Independent Technical Report and Preliminary Economic Assessment Kilgore Project Clark County, Idaho, USA

Prepared for:

OTIS GOLD CORP

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ROWEARTH

Effective Date:
July 30, 2019
Issued Date:
August 26, 2019

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Kilgore Project

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF PHOTOS

1.0 SUMMARY

Otis Gold Corp. (Otis Gold) engaged Global Resource Engineering, Ltd. (GRE), the “Authors”, to perform a Preliminary Economic Assessment (PEA) of the Kilgore Project (“Kilgore Project” or the “project”), Clark County, Idaho, U.S.A.

This report builds on the independent estimation of the mineral resources of the Kilgore deposit as of August 14, 2018 (GRE & Rowearth, 2018). The PEA and this report were prepared according to the guidelines of Form 43-101F1, and Companion Policy 43-101CP, as amended by the Canadian Securities Administrators and enacted on June 30, 2011 (together, the “Instrument”).

David Rowe, Terre Lane, Jeffrey Todd Harvey and J.J. Brown are Qualified Persons under the Instrument. Mr. Rowe and J.J. Brown conducted independent site visits to the Kilgore property on August 9th through 14th, 2017 and August 4th through 5th, 2018, respectively. The conclusions and recommendations in this report are based on information available as of March 31, 2019.

1.1 Location and Property History

The Kilgore deposit is part of Otis Gold’s Kilgore Project, a volcanic-and sediment-hosted epithermal gold property located on the northern margin of the eastern Snake River Plain, approximately 5 miles west-northwest of the small rural hamlet of Kilgore, Clark County, Idaho (Figure 4-1). Otis Gold has a 100% undivided interest in 614 unpatented Federal lode claims totaling 12,150 acres (4,917 hectares) on U.S. Forest Service lands. The area is mountainous; relief on the Kilgore property is 2,000 feet (610 meters) at elevations above 6,400 feet (1,950 meters).

The property’s initial discovery and earliest known gold exploration and production work was reported to have been in the 1930s by the Blue Ledge Mining Company. Evidence of Blue Ledge’s activity remains as several collapsed underground adits, prospect pits, a tram car, and an old foundation, though there is no record of gold production from this period. Six different companies have explored the Kilgore property in modern times, beginning with Bear Creek Mining in 1983, followed by Placer Dome U.S., Pegasus, Echo Bay Exploration (Echo Bay), Latitude Minerals, Kilgore Minerals Ltd., and Otis Gold. Each of the companies conducted one or more campaigns of drilling that presently total 296,246 feet.

The property has excellent three-season access via County and Forest Service gravel roads immediately east of the project, and currently has limited access during winter. Primary electric distribution power is available in Camas Creek Valley south and east of the project and via high voltage transmission lines located in Pleasant Valley 10 miles west of the project. The project is located within the Mud Lake closed basin, runoff within the basin is used for irrigation or it flows to Mud Lake where evaporation and ground water infiltration occur.

1.2 Geology and Mineralization

The Kilgore Project is located in the northeastern portion of the Eastern Snake River Plain (ESRP), locally situated to the south of the Centennial Mountains and regionally along the northern margin of the Miocene-Pliocene Heise Volcanic Field.

The Project is hosted within the northern margin of a Miocene to Pliocene caldera volcanic complex with associated intrusive, extrusive and pyroclastic sequences known as the Heise Volcanic Field, which in turn unconformably overly Cretaceous arkosic sediments. Specifically, the project occurs on the northeastern rim of the Kilgore Caldera, one of four eruptive events that make up the Heise Volcanic Field. To both the north and south, the volcanic rocks are locally blanketed by the tuff of Kilgore, a relatively distinct welded ash flow tuff thought to represent the last major eruptive event of the Kilgore caldera and dated at about 4.45 (±0.05) million years (Morgan, 2005).

The Kilgore deposit is interpreted as being a low sulfidation (LS) epithermal deposit associated with caldera-related volcanic and intrusive activity. The current known resource area is a zone of mineralization approximately 800 meters long, 600 meters wide, and 325 meters deep from ground surface to the maximum inferred mineral resource depth. Mineralized intercepts generally average 40 meters (130 feet) and range up to 100 meters (330 feet) in thickness in the Mine Ridge core and North Target areas. Near surface gold mineralization occurs primarily in rocks of volcanic or subvolcanic origin, including the Tertiary lithic tuff (Tlt) and sub-vertical felsic dikes, dike swarms, and granodioritic bodies that intrude it. These are underlain by sedimentary rocks of the Aspen Formation (Ka), which comprise an additional host of mineralization, one which is characterized by sediment-hosted, low-moderate grade, bulk-mineable type distribution.

Gold mineralization in the volcanic and related intrusive rocks is moderate to high grade as a result of weak to moderate vein development and open space fracture-fill, together within a broad, low grade halo of disseminated gold within variably silicified and argillically altered rocks. Gold content appears to decrease rapidly to lower grades (less than 50 to 100 parts per billion [ppb] gold [Au]) with corresponding decrease in quartz or quartz-adularia alteration and increase in argillic alteration. Exceptions occur in strongly oxidized rock near the topographic surface, where strong to pervasive iron-oxide, yellow-orange to brown staining, is accompanied by higher gold grades. Mineralization in the volcanic and associated intrusive rocks accounts for an estimated 85% of the known mineral resource, with the remaining 15% occurring in the underlying Aspen Formation.

1.3 Exploration and Drilling

Seven companies since 1983 have explored Kilgore with drilling comprising 152 reverse circulation (RC) holes and 229 core holes totaling 306,541 feet (93,434 meters). One of the drill holes is PQ-size for metallurgical tests. Quality Assurance/Quality Control (QA/QC) information is not available for drilling campaigns before Echo Bay in 1994, but drill hole logs and assay certificates are available in Otis Gold's offices, and all assays were performed at commercial laboratories. Echo Bay conducted studies of metallic screen assays, included in-house standards in its sample submissions, and obtained check assay information from a second laboratory.

Otis Gold drilling has been completed during the period of 2008 to 2018. Otis Gold included standards and blanks with its submissions, collected core half pair data, and submitted pulps prepared at the primary laboratory for check assays to secondary commercial laboratories. The programs point to some bias in the primary lab, ALS Global, versus the check labs, but the commercial standards analyzed display no variation of concern from the certified values. Drilling exploration carried out by Otis Gold from 2012 through 2018 consists of 45 reverse circulation (RC) holes and 45 diamond core holes (three of which were drilled for metallurgical testing) for a total of 22,536 meters drilled.

RC and core assays do not compare well with paired data comparisons and separate estimates showing that RC assay samples are generally higher than core. Issues are identified with both types of data, which will not be fully resolved without collecting bulk samples. The various operators of the project have been alerted to recovery and sampling issues and appear to have taken measures to reduce sample bias, reflected in the core drilling techniques used. A bulk sample testing program should be designed, including potential development of an underground bulk sample operation.

In November 2016, Otis Gold contracted Justin Modroo, P.G., to conduct a ground based geophysical magnetic survey in the vicinity of the primary Kilgore resource area (Modroo J. , 2017). The survey was designed to test magnetic signatures surrounding the known deposit to better define local structural characteristics and potentially identify future drilling exploration targets. In 2018, Otis Gold conducted further regional-style exploration, including surface mapping and sampling and soil and stream sediment surveys, with the goal of identifying future drilling targets.

1.4 Metallurgical Testing

A significant amount of metallurgical test work has been conducted on the Kilgore project dating back to 1995. Echo Bay had commissioned the early test work focusing mainly on evaluating the deposit for heap leach treatment. Otis Gold has furthered these investigations with additional, more detailed testing including mineralogy, direct cyanide leach tests, more detailed heap leach testing, and physical material characteristic definition.

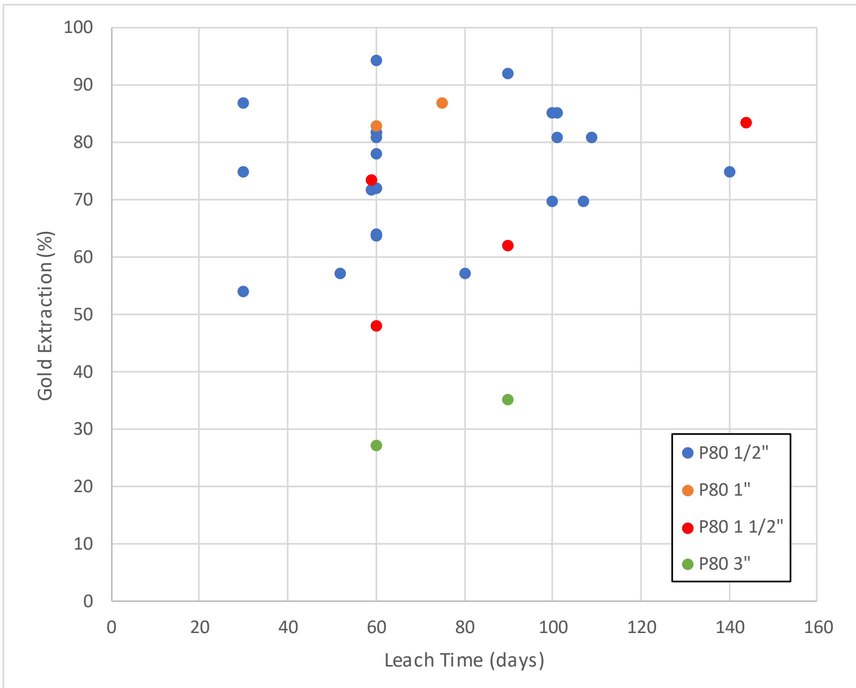
The Kilgore deposit is best characterized as an oxide deposit with three main oxidation states: oxidized, an underlying unoxidized portion, and a mixed or transition type portion lying between these two zones. There are no significant sulfides present in the deposit. A wide range of metallurgical samples have been tested from all zones of the deposit, and the general indication is that it responds very well to heap leaching. There is a deeper, higher-grade zone within the deposit identified as Aspen that contains some carbonaceous material that showed significant “pregnant solution-robbing” tendencies. Subsequent testing indicated that a carbon-based leaching system such as CIL or CIP will overcome negative impact of the natural carbon.

Column leach testing, used as an analog for heap leach testing, has shown excellent results for all material types with recoveries ranging from 64% to 94% in a relatively short leach time of 60 days. Gold grades tested have ranged from 0.03 to 0.055 ounces per ton (opt). P_{80} crush sizes of ½ inch, 1 inch, 1 ½ inches and 3 inches (still underway) have been tested. The results indicate that the overall deposit is moderately sensitive to crush size, with some areas showing potential run-of-mine leachability and others requiring a finer crush size to maintain high recovery and extraction kinetics. Cyanide and lime consumptions have shown wide ranges but there has not been any focus on optimization. Cyanide consumptions range from 0.6 to 5 pounds per ton (lb/t), averaging approximately 2.2 lb/t. Lime has shown a similar variability, ranging from 1.0 to 4.0 lb/t and averaging 2.2 lb/t. Actual cyanide consumption will typically be in the range of 30% of that exhibited in column leach testing.

Crushing work index testing indicates the majority of the deposit is characterized as moderately hard, ranging from 10.2 to 11.7 kilowatt-hours per short ton (kwh/st). Abrasion index testing indicates that some portions of the deposit may be abrasive, with the index ranging from 0.15 to 0.30.

Figure 1-1 shows the summary of all the gold extractions for all column leach tests to date. Both the mid-point (generally 60 days) and final leach period have been plotted.

Figure 1-1: Summary of Column Leach Gold Extractions by Leach Time and Crush Size



The design basis for the heap leach includes crushing to a P_{80} size of ½-inch with a 90-day primary leach cycle. Gold and silver extractions are estimated at 80% and 40%, respectively. Cyanide and lime consumptions of 0.65 and 2.2 lb/t have been assumed, respectively.

As shown by Figure 1-1, the ore shows a fairly high degree of variability at coarser crush sizes but less at the finer crush size of P_{80} ½-inch. Gold extractions tend to be excellent in most cases at periods of 90 days or more for the ½-inch material.

1.5 Mineral Resources

1.5.1 Historic

An NI 43-101 Technical Report by Donald E. Cameron (2012) reviewed historic work and, with the addition of Otis Gold's exploration programs from 2008 through 2011, produced an estimate of mineral resources (Table 1-1):

Table 1-1: Historical Mineral Resources for Kilgore Property (Cameron, 2012)

Resource Category	Metric Tons (T)	Au (g/T)	Au Ounces (Troy)	Short Tons (t)	Au (opt)
Measured	-	-	-	-	-
Indicated	27,352,000	0.59	520,000	30,130,000	0.017
Total Measured and Indicated	27,352,000	0.59	520,000	30,130,000	0.017
Inferred	20,230,000	0.46	300,000	22,290,000	0.014

Mineral resources are at a gold cutoff grade of 0.24 g/T (0.007 opt).
Items are rounded off to reflect the precision of the estimate, thus metal quantity varies slightly from the product of tons and grade.

The information in Table 1-1 is presented as historical information and does not represent the current estimate.

1.5.2 Current

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as: *"a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."* The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. Mineral resources are not mineral reserves and do not have demonstrated economic viability. Mineral reserves can only be estimated based on the results of an economic evaluation as part of a Preliminary Feasibility Study or Feasibility Study. Therefore, no mineral reserves have been estimated as part of this study. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

The requirement of "reasonable prospects for eventual economic extraction" generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at a cutoff grade considering appropriate extraction scenarios and processing recoveries. David Rowe of Rowearth LLC modeled the geology using Leapfrog Geo three-dimensional (3D) modeling software and then created the resource block model using Leapfrog Edge. Rowearth considered that major portions of the Kilgore deposit are amenable for open pit extraction.

To determine the quantities of material offering "reasonable prospects for economic extraction" by an open pit, GRE constructed open pit scenarios developed from the resource block model estimate using Vulcan's Lerchs-Grossman miner "Pit Optimizer" software. Reasonable mining assumptions were applied to evaluate the portions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit. The optimization parameters presented in Table 1-2 were selected based on experience and benchmarking against similar projects. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cutoff grade. Rowearth considers that the blocks located within the resulting conceptual pit envelope show “reasonable prospects for economic extraction” and can be reported as a mineral resource.

The reader is cautioned that the results from the pit optimization are used solely for testing the “reasonable prospects for eventual economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are presently no mineral reserves on the project.

Table 1-2: Kilgore Resource Parameters for Conceptual Open Pit Optimization

Parameter	Unit	Values
Metal Price	US$/oz gold	$1,300.00
Selling cost	US$/oz gold	$2.20
Gold Recovery	%	80.00%
Mining cost	US$/short ton	$2.00
Process cost	US$/short ton includes $1.00 G&A	$4.00
Pit slope	degrees	50

The Kilgore gold resources are reported in Table 1-3. Table 1-4 shows the sensitivity of the resources to cut-off grade.

Table 1-3: Mineral Resource Statement for the Kilgore Deposit

	Imperial Units			Metric Units			
Category	Cutoff (Au opt)	Short tons	Au Grade (opt)	Cutoff (Au g/T)	Metric Tonnes	Au Grade (g/T)	Au Ounces
Indicated	0.006	49,106,000	0.017	0.21	44,556,000	0.58	825,000
Inferred	0.006	10,354,700	0.013	0.21	9,399,000	0.45	136,000

Mineral resources have been classified in accordance with the CIM Definition Standards on Mineral Resources
Gold resources are reported above a 0.21 g/T Au (0.006 opt) cutoff
Mineral resources reported here are constrained within an optimized pit shell.
Pit shell input parameters: Gold price $1,300, Selling price $2.20/oz, Recovery 80%, Mining cost $2/ton, Process cost + G&A $4/ton, Pit slope 50°

Table 1-4: Kilgore Project Resource Statement – Sensitivity to Cut-off Grade

	Imperial Units			Metric Units			
Classification	Cut-off (Au opt)	Short tons	Au Grade (opt)	Cut-off (Au g/t)	Metric Tonnes	Au Grade (g/t)	Au Ounces
Indicated	0.003	62,382,000	0.014	0.10	56,592,000	0.49	886,000
	0.004	58,647,000	0.015	0.14	53,206,000	0.51	873,000
	0.005	53,976,000	0.016	0.17	48,966,000	0.54	852,000
	0.006	**49,106,000**	**0.017**	**0.21**	**44,556,000**	**0.58**	**825,000**
	0.007	44,549,000	0.018	0.24	40,414,000	0.61	796,000
	0.008	40,294,000	0.019	0.27	36,559,000	0.64	764,000
	0.009	36,343,000	0.020	0.31	32,970,000	0.69	730,000
	0.010	32,830,000	0.021	0.34	29,786,000	0.73	697,000

Classification	Imperial Units			Metric Units			Au Ounces
	Cut-off (Au opt)	Short tons	Au Grade (opt)	Cut-off (Au g/t)	Metric Tonnes	Au Grade (g/t)	
Inferred	0.003	16,271,700	0.010	0.10	14,761,000	0.34	163,000
	0.004	14,511,400	0.011	0.14	13,168,000	0.37	157,000
	0.005	12,336,900	0.012	0.17	11,192,000	0.41	147,000
	0.006	**10,354,700**	**0.013**	**0.21**	**9,399,000**	**0.45**	**136,000**
	0.007	8,736,180	0.014	0.24	7,925,000	0.49	126,000
	0.008	7,272,060	0.016	0.27	6,600,000	0.54	115,000
	0.009	6,017,710	0.017	0.31	5,459,000	0.59	104,000
	0.010	5,030,820	0.019	0.34	4,567,000	0.65	95,000

Au block model metal quantities reported at various Au cut-off grades for the Kilgore deposit.

1.6 Mining Methods

For the purposes of this technical report, GRE assumed conventional open pit mining methods using drill, blast, load, and haul mining would be most applicable to the Kilgore deposit.

The PEA evaluated pits that were generated excluding the Aspen material and pits that were generated including the Aspen material. Based on review of metallurgical test work of the Aspen Formation, which shows variable recovery for different zones, GRE assumed 60% of the Aspen material would be leachable, while the other 40% was considered waste. Similarly, 60% of the reported gold ounces within the Aspen rock type were included as contained ounces and the other 40% were considered waste. GRE recommends additional geologic analysis of the Aspen Formation to identify and model the good-performing metallurgical domain relative to the portion that requires CIL or CIP processing efforts.

Four ultimate pit designs were created for the following Lerchs-Grossman pit shells:

- $800/oz Au no Aspen included in pit
- $800/oz Au Aspen included in pit
- $900/oz Au no Aspen included in pit
- $900/oz Au Aspen included in pit

GRE evaluated two leaching options for each of the ultimate pit designs:

- crushing of all material above the mining cutoff grade
- a combination of run-of-mine (ROM) and crushing:
  - ROM of all material above a cutoff grade of 0.004 opt (0.14 g/T) up to the mining cutover grade
  - crushing of all material above the mining cutover grade.

Each pit was evaluated at five mining cutoff grades: 0.006 opt (0.2057 g/T), 0.007 opt (0.24 g/T), 0.008 opt (0.274 g/T), and 0.009 opt (0.309 g/T), and 0.010 opt (0.343 g/T).

The Kilgore deposit includes potentially recoverable silver content as well as gold. Silver assays were conducted on approximately 3,906 samples; however, they were not incorporated into the block model, and the Mineral Resource Estimate in Section 14 did not address silver. Based on the available data and metallurgical test results, silver will be recovered with the gold and will contribute to the economic value of the project. GRE recommends re-assaying all available pulps for silver prior to the next engineering phase and incorporating silver into the Mineral Resource and Economic Model.

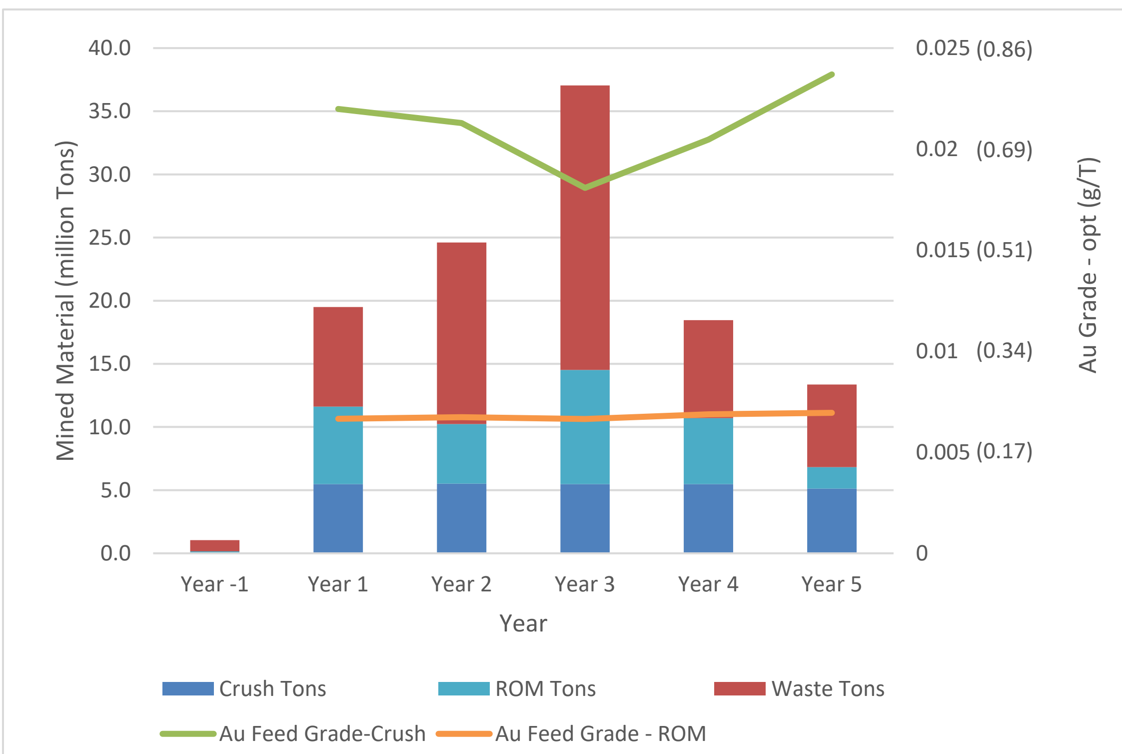
GRE selected the $800/oz Au pit with Aspen included at a cutoff grade of 0.010 opt (0.34 g/T) with a ROM cutoff grade of 0.004 opt (0.14 g/T) and crushing as the base case. This pit has life-of-mine leach material of 54.0 million tons at a strip ratio of 1.1:1.

The mine schedule was broken into three phases of mining. GRE used the following assumptions to generate the schedule:

- Leachable/Crush Material Production Rate: 15,000 tons per day (tpd)
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 10

These assumptions result in a mine life of 5.0 years (Figure 1-2).

Figure 1-2: Mine Production Summary



The assumed mine layout would include a crushing site, leach pad, plant site, and waste dump. The leach material and waste would be drilled and blasted using a rotary crawl driller and ammonium nitrate fuel oil (ANFO) and transported in dump trucks to the primary crusher, which would be located near the pit.

Crushed leach material would be transported to the secondary and tertiary crushing circuits by conveyor and then to the leach pad by conveyor. The analysis uses Caterpillar 777G size trucks, with a heaped capacity of 84 cubic yards, and Caterpillar 992K size loaders, with a bucket capacity of 16 cubic yards.

1.7 Recovery Methods

A conventional heap leach process has been proposed for the Kilgore deposit. Depending on the grade of the material, it will either be crushed to a P_{80} of -1/2-inch prior to being placed on the heap leach or treated directly as ROM. Both ROM and crushed material would have lime added prior to pad placement for pH control. The ROM material would be trucked dumped on the pad and ripped with a dozer after each lift is complete. The crushed material would be conveyed to the heap leach facility (HLF). This pregnant leach solution from the heap leach would be collected in a dedicated pond and either recirculated or processed in the Adsorption-Desorption-Recovery plant (ADR). The gold and silver in the solution would be collected on activated carbon in a series of carbon-in-column (CIC) vessels. Gold and silver recovery would take place through stripping the activated carbon into an enriched solution that reports to an electrowinning circuit where the gold and silver is recovered as a sludge that is ultimately smelted into high purity doré bars.

1.8 Environmental Studies and Permitting

A Golder Associates Preliminary Environmental Report (2010) prepared for Otis Gold provided an overview of studies and permits that will be required to develop the Kilgore Project. The report stated that issues may arise during studies and permitting, but the information available at the time of the report did not identify a fatal flaw. Work on the Kilgore Project is subject to annual United States Forest Service (USFS) Plans of Operation (PoO) that must be submitted in advance. A PoO was approved for exploration work in 2012.

In August, 2017, a Plan of Operation that was prepared for Otis Gold by Klepfer Mining Services and was submitted to the United States Forest Services (USFS) , the lead agency having jurisdiction over the Kilgore Project. The PoO allows for 140 exploration drill sites, up to 420 exploration drill holes and associated drill access roads. The PoO was approved by the USFS in August, 2018.

A coalition of environmental groups lead by the Idaho Conservation League filed an administrative complaint against the USFS in November, 2018 citing failings within the Environmental Assessment (EA) in-lieu of a more comprehensive Environmental Impact Statement (EIS). The administrative legal procedures are currently underway and Otis Gold expects a ruling potentially as soon as December, 2019.

1.9 Capital and Operating Costs

1.9.1 Capital Costs

Capital Costs include:

- The project plans to use contractor mining, and all mining equipment and facilities would be provided by the contractor, so no capital costs are included in the cost estimates for mining equipment and facilities.

- Process capital costs include an ADR recovery system, leach pad, ponds, crushing plant and stacking system, a laboratory, and mobile equipment.
- Development includes pioneering, clearing, grubbing, access road improvements, and haul road construction, assumed to be 20,000 feet of new haul roads.
- Working capital was estimated to be 2 months' operating costs. The working capital was estimated to be recovered the year after production ends. Capital contingency was set at 25%.

The total estimated capital costs for the project, including contingency, are $97.46 million, with initial capital costs of $81.23 million, as shown in Table 1-5.

Table 1-5: Summary of Kilgore Capital Costs (millions)

Capital Cost Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mine	$3.90	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$3.90
Process	$41.12	$0.00	$0.00	$7.87	$0.00	$6.03	$0.00	$0.00	$0.00	$55.03
G&A	$7.21	$1.52	$1.52	$1.52	$1.52	$1.42	$5.00	$0.00	$0.00	$19.71
Sustaining	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Working	$15.46	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	($15.46)	$0.00
Contingency	$13.54	$0.30	$0.30	$1.88	$0.30	$1.49	$1.00	$0.00	$0.00	$18.82
Total	**$81.23**	**$1.52**	**$1.52**	**$9.39**	**$1.52**	**$7.45**	**$5.00**	**$0.00**	**($15.46)**	**$97.46**

1.9.2 Operating Costs

Operating costs include:

- Operating costs for mining production and support equipment were included, as well as recovery of capital costs by the contractor every four years.
- Process operating costs include operation of the ADR plant, leach pad, and crushing plant.
- Manpower includes personnel for mining, process, and overhead.
- Overhead services and supplies were also included.

The total life of mine operating costs were estimated to be $435.57 million, with a mining unit cost of $2.32/mined ton, and process unit cost of $2.90/process ton, and a general and administrative unit cost of $0.51/process ton, as shown in Table 1-6 and Table 1-7.

Table 1-6: Summary of Kilgore Estimated Operating Costs (millions)

Operating Cost Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mine	$1.95	$43.71	$47.14	$81.65	$54.86	$35.68	$264.99
Process	$0.08	$33.12	$30.41	$37.90	$31.31	$24.01	$156.83
G&A	$0.15	$2.76	$2.76	$2.76	$2.76	$2.58	$13.75
Total	**$2.18**	**$79.59**	**$80.31**	**$122.31**	**$88.93**	**$62.26**	**$435.57**

Table 1-7: Summary of Kilgore Estimated Operating Unit Costs

Item		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mine	($/mined ton)	$1.86	$2.24	$1.92	$2.20	$2.97	$2.67	$2.32
Process	($/process ton)	$3.73	$2.85	$2.97	$2.61	$2.92	$3.52	$2.90
G&A	($/mined ton)		$0.47	$0.54	$0.38	$0.51	$0.76	$0.51

The cash operating costs per ounce of Au, all-in sustaining cost per ounce of Au, and all-in cost per ounce of Au are shown in Table 1-8.

Table 1-8: Costs per Gold Ounce

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Cash Operating Cost / Au ounce	$665	$720	$1,099	$809	$585	$780
All-in Sustaining Cost / Au ounce	$681	$737	$1,201	$826	$669	$832
All-in Cost / Au ounce	$681	$737	$1,201	$826	$669	$954

Cash Operating Cost / Au ounce = operating costs/Au ounces
All-in Sustaining Cost / Au ounce = capital costs less initial capital/Au ounces
All-in Cost / Au ounce = (capital costs + operating costs)/Au ounces

1.10 Economic Model

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under National Instrument 43-101. This PEA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under National Instrument 43-101. Readers are advised that there is no certainty that the results projected in this Preliminary Economic Assessment will be realized.

The economic model uses a gold price of $1,300/ounce, which is consistent with the three-year trailing average price through the end of July 2019.

The projected metal recovery rate was 82% for crushed leach material and 50% for ROM material. Recovery was factored as 70% during the first year after placement on the leach pad, 25% during the second year after placement on the leach pad, and 5% during the third year after placement on the leach pad. The resulting base case gold revenue was $726.36 million (Table 1-9).

Table 1-9: Kilgore Project Revenues (millions)

Revenue	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Gold Crush	$0.00	$89.84	$119.07	$111.35	$116.24	$126.68	$38.67	$6.54	$608.39
Gold ROM	$0.00	$16.28	$20.54	$32.36	$27.83	$15.54	$4.65	$0.77	$117.97

GRE included depreciation and depletion deductions from the income before taxes to obtain taxable income. Federal tax at 21% was applied to the taxable income, Idaho corporate tax at 6.925% was applied to the taxable income, Idaho License tax at 1% was applied to net revenue, and Idaho property tax at 0.78% was applied to annual net profit. The taxes were deducted from the taxable income, then the depreciation and depletion allowance were added back from taxable income to obtain net cash flows after taxes. After-tax net present value (NPV) @5%, NPV@7%, NPV@9%, and internal rate of return (IRR) were calculated from the net after-tax cash flow. The total after-tax cash flow over the life of the project was $151.82 million (Table 1-10).

Table 1-10: Kilgore Project Summary of Economic Model

Description	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Net Revenue	$0.00	$106.12	$139.60	$143.72	$144.07	$142.22	$43.31	$7.31	$0.00	$726.36
Total Operating Costs	$2.18	$79.59	$80.31	$122.31	$88.93	$62.26	$0.00	$0.00	$0.00	$435.57
Before Tax Cash Flow	($2.18)	$26.54	$59.29	$21.41	$55.15	$79.96	$43.31	$7.31	$0.00	$290.79
Depreciation	$0.00	$12.75	$13.52	$13.88	$16.14	$16.50	$5.14	$0.00	$0.00	$77.93
Loss Carry Forward	$0.00	($2.18)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Depletion Allowance	$0.00	$6.89	$20.94	$3.76	$19.51	$21.33	$6.50	$1.10	$0.00	$80.03
Taxable Income	$0.00	$4.71	$24.83	$3.76	$19.51	$42.12	$31.68	$6.21	$0.00	$132.83
Total Taxes	$0.00	$1.72	$7.78	$1.39	$6.23	$12.97	$9.55	$1.85	$0.00	$41.51
After Tax Cash Flow	($2.18)	$24.82	$51.51	$20.02	$48.92	$66.99	$33.76	$5.45	$0.00	$249.28
Total Capital Costs	$81.23	$1.82	$1.82	$11.27	$1.82	$8.94	$6.00	$0.00	($15.46)	$97.46
Net Cash Flow	($83.41)	$22.99	$49.69	$8.74	$47.09	$58.04	$27.76	$5.45	$15.46	$151.82

The resulting NPV@5% was $110.39 million, the NPV@7% was $96.82 million, the NPV@9% was $84.64 million, the cash flow was $151.82 million, the IRR was 34.0%, and the payback period was 3.0 years.

A summary of the production includes:

Total leach tons mined:	54.0 million
Total waste tons mined:	60.0 million
Head Grade:	0.014 opt (0.48 g/T)
Mine Life:	5.0 years
Tons per day mined:	15,000
Strip ratio:	1.1
Gold recovery – Crushed Material:	82%
Gold recovery – ROM Material:	50%
Total gold ounces mined:	752,200
Total gold ounces recovered:	558,700
Average annual gold production:	111,700 ounces
Peak annual gold production:	119,600 ounces in year 1

Key economic measurements include the following:

Royalties:	0%
Undiscounted Operating Pre-Tax Cash Flow:	$193.3 million
Pre-tax NPV@5%:	$144.0 million
Pre-tax NPV@7%:	$127.9 million
Pre-tax NPV@9%:	$113.4 million
Pre-tax IRR:	40.6%
After-tax NPV@5%:	$110.4 million
After-tax NPV@7%:	$96.8 million
After-tax NPV@9%:	$84.6 million
After-tax IRR:	34.0%

Undiscounted Operating After-tax Cash Flow:	$151.8 million
After-tax Payback Period:	3.0 years
All-in Sustaining Costs:	$832/Au ounce
All-in Costs:	$954/Au ounce
Total Operating Costs:	$780/Au ounce

Sensitivity analyses were conducted to determine the sensitivity of the economic model to changes in capital costs, operating costs, and gold price. The results are shown in Table 1-11, Figure 1-3, Table 1-12, and Figure 1-4.

Table 1-11: Kilgore Project NPV@7% Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

Variable	NPV@7% at % Of Base Case (millions)				
	80%	90%	100%	110%	120%
Capital Cost	$114.35	$105.71	$96.82	$87.68	$78.28
Operating Cost	$152.78	$125.03	$96.82	$67.05	$39.72
Gold Price	$4.31	$48.34	$96.82	$143.00	$188.71

Figure 1-3: Kilgore Project NPV@7% Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

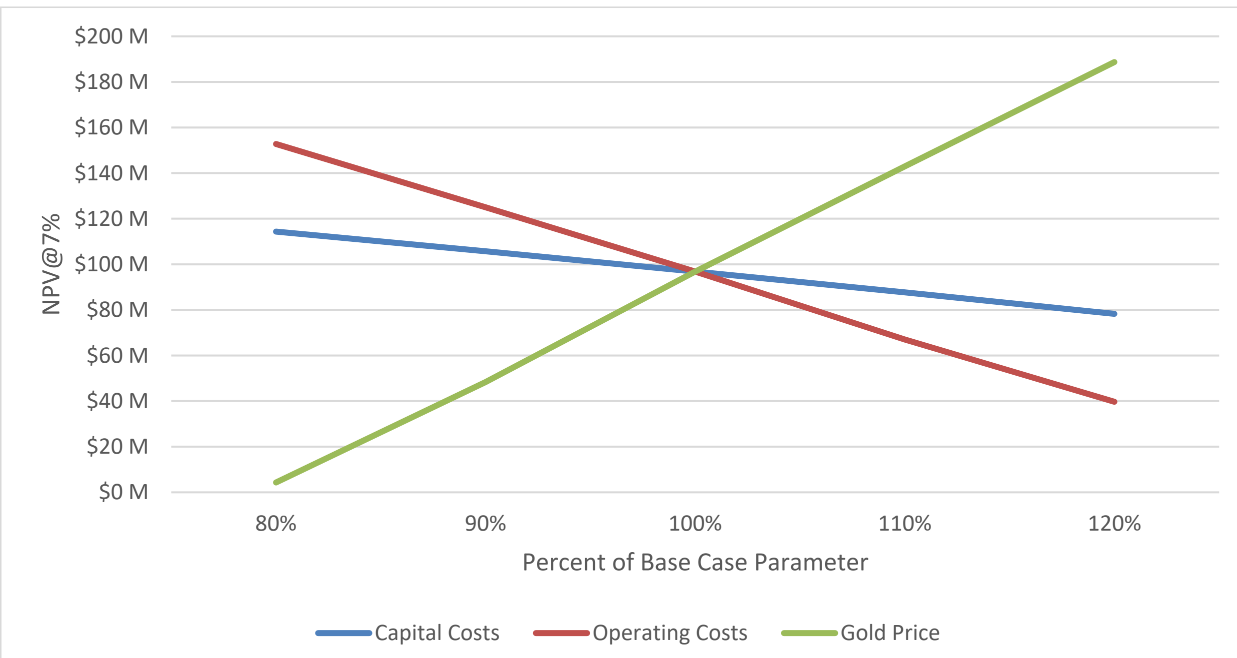
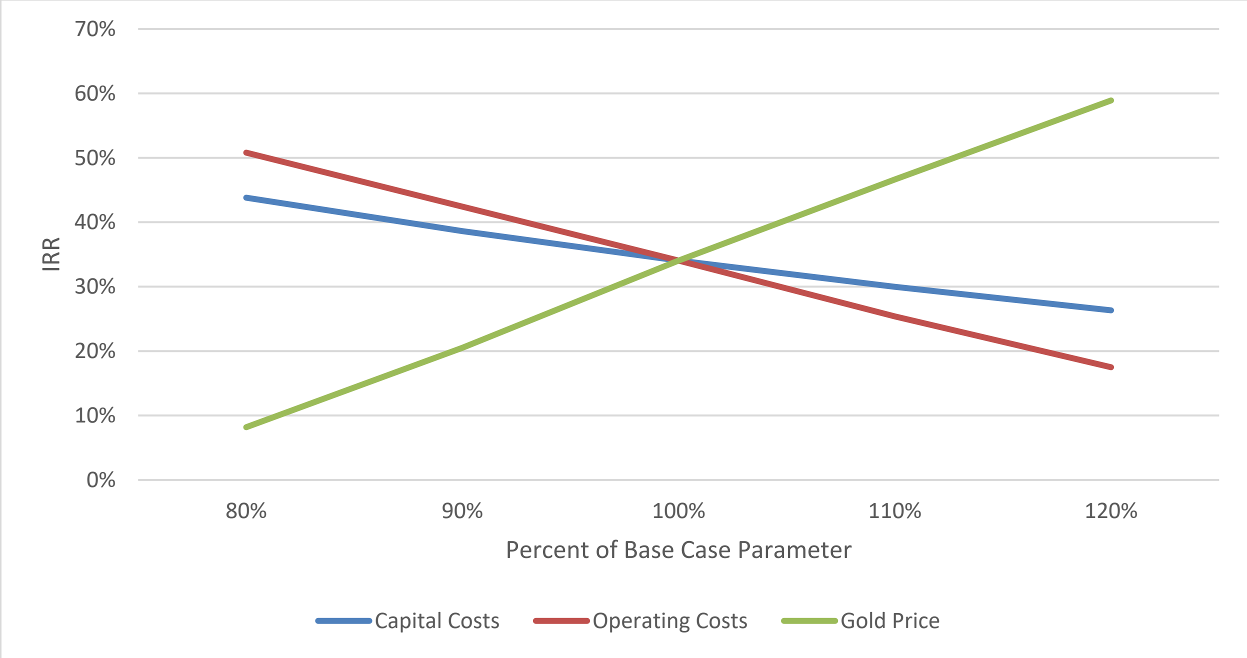


Table 1-12: Kilgore Project IRR Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

Variable	IRR at % Of Base Case				
	80%	90%	100%	110%	120%
Capital Cost	43.81%	38.61%	34.04%	29.98%	26.33%
Operating Cost	50.80%	42.42%	34.04%	25.38%	17.50%
Gold Price	8.19%	20.53%	34.04%	46.65%	58.90%

Figure 1-4: Kilgore Project IRR Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs



A positive valuation is maintained across a wide range of sensitivities on key assumptions.

1.11 Conclusions and Recommendations

The authors have reviewed data and reports supplied by Otis Gold pertaining to the project and have found them to be reasonable in the context in which they are being used. Based our analysis, the authors believe the project has economic potential and should continue to be explored, developed, and advanced to pre-feasibility study.

The individual domain resource estimates are generally contiguous and form a body of mineralization potentially amenable to bulk tonnage mining in an open pit setting. This appears to be supported by the metallurgical studies performed to date by previous companies and Otis Gold.

Exploration in and around the Kilgore Project reveals a large area of hydrothermal alteration that resulted from the geothermal system generated by the magmatism and volcanism associated with the Heise Volcanic Field. Multiple super-volcanic eruptions created both the host rocks and the structural environment to allow precious metal-bearing fluids to be emplaced throughout the Kilgore Project area. Continuing exploration work conducted by Otis Gold has demonstrated gold occurrences over the entire land package currently held. Regional exploration including stream sediment and soil sampling in combination with surface geologic mapping are valuable in identifying further near surface precious metal epithermal style mineralization.

