases for the mine plan development to maximize the grade in the early years, reduce the pre-stripping requirements, and to maintain the process facility at full production capacity. Latte was divided into two-phases. The shell tonnages, grades, and contained metal of the preliminary phases (stages) are summarized in Table 16-5.

Table 16-6: Mine Plan Summary by Pit Phase

Pit/Phase	HL Feed	Waste	Total Material	Gold Grade	SR
	(ktonnes)	(ktonnes)	(ktonnes)	(g/t)	(t:t)
Double Double	1,215	13,697	14,912	3.32	11.3
Latte_1	10,632	22,037	32,669	1.32	2.1
Latte_2	629	2,264	2,893	1.27	3.6
Supremo_1(E)	8,583	33,427	42,010	0.92	3.9
Supremo_2(Mid)	8,186	39,529	47,715	1.26	4.8
Supremo_3(S)	11,543	49,006	60,548	1.28	4.2
Supremo_4(N)	11,929	48,714	60,643	1.08	4.1
Kona	730	3,700	4,430	1.22	5.1
Total	**53,448**	**212,374**	**265,821**	**1.23**	**4.0**

Figure 16.2 further illustrates the phase designs for Coffee, with tonnes, grades, and contained metal shown.

The phases were based on the optimized shells summarized above and were selected based on access logistics and deposit geometry rather than simply on smaller nested shells.

During the active mining and processing of the deposit, the waste will be placed into various waste rock dumps adjacent to the final shell limits. All mineralized material would be hauled to the primary crusher. Year -1 and Year -2 HL feed will be hauled directly to the base of the HL facility (Figure 16-3).

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SCALE:
DESIGN. BY J.D.S. May 2 2014
DRAWN BY: B. Wong May 2 2014
CHECK. BY
APP. BY:

JDS Energy & Mining Inc.

Kaminak Gold Corp.

COFFEE GOLD PROJECT
PROJECT LOCATION MAP
PLAN

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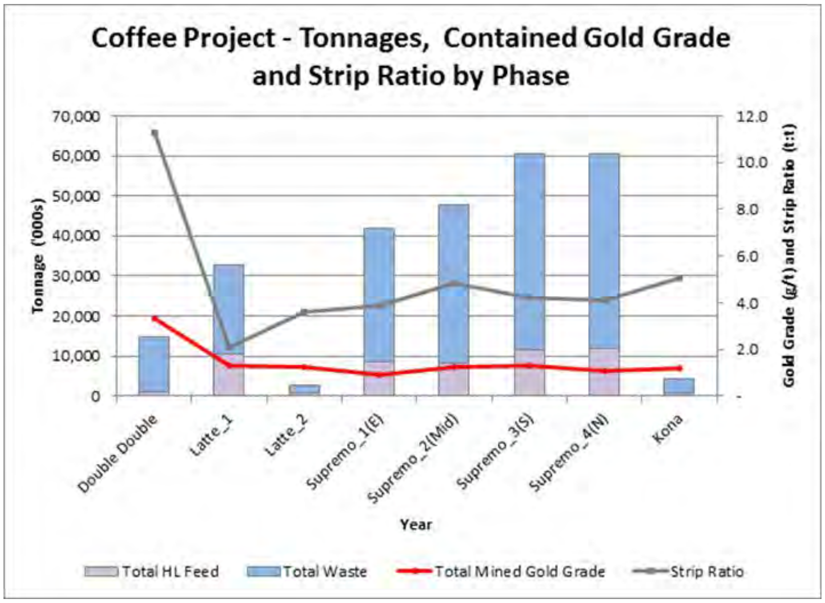
Figure 16-3: Phase Summary



16.5 Mine Production Schedule

The open pit mine production schedule for the Coffee Gold deposit was based on processing higher grade material first, while taking into consideration the deposit geometry and location, topography, and road access from the pits to the waste dumps and primary crusher.

The heap leach throughput was planned at a net annual production of 5.0 mtpa. Pre-production stripping was planned to occur within Year -1 and Year -2, with Year 1 representing the commencement of full-scale processing. The maximum amount of planned total material to be moved from the open pits is approximately 85,000 tpd. The LOM average total open pit mining rate is approximately 65,000 tpd.

Table 16-6 is a summary of total material movement by year for the LOM mine production schedule. In addition, the heap leach processing schedule and stockpile balances are illustrated. The heap leach feed has been split between oxide and transition material.

Table 16-7: LOM Production Schedule

				Year												
Area	**Description**	**Units**	**TOTAL**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Supremo	Oxide HL Feed	ktonnes	**39,000**	-	-	-	763	5,000	5,000	5,000	5,000	5,000	5,000	5,000	4,000	-
	Oxide Mined Gold Grade	g/t	**1.13**	-	-	-	0.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.37	-
	Transition HL Feed	ktonnes	**1,000**	-	-	-	-	42	19	-	32	332	96	95	574	-
	Transition Mined Gold Grade	g/t	**1.64**	-	-	-	-	1.01	1.25	-	1.34	1.04	2.61	3.44	1.60	-
	Total HL Feed	ktonnes	**40,000**	-	-	-	763	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	-
	Total Mined Gold Grade	g/t	**1.14**	-	-	-	0.69	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.40	-
	Waste Mined	ktonnes	**171,000**	-	-	-	5,000	19,000	25,000	22,000	22,000	20,000	20,000	21,000	17,000	-
	Strip Ratio (t:t)	t:t	**4.24**	-	-	-	6.71	3.88	5.00	4.40	4.36	3.90	4.02	4.27	3.68	-
Latte	Oxide HL Feed	ktonnes	**11,000**	852	2,000	5,000	3,000	-	-	-	-	-	-	-	-	-
	Oxide Mined Gold Grade	g/t	**1.28**	1.09	1.13	1.42	1.23	-	-	-	-	-	-	-	-	-
	Transition HL Feed	ktonnes	**725**	-	17	259	450	-	-	-	-	-	-	-	-	-
	Transition Mined Gold Grade	g/t	**1.83**	-	1.24	1.72	1.92	-	-	-	-	-	-	-	-	-
	Total HL Feed	ktonnes	**11,000**	852	2,400	5,000	3,000	-	-	-	-	-	-	-	-	-
	Total Mined Gold Grade	g/t	**1.32**	1.09	1.13	1.44	1.33	-	-	-	-	-	-	-	-	-
	Waste Mined	ktonnes	**24,000**	4,000	7,000	9,000	5,000	-	-	-	-	-	-	-	-	-
	Strip Ratio (t:t)	t:t	**2.16**	4.40	3.03	1.74	1.53	-	-	-	-	-	-	-	-	-
Double Double	Oxide HL Feed	ktonnes	**1,000**	-	-	11	1,200	-	-	-	-	-	-	-	-	-
	Oxide Mined Gold Grade	g/t	**3.32**	-	-	5.05	3.30	-	-	-	-	-	-	-	-	-
	Transition HL Feed	ktonnes	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Transition Mined Gold Grade	g/t	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total HL Feed	ktonnes	**1,000**	-	-	11	1,200	-	-	-	-	-	-	-	-	-
	Total Mined Gold Grade	g/t	**3.32**	-	-	5.05	3.30	-	-	-	-	-	-	-	-	-
	Waste Mined	ktonnes	**14,000**	-	-	2,000	11,000	-	-	-	-	-	-	-	-	-
	Strip Ratio (t:t)	t:t	**11.27**	-	-	211.67	9.49	-	-	-	-	-	-	-	-	-
Kona	Oxide HL Feed	ktonnes	**730**	-	-	-	-	-	-	-	-	-	-	-	730	-
	Oxide Mined Gold Grade	g/t	**1.22**	-	-	-	-	-	-	-	-	-	-	-	1.22	-
	Transition HL Feed	ktonnes	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Transition Mined Gold Grade	g/t	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total HL Feed	ktonnes	**730**	-	-	-	-	-	-	-	-	-	-	-	730	-
	Total Mined Gold Grade	g/t	**1.22**	-	-	-	-	-	-	-	-	-	-	-	1.22	-
	Waste Mined	ktonnes	**3,700**	-	-	-	-	-	-	-	-	-	-	-	4,000	-
	Strip Ratio (t:t)	t:t	**5.07**	-	-	-	-	-	-	-	-	-	-	-	5.07	-
Total Mine	Oxide HL Feed	ktonnes	**52,000**	852	2,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	-
	Oxide Mined Gold Grade	g/t	**1.21**	1.09	1.13	1.43	1.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.34	-
	Transition HL Feed	ktonnes	**2,000**	-	17	259	450	42	19	-	32	332	96	95	574	-
	Transition Mined Gold Grade	g/t	**1.71**	-	1.24	1.72	1.92	1.01	1.25	-	1.34	1.04	2.61	3.44	1.60	-
	Total HL Feed	ktonnes	**53,000**	852	2,400	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	-
	Total Mined Gold Grade	g/t	**1.23**	1.09	1.13	1.44	1.71	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.37	-
	Waste Mined	ktonnes	**212,000**	4,000	7,000	11,000	21,000	19,000	25,000	22,000	22,000	19,000	20,000	21,000	20,000	-
	Strip Ratio (t:t)	t:t	**3.97**	4.40	3.03	2.19	4.24	3.88	5.00	4.40	4.36	3.90	4.02	4.27	3.87	-
	Total Material Mined	ktonnes	**266,000**	5,000	10.000	16,000	26,000	24,000	30,000	27,000	27,000	24,000	25,000	26,000	26,000	-

The Coffee project open pits will produce a total of 53.4 Mt of heap leach feed and 212.4 Mt of waste rock over a twelve year mine operating life (includes 2 years of pre-production), yielding an overall open pit strip ratio of 4:1. The mine schedule focuses on achieving the required heap leach feed production rate, mining of higher grade material early in the schedule, while balancing grade and strip ratios.

A crushed heap leach feed stockpile was designed with a maximum capacity of 1.5 Mt in order to account for the reduced operating time of the heap leach facility during the winter months.

The Coffee deposits are most economical when the open pit phases are mined concurrently. Figure 16-3 summarizes mined heap leach feed and waste tonnages, grade and strip ratio by year, while Figure 16-4 illustrates the total gold mined by year.

Figure 16-4: LOM Mined Tonnes, Grade and Strip Ratio

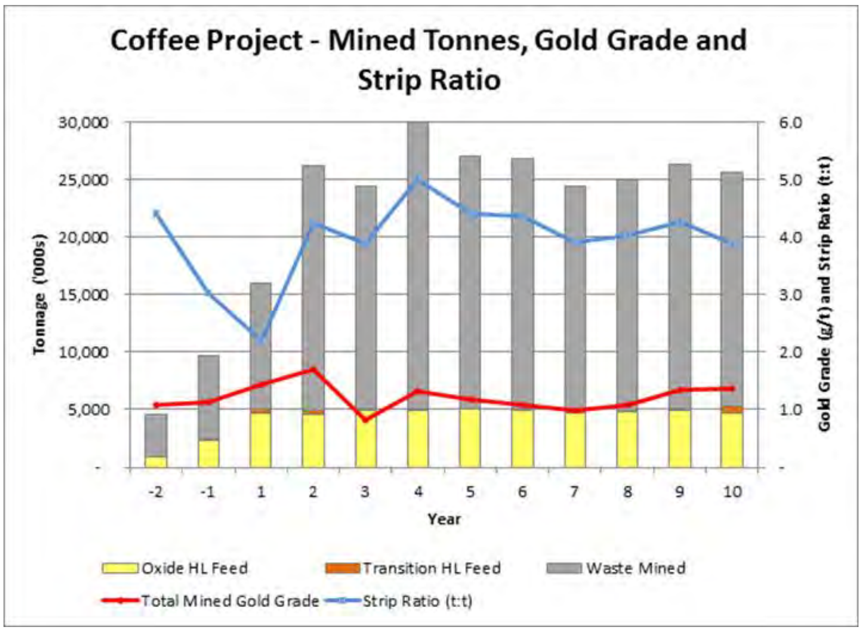
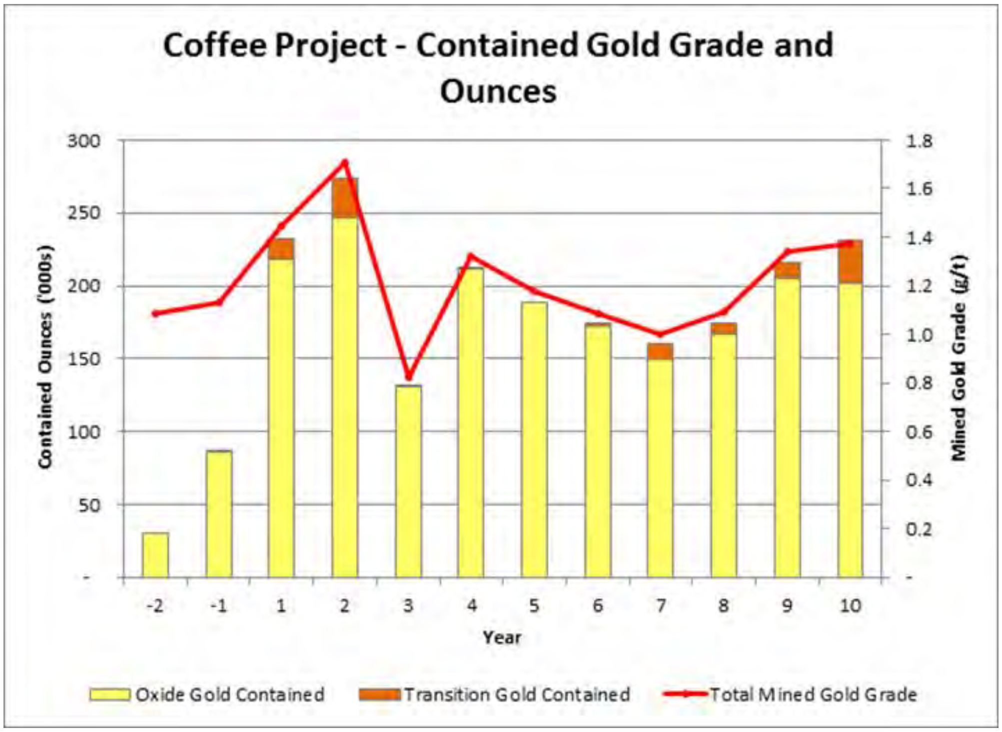


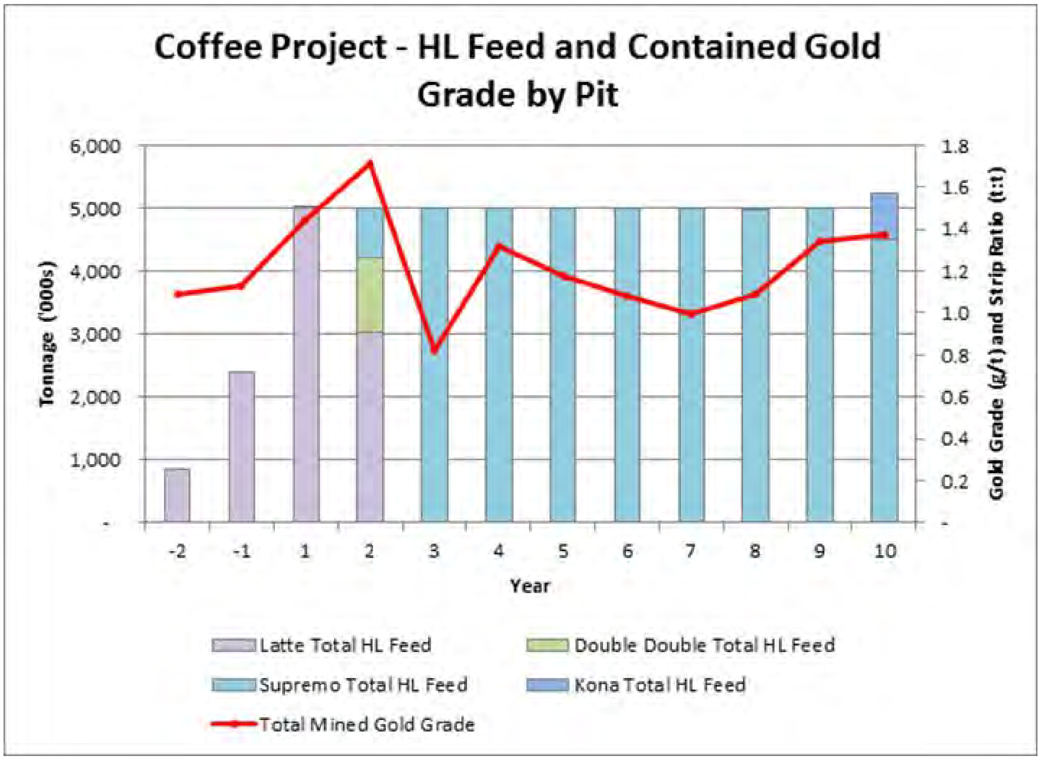
Figure 16-5: Contained Gold by Period



To further illustrate the progression of mining of the Coffee project deposits, Figure 16-6 illustrates the various mined pit tonnages and overall mined gold grade by year.

Layout drawings with the annual year-end statuses of the open pit configuration, waste dump advance, and the heap leach facility are shown in Appendix B.

Figure 16-6 Heap Leach Feed and Gold Grade



16.5.1 Open Pit Development

Year -2: Open Pit mining commences with development and pre-stripping of Phase 1 of the Latte pit. A total of 3.8 Mt of waste is mined in period (for construction of the heap leach embankment) along with a total of 0.8 Mt of heap leach feed at a mined gold grade of 1.1 g/t (this ROM material forms a portion of the 2.0 Mt of material required to fill the pond area of the HLF).

Year -1: Pre-production mining continues with Phase 1 of the Latte pit. Total of 7.3 Mt of waste mined in period (to be placed in appropriate waste dump) along with total of 2.4 Mt of heap leach feed at a gold grade of 1.1 g/t (1.2 Mt of heap leach feed to complete the material required to fill the pond area of the HLF, while remaining feed is sent to primary crusher).

Year 1: First year of full-scale heap leach processing (at planned throughput rate of 5 mtpa) with mining continuing in Latte Phase 1 along with commencement of the Double Double pit. A total of 5.0 Mt of heap leach feed is mined in the year of which 4.7 Mt is oxide (remainder is transition material). Heap leach head grade for the year averages 1.43 g/t Au. 10.9 Mt of waste produced for an open pit mined strip ratio of 2.2:1 (waste tonnes: heap leach feed tonnes).

Year 2: Mining in Latte and Double Double is completed in the period and commences in the Supremo Phase 1(East) pit. Average heap leach head grade achieves LOM maximum of 1.69 g/t Au. The waste produced over the period increases to 26 Mt with heap leach feed target reached for a strip ratio of 4.2:1.

Year 3: Mining is concentrated in the Supremo Phase 1 (East) pit. Average heap leach head grade drops to LOM minimum of 0.82 g/t Au. The waste produced over the period increases to 24 Mt. This phase is mined early in the schedule because due to existing topography, the deposit locations and planned site layout, once mining commences in Supremo Phase 3 (South), and some portions of Supremo Phase 2 (Mid), haul road access will be cut off to both the crusher and waste dumps.

Year 4: Mining in Supremo Phase 1 (East) continues and also commences in Supremo Phase 2 (mid). Average heap leach head grade increases to 1.32 g/t Au. The waste produced for the period totals a LOM maximum of 30 Mt for a strip ratio of 5.0:1. Steady state heap leach feed of 5.0 mtpa is maintained.

Years 5 to 7: Through this period mining in Supremo Phase 1 and 2 is completed, and final phases in Supremo are commenced (Phase 3 (South) and Phase 4 (North). Average heap leach head grade over the period is 1.1 g/t Au. The waste produced over the three year period totals 78 Mt at an average strip ratio of 4.0:1.

Years 8 to 10: Mining is completed in final phases of Supremo along with the smaller deposit at Kona. Average heap leach head grade is 1.3 g/t Au at a feed rate of 5.0 mtpa. Average waste produced is 26 mtpa.

16.6 Mine Operations

The mining activities for the Coffee project were assumed to be undertaken by an owner-operated fleet as the basis for this preliminary economic assessment. The average unit mining costs used in the project economics was $2.48/t of material mined, for pit and dump operations, road maintenance, mine supervision, and technical services. The cost estimate was built from first principles and based on experience of similar sized OP operations and local conditions. The OP mining costs take into account variations in haulage profiles, production schedule and equipment selection. Labour rates were estimated using local information.

16.6.1 Equipment

The major mining equipment requirements, as indicated in Table 16-8, summarize the assumed, all diesel, equipment requirements used for the basis of this study, and are based on similar sized OP operations. The proposed heap leach processing rate of 5.0 Mtpa was used, along with deposit and pit geometry constraints, to estimate the mining equipment fleet needed. The fleet has an estimated maximum capacity of 90,000 t/d total material, which would be sufficient for the LOM plan.

Table 16-8: Major OP Equipment Requirements

Equipment Type	No. of Units
250 mm dia. rotary, crawler drill	3
165 mm dia. rotary, crawler drill	2
16 m^3 front shovel	2
12 m^3 wheel loader	1
136t haul truck	14
16H-class grader	2
D10-class track dozer	4
824H-class wheel dozer	1
115 mm dia. rotary, crawler drill	1
136 t water truck	1

16.6.2 Unit Operations

Blast borehole drills with a diameter of 250 mm are planned to perform the bulk of the production drilling in the mine (both mineralized and waste rock). The 165 mm drills will be primarily used in the mineralized zones to allow for better definition drilling. The hydraulic drill with a 115 mm diameter bit will be used for secondary blasting requirements and may be used on the tighter spaced patterns required for pit development blasts. The main loading and haulage fleet is planned to consist of 136 t haul trucks, loaded primarily with the diesel powered 16 m^3 front shovels or the 12 m^3 wheel loader, depending on pit conditions.

As pit conditions dictate, the D10-class dozers are planned to rip and push material to the excavators and maintain the waste dumps and stockpiles.

The additional equipment listed in Table 16-8 is planned to be used to maintain and build access roads and to meet various site facility requirements, including stockpile maintenance and further exploration development.

16.6.3 Mine Personnel

The open pit labour requirements are based on experience for similar gold operations of this size. The labour requirements are divided into salaried and hourly personnel and the mine operations consists of four areas:

- Supervision – The Supervision area is responsible for the direction of the mine equipment, drilling and blasting operations and the safety and welfare of the equipment operators and blast loading personnel.
- Load and Haul – The Load and Haul area includes equipment operators skilled in running shovels, loaders, excavators, trucks, tracked dozers and graders.
- Drill and Blast (D&B) – The D&B area includes skilled drill operators, as well as blast loading personnel. Also included are any contract explosives personnel.
- Mine Maintenance – The Mine Maintenance area will consist of supervisors who will monitor the skilled maintenance personnel who will be responsible for maintaining, repairing, fuelling and lubricating the mobile mine equipment.
- In addition, Technical Services personnel are responsible for mine engineering, geology and grade control surveying and IT/communication services.

The open pit mine operations require a total average of 100 personnel, mine maintenance requires 35 personnel and supervision/technical needs a total of 30 personnel, for a total of 165 open pit personnel. The work schedule is based on two 12 hour shifts, seven days a week, 365 days per year.

Table 16-9: Mining and Technical Support Personnel

Description	-2	-1	1	2	3	4	5	6	7	8	9	10	11	Average
Driller	3	7	12	18	17	20	18	18	17	17	18	18	-	**15**
Blaster	2	2	2	2	2	2	2	2	2	2	2	2	-	**2**
Blast Helper	4	4	4	4	4	4	4	4	4	4	4	4	-	**4**
Shovel/Loader Operator	2	4	6	10	10	12	11	10	10	10	10	10	-	**9**
Truck Driver	6	9	20	34	32	44	42	34	31	35	33	27	-	**29**
Track Dozer Operator	7	10	13	13	13	13	13	13	13	13	13	13	-	**12**
RT Dozer Operator	4	4	4	4	4	4	4	4	4	4	4	4	-	**4**
Grader Operator	7	7	7	7	7	7	7	7	7	7	7	7	-	**7**
Water Truck Driver	4	4	4	4	4	4	4	4	4	4	4	4	-	**4**
Labourer/Trainee	8	8	8	8	8	8	8	8	8	8	8	8	-	**8**
Vacation/Sick/Absentee Operator	2	3	5	6	6	7	7	6	6	6	6	6	-	**6**
Mining Sub-Total	**49**	**62**	**85**	**110**	**106**	**124**	**120**	**111**	**105**	**110**	**109**	**103**	**-**	**100**
Heavy Equip. Mechanic	4	4	6	7	7	8	8	7	7	7	7	7	-	**7**
Welder/Mechanic	4	4	6	7	7	8	8	7	7	7	7	7	-	**7**
Electrician/Instrumentation	4	4	6	7	7	8	8	7	7	7	7	7	-	**7**
Lube/PM Mechanic	4	4	6	7	7	8	8	7	7	7	7	7	-	**7**
Tireman	2	3	4	4	4	5	5	4	4	4	4	4	-	**4**
Labourer/Trainee	2	3	4	4	4	5	5	4	4	4	4	4	-	**4**
Vacation/Sick/Absentee Mechanic	1	1	1	2	2	2	2	2	2	2	2	2	-	**2**
Maintenance Sub-total	**21**	**23**	**33**	**38**	**38**	**44**	**44**	**38**	**38**	**38**	**38**	**38**	**-**	**35**
Mining Manager	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4	-	4
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Maintenance Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4	-	4
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Engineer	2	2	2	2	2	2	2	2	2	2	2	2	-	2
Ore Control Engineer	2	2	2	2	2	2	2	2	2	2	2	2	-	2
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Mine Geologist	2	2	2	2	2	2	2	2	2	2	2	2	-	2
Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	-	2
Survey Assistant	4	4	4	4	4	4	4	4	4	4	4	4	-	4
Technician/Ore Control	2	2	2	2	2	2	2	2	2	2	2	2	-	2
Clerk	1	1	1	1	1	1	1	1	1	1	1	1	-	1
Technical Support Total	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**30**	**-**	**30**

16.6.4 Mine Equipment Maintenance

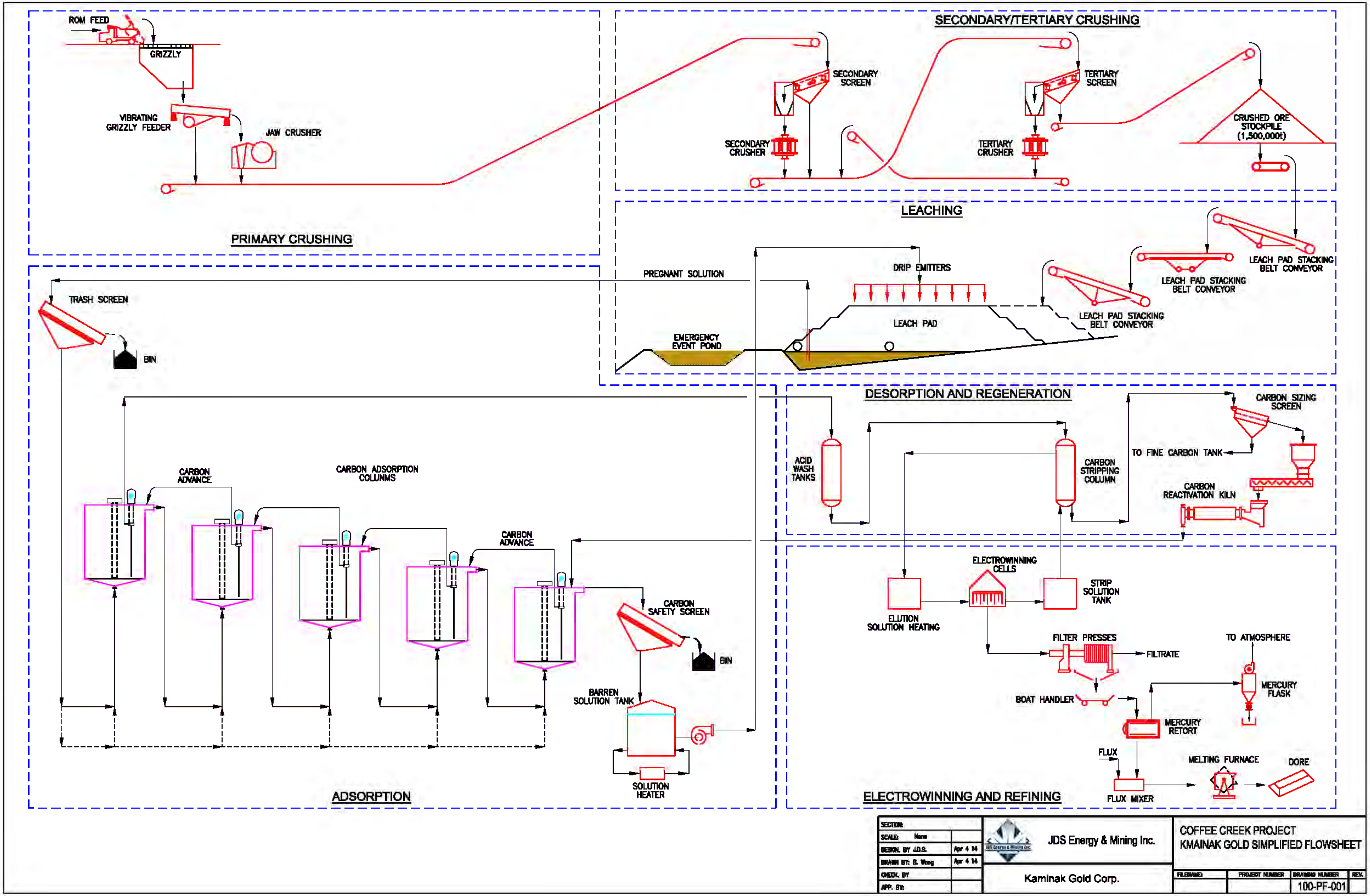
Single original equipment manufacturers (OEM) are recommended for the shovels and excavators, drills, trucks and support equipment. A single site-wide engine power supplier for haul trucks and support equipment is also recommended and serves to significantly reduce maintenance and supply chain direct and indirect costs. The on-site OEM maintenance support personnel are then reduced to one supplier over the entire fleet; parts procurement, shipping and storage is minimized; shop space and tooling is reduced; safety and training is reduced; and parts are interchangeable between all units.