Conventional open pit mining methods using drill, blast, load, and haul mining are applicable to the Kilgore deposit. The deposit is amenable to heap leach gold and silver recovery.

Based on these findings the following recommendations have been presented:

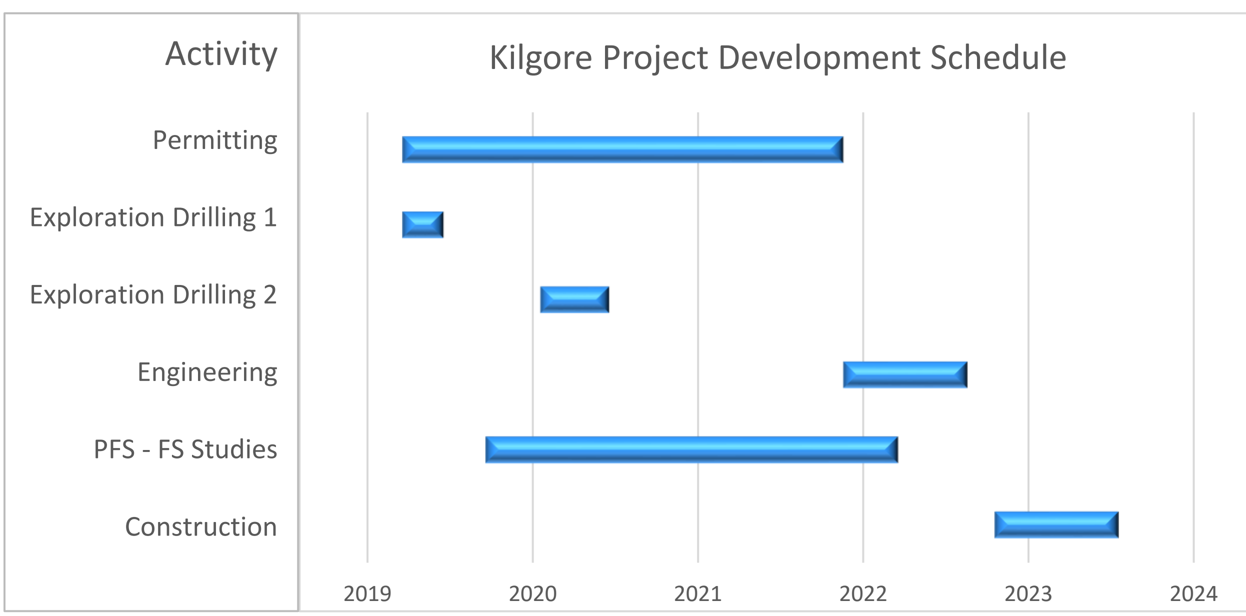
- Continue drill testing the near surface potential of the deposit by drilling to north, south, and west where it remains open including fracture / fault studies to better define the relationship between mineralization and structure, and oriented and geotechnical drilling to assist in mine design studies.
- Assay existing pulps for silver and include silver assays in all new exploration sampling.
- Continue drill testing the lateral and vertical extent of the sediment hosted gold mineralization in the Aspen Formation.
- Drill 3-5 core holes for metallurgical test work including large diameter holes to test ROM potential in the lithic tuff and sill.
- Relog and analyze Aspen Formation intervals to identify good and poor-performing metallurgical domains.
- Ensure that all subsequent metallurgical analysis on new samples utilizes cyanide amenability tests (P_{80} of 10 mesh with a 96-hour bottle roll leach) to define the direction for subsequent testing. This will establish a database of amenability tests for future geometallurgy.
- Ensure that complete carbon assays are undertaken on all mineral domains.
- Quantify the ore types tested to determine the relative abundance of each domain and map the recoveries to those domains.
- Explore the crush size relationship in more detail to allow for the optimization of the ultimate heap leach design.
- A detailed analysis of the material tested and its representativity to the deposit should be conducted to ensure adequate grade, material type, and spatial representativity.
- Increase the tracking of silver in subsequent metallurgical testing and inform the model with additional silver assay details.
- More geotechnical investigation should be undertaken to ensure heap permeability under a multiple lift scenario.
- Based on the metallurgical review, a conservative approach has been taken to ensure maximum gold and silver recovery is obtained.
    - A crush size P_{80} of ½-inch has been selected for the heap design.
    - A primary leach period of 90 days should be employed. Based on these parameters, gold extractions of 82% for crush and 50% for ROM, should be achievable.
    - Cyanide and lime consumptions are moderate. The cyanide consumption has been scaled from the average for all column tests of 2.16 pounds per ton (lb/t) to a projected heap consumption of 0.5 lb/t. The average lime consumption from the column tests has been employed (with the removal of one outlier) to provide an expected consumption of 2.6 lb/t.
    - No agglomeration is necessary as the column tests all exhibited excellent permeability.
    - The silver grade does not appear to be high enough to warrant the use of a Merrill Crowe recovery system. A standard carbon adsorption circuit should be acceptable.
- A Lidar survey will be needed for mine, waste rock, heap leach, and plant facility designs.

- Base line surface water flows and water quality will be needed for design.
- Groundwater monitoring and testing wells will be needed to create a groundwater model and predict pit inflows and pit dewatering requirements.
- In addition to the geologic description, core holes should be logged, and RQD and rock mass rating should be identified.
- Geotechnical testing of soils near the plant, waste rock, and heap leach sites.
- Geotechnical testing of rock for pit wall design.
- Continue Kilgore Project-wide exploration to test for emerging targets both inside and outside the existing land position.
- The recommended budget is $4.4 million and is outlined in Table 1-13.

Table 1-13: Proposed Budget for Kilgore Project

Drilling – Exploration & development	7,500 m	$200/m	$1,500,000
Drilling – metallurgical and water monitoring	2,500 m	$200/m	$500,000
Large diameter bulk samples	600 m	$1,000/m	$600,000
Soils testing			$196,000
Geologic mapping			$140,000
Core studies			$50,000
LiDAR survey			$75,000
Baseline Studies			$250,000
Office Rent			$36,000
Bonding			$371,000
Annual Claim Maintenance Payments			$118,000
Bulk Sample Column Testing			$500,000
Data management			$105,000
Total			**$4,441,000**

The estimated schedule for completing development work on the project is shown in Figure 1-5.



2.0 INTRODUCTION

2.1 Terms of Reference

Otis Gold Corp. (Otis Gold) engaged Global Resource Engineering, Ltd. (GRE), the "Author", to perform a Preliminary Economic Assessment (PEA) of the Kilgore Project, Clark County, Idaho, U.S.A. The PEA builds on the Mineral Resource Estimate prepared by GRE and Rowearth (2018). The 2018 Mineral Resource Estimate is presented within this report in its entirety as presented in the 2018 Technical Report (GRE & Rowearth, 2018). No additional Mineral Resource Estimation has been performed since the issue of that report.

The PEA and this report were prepared according to the guidelines of Form 43-101F1, and Companion Policy 43-101CP, as amended by the Canadian Securities Administrators and enacted on June 30, 2011.

David Rowe, Terre Lane, Jeffrey Todd Harvey, and J.J. Brown are all Qualified Persons under the Instrument. Mr. Rowe and J.J. Brown and conducted independent site visits to the Kilgore property on August 9 through 14, 2017 and August 4 through 5, 2018, respectively. The conclusions and recommendations in this report are based on information available as of June 1, 2019.

The term "Kilgore Project" refers to the entire area covered by the unpatented Federal mining claims upon which the mineral resources are located and exploration programs conducted by Otis Gold. This report makes recommendations for specific work and a budget for the Kilgore Project.

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

This Technical Report was prepared specifically for the purpose of complying with Canadian Securities Administrators National Instrument 43-101 (NI 43-101) and may be distributed to third parties and published without prior consent of the Authors if the Technical Report is presented in its entirety without omissions or modifications, subject to the regulations of NI 43-101. Consent is expressly given for submission of this Technical Report by Otis Gold to all competent regulatory agencies, including but not limited to the British Columbia Securities Commission, the Ontario Securities Commission, the Alberta Securities Commission, the TSX-Venture Exchange, and the Toronto Stock Exchange. However, all reports, publications, exhibits, documentation, conclusions, and other work products obtained or developed by the authors during completion of this Technical Report shall be and remain the property of the author. Unauthorized use or reuse by third parties of reports, publications, exhibits, documentation, conclusions, and other work products obtained or developed by the authors for the purposes of this Technical Report is prohibited. Use of this report acknowledges acceptance of the foregoing conditions.

2.2 Sources of Information and Data Used

Otis Gold provided the Authors with compilations of data used as a basis of this report, principally from geologic mapping, cross-sections, and drilling campaigns, as well as metallurgical test reports.

This report is based, in part, on internal company technical documents, maps, published government reports, company memoranda, data and reports prepared by laboratories and professional consultants in various disciplines, and public documents and statements made by Otis Gold. Cost data is from budgetary quotes from vendors, Infomine Cost Service (InfoMine, 2018), and GRE internal database. The NI 43-101

Technical Report by Rayner and Associates and Van Brunt (2002) and the NI 43-101 Report by Cameron (2012) were relied upon for historical information on technical aspects and mineral resources for the Kilgore deposit up to those dates.

The maps and tables for this report were produced by the Authors, by Otis Gold, or modified from reports written for Otis Gold by others. Illustrations or tables derived from other sources are acknowledged in the caption below the figure or table.

2.3 Site Inspections

Mr. Rowe made a site visit to the Kilgore property on August 9 through 14, 2017, where he inspected pertinent outcrops, mineralization, drill sites, and project setting, and J.J. Brown visited the site on August 4 and 5, 2018. Both Authors visited the Kilgore Project, and Otis Gold's core preparation and storage facilities located in St. Anthony, Idaho, and Spokane, Washington. They inspected drill hole assay logs and certificates, quality control information, geologic maps and sections, and took samples from surface and drill core for which they maintained secure custody and performed independent analysis for gold at a certified laboratory. Although they cannot validate and verify all of the information that composes the Kilgore database, the authors found no issues based on their site visit and other inspections which would preclude estimation of mineral resources.

2.4 Abbreviations and Acronyms

Measurements are generally reported in imperial units in this report. Discrepancies may result in slight variations from the original data in some cases due to rounding of values. Abbreviations, measures, and acronyms used in this report are explained in the list below:

µm	micron
3D	three-dimensional
AARL	Anglo American Research Laboratory
ADR	adsorption-desorption recovery
Ag	silver
ALS	ALS-Chemex
ANFO	ammonium nitrate fuel oil
Au	gold
BLM	Bureau of Land Management
CIC	carbon-in-column
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimeter = 0.3937 inches
CN	cyanide
CSAMT	Controlled Source Audio Magneto-Telluric
CWi	crusher work index
EA	Environmental Assessment
Echo Bay	Echo Bay Exploration
ESRP	Eastern Snake River Plain

g/L	grams per liter
gpm	gallons per minute
GPS	Global Positioning System
g/T	grams per metric tonne
GRE	Global Resource Engineering, Ltd.
Hazen	Hazen Research, Inc.
HEM	Helicopter Electromagnetic
HLF	heap leach facility
Hz	frequency defined as the number of cycles per second
ICP-MS	inductively coupled plasma-mass spectrometry
IMC	Idaho Mining Claim
IRR	internal rate of return
ISO	International Organization for Standardization
Ka	Aspen Formation
K-Ar	Potassium-Argon (referring to age date technique)
kg	kilogram = 2.2046 pounds
km	kilometer = 0.6214 miles
lb/t	pounds per ton
LLDPE	linear low density polyehtylene
LS	low sulfidation
Ma	million years old
mm	millimeter = 0.0394 inches
NaCN	sodium cyanide
NAD	North America Datum
NAE	North American Exploration
NI 43-101	National Instrument 43-101
NN	nearest neighbor
NPV	net present value
OK	ordinary Kriged
opt	troy ounces per short ton, 1.0 opt = 34.2857 g/T
Otis Gold	Otis Gold Corp.
PEA	Preliminary Economic Assessment
PoO	Plan of Operations
ppb	parts per billion
ppm	parts per million = g/T
QA/QC	Quality Assurance/Quality Control
QEMSCAN	Quantitative Evaluation of Materials by Scanning Electron Microscopy
QP	Qualified Person
RC	reverse circulation
RC	Reverse Circulation
RDi	Research Development Inc.
RDi	Research Development Inc.
ROM	run-of-mine

RQD	Rock Quality Designation
SG	Specific Gravity
SNOTEL	snowpack telemetry
Tad	Tertiary intermediate-composition dikes and sills
Tbr	biotite rhyolite
Tct	crystal tuff
Tlt	Tertiary lithic tuff
tpd	tons per day
Tpr	Tertiary rhyolite flows and domes, also referred to as Tbr, Felsic Dike on some graphics
Tqp	Tertiary quartz porphyry
Ttk	Tertiary tuff of Kilgore
Tup	Tertiary upper pyroclastics (sinter and explosion breccia)
USFS	United States Forest Service
UTM	Universal Transverse Mercator

3.0 RELIANCE ON OTHER EXPERTS

The NI 43-101 Technical Reports by Rayner and Associates and Van Brunt (2002) and Cameron (2012) are relied upon for historical information on technical aspects and mineral resources for the Kilgore deposit up to that date but not for the current estimate of resources.

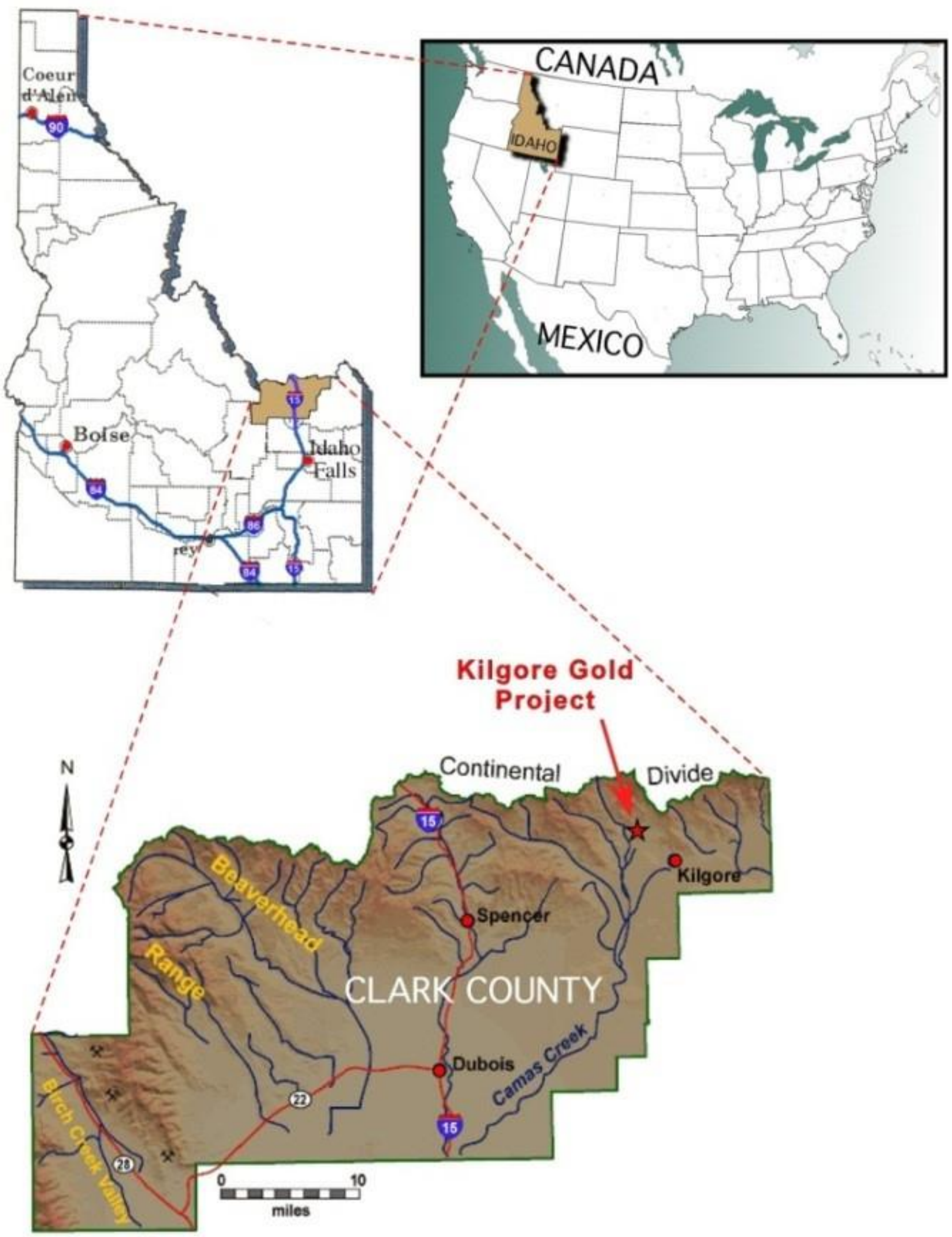
GRE relied upon descriptions, statements, and illustrations by Otis Gold with respect to the status of its mineral claims as described in Section 4 and for environmental studies, issues, and permits described in Section 20. These items are presented for information purposes as required by NI 43-101, and GRE has no opinion with respect to these items. The authors exercised all reasonable due diligence in checking, confirming, and testing project data, but has relied on Otis Gold’s information and presentation in formulating their opinions.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Kilgore Project is situated on the northern margin of the eastern part of the Snake River Plain (ESRP), approximately five miles west-northwest of the small rural community of Kilgore, Clark County, Idaho (Figure 4-1). The core of the Kilgore deposit, known as the "Mine Ridge" area, is centered on longitude 111° 59' 52" W and latitude 44° 25' 53" N. Alternative location coordinates of this core area, as measured in the Universal Transverse Mercator (UTM) Geographic Coordinate System, are 492,556E and 4,920,494N, North American Datum (NAD) 83, Zone 12.

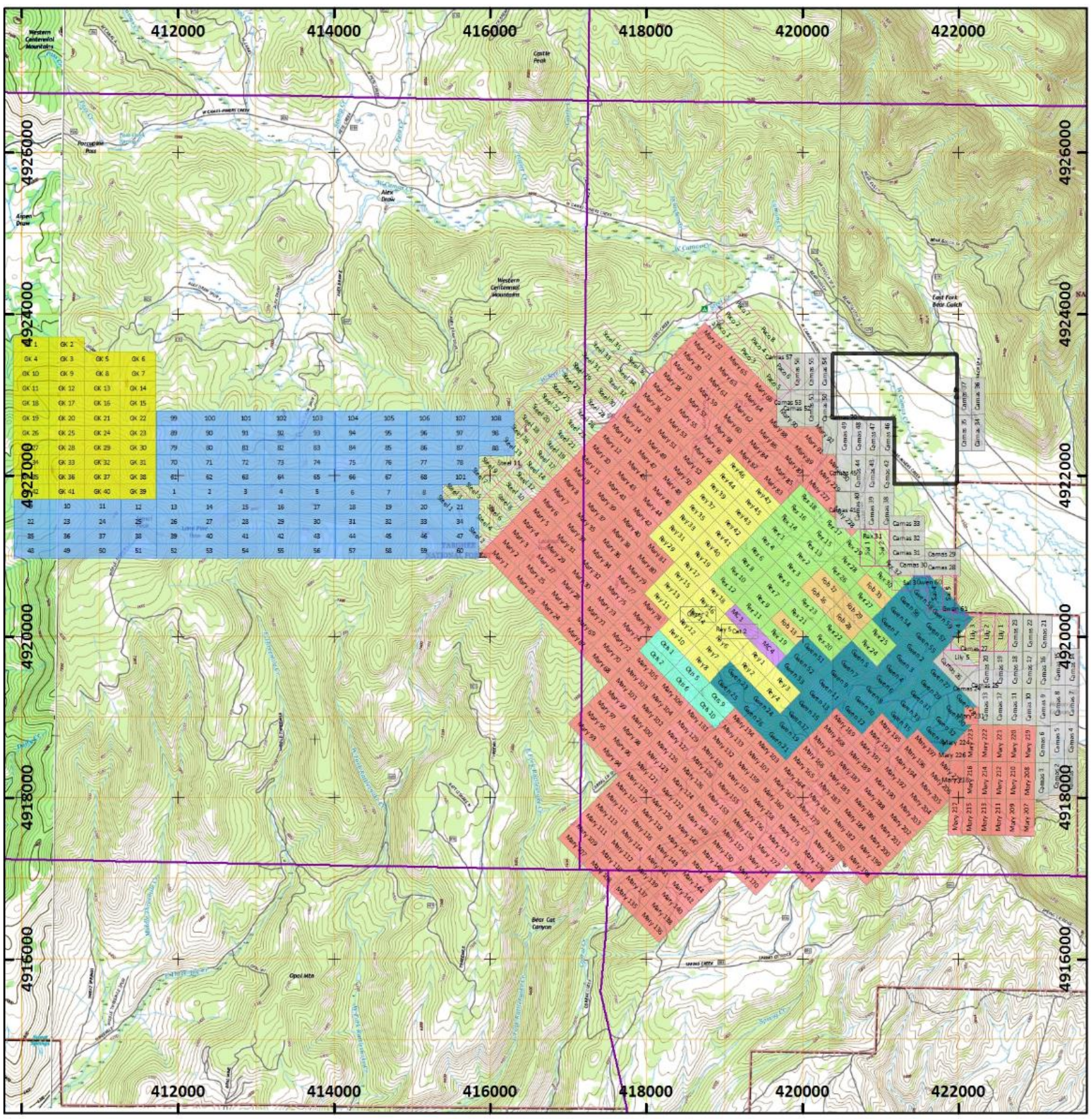
Figure 4-1: Kilgore Location Map (Source: Otis Gold, 2012)



4.2 Land Area

Otis Gold's Kilgore Project land position comprises 614 unpatented Federal lode mining claims located on U. S. Forest Service (USFS) land, Caribou-Targhee National Forest, Idaho (Figure 4-2). Included in this claim position are: 1) a core group of 162 contiguous claims (150 that were obtained 100% by Otis Gold from Bayswater Uranium Corp. in late 2011 and an additional 12 claims staked by Otis Gold in late June of 2010); and 2) an additional and mostly adjacent 70 claims staked by Otis Gold on February 2-7, 2012. These latter claims were staked approximately 1-mile downslope from, and in the West Camas Creek flats area north and east of, the deposit to obtain ground for possible potential future processing facilities and infrastructure. In total, the 614 federal lode mining claims comprise approximately 12,150 acres (19.19 square miles) located in all or parts of Sections 7, 8, 9, 15-22 and 27-34 in township T13NR38E, and Sections 13-24 in township T13NR37E, and parts of townships T12NR37E and T12NR38E; Boise Meridian, Clark County, Idaho. The current mineral resource lies wholly within Otis Gold's property.

Figure 4-2: Otis Gold Kilgore Property Map



Source: Otis Gold, 2017

4.3 Nature and Extent of Issuer's Title and Type of Mineral Tenure

Otis Gold Corp. maintains a 100% ownership interest in the Kilgore Project. Otis Gold's acquisition of the site is outlined in Cameron (2012).

Otis Gold contracted with land person Michelle McKamy of Billings, Montana, to conduct a professional land title search and land title opinion to corroborate and verify its claim ownership on September 14, 2018. Excerpts of the results and findings of Ms. McKamy's work are summarized below:

- The property description and location were taken from: 1) the land status records and mining claim files of the Bureau of Land Management (BLM) in Boise, Idaho; and 2) the indices and records of Clark County Records; and 3) documents provided by Otis Capital USA Corp.
- Otis Capital USA Corp. holds title of record to 614 unpatented lode claims located on USFS lands in accordance with the United States 1876 mining laws 43 CFR Ch11-3800, as amended, and in accordance with Idaho Statutes Title 47, Mines and Mining, Chapter 6, Location of Mining Claims.
- Idaho Mining Claim (IMC) numbers assigned to all of these claims by the BLM are as follows:

Claim Name	**BLM Serial Number**
Camas 1-70	IMC 209304-209373
Cat 2	IMC 177033
CATHY 1-108	IMC 220469-220576
FOB 13	IMC 77092
FOB 16	IMC 77095
FOB 17	IMC 77096
FOB 28	IMC 77106
FOB 29	IMC 77107
FOB 31	IMC 77109
GK #1-42	IMC 220174-220215
Gozer 2	IMC 174947
Lilly 1-5	IMC 218159-218163
Mary 1-231	IMC 216814-217044
MC 1	IMC 161374
MC 4	IMC 161377
Paco 1-8	IMC 218203-218210
Rex 1-32	IMC 185109-185140
Rey 1-47	IMC 185688-186638
Gwen 1-53	IMC 186532-186584
Gwen 54-61	IMC 186639-186646
OTIS 1-10	IMC 201716-201725
SAL 1-5	IMC 218164-218168
Steel 1-34	IMC 218169-218202

- The 614 Certificates of Location for the claims and required maps were timely recorded in the office of the Clark County Clerk, Dubois, Idaho, and with the BLM State Office in Boise, Idaho. The Affidavits of Annual Maintenance Fee payment was timely recorded in the Office of the Clark

County Clerk and Recorder, Dubois, Idaho. The annual payments were received and verified by the BLM and are valid until August 31, 2019, at which time another annual payment will be due.

4.4 Royalties, Back-In Rights, Environmental Liabilities, or Encumbrances

Otis Gold reports that there are no royalties, back-in rights, payments, environmental liabilities, or any other encumbrances affecting its holdings based on the results of McKamy's title review work (McKamy, 2018). McKamy's findings are summarized below:

- There are no valid existing claims of record in conflict with these held by Otis Capital USA Corp. There are no liens, judgments, suits, or any litigation involving either Otis Capital USA Corp. or the claims it holds in Clark County Records, Idaho, nor in the land files at the BLM State Office in Boise, Idaho.
- Otis Capital USA Corp. owns 100% interest in the subject 614 lode mining claims. The property has no recorded reserve royalties, rights, agreements, or encumbrances, and no environmental or tax liabilities of record.

Annual assessment filings/fees continue to be required by the BLM to keep the claims in good standing and for Otis Gold to retain the property mineral rights. Finally, and to the extent known by GRE, there are no environmental liabilities nor are there any significant factors and/or risks that may affect access, title, or the right or ability to perform work on the property.

4.5 Permits and Bonding to Conduct Work

All of the 614 Federal lode mining claims are located on USFS ground, and permits are required to conduct work proposed on the property. Permits are obtained through the Caribou-Targhee National Forest local headquarters in Dubois, Idaho. During the period from 2008 to 2012, Otis Gold performed its drilling on previously established project drill roads. As a result, the annual permitting of a Plan of Operation (PoO) was expedited, and reclamation bonding requirements were minimized. During this period, permits to drill were issued by the USFS within the framework of a Categorical Exclusion, where the permitted activities are deemed acceptable to all local parties, with no substantive or negative comments of appeal received.

In 2015, a new PoO was approved to construct approximately 1,200 meters of new access roads within the Caribou-Targhee National Forest and to conduct drilling at approximately 16 sites. The approval was received after the completion of an Environmental Assessment (EA) by the USFS and subsequent collaborative negotiations with the USFS and other interested parties. This 2015 PoO was subsequently amended several times and expired at the close of 2017. An application for a new PoO was submitted in October 2017. This PoO was permitted with the completion of a new EA, and approval was received in August 2018. The new PoO, which is subject to the receipt of bonding totaling $370,600 (an increase from the prior existing bond of $121,275), covers exploration in the Kilgore deposit, Dog Bone Ridge, Gold Ridge and Mine Ridge areas, including approval for 140 drill sites and the construction of associated access roads. Currently, Otis Gold utilizes a 3rd party agency to maintain its bonding at the Kilgore Project. This new permit is valid for five years.

Otis Gold must maintain a reclamation bond with the USFS, which is rolled forward every year with each new annual PoO and corresponding permit. Otis Gold's exploration plan and the estimated amount of surface disturbance involved are used to calculate a monetary reclamation bond assessment that must be posted by Otis Gold before it can begin its work. As of August 2017, bond monies in the amount of $121,275.00 are held at the Idaho Branch of the East Idaho Credit Union to cover project reclamation. The bonding amount for the 2018 approved permit totals $370,600.

Additional permits needed on an annual basis include temporary permits to obtain water for drilling as administered by the State of Idaho, Department of Water Resources in Idaho Falls, Idaho. Historically, this permit has been is granted within a two-week period. Water Withdrawal permits for 2018 drilling were obtained on April 17, 2018.

Idaho State Land Use Permit No. LU 800561 was acquired by Otis Gold from the Idaho State Department of Lands on February 17, 2012 to conduct exploration on 480 acres of Idaho State Lease Land in the northwest, northeast, and southeast quarters of Section 16, T13N, R38E, Clark County, Idaho. This land is located in the West Camas Creek drainage and flats, approximately one mile north of the deposit.

4.6 Any Other Factors or Risks

The USFS may place access restrictions on the property if extreme danger of forest fire is present but this action has never been required during Otis Gold's drilling of the property. GRE does not know of or been informed of any other significant factors or risks that may affect access, title, or the right or ability to perform work on the property.

A coalition of environmental groups lead by the Idaho Conservation League filed an administrative complaint against the USFS in November 2018 citing failings within the Environmental Assessment (EA) in-lieu of a more comprehensive Environmental Impact Statement (EIS). The administrative legal procedures are currently underway, and Otis Gold expects a ruling potentially as soon as October 2019.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Elevation, and Vegetation

The Kilgore deposit is in a mountainous region on the northern margin of the ESRP between the ESRP to the south and the Centennial Mountains to the north, an east-west range that forms the Continental Divide in this part of the Rocky Mountains. Elevations in the overall project area range from approximately 6,400 feet (1,951 meters) to 8,400 feet (2,560 meters) above sea level, and elevations in the deposit area range from about 7,000 feet (2,134 meters) to 7,800 feet (2,377 meters) above sea level.

Topography defining the project area and its immediate surroundings comprises a gently southwest dipping plateau (Photo 5-1), underlain by a layered, southwest-dipping pile of Miocene-Pliocene volcanic rocks that form a dip-slope. The landform terminates in a northeast-facing slope break at Kilgore in a transition to the lowlands of the West Camas Creek drainage.

Photo 5-1: General Southwesterly Dip of Plateau Containing the Kilgore Deposit on its Up-dip Northeastern Edge

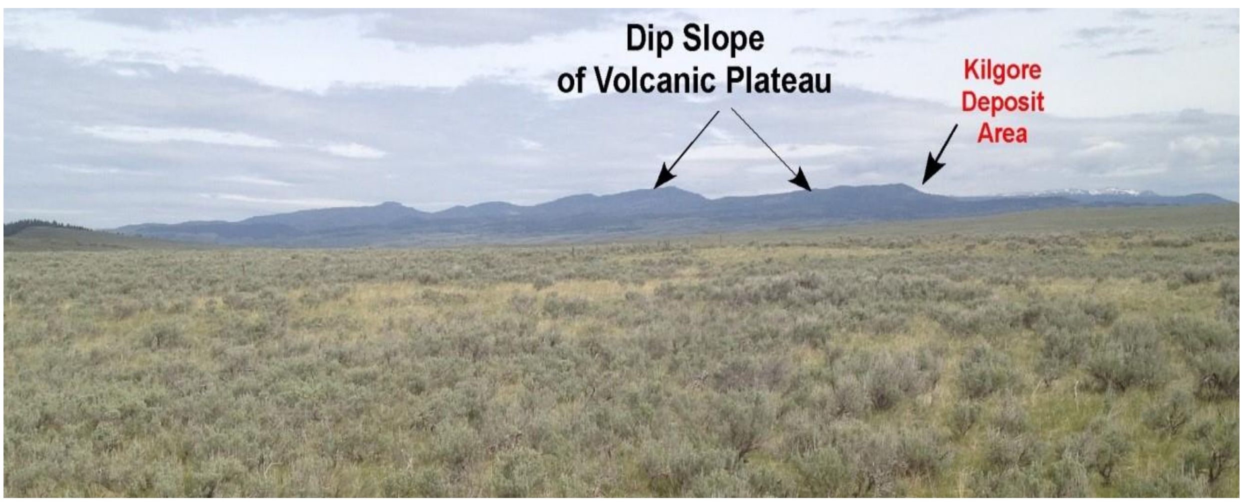


Photo by Otis Gold

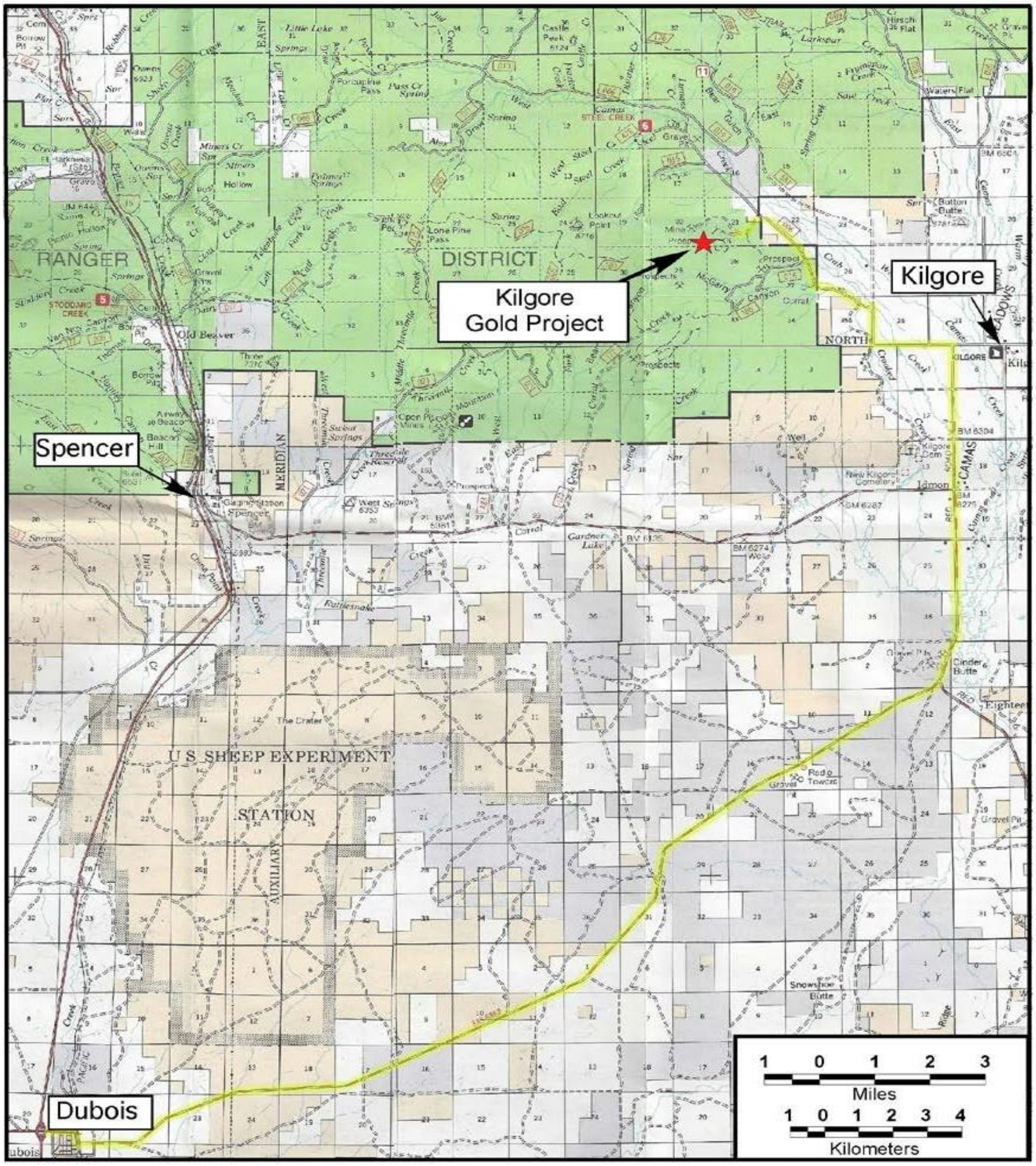
The Kilgore Project has a growing season of less than 70 days, with vegetation characterized by open Douglas fir, lodgepole pine, and subalpine fir forests. Mountain brush and sagebrush cover the lower elevations. Other common native plant species found in the project area include spirea, pinegrass, mountain snowberry, and gooseberry currant. No plant species protected under the Endangered Species Act are known from Clark County, and no special-status plants were found during Golder Associates Preliminary Environmental Study of the area during 2010 (Golder Associates, 2010).

Numerous animal species exist in the project area due to the high level of habitat diversity and large tracts of forested and open land present. Game species noted in the area include mule deer, elk, moose, blue grouse, and mourning dove. Small mammals documented within the project area include red squirrel, beaver, deer mice, shrews, and voles. Carnivores noted in the area include coyote, weasel, mountain lion, black bear, grizzly bear, wolverine, and wolf (JBR Environmental Consultants, 1997). Over nineteen bird species have been recorded within the project area, including the northern goshawk, red-tailed hawk, American kestrel, and great horned owl. Amphibians include spotted frogs and western toad. Domesticated cattle graze in the West Camas Creek drainage area.

5.2 Accessibility

Road access to and through the deposit area is good, with a network of paved and historic unimproved drill roads serving as the direct route to the deposit area (Figure 5-1). Four-wheel drive may be required in early spring or wet weather.

Figure 5-1: Map Showing Access Route from Dubois, Idaho to the Kilgore Project



Source USFS and Otis Gold, 2012

5.3 Demographics, Local Resources, and Infrastructure

The closest infrastructure to the Kilgore Project is the small rural community of Kilgore, Idaho, located approximately five miles east-southeast of the deposit (Figure 5-1), which has a minimum of supplies and resources. Dubois, Idaho, the county seat, is located approximately 26 miles by paved and gravel road from the town of Kilgore, Idaho. Dubois offers a nearly full-service community with a gas station, grocery store, bank, restaurant, two small motels, and other amenities.

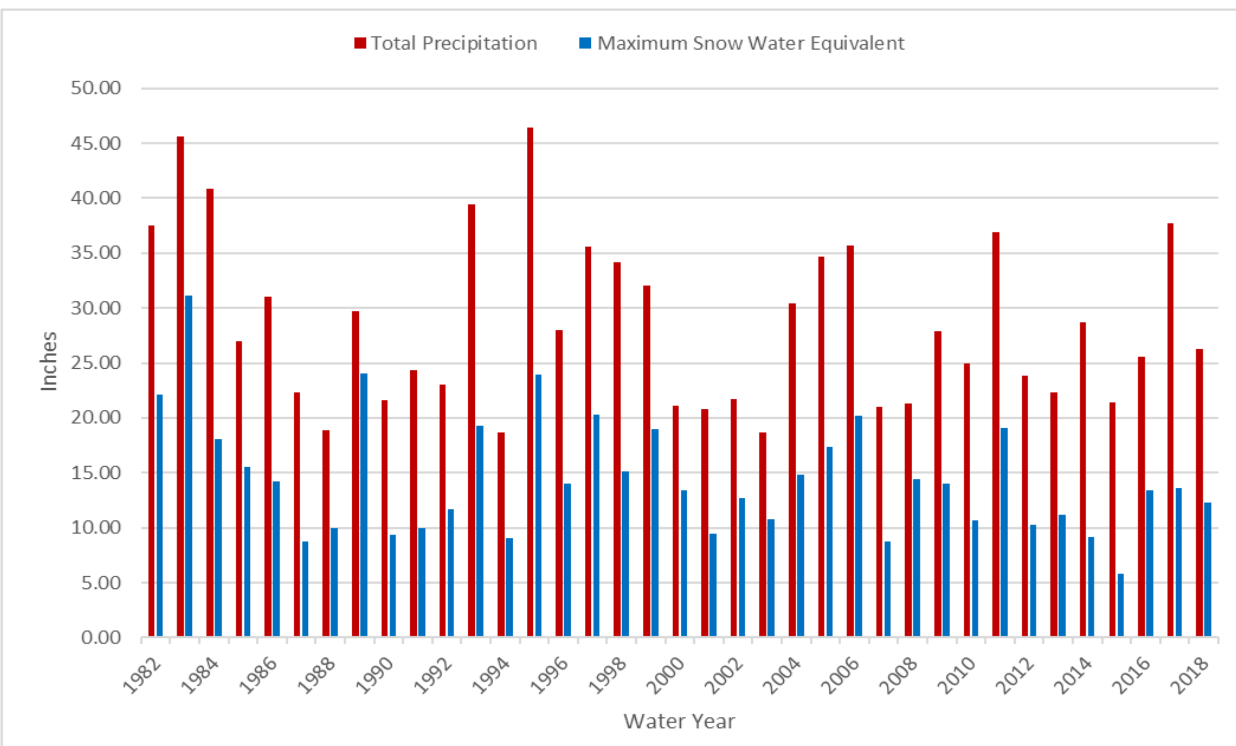
5.4 Nature of Transport

Access to the property is excellent by car, truck, and four-wheel drive vehicle on paved, gravel, and unimproved roads. The Union Pacific Railroad operates a major freight rail line running through Spencer, Idaho, approximately 10 miles southwest of the deposit, and through Dubois, Idaho, approximately 26 miles to the southwest of the deposit.

5.5 Climate and Length of Operating Season

Climate in Clark County in the vicinity of the Kilgore Project is defined by mainly cold, snowy winters and warm and relatively dry summers. The Natural Resource Conservation Service maintains the Crab Creek weather station and Snowpack Telemetry (SNOTEL) site only 0.5 miles north downslope of the Kilgore deposit and northwest of Crab Creek. SNOTEL reports an annual precipitation average of 28.6 inches, but the annual totals vary (Figure 5-2).

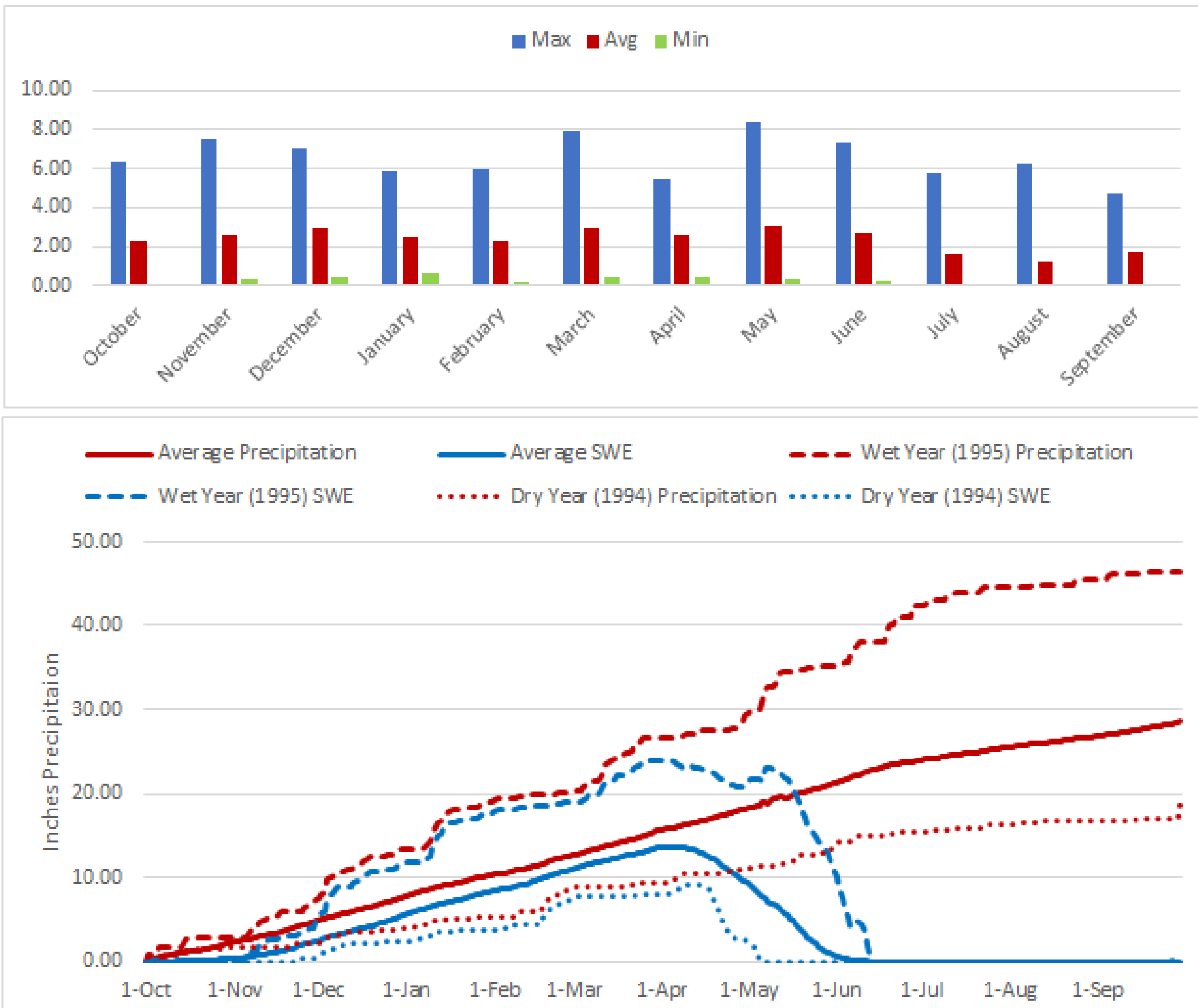
Figure 5-2: Annual Precipitation Totals, Crab Creek (Cabin Creek) SNOTEL Station, Kilgore Project, Clark County, Idaho



Source: Otis Gold

December and May are typically the wettest months of the year, while August and September are the driest months (Figure 5-3). A little more than half of the annual precipitation falls as snow (Figure 5-3).

Figure 5-3: Distribution of Precipitation and Snow Water Equivalent Throughout the Year, Crab Creek (Cabin Creek) SNOTEL Station, Kilgore Project, Clark County, Idaho

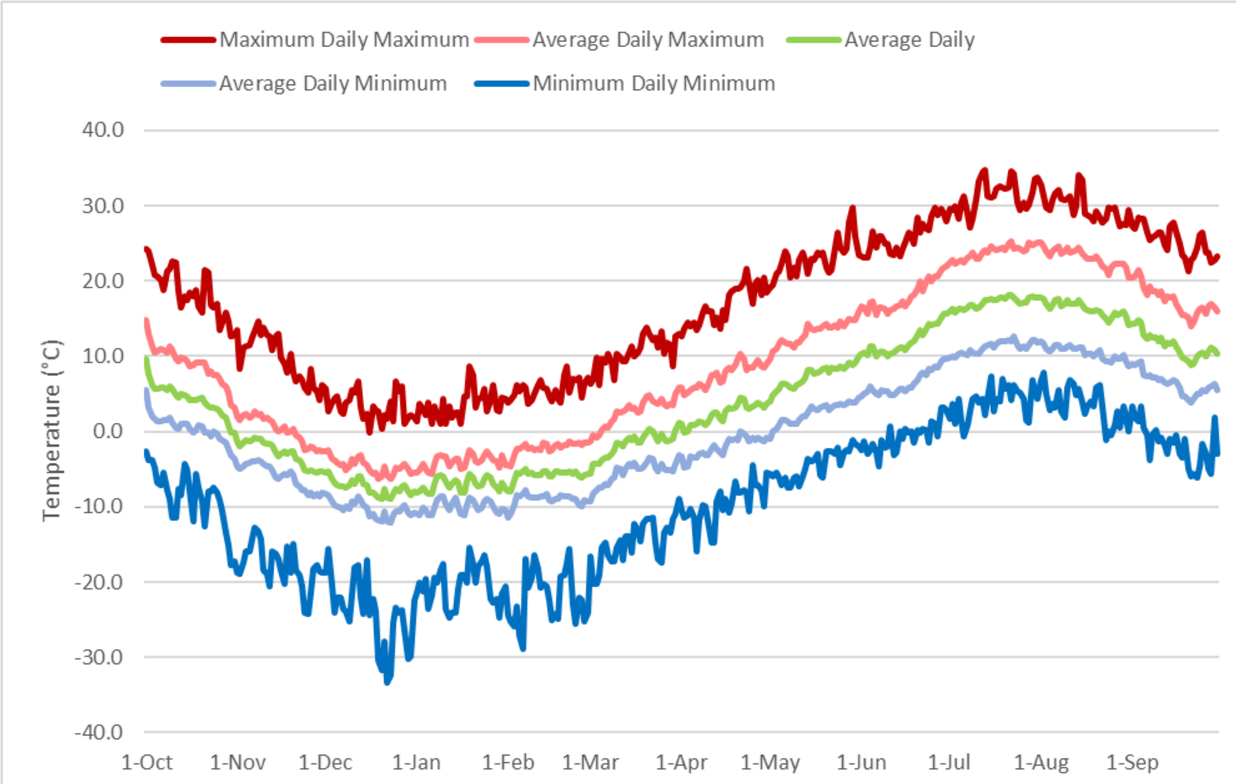


Source: Otis Gold

Average daily temperatures from the SNOTEL site show that, on average, temperatures at Kilgore are below freezing (0°C, or 32°F) from November through April, with daily maximums of 34.7°C (95°F) occurring in July and August (Figure 5-4).

The length of a typical operating/exploration drilling season is generally from about mid-May to early December, depending on snowmelt and associated water runoff conditions in the spring and snow accumulation conditions late in the year.

Figure 5-4: Average Daily Temperatures, Crab Creek (Cabin Creek), SNOTEL Station, Kilgore Project, Clark County, Idaho



Source: Otis Gold

5.6 Water, Power, Mining Personnel, Potential Processing Sites

Otis Gold's 614 Federal lode mining claims allow the Company mineral and surface access rights under established US Mining Law. Power exists as a line paralleling the USFS Road 006, West Camas Creek Road, located one-mile northeast of the deposit. Water is plentiful in the West Camas Creek drainage 1.5 miles north-northeast of the deposit. Results of core drilling by both Echo Bay Exploration (Echo Bay) and Otis Gold reveal that the water table in the deposit area is generally at depths of between 200 feet and 400 feet below the surface, suggesting a possible source for process water that would have less impact than drawing it from West Camas Creek. An ample source of labor is available from the towns of Dubois, Rexburg, Rigby, and Idaho Falls, Idaho, all within 60 miles of the deposit, from southern Montana and northern Nevada, and from the local rural and general population base. The area features potential sites for processing plants, water storage, heap leach pads, and facilities. One site option is Otis Gold's CAMAS federal lode mining claims, located largely in the flats north of the deposit.

6.0 HISTORY

6.1 Pre-Otis Gold

Historic work on the Kilgore deposit is summarized in the Technical Reports by Rayner and Associates and Van Brunt (2002) and Cameron (2012). Early gold exploration at the deposit was conducted in 1937 by the Blue Ledge Mining Company (Campbell, 1937). Evidence of mining activity remains as several underground adits, prospect pits, a mill foundation, and a tramway. Although miners reportedly uncovered “considerable ore of commercial value” (Campbell, 1937), there is no evidence in the form of tailings that metals were ever recovered from the ores. Further, there is no evidence of placer mining in the gulches below the deposit, although it is probable that panning lead to the discovery of the lodes (Benson C. , 1986).

A total of 54 unpatented lode claims were located to cover the core area of the Kilgore deposit by Dennis Forsberg and Foster Howland in 1982. Several mining companies conducted exploration on these claims, including Bear Creek Mining from 1983 to 1985, Placer Dome U.S. from 1990 to 1992, Pegasus Gold from 1993 to 1994, and Echo Bay from 1994 to 1996.

Bear Creek leased the claims from Forsberg and Howland in 1983. The Bear Creek program comprised seven reverse circulation (RC) and core holes during its tenure. Placer Dome drilled 39 holes, including 5 core holes, conducted rock and soil sampling, ran a gradient array IP/resistivity survey, located 82 unpatented lode mining claims, and ran metallurgical tests. The drilling continued through a 50-50 joint venture between Placer Dome and Pegasus with an additional 23 holes.

Echo Bay entered into a Joint Venture agreement with Placer Dome U.S. in June 1994. Echo Bay attained earn-in for 51% of the Kilgore Deposit in June 1996 when project expenditures reached $2.5 million USD. In December 1996 Echo bay completed the purchase of Placer Dome’s remaining 49% stake in the project to hold 100% of the property.

Echo Bay conducted more systematic exploration and evaluation of the Kilgore deposit, spending $4.7 million between 1994 and 1996. Exploration included drilling 122 new drill holes, re-logging all previous drill holes, airborne helicopter electro-magnetic (HEM) surveying, regional geological mapping and soil sampling on the backside sinter, or Dog Bone Ridge target area. Echo Bay performed bottle roll and column leach metallurgical studies, collected environmental baseline data, did resource modeling, and completed initial engineering assessment studies of the main Kilgore deposit area.

In all, between 1984 and 1996, a total of 122,257 feet in 190 holes were drilled on the Kilgore deposit and proximal targets with the goal of defining a bulk-tonnage, open-pitable gold deposit. The majority of this drilling concentrated on the Kilgore deposit, the subject of this NI 43-101 technical report. No further drilling was done on the deposit until Otis Gold began its work in 2008, a hiatus of 12 years since the last historic drilling was completed by Echo Bay in 1996.

Latitude Minerals entered into a 49% - 51% joint venture agreement with Echo Bay on September 2, 1998, and drilled a sinter cap and explosion breccia area, now known as Dog Bone Ridge, located roughly 4,000 feet (1,220 meters) west-southwest of the main Kilgore deposit on Mine Ridge. Latitude drilled six holes, three of which encountered anomalous gold mineralization averaging 300 feet (91.4 meters) thick and extended well-mineralized intercepts at least 300 feet west-northwest of Echo Bay hole 96 EKC-178. The drill results also revealed extensive alteration characteristic of volcanic-hosted gold systems.

In 2002, Kilgore Gold, a wholly-owned subsidiary of Kilgore Minerals, acquired 100% ownership of the property from Forsberg and Howland. From 2002 to 2006, Kilgore Gold conducted detailed field mapping and structural analysis work on Dog Bone Ridge to delineate drill targets to further expand on Echo Bay's and Latitude's work in the area (Caddey, 2003). In 2004, Kilgore Minerals expanded its property position to 3,000 acres and drilled six core holes into the Dog Bone Ridge target area for a total of 5,319 feet (1,566 meters). Significant gold mineralization comprising a 170-foot-thick (51.8-meter) intercept from 370 to 540 feet (112.8 to 164.6 meters) deep and grading 0.036 troy ounces per ton (opt) (1.25 grams per metric tonne [g/T]) Au was encountered in hole KG042; however, no mineralization of significance was found in the other five holes (Pancoast, 2004). The Company drilled eight additional core holes totaling 5,569.4 feet (1,697 meters) in 2006. Drill hole KG06-01, an offset to KG04-02, encountered 41.3 feet (12.6 meters) of mineralized material from 510 feet (155.6 meters) to 552 feet (168.2 meters) grading 0.038 opt (1.30 g/T) Au.

In 2008, Otis Gold formed a joint venture with Bayswater Uranium Corporation and began its exploration programs on the Kilgore deposit.

6.2 Historical Mineral Resource Estimate

Rayner and Associates and Van Brunt (2002) completed a NI 43-101 compliant mineral resource estimate in a Technical Report for Kilgore Gold. That study, following methodology developed for in-house Echo Bay resource estimates, estimated grade to regular 30 x 30 x 15-foot blocks. The resource comprised separate estimates of three lithologic domains, Lithic Tuff, Crystal Tuff, and Aspen Formation, and 12,788 assay intervals. Compositing to 15-foot intervals, two composites greater than 1.0 ounce per ton (opt) Au were excluded from the estimate as outliers. Composites were further restricted to an interpreted 0.010 gold shell. Slightly anisotropic relative gold variograms were used to design search ellipses for the individual domains, with maximum dimensions of 140 to 150 feet and minimum distances of 120 to 140 feet. Blocks estimated by more than six composites and three or more drill holes were considered Indicated Resources, as shown in Table 6-1.

Table 6-1: Historical Mineral Resources for Kilgore Property

Classification	Cutoff Grade (opt)	Au opt	Tons (000s)	Ounces (000s)
Indicated	0.010	0.031	7,043	218
Inferred	0.010	0.028	9,661	269

Units are Imperial: tons are short tons, grade is ounces per short ton, ounces are troy ounces.
Source: Rayner and Associates and Van Brunt, 2002

The estimation technique for the resources in Table 6-1 was ordinary kriging. The mineral resources in Table 6-1 should not be considered a current resource. The authors have not used or relied upon this information in preparing the PEA presented in this report.

An NI 43-101 Mineral Resource Estimate was performed by Don Cameron in 2012 (Table 6-2) based on a then new block model that incorporated historic drilling information, updated geologic interpretations, bulk density test work, and geostatistical modeling of gold grade. The cutoff grade used for reporting and classification was based on a heap leach open pit mining scenario. The estimate was made using Micromine software. The mineral resources in Table 6-2 should not be considered a current resource. The authors have not used or relied upon this information in preparing the PEA presented in this report.

Table 6-2: Historical Mineral Resources for Kilgore Property by Don Cameron, 2012

Resource Category	Tonnes (000s)	Gold Grade g/T	Gold Troy Ounces
Measured	-	-	-
Indicated	27,352	0.59	520,000
Total Measured and Indicated	27,352	0.59	520,000
Inferred	20,230	0.46	300,000

Mineral Resources are at a gold cutoff grade of 0.24 g/T; gold price 1,650 USD/ounce; Au recovery 90%, and pit walls 45°.
Mining cost $1.93/ton, waste mining $1.82/ton, processing cost $7.72/ton, selling cost $5/ton.
Items are rounded off to reflect the precision of the estimate, thus metal quantity varies slightly from the product of tons and grade.
Contained gold ounces are in-situ and include metallurgical recovery losses.
Indicated mineral resources are pit constrained; inferred mineral resources are not.

6.3 Past Production

Other than a few carloads of material mined and stockpiled at the deposit in 1937, no production is known or reported from the property (Campbell, 1937).

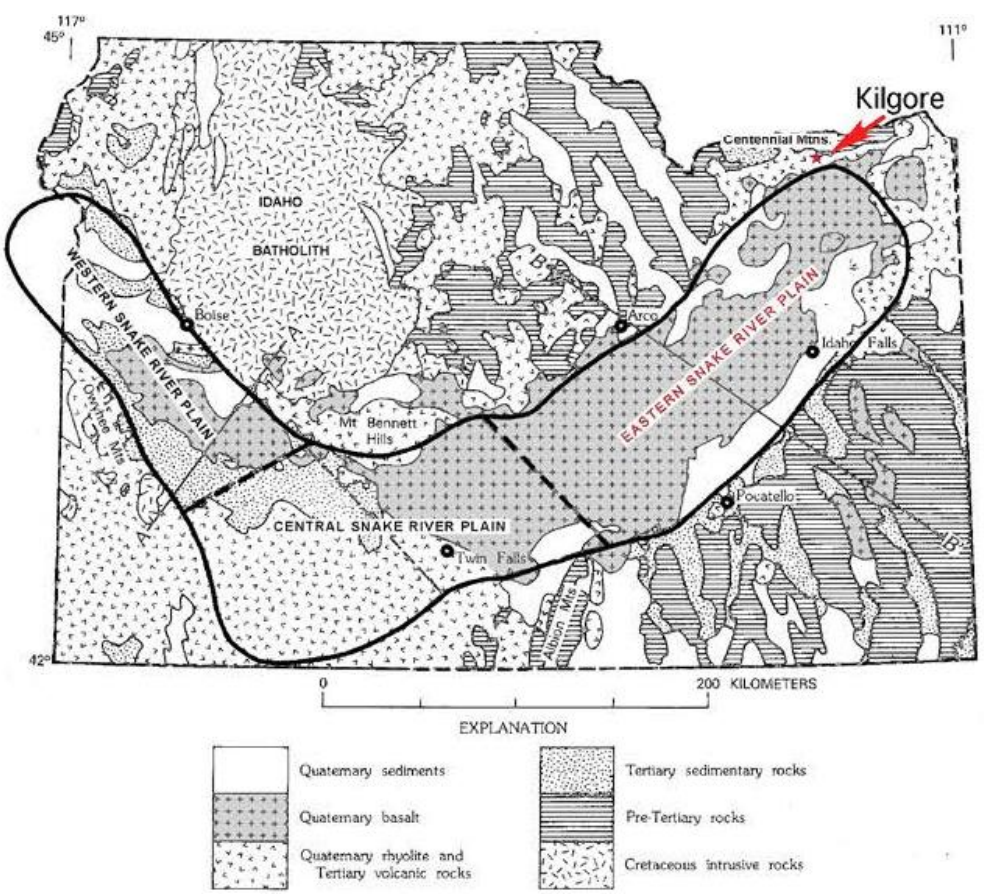
7.0 GEOLOGIC SETTING AND MINERALIZATION

7.1 Regional Geology

The Kilgore Project is located in the northeastern portion of the ESRP, locally situated to the south of the Centennial Mountains, and regionally along the northern margin of the Miocene-Pliocene Heise Volcanic Field. The following description of the regional geologic setting of the Kilgore Project is largely based on work completed by Leeman (1982), Mabey (1982) and Morgan and MacIntosh (2005), and much of the following text is modified and/or directly excerpted from those reports.

The ESRP is an arcuate depression of low topographic relief that extends more than 310.6 miles (500 kilometers [km]) across southern Idaho (Figure 7-1). The plain is distinguished from the surrounding terrain by lower elevation and surface relief and by a complete cover of Cenozoic sedimentary and volcanic rocks. Geologic relationships and recent radiometric dating have demonstrated that since middle Miocene time the ESRP-Yellowstone Plateau province has been characterized by voluminous bimodal rhyolite and basalt volcanism that has progressed eastward with time and is now focused at Yellowstone National Park. The development of this eastward younging bimodal volcanism is attributed to west-southwestward movement of the North American plate over a stationary melting anomaly, or plume-like zone of hot and molten magma rooted at least several hundred km below the surface (Leeman, 1982), commonly referred to as the Yellowstone Hotspot.

Figure 7-1: Regional Geologic Setting of the Kilgore Project



modified from Mabey, 1982

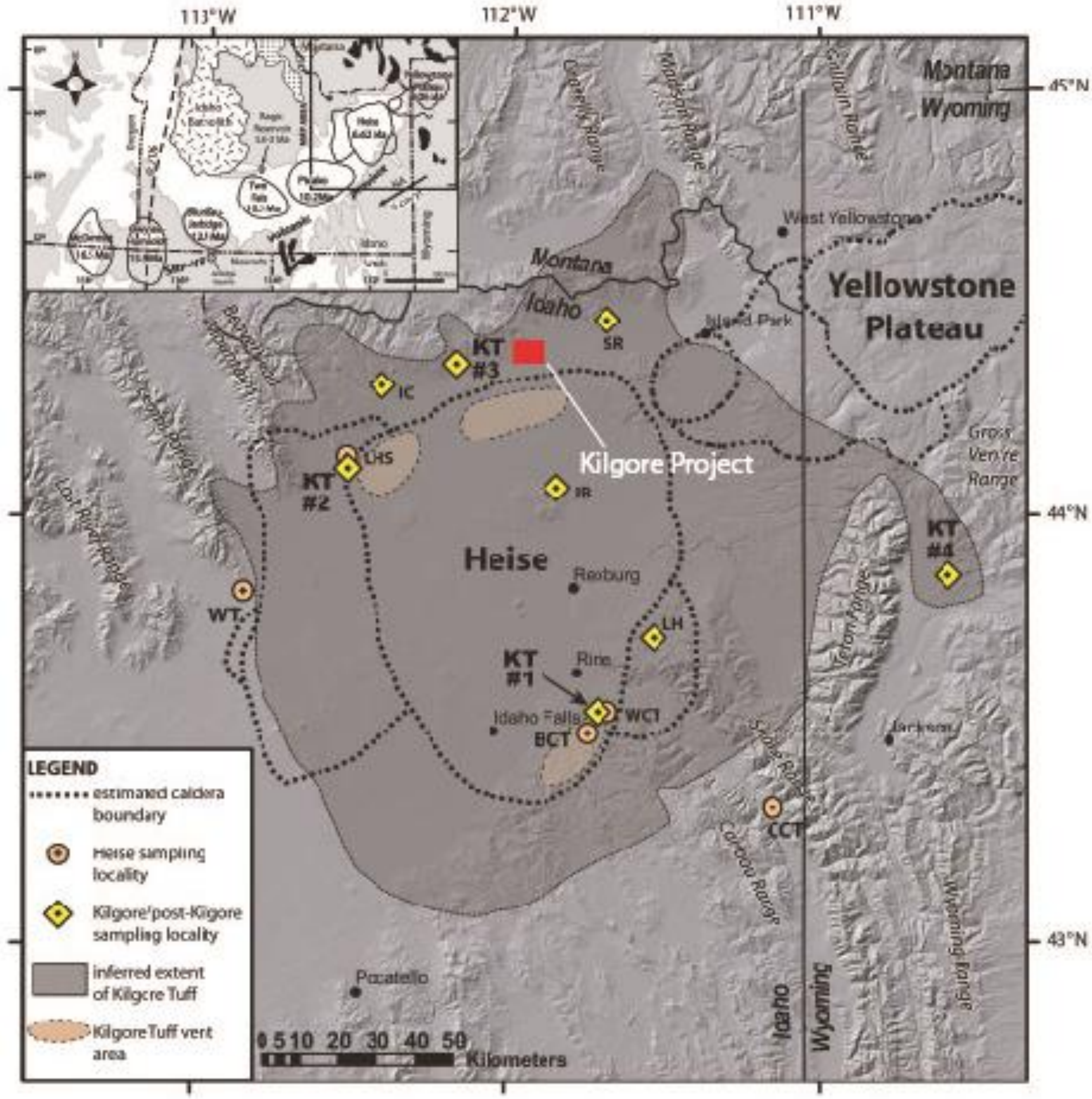
The central and northeast-trending ESRP is interpreted as a structural downwarp based on inward dipping attitudes of volcanic and sedimentary rocks along its margins and the lack of evidence for bounding faults. Most of the ESRP is covered by Quaternary basalt flows and low relief volcanoes that were active until approximately 2,000 years ago. Rhyolite domes ranging in age from 0.3 to 1.5 million years rise above the surface of the basalt in the central part of the eastern plain (Kuntz & Dalrymple, 1979). Rhyolite ash flows are common around the margins of the plain, and Cenozoic sediments are abundant in the surface and shallow subsurface near the margins of the plain. Although cracks and fissures with trends generally normal to or parallel to the axis of the plain are common, evidence of displacement along them is rare (Mabey, 1982).

Four major caldera-forming eruptions occurred within the ESRP during the late Miocene and early Pliocene. These calderas and their cogenetic ignimbrites form the framework of the Heise Volcanic Field, which is slightly older than but analogous to the adjacent (northeast) Yellowstone Plateau Volcanic Field. Field relations and high-precision 40Ar/39Ar age determinations (Morgan & MacIntosh, 2005) establish the four regional ignimbrites of the Heise volcanic field as the Blacktail Creek Tuff (6.62 ± 0.03 million years [Ma]), Walcott Tuff (6.27 ± 0.04 Ma), Conant Creek Tuff (5.51 ± 0.13 Ma), and Kilgore Tuff (4.45 ± 0.05 Ma; all errors reported at ± 2σ).

The Heise Volcanic Field consists of several overlapping or nested calderas that are now buried beneath younger sedimentary and volcanic deposits. Deposits of the Heise Group cover approximately 35,000 square km, with extensive exposures along the margins of the eastern ESRP (Morgan, 1992). Heise Group rocks lie stratigraphically above tuffaceous sediments of the late Tertiary Medicine Lodge Formation on the northern margin of the plain (Skipp, Prostka, & Schleicher, 1979) and above and in places overlapping tuffaceous sediments of the Salt Lake Formation on the southern margin (Allmendinger, 1982; Oriel & Moore, 1985; Love, 1986). Along the northern edge of the ESRP, rhyolites from the Heise Group are distributed from the Arco area in the southwest to southern Montana and Big Bend Ridge on the northeast.

The 431.8 cubic mile (1800 cubic km) Kilgore Tuff is the youngest and most voluminous of the four major caldera-forming eruptions in the Heise Volcanic Field. Exposures of the Kilgore Tuff span ~7,700 square miles (~20,000 square km) across parts of Idaho, Montana, and Wyoming, and range from less than 10 feet (three meters) to greater than 394 feet (120 meters) thick, with the thickest deposits located near three inferred source vent areas along the northern and southern margins of the Kilgore Caldera (Morgan & MacIntosh, 2005). According to anisotropy of magnetic susceptibility measurements and grain-size distributions (Morgan, 1988), the three source vents of the Kilgore Tuff are separated by lateral distances of approximately 31 to 62 miles (50 to 100 km); two of the vents are located in areas where the Kilgore Caldera overlaps the older Heise Caldera boundaries (Figure 7-2).

Figure 7-2: Local Geologic Setting of the Kilgore Project

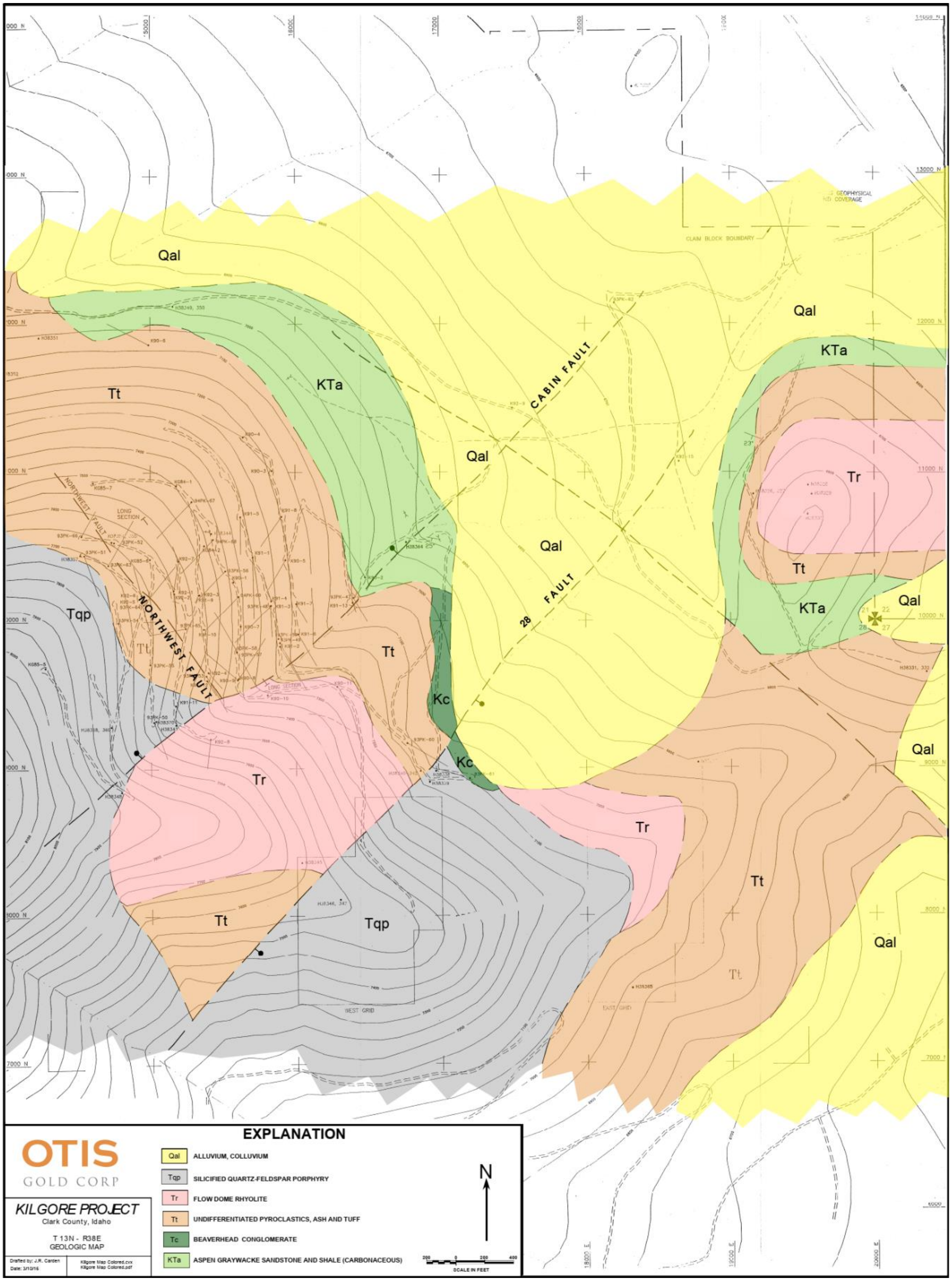


from Watts et. al, 2011

7.2 Local and Property Geology

The Kilgore Project is situated in an area of Miocene to Pliocene rhyolite flow-dome complexes and associated pyroclastic sequences along the northern margin of the ESRP, coincident with the northern margin of the Heise Volcanic Field and specifically with the interpreted north-eastern rim of the Kilgore Caldera Complex (Figure 7-2). The rhyolitic rocks unconformably overlie folded Cretaceous to early Tertiary clastic sedimentary rocks. Toward the project perimeter to both the north and south, the volcanic rocks are locally blanketed by the Tuff of Kilgore, a relatively distinct welded ash flow tuff thought to represent the last major eruptive event of the Kilgore Caldera. A geologic map of the Kilgore Project area is presented as Figure 7-3.

Figure 7-3: Geologic Map of the Kilgore Project Area



Modified from Benson, 1986

7.2.1 Lithology

The following paragraphs describe the individual lithologic units important to the Kilgore Project. Descriptions are based on distinguishing characteristics observed by Otis Gold geologists in hand specimen, outcrop, drill core, and road cuts. The local lithologies described here generally correlate well with published accounts of presumed equivalent bedrock formations in reasonably close proximity to the project area.

Aspen Formation (Ka)

Clastic sedimentary rocks of the Cretaceous Aspen Formation (Ka) are the oldest rocks known to occur in the project vicinity. These rocks outcrop in the northern portion of the project area but are more commonly present beneath thick volcanic cover and extend to the depths explored to date. The Aspen Formation, as it is referred to here, was formally described by Scholten et al. (1955), and later variably included in or as a member of the Late Cretaceous Frontier Formation (Mansfield, 1920), the upper (also Late Cretaceous) Kootenai Formation (Mitchell & Bennett, 1979), and the Late Cretaceous-to-Paleocene Beaverhead Formation (Witkind & Prostka, 1980). Total thickness of the Aspen Formation is unknown, but Scholten et al. (1955) reports a thickness in excess of 3,500 feet (1,067 meters) in the southeastern Tendoy Range, just 25 miles (40 km) west of the Project area. Contours of the top of the Aspen Formation based on drill hole data and outcrop indicate that the surface dips moderately to the southwest.

In outcrop, the Aspen Formation is generally gray to greenish-gray, strongly weathered, and pulverulent, with subtle to indistinct bedding. The individual sedimentary strata are comprised mostly of immature coarse- to fine-grained salt-and-pepper-textured lithic graywacke interbedded with lesser amounts of locally carbonaceous black siltstone and shale, all of which are locally calcareous. In drill core, the Aspen is represented by compact, thin-bedded layers of tightly packed and rounded to subrounded sand and silt particles within well-indurated and intercalated graywacke and siltstone sequences (Photo 7-1). Graywacke generally varies from light to dark gray and sometimes contains 0.08- to 0.16 inch (2- to 4-mm) dark streaks resembling carbon leaders or shale “rip-ups.” Coarser conglomeratic phases occur locally and are thought to have developed in stream channels cut into and preserved at the top of the Aspen. These conglomerates may represent the base of the Beaverhead Formation, as described by Witkind and Prostka (1980).

Repetitive fining-upward cycles are observed in the Aspen in drill core, with some of the rocks, particularly siltstones and shales, revealing considerable penecontemporaneous soft sediment deformation such as warping, slumping, and folding, along with slump breccias and chaotic textural intermixing. Locally, angular rip-up clasts of black siltstone and shale are present within graywacke; some graywackes have been “forcefully injected” by underlying siltstones and shales.

Gold mineralization occurs throughout the Aspen Formation as deep as drilling has been completed. The calcareous matrix of the arkosic sediment has been replaced by silica, adularia, and calc-silicate minerals, e.g., epidote; mineralization is also encountered at depth and away from the current resource area where the Aspen is intersected by fluid pathways such as the Cabin Fault or Mine Ridge Fault, and other east or northeast trending faults. Drill hole OKC-371 intersected a 75-foot (23-meter) interval of mineralization adjacent to the Cabin fault and at a vertical depth of 899 feet (274 meters) below the contact of the Aspen

Photo 7-1: Typical Aspen Formation (Ka) in Drill Core

and the overlying volcanic rocks. Mineralized Aspen Formation typically contains local quartz microveins, is iron-stained, and is occasionally cut by gold-bearing mafic dikes. Matrix replacement style mineralization is thought to be more common at depth within the Aspen in the vicinity of structural intersections.

Tertiary Andesite Flows (Tap)

Sill-like bodies occurs at the base of the volcanic pile where they lie both conformably and unconformably in contact with the underlying Aspen Formation arkosic sediments. Identified previously as the Tct (crystal tuff), Otis Gold geologists now apply the designation Tertiary andesite porphyry (Tap) and interpret them as faulted volcanic flows. This is applied to hypabyssal sills and dikes (faulted flows?) of intermediate, andesitic composition, parts of which have been subjected to quartz-adularia alteration subsequent to emplacement. Thin-section analysis of material from this unit shows none of the features found in a typical tuff such as welded glass shards. Thickness of the principal flows in the deposit area range between 100 and 300 feet (30 and 90 meters). A northwest-trending altered hornblende granodiorite intrusive body locally in excess of 300 feet (90 meters) wide and at least 500 feet (150 meters) long occurs northwest of the Mine Ridge fault and underlies the North Target area.

In drill core, Tap is green-gray to gray in color (Photo 7-2). Its porphyritic texture is typified by a fine grain matrix containing 5-10% rectangular feldspar phenocrysts ranging from 1 to5 mm in size, and 1 to 3% very fine grain, dark grey to black, elongate hornblende phenocrysts. The feldpsars are often selectively altered or replaced, and as a result Tap can exhibit a pitted texture where feldspar phenocrysts are argillically altered to clay and occasionally dissolved. Zones of strong silicification are common, often as flooding throughout the rock, but also as microveinlets, as crystals in vugs and fracture fillings, and as druzy fracture coatings. Quartz veinlets vary in composition from a translucent-grey quartz to a dense milky white variety. The typical greenish-grey color is attributed to a pervasive propylitic alteration of the fine grained matrix. Trace amounts of fine-grained, disseminated pyrite are common in the matrix, and are observed in drill core selectively replacing the feldspar phenocrysts. Pyrite, quartz-pyrite veinlets, and concentrated areas of disseminated pyrite blebs are frequently observed near the contacts with surrounding rock.

Photo 7-2: Andesite Flows (Tap)

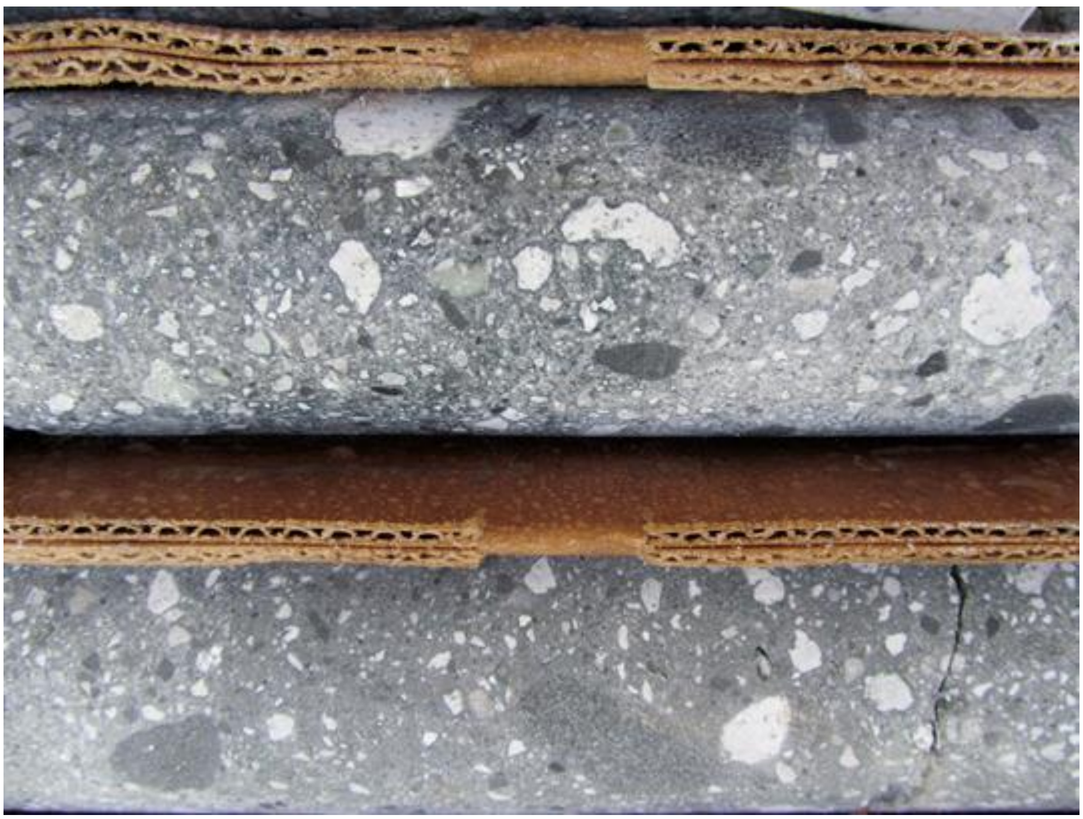
Based on similarity in appearance, it is often difficult to distinguish between the Tap and rhyolitic rocks of the group Tpr (described below) in drill core. The primary distinguishing factor relied upon by Otis Gold geologists is the presence or lack of rounded quartz eyes, which are absent in the Tap but considered somewhat ubiquitous of the Tpr.