17.0 RECOVERY METHODS

17.1 Process Selection and Description

Flowsheet selection was based upon results of laboratory test work. The process flowsheet includes a three-stage crushing plant followed by a heap leach operation. Gold is extracted by an ADR carbon plant. The process flowsheet is based on an HL processing rate of 5.0 million dry tonnes per year (14,286 dry tonnes per day assuming 350 operating days per year). The three-stage crushing plant will also operate at the 14,286 tonnes per day capacity. During the coldest part of the year heap loading activities will be suspended and heap loading is only scheduled for 250 days per year at a design rate of 20,000 tonnes per day. The process plant will be located near the HLF to minimize the pumping and pipeline requirements for pregnant and barren solutions.

A simplified process flow sheet is presented in Figure 17-1.


PRIMARY CRUSHING
ROM FEED
GRIZZLY
VIBRATING GRIZZLY FEEDER
JAW CRUSHER
SECONDARY/TERTIARY CRUSHING
SECONDARY SCREEN
SECONDARY CRUSHER
TERTIARY SCREEN
TERTIARY CRUSHER
CRUSHED ORE STOCKPILE (1,500,000t)
LEACHING
PREGNANT SOLUTION
DRIP EMITTERS
LEACH PAD
EMERGENCY EVENT POND
LEACH PAD STACKING BELT CONVEYOR
LEACH PAD STACKING BELT CONVEYOR
LEACH PAD STACKING BELT CONVEYOR
ADSORPTION
TRASH SCREEN
BIN
CARBON ADVANCE
CARBON ADSORPTION COLUMNS
CARBON ADVANCE
CARBON SAFETY SCREEN
BIN
BARREN SOLUTION TANK
SOLUTION HEATER
DESORPTION AND REGENERATION
ACID WASH TANKS
CARBON STRIPPING COLUMN
CARBON SIZING SCREEN
TO FINE CARBON TANK
CARBON REACTIVATION KILN
ELECTROWINNING AND REFINING
ELECTROWINNING CELLS
ELUTION SOLUTION HEATING
STRIP SOLUTION TANK
FILTER PRESSES
FILTRATE
TO ATMOSPHERE
MERCURY FLASK
BOAT HANDLER
MERCURY RETORT
FLUX
FLUX MIXER
MELTING FURNACE
DORE
SECTION:
SCALE: None
DESGN. BY: J.D.S. Apr 4 14
DRAWN BY: B. Wong Apr 4 14
CHECK. BY
APP. BY:
JDS Energy & Mining Inc.
Kaminak Gold Corp.
COFFEE CREEK PROJECT
KMAINAK GOLD SIMPLIFIED FLOWSHEET
FILENAME
PROJECT NUMBER
DRAWING NUMBER
REV.
100-PF-001

17.1.1 Primary Crushing

Run-of-mine (ROM) HL feed is trucked from the mine and normally dumped directly into a primary feed hopper. HL feed is drawn out of the feed hopper by an apron feeder discharging onto a vibrating grizzly. The grizzly oversize feed the primary jaw crusher. A small stockpile of ROM HL feed will be available to allow mining operations to continue if the primary jaw crusher is not operating. The grizzly underflow and jaw crusher product are transported to the secondary crusher screen on the secondary crushing circuit feed conveyor, which is equipped with a metal detector and magnet.

17.1.2 Secondary Crushing and Screening

Ore from the secondary crushing plant feed conveyor is transported to the secondary crusher vibrating screen. Screen undersize material by-passes the secondary cone crusher directly to the tertiary screen feed conveyor. Screen oversize material feeds the secondary cone crusher. The secondary cone crusher discharge and screen underflow are transported to the tertiary circuit on the tertiary screen feed conveyor.

17.1.3 Tertiary Crushing and Screening

The tertiary crushing circuit includes two cone crushers in closed circuit with a two horizontal vibrating screens. The circuit feed reports to the tertiary vibrating screen feed box where the screen feed is split to the two screens. Tertiary screen undersize material is dropped directly on the final crusher product conveyor. Tertiary screen oversize material reports to a transfer conveyor that in turn feeds the tertiary crusher feed conveyor. The feed conveyor discharges to a surge bin from which it is drawn by vibrating feeders to the tertiary crusher feed conveyors. Product discharged from the tertiary cone crushers is collected on the tertiary screen feed conveyor and delivered to the screen feed surge bin from which it is fed the tertiary screens.

The need for tertiary crushing for the oxide mineralized material types is considered marginal; however, for this study tertiary crushing is included. The final product conveyor discharges to a final crusher product stockpile via a radial stacker or alternatively directly to the heap stacking conveyor system. The tertiary crusher could be by-passed if the tertiary crusher vibrating screen is opened.

17.1.4 Crushed HL feed Stockpile

During the heap construction season the heap will be constructed from two sources. The crushing plant product will be directly placed on the heap at the nominal rate of 15,000 tpd. In addition 5,000 tpd will be reclaimed from the crushed HL feed stockpile by front end loader and portable conveyors.

When heap loading operations are suspended all crushed HL feed will be fed to the crushed HL feed stockpile.

17.1.5 Heap Loading

Final crushed HL feed is reclaimed from the final crusher product stockpile by a front end loader and portable conveyors and fed onto the heap loading main header feed conveyor to which a lime is added for pH control. The heap loading main header feed conveyor discharges onto a series of for heap construction grasshopper-type portable conveyors and utilizes the same self-propelled radial stacker used to build the stockpile.

17.1.6 Heap Leach Facility (HLF)

Kaminak engaged Knight Piésold and Co. (KP) to develop pre-conceptual plans, for a valley fill heap leach facility (HLF). Details of KP work are summarized below from the report titled: "Kaminak Gold Corporation, Coffee Project, Heap Leach Facility, Report on Pre-Conceptual Design, May 21, 2014".

As currently designed, the HLF would allow HL feed stacking to a height of approximately 128 m, which results in a capacity of 60 million tonnes. The HLF components consist of the following:

- Underdrain system to capture and transport flow from seeps and springs from under the HLF
- In-heap pond storage of pregnant solution
- In-heap pond liner system that comprises from bottom to top:
  - prepared sub-base of 300 mm of compacted fine-grained soil
  - secondary 80-mil double-sided textured linear low-density polyethylene (LLDPE) geomembrane
  - leachate collection and recovery system (LCRS)
  - primary 80-mil double-sided textured LLDPE geomembrane
- Heap leach pad liner system that comprises from bottom to top:
  - prepared sub-base 300 mm of compacted fine-grained soil
  - 80-mil double-sided textured LLDPE geomembrane
  - Solution collection system
  - Vertical solution collection wells
  - Lined event pond proposed to be located downstream of the in-heap pond embankment
  - Permanent and interim perimeter diversion channels to manage surface water flows

Barren solution would be applied to the heap using drip emitters (that would be buried for winter operations) which would flow through the leach HL feed to the in-heap pond. Pregnant solution would be recovered from the in-heap pond through a series of three solution collection wells located at the lowest portion of the in-heap pond. Pregnant solution would be pumped directly to the process plant for gold extraction and re-circulation back to the HLF.

The proposed HLF would be constructed in four stages, with the initial stage beginning in Year -2 and the final stage commencing by Year 6. Work in Years 8, 10 and 12 simply comprises stacking of additional HL feed within the established facility footprint. A construction schedule developed contemplates approximately 20 months of total construction duration for the initial stage. It was assumed that construction would begin in late March of Year -2 and that leaching of leach mineralized materials within the in-heap pond would commence in the fourth quarter of Year -1.

Construction quantities for earthworks, geosynthetics, pipework and concrete work were estimated based on staged construction. Unit rates were established for each of the major construction activities and a CAPEX cost estimate was developed. The estimated cost for the initial stage of the HLF construction is US$30.9 million. The total CAPEX cost for the HLF (including staged expansion) is estimated to be US$47.2 million.

17.1.7 Carbon Adsorption

The pregnant solution will be pumped to the carbon adsorption circuit. The solution will be pumped across a stationary trash screen for trash removal. The carbon adsorption circuit consists of two parallel trains of five cascading carbon columns. The solution flows counter current to the flow of carbon and the solution overflow from the final column will discharge onto a carbon safety screen to catch any entrained carbon. The barren solution that will discharge from the final carbon column drains to barren solution tanks. Cyanide solution, liquid caustic and antiscalent are added to the barren as needed. Barren solution is then pumped back to the leach pad. The loaded carbon from the first carbon column is advanced to the desorption circuit. At this point of the process, the carbon in each column will be advanced one column and fresh carbon will be placed in the fifth carbon column. Recessed impeller pumps will be used to move carbon between the columns and to the loaded carbon screen to minimize carbon abrasion.

17.1.8 Gold Recovery and Refining

The loaded carbon will be advanced to the loaded carbon screen for dewatering and then the loaded carbon is transferred into an acid wash tank. The acid wash removes any scale that accumulates on the surface of the activated carbon. At the conclusion of the acid wash cycle, a dilute caustic solution will be used to wash the carbon and to neutralize the acid solution. The wash solution will be discharged to the barren solution for use in the heap leaching circuit. Then, the washed carbon transfer pump will move the carbon from the acid wash tank to the carbon strip vessel.

The caustic soda strip solution is pumped through in-line heaters and a heat exchanger before entering the bottom of the strip vessel. The pregnant solution that flows out of the top of the strip vessel will flow to an electrowinning cell. At the conclusion of the strip cycle, the stripped carbon will be pumped to the stripped carbon dewatering screen.

The dewatered carbon will discharge into the carbon reactivation kiln feed hopper. The processed carbon will be thermally regenerated in the carbon reactivation kiln to maintain its activity. Regenerated carbon is discharged into a carbon quench tank.

Gold will be plated onto knitted-mesh steel wool cathodes in the electrowinning cell. Cathodes go to a wash station where the gold-bearing sludge and any remaining steel wool will be removed. The gold-bearing sludge and steel wool will be retorted to remove any mercury. The retort residue will be mixed with fluxes and then smelted in an induction furnace to produce gold doré and slag. The doré will be transported to a refiner for further purification. Slag is processed to remove prills for re-melting in the furnace.

17.2 Process Design Criteria

The design particle crush size is minus 16 millimeters (80% passing 12.5 mm), however, mineralized material types found to leach adequately at coarser sizes could bypass the tertiary crushing circuit. Based on experience gained during actual operations, the crush size for each HL feed type may be modified as conditions permit.

Several considerations to adequately mitigate the Yukon climate have been included in the general design criteria:

- All conveyors will be fitted with covers
- A valley fill heap configuration with an in-heap solution pond and temperature monitoring for pregnant leach solution
- Burying the drip emitter lines
- Ability to heat a portion of the barren solution
- Heat tracing of and insulation of the barren solution tank and pipelines
- Dedicated stand by generators for backup power supply to pregnant and barren solution pumps.
- All crushing and process buildings will be a pre-engineered steel structures with insulated steel roofs and walls.

The process design criteria are summarized in the Table 17-1.

Table 17-1: Process Design Criteria

	Unit	Value
General		
Annual Treatment Rate	mtpy	5,000,000
Crushing Plant Operation	days/year	350
Crushing Plant Operation	mtpd	14,286
Crushing Plant Operation	hrs/day	18
Design Rate Crushing Plant Operation	mtph	794
Heap Loading Operation	days/year	250
Heap Loading Operation	mtpd	20,000
Heap Loading Operation	hrs/day	18
Design Rate Heap Loading Operation	mtph	1,111
Ore Characteristics		
Specific Gravity (Average)	mt/m3	2.4
Dry Crushed HL feed Bulk Density	mt/m3	1.6
Run-of-Mine Moisture	%	5.0
Crushing		
Days per Week	d	7
Days per Year	d	350
Shifts per Day	shifts	2
Shift Length	hrs	12
Crusher Availability	%	75
Hours per Day	hrs	18
Primary Crusher Feed		
Type		Vibrating Grizzly
Size	mm	1470 x 6100
Motor	hp	37
Type		Vibrating Grizzly Scalper
Size	mm	1400 x 2440
Motor	hp	7
Primary Crusher		
Type		Jaw
Size	mm	1070 X 1370
Closed Side Setting	mm	102
Motor	hp	225
Product Size, 80 % Passing	mm	
Secondary Screen Feed Conveyor		
Type		Belt
Width	mm	1,220
Motor	kW	36
Secondary Crusher Screen		
Type		Vibrating, Double Deck
Size	mm	2440 x 6100
Top Screen Deck Aperture	mm	64
Bottom Screen Deck Aperture	mm	38
Motor	kW	37
Secondary Crusher		
Type		Standard Cone Crusher
Size		TRIO TC66
Motor	kW	260
Closed Side Setting	mm	25
Tertiary Screen Feed Conveyor		
Type		Belt
Width	mm	1,220
Motor	kW	37
Tertiary Crusher Screen		
Number of Screens	2	
Type		Vibrating, Double Deck
Size	mm	2440 x 6096
Top Screen Deck Aperture	mm	25
Bottom Screen Deck Aperture	mm	19
Motor	kW	37
Tertiary Crusher		
Number of Crushers	2	
Type		Short-Head Cone Crusher
Size		TRIO TC66
Motor	kW	260
Closed Side Setting	mm	13
Tertiary Crusher Transfer Conveyor		
Type		Belt
Width	mm	1,220
Motor	kW	33.5
Final Crusher Product Conveyor		
Type		Belt
Width	mm	1,220
Product, 80% minus	mm	12
Motor	kW	37
Crushed HL feed Radial Stacker		
Width	mm	1,220
Motor	kW	90
Crushed HL feed Stockplie	mt	1,500,000
Conveyors To Leach Pad		
Number of Grasshopper Conveyors		24
Type		Belt
Width	mm	1,067
Quantity		24
Motor	kW	30

Telescoping Radial Stacker		
Type		Belt
Width	mm	1,067
Motor	kW	75
Barren Solution Pumps		
Number of Units Installed		2
Number of Units Operating		2
Type		Horizontal
Motor	kW	100
Capacity per Pump	m3/hr	567
Operating Capacity	m3/hr	1,134
Leach Pad		
Ultimate Design	mt	60,000,000
Ultimate Height	m	125
Lift Height	m	10
Leach Cycle	d	60
Solution Application Rate	l/hr/m2	10
Solution Flow Rate	l/s	315
Area Under Leach	m2	113,400
Pregant Solution Pumps		
Number of Units Installed		3
Number of Units Operating		2
Type		Vertical Turbine
Motor	kW	100
Capacity per Pump	m3/hr	567
Operating Capacity	m3/hr	1,134
Carbon Columns & Handling		
Quantity per Train	5	5 Cascade Type
Number of Trains	2	2
Power	kW	20
Capacity per Column Train	m3/hr	567
Total Capacity	m3/hr	1,134
Acid Wash System		
Power	kW	5
Carbon Capacity	mt	3
Carbon Strip Vessel		
Power	kW	4
Carbon Capacity	mt	3
Heat Exchanger		
Power	kW	20
Type		Skid Mounted
Electrowinning Cell		
Power	kW	15
Capacity	m3	3.0
Regeneration Furnace		
Power	kW	15
Carbon Capacity	mt	3
Mercury Retort Oven		
Power	kW	45
Dore Furnace		
Power	kW	45

All equipment sizes and power requirements are approximate

17.3 Process Services

17.3.1 Electrical Power

All power for the project will be supplied by diesel generators. Estimated power required for the process plant is 3.9 megawatts.

17.3.2 Water Supply

Water will be supplied from wells developed near the plant facilities. The preliminary estimate of the initial annual required water supply is 350,000 m^3. The annual supply requirement is anticipated to be reduced yearly as more water for recycle becomes available. This quantity of water will be adequate to supply all process, camp and mine facilities. A chlorination system for all camp and potable needs is included.

17.3.3 Reagents

Sodium cyanide briquettes will be delivered in one tonne super sacks and mixed in the cyanide mix tank and subsequently transferred to the cyanide mix storage tank. This concentrated cyanide solution will be added to the barren solution tank, the carbon columns, and to the gold recovery circuit as required.

Hydrated lime will be delivered to a lime silo which will be fed by a screw feeder onto the heap leach feed conveyor during heap loading operations.

17.3.4 Laboratory

An assay and metallurgical laboratory facility is equipped to perform sample preparation and assays, by atomic absorption, fire assay, and CN soluble analyses. A metallurgical test work area for process optimization is also included.

17.4 Gold Production Estimates

17.4.1 Ultimate Gold Recovery

The estimates for ultimate gold recovery for various mineralized types were based on the results of laboratory column leach tests conducted by KCA. Table 17-2 presents the results of these column leach tests.

Table 17-2: Column Leach Test Work Used for Recovery Projections

Sample	Crush Size P80 (mm)	Au Extracted (%)	Reagent Consumptions	
			NaCN (kg/t)	$CaOH_2$ (kg/t)
Supremo Oxide Core Composite	25	92	0.17	1.51
Supremo Oxide Core Composite	12.5	94	0.28	1.50
Supremo Oxide Core Composite	12.5	95	0.52	1.57
Supremo Oxide Surface Sample	25	92	0.93	1.51
AVERAGE		**93.3**	**0.48**	**1.52**
Latte Oxide Core Composite	25	90	0.19	1.51
Latte Oxide Core Composite	12.5	90	0.27	1.51
Latte Oxide Surface Sample	25	92	1.08	1.00
AVERAGE		**91.3**	**0.51**	**1.34**
Supremo Upper Transition Core	12.5	**73**	**0.31**	**1.00**
Latte Upper Transition Core	12.5	**47**	**0.46**	**2.01**

For reference purposes the distribution of mined material the final mine plan is presented in Table 17-3.

Table 17-3: Distribution of Material Types in Final Mine Plan

Material Type	Tonnage Mined	Distribution
Supremo Oxide	39,100,000	73
Supremo Upper Transition	1,200,000	2
Latte Oxide	10,500,000	20
Latte Upper Transition	700,000	1
Double Double Oxide	1,200,000	2
Kona Oxide	700,000	1
Total	**53,400,000**	**100.0**

Projected ultimate gold recoveries were all reduced by 3% from the laboratory column test data to account for operational scale up, as recommended by KCA historical experience. Cyanide consumptions were all set at 0.20 kg/t which is generally reducing laboratory data by a factor of 0.33 (or slightly higher), again as recommended by KCA historical experience. The 1.50 kg/t lime consumption for all areas used the laboratory column data. Recoveries of Supremo were used to project recoveries of both the Double Double and Kona areas.

A summary of the ultimate recoveries and reagent consumptions used for each area is given below in Table 17-4.

Table 17-4: Ultimate Gold Recovery and Reagent Consumption Used in the Production Model for Each Area

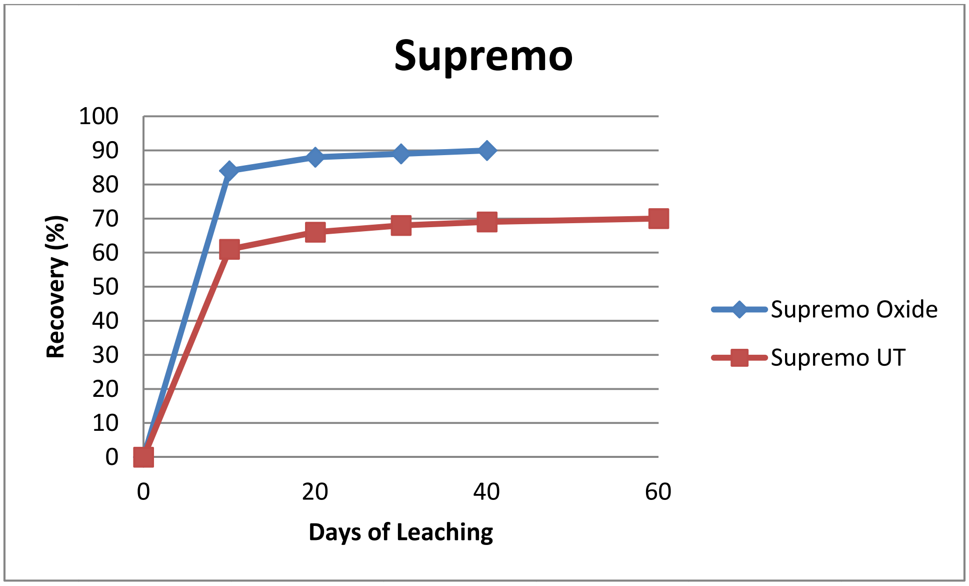
Sample	Ultimate Au Recovery (%)	Reagent Consumptions	
		NaCN (kg/t)	CaO (kg/t)
Supremo Oxide	90	0.20	1.50
Supremo Upper Transition	70	0.20	1.50
Latte Oxide	88	0.20	1.50
Latte Upper Transition	44	0.20	1.50
Double Double Oxide	90	0.20	1.50
Kona Oxide	90	0.20	1.50

Kaminak requested that Mr. Dan Kappes, President of KCA, review the available test work and provide a letter to present his estimate of both recoveries and reagent consumptions suitable for use in the PEA. His letter generally confirmed the values of the both recovery and reagent consumption for the Supremo and Latte material types as KCA has not tested the Double Double or Kona rock types. A copy of his letter is found in Appendix C.

17.4.2 Leaching Time and Recovery Profiles

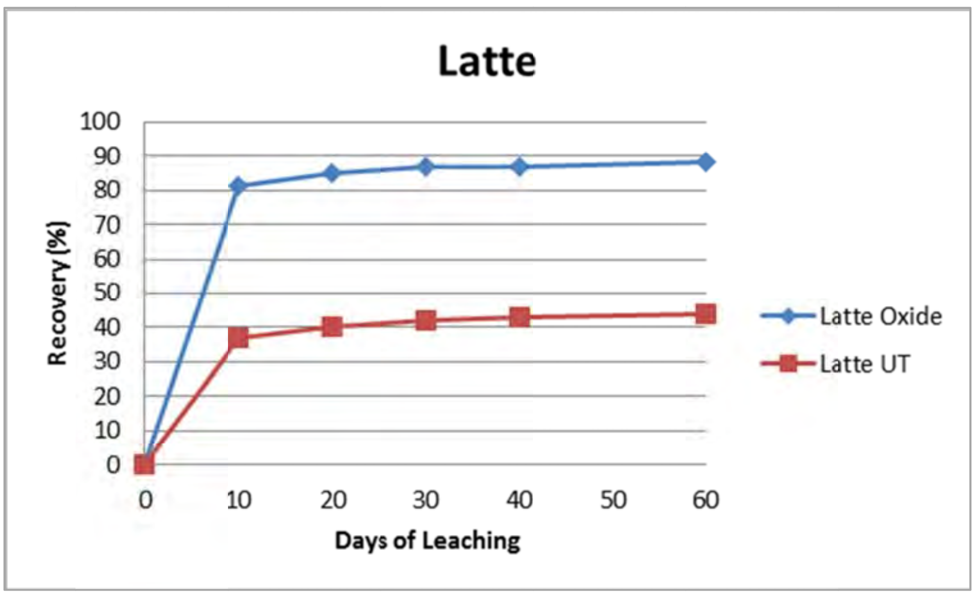
Actual laboratory data of gold recovery and leach time was used to generate the leach profiles. The leach time actually experienced in the laboratory was doubled to generate the leach profiles for both the Supremo and Latte mine areas presented below in Figures 17-1 and 17-2.

Figure 17-2: Supremo Leaching Profiles



As an example in Figure 17-1 above a leach time of 40 days indicates an ultimate 90% recovery for Supremo Oxide. The actual laboratory leach time to achieve this 90 % recovery was only 20 days. A safety factor of 100% has been added to the laboratory leach time.