In late 2018 Otis Gold submitted 491 samples to ALS global for analysis by whole rock, rare earth element, and trace element geochemistry; detailed review of the results indicate that analysis of Tct, Tad and Tap labeled samples places all these, and spatially associated rocks, within the field of andesite volcanic chemistry. In light of these results, Otis Gold geologists now characterize these rocks as intermediate andesite porphyry volcanic flows and represent part of the magmatic evolution of the Heise Volcanic Field. Previously interpreted sill like bodies may actually represent irregular caldera margin faulting and related fault bounded slump blocks.

Undifferentiated Tuff (Tlt)

The Ka sediments and Tap flows are unconformably overlain by a thick (locally in excess of 300 meters) layer of undifferentiated lithic and crystalline tuffs. This unit (Tlt) is comprised of a complex series of lithic lapilli tuffs, locally crystal-rich ash-fall tuffs, and other pumice-rich pyroclastic rocks, all of reportedly local extent (Benson C. , 1986). Together, these rocks represent a significant host of gold mineralization at Kilgore and are thought to be the products of activity associated with the Heise Volcanic Field, that is four successive super-volcanic eruptions between 6.62 and 4.45 Ma (Morgan & McIntosh, 2005). In general, the tuffs are composed of medium-gray, ashy tuffaceous material with variable amounts of lapilli-sized angular to sub-angular fragments of dark gray Aspen Formation sandstone and siltstone, pumice fragments, quartzite, and various volcanic and rhyolitic rocks ranging in size from 2 mm to 64 mm, as well as rare larger clasts greater than 64 mm in diameter (Photo 7-3). Lithic fragments are supported by an ashy matrix with broken fine-grained plagioclase crystals and quartz crystal fragments. Tuffs of the Tlt can be widely variable in appearance, largely due to the quantity and composition of the lithic fragments and degree of welding, but also as a result of superimposed quartz and quartz-adularia alteration and silica flooding that tends to bleach the rock from its characteristic medium-gray to a lighter gray or cream color in places.

Photo 7-3: Lithic Lapilli Tuff of the Tlt



Biotite Rhyolite (Tpr)

A series of flow-domes, plugs, and dikes of rhyolitic composition comprise the Tpr, which occurs in a wide northwest-trending belt or zone through the Project vicinity. This The rhyolite is reddish-brown to light pink in color with coarse flow foliation commonly present. The flows and domes generally have well-developed vitrophyric margins with local spherulitic and lithophysae zones. The rhyolite is composed of trace to 1% fine-grained biotite, 1% to 5% plagioclase and sanidine, and scattered fine-grained distinctive quartz eyes in an aphanitic, locally pilotaxitic groundmass (Photo 7-4). The presence of biotite is diagnostic where the rhyolite is fresh, though weathering, hydrothermal alteration, and bleaching frequently removes or alters most of the biotite, replacing it completely and/or liberating iron to form rusty-red and yellow-brown oxide stains. Age of the Tpr is 7.9 ± 0.4 Ma based on potassium-argon (K-Ar) age dating of biotite in a vitrophyre by (Benson C. , 1986).

Photo 7-4: Typical bleached, hydrothermally altered Tertiary biotite rhyolite (Tpr)

Rhyolite Quartz Porphyry (Tqp)

A thick, relatively crystal-rich rhyolite flow unit is encountered central to and northwest of the Kilgore Project area. This unit is described by Benson (1986) as a quartz porphyry lava erupted onto a highly irregular topographic surface. The unit designation reflects the presence of distinctive quartz phenocrysts throughout the rock, which is generally massive with occasional local coarse flow foliation. Tqp is commonly light gray with 2% to 10% coarse-grained quartz and sanidine phenocrysts in a micro-spherulitic groundmass (Photo 7-5). Some of the phenocrysts are at least 4 mm in diameter, with many of the quartz phenocrysts containing central inclusions of what appear to be microphenocrysts of alkali feldspar (Benson C. , 1986).

Exposures of the Tqp within the Project area form coarse talus slopes and craggy, resistant outcrops of highly silicified material. The unit includes a basal spherulitic vitrophyre that is intensely clay altered and thought to have acted as a partial barrier to the ascending mineralizing fluids that created the Kilgore deposit. The unit attains a thickness of as much as 600 feet (180 meters) based on an intersection in Bear Creek drill hole KG-3.

According to Benson (1986), local stratigraphic relations and K-Ar age dates of the Tpr and of an aphyric rhyolite in the extreme northeastern and southwestern parts of the Kilgore area constrain the age of the Tqp between 5.9 ± 0.3 Ma (younger aphyric rhyolite) and 7.9 ± 0.4 Ma (older Tpr). The rhyolite of Spring

Creek is a similar, well described rhyolite flow (Morgan, Doherty, & Leeman, 1984) that outcrops to the northeast of the Kilgore Project. Otis Gold considers the rhyolite of Spring Creek, K-Ar age dated at 6.3 ± 0.3 Ma (Morgan, Doherty, & Leeman, 1984), to be correlative with the Tqp.

Photo 7-5: Tertiary Rhyolite Quartz Porphyry (Tqp)

Upper Pyroclastics (Tup)

A discrete and distinct upper pyroclastic unit (Tup) is mapped over a surface area of approximately one square mile just southwest of the Kilgore deposit. This unit is comprised of hot-spring sinter material, silicified explosion breccia, crumble breccia, clast-supported breccia, and unsorted, non-bedded to poorly-bedded lithic breccia at least 300 feet (90 meters) thick (Photo 7-6). It forms the capping stratigraphic unit at the top the southwest-dipping plateau just up-slope and southwest of the Kilgore deposit. The Tup is a relatively widespread and well-preserved silica cap and silicified explosion breccia layer that represents the surface expression of a hot spring-type epithermal system. The Tup overlies much of the Dog Bone Ridge target area, and it is interpreted by Otis Gold as a large vent zone that has broken through an existing silica cap to form a fallout apron above the Tqp unit. Otis Gold's interpretation is based in part on the composition of the silicified explosion breccia, which includes local fragments of coarse sand-sized silicified material, Tqp, Tpr, and much clast-supported breccia. Berger and Eimon (1982), Silberman (1982), and Berger (1985), describe similar sinters, breccias, and fallout aprons related to numerous other classic hot spring epithermal systems and related precious metals deposits. Morgan et al (2009) describe very similar deposits that occur as the result of large hydrothermal explosion breccias above the Yellowstone magma chamber.

Photo 7-6: Sinter and Explosion Breccia of the Tup Exposed in Outcrop

Tuff of Kilgore (Ttk)

The tuff of Kilgore is a widespread welded ash-flow tuff that forms a gently southwest-dipping (less than 5°) blanket over the rhyolitic and pyroclastic strata of the Kilgore Project to the south, west, and northwest of the Project area. The tuff of Kilgore is post mineral, as it mantles the hydrothermally altered rocks of the Kilgore mineralizing system but lacks hydrothermal alteration itself. The average age of the tuff of Kilgore is 4.45 ±0.05 Ma (Morgan & MacIntosh, 2005), while a K-Ar age determination on hydrothermal adularia at the Kilgore deposit dates mineralization at 5.3 ± 0.2 Ma (Benson C. , 1986).

The tuff of Kilgore is generally purple gray to dark reddish-brown in color, with 1% to 7% medium-grained crystals of sanidine, plagioclase, and rare quartz in a very fine-grained glassy, locally devitrified matrix. Unit thickness is locally up to 150 meters (500 feet), with a black to reddish-brown vitrophyric base that is up to 12 meters (40 feet) thick (Benson C. , 1986). The Ttk is moderately to strongly welded, generally eutaxitic, and locally rheomorphic, with strong lineation.

Petrographic, radiometric, and field studies reveal that the tuff of Kilgore is equivalent to the tuff of Heise, which has a K-Ar age date of 4.3 ± 0.15 Ma (Armstong, Harakel, & Neill, 1980). The tuff of Heise crops out within the neighboring Rexburg Caldera complex. Embree and others (1982) suggest that the caldera source for both tuff units is near Kilgore, Idaho, with the tuff of Heise representing the distal facies of the tuff of Kilgore where the former ponded in the Rexburg Caldera complex.

7.2.2 Structure

Three main structural trends are recognized in the Kilgore deposit area: 1) Azimuth 300°-320°, 2) 030°-060°; and 3) 090°-110°. These are corroborated by detailed local geological and structural mapping by Benson (1986), field structural investigation work by Caddey (2003), detailed local geological and structural mapping by Echo Bay geologists in 1995 (unpublished), and recent geological and geophysical field studies by Otis Gold including the 3D geologic modeling of drill data by Rowearth in the course of the preparation of the 2018 mineral resource model (GRE & Rowearth, 2018).

To date the dominant hypothesis of structural control of mineralization has centered upon 300°-320° structures and this was evidenced by: 1) the direct association of the emplacement of gold mineralization at the Kilgore deposit with the Northwest Fault and a sub-parallel suite of structures; 2) additional scattered areas of gold mineralization and voluminous silicified vent breccia associated with the Dog Bone Ridge target area, localized along the northwesterly extension of the 300° trending McGarry Canyon Northwest Fault. The McGarry Canyon Northwest Fault lies approximately 2,500 feet (760 meters) southwest of and parallel to the Northwest Fault (Figure 7-3).

The Northwest Fault and a number of parallel structures comprise a 300°-320° trending fault zone (Figure 7-4) at least 6 km long containing: 1) the Kilgore deposit area; 2) its southeasterly extension into the Prospect Ridge area; and 3) Otis Gold's Gold Ridge target located approximately 3,300 feet (1.0 km) northwest of the deposit. Overall, the trend of this fault zone is characterized by the emplacement of a belt of shallowly-emplaced rhyolite plugs, granitoid dikes, and domes, and granodioritic bodies; this also coincides with the geophysical interpretation of the 3.4-mile- (5.5-km-) long arcuate toe of a volcanic terrace composed dominantly of lithic tuff (Tlt) that approximately parallels this fault trend. The Northwest Fault zone is partially or completely capped by Tqp, much of which is highly silicified (Benson C. , 1986). Geologic 3D modeling of the deposit carried out in the course of resource modeling by Rowearth (GRE & Rowearth, 2018) used information gathered from geologic logging of drill cores; from that modeling, a suite of sub-parallel faults was identified that hosts a significant portion of the mineralization in the Kilgore Deposit.

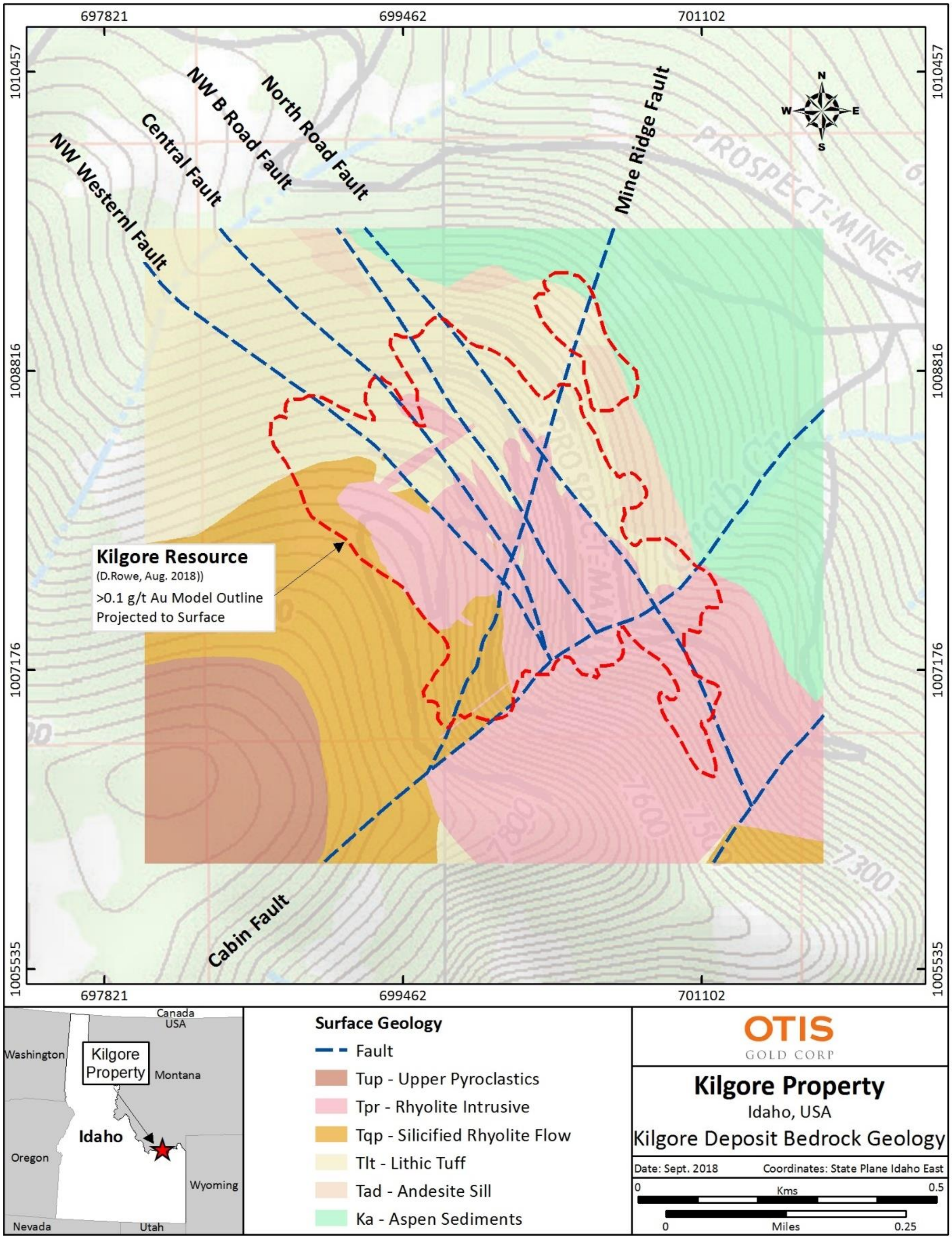
The McGarry Canyon Northwest Fault comprises a zone at least 1.8 miles (3 km) in length that includes a northwest-trending silicified vent zone and hydrothermal fluid conduit with related explosive pyroclastic volcanism, rhyolitic volcanism, dike emplacement, and epithermal activity (explosion breccia and sinter) in the Dog Bone Ridge area (Caddey, 2003). Surface exposures along the central part of Dog Bone Ridge consist of linear, siliceous, tectonic, and phreatic explosion breccias localized along a 1 mile (1.6 km) length of the McGarry Canyon Northwest Fault. Erosion-resistant surface outcrops forming the ridgeline are intensely silicified, brecciated, and healed with at least three generations of low-temperature varieties of chalcedonic and opaline quartz (Caddey, 2003).

Northeast of the Kilgore deposit, the West Camas Creek drainage may be the expression of a ring fracture related to the Kilgore Caldera margin; these northwest-trending structures may represent structures that are contemporaneous with the development of the Heise Volcanic Field and therefore development of the Kilgore Caldera. The recently identified toe of a volcanic terrace also approximately parallels the arcuate trend of the West Camas Creek drainage, implying that it too is controlled by ring structures associated with the development of the Kilgore Caldera. Interpretations based on Otis Gold drilling and surface geophysics suggest that these structures served as the secondary hydrothermal fluid conduits and focus for the emplacement of northwest-trending intrusive dikes with which the gold mineralization appears to be directly associated, though this has yet to be confirmed.

Prominent 030°-060° trending structures in the Kilgore deposit area include the Mine Ridge Fault, Cabin Fault, Bear Cat Fault system and 28 Faults. Figure 7-4 The Kilgore Deposit gold-silver mineralization appears to be associated with a complex network of cross-cutting radial faults and ring structures; the radial faults have high-grade mineralization associated with them that was initially exploited during early development of the Kilgore Deposit. The ring structures and sub-parallel fault swarms may have acted as secondary fluid pathways for the distribution of hydrothermal fluids into the receptive host rock units that constitute the Kilgore Deposit, i.e., the lithic tuffs (Tlt), quartz porphyry intrusives (Tpr), and andesite flows (Tap) - the contact zone between the lithic tuff and underlying Aspen Formation, and sub-vertical northwest trending faults.

Gold grade-thickness maps and Echo Bay geophysical/airborne magnetic data (Woolham, 1996) suggest the presence of an 090°-110° structure that crosses the heart of the Kilgore deposit into the West Camas drainage. The regional and local context of this apparent structure is not clear.

Figure 7-4: Plan View of the Kilgore Resource Area



7.2.3 Mineralization and Alteration

The following description of mineralization and alteration specific to the Kilgore Project is largely excerpted and/or modified from Cameron (2012). It has been modified and updated to reflect exploration carried out from 2012 through 2018 (GRE & Rowearth, 2018).

Mineralization

Gold mineralization at Kilgore occurs within two suites of receptive host rocks: 1) in rocks of volcanic or subvolcanic origin, including the Tlt, Tap, and the sub-vertical granitoid dikes, dike swarms, and granite to granodioritic bodies that intrude it, and 2) the sedimentary turbidites composed of arkosic sandstones and siltstones, and carbonaceous shales of the Aspen Formation.

Gold mineralization in the volcanic and related intrusive rocks contains high grade zones as a result of weak to moderate vein development and open space fracture-fill, together within a broad, low grade halo of disseminated gold within variably silicified and argillically altered rocks. Gold content appears to decrease rapidly to lower grades (less than 50 to 100 parts per billion [ppb] gold [Au]), with corresponding decrease in quartz or quartz adularia as silicification and increase in argillic alteration. Exceptions occur in strongly oxidized rock near the topographic surface where strong to pervasive iron-oxide, yellow-orange to brown staining is accompanied by high gold grades.

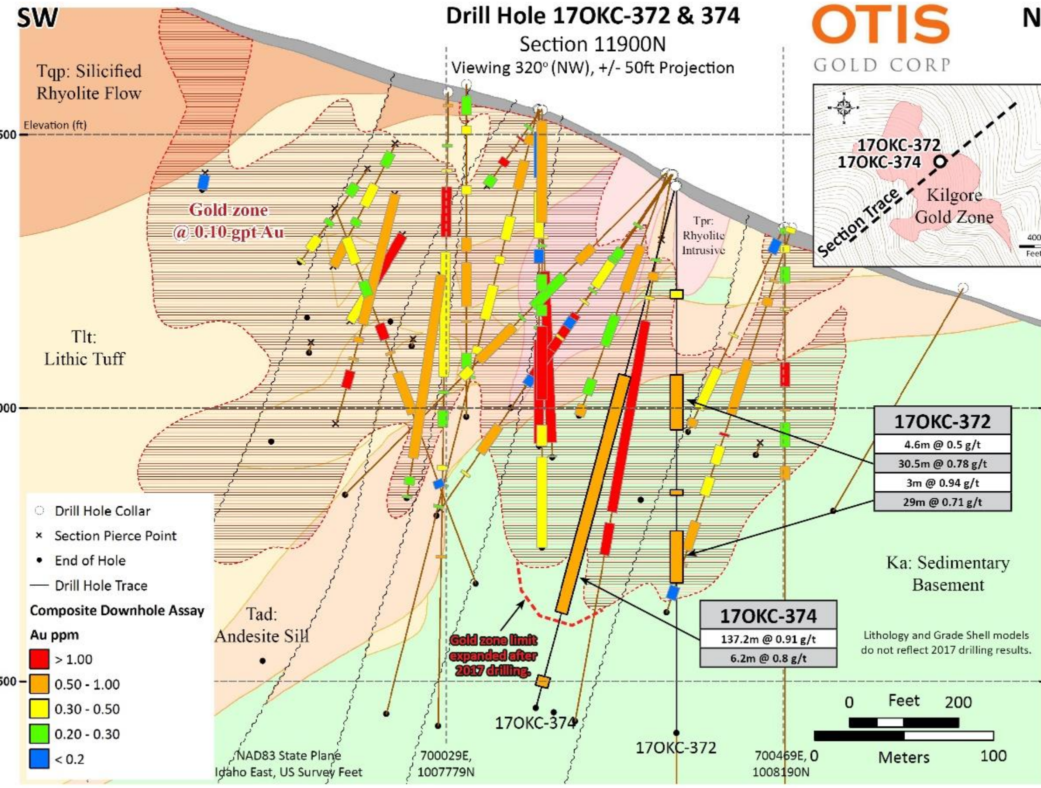
Mineralization in the volcanic and associated intrusive rocks accounts for an estimated 85% of the known mineral resource, with the remaining 15% occurring in the underlying Aspen Formation sediments.

The Aspen Formation sedimentary rocks are arkosic sandstones with irregularly distributed carbonaceous shale layers; gold mineralization occurs associated with quartz, adularia, and epidote and appears to have been the result of replacement of the carbonate cement of the arkosic sandstone. Recent petrologic work carried out by Otis Gold has revealed 5 to 15 micron (µm) sized gold particles between quartz and adularia crystals that now cement the quartz and feldspar grains of the arkosic sandstone. To date, a limited number of drill holes has tested the sediment hosted mineralization. Further testing of the Aspen Formation sediment hosted mineralization is necessary to assess the potential as a future target for bulk tonnage, low-grade mining operations.

The Kilgore deposit is a zone of mineralization approximately 2,600 feet (800 meters) long, 2,000 feet (600 meters) wide, and 1,070 feet (325 meters) deep from ground surface to the maximum inferred mineral resource depth. Mineralized intercepts generally average 130 feet (40 meters) and range up to 300 feet (90 meters) in thickness in the Mine Ridge core and North Target areas. Figure 7-5 through Figure 7-7 illustrate the distribution and orientation of mineralization of the Kilgore deposit in cross section and relative to individual intercepts and host rock geology and structure.

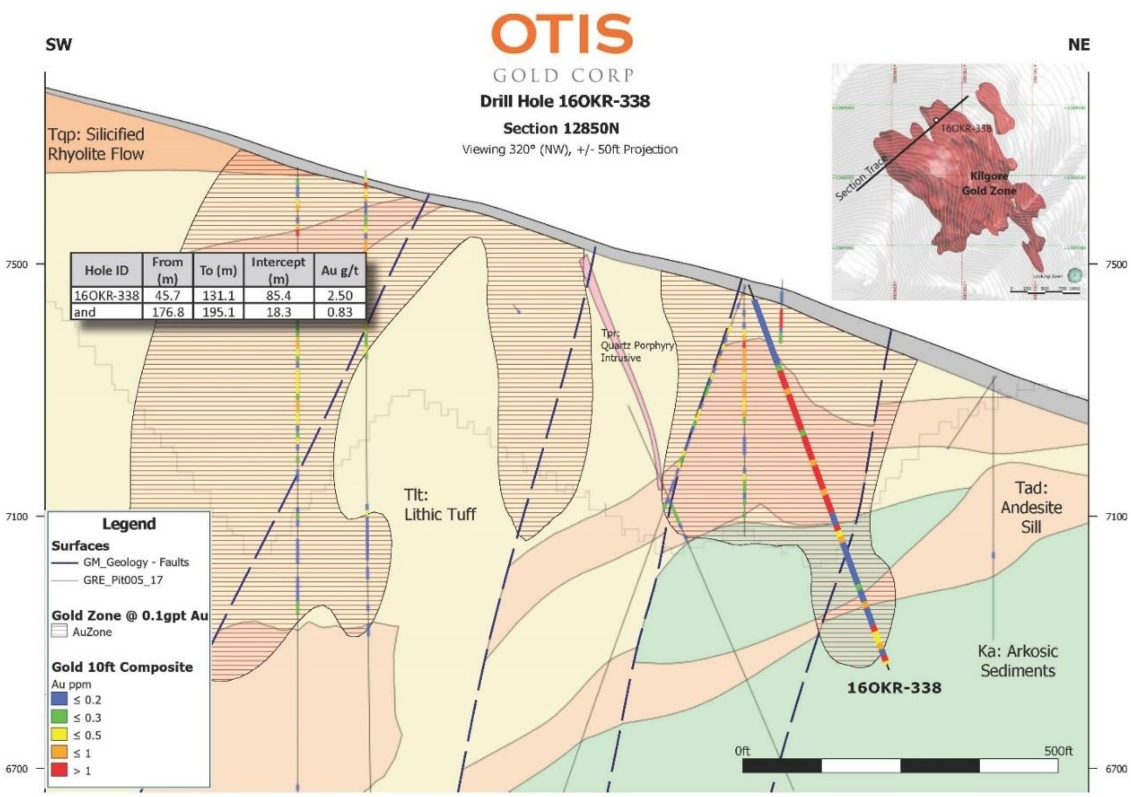
Significant mineralized zones within the Project area are typically associated with structures and the mineralized halos around them in the surrounding rocks. The geology and apparent detailed mineralization controls vary from one area to the next. Mine Ridge comprises the core of the Kilgore deposit and the bulk of the gold mineralization contained within it. Major geologic controls include the northwest trending fault swarms and the northeast trending radial fault structures. Gold mineralization is

Figure 7-5: Representative Cross Section of the Kilgore Resource Area, Section 11900N



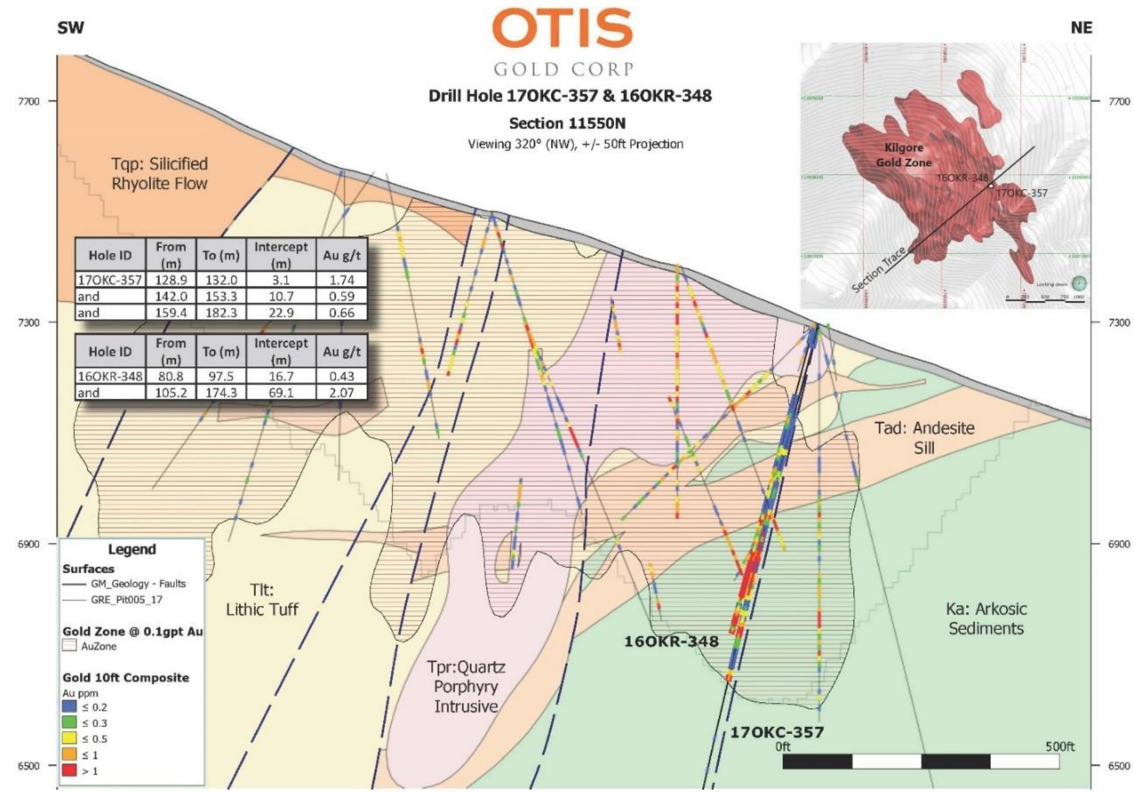
Otis Gold, 2018

Figure 7-6: Representative Cross Section of the Kilgore Resource Area, Section 12850N



Otis Gold, 2018

Figure 7-7: Representative Cross Section of the Kilgore Resource Area, Section 11550N



Otis Gold, 2018

spatially associated with the intrusive rocks and their contacts with porous and permeable lithic tuffs. Some of the higher-grade mineralization is localized in sub-vertical to vertical fissure, shear, and fault/fracture zones. The Tlt hosts significant disseminated mineralization that forms more extensive zones away from the dike contacts.

The system of northwest trending faults in the areas of the north and B roads (locally identified at the north road/rhyolite fault, northwest B-Road fault, northwest central fault, and northwest western fault) may represent a fault or shear zone several hundreds of meters wide and comprised of several sub-parallel structures. These faults and the cross-cutting Cabin, Mine Ridge, and other east-northeast trending faults contribute significantly to the overall distribution of mineralization.

The upper 100 to 200 feet (30 to 60 meters) of the Aspen Formation serves as a major host environment to gold mineralization in the Mine Ridge area, especially at the upper contact of the Aspen Formation with the overlying Tap. Here the unit is variably silicified, displaying quartz microveining, development of iron oxides along micro fractures, oxidation of sulfides, the presence of pyrite stringers, and chlorite, ankerite alteration. Quartz veins and sheared quartz vein zones cutting Aspen rocks, as well as the edges and margins of mafic dikes intruded into the Aspen Formation, all serve as environments for the deposition of higher-grade gold values. Typical thicknesses and grades of mineralized intercepts found in the upper part of the Aspen Formation in Otis Gold core holes include:

- 100 feet (30.4) meters @ 0.074 opt (2.53 g/T) Au from 283 feet (86.3 meters) to 382 feet (116.7 meters) in 10 OKC-210

- 109 feet (33.1 meters) @ 0.037 opt (1.27 g/T) Au from 676 feet (206.0 meters) to 784 feet (239.1 meters) in 11 OKC-253.

Core drilling in 2017 demonstrated significant lengths of low to moderate grade mineralization below the contact, including 17OKC-356, which returned an average 0.048 opt (1.66 g/T) Au over 424.5 feet (129.4 meters), including 80 feet (24.4 meters) averaging 0.10 opt (3.45 g/T) Au, and 17OKC-374, which returned an average of 0.027 opt (0.91 g/T) Au over 450 feet (137.2 meters).

Sulfide and precious metal mineral species identified in core from the Mine Ridge area include pyrite, electrum, native gold (some visible), galena, arsenopyrite, sphalerite, stibnite, cinnabar, naumannite (Ag_2Se), aguilarite (Ag4SeS), argentite-acanthite, chalcopyrite, wolframite (ferberite), and rare pyrargyrite (Ag_3SbS_3) (Benson C. , 1986; Otis Gold Corp., 2012). Benson's (1986) scanning electron microscope and energy dispersive X-ray spectrometry studies on heavy mineral concentrate grains from historic Bear Creek drill samples in the area found additional mineral species, including an unknown mineral composed of Ag-Pb-Bi-Se-S, galena with minor selenium and silver, gersdorffite (NiAsS), and cobalt-nickel-pyrite ($(Ni, Co, Fe)S_2$) with minor chalcopyrite. All of the silver minerals, electrum, and gold occur as discrete grains and within pyrite. Panned concentrate studies of gold grains conducted by Hazen Research, Inc. (1995) for Echo Bay found spongy, lacy, rectangular, splinter, and amoeboid morphologies with rich yellow color and sizes in the 25- to 150-micron range.

Petrographic work in early 2019 on selected samples from 17OKC-379 have revealed an association between jarosite and gold in lithic tuffs whereby small gold inclusions occur within jarosite masses. The mineral jarosite is the most common member of the alunite group; it is a hydrated potassium-iron sulfate typically associated with high sulfidation epithermal or porphyry associated epithermal systems and is formed by the direct oxidation of iron sulfides.

Gangue minerals identified in the area are mostly quartz, adularia, jarosite, manganiferous siderite, pyrite, pyrrhotite, arsenopyrite, illite-sericite, kaolinite, barite, and dumortierite/tourmaline.

<u>Alteration</u>

Throughout the Kilgore Project area the felsic and intermediate volcanic and intrusive rocks, volcaniclastic rocks, epiclastic rocks and underlying Tertiary-Cretaceous sedimentary rocks are pervasively altered – that is to say that all rocks in the project area exhibit a degree of hydrothermal alteration varying from strong, through moderate, to weak, but pervasive alteration. There are few examples of fresh unaltered rocks within the whole Kilgore Project area.

Quartz forms the dominant hydrothermal alteration mineral as silicification and often occurs with opaline silica in the major rock types. Silicification generally occurs as aggregates of very fine to coarse grained quartz with a mosaic texture. Silicification is also present throughout the Mine Ridge core area as irregular quartz veins, quartz stockwork vein zones, sheared quartz-vein zones on and along dike margins, quartz microveinlets and microveinlet zones, and late-stage cavity-filling quartz crystals, locally coated with rare, late-stage, visible gold. Some visible gold grains occur in oxidized selvage material along the margins of late-stage quartz veins (Photo 7-7). Some of the best developed areas of quartz veining generally occur along the margins of the northwest-trending dikes in the Mine Ridge area, along and parallel to the northeast-trending Mine Ridge fault, and near, and at the intersection of northwest- and northeast-trending structures.

Photo 7-7: Visible Gold in Late-Stage Quartz Veinlet in Drill Hole 08 OKC-193

Adularia is a common alteration mineral, often occurring with quartz or fine-grained silica. Felsic-to intermediate dike rocks commonly show pervasive quartz–adularia alteration and replacement, and some adularia occurs on fractures (Larabee, 2012; Benson C. , 1986).

Dumortierite, commonly found in the Mine Ridge area and logged in numerous historic core holes as tourmaline, is closely associated with higher-level quartz stockwork vein zones and also exists as radial sprays and needle-like replacements of spherulites in vitrophyre at the base of Tqp where it caps the deposit in the Mine Ridge area. It occurs distal to, and along the margins of, silicified and mineralized quartz-vein material in the upper 100 to 300 feet (30 to 90 meters) of core holes 95 EKC MET-5 and 08 OKC-191, near the intersection of the Northwest Fault zone and Mine Ridge fault. Replacement of feldspars and matrix material in Tlt is also a common mode of occurrence. The species generally forms blue-green radiating bundles, spots, and clots of acicular crystals with an average grain size of less than 1 mm. An analog for the occurrence of dumortierite at Kilgore is in the gold-dominant areas of epithermal precious metal deposits of the Rochester District, Nevada (Knopf, 1924).

Other alteration minerals reported by Otis Gold include illite, sericite, kaolinite, chalcedony, opal, amethyst, barite, calcite, Mn-Fe carbonate minerals, limonite, goethite, and hematite. Sericite is a generic term for fine-grained white mica or ordered clay mineral. Limonite is a mineral nomenclature typically used by geologists to describe occurrences of yellow to orange-brown colored oxidation products rather than a formal mineral name, sensu stricto. Jarosite, a yellowish primary or secondary iron-aluminum sulfate, has also been observed in petrological analysis (see Section 13.6.1) and probably accounts for most of the descriptions of Limonite within the Tlt.

Kilgore shows typical alteration zoning from proximal quartz-adularia through argillic to distal propylitic. Major alteration types commonly found in the Mine Ridge area include quartz-adularia, silicification, argillic, propylitic, and tourmalinization (in part dumortierite, see above).

Quartz-adularia is a dominant alteration style in parts of the Kilgore deposit, where it is present as flooding, a component of quartz veinlets, and in breccias, as well as local fracture coatings. Recent petrologic work outlined in Section 13.4 of this report shows that quartz-adularia alteration is associated directly with gold mineralization. Quartz, adularia, epidote, and chlorite have replaced the matrix of the arkosic sandstone. Less altered sandstones observed on the property have a carbonate cement composed dominantly of calcite, and it would appear that alteration has replaced it with quartz-adularia. It is possible that the calcite is a form of advanced propylitic alteration that preceded the quartz-adularia alteration by filling the primary pore spaces in the arkosic sands; calcite veining is commonly observed in drill core associated with chlorite and calc-silicate minerals away from zones of intense quartz-adularia alteration.

Argillic alteration includes bleaching of host rocks in the Mine Ridge area, with the development of illite, sericite, and kaolinite (Larabee, 2012). Feldspars and groundmass in tuff and dike host rocks are commonly partially to completely replaced, with feldspar crystals, revealing crystal casts and ghost crystal outlines where nearly totally replaced. In general, the extent of argillic alteration ranges from pervasive to structurally controlled replacement, depending on the host rock.

Propylitic alteration is mostly evident deeper in Otis Gold core holes, particularly in the sediments, as calcite, chlorite, epidote, and pyrite. These minerals mostly occur on fractures and as disseminations, particularly throughout parts of Tad, in gold-bearing hornblende andesite bodies, and in altered Aspen Formation siltstones and sandstones, imparting a light to dark green color to them. Detailed logging of Otis Gold core holes by staff geologists reveals that groundmass and mafic phenocrysts in late-stage, gold-bearing mafic dikes are also commonly chloritized, particularly where they cut the Aspen Formation and, as a result, are barely distinguishable from the latter.

The geochemical signature of Kilgore is consistent with an epithermal chemical signature, one high in gold, arsenic, antimony, mercury, and selenium. Arsenic exhibits the strongest correlation to the deposit, where it is clear that there is a significant arsenic anomaly on top of, and down-slope from the deposit, as well as along major northeast trending faults.

8.0 DEPOSIT TYPES

The Kilgore deposit is a zoned, low sulfidation (LS) epithermal hot spring precious metals (Au, silver [Ag]) deposit the result of caldera-related volcanic activity. These deposits are commonly bulk-tonnage, low-grade, and amenable to open-pit mining. Numerous scientific articles have been written and published on this deposit type concerning its origins, physical, chemical, and geological settings, recognition criteria, major- and trace-element geochemistry, zoning, alteration types, ore mineralogy, ore grades and distribution of ore, and mining characteristics. Models are described in papers by Buchanan (1981), Silberman (1982), and Berger (1985), among others, and the reader is referred to these for more information on the subject.

Epithermal hot spring-type precious metal (Au, Ag) deposits form at low to moderate temperatures in the near-surface environment. They generally form at depths of less than 5,000 feet (1.5 km) and temperatures of less than 300° C in subaerial environments within volcanic arcs at convergent plate margins, intra- and back-arc settings, and in post-depositional settings (Robert, et al., 2007). Epithermal deposits are found in all rock types, but, historically, some of the largest occur as disseminated bulk-tonnage and/or stockwork-type vein deposits in volcanic rocks (e.g., Round Mountain, Nevada; McDonald Meadows, Montana). Nearby deposits of this type are the Grassy Mountain deposit, Oregon, and the deposits at Sunbeam, Idaho (Grouse Creek and Sunbeam). Active geothermal areas such as Steamboat Springs, Nevada, and Broadlands, New Zealand (White, 1974), and Norris Geyser Basin in Yellowstone Park, Wyoming, are modern-day analogs of epithermal hot spring-type precious metal deposits that are currently forming.

Kilgore presents distinct features common to many epithermal hot spring deposits such as:

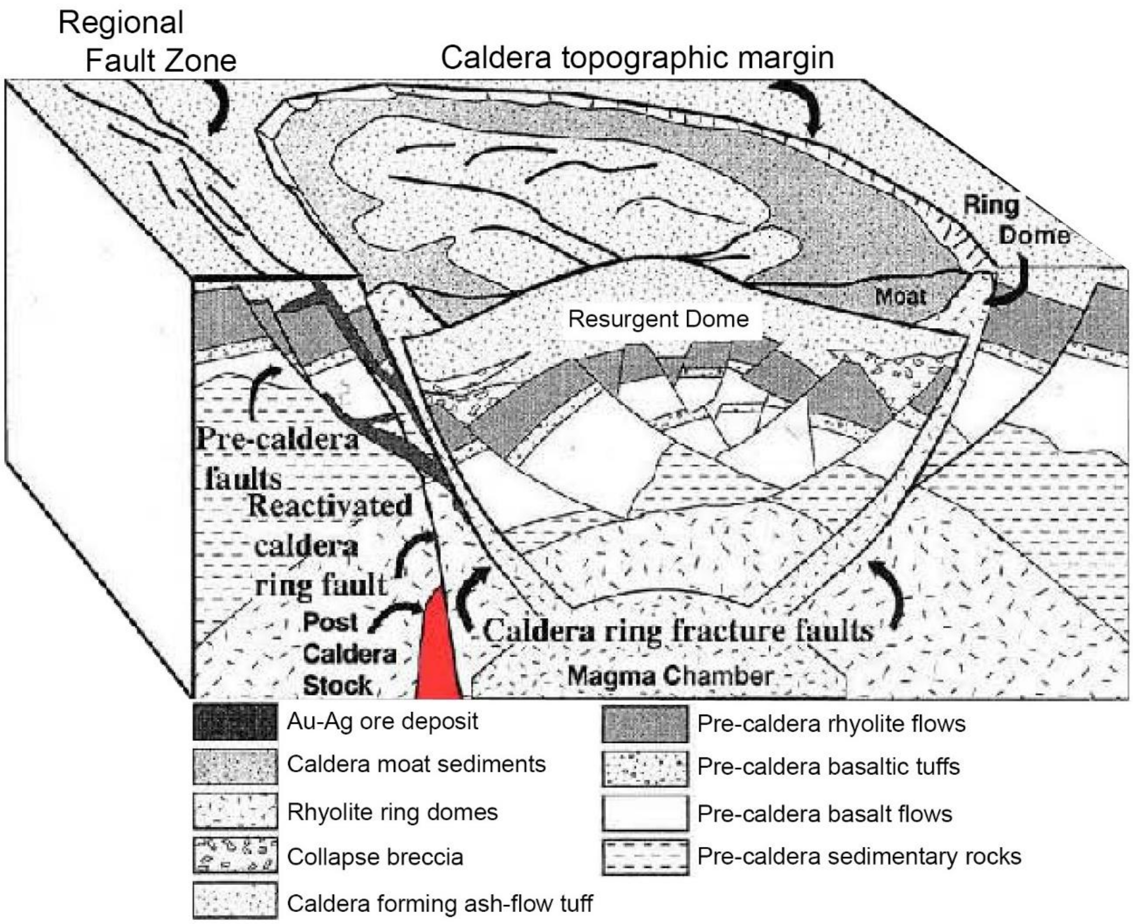
- Relationship to caldera activity and structures (Rytuba, 1994)
- Extensional structural environments characterized by high-angle normal faulting and/or dilational zones proximal to strike-slip fault structures
- Proximity and temporal association with shallow rhyolitic intrusions and effusive volcanic centers.

Figure 8-1 presents a schematic representation of the structural and volcanic features related to the deposition of epithermal precious metals deposits in a caldera-related environment.

At the deposit scale, LS gold deposits can be hosted in volcanic, volcaniclastic, lacustrine, or epiclastic units, but can also be hosted by volcanic or sedimentary rocks, as at Kilgore. Both low-grade disseminated and structurally controlled high-grade deposits can form, e.g., the Mine Ridge fault zone. Syn-mineral mafic dikes are also common in these deposits (Sillitoe & Hedenquist, 2003) and they occur at Kilgore where they host high-grade mineralization.

Mineralization at Kilgore comprises an epithermal assemblage of small quantities of pyrite, electrum, native or free gold, silver, mercury, and base metal sulfides, sulfosalts, and selenides. Otis Gold considers the recognition criteria for epithermal volcanic-hosted LS model (Table 8-1) essential to its exploration of the Kilgore deposit.

Figure 8-1: Structural and Volcanic Features Related to Deposition of Epithermal Precious Metal Deposits in a Caldera-Related Environment



From Rytuba, 1984

Table 8-1: Comparison of LS-Type Deposit Model and Kilgore Deposit Recognition Criteria

Recognition Criteria Type	LS-Type Deposit Model Recognition Criteria	Kilgore Deposit Recognition Criteria
Deposit Form/Styles of Mineralization	Disseminated and structurally controlled mineralization, open-space veins (high grade) and stockwork mineralization common	Disseminated and structurally controlled mineralization, late open-space quartz veins (some with high grade Au) and high-angle fractures, quartz stockwork veining in Mine Ridge core area
Textures	Veins, cavity fillings (bands, colloforms, druses), breccias	Disseminations, veins and shear zones (on edges of high-angle dikes), hydrothermal breccias, minor banded quartz veins, silica replacement and fracture filling, stockworking, fine-grained quartz-adularia flooding and replacement, quartz microveining
Ore Mineralogy	Pyrite, electrum, gold, sphalerite, galena, arsenopyrite	Pyrite, electrum, native gold (some visible), galena, arsenopyrite, sphalerite, stibnite, cinnabar, naumannite, aguilarite, argentite-acanthite, chalcopyrite

Recognition Criteria Type	LS-Type Deposit Model Recognition Criteria	Kilgore Deposit Recognition Criteria
Gangue Mineralogy	Quartz, adularia, chalcedony, calcite, illite, carbonates	Quartz, adularia, illite, sericite, kaolinite, chalcedony, opal, calcite, chlorite, pytire, tourmaline (dumortierite), iron oxides and hydroxies (limonite, goethite, hematite), amethyst, barite, carbonates (manganosiderite?)
Geochemical Suite	Gold, silver, zinc, lead, copper, tin, arsenic, mercury, selenium	Gold, silver, zinc, lead, arsenic, tin, boron, mercury, copper, selenium
Alteration	Quartz-adularia through argillic to distal propylitic	Quartz-adularia through argillic to distal propylitic, silicification, tourmalinization, pyritization, chloritization
Host rocks	Volcanic and basement sedimentary rocks	Volcanic and basement sedimentary rocks
Mafic Dikes	Commonly present	Present and often associated with high-grade (+10 g/T) gold values

Table 8-1 does not include all the characteristics of LS-type epithermal deposits but instead presents eight criteria found at Kilgore, and effectively demonstrate the correlation of lithologies, structure, mineral paragenesis, and mineralization styles at Kilgore with those of other known deposits of this class.

The Kilgore deposit lies along the intersection of major northwest-trending regional structures, the Northwest Fault Zone, and northeast-trending radial faults from the core of the Heise Volcanic Field. These zones lie just inside of, and tangential to, the arcuate northeast part of the Kilgore Caldera margin and structural ring fracture zone. While the Kilgore deposit is located along the inferred margin of the Kilgore Caldera, the age of mineralization is dated at approximately 5.3 Ma (Benson C. , 1986), which pre-dates the age of the final phases of the Heise Volcanic Field and are attributed to the formation of the Kilgore Caldera itself (i.e., the Kilgore tuff dated at approximately 4.45 Ma) (Morgan & MacIntosh, 2005; Watts, Bindeman, & Schmitt, 2011).

Otis Gold’s present conceptual model associates gold mineralization with the processes that lead to the formation and eruption of the Kilgore Caldera, the final of four major phases of the Heise Volcanic Field; a near surface injection of large volumes of magma, related to the apparent east-northeast migration of the Yellowstone Hotspot, was responsible for creating a large hydrothermal system. The eruption of the Kilgore Caldera has complicated further exploration of the Kilgore Project by covering the project area in a thick layer of volcanic and volcaniclastic rocks. Continued exploration of the Kilgore Project may lead to further discoveries of epithermal deposits related to the pre-, syn- and post-eruption hydrothermal activity.

9.0 EXPLORATION

9.1 Historic Exploration

With the exception of exploration carried out by Echo Bay in the mid to late 1990s, very little detail is available regarding exploration activity (other than drilling) conducted by previous operators of the Kilgore Project. From 1994 through 1997, Echo Bay conducted soil sampling, regional geologic mapping, an airborne magnetic and HEM survey, and false color satellite imaging.

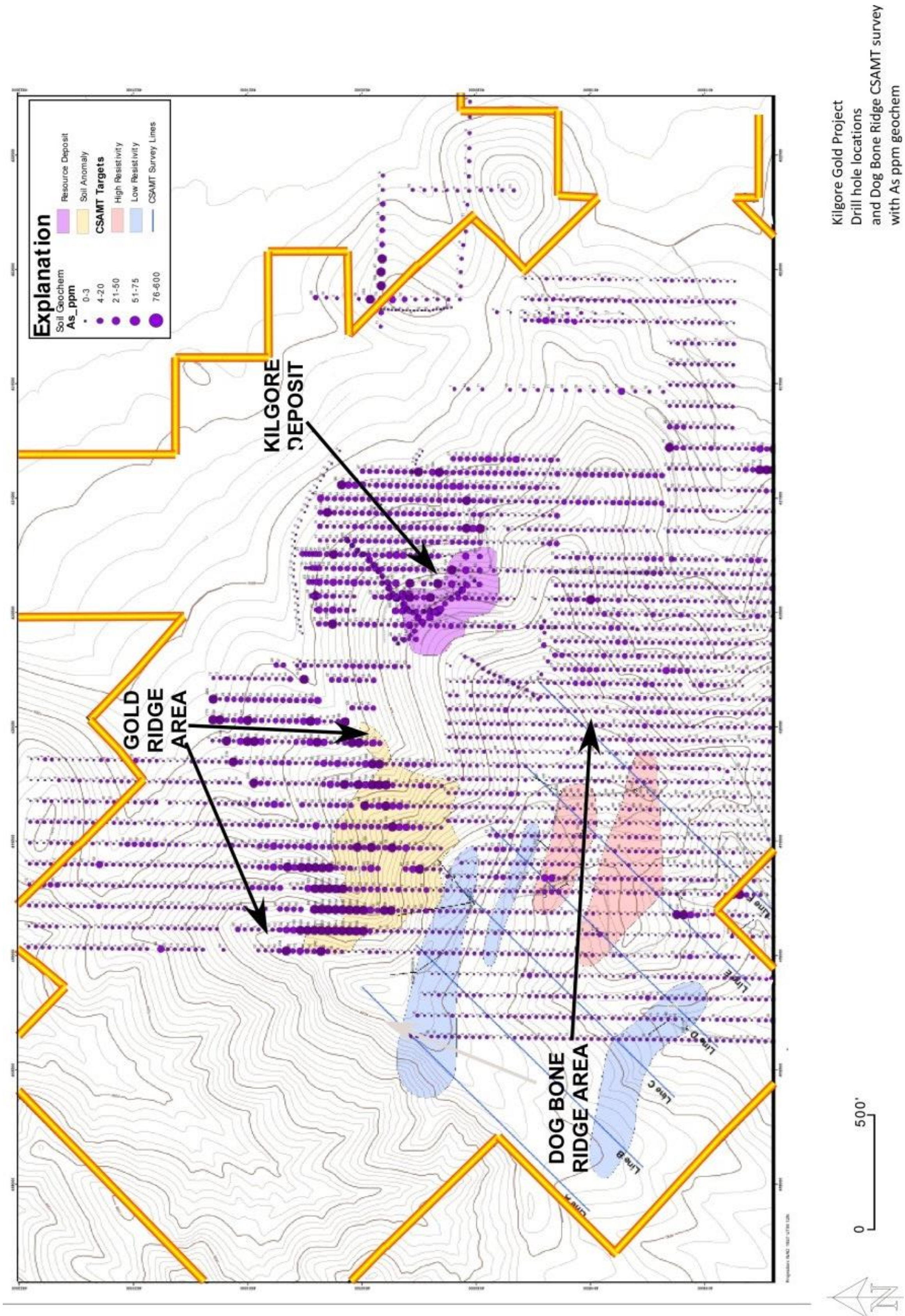
In 1996, Echo Bay collected 1,857 soil samples over the Kilgore property. The data was compiled, but a rigorous interpretation was never made. The samples demonstrate anomalies in gold, arsenic (Figure 9-1), antimony, mercury, and selenium over the deposit. Arsenic exhibits the strongest correlation to gold and silver, and it is clear that there is a significant arsenic anomaly immediately adjacent to, and down-slope from, the deposit. The Echo Bay data also showed that strongly anomalous arsenic occurs on the forested slope lying one km to the northwest of the Kilgore deposit. This is identified by Otis Gold as its Gold Ridge target.

In 1996, Echo Bay contracted Aerodat of Toronto, Canada, to complete a 70-square-mile (180-square-km) helicopter-borne electromagnetic, magnetic, radiometric, and Very Low Frequency-Electromagnetic survey. While the data were never fully reduced or followed-up, the results of Aerodat's work, as reported by Rayner and Associates and Van Brunt (2002), are summarized below:

- A large resistivity high covers the property, with the Kilgore deposit located on the northeast flank of the high
- A circular resistivity high exists 3 miles (4.8 km) west of the Kilgore resource and is surrounded by a ring-shaped magnetic feature
- The Kilgore deposit and core claims lie on the northeast flank of a large, round magnetic anomaly thought to be a caldera margin that is at least 8.7 miles (14 km) in diameter
- A large ring-shaped magnetic feature to the north of the claim block may be the edge of another caldera
- A linear northeast-trending magnetic low at least 3 miles (4.8 km) in length extends through the property in the area of Dog Bone Ridge
- An east-west magnetic low parallels the southern margin of the Kilgore resistivity high.

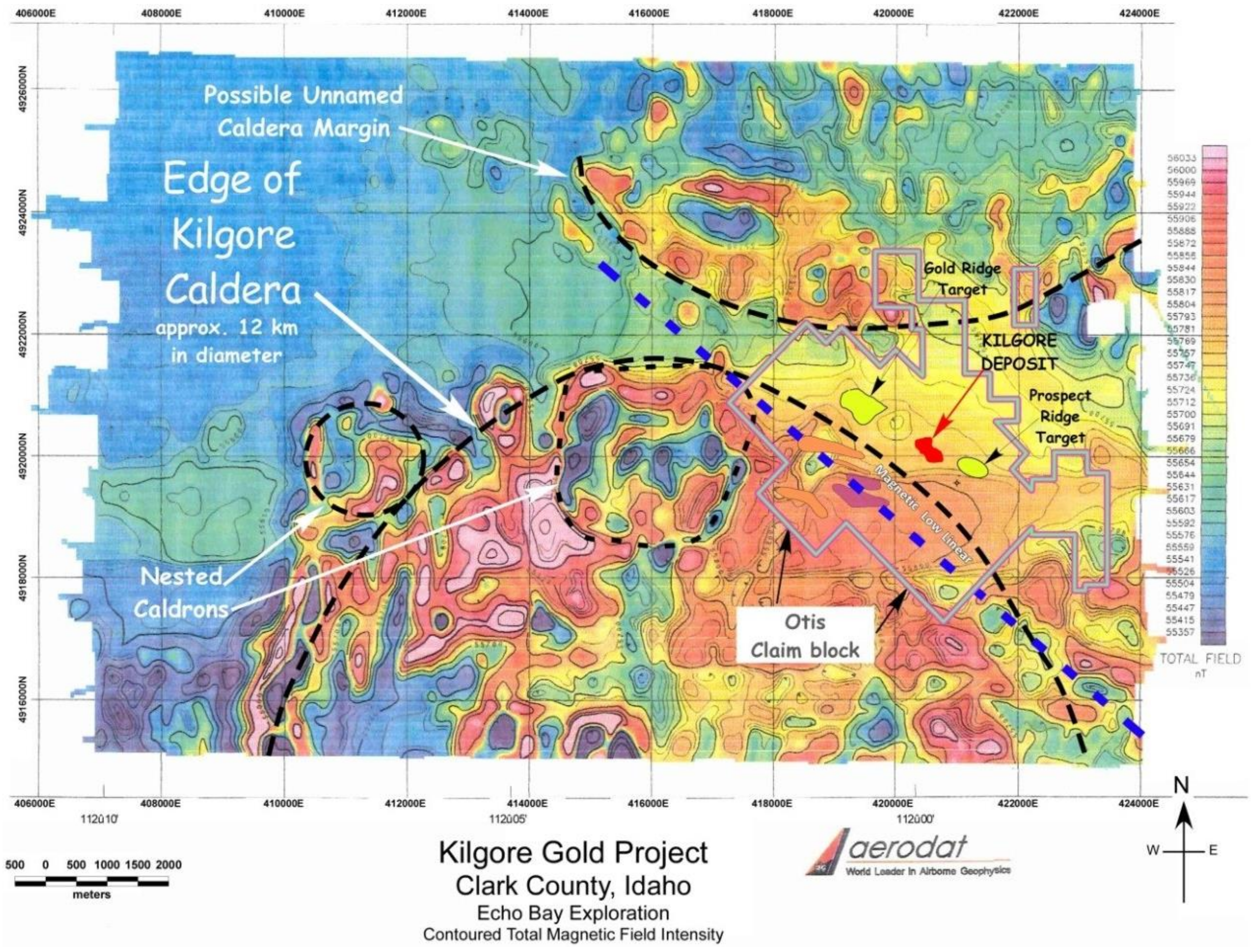
The relationship of the magnetic features to the Kilgore deposit is shown in Figure 9-2.

Figure 9-1: Bubble Map Showing Arsenic in Soil Anomalies in the Gold Ridge Area, Echo Bay, 1996



Source: Otis Gold

Figure 9-2: Airborne Magnetics Flown by Aerodat for Echo Bay in 1996



Interpretation and Annotation are by Otis Gold

9.2 Otis Gold Exploration

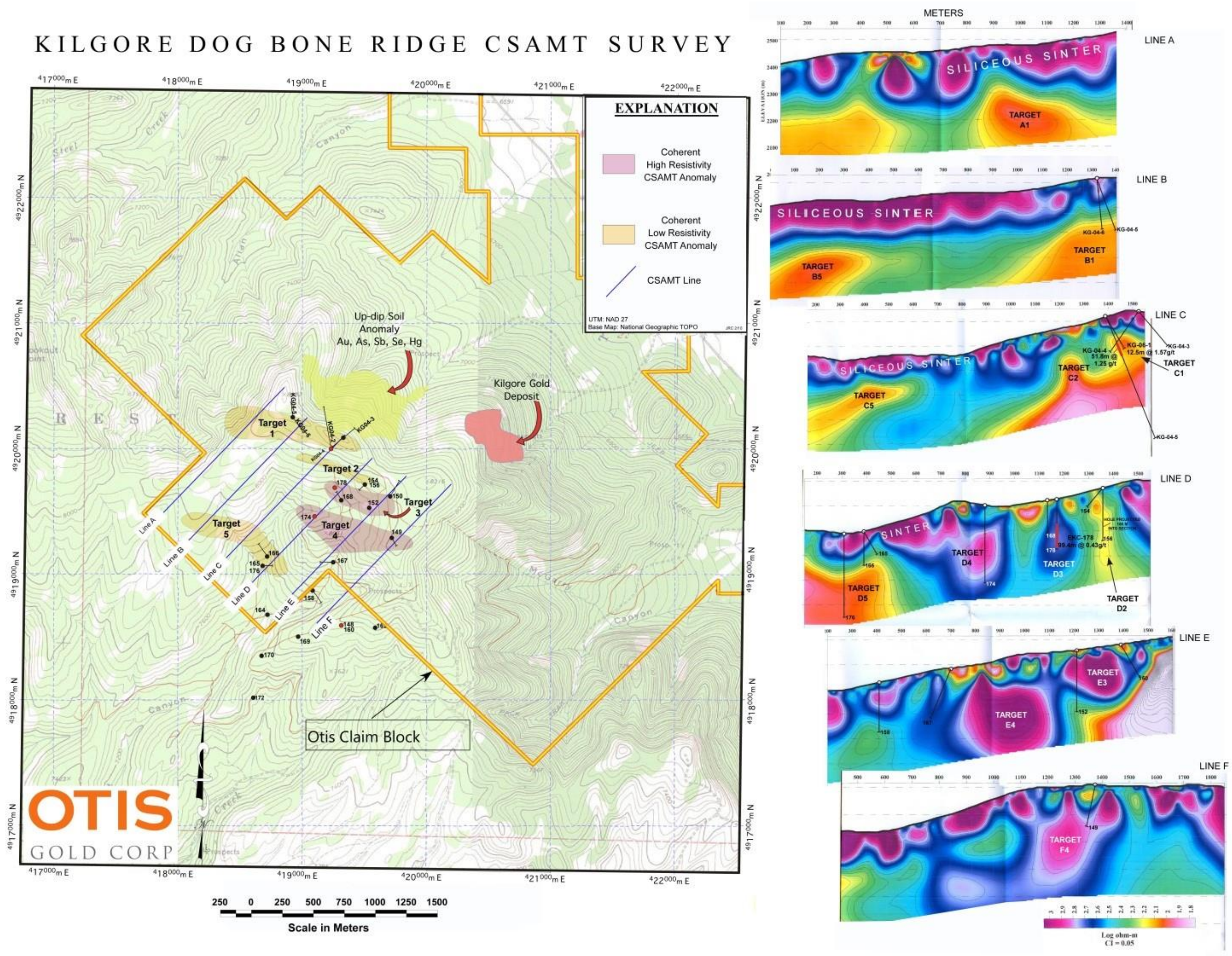
Outside of drilling, exploration activities carried out by Otis Gold to date include geophysical survey and regional surface sampling of rock, soils, and stream sediments.

9.2.1 Geophysical Exploration

9.2.1.1 2009 CSAMT Survey

In October 2009, Otis Gold commissioned Zonge Geoscience, of Reno, Nevada, to perform a Controlled Source Audio Magneto-Telluric (CSAMT) survey in the Dog Bone Ridge target area (Figure 9-3). Six lines were oriented N45E for a total of 5.3 line-miles (8.5 line-km) of data coverage. The objective of the survey was to delineate near-surface alteration as well as underlying structures and potential feeder zones.

Figure 9-3: Dog Bone Ridge CSMAT Survey



Cameron, 2012

The survey was conducted using a 165-foot (50-meter) electric-field receiver dipole in spreads consisting of four electric-field dipoles with a magnetic field antenna located in the center of the spread. The magnetic antenna was oriented perpendicular to the survey line. Measurements were made at frequencies ranging from 1 hertz (Hz) to 8,192 Hz, in binary steps. Each current electrode consisted of three pits lined with aluminum foil and soaked with salt water. The electrodes were connected to the transmitter with lengths of insulated 14-gauge wire, separated by approximately two meters.

The survey tested for low-to-moderately resistive bodies containing higher resistivity core associated with structures that may have acted as conduits for gold mineralization. Resistivity of a rock is generally controlled by rock porosity. Dense compact rocks, such as those affected by silicification, tend to be highly resistive. Structural zones often produce relatively low resistivity due to increased porosity resulting from broken rock. Mixtures of rock and alteration types produce resistivity results that are difficult to interpret.

Figure 9-3 presents all six inverted resistivity sections overlain by a structural interpretation. Clearly evident on all sections is a surface layer with predominantly high resistivity overlying variable, but predominantly less resistive material. A notable exception is Line B, which exhibits a uniform high resistivity layer for most of its length. This high surface resistivity is interpreted as representing the sinter cap and explosion breccia – rock unit Tup. The dotted lines separate the two resistivity domains, and the dashed lines identify interpreted structures. For reference, the geochemical signature of the Echo Bay soil anomaly is shown in yellow, with the Kilgore deposit to the northeast shown in salmon color.

Echo Bay Hole EKC-178, collared on Dog Bone Ridge, drilled into the core of the low resistivity anomaly of line D and encountered 324 feet (99 meters) @ 0.012 opt (0.418 g/T) Au. Wright (2009) discusses this intercept and recommends testing other anomalies. Kilgore Gold intercepted 170 feet (51.8 meters) @ 0.036 opt (1.25 g/T) Au in hole KG-04-4, which was nearly coincident with the low resistivity anomaly detected at the end of Line C.

Based on the results of the 2009 CSAMT survey, Otis Gold selected five targets on Dog Bone Ridge and drilled four of them (Table 9-1) in 2010 with mixed results.

Table 9-1: Hole Drilled into Anomalies on Dog Bone Ridge (Cameron, 2012)

Hole ID	Site	Azimuth (N=0)	Angle (deg)	Total Depth (ft)	Total Depth (m)	Target Tested	CSAMT Line	Anomaly Type
10 OKC-240	4	45	-62	2,618	798	E4	Line E	High Resistivity
10 OKC-241	3	45	-63	2,418	737	E3	Line E	High Resistivity
10 OKC-242	1	350	-65	1,965	599	C1	Line C	Low Resistivity
10 OKC-243	1	45	-45	2,749	838	C1	Line C	Low Resistivity
10 OKC-244	2	255	-80	1,834	559	C2	Line C	High Resistivity

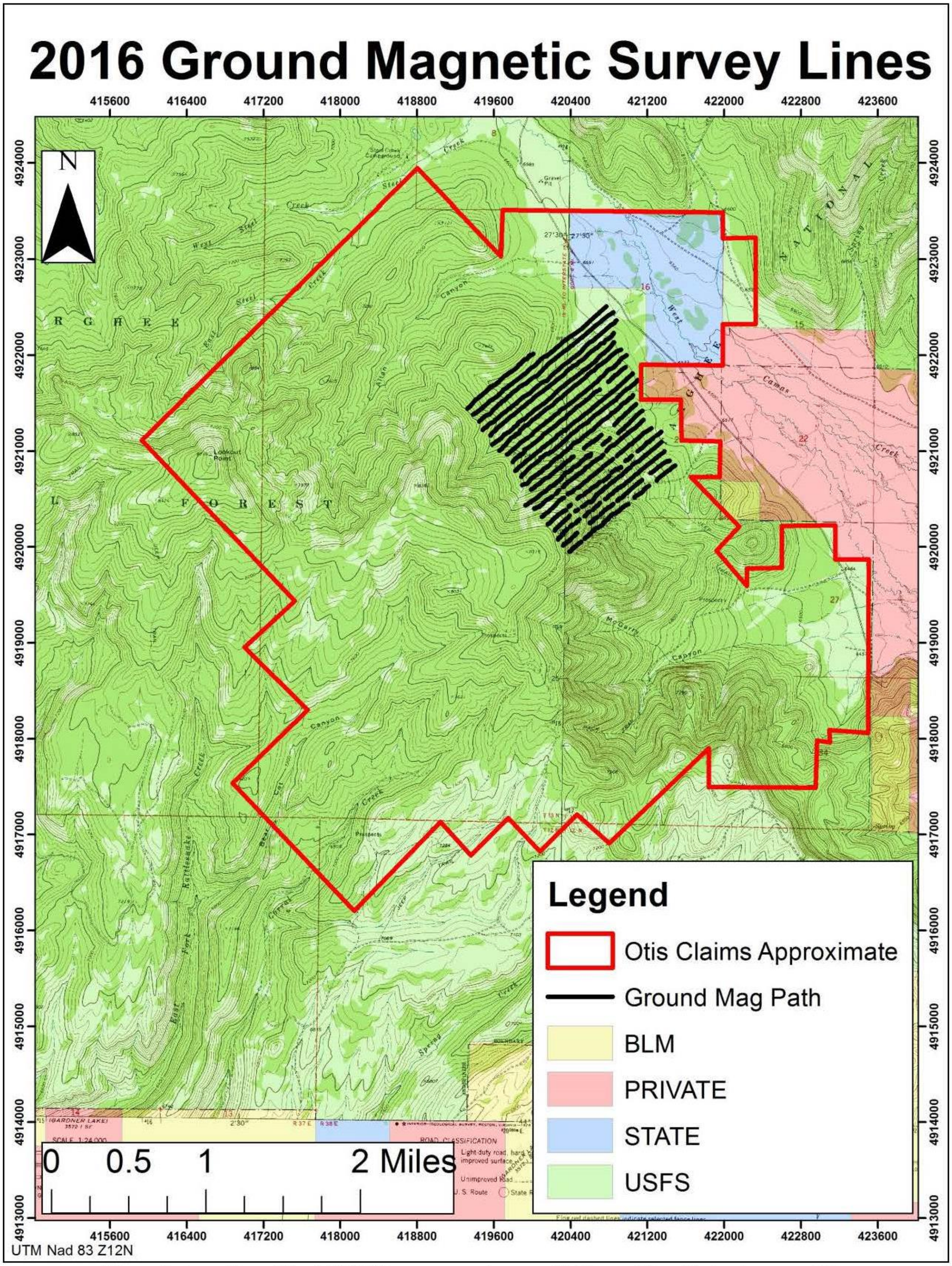
10OKC-240 drilled into high resistivity anomaly E4. From surface to 341 feet (104 meters), the hole encountered barren siliceous sinter and explosion breccia that contained no detectable gold. From 341 feet (104 meters) to 797 feet (243 meters), the hole cut a felsic dike that contained slightly anomalous gold. Drill hole OKC-242, drilled into Anomaly E3, encountered mostly siliceous sinter, but very little detectable gold. The most interesting hole was OKC-243, collared near Kilgore Gold's hole KG-04-4. This hole encountered a 98-foot (30-meter)-thick hydrothermally altered and brecciated felsic dike with gold grades up to 0opt (0.731 g/T) Au.

9.2.1.2 2016 Ground Magnetic Survey

In November 2016, Otis Gold contracted Justin Modroo, P.G., to conduct a ground based geophysical magnetic survey in the vicinity of the primary Kilgore resource area (Modroo J. , 2017). The survey was designed to test magnetic signatures surrounding the known deposit to better define local structural characteristics and potentially identify future drilling exploration targets.

The ground magnetic survey successfully recorded data over approximately 20.5 line-miles (33 line-km) (Figure 9-4) using Geometrics G-859 (rover) and G-856ax (base station) magnetometers. Line directions were collected along 50°and 230° azimuth and were spaced 295 feet (90 meters) apart. Base station data were recorded every 30 seconds, while rover data were recorded every second. The survey successfully mapped local- and regional-scale magnetic anomalies, providing greater insight into the structural components of the primary Kilgore deposit and surrounding area. The survey identified several significant magnetic features that may be directly related to known faulting and other structural features which contribute to or control the distribution of mineralization.

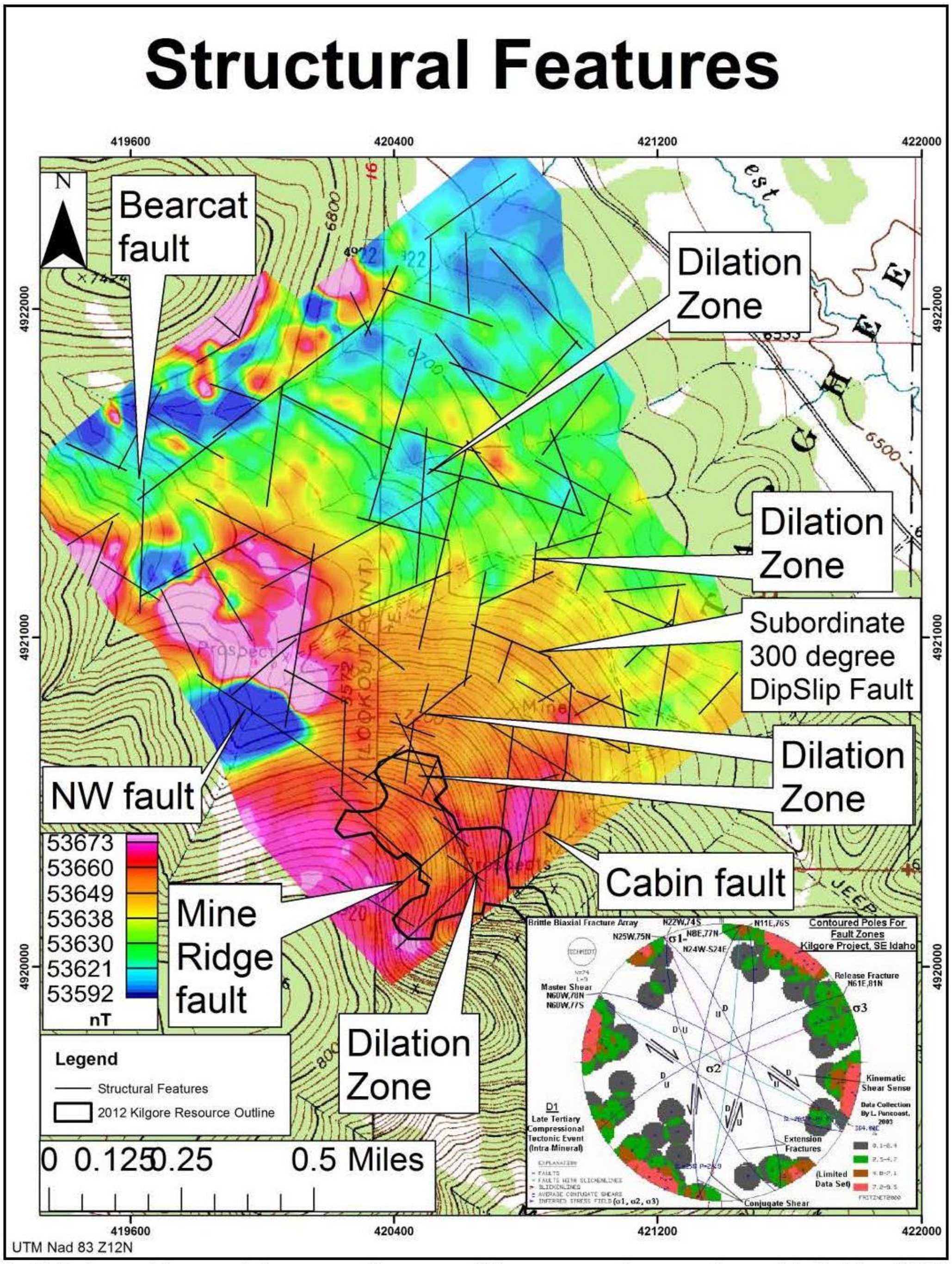
Figure 9-4: 2016 Ground Magnetometer Survey Lines



Modroo, 2017

Based on the results of the 2016 ground magnetic survey, Modroo (2017) suggests that the Kilgore resource and region was preferentially structurally prepared prior to magmatic and meteoric epithermal gold and silver mineralization. The hydrothermal fluids used zones of weakness and dilation due to the conjugate fault system (300° and 10°) and extensional and relaxation faulting (335° and 60°). These structural features are interpreted in the magnetic data and include the dextral and dip-slip NW fault, the sinistral Bear Cat, Cabin, and Mine Ridge faults, and a number of subordinate faults with similar regional trends (Figure 9-5).

Figure 9-5: 2016 Ground Magnetometer Survey RTP Structural Interpretation



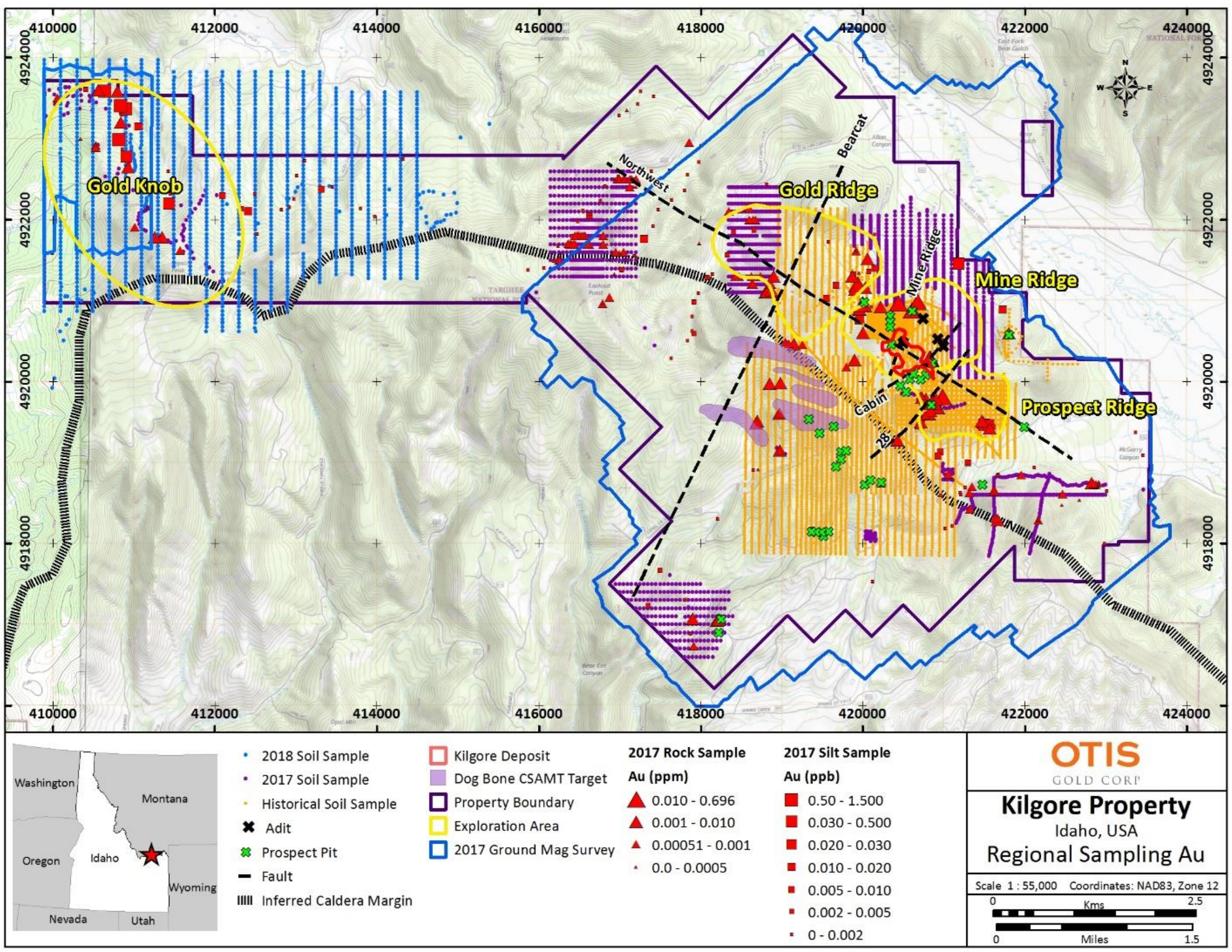
Modroo, 2017

The structural features and boundaries create a network of structural compartments, some of which could be favorable hosts for hydrothermal gold mineralization. The Kilgore resource area is interpreted to be located within one such mineralized compartment, bounded to the southwest by the NW fault, to the southeast by the Cabin fault, and to the northwest by an un-named fault that strikes 60°.

9.2.1.3 2017 Ground Magnetic Survey

Encouraging results from the 2016 ground magnetic survey prompted Otis Gold to contracted Justin Modroo, P.G., to conduct a second, expanded ground based magnetic survey in 2017 (Modroo, 2018). The 2017 ground magnetometer work consisted of two discrete survey grids, one in the immediate vicinity of the Kilgore resource area for a total of 295 line-miles (474 line-km), and a second, 25-line-mile (40-line-km) grid in the Gold Knob area roughly 6.2 miles (10 km) west of the Kilgore deposit. The 2017 ground magnetic survey boundaries are shown in blue on Figure 9-6.