Figure 17-3: Latte Leaching Profiles



17.4.3 Process Circuit Inventory

Even with all the different mined areas, all reached ultimate extraction in 60 days or less. This rapid leach cycle will help to reduce the inventory of ultimate recoverable gold contained in the heap leach pad during normal operations, but an inventory of recoverable gold will be present. The inventory of recoverable gold in the process circuit will mainly be attributed to the unleached mineralized materials placed on the leach pad as cover for the drip emitter system, estimated be about 550,000 t of HL feed and contain 22,000 oz of gold.

Although the leach cycle should minimize the amount of gold from partially leached material an allowance of 5,000 oz has been made.

Gold inventories contained in process solutions and on carbon are estimated to contain 3,000 oz.

A process inventory of 30,000 oz of gold is expected.

17.4.4 Gold Production Model

Based on the final mine plan a gold production model was developed. The gold production model considers the different mining areas and their ultimate gold extractions for each area. Since the heap leach facility will not be loaded for 100 days per year, a stockpile of crushed HL feed will be required. Usually the stockpile will be full at the start of leaching in the spring. Since the heap loading equipment has a higher capacity than the crushing circuit the stockpile will be depleted by the time heap loading operations are curtailed in the latter part of the year. The stockpile will then be filled over the winter months. A mixture of HL feeds at varying recoveries and grades will be contained in the stockpile. The stockpile inventory is calculated by keeping a running total of tonnage and recoverable ounces of gold. As the stockpile is reclaimed these tonnes and recoverable ounces are then loaded onto the HLF.

Table 17-5 presents the annual gold production. In Year -1, the calculated gold production of 63,000 oz will be reduced by 10,000 oz of process inventory. Of the produced 53,000 oz, only 50% or 26,000 oz are credited as sold. The remaining 27,000 ounces are carried over to Year 1 production to allow for plant start-up.

Table 17-5: Annual Gold Production

	Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	TOTAL
Total Mined and Crushed To Crushed HL feed Stockpile	**Oxide HL feed**	852,000	2,383,000	4,741,000	4,538,000	4,958,000	4,981,000	5,001,000	4,969,000	4,660,000	4,878,000	4,906,000	4,668,000	-	**51,534,000**
	Oxide Grade	1.09	1.13	1.43	1.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.34	-	**1.21**
	Oxide Oz	30,000	86,000	218,000	247,000	130,000	211,000	189,000	173,000	149,000	166,000	205,000	202,000	-	**2,006,000**
	Oxide Recovered Oz	26,000	76,000	192,000	220,000	117,000	190,000	170,000	155,000	134,000	150,000	185,000	181,000	-	**1,797,000**
	Transition HL feed	-	17,000	259,000	450,000	42,000	19,000	-	32,000	332,000	96,000	95,000	574,000	-	**1,914,000**
	Transition Grade	-	1.24	1.72	1.92	1.01	1.25	-	1.34	1.04	2.61	3.44	1.60	-	**1.71**
	Transition Oz	-	664	14,000	28,000	1,371	774	-	1,000	11,000	8,000	10,000	30,000	-	**105,000**
	Transition Recovered Oz	-	292	6,300	12,000	960	542	-	951	8,000	6,000	7,000	21,000	-	**63,00**
	Total Tonnes	852,000	2,400,000	4,999,000	4,987,000	5,587,000	5,587,000	5,001,000	5,001,000	4,992,000	4,973,000	5,000,000	5,242,000	-	**53,448,000**
	Total Grade	1.09	1.13	1.44	1.71	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.37	-	**1.23**
	Total Recovered Oz	26,000	76,000	198,000	232,000	118,000	191,000	170,000	156,000	142,000	155,000	192,000	202,000	-	**1,859,000**
Beginning Stockpile Inventory	**Total Tonnes**	852,000	3,252,000	6,251,000	5,587,000	5,587,000	5,600,000	5,588,000	5,589,000	5,581,000	5,554,000	5,554,000	5,797,000	797,000	**-**
	Total Recovered Oz	26,000	103,000	237,000	255,000	145,000	206,000	192,000	177,000	161,000	172,000	209,000	223,000	31,000	**-**
From Crusher Stockpile to Heap Leach	**Total Tonnes**	-	2,000,000	5,651,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	797,000	**53,448,000**
	Total Recovered Oz	-	63,000	215,000	228,000	130,000	184,000	172,000	158,000	144,000	155,000	188,000	192,000	31,000	**1,859,000**
Closing Stockpile Inventory	**Total Tonnes**	852,000	1,252,000	600,000	587,000	587,000	587,000	588,000	589,000	581,000	554,000	554,000	797,000	-	**7,276,000**
	Total Recoverable Oz	26,000	39,000	23,000	27,000	15,000	22,000	20,000	19,000	17,000	17,000	21,000	31,000	-	**250,000**
Gold Production	**Gold in Heap Inventory**		10,000	10,000	10,000	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	(30,000)	
	Recovered Oz	-	53,000	205,000	218,000	130,000	184,000	172,000	158,000	144,000	155,000	188,000	192,000	61,000	**1,859,000**
Plant Start Up Allowances	**Ounces**		27,000	(27,000)											
Gold Sales	**Ounces**		**27,000**	**231,000**	**218,000**	**130,000**	**184,000**	**172,000**	**158,000**	**143,853**	**155,000**	**188,000**	**192,000**	**61,000**	**1,859,000**

18.0 PROJECT INFRASTRUCTURE

18.1 Waste Management

The waste rock generated from the open pits not used for constructing the initial heap leach embankment will be disposed of in various waste dumps located adjacent to the various deposits at the Coffee project. Table 18-1 summarizes the waste rock allocation by pit phase and waste dump. If required, prior to waste rock disposal topsoil and growth media will be removed from the footprint area and stockpiled.

Kaminak is currently undertaking a comprehensive acid-base accounting program, which is being executed by Lorax Environmental of Vancouver, B.C. Preliminary results indicate that the rocks are non-acid-generating (NAG) due to the low abundance of sulphide within both mineralized and unmineralized rock. In addition, there is a high abundance of carbonate minerals in the rock, which further reduces the potential for acid generation.

The initial waste rock generated in Year -2 will be used to construct a containment embankment for the heap leach facility. The majority of this rock will be spread in lifts and compacted to provide the structural strength of the containment embankment required.

Surface diversion ditches will be provided, as necessary, around the outside of deposited waste rock dumps to minimize any run-off.

Table 18-1: Waste Rock Allocation

Pit/Phase	Units	Total	Waste Dump			
			South	West	North	Kona
SU1(E)	Mt	**33**	18	-	15	-
SU2(M)	Mt	**40**	-	40	-	-
SU3(S)	Mt	**49**	39	10	-	-
SU4(N)	Mt	**49**	-	27	22	-
LT1(S)	Mt	**22**	-	22	-	-
LT2(N)	Mt	**2**	-	2	-	-
DBLDBL	Mt	**14**	14	-	-	
KONA	Mt	**4**	-	-	-	4
Total	**Mt**	**212**	**71**	**101**	**37**	**4**

18.2 Infrastructure

The project envisions the construction of the followings key infrastructure items:

- Approximately 250 km all-season access road from Carmacks to the project site
- Approximately 7 km of new on-site access roads for light vehicles to by-pass the active mining areas
- New airstrip
- Primary, secondary and tertiary crushing systems
- 1.5 M t crushed HL feed stockpile
- Carbon adsorption plant and gold refinery
- Truck shop, warehouse and camp
- Fresh water supply developed from groundwater
- Bulk explosives storage and magazines
- Power plant and bulk fuel storage
- Potable, fire and sewage water systems.

18.3 General Site Arrangement

The general site arrangement is shown in Figure 18-1.

The site has been configured to place the heap and other site infrastructure away from mineralized and potentially mineralized zones. The primary crusher is located approximately 2.3 km from the currently planned mining area. The largest structure on site, other than the valley-fill heap leach facility and waste rock dumps will be the 1.5 M t crushed HL feed stockpile. The stockpile, crushing and gold recovery systems have been located near the HLF to minimize pumping and pipeline requirements for the pregnant and barren solutions and to reduce conveying distances.

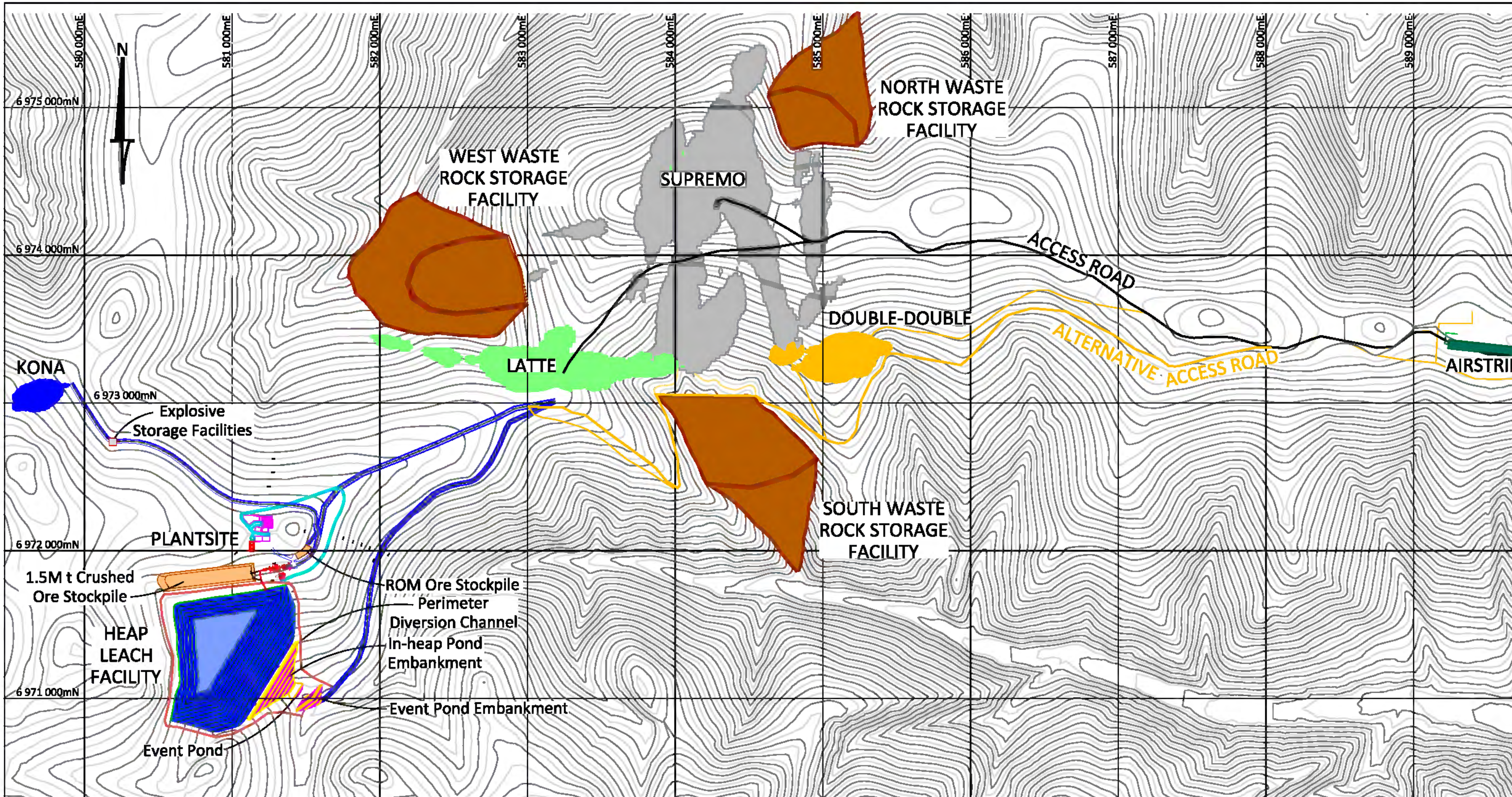
18.4 Site Access Road

An all-season access road has been proposed to be constructed to connect the project site to Carmacks, Yukon Territory, Canada. The proposed route is highlighted in Figure 18-2.

The conceptual design for the road includes upgrading approximately 74 km of existing public road and 179 km of new road construction (250 km total). The road will be designed as a single-lane (5 m width), mine access road with intermittent turnouts and traffic will be radio controlled.

The road will be suitable for semi-trailer trucks and will eliminate the need to supply the site by barge, ice-road or by air.

The mine will own and operate a fleet of snow plows and graders used to maintain the road through the winter.



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ISSUE												

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REVISIONS												

SECTION:	
SCALE:	
DESIGN. BY J.D.S.	May 2 2014
DRAWN BY: B. Wong	May 2 2014
CHECK. BY	
APP. BY:	

JDS Energy & Mining Inc.

Kaminak Gold Corp.

COFFEE GOLD PROJECT
PROJECT OVERALL SITE PLAN

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
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PROJECT	PROCESS	CIVIL	MECH.	STRUCT.	PIPING	SERVICES	ELECT.	INSTR.	NO	DESCRIPTION	BY	DATE
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SECTION:	
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CHECK. BY	
APP. BY:	

JDS Energy & Mining Inc.

Kaminak Gold Corp.

COFFEE GOLD PROJECT
PROJECT LOCATION MAP
PLAN

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
		100-10-001	

18.5 Light Vehicle Roads

Approximately 7 km of new road for light vehicles will be constructed at the project site. New roads will by-pass active mining areas and will be used to move personnel and supplies around the site. Double-lane light vehicle roads will be 12 m wide, including berms and a ditch, and will have a maximum grade of 10 %. Light vehicles are kept on separate roads away from mine mobile equipment where possible. This is done in order to improve safety for personnel in light vehicles and to increase the efficiency of the mine operation.

18.6 Surface Haul Roads

Surface haul roads will connect the mining areas to the primary crusher, truck shop and HLF. Roads will be approximately 25 m wide to accommodate two-way haul truck traffic. The maximum grade is 10 %. Roads will include a berm of height equal to 75 % of the diameter of the largest tire hauling on the road. The berm will have cut-outs for snow removal during the winter.

18.7 Airstrip

A new gravel airstrip will be constructed on the ridgeline, approx. 4 km to the east of the project site and will be designed to accommodate Dash-8 (or equivalent) aircraft. The runway will be approximately 1,250 m long and 30 m wide. The airstrip will include an apron sized to accommodate two Dash-8 sized aircraft and a garage to house a forklift and de-icing equipment.

Personnel will be transported between the camp and airstrip using the site crew vans.

The airstrip will be supplied with power from a generator located at the site. The generator will have an independent fuel tank and fuel will be delivered by the fuel and lube truck. An emergency generator will be on standby to supply power in the event of a failure.

Emergency jet fuel will be stored at the airstrip.

Aircraft de-icing will be performed on the apron. The apron will be design to collect and contain de-icing chemicals. Used chemicals will be recycled where possible and otherwise shipped off-site for disposal.

The airstrip will include all necessary lighting and navigational equipment required for aircraft to take-off and land safely.

18.8 Buildings and Structures

18.8.1 Primary Crusher and Pad

Run-of-mine mineralized material will be hauled by truck to the primary crusher pad. When possible, material will be dumped directly into the crusher, otherwise it will be stored in the ROM stockpile.

The primary crusher, grizzly, feeder, bins and conveyors will be placed next to a mechanically stabilized earth (MSE) wall. Equipment will dump from the top of the MSE wall (the crusher pad) into the primary crushing system. The other three sides of the primary crushing system will be shielded from the weather by a steel, pre-engineered building and roof.

18.8.2 External Conveyors

Plant site conveyors that pass outside buildings will be protected from rain and snow by plywood or steel half-culvert “dog house” covers.

18.8.3 ROM Stockpile

The ROM stockpile is located adjacent to the primary crusher pad. The stockpile has sufficient footprint for a maximum of 100,000 t of ROM material (at 10 m stockpile height). Trucks that are not routed directly to the primary crusher will travel up a short single-lane ramp to dump to the stockpile. If the supply of mineralized material from the pit is disrupted, a wheel loader will be used to continue to feed the primary crusher from the ROM stockpile.

18.8.4 Secondary and Tertiary Crushers

The secondary and tertiary crusher building will be a 20 x 30 m pre-engineered steel structure that will house the secondary screens, secondary crusher and tertiary crushers. The tertiary screens will be housed in a separate 7 x 13 m pre-engineered building.

The design criteria assume the HLF will be loaded for 250 days per year to avoid loading during the coldest period of the year. Product from the tertiary screens will be conveyed directly to the HLF or to the 1.5 Mt crushed HL feed stockpile depending on temperature and weather conditions.

18.8.5 1.5 Million Tonne Crushed HL Feed Stockpile

The 1.5 Mt crushed HL feed stockpile will be approximately 120 m wide x 625 m long x 10 m tall.

The stockpile will be constructed of tertiary crushed material using the grasshopper conveyors and mobile stacker during the winter months when placement on the heap has been suspended.

The stockpile will be reclaimed back onto the grasshopper conveying system using a wheel loader at a nominal rate of 5,000 tpd during the 250 day heap construction season.

18.8.6 Carbon Adsorption Plant and Gold Refinery

The layout for the carbon adsorption plant and gold refinery has been provided by Kappes Cassiday. The majority of the process equipment will be housed within a 42m x 26m pre-engineered steel building. The carbon-in-column (CIC) tanks will be insulated and placed outside the primary carbon-adsorption building.

18.8.7 Reagent Storage

Bulk reagents will be stored in a reagents warehouse near the carbon adsorption plant / gold refinery building.

18.8.8 Metallurgical and Assay Lab

The metallurgical and assay lab will be located in a pre-engineered building near the carbon adsorption plant / gold refinery building. Refer to Section 17.3 for further details.

18.8.9 Truck Shop, Warehouse and Administration

The truck shop, warehouse and administration offices will be housed in a single modular “tent” structure, 100m long x 30m wide x 12m tall. A single building has been selected in order to minimize the capital cost of the building. The skin on the structure has an expected life of 12 years and can be re-skinned at relatively low cost.

The truck shop will have four bays for maintenance and repair of large mobile equipment including a wash bay. The truck shop includes two bays for smaller equipment (loaders, graders, pickups, etc.). A mobile rough terrain crane will be used to support maintenance activities.

Offices will be located within the combined truck shop / warehouse building. If additional office space is necessary, a second story will be added above the mine dry complex.

Cold storage is located near the warehouse.

18.8.10 Camp

The camp will have 220 single occupancy rooms in “Jack & Jill” bathroom arrangement (one bathroom shared between two rooms). It is expected that approximately 180 personnel will be on-site at any one time. There is sufficient spare capacity to clean and maintain rooms between on-coming crews. The kitchen, cafeteria, dry and catering will be sized to accommodate approximately 90 personnel at a time.

The camp includes a recreation area, mine dry, wash and laundry facilities and potable and sewage water treatment systems.

Domestic garbage will be incinerated on-site.

18.8.11 Communications / IT

The camp and offices will include a wired and wireless computer network, satellite phone and satellite TV.

A hand-held radio system will be used for voice-communication between personnel in the field.

18.8.12 First Aid / Emergency Services

Catering staff will be trained and act as first-aid attendants. The first aid room will be located in the warehouse. The ambulance and fire truck will be parked at the ready in the truck shop.

Buildings will be equipped with smoke, carbon monoxide and heat detectors, overhead sprinklers, hydrants / hoses and appropriate chemical fire extinguishers.

18.8.13 Explosives Storage and Magazines

Explosives will be stored at a secured and monitored site located approximately 1 km from populated, high traffic areas. The final location of the explosives storage site will be determined as part of future pre-feasibility or feasibility studies. Boosters and detonators will be stored in barricaded magazines and separated according to Natural Resource Canada guidelines.

18.8.14 Bulk Explosives Storage

Ammonium nitrate (AN) prill will be stored in a 60 tonne steel silo. AN will be loaded into bulk delivery trucks and only combined with fuel oil (FO) to make ANFO explosive at the blast hole.

18.8.15 Powder Magazine

High explosive boosters and packaged explosives will be stored in a 24 tonne powder magazine.

18.8.16 Detonator Magazine

Detonators (caps) will be stored in a 3.6 tonne cap magazine.

18.9 Power Supply

Total installed power, including process equipment and other site infrastructure is estimated to be 5.0 MW. The average operating power demand is 3.6 MW. Electricity will be supplied by five on-site diesel generators required for an N+2 configuration.

Three generators (N = 3) will be required to meet the average demand with one on standby and one for scheduled maintenance (N + 2). Each generator will be rated for 1,450 ekW / 4,160 V, three phase, 60 Hz output.

Each diesel engine and generator will be mounted on a modular, insulated and sound attenuated enclosure including the radiator and external 8-hour fuel tank. The external fuel tanks will be tied to the bulk fuel storage system. Each enclosure will have an independent fire suppression system.

If necessary, a heat recovery system can be added to each unit. This will allow heat to be recovered from the engine exhaust and used to heat buildings or process solutions.

18.10 Power Distribution

Overhead power lines will connect the diesel power station to fixed structures (buildings, process equipment, conveyors etc.) at the project site.

18.11 Bulk Fuel Storage

Fuel consumption for the diesel power plant and mobile equipment is estimated as follows:

Diesel power plant	125,000 l / week,
Mine mobile equipment	250,000 l / week,
Total	375,000 l / week.

Bulk fuel tanks have been sized to store two weeks of diesel on-site, approximately 750,000 l.

The fuel storage area will have two 430,000 l tanks for a total diesel storage capacity of 860,000 l and a single 49,000 l tank for gasoline. Tanks will be double-walled and located within a lined compound that meets Yukon and Environment Canada regulations for containment in the event of a spill.

18.12 Fuel Distribution

Diesel will be delivered to mine mobile equipment and the airstrip by the fuel & lube truck.

Light vehicles will fuel up at the fuel storage area. The bulk diesel and gasoline tanks will include integrated dispensing systems.

18.13 Surface Water Management

A network of ditches will be excavated above the plant site, open pits and waste rock dumps to capture and divert clean surface water before it enters the mine system.

18.14 Water Supply, Raw and Process Water

It is assumed that a supply of groundwater is available near the plant site for use in the process and for other purposes around the site. Ground water will be pumped to two storage tanks located on the ridge above the plant site (raw & process water).

All water tanks will be insulated to prevent freezing. Water lines (raw, process, fire water, sewage, etc.) will be buried underground or heat traced where they appear at surface.

18.15 Fire Water

Fire water will be stored in a dedicated 430,000 l tank on the ridge above the plant site. This capacity will allow for approximately two hours of fire-fighting. If additional water is necessary for extended fire-fighting, remaining water in the raw and process water tanks can be used.

All buildings will be connected to the fire water system.

Two 250,000 l/hr pumps (one electric, one diesel backup) will provide pressure and volume for fire-fighting in any one of the main buildings.

18.16 Potable Water

A potable water treatment system is included in the camp facility.

18.17 Sewage Treatment

A sewage water treatment system is included in the camp facility. The sewage treatment system will be sized to handle approximately 200 personnel on-site.

Contaminated water from the heavy equipment wash bay will pass through an oil water separator. Oily sludge will be stored in a transfer tank and back-hauled off-site for disposal. The remaining water will be stored in a pond and recycled to the wash bay.

18.18 Freight

Freight will be delivered to site on the all-season access road and offloaded at the warehouse or cold storage area.

18.19 Waste Management

All methods of reducing waste and recycling on site will be examined and implemented to minimize the amount of industrial waste generated by the site.

Solid waste (scrap steel, wood, etc.) will be collected in bins; empty chemical totes, lubricant drums, etc. will be compacted and stored in the cold storage area.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

At this time, no market studies have been completed as the gold to be produced at Kaminak can be readily sold in the open market. Gold refining charges were estimated to be US$7.50/payable ounce.

19.2 Contracts

No contractual arrangements for concentrate trucking, port usage, shipping, smelting or refining exist at this time. Furthermore, no contractual arrangements have been made for the sale of gold doré at this time.

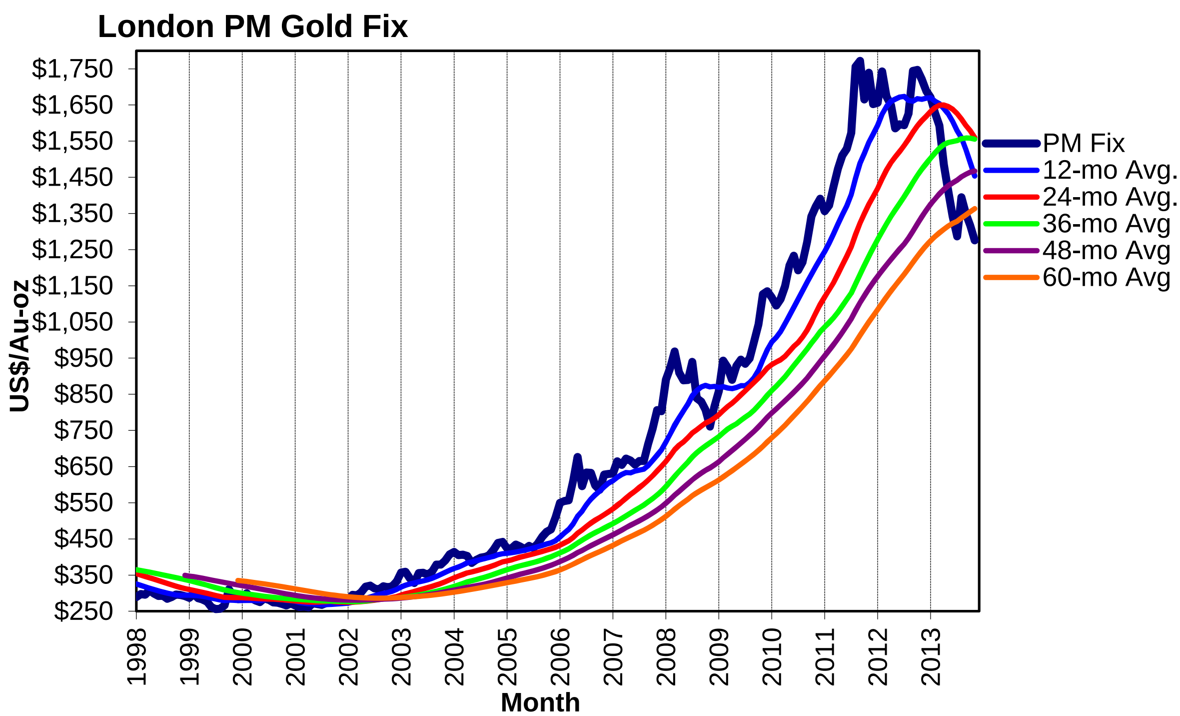
19.3 Royalties

There is a 2% net smelter returns royalty (NSR) on the property, payable to prospector Mr. Shawn Ryan of Dawson City, YT. The NSR is subject at any time to a 1% buy-back for $2 million, with annual advance royalty payments of $20,000 commencing December 31, 2013.

19.4 Metal Prices

The base and precious metal markets benefit from terminal markets around the world (London, New York, Tokyo, Hong Kong) and fluctuate on an almost continuous basis. Historical metal price for gold is shown in Figure 19-1 and demonstrate the change in metal price from 1998 through to November 2013.