Figure 9-6: 2017 Ground Magnetic Survey Boundaries (Otis Gold, 2018)

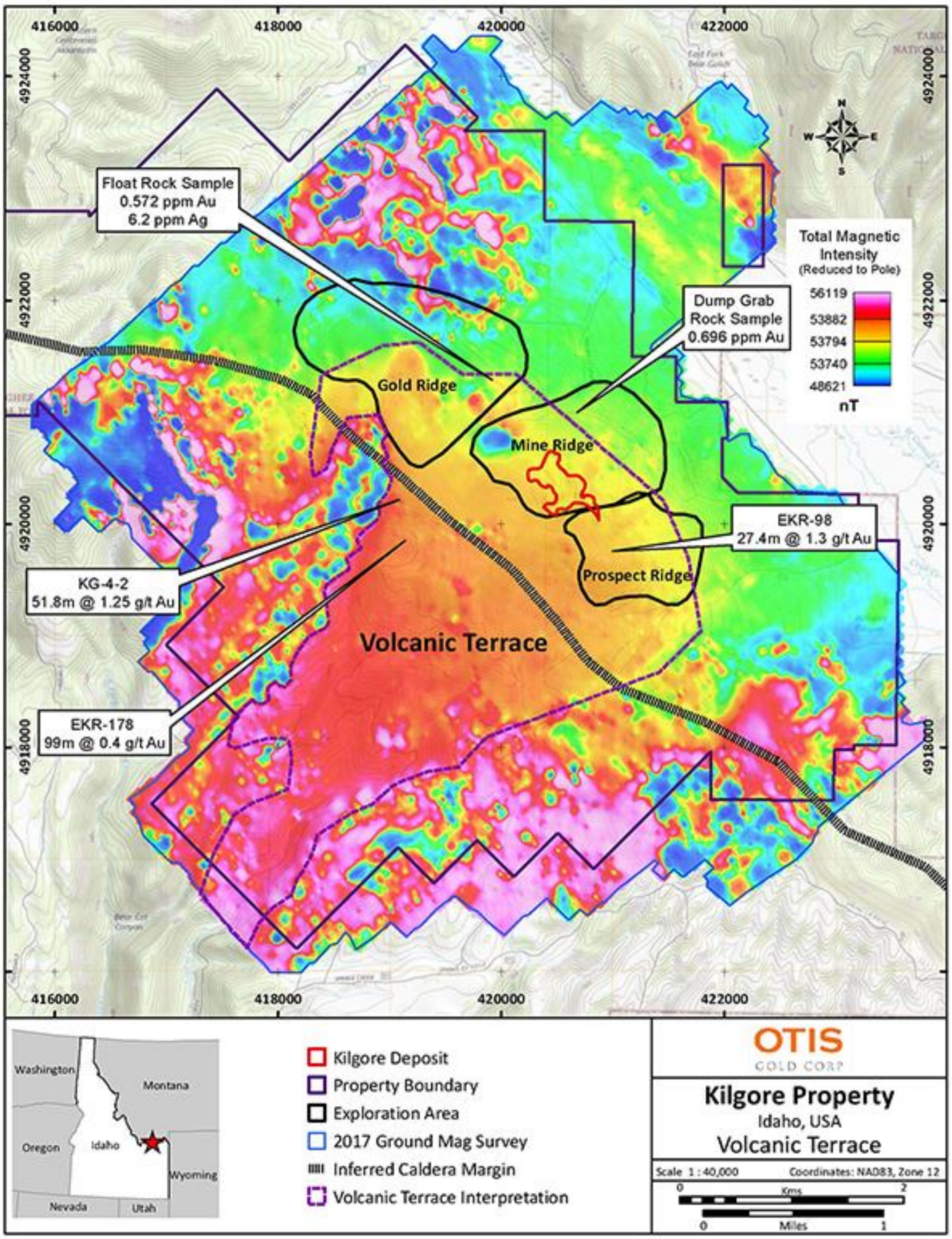


In the Kilgore resource area, survey line directions were collected along 50°and 230° azimuth and were spaced 300 feet (91 meters) apart, following the 2016 design. The Gold Knob survey was designed to obtain general geologic and structural information based on a limited amount of existing data in the area. Line directions were collected along 0°and 180° azimuth and were spaced 328 feet (100 meters) apart (except in the GK 32 discovery area, where two extra lines were acquired 108 feet (33 meters) apart). All

2017 location data was recorded in the NAD 83, Zone 12, North UTM grid system. Base station data were recorded every 60 seconds, while rover data were recorded every second (1 Hz).

Figure 9-7 shows the reduced-to-pole total magnetic intensity for the main Kilgore grid (Modroo, 2018) with interpretation and annotation overlaid by Otis Gold.

Figure 9-7: 2017 Kilgore Ground Magnetic Survey RTP



Otis Gold, 2018

The 2016 and 2017 surveys together provide a total of 340.5 line-miles (548 line-km) of ground magnetic data within the Kilgore Project area. Results and interpretations of the combined survey efforts are summarized by Modroo (2018), as follows:

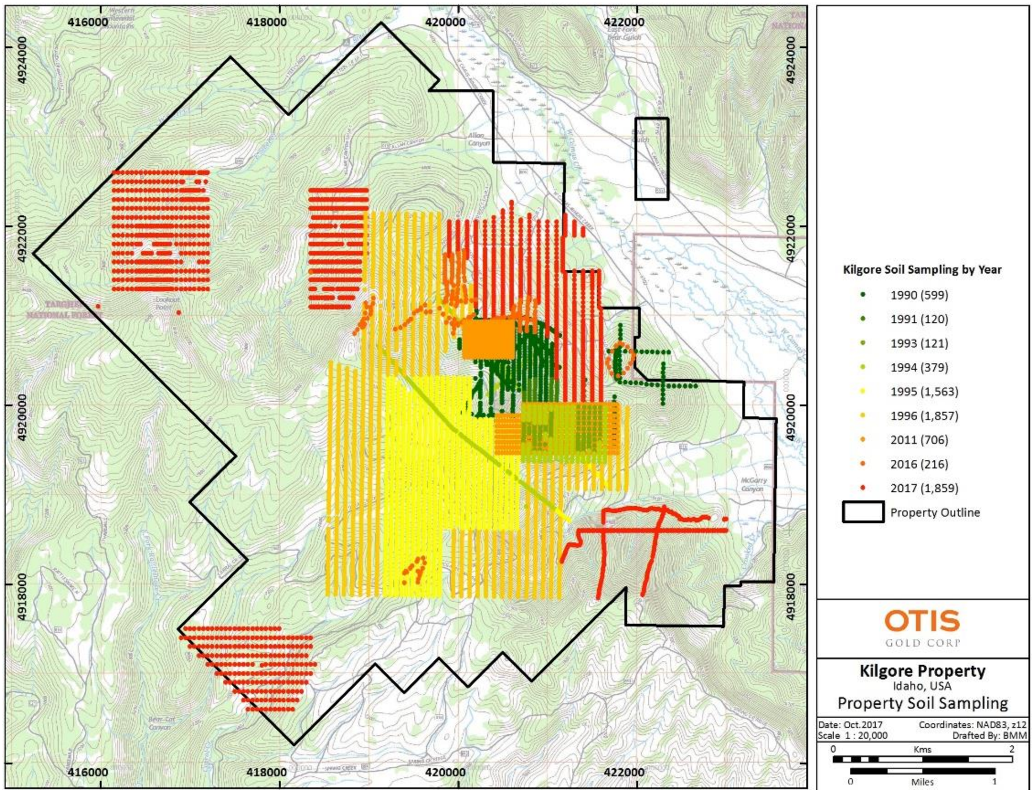
- The magnetic data illustrates the structural regime (confirming Caddey's (2003) work/interpretation) important for locating fault intersections and resultant dilation zones hosting mineralization, but also to track post mineral fault movements and help decipher the local structures within the Kilgore deposit.
- The magnetic data shows a conjugate faulting system that created a network of pathways for fluid movement along blocks/compartments of the shattered crust and brittle volcanic geology. It will be important to test mineralization extent within the local structural setting to help determine favorable geologic/meteoric conditions for gold mineralization.
- The data has clearly defined an 3.1-square mile (8-square km) "volcanic pile/terrace" that is highly prospective for gold exploration associated with the Crystal Tuff (Tad) and areas in contact with reactive and permeable country rocks, like the Aspen (Ka) and Biotite Rhyolite (Tbr). The "toe" of this volcanic terrace hosts the current Kilgore deposit and also includes other positive historical drilling results and anomalous gold surface sampling results.
- The magnetic data is also a valuable tool for mapping alteration, with many obvious localized alteration lows throughout the data, highlighting the regional extent of this hydro-thermal system. Closely monitoring the alteration and gold bearing mineral assemblage will help guide local exploration around new and existing gold occurrences.
- The data shows multiple geologic events that occurred post mineralization and have essentially covered up the most prospective geology. Sporadic surface geochemical sampling results obtained over these younger features show no elevated mineralization.
- Utilizing all available exploration data, over 20 excellent drill targets have been identified with an additional 40+ locations selected to follow up on initial success. These exploration targets cover a prospective 2.3-square mile (6-square km) area surrounding the Kilgore deposit.

9.2.2 Soil Sampling

To date, approximately 7,420 soil samples have been collected throughout the Kilgore Project area. Of that total, 2,781 were collected on behalf of Otis Gold beginning in 2011 (Figure 9-8 and Figure 9-9).

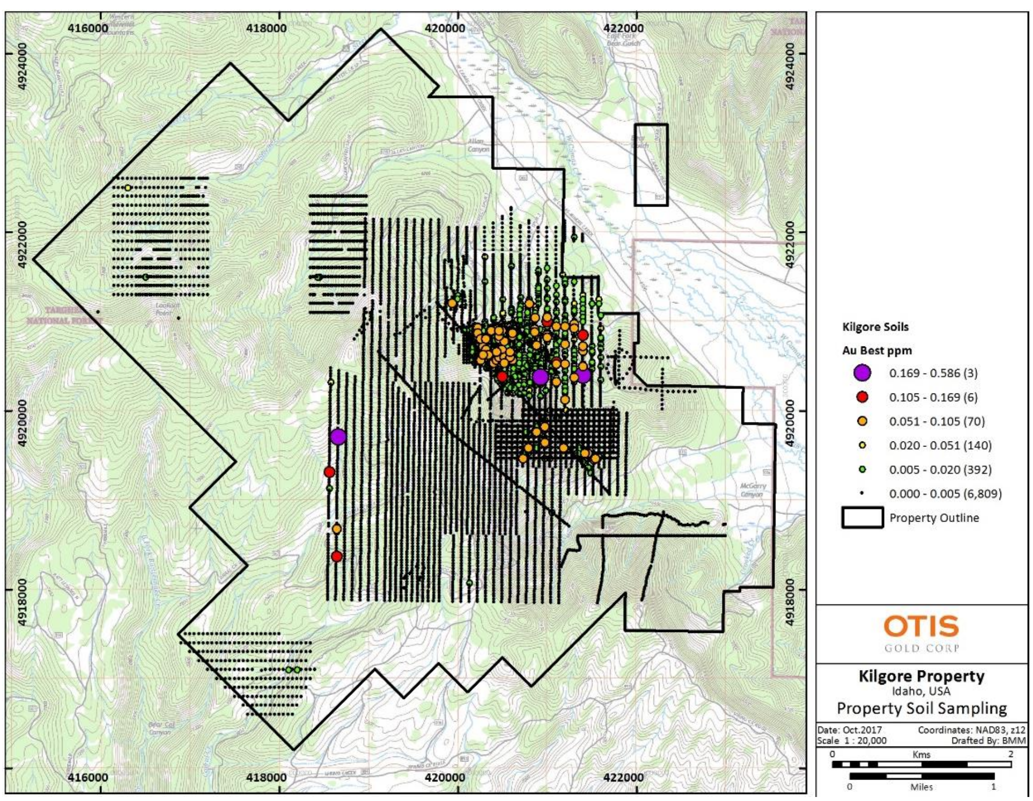
In October 2011, Otis Gold contacted North American Exploration (NAE) and commissioned two soil surveys at Kilgore. The surveys were conducted along the trace of the Northwest Fault to determine if this structural feature could be traced further northwest or southeast of the existing road system. If anomalies could be detected, then road construction would be warranted to test them. The "C" soil horizon was sampled just above bedrock. Samples were collected with a sharp-shooter type shovel. Any large roots and/or other organic matter and small stones over the size of about 0.6 inches (1.5 centimeters (cm)) were removed on the shovel blade, and the soil was placed into clean 2.5 x 8-inch (6 x 20-cm) cloth bags.

Figure 9-8: Kilgore Soil Sampling by Year



(Otis Gold, 2018)

Figure 9-9: Kilgore Soil Sampling Results

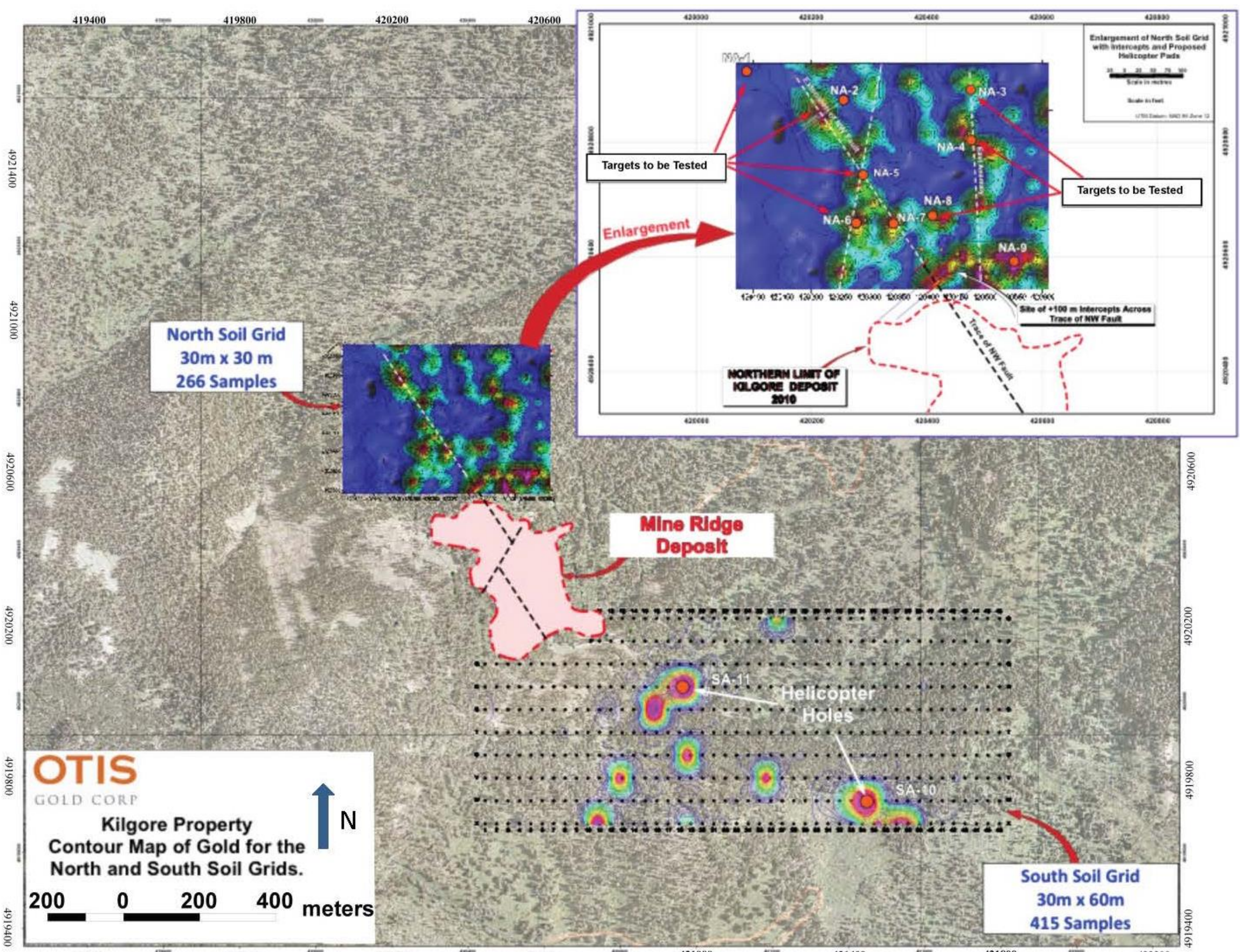


(Otis Gold, 2018)

After each soil sample was collected, the corresponding waypoint number was written on the sample bag, and a Global Positioning System (GPS) coordinate was collected using the NAD 83 Continental Datum. At each sample location, a 1 x 3-inch (2.5 x 7.5-cm) aluminum tag with a waypoint number was scribed on it and attached to vegetation along with a ribbon of pink flagging so that the site could be relocated. NAE collected 266 samples from the North Soil Grid at 100-foot x 100- foot (30-meter x 30-meter) spacing, and 415 samples from the South Soil Grid at 100-foot x 200-foot (30-meter x 60-meter) spacing. The samples were put in rice bags by NAE and transported directly to ALS's "clean lab" in Winnemucca, Nevada, where they were prepped and shipped to Reno, Nevada, for analysis by ALS Labs.

After results were received from the North Soil grid, the data were contoured using Golden Software's SURFER, with results plotted on an orthophoto map. The data display a strong and significant linear gold-in-soil anomaly that closely aligns to the extension of the Northwest Fault (the apparent structural conduit to the system) controlling the overall northwest trend of the deposit (Figure 9-10).

Figure 9-10: Gold-in-Soil Anomalies Associated with the North and South Soil Grids, 2011



Cameron, 2012

The anomaly opens the possibility of a 1,312-foot (400-meter) extension of the Kilgore deposit to the northwest beyond where +100-meter-thick intercepts of 0.89 g/T Au were discovered in holes 11 OKC-258 and 259. The gold anomaly is supported by trace-element geochemistry characteristic of a typical epithermal gold system. The North area anomaly fills a portion of the gap between the Gold Ridge target further to the northwest and the Kilgore deposit to the southeast.

The South Soil Grid (Figure 9-10) displays a very strong and coherent gold-in-soil anomaly that covers approximately 18,000 square yards (15,000 square meters) in the Prospect Ridge target area. This anomaly overlies a section of the lithic tuff that is identical to rock that hosts the majority of the Kilgore deposit.

Sample collection on the Kilgore Project from 2012 to 2016 was limited to localized prospecting north of the Kilgore resource area. That work produced 20+ rock and soil samples, most of which returned anomalous Au values, including up to 0.5 parts per million (ppm) from a single grab sample.

The anomalous Au samples north of the primary deposit prompted collection of 213 soils samples in 2016. The soil samples were collected from the "B" or "C" horizon using a shovel and occasionally a hand auger in areas of thick organic overburden. Soil samples were collected opportunistically and in general with a north south line direction with spacing up to 328 feet (100 meters). Soils samples were collected approximately 82 feet (25 meters) apart along the selected line path. All geochemical samples (rocks, silts, and soils) were submitted to ALS Minerals in Reno, Nevada, and treated by Aqua Regia digestion and analyzed by Fire Assay with either inductively coupled plasma-mass spectrometry (ICP-MS) or atomic absorption spectrometry (AAS) finish. The soil sample results revealed a growing and open-ended Au anomaly north of the deposit and other anomalous soil geochemistry along defined structures, providing good correlation between data sets.

Sampling in 2017 produced 2,125 soil samples, 268 stream/silt samples, and 151 rock/grab samples (Figure 9-11). Acquisition protocol and personnel were the same as the 2016 exploration program. The 2017 soil geochemistry work was designed to follow-up open-ended geochemical anomalies, to better define drill targets, and to generate new exploration targets outside of the primary Kilgore resource area. The soil sample results reveal a 3,280-foot x 1,640-foot (1,000-meter x 500-meter) area of anomalous gold northeast of the Kilgore deposit (Figure 9-12), highlighting the mineralizing potential of subordinate structures (e.g., Gold Ridge, Northwest, Snotel, and the Vortex faults) and conjugate structures (e.g., McGarry, 28, Cabin, Mine Ridge, Dog Bone, and Bear Cat faults) and their corresponding intersections and resultant dilation zones.

In 2018, as part of Otis Gold's ongoing program of regional exploration, Otis Gold conducted a soil sample survey across the northwestern end of the claim block. A total of 750 soil samples were collected along lines spaced 200m apart and at 50m intervals (Figure 9-13). Samples were sent to BV-Inspectorate in Reno, Nevada for multi-element geochemical analysis. The results showed a number of localized gold anomalies and associated silver and arsenic anomalies. Other base element anomalies including mercury, selenium and antimony that picked out north-south trending structures interpreted as being caldera related radial faults. Of note was the silver anomaly in soils that appears to coincide with the Tertiary volcanic and Cretaceous sediment contact.

Figure 9-11: Regional Soil and Rock Sampling, 2018

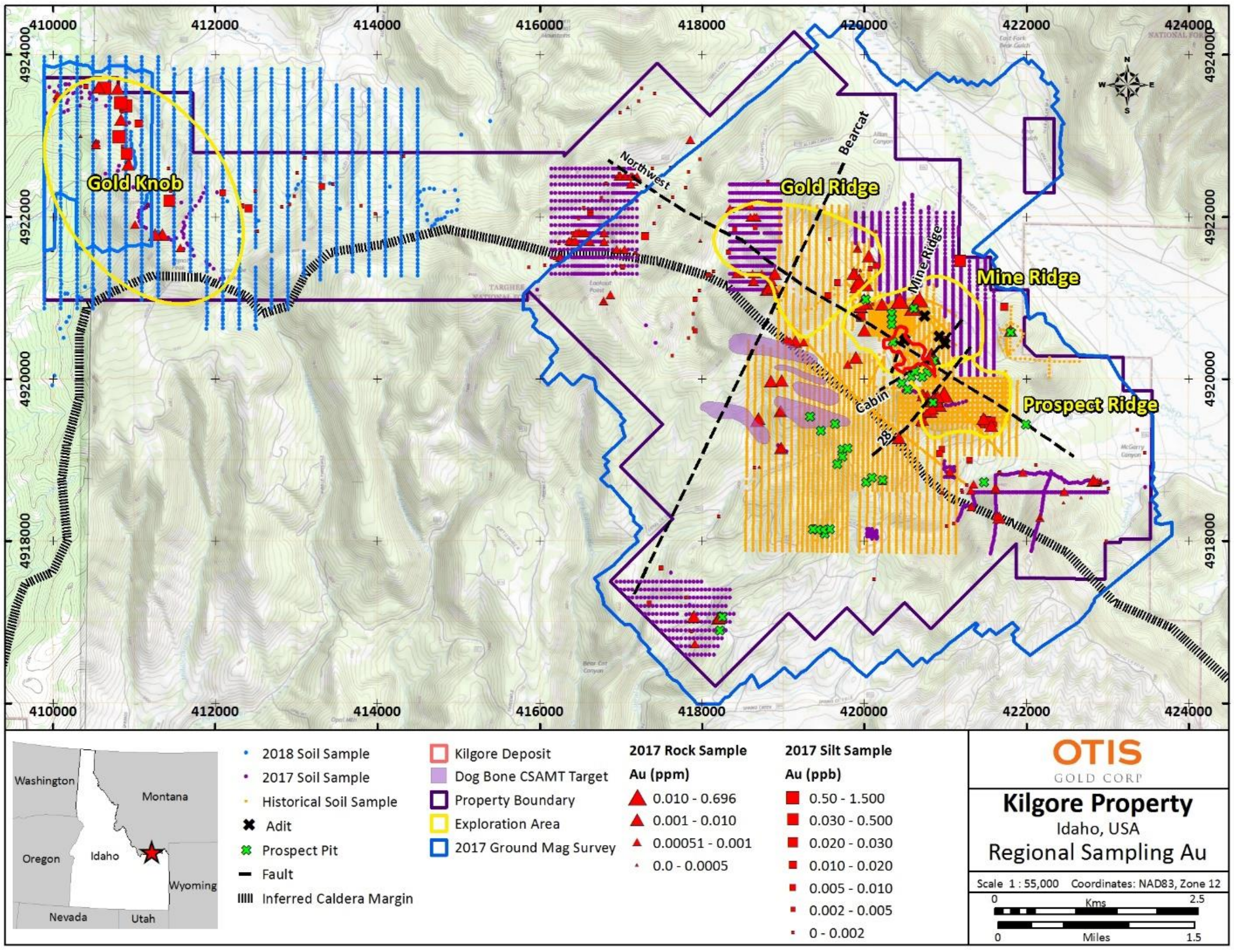
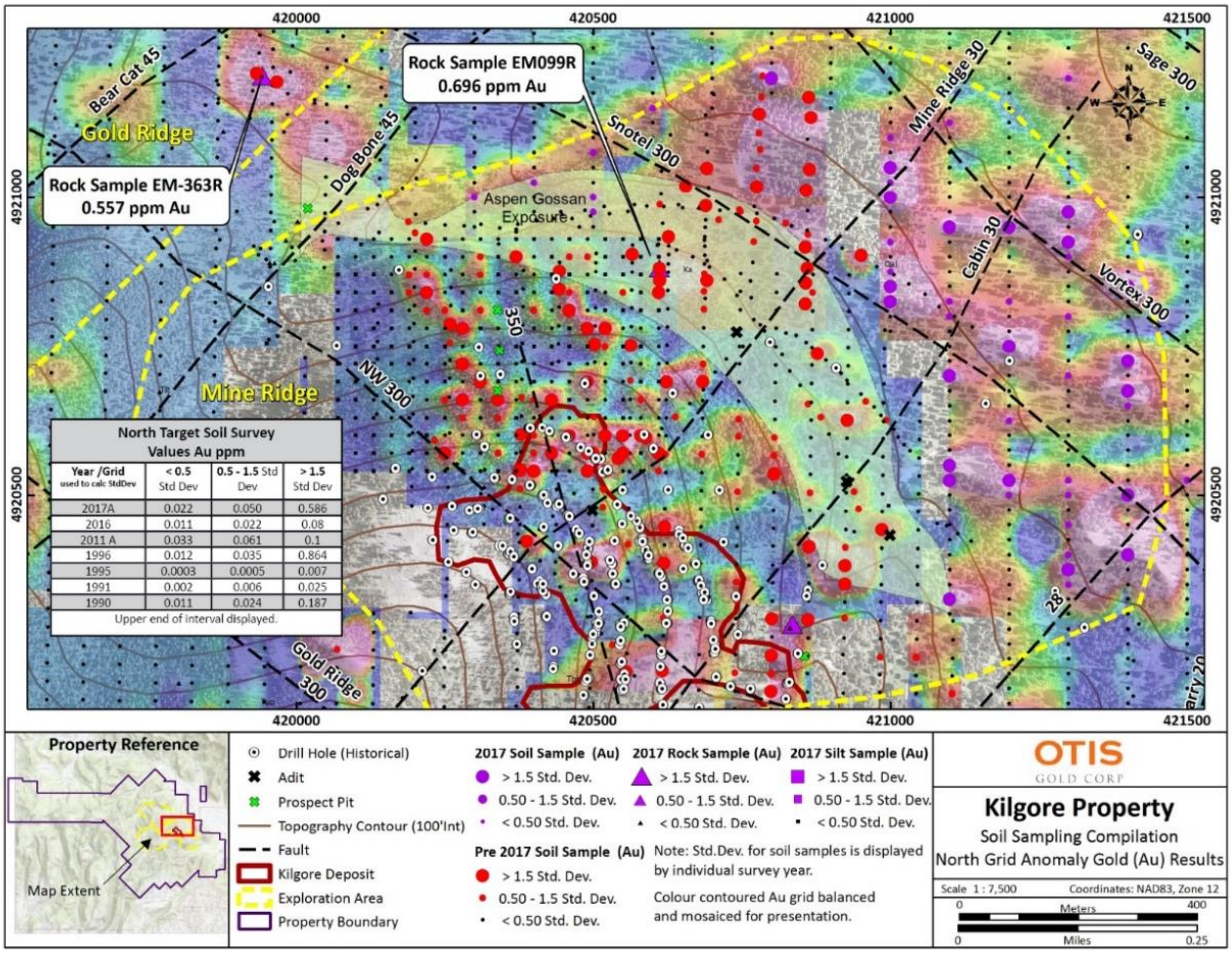
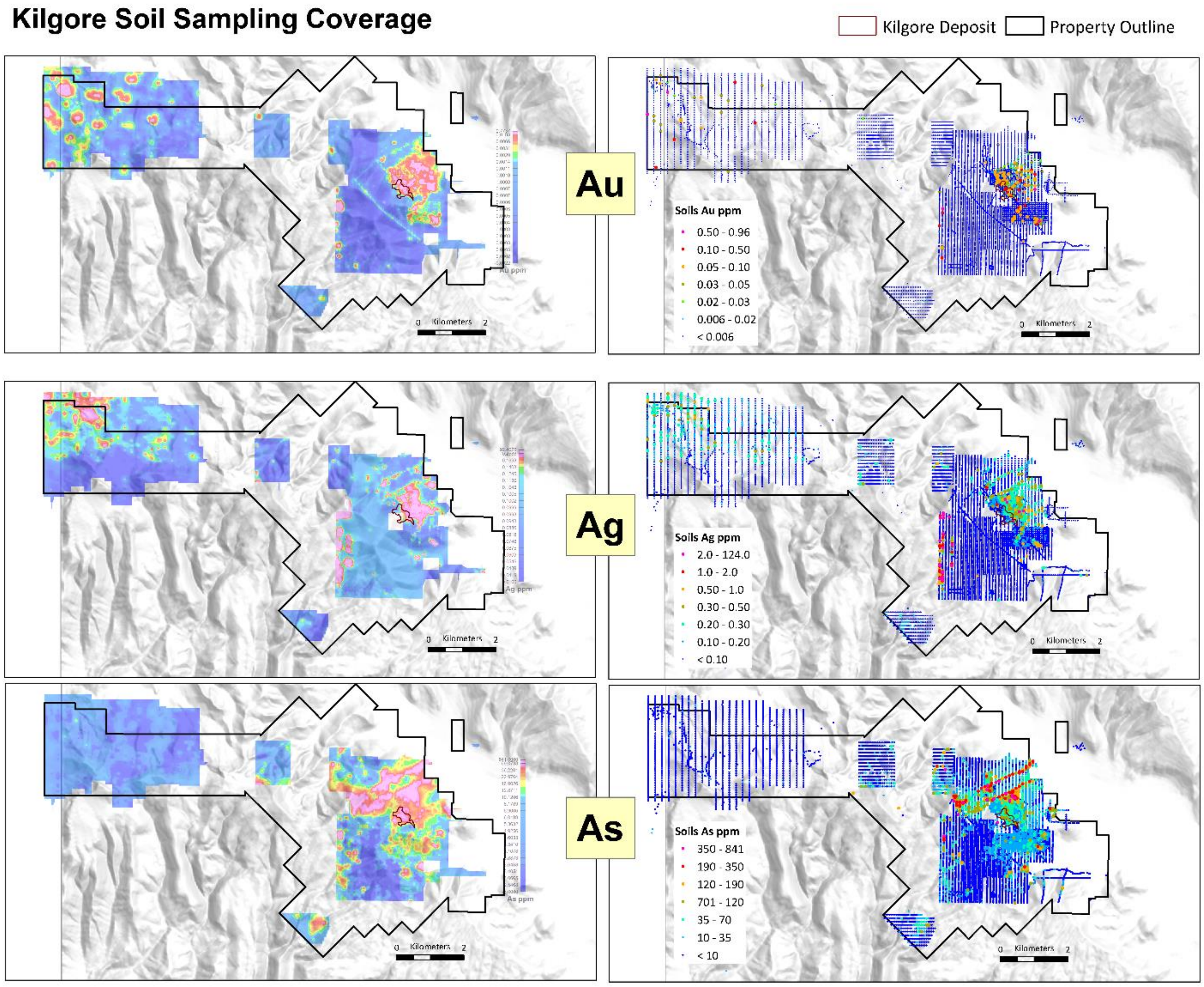


Figure 9-12: 2017 Soil Sampling Results, North Gold Anomaly (Otis Gold 2017)



North Target Soil Survey Values Au ppm			
Year /Grid used to calc StdDev	< 0.5 Std Dev	0.5 - 1.5 Std Dev	> 1.5 Std Dev
2017A	0.022	0.050	0.586
2016	0.011	0.022	0.08
2011 A	0.033	0.061	0.1
1996	0.012	0.035	0.864
1995	0.0003	0.0005	0.007
1991	0.002	0.006	0.025
1990	0.011	0.024	0.187
Upper end of interval displayed.			

Figure 9-13: Multi-element Soil Results Including Au, Ag and As



10.0 DRILLING

A total of 381 drill holes (152 RC and 229 core) have been drilled in the Kilgore Project to date. Drill hole locations are identified according to year drilled on Figure 10-1, and Table 10-1 summarizes the individual drilling campaigns, broken out by operator.

Figure 10-1: Kilgore Exploration Drill hole Locations

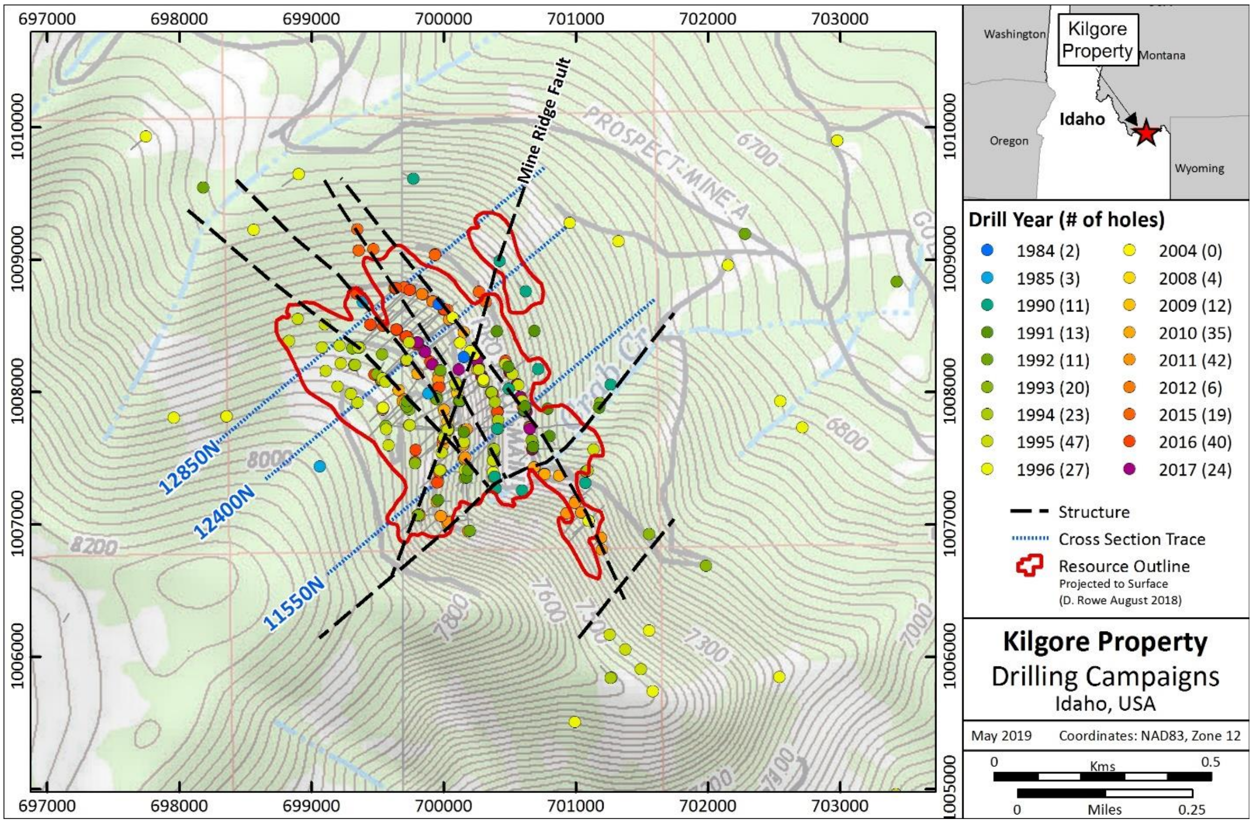


Table 10-1: Kilgore Project Drilling Summary

Operator	Year(s)	No. of Drill holes	Drill hole Type	Approx. Feet Drilled	Approx. Meters Drilled
Bear Creek	1983-1985	7	RC	8,199	2,499
Placer Dome U.S.	1990-1992	34	RC	17,205	5,244
		5	Core	3,835	1,169
Pegasus	1993-1994	23	RC	9,921	3,024
Echo Bay	1994-1996	37	RC	33,753	10,288
		67	Core	49,144	14,979
Latitude	1998	6	RC	4,072	1,241
Kilgore Gold	2002-2004	14	Core	10,889	3,319
Otis Gold	2008-2017	45	RC	27,835	8,484
		143	Core	141,690	43,187
TOTALS		**381**	**--**	**306,543**	**93,434**

10.1 Otis Gold Drilling Exploration 2012 through 2017

Drilling exploration carried out at the Kilgore Project prior to 2012 is described in detail in the 2012 NI 43-101 Technical Report prepared by Cameron (2012). This report documents drilling carried out by Otis Gold from 2012 through 2017.

10.1.1 Type and Extent

Drilling exploration carried out by Otis Gold from 2012 through 2017 consists of 45 RC holes and 45 diamond core holes (one of which was drilled for metallurgical testing) for a total of 22,536 meters drilled. Drill hole locations are shown on Figure 10-2, and drill hole details for each of the 2012, 2015, 2016, and 2017 drill campaigns are summarized in Table 10-2 through Table 10-5, respectively.