Figure 19-1: Average Gold Cash Price as at November 30, 2013



Two metal price scenarios were evaluated as part of the economic analysis. In addition to the Base Case metal pricing scenario, a scenario was evaluated utilizing three-year trailing average metal prices. Base Case pricing for gold is based on approximately 20% less than the three-year trailing average. The Base Case pricing is also approximate to current gold spot prices.

Table 19-1 summarizes the metal prices and exchange rates used to run various scenarios in the economic analysis.

Table 19-1: Metal Price and Foreign Exchange Rate Used in Economic Analysis Scenarios

Parameter	Units	Three-Year Trailing Average (as of Nov 30, 2013)	Base Case
Gold Price	US$/oz	1,555	1,250
Exchange Rate	US$:C$	0.95	0.95

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Environmental management issues associated with the Coffee project are primarily, but not limited to, aquatic resources and proximity to sensitive wildlife areas. Baseline environmental studies were initiated in 2010 and have been expanded upon over the years.

20.1 Existing Permits

Kaminak does not hold any of the permits required to operate a mine. Kaminak carries out its site and exploration activities under the following permits/licences:

- Class 4 Quartz Mining License, LQ00312
- Type B Water Licence, MN12-014
- Storage Tank System Permit, 02012-26
- Waste Management Permit, 81-027

20.2 Baseline Environmental Studies

Baseline environmental studies were first undertaken in 2010. In 2014, a comprehensive baseline program has been initiated.

Work undertaken to date includes:

- Weather
- Hydrology, including initial hydrogeology
- Initial ML/ARD
- Monthly water quality
- Initial Fishery studies
- Wildlife – aerial ungulate surveys and wildlife observations
- Traditional Land Use and Archaeology

20.2.1 Weather

There is a weather station on site, located along the road to the main deposit. The data logger has been in place since 2012.

20.2.2 Hydrology

The main objective of the hydrologic monitoring program has been to characterize the timing and magnitude of stream flow at various locations within the project area. This data will be used to help make management, design and development decisions in the future. Two hydrology stations were added in 2013, bringing the total number of stations to four. Additional stations and further hydrologic monitoring will be required moving forward toward the development of a project proposal.

20.2.3 Aquatic Resources and Fishery Studies

In order to monitor potential changes related to the development of this project, water, sediment quality and aquatic biology baseline studies are being conducted in 2014. Monthly Water Quality sampling has been undertaken since 2010 at a number of locations in the project area. A total of 42 monthly datasets were obtained October 2010 to May 2014 from locations on and tributaries to Coffee, Halfway and Independence Creeks, and from the Yukon River both upstream and downstream of the Project. Visual fish surveys have also been conducted.

These studies will continue and are being expanded upon in 2014 toward the development of a project proposal.

20.2.4 Wildlife Monitoring

Wildlife baseline information has been collected since 2010. This information has been collected to document the wildlife species sighted in the area. The collection of baseline data also contributes to the identification of any species at risk. The evaluation of the information collected will contribute to the comprehensive wildlife baseline study being conducted in 2014.

The wildlife monitoring program will need to be expanded in 2014 to include breeding bird surveys, raptor surveys, carnivore/den surveys, vegetation and habitat mapping, and continued effort on aerial ungulate surveys.

20.3 Environmental Management

Kaminak has developed a number of management plans as part of an Environmental Action Plan, to assist with management and operational decisions during the exploration phase. These plans will need to be expanded upon as the project moves toward development. These management plans include:

- Spill Response
- Emergency Response
- Reclamation
- Wildlife
- Fuel Storage
- Water Quality
- Hazardous Materials and Waste Management
- Environmental Awareness and Education
- Heritage and Archaeological Sites

20.4 Site Reclamation and Closure

A site reclamation plan will be required as part of the design and project proposal submission. In this area, the expectation would be that all facilities would be removed from the site and that surface disturbance would be modified to minimize the impact upon wildlife and other land users. As part of the project design, the area of disturbance will be minimized and, as much as possible, there will be progressive reclamation work concurrent with operations.

Facilities that cannot be used in the future will be demolished and removed from site for salvage to the extent possible. With an open pit excavation available, the option to dispose of non-toxic materials in the pit is being investigated. Site monitoring of conditions and water quality would continue for a period after the cessation of operations and removal of facilities.

Financial assurance must be posted to secure the rehabilitation works, and the determination of the outstanding mine reclamation and closure liability, associated with the project technical features and structures, must be sealed by a professional engineer who is licensed to practice in the Yukon (Yukon Energy, Mines, and Resources 2006).

The Government of Yukon determines the amount and form of security to be provided by the proponent. The government will also ensure that security is maintained at all times. Financial security will comprise an initial payment, prior to commencement of development, and a periodic adjustment to ensure that full security is held for outstanding reclamation and closure liability throughout the development, operation, and closure of a mine site. Progressive reclamation may reduce the amount of financial security required to be provided and maintained by the proponent.

The proponent must file an annual report stating what progressive reclamation has been accomplished and the results of environmental monitoring programs. The proponent will monitor to determine the effectiveness of the mitigation measures as progressive reclamation and closure work is completed. (Yukon Energy, Mines, and Resources 2006).

The current environmental liabilities associated with the project are:

- Removal of material that cannot be disposed of on site
- Potential hydrocarbon contaminated soils
- Empty fuel drums and propane cylinders
- Assorted diamond drill set up materials and refuse
- Reclamation of the airstrip
- Removal and disposal of the drill core
- Removal and disposal of facilities that will not be sold with the camp

In 2013, Kaminak initiated a re-vegetation study on drill fences. The excavator was used to pull organic material that had been set-aside back over the drill fence. The organic material was worked by hand to promote regrowth. Plots were established to trial a number of re-vegetation methods, including: seed dispersal, transplanting locally growing vegetation, salvaging plants that had been removed during the construction of the drill fence, and various combinations of these methods. This work will continue in 2014 and test plots will be monitored.

20.5 Environmental Assessment and Permitting

Before projects proceed to the licensing phase, they are first assessed through an environmental assessment (EA). The Yukon Environmental and Socio-economic Assessment Board (YESAB) administer EAs in Yukon. The Coffee project will be subject to an EA under the *Yukon Environmental and Socio-economic Assessment Act* (YESAA).

20.5.1 Environmental Assessment

The Project requires an Executive Committee screening because it is a quartz exploration program that involves the movement of 250,000 t or more rock. Projects assessed by the Executive Committee of YESAB generally require between one and three years (not more than 918 days, including time required for a government decision).

Detailed information requirements for this process are outlined in the Information Requirements for Executive Committee Project Proposal Submissions under the YESAA, which is available through the YESAB office.

Once assessments are complete, recommendations are forwarded to a decision body or bodies. The recommendations will be one of the following (YESAB 2011):

- The Project will not have significant adverse effects and should proceed.
- The Project will have significant adverse effects that cannot be mitigated and should not proceed.
- The Project should proceed with terms and conditions that will mitigate the effects.
- The Project should be assessed at a higher level. (Note: This can only occur when the assessment was done at the Designated Office (DO) or Executive Committee level.)

In some cases, assessments may also recommend project audits or effects monitoring.

20.5.2 Licensing

The Project will be subject to territorial legislation, and will require a number of permits and approvals. The Project may also be subject to federal legislation, depending upon specific project features and details.

Quartz Mine License

All hard rock mining claims are administered through the Quartz Mining Act (QMA) in Yukon. The QMA enables the Government of Yukon to issue licenses and regulate mining developments. The Government of Yukon Department of Energy, Mines and Resources administer the Quartz Mine License (QML) following the EA. Although permits and licenses cannot be issued in advance of completing the assessment, regulatory processes can be initiated simultaneously while the assessment is underway (Yukon Energy, Mines and Resources 2010).

Water License

The Yukon Water Board is responsible for licensing the use of water and the discharge of wastes into waters within the Yukon Territory under the *Yukon Waters Act* (Yukon Water Board 2006). The Project will require a Type A water license.

Storage Tank Systems Permit

All fuel storage is regulated under the Storage Tank Regulation of the *Yukon Environment Act*. All storage tanks require a Storage Tank Permit and must be installed according to territorial and federal standards.

20.6 Socioeconomic Considerations

20.6.1 First Nations and Project Location

The Project is located on Crown Land within the traditional territory of the Tr’ondëk Hwëch’in First Nation. A portion of the claim block is located within the overlap area with Selkirk First Nation; however, Kaminak has conducted limited exploration within the overlap area. The proposed road alignment is located within the traditional territories of Little Salmon Carmacks First, Selkirk First Nation and Tr’ondëk Hwëch’in First Nation.

During August 2010, in collaboration with the Tr’ondëk Hwëch’in, Kaminak retained Matrix Research Limited (Matrix Research) of Whitehorse, Yukon, to conduct a heritage resources overview assessment and preliminary field reconnaissance over the Coffee property. One historical site and three pre-contact First Nations sites were identified within the property. Buffers were set up around the sites. However, none occur in proximity to established zones of gold mineralization and there was no impact on exploration programs during the remainder of the 2010 season or preceding years.

In 2011, as follow-up to the overview assessment, a more detailed heritage assessment was undertaken including archaeological fieldwork. The survey confirmed that no heritage sites were located within or overlapping with zones of established gold mineralization or planned exploration, nor areas of ancillary infrastructure including the Coffee camp area and the proposed access road.

In 2012, an Oral History project of the Coffee project area was undertaken by the Tr’ondëk Hwëch’in Heritage Department and supported by Kaminak. The project included a site visit in July 2012 by the project participants including Tr’ondëk Hwëch’in First Nation Heritage Officers and the lead researcher/author of the Oral History project.

On May 16th, 2013, Kaminak and Tr'ondëk Hwëch'in signed an Exploration Communication and Cooperation Agreement.

In 2013, White River First Nation informed Kaminak that the Project was located within their asserted traditional territory. On April 8th, 2013, Kaminak received a map from White River First Nation showing the extent of their asserted traditional territory. The Coffee project is located within WRFNs asserted traditional territory.

20.6.2 Communities

The primary communities affected by the Coffee project and related infrastructure are Dawson City, Pelly Crossing, Beaver Creek and Carmacks. The project is located in west-central Yukon, within the Whitehorse Mining District 130 km south of Dawson City.

20.6.3 Studies and Consultation

Kaminak initiated and supported a collaborative study in 2013 with Tr'ondëk Hwëch'in, Selkirk First Nation and White River First Nation to compile a list of the studies that had been previously conducted for the Coffee area and identify any information gaps. The collaborative studies are ongoing.

In 2014 Kaminak will undertake data collection towards a socio-economic assessment.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs Summary

The capital cost estimate (CAPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a preliminary study.

The CAPEX estimate includes the costs required to develop, sustain, and close the operation for the planned 12 year mine life. The construction schedule is based on an approximate 2-year build period. The intended accuracy of this estimate is +/-25%.

The CAPEX estimate summary is shown in Table 1-12.

Table 21-1: LOM Capital Costs

Capital Cost		Pre-Production (C$M)	Sustaining/ Closure (C$M)	Total (C$M)
Capitalized Mining		50.3	0.0	50.3
Pre-Production Operating Costs		16.7	0.0	16.7
Site		57.1	4.8	61.9
Mining Equipment		46.3	64.5	110.8
Leach Facility		56.1	17.2	73.3
Camp		10.3	0.0	10.3
Indirects		37.0	0.0	37.0
Closure		0.0	40.0	40.00
Subtotal		**273.8**	**126.5**	**400.3**
Contingency	15%	31.0	19.0	50.0
Total Capital Costs		**304.8**	**145.5**	**450.3**

21.2 Mine Capital Costs

Mining equipment requirements were generated using the resource model provided, current economic considerations, and a number of assumptions based on industry knowledge. The LOM production schedule was focused on producing higher grade heap leach feed in the early years of the mine life and as such, was used as the basis for the capital cost estimate for the open pit operation (as well as comparing to similar sized open pit gold operations). The open pit mining activities for the Coffee pits were assumed to be undertaken by an owner-operated fleet as the basis for this preliminary economic assessment with the fleet having an estimated maximum capacity of 90,000 tpd total material, which will be sufficient for the proposed LOM plan. Derivation of the LOM plan is discussed further in Section 16 above.

The annual open pit equipment fleet requirements to achieve the target mining and processing rates are summarized in Table 21-1 below (in terms of number of units required per year). Mining cost service information and factors based on experience were taken into consideration in determining the open pit capital cost estimate. Estimated freight costs to site are included. The primary mine equipment and ancillary units will be purchased by the mine, and fall under capital costs.

Mine pre-production costs have been capitalized in years -2 and -1 and total $21.1M and $29.3M respectively.

Table 21-2: Mine Equipment Capital Costs

				YEAR												
Item	**Unit**	**Unit Cost**	**LOM Total units**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**Total**
Primary																
Crawler-Mounted, Rotary Tri-Cone, 250 mm Dia.	C$M	$ 2.6	3	$2.6		$2.6	$2.6									**$7.9**
Crawler-Mounted, Rotary Tri-Cone, 165 mm Dia.	C$M	$ 1.1	3	$1.1		$1.1						$1.1				**$3.2**
Crawler-Mounted, Rotary Tri-Cone, 115 mm Dia.	C$M	$ 0.7	1	$0.7												**$0.7**
Diesel, 16-cu-m Front Shovel	C$M	$ 5.9	2	$5.9			$5.9									**$11.8**
Diesel 12-cu-m Wheel Loader	C$M	$ 3.2	1	$3.2												**$3.2**
136-t class Haul Truck	C$M	$ 2.9	14	$5.9	$2.9	$11.8	$11.8		$8.8							**$41.3**
D10-class	C$M	$ 1.9	8	$3.7	$1.9	$1.9					$3.7	$3.7				**$14.8**
824H-class	C$M	$ 1.3	2	$1.3							$1.3					**$2.5**
16H-class	C$M	$ 1.2	4	$2.3							$1.2	$1.2				**$4.6**
136 t-class	C$M	$ 2.9	2	$2.9							$2.9					**$5.8**
Subtotal Primary				**$29.6**	**$4.8**	**$17.3**	**$20.3**		**$8.8**		**$9.0**	**$5.9**				**$95.8**
Ancillary																
ANFO/Slurry Truck, 12-t	C$M	$ 0.2	2	$0.5												**$0.5**
Stemming truck, 15-t	C$M	$ 0.1	2	$0.2												**$0.2**
Powder Truck, 1-t	C$M	$ 0.1	2	$0.1												**$0.1**
AN Storage Bin, 60-t	C$M	$ 0.1	1	$0.1												**$0.1**
Powder magazine, 24-t	C$M	$ 0.1	1	$0.1												**$0.1**
Cap magazine, 3.6-t	C$M	$ 0.0	1	$0.0												**$0.0**
385C Excavator (backhoe), 3 cu-m	C$M	$ 0.6	1	$0.6												**$0.6**
Haul Truck (road constr), 30-t	C$M	$ 0.6	3	$1.8												**$1.8**
Backhoe/Loader, 1.0 cu-m	C$M	$ 0.2	1	$0.2												**$0.2**
Portable Aggregate Plant,30 tph	C$M	$ 0.4	1	$0.4												**$0.4**
Transporter w/Tractor, 100-t	C$M	$ 0.5	1	$0.5												**$0.5**
Fuel truck, 20000-l	C$M	$ 0.3	1	$0.3												**$0.3**
Lube/Service Truck	C$M	$ 0.3	2	$0.7												**$0.7**
Mechanic Field Service Truck	C$M	$ 0.2	3	$0.6												**$0.6**
Off-Road tire handling Truck	C$M	$ 0.4	1	$0.4												**$0.4**
Int. Tool Carrier, 140-hp	C$M	$ 0.2	1	$0.2												**$0.2**
Light Plant, 6-kW	C$M	$ 0.0	10	$0.2												**$0.2**
Pickup Truck, 0.75-t, 4-WD	C$M	$ 0.1	10	$0.5												**$0.5**
Crew Van, 1-t, 4-WD	C$M	$ 0.1	5	$0.3												**$0.3**
Mobile Radio, installed	C$M	$ 0.0	74	$0.0	$0.0	$0.0	$0.0		$0.0		$0.0	$0.0				**$0.1**
Subtotal Ancillary				**$7.7**	**$0.0**	**$0.0**	**$0.0**		**$0.0**		**$0.0**	**$0.0**				**$7.7**
Miscellaneous																
Shop Equipment	C$M	$ 0.8	1	$0.8												**$0.8**
Eng & Office Equip plus Software	C$M	$ 0.7	1	$0.7												**$0.7**
Radio Communications System + GPS	C$M	$ 0.6	1	$0.6												**$0.6**
Subtotal Miscellaneous	**C$M**			**$2.1**												**$2.1**
Total Equipment & Misc.				**$39.3**	**$4.8**	**$17.3**	**$20.3**		**$8.8**		**$9.0**	**$5.9**				**$105.5**
Spares Inventory @ 5%	C$M			$2.0	$0.2	$0.9	$1.0		$0.4		$0.5	$0.3				**$5.3**
Sustaining	C$M															
TOTAL MINE EQUIPMENT CAPITAL	**C$M**			**$41.2**	**$5.0**	**$18.2**	**$21.3**		**$9.3**		**$9.5**	**$6.2**				**$110.8**

21.3 Processing and Infrastructure Capital Costs

Heap leach processing and infrastructure capital costs are recorded in Table 21-3.

Table 21-3: Heap Leach, Processing and Infrastructure Capital Costs

Item	Unit	Capital
Site	C$M	$61.9
Heap Leach & Processing	C$M	$73.3
Camp	C$M	$10.3
Indirects	C$M	$37.0
Total	**C$M**	**$182.3**

The following items are included in each capital cost category:

Site – General earthworks, site access road, airstrip, power supply and distribution, fuel storage, communications, water supply and distribution, metallurgical lab, truck shop / warehouse, communications, incinerator and site mobile equipment.

Heap Leach & Processing – Conveying system, crushing, carbon adsorption plant, gold refinery and HLF.

Camp – Sleepers, kitchen / dining, recreation, mine dry, laundry and sewage and potable water treatment systems.

Indirects – Engineering procurement, construction management (EPCM), construction indirects, freight and logistics, commissioning spares, initial fills, commissioning and owner's costs.

21.4 Operating Costs Summary

The OPEX estimate is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a preliminary study.

The operating cost estimate in this study includes the costs to mine, handling and transport HL feed to the process plant, process the mineralized material to doré and general and administrative expenses (G&A). These items total the mine operating costs. The total life-of-mine costs are summarized in Table 21-4.

Table 21-4: LOM Operating Costs

Operating Cost	$/tonne milled	LOM C$M
Mining & Rehandle ($2.29/t mined)	11.86	610.0
Processing	6.67	343.2
G&A	4.00	205.8
Total Operating Costs	**22.53**	**1,159.0**

21.5 Leach Processing Costs

Process operating costs were estimated to include all crushing, heap loading, and gold recovery activities to produce unrefined gold bullion (doré). All costs are in 2014 Canadian dollars and all tonnages are metric tonnes. The annual mineralized material treatment rate is 5 million tonnes. The crushing plant is designed at 794 tonnes per hour and heap loading is designed at 1,111 tonnes per hour. Labour rates and benefit loadings at 35% are based on information supplied by JDS. All reagents cost estimates are under the in Leach & Plant section. All power is supplied by diesel generators and power costs are estimated at $0.35 per kilowatt hour.

The process operating costs are summarized in Table 21-3.

Table 21-5: Summary of Process Operating Costs

	C$M / Year	$ / Tonne HL feed
Crushing & Conveying	11.6	2.32
Leach & Plant	18.5	3.70
Refine & Laboratory	3.2	0.65
Total Processing Cost	**33.3**	**6.67**

21.5.1 Manpower Requirements

All Supervision for Process Operations are considered to be staff positions and all other functions are hourly positions. Manpower was based on a 4 crew working 12 hour shifts on a two week on/ two week off rotation.

Process manpower including burden and annual costs are presented in Table 21-4.

Table 21-6: Process Manpower

SUPERVISION	Quantity
Leach Process Superintendent	1
Metallurgical Superintendent	1
Metallurgist	1
Leach General Foreman	1
Leach Pad Planner	1
Shift Supervisor	4
Chief Chemist	1
Maintenance Superintendent	1
Maintenance Planner	1
Administrative Assistant	4
	Quantity
CRUSHING & CONVEYING	
Crushing Operator	8
Crushing Helpers	8
Stacker Operators	4
Conveyor Operators	4
Dozer/FEL Operators	4
Labourers	8
LEACHING AND PLANT	
Leach Plant Operator	4
Leach Plant Helper	4
Piping Crew	8
Labourers	4
Treatment Plant Operators	4
Treatment Plant Helpers	4
Labourers	4
REFINERY AND LABORATORY	
Gold Recovery Operators	6
Assay Technicians	8
Sample Preparation	8
MAINTENANCE	
Mechanics/Welders	3
Mechanic Apprentice	2
Lead Electrician (Shared)	1
Electricians	3
Electrical Apprentices	2
Instrument Technicians	1
General Labour for Maintenance	8
Totals	**126**

Table 21-5 presents the allocation of supervision and maintenance manpower n to process areas.

Table 21-7: Supervision and Maintenance Manpower Allocation

Process Area	Percentage Allocation
Crushing & Conveying	25%
Leach & Plant	40%
Refine & Laboratory	35%
Total	**100 %**

21.5.2 Crushing and Conveying

The estimated costs for crushing and conveying are given in Table 21-6.

Table 21-8: Estimated Costs for Crushing and Conveying

Crushing & Conveying Costs	$ / Year
Manpower	3,144,000
Primary Wear Parts	500,000
Secondary Wear Parts	500,000
Tertiary Wear Parts	500,000
Miscellaneous Wear Parts	100,000
Screens	50,000
Conveyor Belting	250,000
Conveyor Parts	250,000
Miscellaneous Supplies / Services	250,000
Sub Total	**5,544,393**
Power Cost	
Connected Power (kw)	3,000
Power Consumed (kwh/year)	17,325,000
Power Cost @ $0.35 per kwh	**6,064,000**
Total Crush and Convey Costs	**11,608,000**
Cost per Tonne HL feed	**2.32**

21.5.3 Leach and Plant

Table 21-9: Leach and Plant Operating Costs

Leach & Plant	**$ / Year**
Operating and Maintenance Manpower	3,335,000
Piping and Solution Distribution	750,000
Rehandling Costs Stockpile to Heap	750,000
Sub Total	**4,835,000**
Power Cost	
Connected Power (kw)	814
Power Consumed (kwh/year)	6,838,000
Power Cost @ $0.35 per kwh	**2,393,000**
Sub Total Leach & Plant Costs	**7,228,000**
Sub Total Costs Per Tonne	1.45
Plus Reagents Per Tonne	2.11
Plus Water (allowance) Per Tonne	0.15
Total Leach & Plant Costs Per Tonne	**3.70**

Reagent costs in the leach and plant area also include all regent costs for the refinery and laboratory. Reagent consumptions were taken from the column leach tests done by KCA. As recommended by KCA average sodium cyanide consumptions were factored by 0.33 since operating experience from many heap leach operations actually experience much lower consumptions in actual operations. Reagent costs were based on recent quotes. Table 21-8 summarizes the major reagent and chemical requirements.

Table 21-10: Reagent Costs

Reagent	**Cost ($/kg)**	**Consumption (kg/t)**	**$/t HL feed**
Lime	$0.60	1.50	0.90
NaCN	$4.50	0.20	0.90
NaOH	$1.00	0.10	0.10
Carbon	$3.50	0.01	0.04
Antiscalant	$3.50	0.02	0.07
Refinery	Allowance		0.05
Laboratory	Allowance		0.05
Total			**2.11**

21.5.4 Refine and Laboratory

The refining and laboratory costs are given in Table 21-9.

Table 21-11: Refining and Laboratory Costs

Refine & Laboratory	**$ / Year**
Operating and Maintenance Manpower	2,762,000
Miscellaneous Supplies Allowance	250,000
Sub Total	**3,012,000**
Power Cost	
Connected Power (kw)	205
Power Consumed (kwh/year)	615,000
Power Cost @ $0.35 per kwh	**215,000**
Total Refine & Laboratory Costs	**3,228,000**
Cost Per Tonne	**0.65**

21.6 Mine Operating Costs

The open pit mining activities for the Coffee project were assumed to be undertaken by the owner as the basis for this study. They are presented in Q2-2014 Canadian dollars and do not include allowances for escalation or exchange rate fluctuations. Open pit mining costs are a summation of operating and maintenance labour, administrative labour, parts and consumables, fuel, and miscellaneous operating supplies.

The mining unit rate was calculated from first principles based on equipment required for the mining configuration of the operation as described in Section 16 of this report, as well as a comparison to similar sized open pit gold operations in the region. Local labour rate assumptions and an assumed diesel fuel price of C$1.16/litre (delivered) were taken into consideration in determining the operating mining cost. The open pit mining costs encompass pit and dump operations, road maintenance, mine supervision and technical services cost.

The average open pit operating costs for the LOM plan are presented in Table 21-10 and Table 21-13, both by mining function and category. These costs are based on the LOM schedule presented in Section 16 of this report and account for the material tonnages mined and their associated costs.

Table 21-12 Open Pit Operating Cost Estimate – by Function

Cost Function	Average Cost (C$)/Tonne Mined
Drilling	0.28
Blasting	0.39
Loading	0.36
Hauling	0.71
Roads & Dumps	0.35
General Mine/Maintenance	0.17
Supervision & Technical	0.22
Total Open Pit Operating Cost	**2.48**

Table 21-13 Open Pit Operating Cost Estimate – by Category

Cost Category	Cost $/Tonne Mined
Operating Labour	0.39
Maintenance Labour	0.27
Supervision & Technical	0.20
Non-energy Consumables	0.85
Fuel	0.76
Leases, Outside Services, Misc.	0.02
Total Open Pit Operating Cost	**2.48**

21.7 General Administrative Costs

General administrative (G&A) annual operating costs are summarized in Table 21-14.

The following items are included in each G&A operating cost category:

Labour – Administration, Infrastructure Maintenance, First Aid, Environment, Health & Safety, Human Resources, IT and Security staff and hourly personnel.

Equipment – Operating costs for the site mobile equipment (small loaders, forklifts, etc.).

Materials – Communication, first aid and office supplies.

Expenses – Insurance, business travel, site access road maintenance, storage, legal, recruitment / training / safety programs, consultants, community engagement, freight and infrastructure power (power cost attributed to camp, truck shop / warehouse, etc.).

Services – Catering and employee air travel.

Table 21-14: G&A Operating Costs Summary

Item	M$ / Year	$ / t HL feed
Labour	4.5	0.91
Equipment	0.5.	0.09
Materials	0.3	0.06
Expenses	6.1	1.22
Services	6.0	1.20
Subtotal G&A	**17.4**	**3.49**
Contingency	2.6	0.51
Total G&A	**20.0**	**4.00**

22.0 ECONOMIC ANALYSIS

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely 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 variations in metal prices, 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 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 Section 21 of this report (presented in 2014 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

All costs, metal prices and economic results are reported in Canadian dollars (CDN). LOM plan tonnage and grade estimates are illustrated on Table 22-1.

Table 22-1: LOM Plan Summary for All Cases

Category	Unit	Value
Mine Life	Years	11.1
Total Oxide HL feed	M tonnes	51.5
Total Transition HL feed	M tonnes	1.9
Total HL feed	**M tonnes**	**53.4**
Total Waste	M tonnes	212.4
Strip Ratio	w:o	4.0
Throughput Rate	tpd	13,251
Average Oxide Grade	g/t	1.21
Average Transition Grade	g/t	1.71
Total Grade	**g/t**	**1.23**
Au Payable	LOM k oz	1,859
	Avg k oz/yr	167

Other economic factors include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated for each scenario)
- Closure Cost of $46.0M which includes a 15% contingency
- Nominal 2014 dollars
- Revenues, costs and 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.)
- Costs from operations incurred in the pre-production period have been capitalized and assumed to cover the working capital necessary

Table 22-2 outlines the metal prices and CDN:USD exchange rates assumptions used in the economic analysis.

Table 22-2: Metal Price and Foreign Exchange Rates used in Economic Analysis Scenarios

Parameter	Units	Three-Year Trailing Average (as of Nov. 30, 2013)	Base Case
Gold Price	US$/oz	1,555	1,250
Exchange Rate	USD:CDN	0.95	0.95

22.2 Revenues and NSR Parameters

Mine revenue is derived from the sale of doré into the international marketplace. No contractual arrangements for concentrate smelting or refining exist at this time. Details regarding the terms used for the economic analysis can be found in the Market Studies Section 19 of this report. Revenues from doré production are assumed to begin in Year -1 and end in Year 11, in line with the 11.1-year mine life.

Table 22-3 indicates the NSR parameters that were used in the economic analysis. Figure 22-1 shows a breakdown of the amount of gold produced during the mine life – a total 1,859 koz of gold is produced during the mine life.

Table 22-4 shows the amount of payable metal for the project. LOM Net Smelter Return (net of royalty payments) for the Base Case amounted to $2,405.3M. Royalties amount to $26.3M. Royalties were calculated as 1% of NSR with a $2M upfront payment in year -1.

Table 22-3: NSR Parameters Used in Economic Analysis

Assumptions	Unit	Value
Recoveries		
Supremo		
Oxide HL feed	%	90.0
Transition HL feed	%	70.0
Latte		
Oxide HL feed	%	88.0
Transition HL feed	%	44.0
Double Double + Kona		
Oxide HL feed	%	90.0
Transition HL feed	%	70.0
NSR Parameters		
Au Payable	%	100
Au Refining Charge	US$/oz	7.50

Figure 22-1: Gold Doré Production by Year

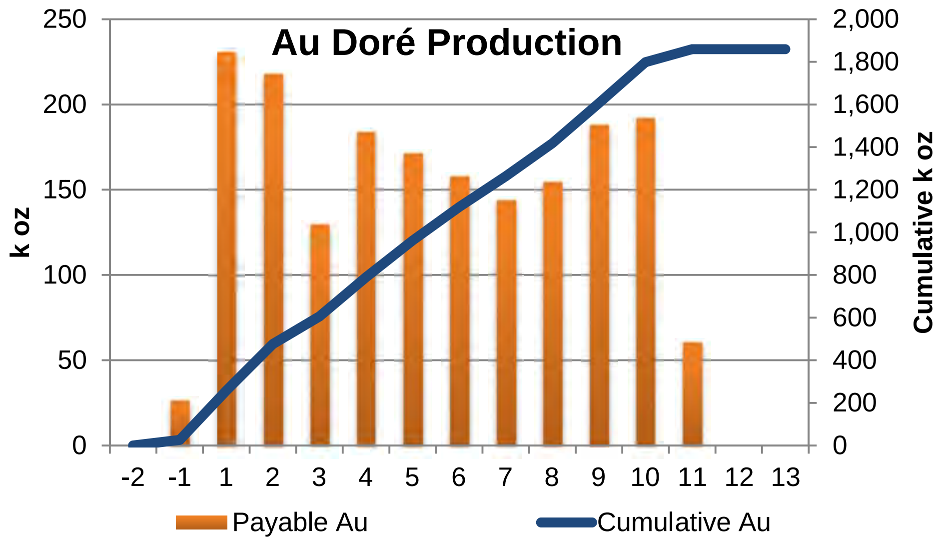


Table 22-4: LOM Payable Metal Production

Payable Metal	Unit	Value
Au	LOM k oz	1,859.2
	LOM NSR $M	2,446.3
Refining Costs	LOM $M	14.7
Total Pre-Royalty NSR	**LOM NSR $M**	**2,431.6**
Royalties	LOM $M	26.3
Total LOM NSR (Net of Royalties)	**LOM $M**	**2,405.3**

22.3 Summary of Capital Cost Estimate

During pre-production (assumed to be Years -2 and -1), the initial capital costs amount to $304.8M. This includes costs for site development, processing plant, on-site infrastructure, camp construction, etc.

A 15% contingency is included in the initial capital costs. A breakdown of the initial capital costs is shown in Table 22-5 and Figure 22-2.

Sustaining and closure capital cost estimates amount to $145.5M and were assumed to occur from Year 1 to Year 11 with a majority of these costs for mining equipment and the leach facility. A 15% contingency is included in the sustaining and closure capital expenditures. A breakdown of the sustaining and capital costs is shown in Table 22-5 and Figure 22-3.

Owner closure costs amount to $46.0M and were assumed to occur in Year 11 through Year 15.

Details on the Capital Costs can be found in Section 21 of this report.

Table 22-5: Summary of LOM Capital Costs

Capital Cost		Pre-Production (C$M)	Sustaining/ Closure (C$M)	Total (C$M)
Capitalized Mining		50.3	0.0	50.3
Pre-Production Operating Costs		16.7	0.0	16.7
Site		57.1	4.8	61.9
Mining Equipment		46.3	64.5	110.8
Leach Facility		56.1	17.2	73.3
Camp		10.3	0.0	10.3
Indirects		37.0	0.0	37.0
Closure		0.0	40.0	40.0
Subtotal		**273.8**	**126.5**	**400.3**
Contingency	15%	31.0	19.0	50.0
Total Capital Costs		**304.8**	**145.5**	**450.3**

Figure 22-2: Breakdown of Pre-Production Capital Costs

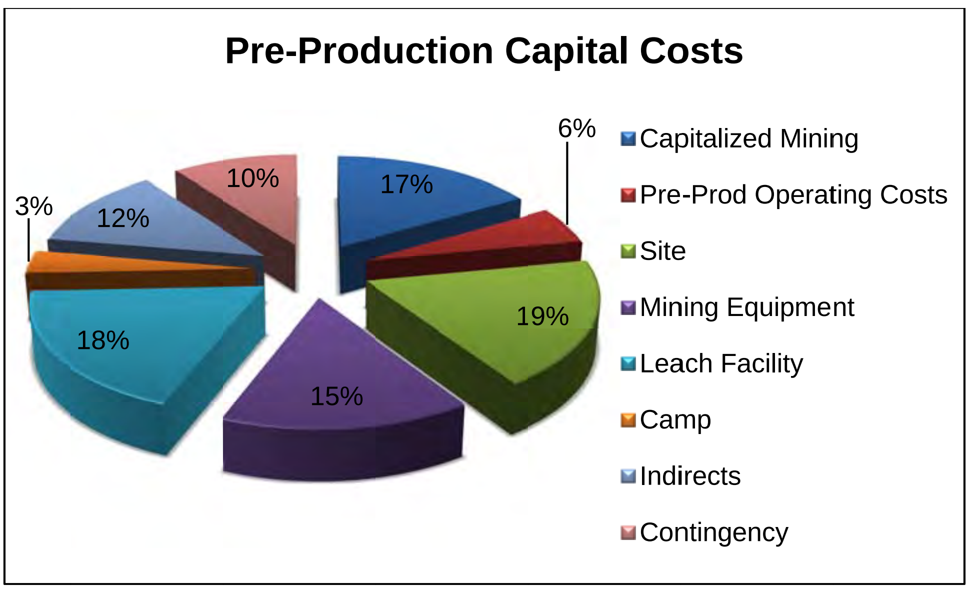
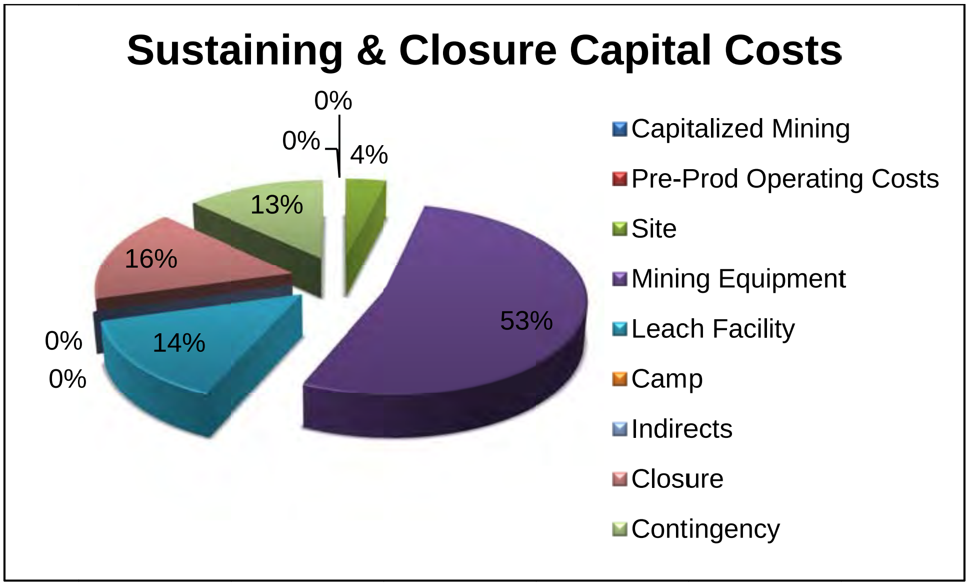


Figure 22-3: Breakdown of Capital Expenditures During Production



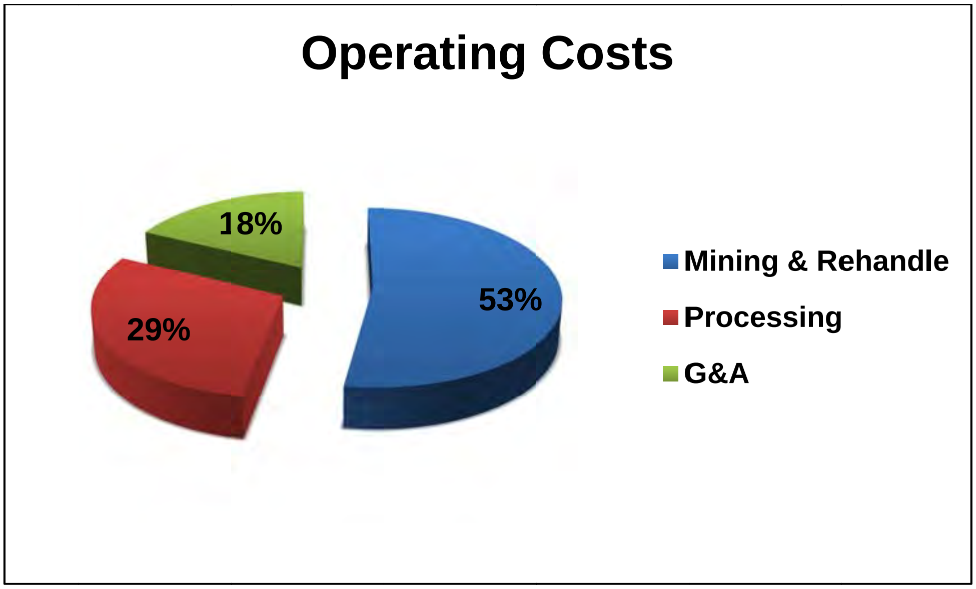
22.4 Summary of Operating Cost Estimates

Total LOM operating costs amount to $1,159.0M. This translates to an average cost of $22.53/tonne milled. A breakdown of these costs is outlined in Table 22-6 and Figure 22-4.

Table 22-6: Operating Costs

Operating Cost	$/tonne milled	LOM $M
Mining & Rehandle ($2.29/t mined)	11.86	610.0
Processing	6.67	343.2
G&A	4.00	205.8
Total Operating Costs	**22.53**	**1,159.0**

Figure 22-4: Breakdown of Operating Costs



22.5 Taxes

The project has been evaluated on an after-tax basis in order to provide a more indicative value of the potential project economics. A specialized mining tax professional was commissioned to review and assist in preparing a tax model for the post tax economic evaluation of the project with the inclusion of applicable federal and provincial income taxes. The tax calculations account for opening tax pools, Yukon Quartz Mining Royalties, provincial and federal income taxes. The tax calculations also assume appropriate capital cost allowance for each of the capital cost class. Total taxes for the life of the project amount to $280.8M.

22.6 Economic Results

The project is economically viable with an after-tax internal rate of return (IRR) of 26.2% and an after-tax net present value using a 5% discount rate ($NPV_{5\%}$) of $330.4 using the Base Case metal pricing. Table 22-7 summarizes the economic results of each scenario evaluated.

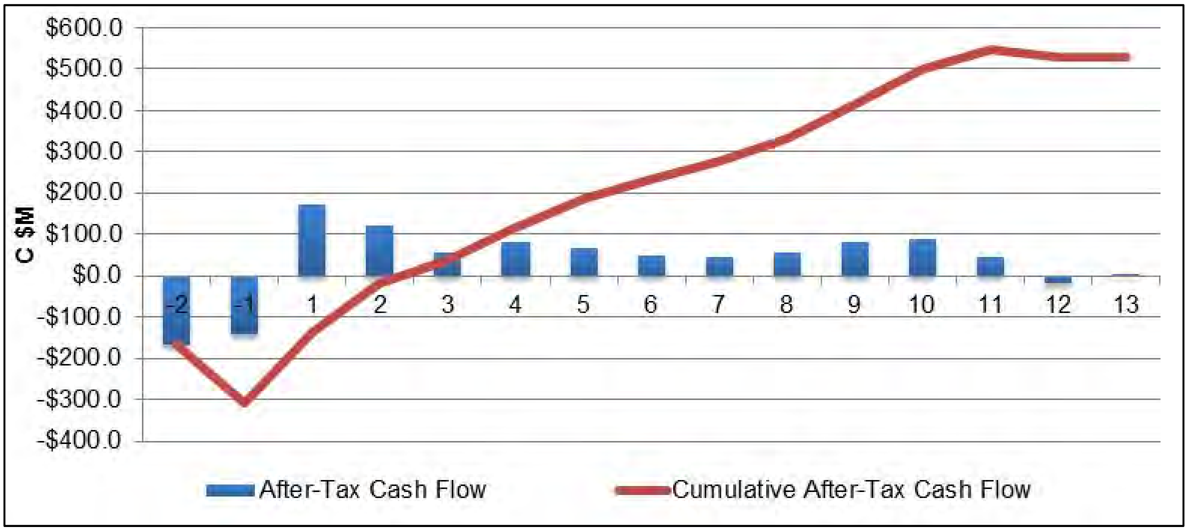
Figure 22-5 shows the projected cash flows for the project used in the different scenarios of the economic analysis.

Table 22-7: Summary of Results for Base Case Scenario

Au @ US$1,250/oz; FX Rate $0.95 USD:CDN

Parameter	Unit	Value
Production		
Mine Life	Yrs	11.1
Au Produced	LOM k oz	1,859
	Avg k oz/yr	168
LOM NSR (after royalties)	$M	2,405.3
Operating Costs	LOM $M	1,246.3
	$/t	23.09
Capital Costs		
Pre-Production	$M	273.8
Sustaining & Closure	$M	126.5
Subtotal	$M	400.3
Contingency 15%	$M	50.0
Total Capital Costs	**$M**	**450.3**
Operating Cash Flow	$M	1,246.3
	$M/yr	112.8
Cash Cost	$/oz	645.43
Economic Results		
After-Tax Free Cash Flow	$M	515.2
	Avg $M/yr	46.6
Discount Rate		5%
Pre-Tax NPV$_{5\%}$	**$M**	**522.4**
Pre-Tax IRR	**%**	**32.8**
Pre-Tax Payback	**Yrs**	**1.8**
After-Tax NPV$_{5\%}$	**$M**	**330.4**
After-Tax IRR	**%**	**26.2**
After-Tax Payback	**Yrs**	**2.0**

Figure 22-5: Annual After-Tax Cash Flows for Base Case Scenario



22.7 Sensitivities

A sensitivity analysis was performed to determine what factors most affected the project economics. The analysis revealed that the project is most sensitive to metal prices, followed by head grades and operating costs. The project showed the least sensitivity to capital costs. Table 22-8 along with Figure 22-6 outline the results of the sensitivity tests performed on after-tax $NPV_{5\%}$ for each of the metal price scenarios evaluated.

The project was also tested under various discount rates. The results of these tests for each of the metal pricing scenarios are demonstrated in Table 22-10. The sensitivity analysis revealed that the Coffee project's break-even gold price is US$900/oz.

Table 22-8: Sensitivity Results for Base Case Scenario

Au @ US$1,250/oz; FX Rate $0.95 USD:CDN

	After-Tax $NPV_{5\%}$ ($M)		
Variable	**-15%**	**100%**	**+15%**
Metal Prices	158.9	330.4	497.8
Head Grade	158.4	330.4	498.1
Operating Costs	410.2	330.4	249.6
Capital Costs	391.1	330.4	269.8

Figure 22-6: Sensitivity Results for Base Case Scenario

Au @ US$1,250/oz; FX Rate $0.95 USD:CDN

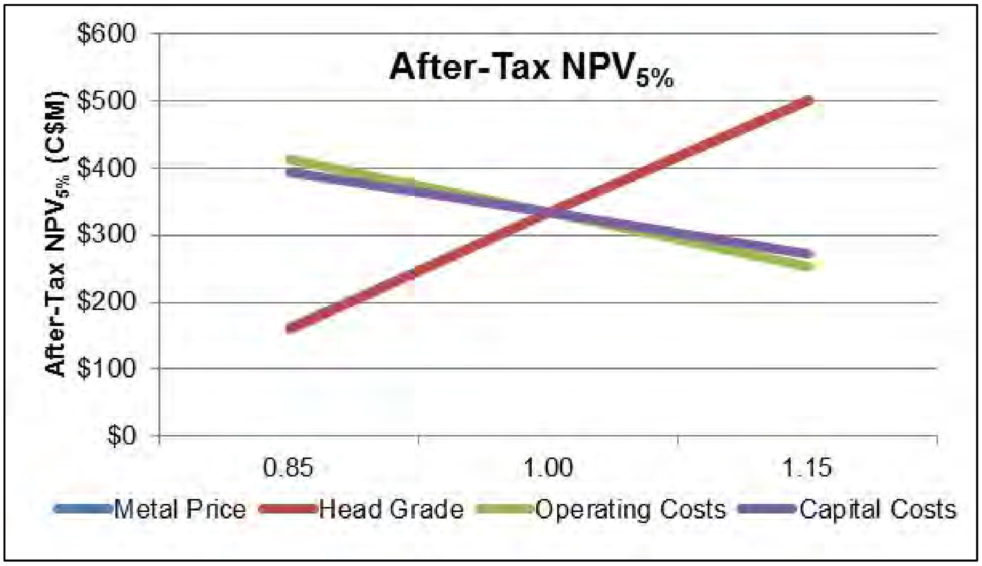


Table 22-9: Project Sensitivity to Au Price

Au US $/oz	After Tax $NPV_{5\%}$ ($M)	After-Tax IRR (%)	After-Tax Payback Period (years)
$1,000	99.4	12.0	4.7
$1,100	193.7	18.1	3.5
$1,200	285.3	23.6	2.5
$1,250	330.4	26.2	2.0
$1,300	375.3	28.7	1.8
$1,400	464.5	33.5	1.6
$1,500	553.1	37.9	1.4

Table 22-10: Base Case Scenario Discount Rate Sensitivity

Au @ US$1,250/oz; FX Rate $0.95 USD:CDN

Discount Rate	After-Tax NPV ($M)
0%	515.2
5%	330.4
7%	274.8
8%	250.0
10%	205.6
12%	167.1

23.0 ADJACENT PROPERTIES

There are no adjacent properties considered relevant to this technical report.

24.0 OTHER RELEVANT DATA AND INFORMATION

This section is not applicable.

25.0 INTERPRETATIONS AND CONCLUSIONS

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the Coffee project. Standard industry practices, equipment and design methods were used in the PEA.

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

Using the assumptions contained in this report, the project is economic and should proceed to the pre-feasibility or feasibility stage.

There is also a likelihood 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.

To date, the QPs are not aware of any fatal flaws for the Coffee project.

25.1 Risks

As with most mining projects there are many risks that could affect the economic viability of the project. Many of these risks are based on lack of detailed knowledge and can be managed as more sampling, testing, design, and engineering are conducted at the next study stages. Table 25-1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches.

The most significant potential risks associated with the project are uncontrolled dilution, uncontrolled groundwater inflow in the pit, lower gold recoveries than those projected, 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.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. 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. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource and reserve estimates.

Table 25-1: Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Water Inflow into Pits	The management of water on-site is a critical component of the project design. Basic assumptions were made for surface and ground water flows based on preliminary drilling and hydrogeologic information.	Continued collection and analysis of data relating to ground and surface water needs to be continued on-site over the near-term to enhance the local hydrological knowledge.
Dilution	Higher than expected dilution has a severe impact on project economics. The open pit mine must ensure accurate drilling and blasting practices are maintained to minimize dilution, minimize secondary breaking and optimize extraction. The ability to segregate higher grade material, early in the mine life, is critical to project economics.	A well planned and executed grade control plan is necessary immediately upon commencement of mining.
Resource Modelling	All mineral resource estimates carry some risk and are one of the most common issues with project success. 80% of the resources in the mine plan are Inferred.	Infill drilling may be recommended in order to provide a greater level of confidence in the resource.
Metallurgical Recoveries	Changes to metallurgical assumptions could lead to reduced metal recovery, increased processing costs, and/or changes to the processing circuit design. If LOM gold recovery is lower than assumed, the project economics would be negatively impacted.	Additional sampling and test work could be conducted if applicable.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. If OPEX increases then the mining cut-off grade would increase and, all else being equal, the size of the optimized pit would reduce yielding fewer mineable tonnes.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Permit Acquisition	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Geochemistry and Water Management	Potentially Acid-Generating (PAG) material is not currently defined in 3D geological block model. Acid-based accounting (ABA) testing needs to be completed to verify that PAG doesn't exist. If PAG material is present it will result in increased handling costs (liners for dump, etc.) Seepage control may need to be beyond the level stated in this report for the tailings facility and the waste rock dump. Water treatment at closure may last longer than anticipated due to seepage and pit filling.	Further test work could be conducted to help build a block model of PAG material. Further hydrology work may be needed along with water treatment plans.
Development Schedule	The project development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the project economics.	If an aggressive schedule is to be followed, PFS field work should begin as soon as possible.
Mine Geotech	The geotechnical nature of the open pit wall rock, including the nature and orientation of faults and secondary geological structures could impact pit slopes. Pit slopes could be increased or decreased and thus alter the pit designs, mineable tonnes, and strip ratio.	Improved geotechnical knowledge and modeling.
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the project. High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	The early search for professionals as well as competitive salaries and benefits identify, attract and retain critical people.

25.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the project. The major opportunities that have been identified at this time are summarized in Table 25-2, excluding those typical to all mining projects, such as changes in metal prices, exchange rates, and etcetera. Further information and assessments are needed before these opportunities should be included in the project economics.

Table 25-2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of the Pits	The mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource.	Increased mine life.
Backfilling Pits	Backfilling mined pits with waste.	Reduce surface dump sizes and possibly haul distances.
Optimize Mine Plan	Optimize the mine plan and pit sequence.	Smooth waste profiles and increase heap leach feed grade.
Contract Mining	Contract mining instead of owner mining.	Reduce Capex
Run of Mine Leaching	Leach run of mine material	Reduce crushing costs
Optimize Heap Leach Construction	Optimize heap leach construction schedule	Reduce initial Capex
Project Access and Transportation	Barging of construction and operations equipment & supplies.	Reduce construction schedule and construction Capex & Opex.
Pit Slope Steepening	Pit slope angles could potentially be improved which may increase slope angles (conversely it could also make them shallower).	An increase in overall pit slopes for all domains in all pits would reduce the strip ratio and increase the ounces mined.
Project Strategy and Optimization	With additional detailed planning and a series of strategic option reviews the project may be able to add value.	Planning and executing the project with the optimum mine design/schedule and processing systems would result in the maximum possible value to shareholders and other economic stakeholders.
Liquefied Natural Gas (LNG) Power Generation	LNG power generation could reduce Opex.	Reduced Opex and increased net cash flow.

Transitional material /additional metallurgical testing may increase the amount of transitional material that may be treated at a profit // Increase in mine life.

26.0 RECOMMENDATIONS

It is recommended that the project proceed to the feasibility study stage in line with Kaminak's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Kaminak's project development plans.

It is estimated that a feasibility study and supporting field work would cost approximately $25.5 million. A breakdown of the key components of the next study phase is as follows in Table 26-1.

Table 26-1 Cost Estimate to Advance Project to Feasibility Stage

Component	Estimated Cost (M$)	Comment
Infill drilling	14.0	Conversion of inferred resources to indicated within and immediately adjacent to the pit shells. Drilling will include holes for combined resource, geotech and hydrogeology purposes plus additional 4 geotechnical holes
Metallurgical testing	0.5	Variability test work and crush size trade off study
Condemnation drilling	3.0	Drilling under waste dumps, HLF & infrastructure to ensure no sterilization of resources
Geochemistry	0.5	ABA accounting tests and humidity cell testing to determine acid generating potential of all rock units and mitigation plans
Geotechnical/ Hydrology/Hydrogeology	1.0	Mine and surface facilities geotechnical investigations (logging, test pitting, sampling, lab tests, etc.)
Engineering	3.5	FS-level mine, infrastructure & process design, cost estimation, scheduling & economic analysis
Environment	3.0	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**25.5**	Excludes corporate overheads and future permitting activities

Key components of the next study phase will include the following activities:

- Approximately four additional oriented geotechnical drillholes should be planned to an average approximate depth of 150 m. The geotechnical drilling will require a HQ diameter, 1.5 m triple tube core barrel. Core retrieved from the four holes should have detailed geotechnical information logged such as intact rock strength, and frequency and conditions of natural discontinuities.
- Condemnations drilling to ensure mineral resources are not sterilized by infrastructure and waste dump facilities.
- Continue to conduct geotechnical investigations of the selected heap leach, waste dump and plant site locations.
- Carry out metallurgical test work to confirm recoveries for all significant rock types.
- Carry out acid base accounting testing to quantify PAG and NAG material.
- Further hydrology study to better understand and manage water balance and potential water treatment for any surplus water discharge.
- Conduct investigation of the natural gas powered generator plant at site for power source.
- Infill drilling to increase Inferred resources to Indicated in order to form the basis of future economic calculations.