Figure 10-2: 2012-2017 Otis Gold Drilling Exploration

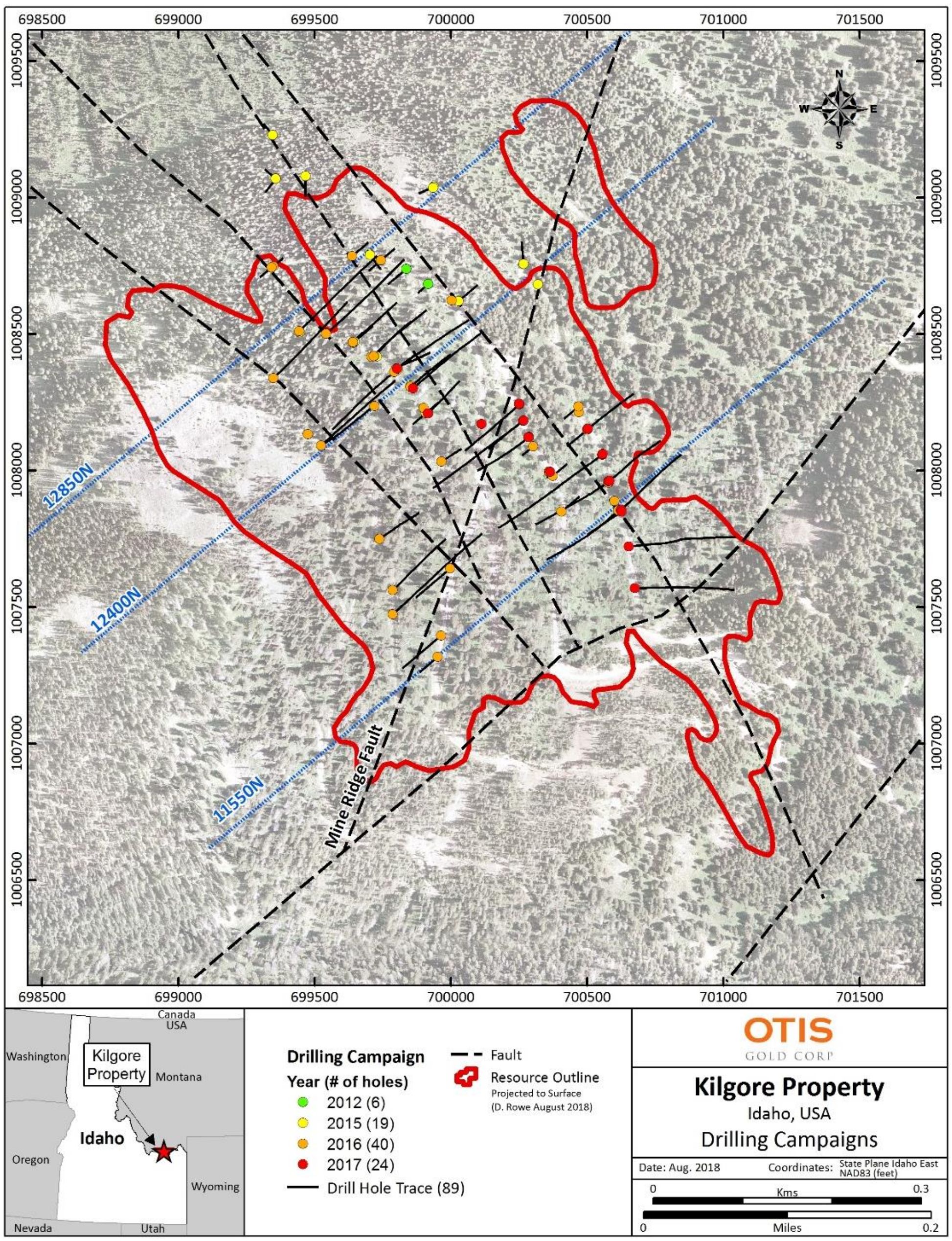


Table 10-2: 2012 Otis Gold Drilling Exploration – Drill Hole Summary Table

Hole ID	Type	Depth (m)	Collar Location (NAD83, UTM Z12)		Elevation (m)	Azimuth	Dip
			East	North			
12OKR-290	RC	166	420454.39	4920597.1	2275.8	0	-90
12OKR-291	RC	186	420454.39	4920597.1	2275.8	230	-70
12OKR-292	RC	137	420455.39	4920598.1	2275.8	230	-50
12OKR-293	RC	200	420478.89	4920579.9	2277.0	0	-90
12OKR-294	RC	152	420478.89	4920579.9	2277.0	230	-70
12OKR-295	RC	168	420425.8	4920606.9	2275.9	230	-70

Table 10-3: 2015 Otis Gold Drilling Exploration – Drill Hole Summary Table

Hole ID	Type	Depth (m)	Collar Location (NAD83, UTM Z12)		Elevation (m)	Azimuth	Dip
			East	North			
15OKR-296	RC	198	420419.14	4920607.66	2278.4	144	-70
15OKR-297	RC	183	420514.36	4920556.75	2276.9	50	-60
15OKR-298	RC	107	420485.84	4920699.84	2231.1	246	-55
15OKR-299	RC	128	420485.55	4920687.07	2232.1	0	-90
15OKR-300	RC	165	420585.41	4920600.92	2234.7	355	-60
15OKR-301	RC	91	420601.39	4920577.43	2233.5	0	-90
15OKR-302	RC	236	420473.75	4920435.93	2313.1	50	-75
15OKR-303	RC	182	420473.75	4920435.93	2313.1	0	-90
15OKR-304	RC	223	420443.36	4920484.88	2312.6	0	-90
15OKR-305	RC	192	420393.94	4920515.65	2312.6	0	-90
15OKR-306	RC	162	420334.46	4920528.28	2311.6	0	-90
15OKR-307	RC	244	420473.75	4920435.93	2313.1	230	-75
15OKR-308	RC	235	420457.13	4920466.47	2312.5	0	-90
15OKR-309	RC	213	420394.94	4920516.67	2312.7	50	-65
15OKR-310	RC	219	420420.03	4920500.1	2313.2	0	-90
15OKR-311	RC	107	420312.55	4920696.07	2234.2	311	-60
15OKR-312	RC	152	420315.54	4920695.07	2234.2	217	-65
15OKR-313	RC	122	420332.54	4920696.07	2234.2	180	-50
15OKR-314	RC	107	420306.55	4920736.07	2209.8	320	-60

Table 10-4: 2016 Otis Gold Drilling Exploration – Drill Hole Summary Table

Hole ID	Type	Depth (m)	Collar Location (NAD83, UTM Z12)		Elevation (m)	Azimuth	Dip
			East	North			
16OKC-321	Core	393	420433.98	4920238.97	2335.9	50	-60
16OKC-322	Core	307	420433.3	4920211.89	2334.8	50	-70
16OKC-326	Core	332	420610.57	4920368.96	2260.3	0	-90
16OKC-327	Core	307	420419.78	4920296.17	2339.7	50	-80
16OKC-331	Core	305	420393.46	4920516.47	2312.5	0	-90
16OKC-332	Core	335	420356.09	4920401.19	2348.6	50	-70
16OKC-333	Core	322	420416.33	4920444.61	2328.0	230	-75
16OKC-334	Core	261	420393.46	4920516.47	2312.5	50	-70
16OKC-335	Core	328	420393.46	4920516.47	2312.5	50	-80
16OKC-337	Core	173	420303.91	4920477.31	2330.3	50	-45
16OKC-340	Core	306	420333.36	4920529.52	2311.7	50	-65
16OKC-341	Core	143	420341.65	4920414.25	2348.3	0	-90
16OKC-344	Core	299	420363.6	4920526.04	2311.6	50	-65
16OKC-345	Core	300	420414.01	4920499.48	2313.3	50	-70
16OKC-349	Core	322	420441.2	4920485.98	2312.5	50	-80
16OKC-350	Core	332	420416.84	4920499.85	2313.1	50	-80
16OKC-351	Core	372	420438.83	4920482	2312.6	230	-75
16OKC-352	Core	305	420457.12	4920465.63	2312.5	50	-80
16OKC-353	Core	305	420457.12	4920465.63	2312.5	0	-90
16OKC-354	Core	335	420490.37	4920381.64	2315.4	50	-80
16OKR-315	RC	198	420610.74	4920367.88	2260.0	0	-90
16OKR-316	RC	183	420614.64	4920363.81	2259.5	50	-77
16OKR-317	RC	183	420624.04	4920324.09	2257.5	50	-75
16OKR-318	RC	229	420593.13	4920397.11	2261.7	230	-80
16OKR-319	RC	259	420471.53	4920441.56	2312.8	0	-90
16OKR-320	RC	283	420416.35	4920500.38	2313.3	0	-90
16OKR-323	RC	174	420644.99	4920434.52	2233.8	0	-90
16OKR-324	RC	183	420676.7	4920355.12	2227.9	0	-90
16OKR-325	RC	219	420683.13	4920334.62	2224.2	0	-90
16OKR-328	RC	213	420483.33	4920163.53	2308.3	230	-73
16OKR-329	RC	213	420487.13	4920187.26	2307.9	230	-55
16OKR-330	RC	226	420498.6	4920262.05	2308.8	230	-60
16OKR-336	RC	198	420305.54	4920602.02	2279.6	50	-75
16OKR-338	RC	198	420393.87	4920612.52	2275.9	50	-70
16OKR-339	RC	198	420425.92	4920607.47	2275.8	50	-75
16OKR-342	RC	213	420504.49	4920561.37	2277.0	0	-90
16OKR-343	RC	198	420303.7	4920600.92	2279.6	230	-75
16OKR-346	RC	244	420644.57	4920440.69	2234.1	230	-72
16OKR-347	RC	223	420653.79	4920414.92	2232.6	230	-61
16OKR-348	RC	174	420687.41	4920324.25	2222.0	230	-75

Table 10-5: 2017 Otis Gold Drilling Exploration – Drill hole Summary Table

Hole ID	Type	Depth (m)	Collar Location (NAD83, UTM Z12)		Elevation (m)	Azimuth	Dip
			East	North			
17OKC-355	Core	222	420654.57	4920415.86	2232.8	0	-90
17OKC-356	Core	460	420442.5	4920485.9	2312.6	50	-86
17OKC-357	Core	429	420690.42	4920324.63	2222.0	230	-75
17OKC-358	Core	319	420442.38	4920486.01	2312.7	50	-72
17OKC-359	Core	386	420690.63	4920324.79	2222.0	50	-75
17OKC-360	Core	298	420460.12	4920463.84	2312.6	50	-70
17OKC-361	Core	401	420676.93	4920356.74	2226.5	230	-75
17OKC-362	Core	305	420476.29	4920435.51	2313.2	50	-80
17OKC-363	Core	341	420678.08	4920357.48	2226.6	50	-75
17OKC-364	Core	299	420476.07	4920435.44	2313.2	0	-90
17OKC-365	Core	335	420670.38	4920386.79	2228.9	230	-65
17OKC-366	Core	335	420535.9	4920422.91	2290.9	0	-90
17OKC-367	Core	306	420670.99	4920387.36	2228.8	0	-90
17OKC-368	Core	338	420582.68	4920425.74	2264.8	230	-65
17OKC-369	Core	289	420654.64	4920415.67	2232.7	50	-75
17OKC-370	Core	333	420583.14	4920426	2264.9	230	-80
17OKC-371	Core	420	420698.68	4920283.76	2217.6	85	-71
17OKC-372	Core	305	420588.72	4920407.8	2263.2	0	-90
17OKC-373	Core	319	420704.53	4920237.02	2212.8	85	-71
17OKC-374	Core	301	420588.25	4920407.42	2263.1	230	-75
17OKC-375	Core (PQ-met)	186	420691.21	4920322.86	2222.0	230	-75
17OKC-376	Core	277	420578.76	4920444.34	2266.5	230	-75
17OKC-377	Core	283	420578.35	4920444.74	2266.9	0	-90
17OKC-378	Core	198	420611.81	4920368.31	2260.0	0	-90
17OKC-379	Core (PQ)	288	420442.96	4920485.3	2312.7	50	-86

10.1.2 Procedures

Core holes were drilled by Timberline Drilling of Coeur d' Alene, Idaho, using (depending on campaign) a Longyear LF-90 on tracks, a Sandvick DE-140 on skids, and two Atlas Copco CS14 track-mounted core rigs, with support equipment consisting of a water truck and a 10,000-pound, all-wheel drive forklift (Photo 10-1). Timberline employed standard core drilling methods incorporating triple-tube core recovery, face-discharge bits, and mixed water and bentonite downhole muds. RC holes were drilled by Okeefe Drilling Company out of Butte, Montana, using a Foremost 650 Prospector drill rig outfitted with a circulating wet splitter and support equipment consisting of a water buggy and a skidder to carry the rods. GRE knows of no drilling, sampling, or recovery factors that could materially impact the reliability of the results.

Photo 10-1: Typical Timberline Core Drilling Set Up

10.1.3 Interpretation and Relevant Results

Otis Gold's 2012 drilling program consisted of 1,009 meters of drilling in six RC holes designed to offset and extend the greater than 100-meter-thick, near surface intercepts encountered in 2011 in the North Target area located just north of the northwestern-most extent of the primary Kilgore resource area. All six of the 2012 holes encountered mineralization, with four holes returning significant bulk-tonnage thicknesses and grades (Table 10-6). The 2012 drill results served to better define and extend the North Target portion of the Kilgore resource area, which remains open to the northwest along the strike of the deposit.

Table 10-6: 2012 Significant Intercepts

Hole ID	From (meters)	To (meters)	Thickness (meters)	Grade (g/T Au)
12 OKR-290	13.7	15.2	1.5	1.03
	35.1	26.6	1.5	0.98
	61.0	74.7	13.7	0.311
	102.1	112.8	10.7	0.55
12 OKR-291	3.0	7.6	4.6	0.67
	12.2	16.8	4.6	1.32
	45.7	128.0	82.3	0.95
Includes	105.2	115.8	10.6	2.21
12 OKR-292	6.1	128.0	121.9	1.04
Includes	38.1	83.8	45.7	1.52

Hole ID	From (meters)	To (meters)	Thickness (meters)	Grade (g/T Au)
12 OKR-293	33.5	39.6	6.1	0.45
	45.8	48.8	3.0	0.63
	76.2	89.9	13.7	0.30
	111.3	114.3	3.0	1.17
	164.6	166.1	1.5	1.48
12 OKR-294	15.2	29.0	13.8	1.09
	61.0	144.8	83.8	1.12
Includes	96.0	126.5	30.5	2.10
12 OKR-295	3.0	13.7	10.7	0.42
	38.1	73.2	35.1	0.77
	112.8	129.5	16.7	0.53

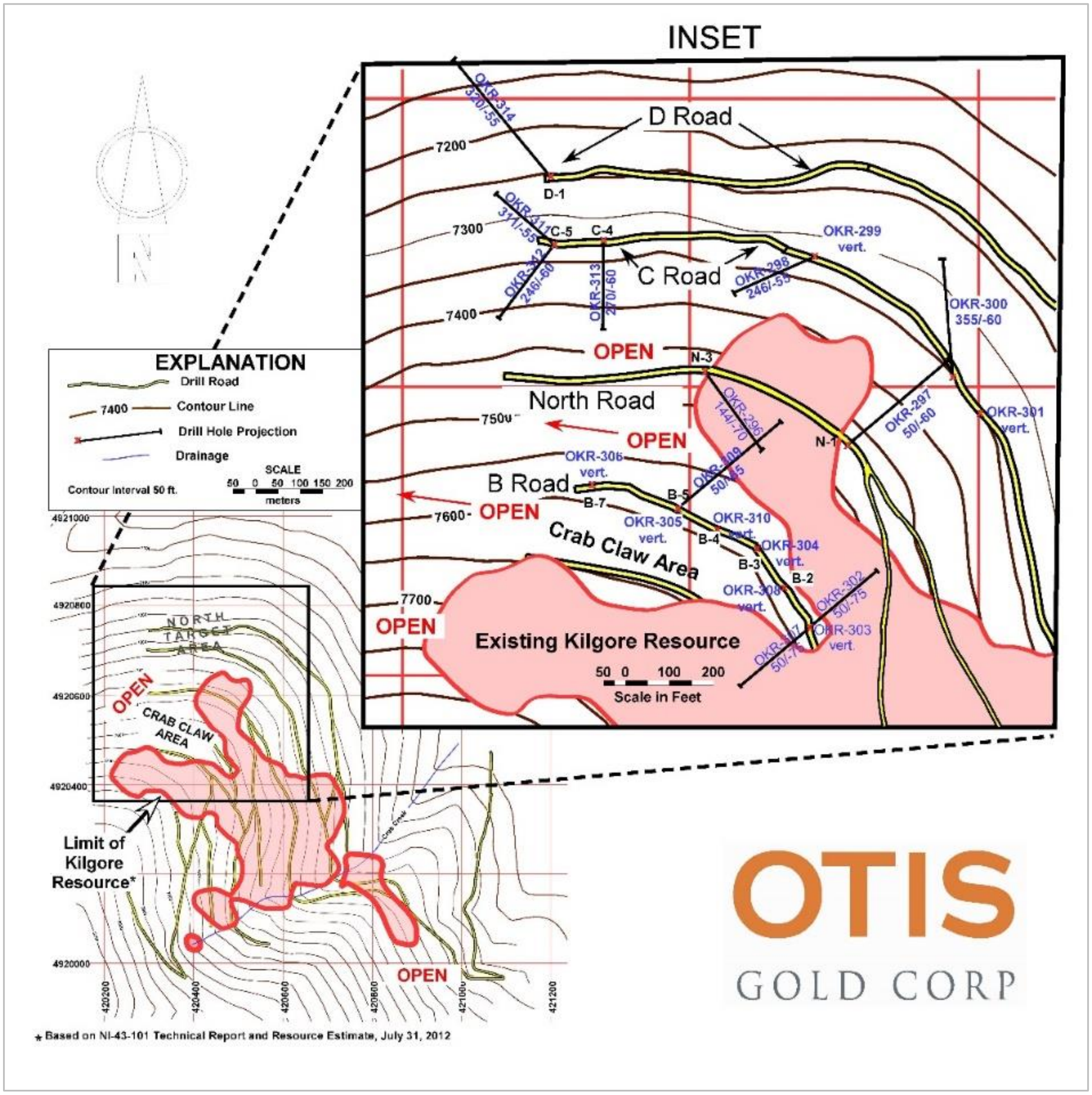
Note: The gold grade calculation is a weighted mean with a 0.250 g/T top and bottom cutoff. The grade calculation includes internal waste and low grade sections. Holes OKR-290 and OKR-293 are vertical; the remainder were drilled at a 230° azimuth to intercept the general strike of the deposit and structural features (i.e., Northwest Fault) at right angles so as to provide a close approximation to true thickness.
RC hole 12 OKR-292 was drilled as a twin to Otis Gold core hole 11 OKC-258, which contains 114.3 meters of 0.89 g/T Au from 6.1 meters to 120.4 meters. Assay results from both holes compare relatively well with one another in terms of overall average grade, thickness, depth, and continuity of mineralization.

In 2015, Otis Gold drilled a total of 10,712 feet (3,265 meters) in 19 RC holes. Drilling targeted two areas, the “Crab Claw,” a large and untested gap along the western boundary of the Kilgore resource area, and the North Target area immediately to the north (Figure 10-3).

Of the 10 holes drilled in the Crab Claw area, drill hole 15 OKR-296 was drilled along the North Road, and the remaining nine, holes 15 OKR-302 through 15 OKR-310, were drilled along the newly constructed B Road. The holes were designed to test a roughly 820-foot (250-meter) long by 394-foot (120-meter) wide, previously untested gap in the western part of the known resource area just northwest of the northeast-trending Mine Ridge Fault. All 10 holes intersected gold mineralization, and six holes encountered intercepts ranging from 164 to 328 feet (50 to 100 meters), with associated gold grades of 0.0175 to 0.1225 opt (0.6 to 4.2 g/T) (Table 10-7). Six of the 10 holes drilled in the Crab Craw area terminated in mineralization. These six holes (identified by an asterisk in Table 10-7) include four vertical holes and two holes oriented N50°E, together spanning a lateral distance of approximately 820 feet (250 meters). Based on the drilling results, Otis Gold interprets the Crab Claw as a western extension, largely open at depth, of the higher-grade core of the primary Kilgore resource area.

All 9 holes drilled in the North Target area were intended to test gold-in-soil anomalies generated during a 2013 soil survey. Two of the nine holes encountered low grade anomalous gold over intercepts of roughly 164 feet (50 meters) (Table 10-8). The mixed results from the North Target area drilling indicate at least some potential northern expansion of the known resource area, which Otis Gold followed up on in 2016.

Figure 10-3: 2015 Drilling in the Crab Claw and North Target Areas



Otis Gold, 2018

Table 10-7: 2015 Significant Intercepts, Crab Claw Target Area

Hole ID	Site	TD (meter)	Azimuth/ Angle	From-To (m)	Intercept (meter)	Au Grade (g/T)	Ag Grade (g/T)
15 OKR-296	N-3	198	144/-70	16.8-117.3 161.5-173.7	100.5 12.2	0.60 0.63	N/D
15 OKR-302*	B-1	236	50/-75	135.6-231.6	100.0	0.57	N/D
15 OKR-303*	B-1	182.3	-90	61.0-85.3 166.1-182.3	24.3 16.2	1.30 0.76	N/D
15 OKR-304*	B-3	222.5	-90	21.3-41.1 166.1-222.5	19.8 56.4	1.22 2.05	N/D
15 OKR-305	B-5	192	-90	128.0-187.5	59.5	3.79	N/D
15 OKR-306	B-7	161.5	-90	16.8-47.2	30.6	Anomalous (0.22)	N/D
15 OKR-307	B-1	244	230/-75	89.9-112.8 167.6-195.1	22.9 27.5	0.84 0.83	N/D
15 OKR-308*	B-2	234.7	-90	53.3-57.9 184.4-234.7	4.6 50.3	1.03 4.24	2.3 6.8
15 OKR-309*	B-5	213.4	50/-65	32.0-83.8 118.9-213.4	51.8 94.5	0.64 4.21	0.64 29.6
15 OKR-310*	B-4	219.5	-90	86.9-103.6 195.1-219.5	16.8 24.4	0.60 0.94	N/D

Table 10-8: 2015 Significant Intercepts, North Target Area

Hole Number	Site	TD (meters)	Azimuth/Angle	From-To (meters)	Intercept (meters)	Au Grade (g/T)
15 OKR-297	N-1	183	50/-60	19.8-70.1	50.3	Anomalous (0.2)
15 OKR-298	C-3	107	246/-55	No Significant Intercepts		
15 OKR-299	C-3	128	-90	No Significant Intercepts		
15 OKR-300	C-2	165	355/-60	No Significant Intercepts		
15 OKR-301	C-1	91	-90	No Significant Intercepts		
15 OKR-311	C-5	107	311.-60	No Significant Intercepts		
15 OKR-312	C-5	153	217/-65	No Significant Intercepts		
15 OKR-313	C-4	122	180/-50	70.1-120.4	50.3	Anomalous (0.16)
15 OKR-314	D-1	107	320/-60	No Significant Intercepts		

In 2016, Otis Gold completed a 40-hole, combined RC and core drilling program. The drilling program was designed based on an updated set of geologic cross sections and long sections incorporating all previous Kilgore drill results through 2015. The 2016 drill holes are located along the Main Road, Segment 1 Road, North Road and B Road (the location of the Crab Claw drilling completed in 2015). Drill hole locations were selected to target mineralization at depth in the Aspen Formation and to infill and define the limits of known mineralization, particularly in the southwestern portion of the deposit where historic drilling is sparse.

Twenty-five of the 40 holes drilled in 2016 encountered mineralization in the Aspen Formation, and an additional 11 holes encountered mineralization in Tertiary lithic tuff and dikes, the primary host of gold mineralization in the defined resource area. Significant intercepts from the 2016 drilling exploration are presented in Table 10-9. True widths are estimated at between 80% and 100% of the drilled interval, based on their estimated dip, association with diking and the orientation of sedimentary bedding, and continuity of mineralization between drill holes.

Table 10-9: 2016 Significant Intercepts

Hole Number	Hole Type	TD (meter)	Azimuth/ Angle	From - To (m)	Intercept (m)[4]	Au Grade (g/T)	Primary Host Rock Unit(s)
16 OKR-315	RC	126.5	-90˚	96.0 - 126.5	30.5	5.37 [1]	Tertiary Sill and Aspen Formation
16 OKR-316	RC	182.9	50/-77	126.5-182.9	56.4	0.85	Bottomed in Aspen Formation
16 OKR-317	RC	182.9	50/-75	32.0-50.3 64.0-67.1 103.6-115.8 120.4-181.4	18.3 3.1 12.2 61.0	0.50 0.72 0.34 1.03[2]	Felsic Dike Tertiary Sill Tertiary Sill Aspen Formation
16 OKR-318	RC	228.6	230/-80	53.3-57.9 80.8-86.9 93.0-213.4	4.6 6.1 120.4	0.51 0.46 1.55 [3]	Tertiary Sill Tertiary Sill Sill and Aspen
16 OKR-319	RC	259.0	-/-90	57.9-88.4 195.1-227.1 233.2-259.1	30.5 32.0 25.9	0.35 0.40 0.51	Dike Sill and Aspen Bottomed in Aspen
16 OKR-320	RC	283.5	-90˚	65.5 – 76.2 266.7 – 275.8	10.7 9.1	0.31 0.57	Lithic Tuff Aspen Formation
16 OKC-321	Core	392.6	50/-60	54.3-80.2 122.2-131.1 144.5-148.7 162.4-217.9	25.9 8.8 4.3 55.5	0.51 0.51 0.51 0.82	Lithic Tuff Tertiary Sill Lithic Tuff Lithic Tuff and Sill
16 OKC-322	Core	314.9	50˚/-70˚	79.2 – 86.9 100.6 – 111.3 165.5 – 180.7 207.3 – 219.5	7.6 10.7 15.2 12.2	0.54 0.52 0.65 0.92	Lithic Tuff Lithic Tuff Lithic Tuff Tertiary Sill
16 OKR-323	RC	173.7	-/-90	74.7-85.3 108.2-121 132.6-140.2	10.7 13.7 7.6	1.45 0.53 0.96	All in Aspen Formation
16 OKR-324	RC	182.9	-90˚	93.0 – 158.5 181.4-182.9	65.5 1.5	0.69 4.00	Aspen Formation (Hole ended in 4.00 g/T Au @ 182.9m)
16 OKR-325	RC	219.5	-90˚	96.0 – 117.3 138.7 – 185.9	21.3 47.2	1.27 0.81	All in Aspen Formation
16 OKC-326	Core	331.6	-90˚	89.9 – 105.2 112.8 – 125.0 129.5 – 146.3 160.0 – 198.1	15.2 12.2 16.8 38.1	0.68 0.51 0.53 0.81	All in Aspen Formation
16 OKC-327	Core	307.2	50˚/-80˚	57.9-172.2 185.0-211.8 257.6-277.4	114.3 26.8 19.8	1.00 0.67 1.09	Lithic Tuff Tertiary Sill Tertiary Sill
16 OKR -328	RC	213.4	230˚/-73˚	96.0 – 109.7 120.4 – 128.0	13.7 7.6	0.80 0.40	Lithic Tuff

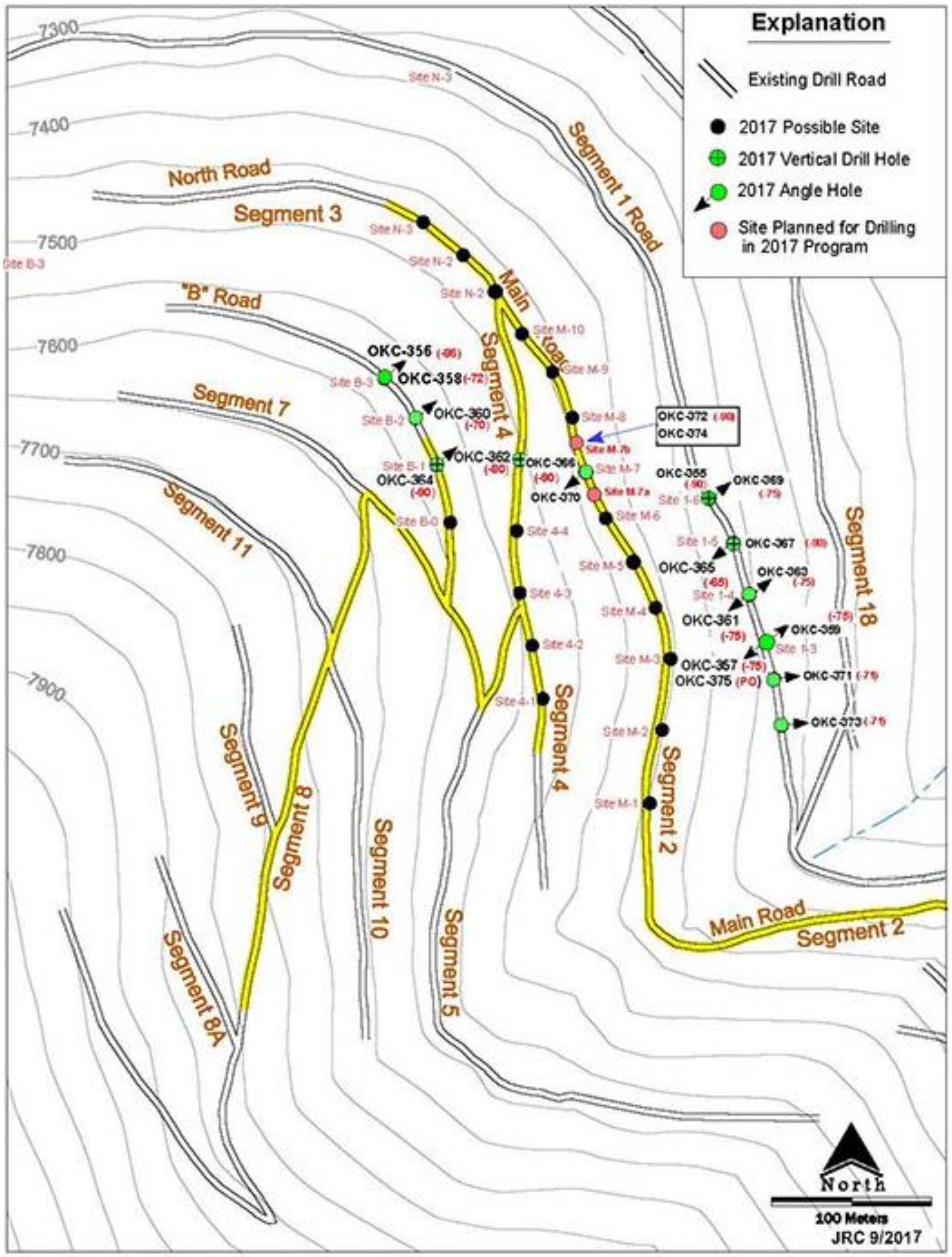
Hole Number	Hole Type	TD (meter)	Azimuth/ Angle	From - To (m)	Intercept (m)[4]	Au Grade (g/T)	Primary Host Rock Unit(s)
16 OKR-329	RC	213.4	230˚/-55˚	103.6 – 149.4	45.7	0.67	Lithic Tuff
16 OKR-330	RC	225.6	230˚/-50˚	65.5 – 115.8	50.3	2.04	Lithic Tuff
16 OKC-331	Core	304.8	-/-90˚	123.4 – 139.6 164.6 – 169.2	16.2 4.6	0.69 0.92	Lithic Tuff Tertiary Sill
16 OKC-332	Core	335.5	50˚/-70˚	53.3 – 103.6 131.1 – 141.7 285.0 – 319.4	50.3 10.6 34.4	0.63 0.91 1.28	Lithic Tuff Lithic Tuff Aspen Formation
16 OKR-336	RC	198.1	50˚/-75˚	No significant intercepts			
16 OKR-338	RC	198.1	50˚/-70˚	45.7 – 131.1 176.8 – 195.1	85.4 18.3	2.50 0.83	Tertiary Sill and Aspen Aspen Formation
16 OKR-339	RC	182.9	50˚/-75˚	4.5 – 33.5 96.0 – 114.3 153.9 – 158.5	29.0 18.3 4.6	1.17 0.75 0.95	Lithic Tuff Tertiary Sill Tertiary Sill
16 OKC-340	Core	305.7	50˚/-65˚	No significant intercepts			
16 OKR-342	RC	213.4	-/-90˚	67.1 – 76.2 189.0 – 205.7	9.1 16.7	0.69 0.91	Lithic Tuff Aspen Formation
16 OKR-343	RC	198.1	50˚/-75˚	No significant intercepts			
16 OKC-344	Core	298.7	50˚/-65˚	108.2 – 171.3	63.1	0.66	Tertiary Sill and Aspen
16 OKR-346	RC	243.8	230˚/-61˚	41.1 – 45.7 77.7 – 109.7 134.1 – 138.7 172.2 – 189.0	4.6 32.0 4.6 16.8	0.51 0.89 0.57 0.79	Tertiary Sill Aspen Formation Aspen Formation Aspen Formation
16 OKR-347	RC	222.5	230˚/-72˚	120.4 – 170.7	50.3	0.97	Aspen Formation
16 OKR-348	RC	174.7[2]	230˚/-75˚	80.8 – 97.5 105.2 – 174.3	16.7 69.1	0.43 2.07[2]	Tertiary Sill Aspen Formation

Includes 13.7 meters @ 8.71 g/T Au. This drill hole has been capped at 34.25 g/T Au (or 1.0 opt Au).
Includes 6.1 meters @ 2.26 g/T Au.
Includes 7.6 meters @ 8.86 g/T Au.
True widths are estimated at between 80% and 100% of the drilled interval, based on their estimated dip, association with diking and the orientation of sedimentary bedding, and continuity of mineralization between drill holes.
Hole OKR-348 was lost in Aspen Sandstone and ended in rock containing 5.63 g/T Au.

The 2016 drilling results indicate that gold mineralization in the Aspen Formation is more extensive than revealed by previous drill testing. Mineralization in the Aspen Formation appears to lie along a northwesterly-trending belt or corridor in the northern half of the resource area, much of which remains open for further drilling. Mineralization in the Aspen Formation is typically higher-grade and displays thicker mineralized intercepts than those comprising the current bulk of the resource hosted in the overlying volcanic rocks. Reported intercepts in the Aspen demonstrate that mineralization exists to depths of up to 984 feet (300 meters) below the surface of the deposit, and in places remains open at depth. The Tertiary intrusive sill (Tad), which directly overlies the Aspen Formation and locally intrudes the upper portion of it, is an important host of gold mineralization. Based on the results of the 2016 drilling exploration, Otis Gold considers the contact between the Tad and the Aspen an important target for future exploration. Significant intercepts exist in both the Aspen Formation and the overlying Tap, and in many cases straddle the contact between them. Some of the intercepts drilled in 2016 in the Aspen Formation along the mineralized northwest-trending corridor contain coarse-grained visible gold.

In 2017, Otis Gold completed 25 diamond core holes for a total of 26,161 feet (7,974 meters) of drilling (Figure 10-4). The primary goal of the drilling program was to follow up on open-ended drilling at depth and laterally as infill of 2015 and 2016 drill intercepts in the primary resource area, and specifically at depth in the Aspen Formation.

Figure 10-4: 2017 Drill hole Locations



In summary, the 2017 drilling results extended mineralization in the Aspen Formation 197 to 295 feet (60 to 90 meters) deeper than was previously known, largely in the central part of the deposit southeast of the Mine Ridge Fault (Figure 10-5 through Figure 10-7). Average grades in this area are generally higher than the overall average grade of the Kilgore deposit reported by Cameron (2012), and mineralization appears to be fairly continuous between holes within sections and from section to adjacent section.

Figure 10-5: 2017 Drilling Results, Section 12850N

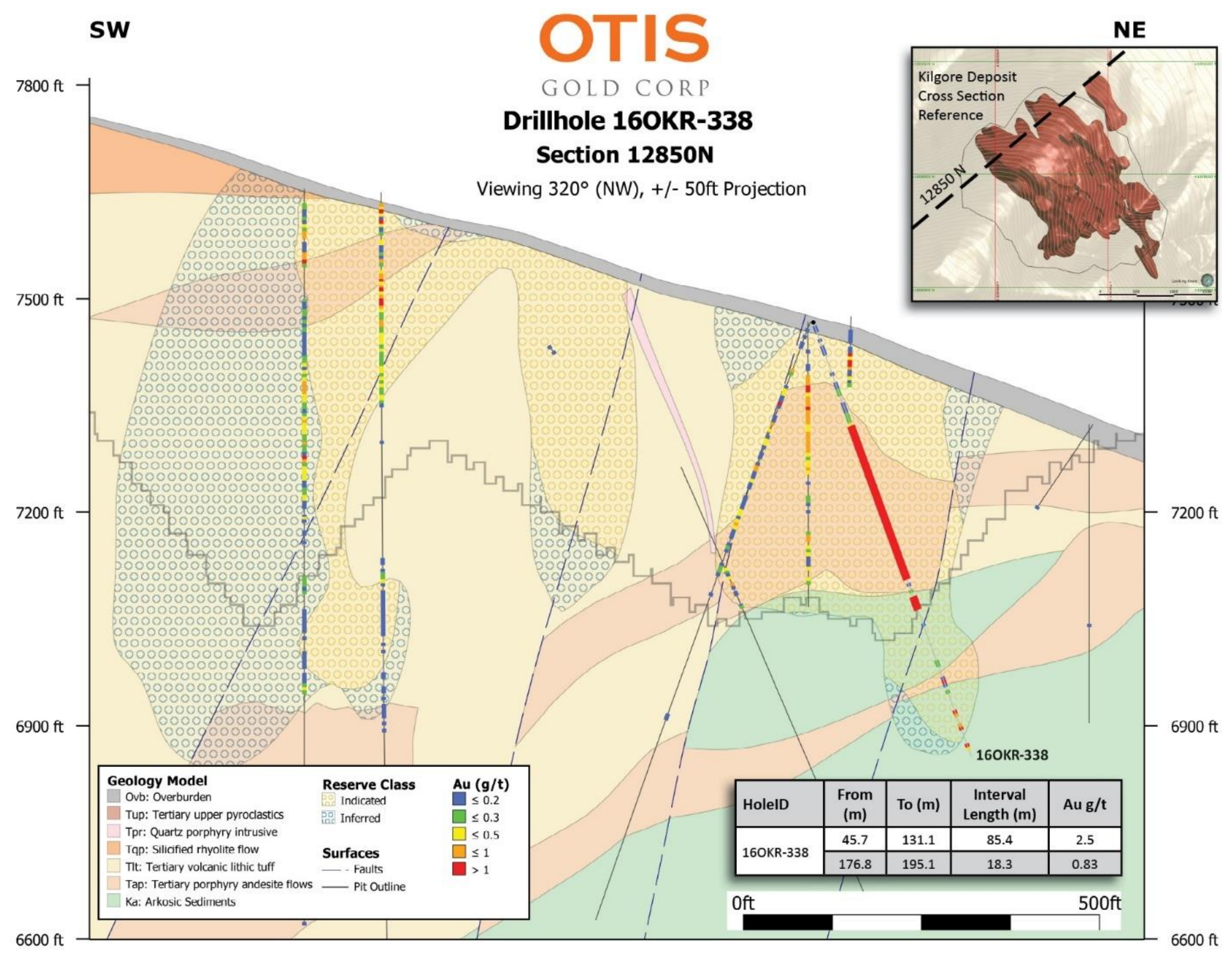
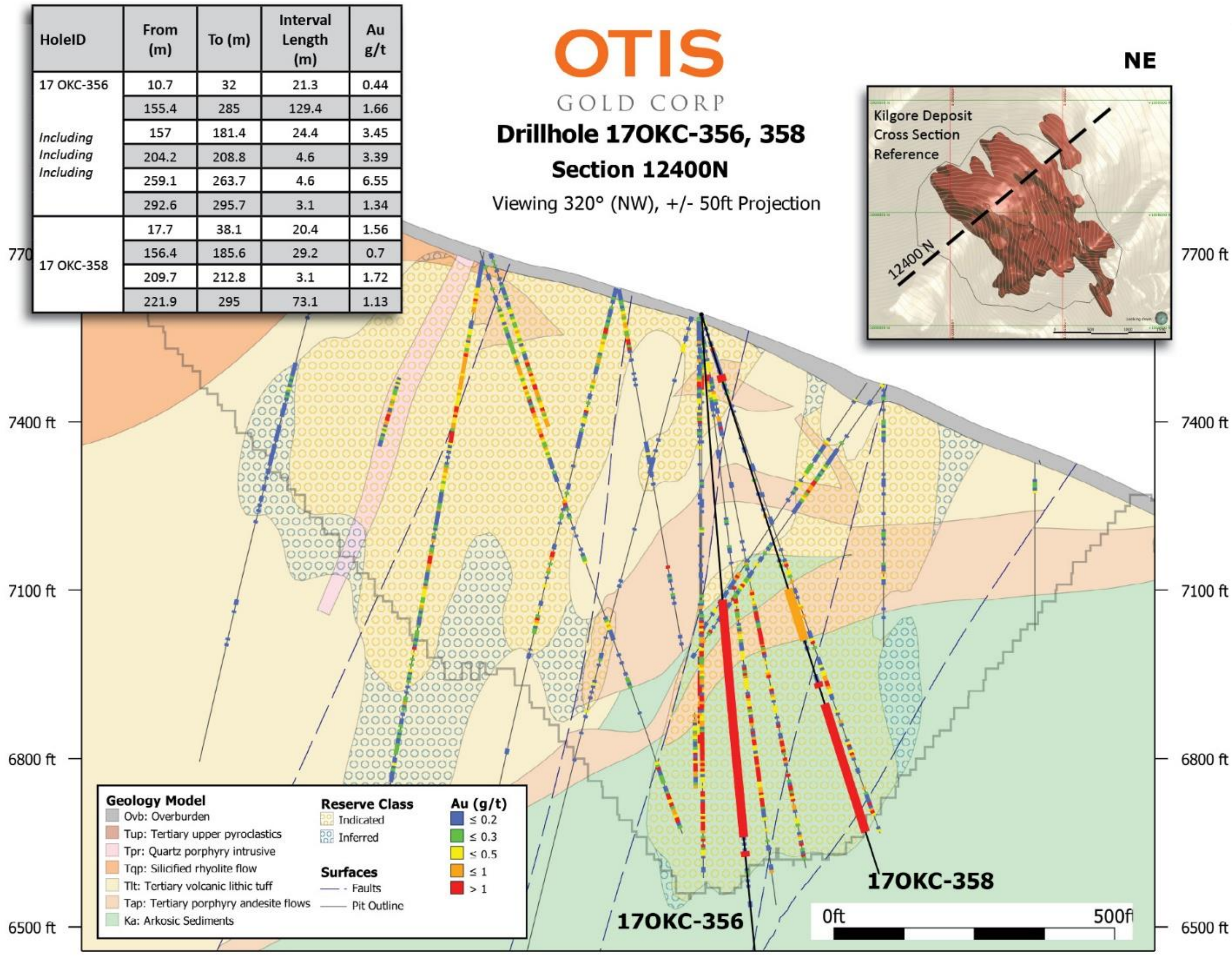
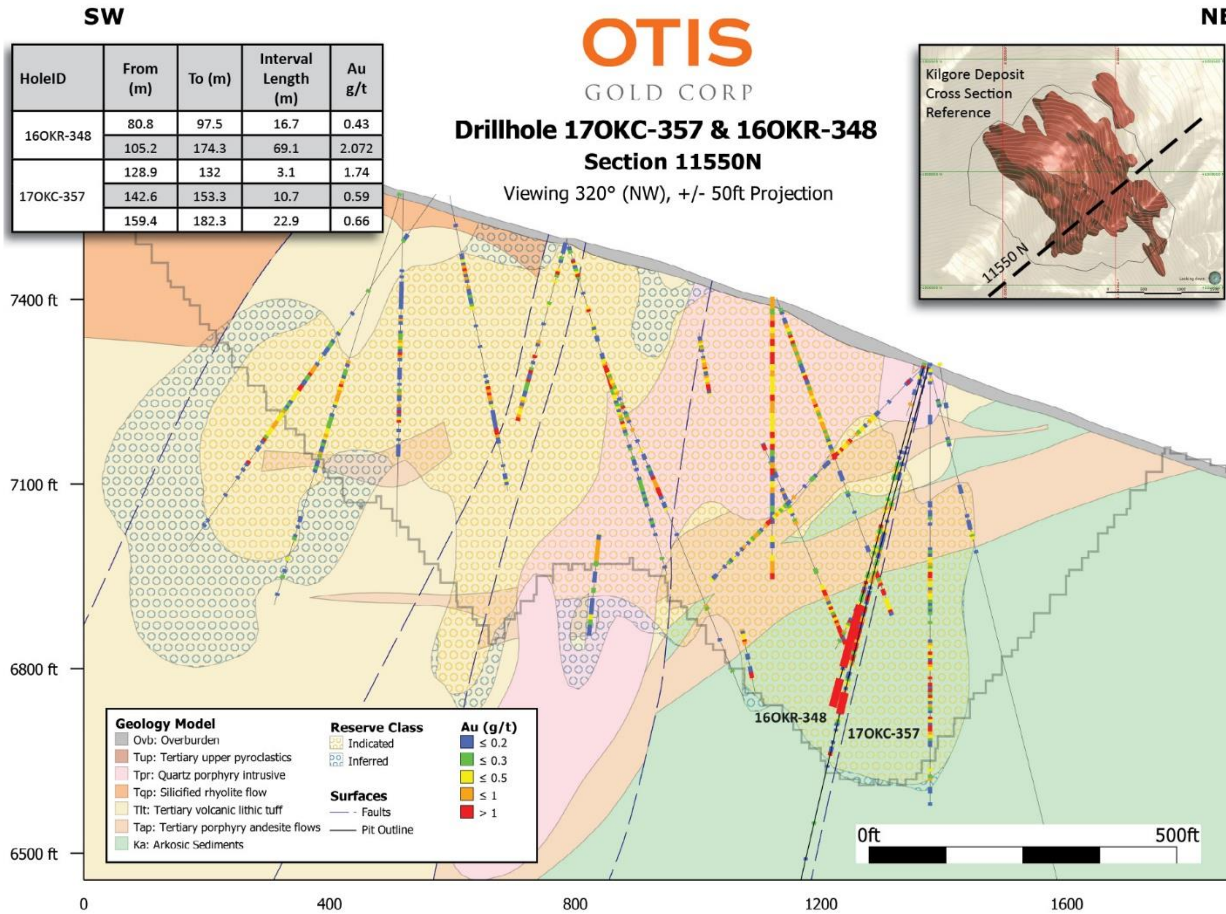


Figure 10-6: 2017 Drilling Results, Section 12400N



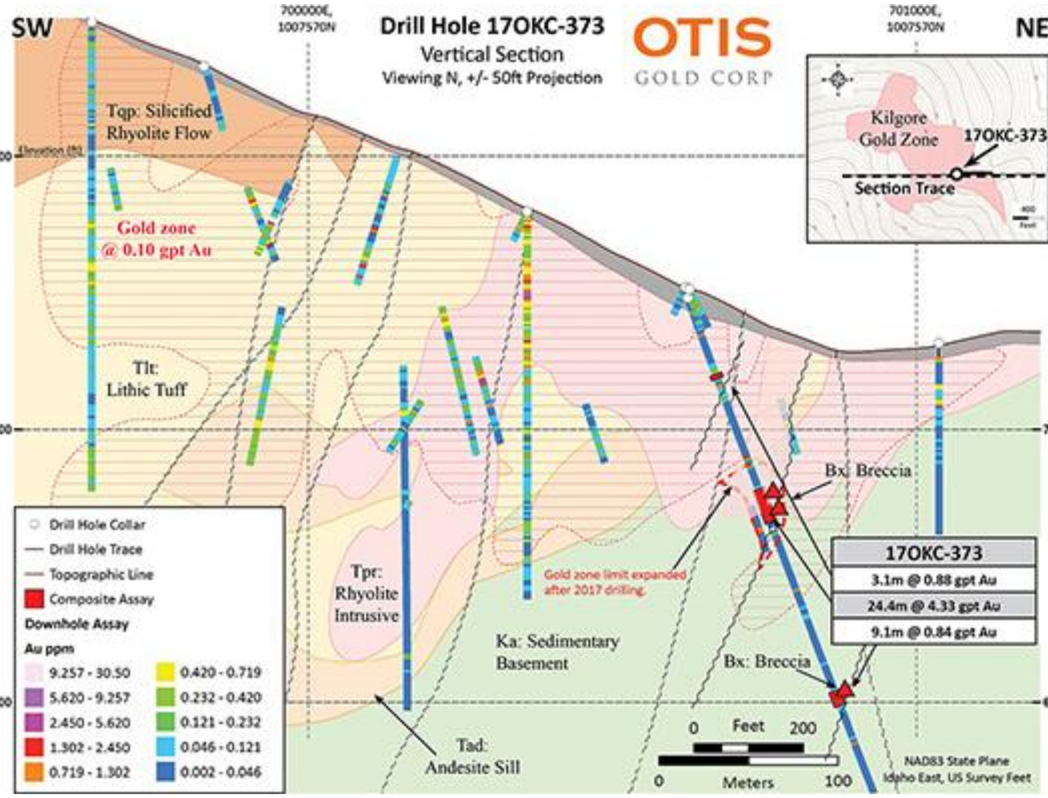
HoleID	From (m)	To (m)	Interval Length (m)	Au g/t
17 OKC-356	10.7	32	21.3	0.44
	155.4	285	129.4	1.66
Including	157	181.4	24.4	3.45
Including	204.2	208.8	4.6	3.39
Including	259.1	263.7	4.6	6.55
	292.6	295.7	3.1	1.34
17 OKC-358	17.7	38.1	20.4	1.56
	156.4	185.6	29.2	0.7
	209.7	212.8	3.1	1.72
	221.9	295	73.1	1.13

Figure 10-7: 2017 Drilling Results, Section 11550N



Drilling results also bound known mineralization to the northeast (drill hole 17 OKC-363) and to the southeast (drill hole 17 OKC-369). Hole 17 OKC-373, drilled near the Cabin fault, encountered a highly altered and brecciated intercept of 80 feet (24.4 meters) @ 0.126 opt (4.33 g/T) Au at the contact between the Tpr (rhyolite dome) and adjacent Aspen Formation (Figure 10-8). Based on this intercept, Otis Gold considers the contact between the rhyolite dome and Aspen Formation in the Cabin Fault area and beyond to the southeast, which remains largely untested, a high priority exploration target for future exploration.