27.0 UNITS OF MEASURE, CALCULATIONS AND ABBREVIATIONS

Table 27-1: Units of Measure

'	Foot
"	Inch
µm	Micron (micrometre)
Amp	Ampere
Ac	Acre
C$	Canadian dollars
cfm	Cubic feet per minute
cm	Centimetre
dpa	Days per annum
dmt	Dry metric tonne
ft	Foot
ft^3	Cubic foot
g	Gram
hr	Hour
ha	Hectare
hp	Horsepower
In	Inch
kg	Kilogram
km	Kilometre
km^2	Square kilometer
KPa	Kilopascal
kt	Kilotonnes
Kw	Kilowatt
KWh	Kilowatt-hour
l	Litre
lb or lbs	Pound(s)
m	Metre
M	Million
m^2	Square metre
m^3	Cubic metre
mi	Mile
min	Minute
mm	Millimetre
Mpa	Mega Pascal
mph	Miles per hour
Mtpa	Million tonnes per annum

Mt	Million tonnes
ºC	Degree Celsius
oz	Troy ounce
ppb	Parts per billion
ppm	Parts per million
s	Second
t	Metric tonne
tpd	Tonnes per day
tph	Tonnes per hour
US$	US dollars
V	Volt
W	Watt
wmt	Wet metric tonne

Table 27-2: Abbreviations & Acronyms

% or pct	Percent
AAS	Atomic absorption spectrometer
ABA	Acid base accounting
ADIS	Automated Digital Imaging System
Ag	Silver
Au	Gold
AMSL	Above mean sea level
ANFO	Ammonium Nitrate/Fuel Oil
AP	Acid potential
ARD	Acid rock drainage
BIF	Banded iron formation
BLS	Barren leach solution
Btu	British Thermal Unit
BWi	Bond work index
$CaCO_3$	Calcium carbonate
CAPEX	Capital costs
CAT	Caterpillar
CCME	Canadian Council of Ministers of the Environment
CEAA	Canadian Environmental Assessment Act
CIC	Carbon-in-Column
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining
CIP	Carbon-in-Pulp
Class EA	Class Environmental Assessment

CLU	Change of land-use authorization
CPM	Critical path method
COSEWIC	Committee on the Status of Endangered Wildlife in Canada
CRM	Certified reference material
Cu	Copper
CV	Coefficient of variation
CWi	Bond crusher work index
DFO	Department of Fisheries and Oceans
DO	Dissolved oxygen
EA	Environmental Assessment
EAA	Environmental Assessment Act
EASR	Environmental Activity and Sector Registry
ECA	Environmental Compliance Approval
EEM	Effluent Effects Monitoring
Elev	Elevation above sea level
ESIA	Environmental-Social Impact Assessment
ETF	Exchange traded fund
FA/grav	Fire assay with gravimetric finish
FEL	Front-end loader
FLOT	Flotation
FS	Feasibility Study
GMV	Gross metal value
GPS	Global positioning system
H:V	Horizontal to vertical
HADD	Harmful Alternation or Disruption or Destruction
HDPE	High density polyethylene
HVAC	Heating, ventilation and air conditioning
ICP-MS	Inductively coupled plasma mass spectrometry
ID^2	Inverse distance squared
IMSS	Immigrant and Multicultural Services Society
IRA	Inter-ramp angles
IRR	Internal rate of return
ISN	Payroll tax
ISRMR	In situ rock mass rating
JDS	JDS Energy & Mining Inc.
LGEEPA	General Law of Equilibrium and Environmental Protection
LGPGIR	General Law for Prevention and Integral Management of Waste
LOM	Life of mine
LSA	Local Study Area
MARC	Maintenance and repair contract
MIA-P	Environmental impact manifest

MIBC	Methyl isobutyl carbinol
ML/ARD	Metal leaching/acid rock drainage
MLI	McClelland Laboratories, Incorporated
MMER	Metal Mining Effluent Regulations
MNR	Ministry of Natural Resources
MoE	Ministry of Environment
MSE	Mechanically stabilized earth
N,S,E,W	North, South, East, West
NI 43-101	National Instrument 43-101
NAD	North American Datum
NAG	Non-acid generating
NN	Nearest neighbour
NP	Neutralization potential
NPC	Noise Pollution Control
NPRI	National Pollutant Release Inventory
NPV	Net present value
NSR	Net Smelter Return
Ø	Diameter
OEM	Original equipment manufacturer
OK	Ordinary Kriging
OP	Open Pit
OPEX	Operating costs
PAG	Potentially acid-generating
PAX	Potassium Amyl Xanthate
PDR	Project Description Report
PEA	Preliminary economic assessment
PFS	Preliminary feasibility study
PLS	Pregnant leach solution
PM	Project management
POX	Pressure oxidation
PPM	Project procedures manual
PPV	Peak Particle Velocity
PSA	Project Study Area
QA/QC	Quality assurance/quality control
QMS	Quality management system
RC	Reverse circulation
RISC	Resource Inventory Standards Committee
ROM	Run-of-mine
RQD	Rock quality designation
RSA	Regional Study
SARA	Species at Risk Act

SG	Specific gravity
SAG	Semi-autogenous grinding
STP	Sewage treatment plant
ToR	Terms of Reference Report
TMF	Tailings management facility
The Agency	Canadian Environmental Assessment Agency
UG	Underground
UPS	Uninterrupted power system
UTM	Universal Transverse Mercator
VA	Voluntary Agreement
VOC	Volatile Organic Compound
Vulcan	Maptek Vulcan ™
Whittle	Gemcom Whittle- Strategic Mine Planning ™
WRMF	Waste rock management facility
X,Y,Z	Cartesian coordinates, also Easting, Northing and Elevation

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Appendix A – Qualified Persons Certificates

PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada", with an effective date of June 10, 2014, prepared for Kaminak Gold Corporation;

2. I am currently employed as a Senior Project Manager, with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C2T6;

3. I am a graduate of the University of Alberta with a BSc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I have worked in technical, operations and management positions at mines in Canada, United States, Mexico, Brazil, and Australia. I have been an independent consultant for over 7 years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

6. 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 the NI 43-101;

7. I have visited the Coffee Project site September 29-October 1, 2013;

8. I am responsible for Sections 15, 21, 24 and share responsibility for Sections 1, 2, 3 and 28;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. As of 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;

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: June 10, 2014
Signing Date: July 8, 2014

Original signed and sealed

Michael E. Makarenko, P. Eng.

CERTIFICATE OF AUTHOR

I, Dino Giacobbe Pilotto do hereby certify that:

1. This certificate applies to the Technical Report entitled ”Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada”, with an effective date of June 10, 2014, prepared for Kaminak Gold Corporation;

2. I am currently employed as a Senior Engineer with JDS Energy & Mining Inc. with an office at Suite 860, 625 Howe Street, Vancouver BC V6C 2T6;

3. I am a graduate of the University of British Columbia with a B.A.Sc (Mining & Mineral Process Engineering). I have practiced my profession continuously since 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 am a Registered Professional Mining Engineer in Alberta, Saskatchewan and Northwest Territories/Nunavut;

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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have not visited the Coffee Project site;

7. I am responsible for Section 16 (except 16.3);

8. I have had no prior involvement with the property that is the subject of this Technical Report;

9. As of 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;

10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: June 10, 2014
Signing Date: July 8, 2014

Original signed and sealed

____

Dino G. Pilotto, P. Eng.

CERTIFICATE OF AUTHOR

I, Scot G. Klingmann, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada", with an effective date of June 10, 2014 prepared for Kaminak Gold Corporation;

2. I am currently employed as Senior Engineer by JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver BC V6C 2T6;

3. I am a graduate of Queen's University with an M.Sc. in Mining Engineering, 1999. I have practiced my profession continuously since 1999;

4. I have worked in technical and management positions at mines in Canada. I have been an independent consultant for two years and have performed mine design, planning, cost estimation, technical due diligence reviews and report writing for mining projects around North America.

5. I am a Registered Professional Mining Engineer in British Columbia (#32339);

6. 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 the NI 43-101;

7. I have not visited the Coffee Project site;

8. I am responsible for Section 18;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. As of 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;

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: June 10, 2014
Signing Date: July 8, 2014

Signed and Sealed
Scot Klingmann, July 8th, 2014

Scot G. Klingmann, P.Eng.

CERTIFICATE OF AUTHOR

I, Gordon Doerksen do hereby certify that:

1. I am currently employed as V.P. Technical Services with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver, BC, V6C 2T6;

2. This certificate applies to the technical report titled "Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada", with an effective date of June 10, 2014, prepared for Kaminak Gold Corporation;

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 invlolved 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 did not visit the Coffee Project site;

5. I am responsible for Sections numbers 19, 22, 25, 26, 27;

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 not had prior involvement with the property that is the subject of this 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 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: June 10, 2014
Signing Date: July 8, 2014

ORIGINAL SIGNED AND SEALED

Gordon Doerksen, P.Eng.

Vancouver Office
T 604.687.7545 f 604.689.5041
#860 – 625 Howe Street Vancouver, BC V6C 2T6

Kelowna Office
t 250.763.6369 f 250.763.6302
#200 – 532 Leon Avenue, Kelowna, BC V1Y 6J6

SRK Consulting (U.S.) Inc.
7175 West Jefferson Avenue, Suite 3000
Lakewood, Colorado
80235

denver@srk.com
www.srk.com

Tel: 303.985.1333
Fax: 303.985.9947

CERTIFICATE OF QUALIFIED PERSON

Michael Levy, P.E., P.G.

I, Michael E Levy, am a Professional Engineer, employed as a Principal Geotechnical Engineer with SRK Consulting Inc.

This certificate applies to the technical report titled "Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada", with an effective date of June 10, 2014, prepared for Kaminak Gold Corporation;

I am a registered Professional Engineer in the states of Colorado (#40268) and California (#70578) 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.

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 have not visited the Coffee Project site.

I am responsible for preparation of sections 16.3 of the Technical Report.

I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the National Instrument 43-101;

I have not had prior involvement with the property that is the subject of this Technical Report;

I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: June 10, 2014
Signing Date: July 8, 2014

"Original Signed"

Michael E. Levy, P.E, P.G.

Group Offices:
Africa
Asia
Australia
Europe
North America
South America

Canadian Offices:
Saskatoon 306.955.4778
Sudbury 705.682.3270
Toronto 416.601.1445
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Denver 303.985.1333
Elko 775.753.4151
Fort Collins 970.407.8302
Reno 775.828.6800
Tucson 520.544.3688

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 30 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 "Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada", with an effective date of June 10, 2014, prepared for Kaminak Gold Corporation (the "Technical Report"), and accept professional responsibility for sections 4 through 12, 14 and 23.
7. I visited the Coffee property on September 12-14, 2011; on August 28-29, 2012; and on May 15-16, 2013.
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 Kaminak Gold Corp. 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 8 of July, 2014.

Original signed and sealed

____

Robert Sim, P.Geo

CERTIFICATE OF AUTHOR

I, Fred H. Lightner do hereby certify that:

1. This certificate applies to the Technical Report entitled ”Preliminary Economic Assessment Technical Report Coffee Project Yukon Territory, Canada”, with an effective date of June 10, 2014, prepared for Kaminak Gold Corporation;

2. I am currently engaged as an independent contractor to serve as the Director of Mine Development with Kaminak Gold Corporation with an office at Suite 1020 – 800 West Pender Street, Vancouver, BC V6C 2V6;

3. I am a graduate of the Colorado School of Mines with the degree of Metallurgical Engineer, 1968. I practiced my profession continuously from 1968 until 2004. I retired in 2004 but I have continued to consult to various companies on the operation, development and evaluation of gold heap leach mining properties.

4. I am a Registered Professional Metallurgical Engineer in the State of Colorado:

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 as having held the previous positions of Sr. Project Metallurgist for Amoco Minerals, General Manager of Operations and Development for Tenneco Minerals, Vice President of Operations for Pegasus Gold Corporation, President and Director of Wharf Resources, and Senior Vice President and Chief Operating Officier of Metallica Resources, 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 the NI 43-101;

6. I have visited the Coffee Project site on the dates of August 27-28, 2013 and September 29-October 1, 2013;

7. I am responsible for Sections 13, 17, 20;

8. I have had no prior involvement with the property that is the subject of this Technical Report;

9. As of 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;

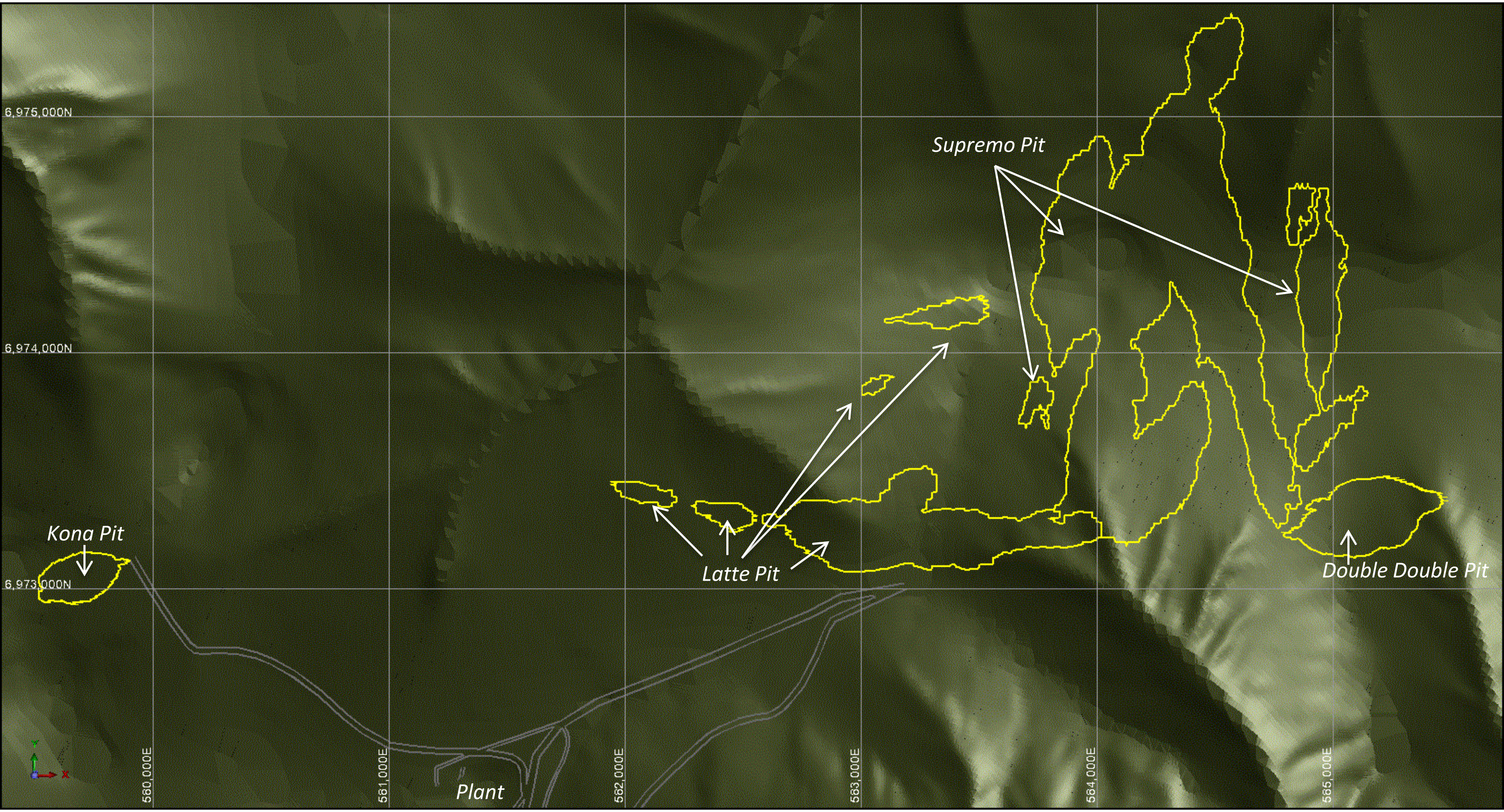
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: June 10, 2014
Signing Date: July 8, 2014

“Signed and Sealed”

____
Fred H. Lightner

Appendix B – Mining End of Year Progress Maps



JDS Energy & Mining Inc.

LEGEND:

Active Areas

Previously Mined/Placed

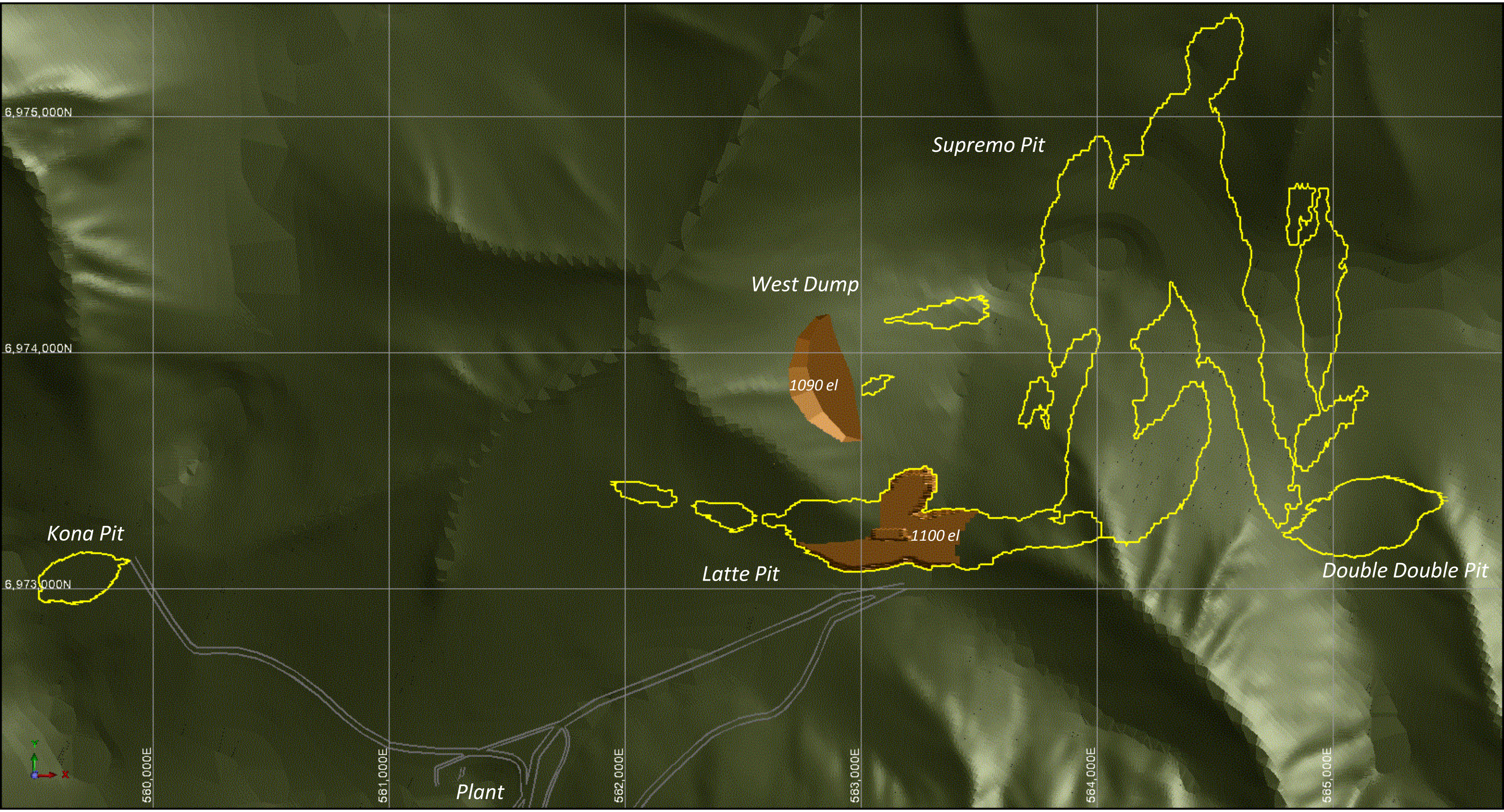
SCALE:	
DSN by:	TFH
DATE:	May 2, 2014
PROJECT #:	
REV #:	2

TITLE

Kaminak - Coffee Gold Project

2014 PEA

Mine Status Map: Mining Areas



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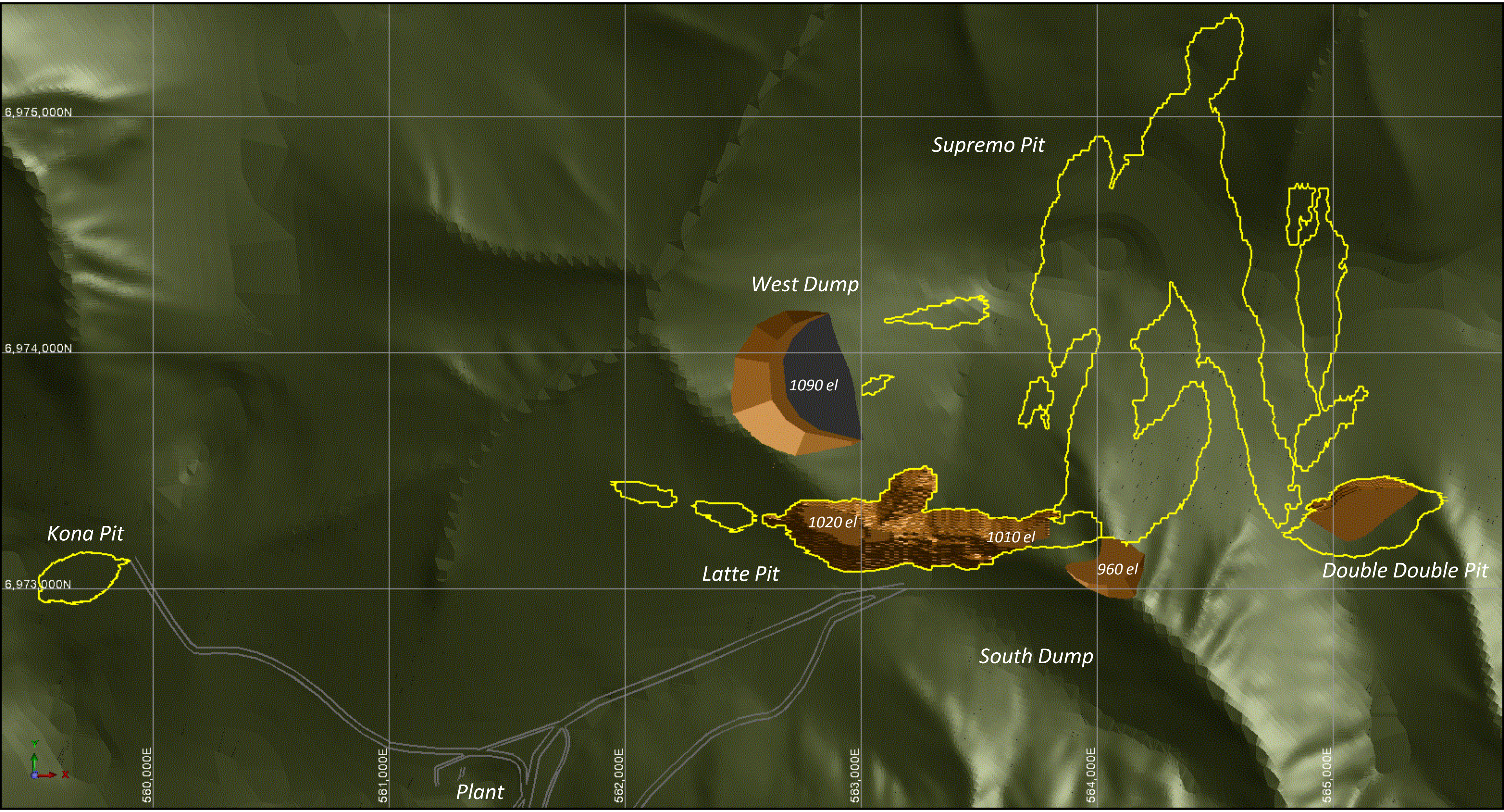
Previously Mined/Placed	DATE: May 2, 2014	**2014 PEA**
	PROJECT #:	**Mine Status Map: End of Year -2**
	REV #: 2	

LEGEND:	SCALE:		TITLE
Active Areas	DSN by:	TFH	**Kaminak - Coffee Gold Project**
Previously Mined/Placed	DATE:	May 2, 2014	**2014 PEA**
	PROJECT #:		**Mine Status Map: End of Year -1**
	REV #:	2	

JDS Energy & Mining Inc.



JDS Energy & Mining Inc.

LEGEND:

- Active Areas
- Previously Mined/Placed

SCALE:

DSN by: TFH

DATE: May 2, 2014

PROJECT #:

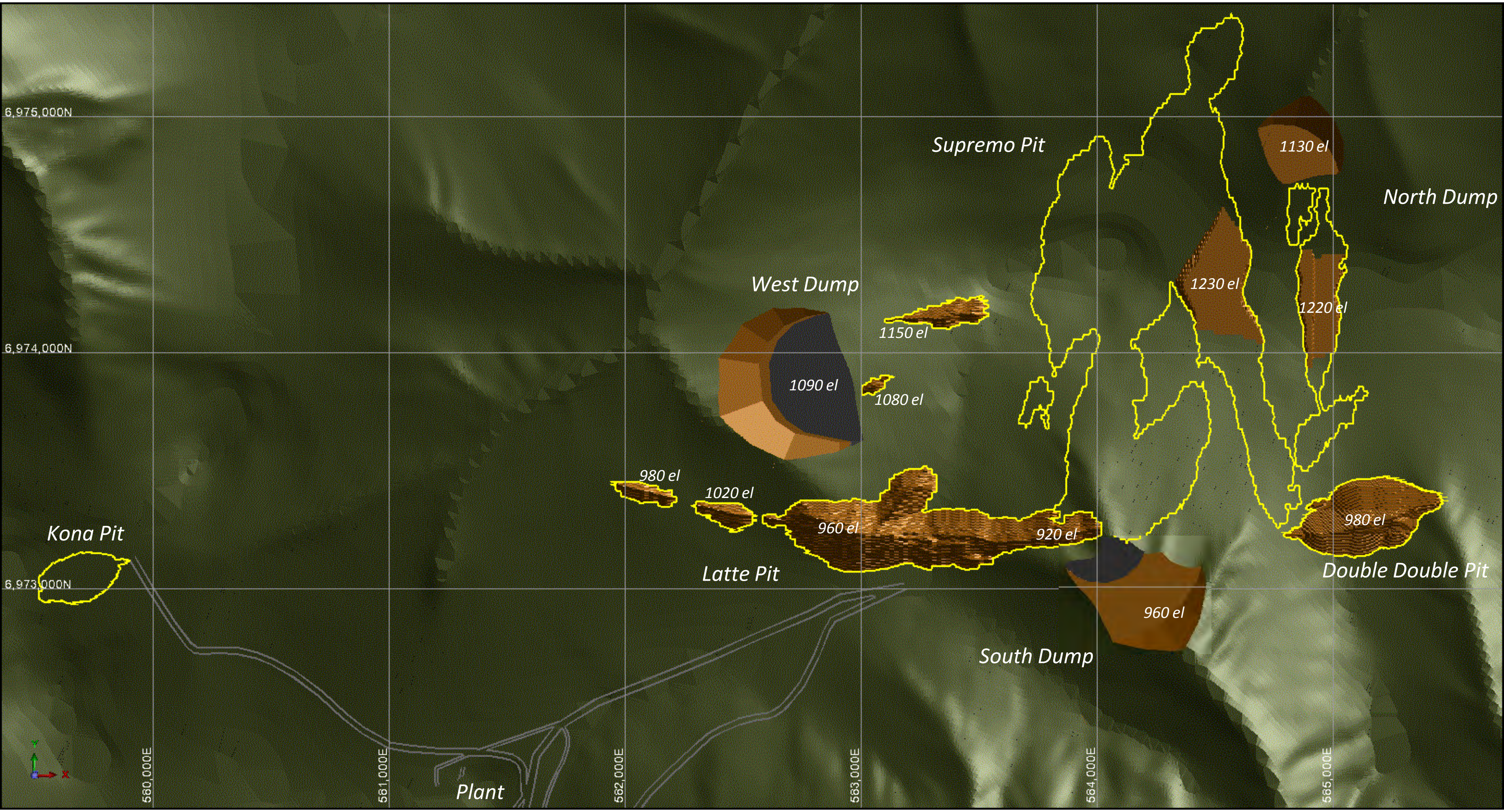
REV #: 2

TITLE

Kaminak - Coffee Gold Project

2014 PEA

Mine Status Map: End of Year 1



SCALE:

DSN by: TFH

DATE: May 2, 2014

PROJECT #:

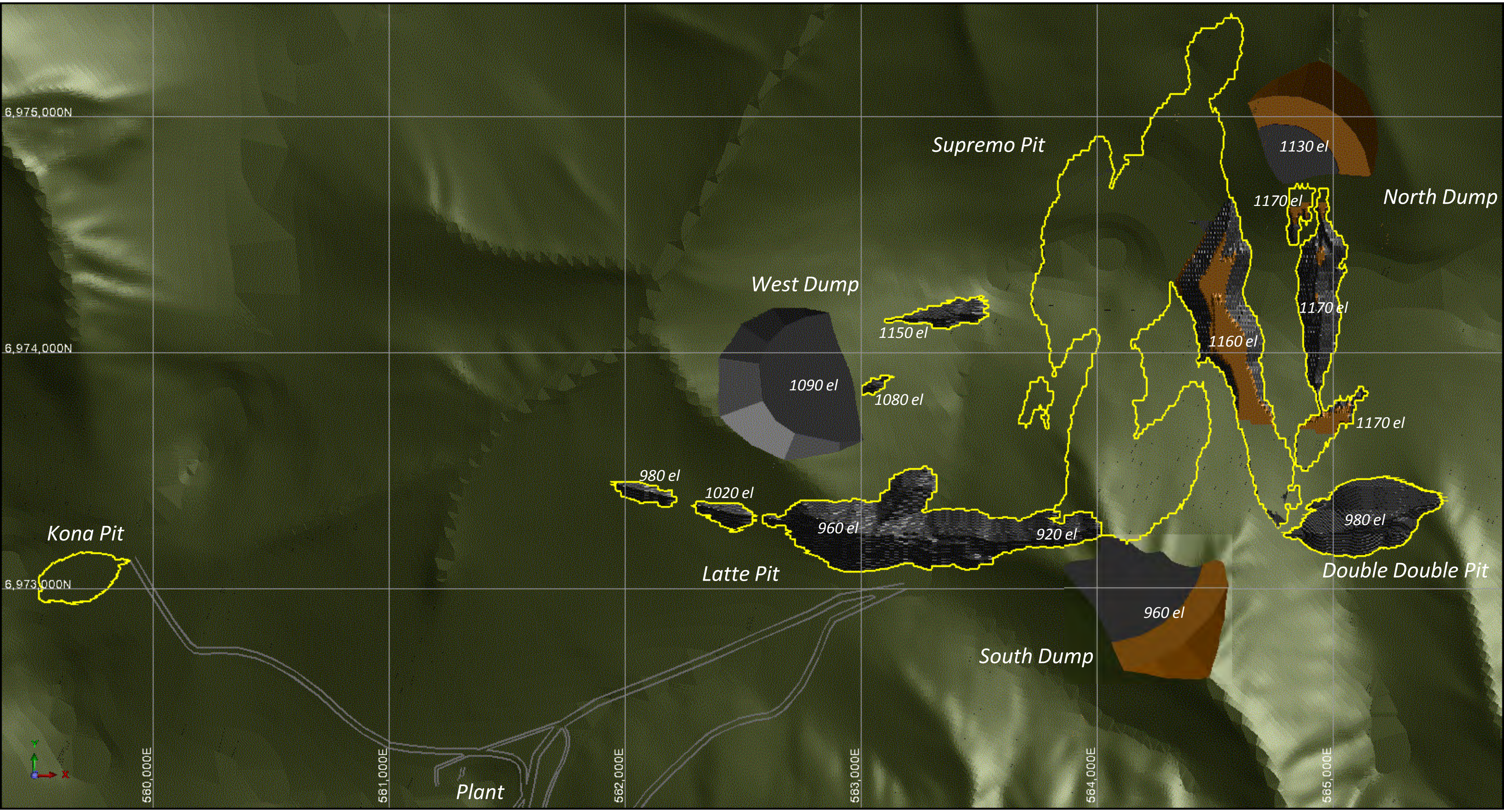
REV #: 2

TITLE

Kaminak - Coffee Gold Project

2014 PEA

Mine Status Map: End of Year 2



LEGEND:

- Active Areas
- Previously Mined/Placed

SCALE:

DSN by: TFH

DATE: May 2, 2014

PROJECT #:

REV #: 2

TITLE

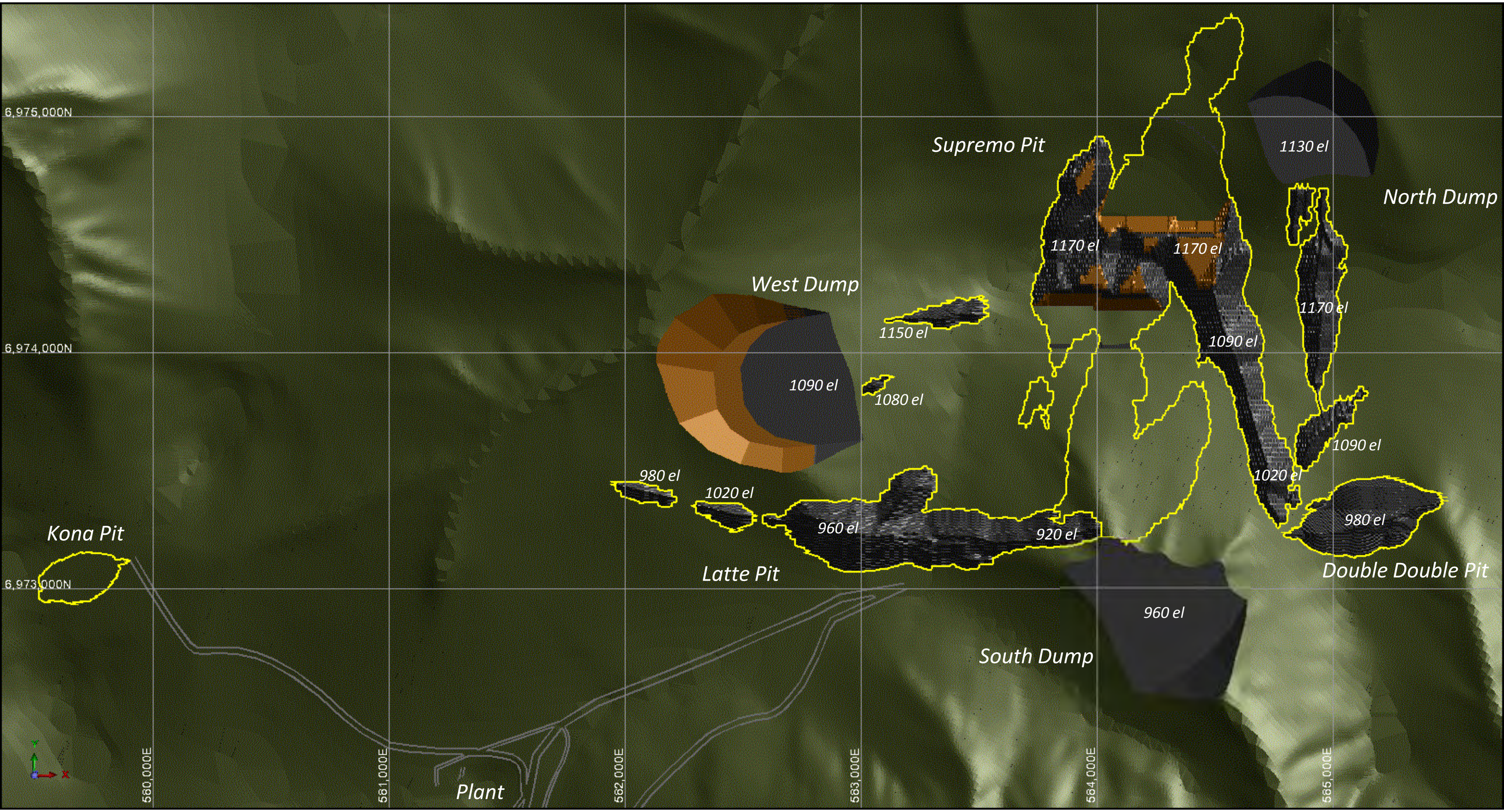
Kaminak - Coffee Gold Project

2014 PEA

Mine Status Map: End of Year 3

JDS Energy & Mining Inc.

LEGEND:	SCALE:	TITLE
Active Areas	DSN by: TFH	**Kaminak - Coffee Gold Project**
Previously Mined/Placed	DATE: May 2, 2014	**2014 PEA**
	PROJECT #:	**Mine Status Map: End of Year 4**
	REV #: 2	



LEGEND:	SCALE:	TITLE
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Previously Mined/Placed	DATE: May 2, 2014	**2014 PEA**
	PROJECT #:	**Mine Status Map: End of Year 5**
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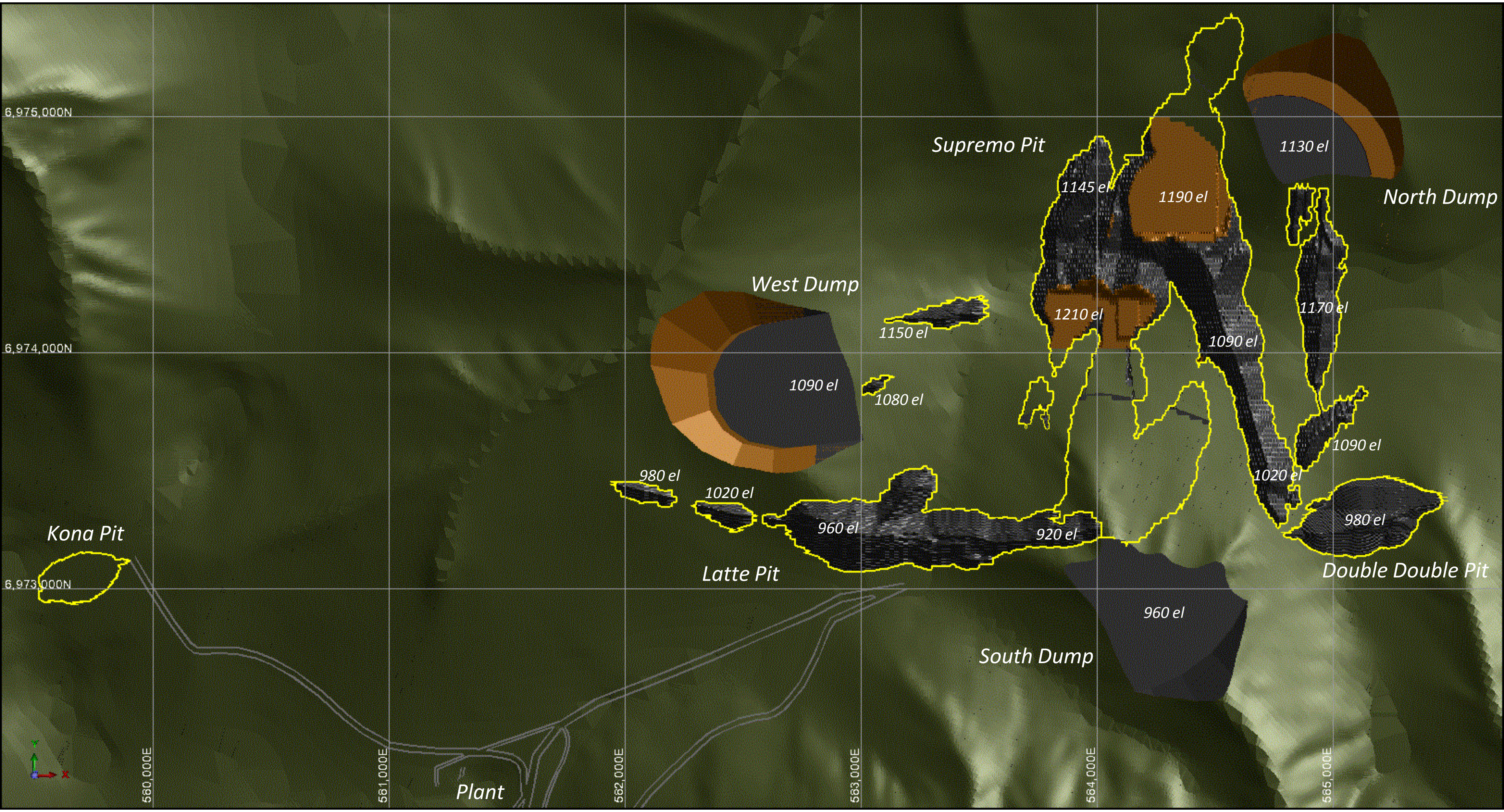

JDS Energy & Mining Inc.	LEGEND:	SCALE:	TITLE
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	Previously Mined/Placed	DATE: May 2, 2014	**2014 PEA**
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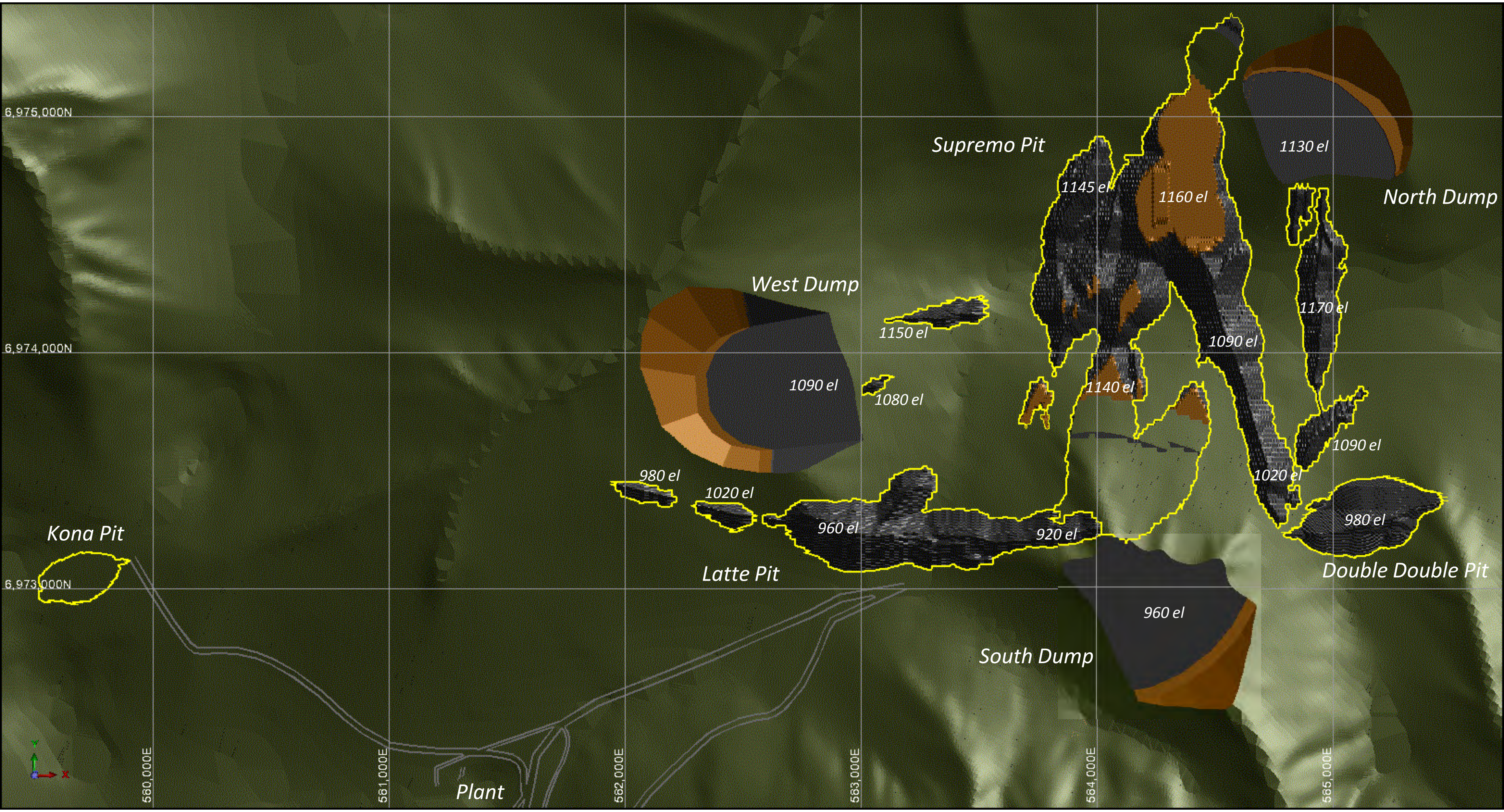
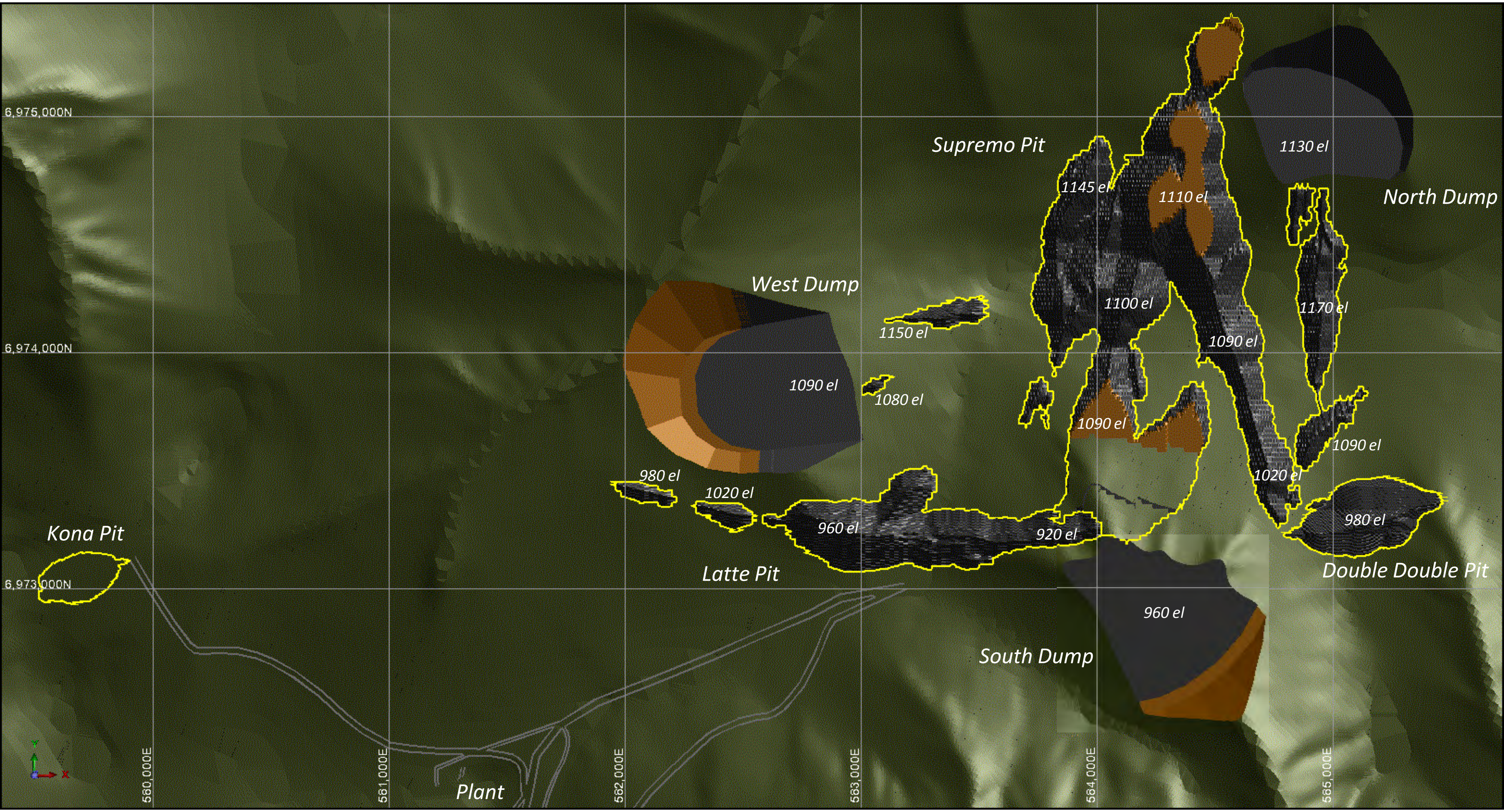


LEGEND: Active Areas Previously Mined/Placed	SCALE: DSN by: TFH DATE: May 2, 2014 PROJECT #: REV #: 2	TITLE **Kaminak - Coffee Gold Project** **2014 PEA** **Mine Status Map: End of Year 7**

JDS Energy & Mining Inc.



LEGEND:

Active Areas

Previously Mined/Placed

SCALE:

DSN by: TFH

DATE: May 2, 2014

PROJECT #:

REV #: 2

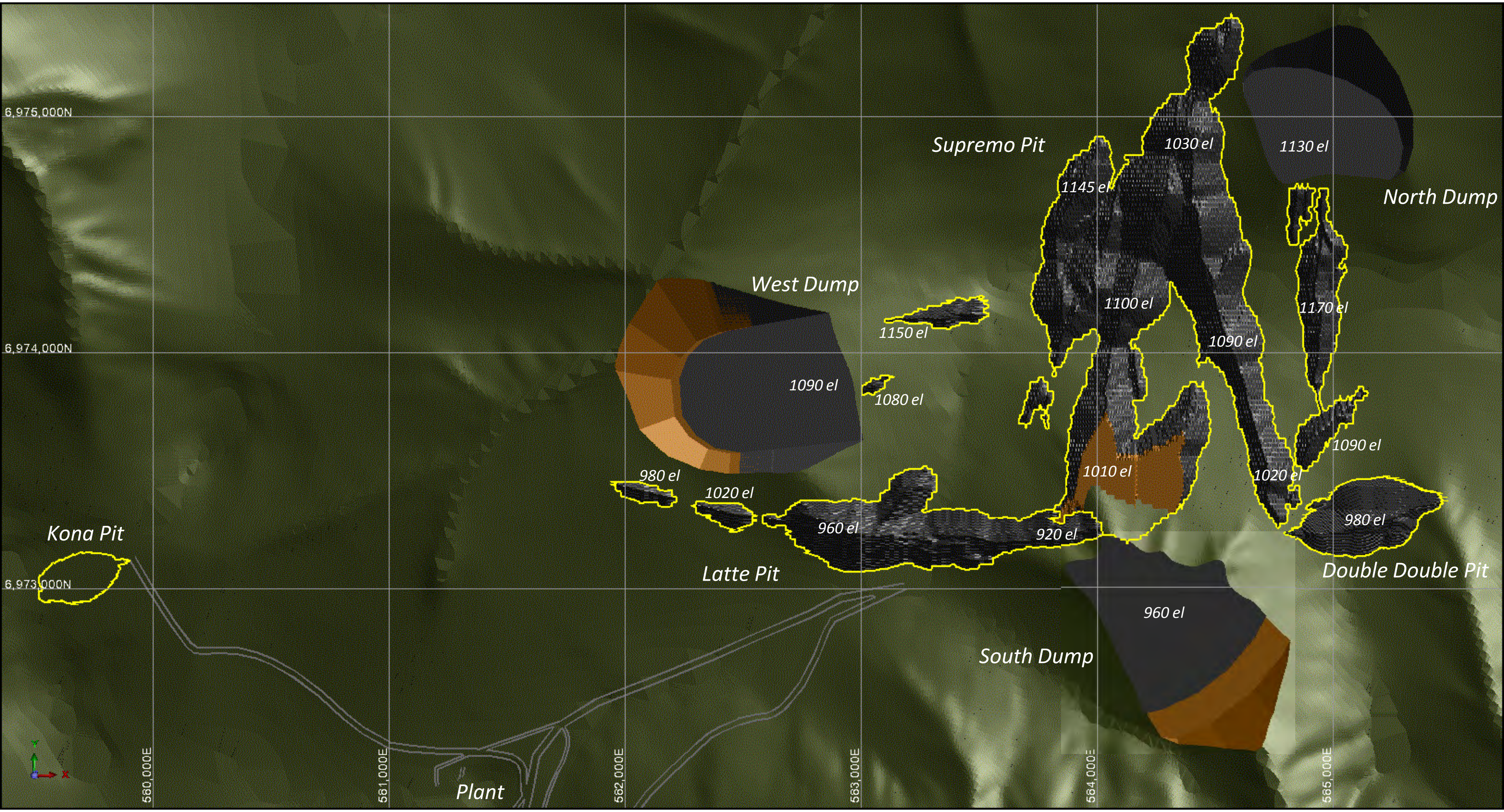
TITLE

Kaminak - Coffee Gold Project

2014 PEA

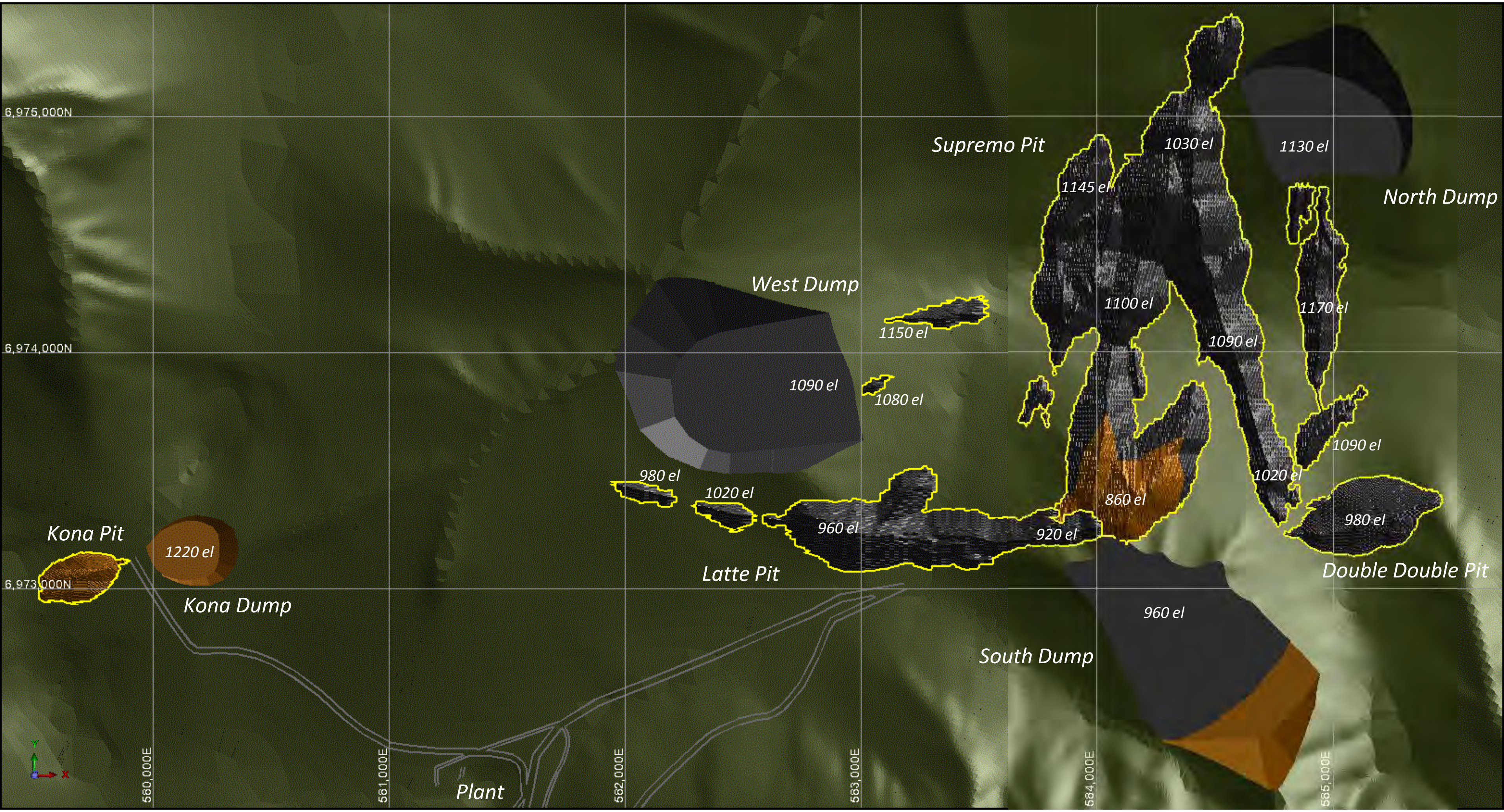
Mine Status Map: End of Year 8

JDS Energy & Mining Inc.



LEGEND:	SCALE:	TITLE
Active Areas	DSN by: TFH	**Kaminak - Coffee Gold Project**
Previously Mined/Placed	DATE: May 2, 2014	**2014 PEA**
	PROJECT #:	**Mine Status Map: End of Year 9**
	REV #: 2	

JDS Energy & Mining Inc.



LEGEND:	SCALE:	TITLE
Active Areas	DSN by: TFH	**Kaminak - Coffee Gold Project**
Previously Mined/Placed	DATE: May 2, 2014	**2014 PEA**
	PROJECT #:	**Mine Status Map: End of Year 10**
	REV #: 2	

JDS Energy & Mining Inc.

Appendix C – KCA Letter

KCA
Kappes, Cassiday & Associates www.kcareno.com
7950 Security Circle • Reno, Nevada USA 89506 • Telephone: (775) 972-7575 • Email: kca@kcareno.com

May 9, 2014

Mr. Fred Lightner
Director, Mine Development
Kaminak Gold Corporation
Suite 1020 – 800 West Pender St.
Vancouver, BC V6C 2V6
CANADA

Dear Fred,

At your request we have reviewed the column leach tests for the Coffee Project of Kaminak Gold to comment on gold recovery and reagent consumption estimates to be used for a PEA. Below is a table which presents a summary of our laboratory test results for the Oxide and Upper Transition material types.

KCA Test No.	Description	Material Type	Column Tail p80 Size mm	Au Extracted %	Cons. NaCN, kg/MT	Addition $Ca(OH)_2$ kg/MT
69157	Latte Oxide	Core	24.9	90	0.19	1.51
69160	Latte Oxide	Core	12.3	90	0.27	1.51
69911	Latte Oxide	Bulk*	111	88	0.56	1.01
69914	Latte Oxide	Bulk	24	92	1.08	1.00
68155	Latte Upper Transition	Core	12.2	47	0.46	2.01
69145	Supremo Oxide	Core	24.8	92	0.17	1.51
69148	Supremo Oxide	Core	12.5	94	0.28	1.50
69554	Supremo Oxide	Core*	12.6	95	0.52	1.57
69917	Supremo Oxide	Bulk*	83	85	0.91	1.52
69920	Supremo Oxide	Bulk	24	92	0.93	1.51
69151	Supremo Upper Transition	Core	12.3	73	0.31	1.00

***tests run at 22 degrees C, all other tests run at 4 degrees C**

The table below presents our prediction of actual field recoveries and reagent consumptions to be used for the PEA based on our experience. The recoveries have been reduced by 3%, NaCN consumptions have been factored by 33%, and hydrated lime requirements are taken directly from laboratory test results. Naturally these projections assume similar samples to those we tested and a well managed heap leach operation.

KCA Test No.	Description	Field Adjusted Recovery %	Field Adjusted NaCN, kg/MT	Field Hydrated Lime kg/MT
69157	Latte Oxide	87	0.06	1.51
69160	Latte Oxide	87	0.09	1.51
69911	Latte Oxide	85	0.18	1.01
69914	Latte Oxide	89	0.35	1.00
68155	Latte Upper Transition	44	0.15	2.01
69145	Supremo Oxide	89	0.06	1.51
69148	Supremo Oxide	91	0.09	1.50
69554	Supremo Oxide	92	0.17	1.57
69917	Supremo Oxide	82	0.30	1.52
69920	Supremo Oxide	89	0.31	1.51
69151	Supremo Upper Transition	70	0.10	1.00

If you have questions or need clarification please contact me.

Yours truly,

Daniel W. Kappes
President
Kappes, Cassiday & Associates
Nev. Reg. Professional Engineer # 3223

PROFESSIONAL ENGINEER - STATE OF NEVADA
DANIEL W. KAPPES
METALLURGICAL
No. 3223

expires 30 June 2015

Appendix D – Economic Model

		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	LOM TOTAL
Kaminak - Coffee Creek PEA Economic Model	**Unit**	0	365	365	365	365	365	365	365	365	365	365	365	365	365	0	0	0	**LOM Total**
Metal Price																			
Au	US $/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	**1,250**
Exchange Rate	CAD:USD	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	0.95	**0.95**
Supremo																			
Oxide Ore	tonnes	0	0	0	762,555	4,957,713	4,980,680	5,000,906	4,969,105	4,659,956	4,877,577	4,905,693	3,938,491	0	0	0	0	0	**39,052,676**
Oxide Grade	g/t	0.00	0.00	0.00	0.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.37	0.00	0.00	0.00	0.00	0.00	**1.13**
Contained Oxide Au	oz	0	0	0	16,993	130,496	211,103	188,984	172,669	149,102	166,227	205,243	172,920	0	0	0	0	0	**1,413,735**
Recovered Oxide Au	90%	0	0	0	15,293	117,446	189,992	170,086	155,402	134,192	149,604	184,719	155,628	0	0	0	0	0	**1,272,362**
Transition Ore	tonnes	0	0	0	0	42,044	19,294	0	31,523	332,158	95,530	94,670	573,819	0	0	0	0	0	**1,189,038**
Transition Grade	g/t	0.00	0.00	0.00	0.00	1.01	1.25	0.00	1.34	1.04	2.61	3.44	1.60	0.00	0.00	0.00	0.00	0.00	**1.64**
Contained Transition Au	oz	0	0	0	0	1,371	774	0	1,359	11,109	8,029	10,455	29,560	0	0	0	0	0	**62,658**
Recovered Transition Au	70%	0	0	0	0	960	542	0	951	7,777	5,620	7,319	20,692	0	0	0	0	0	**43,861**
Total Ore	tonnes	0	0	0	762,555	4,999,757	4,999,974	5,000,906	5,000,628	4,992,114	4,973,107	5,000,363	4,512,310	0	0	0	0	0	**40,241,714**
Waste	tonnes	0	0	0	5,113,178	19,393,776	24,996,285	21,980,534	21,797,143	19,454,304	20,000,858	21,336,538	16,602,912	0	0	0	0	0	**170,675,528**
Strip Ratio	w:o	0.00	0.00	0.00	6.71	3.88	5.00	4.40	4.36	3.90	4.02	4.27	3.68	0.00	0.00	0.00	0.00	0.00	**4.24**
Total Supremo Mined	**tonnes**	**0**	**0**	**0**	**5,875,733**	**24,393,533**	**29,996,259**	**26,981,440**	**26,797,771**	**24,446,418**	**24,973,965**	**26,336,901**	**21,115,222**	**0**	**0**	**0**	**0**	**0**	**210,917,242**
Latte																			
Oxide Ore	tonnes	851,724	2,383,292	4,729,750	2,571,192	0	0	0	0	0	0	0	0	0	0	0	0	0	**10,535,958**
Oxide Grade	g/t	1.09	1.13	1.42	1.23	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**1.28**
Contained Oxide Au	oz	29,769	86,471	215,947	101,613	0	0	0	0	0	0	0	0	0	0	0	0	0	**433,799**
Recovered Oxide Au	88%	26,196	76,094	190,034	89,419	0	0	0	0	0	0	0	0	0	0	0	0	0	**381,744**
Transition Ore	tonnes	0	16,682	258,956	449,526	0	0	0	0	0	0	0	0	0	0	0	0	0	**725,164**
Transition Grade	g/t	0.00	1.24	1.72	1.92	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**1.83**
Contained Transition Au	oz	0	664	14,290	27,684	0	0	0	0	0	0	0	0	0	0	0	0	0	**42,638**
Recovered Transition Au	44%	0	292	6,288	12,181	0	0	0	0	0	0	0	0	0	0	0	0	0	**18,761**
Total Ore	tonnes	851,724	2,399,974	4,988,706	3,020,718	0	0	0	0	0	0	0	0	0	0	0	0	0	**11,261,122**
Waste	tonnes	3,749,804	7,263,452	8,662,440	4,625,860	0	0	0	0	0	0	0	0	0	0	0	0	0	**24,301,556**
Strip Ratio	w:o	4.40	3.03	1.74	1.53	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**2.16**
Total Latte Mined	**tonnes**	4,601,528	9,663,426	13,651,146	7,646,578	0	0	0	0	0	0	0	0	0	0	0	0	0	**35,562,678**
Double Double																			
Oxide Ore	tonnes	0	0	10,745	1,204,141	0	0	0	0	0	0	0	0	0	0	0	0	0	**1,214,886**
Oxide Grade	g/t	0.00	0.00	5.05	3.30	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**3.32**
Contained Oxide Au	oz	0	0	1,743	127,915	0	0	0	0	0	0	0	0	0	0	0	0	0	**129,658**
Recovered Oxide Au	90%	0	0	1,569	115,123	0	0	0	0	0	0	0	0	0	0	0	0	0	**116,692**
Transition Ore	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**0**
Transition Grade	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**0.00**
Contained Transition Au	oz	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**0**
Recovered Transition Au	70%	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**0**
Total Ore	tonnes	0	0	10,745	1,204,141	0	0	0	0	0	0	0	0	0	0	0	0	0	**1,214,886**
Waste	tonnes	0	0	2,274,382	11,422,242	0	0	0	0	0	0	0	0	0	0	0	0	0	**13,696,624**
Strip Ratio	w:o	0.00	0.00	211.67	9.49	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**11.27**
Total Double Double Mined	**tonnes**	0	0	2,285,127	12,626,383	0	0	0	0	0	0	0	0	0	0	0	0	0	**14,911,510**
Kona																			
Oxide Ore	tonnes	0	0	0	0	0	0	0	0	0	0	0	730,006	0	0	0	0	0	**730,006**
Oxide Grade	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.22	0.00	0.00	0.00	0.00	0.00	**1.22**
Contained Oxide Au	oz	0	0	0	0	0	0	0	0	0	0	0	28,636	0	0	0	0	0	**28,636**
Recovered Oxide Au	90%	0	0	0	0	0	0	0	0	0	0	0	25,772	0	0	0	0	0	**25,772**
Transition Ore	tonnes	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**0**
Transition Grade	g/t	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	**0.00**
Contained Transition Au	oz	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**0**
Recovered Transition Au	70%	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**0**
Total Ore	tonnes	0	0	0	0	0	0	0	0	0	0	0	730,006	0	0	0	0	0	**730,006**
Waste	tonnes	0	0	0	0	0	0	0	0	0	0	0	3,699,916	0	0	0	0	0	**3,699,916**
Strip Ratio	w:o	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	5.07	0.00	0.00	0.00	0.00	0.00	**5.07**
Total Kona Mined	**tonnes**	0	0	0	0	0	0	0	0	0	0	0	4,429,922	0	0	0	0	0	**4,429,922**
Ore Mined to Crusher Stockpile																			
Oxide Ore	tonnes	851,724	2,383,292	4,740,495	4,537,888	4,957,713	4,980,680	5,000,906	4,969,105	4,659,956	4,877,577	4,905,693	4,668,497	0	0	0	0	0	**51,533,526**
Oxide Grade	g/t	1.09	1.13	1.43	1.69	0.82	1.32	1.18	1.08	1.00	1.06	1.30	1.34	0.00	0.00	0.00	0.00	0.00	**1.21**
Oxide Oz	oz	29,769	86,471	217,690	246,520	130,496	211,103	188,984	172,669	149,102	166,227	205,243	201,556	0	0	0	0	0	**2,005,829**
Oxide Recovered Oz	oz	26,196	76,094	191,602	219,836	117,446	189,992	170,086	155,402	134,192	149,604	184,719	181,400	0	0	0	0	0	**1,796,570**
Transition Ore	tonnes	0	16,682	258,956	449,526	42,044	19,294	0	31,523	332,158	95,530	94,670	573,819	0	0	0	0	0	**1,914,202**
Transition Grade	g/t	0.00	1.24	1.72	1.92	1.01	1.25	0.00	1.34	1.04	2.61	3.44	1.60	0.00	0.00	0.00	0.00	0.00	**1.71**
Transition Oz	oz	0	664	14,290	27,684	1,371	774	0	1,359	11,109	8,029	10,455	29,560	0	0	0	0	0	**105,296**
Transition Recovered Oz	oz	0	292	6,288	12,181	960	542	0	951	7,777	5,620	7,319	20,692	0	0	0	0	0	**62,621**
Total Tonnes	tonnes	851,724	2,399,974	4,999,451	4,987,414	4,999,757	4,999,974	5,000,906	5,000,628	4,992,114	4,973,107	5,000,363	5,242,316	0	0	0	0	0	**53,447,728**
Total Grade	g/t	1.09	1.13	1.44	1.71	0.82	1.32	1.18	1.08	1.00	1.09	1.34	1.37	0.00	0.00	0.00	0.00	0.00	**1.23**
Total Recovered Oz	oz	26,196	76,386	197,890	232,017	118,406	190,534	170,086	156,353	141,968	155,224	192,037	202,092	0	0	0	0	0	**1,859,191**
Beginning Stockpile Inventory																			
Total Tonnes	tonnes	851,724	3,251,698	6,251,149	5,587,414	5,587,171	5,587,145	5,588,051	5,588,679	5,580,793	5,553,900	5,554,263	5,796,579	796,579	-	-	-	-	
Total Recovered Au	oz	26,196	102,583	237,378	254,801	145,194	205,793	191,712	176,528	160,563	171,934	209,185	222,967	30,641	-	-	-	-	
From Crusher Stockpile to Heap Leach																			
Total Tonnes	tonnes	-	2,000,000	5,651,149	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	796,579	-	-	-	-	**53,447,728**
Total Recovered Au	oz	-	63,095	214,594	228,013	129,935	184,167	171,538	157,934	143,853	154,787	188,310	192,326	30,641	-	-	-	-	**1,859,191**
Closing Inventory																			
Total Tonnes	tonnes	851,724	1,251,698	600,000	587,414	587,171	587,145	588,051	588,679	580,793	553,900	554,263	796,579	-	-	-	-	-	
Total Recovered Au	oz	26,196	39,488	22,784	26,788	15,259	21,626	20,175	18,594	16,710	17,147	20,875	30,641	-	-	-	-	-	

		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	LOM TOTAL
Totals																			
Total Ore Mined	**tonnes**	851,724	2,399,974	4,999,451	4,987,414	4,999,757	4,999,974	5,000,906	5,000,628	4,992,114	4,973,107	5,000,363	5,242,316	0	0	0	0	0	**53,447,728**
Throughput Rate	tpd	2,333	6,575	13,697	13,664	13,698	13,699	13,701	13,700	13,677	13,625	13,700	14,363	0	0	0	0	0	**13,251**
Total Waste Mined	**tonnes**	3,749,804	7,263,452	10,936,822	21,161,280	19,393,776	24,996,285	21,980,534	21,797,143	19,454,304	20,000,858	21,336,538	20,302,828	0	0	0	0	0	**212,373,624**
Strip Ratio	**w:o**	4.40	3.03	2.19	4.24	3.88	5.00	4.40	4.36	3.90	4.02	4.27	3.87	0.00	0.00	0.00	0.00	0.00	**3.97**
Total Mined	tonnes	4,601,528	9,663,426	15,936,273	26,148,694	24,393,533	29,996,259	26,981,440	26,797,771	24,446,418	24,973,965	26,336,901	25,545,144	0	0	0	0	0	**265,821,352**
Rehandle	tonnes	0	0	651,698	12,586	243	26	0	0	7,886	26,893	0	0	796,579	0	0	0	0	**1,495,911**
Total Ore to Heap	tonnes	0	2,000,000	5,651,149	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	5,000,000	796,579	0	0	0	0	**53,447,728**
Au in Heap Inventory	oz	0	10,000	10,000	10,000	0	0	0	0	0	0	0	0	-30,000	0	0	0	0	**0**
Recovered Au	oz	0	53,095	204,594	218,013	129,935	184,167	171,538	157,934	143,853	154,787	188,310	192,326	60,641	0	0	0	0	**1,859,191**
Payable Metal																			
Au Payable	%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	**100%**
	oz	0	26,547	231,141	218,013	129,935	184,167	171,538	157,934	143,853	154,787	188,310	192,326	60,641	0	0	0	0	**1,859,191**
Payable Au Value USD	US $	0	33,184,352	288,926,723	272,516,553	162,418,422	230,208,196	214,421,949	197,416,937	179,816,185	193,483,603	235,387,514	240,407,863	75,800,771	0	0	0	0	**2,323,989,067**
Payable Au Value CAD	C $	0	34,930,897	304,133,392	286,859,529	170,966,760	242,324,416	225,707,314	207,807,302	189,280,195	203,666,950	247,776,331	253,060,909	79,790,285	0	0	0	0	**2,446,304,281**
Refining Costs	US$/payable oz	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	7.50	**7.50**
	C $	0	209,585	1,824,800	1,721,157	1,025,801	1,453,946	1,354,244	1,246,844	1,135,681	1,222,002	1,486,658	1,518,365	478,742	0	0	0	0	**14,677,826**
Royalties	C $	0	347,213	5,023,086	2,851,384	1,699,410	2,408,705	2,243,531	2,065,605	1,881,445	2,024,449	2,462,897	2,515,425	793,115	0	0	0	0	**26,316,265**
	C $			2,000,000															**2,000,000**
Net Smelter Return (NSR) after Royalties	**C $**	**0**	**34,374,098**	**297,285,506**	**282,286,988**	**168,241,549**	**238,461,765**	**222,109,540**	**204,494,853**	**186,263,069**	**200,420,499**	**243,826,776**	**249,027,118**	**78,518,428**	**0**	**0**	**0**	**0**	**2,405,310,190**
OPEX																			
Mining & Rehandle	C $/tonne mined	4.58	3.03	2.73	2.39	2.44	2.40	2.52	2.35	2.41	2.47	2.36	2.26	0.00	0.00	0.00	0.00	0.00	**2.43**
	C $			43,864,717	62,594,974	59,548,659	72,082,409	67,977,721	63,083,025	58,860,490	61,659,692	62,238,744	57,719,634	398,290	0	0	0	0	**610,028,354**
Processing	C $/tonne ore	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	6.67	**6.67**
	C $			37,693,164	33,350,000	33,350,000	33,350,000	33,350,000	33,350,000	33,350,000	33,350,000	33,350,000	33,350,000	5,313,182	0	0	0	0	**343,156,346**
G&A	C $/tonne ore	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	**4.00**
	C $			22,604,596	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	20,000,000	3,186,316	0	0	0	0	**205,790,912**
Total Opex	**C $**	**0**	**0**	**104,162,477**	**115,944,974**	**112,898,659**	**125,432,409**	**121,327,721**	**116,433,025**	**112,210,490**	**115,009,692**	**115,588,744**	**111,069,634**	**8,897,787**	**0**	**0**	**0**	**0**	**1,158,975,611**
	C $/payable oz	**0**	**0**	**451**	**532**	**869**	**681**	**707**	**737**	**780**	**743**	**614**	**578**	**147**	**0**	**0**	**0**	**0**	**623**
Net Operating Cashflow	**C $**	**-**	**34,374,098**	**193,123,029**	**166,342,014**	**55,342,891**	**113,029,356**	**100,781,819**	**88,061,829**	**74,052,579**	**85,410,807**	**128,238,032**	**137,957,484**	**69,620,641**	**-**	**0**	**0**	**0**	**1,246,334,579**
CAPEX																			
Capitalized Mining	C $	21,061,248	29,266,629	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**50,327,877**
Pre-Production Operating Costs	C $	-	16,700,000			-	-	-	-	-	-	-	-	-	-	-	-	-	**16,700,000**
Site	C $	43,608,000	13,526,000	4,806,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**61,940,000**
Mining Equipment	C $	41,240,132	5,042,211	18,200,368	21,338,211	-	9,287,526	-	9,488,684	6,216,000	-	-	-	-	-	-	-	-	**110,813,132**
Leach Facility	C $	28,726,903	27,355,324	-	7,830,837	-	5,163,626	-	4,191,428	-	-	-	-	-	-	-	-	-	**73,268,118**
Camp	C $	5,499,200	4,798,800	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**10,298,000**
Indirects	C $	22,904,378	14,062,034	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**36,966,412**
Closure	C $	-	-	-	-	-	-	-	-	-	-	-	-	8,000,000	8,000,000	8,000,000	8,000,000	8,000,000	**40,000,000**
Subtotal	**C $**	**163,039,860**	**110,750,997**	**23,006,368**	**29,169,047**	**-**	**14,451,153**	**-**	**13,680,113**	**6,216,000**	**-**	**-**	**-**	**8,000,000**	**8,000,000**	**8,000,000**	**8,000,000**	**8,000,000**	**400,313,538**
Contingency	C $	21,296,792	9,717,655	3,450,955	4,375,357	-	2,167,673	-	2,052,017	932,400	-	-	-	1,200,000	1,200,000	1,200,000	1,200,000	1,200,000	**49,992,849**
CAPEX incl. Contingency	**C $**	**184,336,652**	**120,468,652**	**26,457,324**	**33,544,404**	**-**	**16,618,826**	**-**	**15,732,130**	**7,148,400**	**-**	**-**	**-**	**9,200,000**	**9,200,000**	**9,200,000**	**9,200,000**	**9,200,000**	**450,306,388**
Pre-Production	*C $*	*184,336,652*	*120,468,652*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	*0*	***304,805,304***
Sustaining & Closure	*C $*	*0*	*0*	*26,457,324*	*33,544,404*	*0*	*16,618,826*	*0*	*15,732,130*	*7148400*	*0*	*0*	-	*9200000*	*9200000*	*9200000*	*9200000*	*9200000*	***145,501,083***
Net Cashflow	**C $**	**-184,336,652**	**-86,094,554**	**166,665,705**	**132,797,610**	**55,342,891**	**96,410,531**	**100,781,819**	**72,329,699**	**66,904,179**	**85,410,807**	**128,238,032**	**137,957,484**	**60,420,641**	**-9,200,000**	**-9,200,000**	**-9,200,000**	**-9,200,000**	**796,028,191**
Cumulative Net Cashflow	C $	-184,336,652	-270,431,206	-103,765,500	29,032,110	84,375,000	180,785,531	281,567,350	353,897,049	420,801,228	506,212,035	634,450,067	772,407,551	832,828,191	823,628,191	814,428,191	805,228,191	796,028,191	
Pre-Tax Payback Period	**Years**																		**1.8**
Pre-Tax IRR	**%**																		**32.8%**
Pre-Tax NPV$_{5\%}$	**C $**																		**$522,440,799**
Income Taxes	**C $**	**0**	**2,029,656**	**16,459,400**	**12,399,121**	**180,125**	**23,269,159**	**33,877,696**	**24,434,393**	**23,143,356**	**29,781,320**	**46,821,860**	**51,006,245**	**24,180,236**	**-3,272,088**	**-3,143,771**	**-346,803**	**0**	**280,819,905**
After-Tax Cash Flow	**$**	**-184,336,652**	**-88,124,210**	**150,206,306**	**120,398,489**	**55,162,765**	**73,141,372**	**66,904,122**	**47,895,306**	**43,760,823**	**55,629,487**	**81,416,172**	**86,951,239**	**36,240,404**	**-5,927,912**	**-6,056,229**	**-8,853,197**	**-9,200,000**	**515,208,286**
Cumulative After-Tax Cash Flow	$	-184,336,652	-272,460,862	-122,254,557	-1,856,067	53,306,698	126,448,070	193,352,193	241,247,499	285,008,322	340,637,809	422,053,981	509,005,220	545,245,624	539,317,712	533,261,483	524,408,286	515,208,286	
After-Tax Payback Period	**Years**																		**2.0**
After-Tax IRR	**%**																		**26.2%**
After-Tax NPV$_{5\%}$	**C $**																		**$330,439,435**