Figure 10-8: 2017 Drilling Results, Drill hole 17 OKC-373



Significant intercepts from the 2017 drilling campaign are presented in Table 10-10. True widths are estimated at between 80% and 100% of the drilled interval, based on their estimated dip, association with diking, and the orientation of sedimentary bedding, and continuity of mineralization between drill holes.

Table 10-10: 2017 Significant Intercepts

Hole ID	TD (meters)	Azimuth/ Angle		From (meters)	To (meters)	Intercept (meters)	Au (gp/t)	Primary Host Rock Unit(s)
17 OKC-355	221.9	-/-90˚		85.0	96.9	11.9	0.63	Aspen
				121.3	124.4	3.1	1.56	Aspen
				139.6	162.5	22.9	0.74	Aspen
				177.7	186.8	9.1	0.78	Aspen
17 OKC-356	459.9	50˚/-86˚		10.7	32.0	21.3	0.44	Lithic Tuff
				155.4	285.0	129.4	1.66	Aspen
			Including	157.0	181.4	24.4	3.45	Aspen
			Including	*204.2*	*208.8*	*4.6*	*3.39*	Aspen
			Including	*259.1*	*263.7*	*4.6*	*6.55*	Aspen
				292.6	295.7	3.1	1.34	Aspen
17 OKC-357	429.2	230˚/-75˚		128.9	132.0	3.1	1.74	Aspen
				142.6	153.3	10.7	0.59	Aspen
				159.4	182.3	22.9	0.66	Aspen
17 OKC-358	319.4	50˚/-72˚		17.7	38.1	20.4	1.56	Lithic Tuff
				156.4	185.6	29.2	0.7	Sill & Aspen
				209.7	212.8	3.1	1.72	Aspen

Hole ID	TD (meters)	Azimuth/ Angle		From (meters)	To (meters)	Intercept (meters)	Au (gp/t)	Primary Host Rock Unit(s)
				221.9	295.0	73.1	1.13	Aspen
17 OKC-359	389.5	50˚/-75˚		No Significant Values				
17 OKC-360	298.1	50˚/-70˚		141.7	155.4	13.7	0.43	Aspen & Sill
				185.9	192.0	6.1	0.47	Aspen
				201.2	217.9	16.8	0.77	Aspen
				245.4	254.5	9.1	0.85	Aspen
				280.4	288.0	7.6	0.91	Aspen
17 OKC-361	401.4	230˚/-75˚		79.2	82.3	3.1	0.84	Andesite Sill
				93.0	203.6	110.6	0.90	Aspen
17 OKC-362	305.1	50˚/-80˚		234.1	299.6	65.5	1.21	Aspen
				68.0	82.9	14.9	0.40	Rhyolite Dike
17 OKC-363	341.4	50˚/-75˚		No Significant Values				
17 OKC-364	299.3	-/-90˚		190.5	222.5	32.0	1.09	Aspen & Sill
				231.6	294.1	62.5	1.11	Aspen
17 OKC-365	334.7	230˚/-65˚		70.1	73.2	3.1	1.22	Sill
				93.0	123.4	30.4	0.68	Sill & Aspen
				134.1	172.2	38.1	0.69	Aspen
17 OKC-366	334.7	-/-90˚		16.8	36.6	19.8	0.47	Rhyolite
				141.7	151.1	9.4	1.53	Sill
				181.4	268.2	86.8	1.05	Aspen
17 OKC-367	306.0	-/-90˚		76.9	85.3	8.4	0.33	Aspen
				106.7	114.3	7.6	0.36	Aspen
				123.4	157.0	33.6	0.56	Aspen
17 OKC-368	337.7	230˚/-65˚		46.6	67.4	20.8	0.56	Tuff, Aspen
				123.1	146.0	22.9	0.48	Sill & Aspen
				187.6	206.7	19.1	0.59	Aspen
17 OKC-369	289.3	50˚/-75˚		No Significant Values				
17 OKC-370	333.1	230˚/-80˚		57.3	64.9	7.6	0.59	Tuff
				109.1	121.6	12.5	0.43	Sill
				138.1	224.9	86.8	0.76	Aspen
				232.6	235.6	3.0	0.63	Aspen
				246.3	258.0	11.7	1.15	Aspen
17 OKC-371	420.3	85˚/-71˚		30.2	33.2	3.0	0.80	Rhyolite
				80.2	86.3	6.1	0.57	Sill
				356.5	359.4	2.9	1.53	Rhyolite
17 OKC-372	304.8	-/-90˚		57.9	62.5	4.6	0.50	Sill & Aspen
				105.2	135.6	30.5	0.78	Sill & Aspen
				169.2	172.2	3.0	0.94	Aspen
				192.0	221.0	29.0	0.71	Aspen
17 OKC-373	318.5	85˚/-71˚		45.1	48.2	3.1	0.88	Lithic Tuff
				113.7	133.5	24.4	4.33	Dike & Aspen
				234.1	243.2	9.1	0.84	Dike & Aspen
17 OKC-374	301.1	230˚/-75˚		109.1	246.3	137.2	0.91	Sill & Aspen
				282.9	289.1	6.2	0.80	Aspen

Hole ID	TD (meters)	Azimuth/ Angle		From (meters)	To (meters)	Intercept (meters)	Au (gp/t)	Primary Host Rock Unit(s)
17 OKC-375	185.9	230˚/-75˚		98.5	151.8	53.3	1.23	Sill & Aspen
				165.5	174.7	9.1	0.87	Aspen
17 OKC-376	276.8	230˚/-75		109.7	189.0	79.3	0.79	Sill & Aspen
				204.2	275.8	71.6	1.05	Aspen
17 OKC-377	282.9	-/-90˚		103.6	113.6	10.0	0.30	Sill
				121.3	240.2	118.9	1.41	Aspen
17 OKC-378	197.7	-/-90˚		4.6	21.3	16.7	0.78	Lithic Tuff
				97.5	197.7	100.2	0.74	Aspen

True widths are estimated at between 80% and 100% of the drilled interval, based on their estimated dip, association with diking and the orientation of sedimentary bedding, and continuity of mineralization between drill holes.

10.2 RC and Core Comparisons

The resource database includes 217 core drill holes and 168 RC drill holes. The data populations of both types of drilling samples were analyzed by creating an Ordinary Kriged (OK) block model and a Nearest Neighbor (NN) block model, and the blocks containing data from both drilling sets were compared. The input data was the same 10-foot composites used to create the resource model. The OK block model was 100-foot by 100-foot by 100-foot blocks limited to lithologies 3Tpr, 5Tad, 6Tlt, and 7Ka; the investigated area was also limited by the 1500 $/oz pit shell. An isotropic search of 80 feet was used. The NN block model was 20 feet by 20 feet by 10 feet and limited by the same lithology types and pit. The search was also isotropic, but a shorter 75-foot search distance was used.

Figure 10-9 and Figure 10-10 show Q-Q plots of the two different drilling models. Clearly, at any gold grade range the RC composites show a positive bias compared to the core holes.

Figure 10-9: QQ Plot of OK Blocks

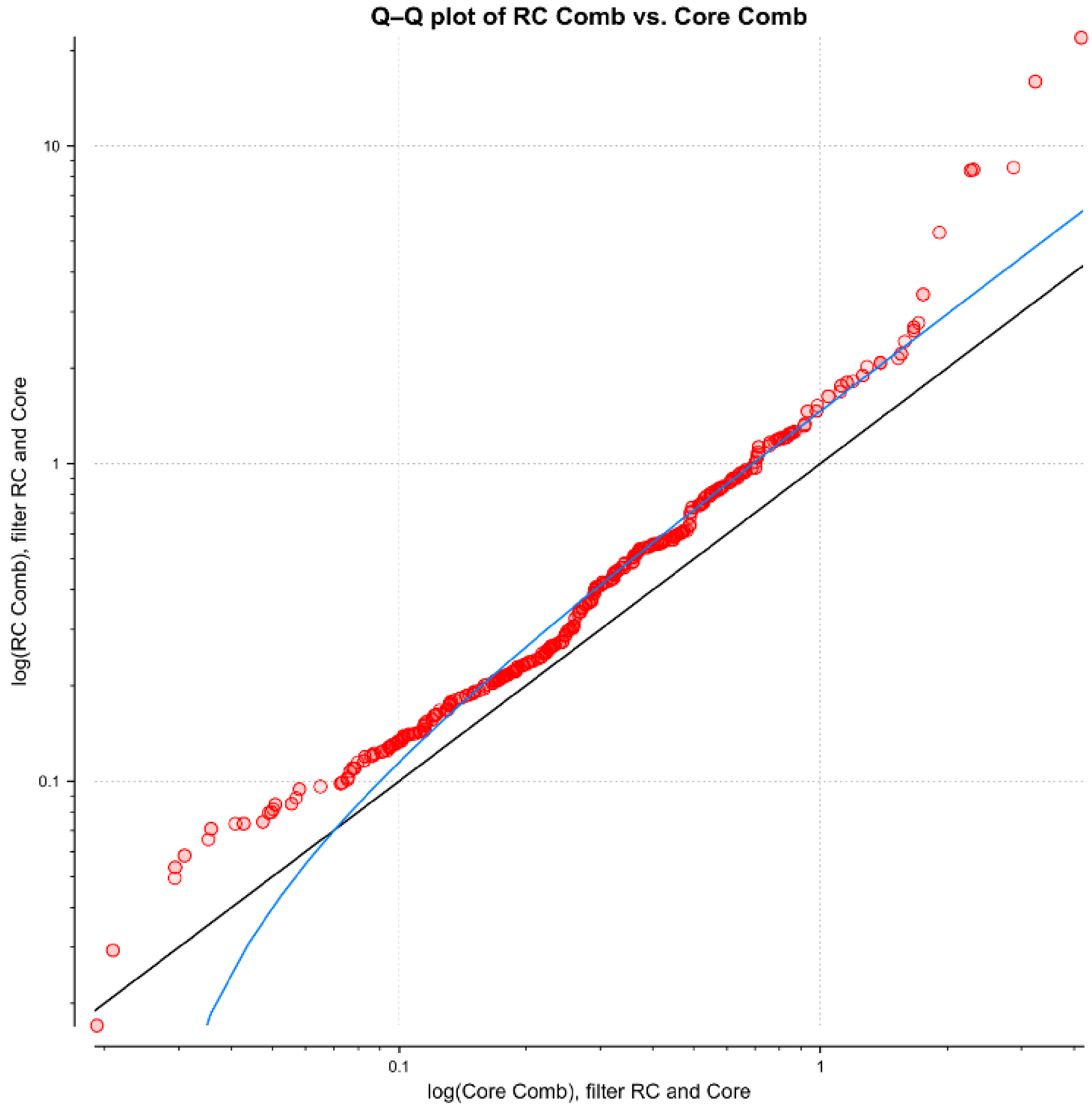


Figure 10-10: QQ Plot of NN Blocks

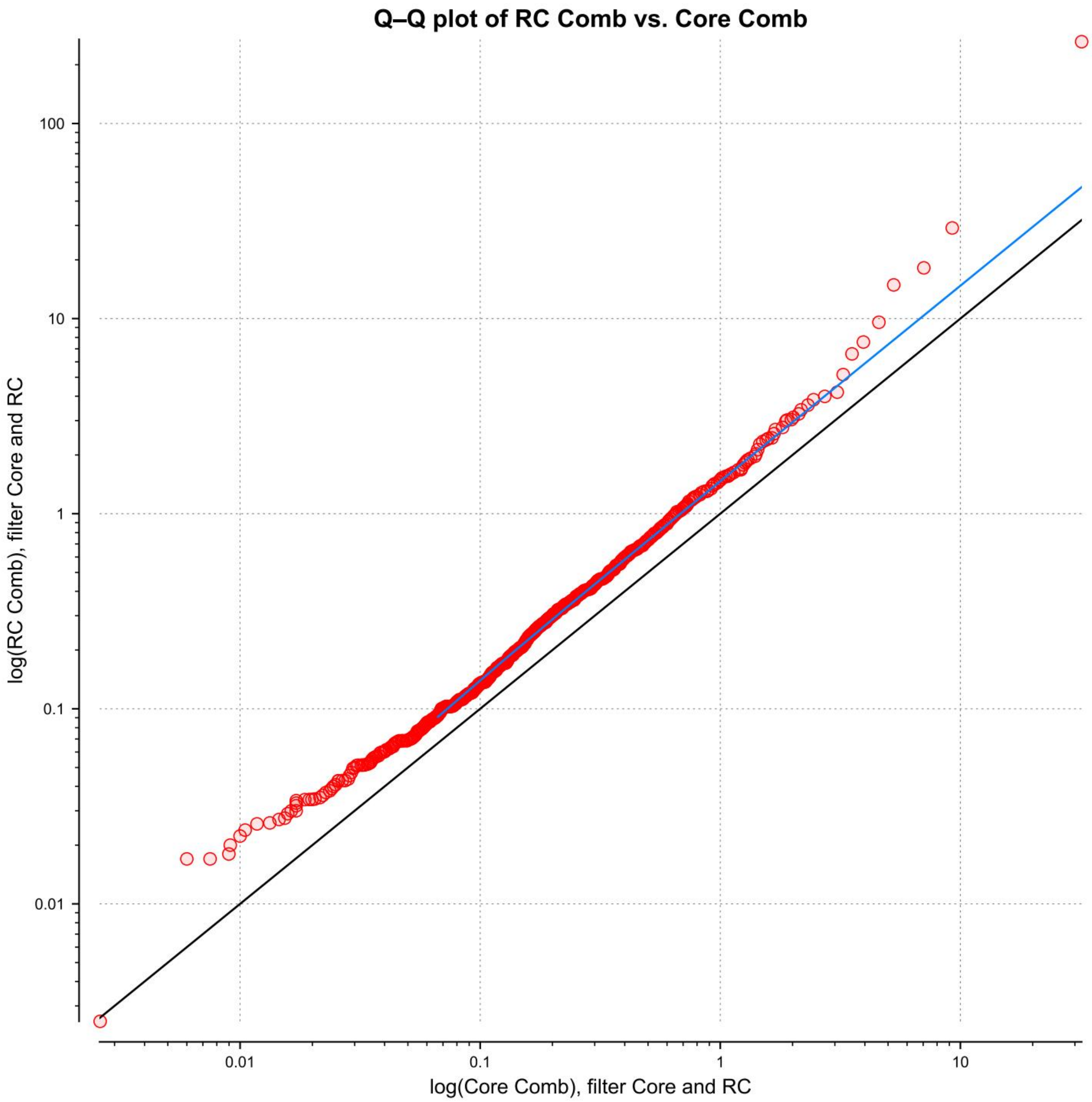


Figure 10-11 and Figure 10-12 show the blocks with common core and RC modeled grade in each of the OK and NN methods.

Figure 10-11: OK Blocks with both Core and RC Modeled Grade

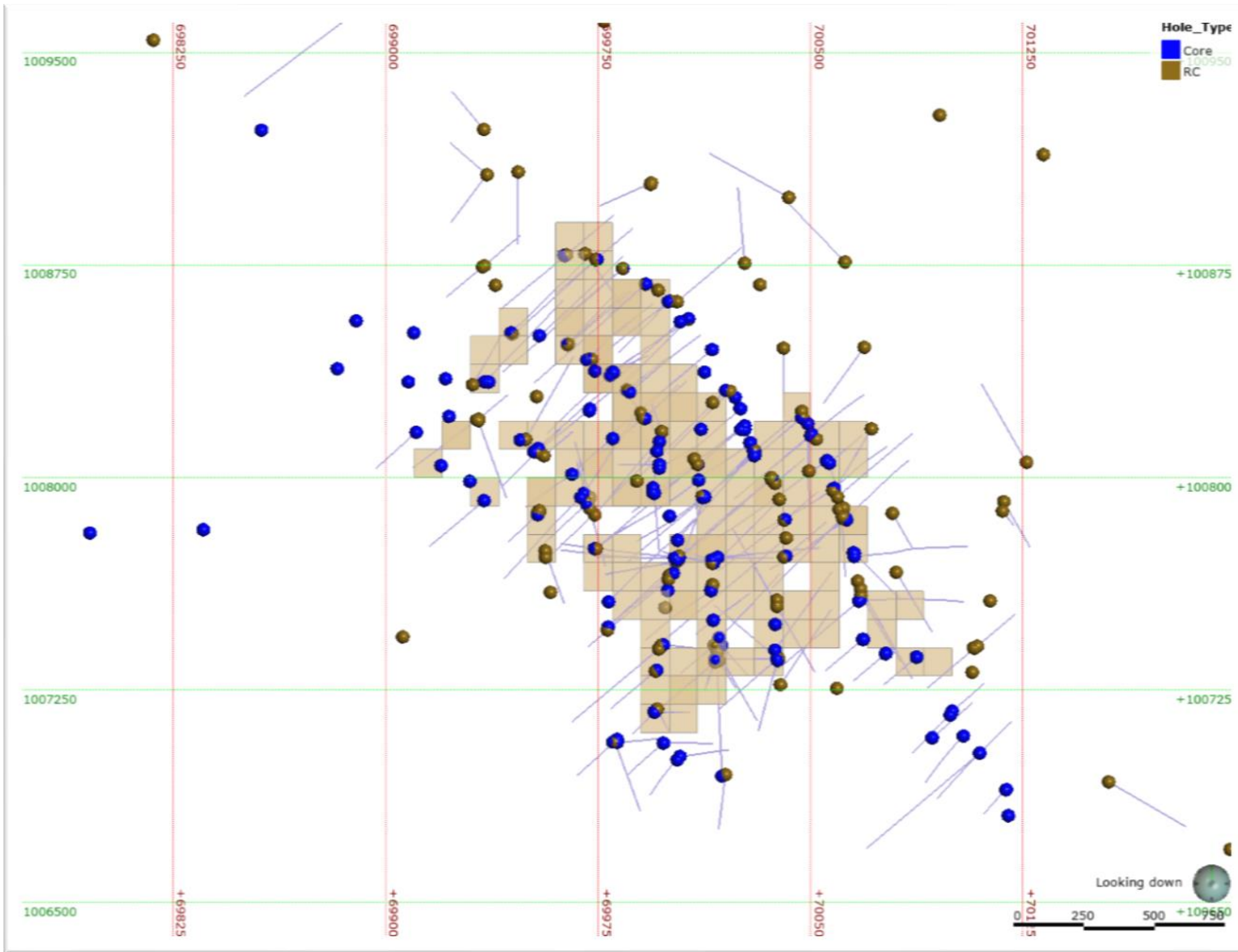
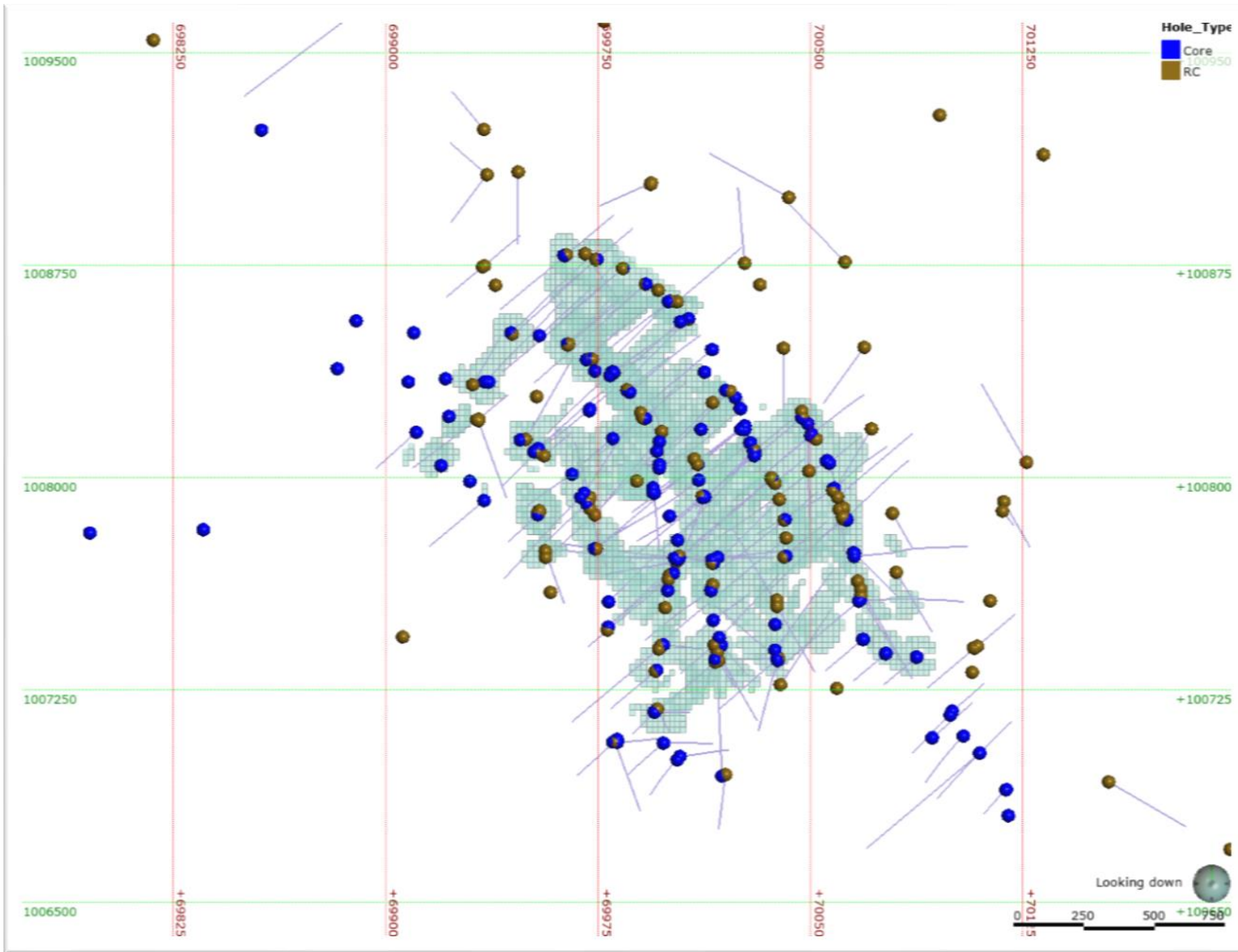


Figure 10-12: NN Blocks with both Core and RC Modeled Grade



Otis Gold geologists are aware of issues related to water inflow-induced contamination in RC holes and washing-out of soft matrix material and fracture coatings in core samples. At this time, it is evident that there is a positive grade bias in the paired RC data and a contained metal bias for RC data versus core. Unfortunately, it is not possible to know which data set is more representative of the *in-situ* mineralization without adding larger diameter core or separate bulk sample data, such as a decline, trenching, or test pits. Large diameter core drilling combined with bulk sampling of gold and silver bearing rocks as indicated by previous drilling will provide additional information for the data analysis, and would potentially lead to better understanding of distribution and variability of gold and silver. The most obvious examples of compromised drill holes are removed from the data analysis. The use of triple-tube core recovery systems combined with face-discharge drill bits used by Echo Bay and Otis Gold since 1995 is an effort to improve the quality of gold sampling in the Kilgore deposit, representing approximately 57.4% of the total footage drilled. At Kilgore, RC samples compose only 33% of the assays in the final resource database.

11.0 SAMPLE PREPARATION AND SECURITY

Sampling and analytical procedures prior to 2012 are described in detail in the 2012 NI 43-101 Technical Report prepared by Cameron (2012). This report documents the sample preparation, analysis, and security measures employed by Otis Gold from 2012 through 2017.

11.1 Sample Preparation

Otis Gold employs standard sample handling and preparation procedures during all core and RC drilling programs. RC samples are logged, bagged, and tagged at the drill rig by an Otis Gold geologist. The chip samples are collected continuously over 5-foot intervals directly from the rotary splitter, with a sample bag catching one side of the split and a sample sieve catching the other. The standard split ratio is 50/50, though that ratio is occasionally adjusted to maintain sample volume. At the end of each sample interval, the sample bag is sealed and laid out for double checking and later transport. The sample sieve is rinsed and logged by the geologist, and a representative portion is placed in the chip tray. Duplicate samples are collected every 100 feet, with the duplicate sample bag sharing the same stand as the sieve. All RC samples are transported to St. Anthony by Otis Gold staff prior to shipment to the lab.

Drill core is collected, cleaned, and placed into wax-coated core boxes at the drill site by the drill crew. The number and depth of each core run is indicated by marked wooden blocks placed at the end of the run. Core boxes are labelled in the field with the drill hole number, box number, and the associated footage interval. Filled core boxes are transported to the core storage facility in St. Anthony once daily by Otis Gold personnel.

In St. Anthony, each box of whole core is photographed and logged, and selected sample intervals are prepared for shipment to the lab. Core recovery, rock quality designations, lithology, structure, alteration and other pertinent details are recorded by Otis Gold geologists on a standard, hand-written log form. Samples are selected by Otis Gold geologists during logging and are identified by a red ribbon, marked with the sample number, placed at the beginning and end of each assay interval.

Drill core sample intervals are split with a hydraulic core splitter. Care is taken to ensure that the core is oriented appropriately to produce unbiased and representative split samples when veining is present, and the hydraulic splitter is cleaned between samples to avoid cross-contamination. One half-core of each sample interval is retained in the core box, and the other is bagged in a clean 45 x 60 cm (18 x 24-inch), 8-mil, industrial-strength, polyethylene sample bag secured with a wire tie. The hole number and sample ID are written on each bag in indelible marker, and the individual samples are then consolidated into 60 x 90 cm rice bags for transport to the lab. All samples are delivered directly to ALS-Chemex (ALS) in Elko, Nevada, by Otis Gold personnel.

11.2 Analytical Procedures

All assay work is performed at ALS Global laboratories located in Elko and Reno, Nevada. ALS is an International Organization for Standardization (ISO)-certified lab, with an ISO 9001:2008 quality management system certification and ISO 17025:2005 technical capability accreditation.

Samples received at ALS are logged into a tracking system and a bar code label is attached to each individual sample. Excessively wet or damp samples are dried in drying ovens. Samples are crushed to a standard of 70% passing a 2-mm sieve and split using a riffle splitter. A sample split of up to 1,000 grams is pulverized to greater than 85% of the sample passing a 75-µm sieve. One sample pulp is retained by the lab for analysis, and the other is sent to Otis Gold's office facility in Spokane, Washington, for cataloguing and storage.

Gold content is determined by a 50-gram fire assay with atomic absorption finish (ALS method Au-AA24) to an upper limit of 10 grams. Over-limit samples are analyzed by fire assay with gravimetric finish (Au-GRA22). Internal quality control measures employed by ALS include insertion of standards, duplicates, and blanks (about 10% of the total samples in each analytical run), and the QC data are routinely monitored to ensure that reference materials and duplicate analyses meet specific precision and accuracy requirements.

11.3 Quality Assurance and Quality Control

Otis Gold's quality assurance and quality control measures include routine insertion of blank, standard, and duplicate samples into the sample stream, and subsequent monitoring of associated analytical results.

11.3.1 Reference Materials

Certified commercial standard samples are supplied by Rocklabs, and blank material is derived from Columbia River flood basalt acquired near Spokane. A sealed kraft envelope of reference material is labelled sequentially and inserted into the sample stream at a frequency of one in every 15 to 20 samples, and the lab is instructed to analyze all samples and pulps in numerical order.

Standard and blank assay results in excess of plus or minus three standard deviations from the expected mean for the material are considered a failed result. In cases of failure, the reference sample and a select number of surrounding samples are re-analyzed, and the assay data within the database updated accordingly.

In 2017, Otis Gold analyzed a total of 236 blanks and 243 standard samples. Analytical results for eight of the 236 blank samples failed to meet the QA criteria. In all cases, the failed reference and surrounding samples were re-analyzed with positive results. Eight of the 243 standard samples also failed. The failed standards and surrounding samples were re-analyzed, with the results of the re-run superseding the original assay results. In all but one instance, the results of the re-run fell within the control limits defined for the standard material.

11.3.2 Duplicate Samples

RC field duplicates are inserted into the sample stream at a typical rate of one in every 20 samples. Core duplicates are selective, and the insertion rate is widely variable. Duplicate sample results are reviewed statistically using an average relative difference comparison and scatter plots to illustrate the strength of correlation.

In 2017, Otis Gold analyzed a total of 60 duplicate core samples. Results of the duplicate analysis are presented in Figure 11-1. Of the 60 duplicate pairs analyzed, seven have an average relative difference in excess of 1.0 (Table 11-1). Based on visual examination of the original and duplicate core samples (prior to submission to the lab), Otis Gold attributes the variable results to local variation in sample mineral quality and possibly the presence of coarse gold.

Figure 11-1: Scatter Plot of 2017 Duplicate Sample Results

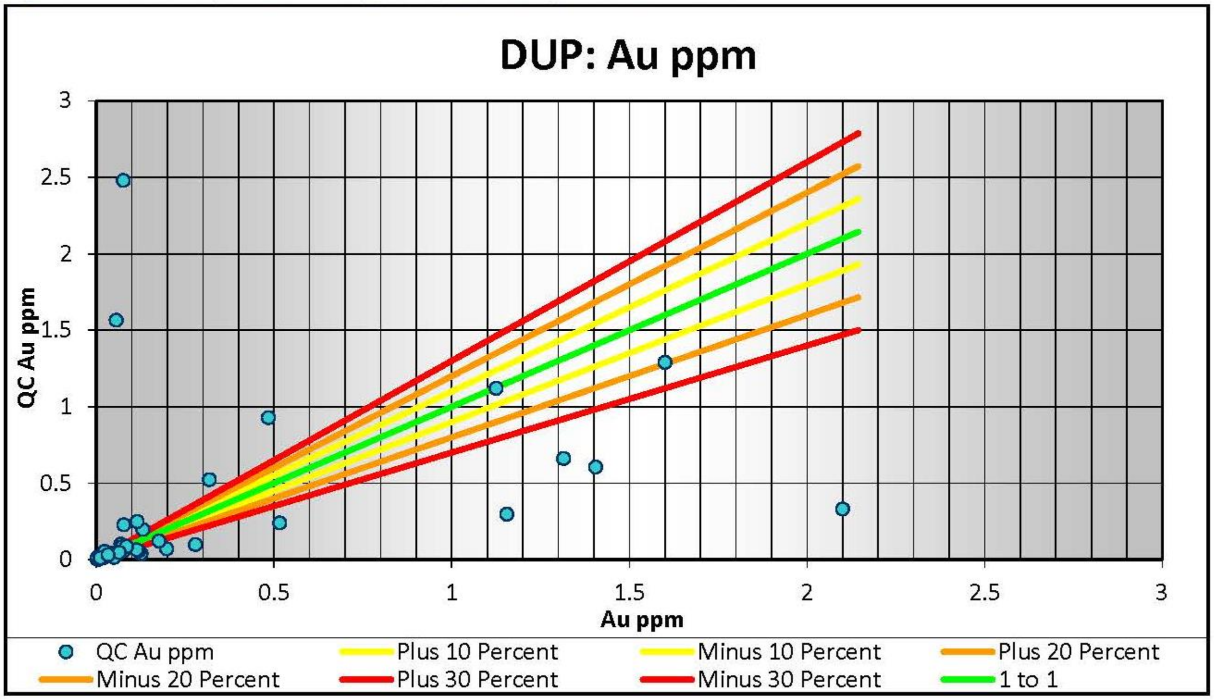


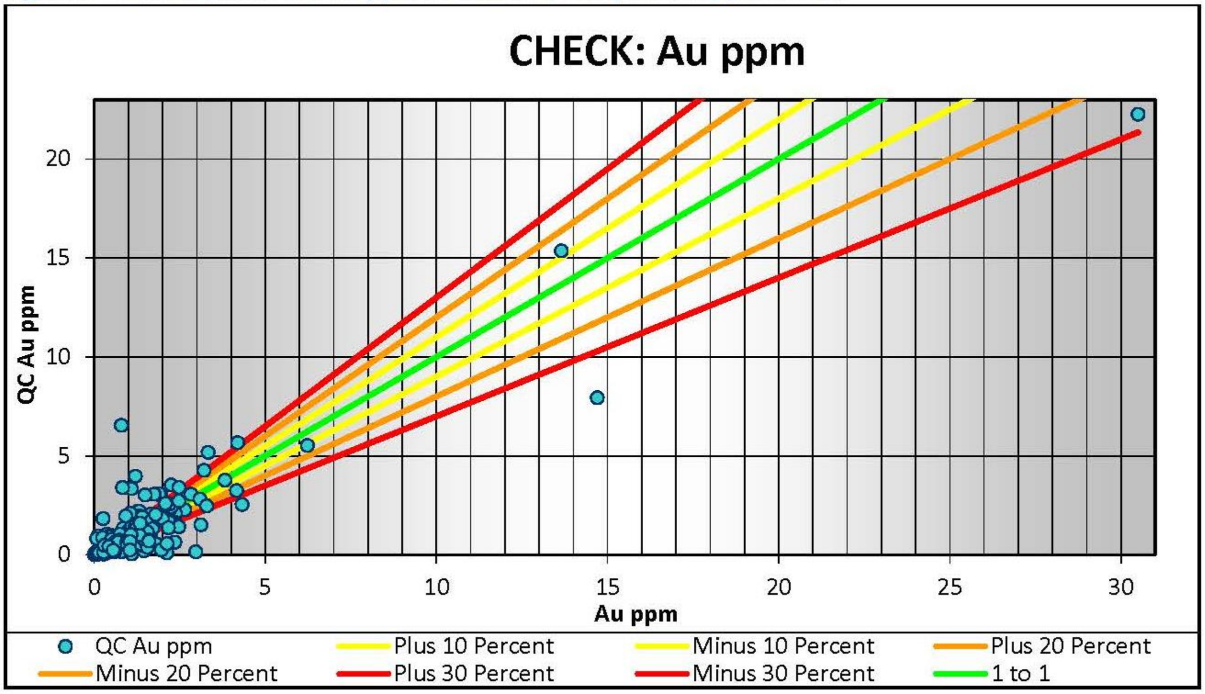
Table 11-1: 2017 Field Duplicates with Poor Correlation

Hole ID	From (feet)	To (feet)	Sample	QC Sample	QC Type	Certificate	Au Best (ppm)	QC Au Best (ppm)	ARD
17 OKC-363	243	248	17OKC-363_42r	17OKC-363_42DR	DUP	EL18027376	0.076	2.48	1.88
17 OKC-374	515	520	17OKC-374_106	17OKC-374_106D	DUP	EL18013178	0.056	1.565	1.86
17 OKC-377	668	673	17OKC-377_132	17OKC-377_132D	DUP	EL18004208	2.1	0.33	1.46
17 OKC-373	228	233	17OKC-373_42	17OKC-373_42D	DUP	EL17239910	0.0025	0.013	1.35
17 OKC-364	820	825	17OKC-364_160	17OKC-364_160D	DUP	EL17221093	1.155	0.297	1.18
17 OKC-374	278	283	17OKC-374_57	17OKC-374_57D	DUP	EL18013178	0.05	0.015	1.08
17 OKC-359	198	203	17OKC-359_38	17OKC-359_38D	DUP	EL17215119	0.126	0.038	1.07

11.3.3 Check Samples

A total of 215 check samples were analyzed in 2017 by a secondary laboratory, American Assay Laboratories, Inc., of Sparks, Nevada. The samples were analyzed by fire assay with an ICP finish, slightly different than the ALS method that utilizes an atomic absorption finish. Check sample results are shown on Figure 11-2. Results of the check sample analysis generally fall within an acceptable range, though of the total 215 samples, 29 show an average relative difference of greater than 1.0.

Figure 11-2: Scatter Plot of 2017 Check Sample Results



Otis Gold again attributes the high acid rock drainage results to local variation in sample mineral quality and/or the presence of coarse gold, and reports no apparent bias between the two laboratories.

11.4 Sample Security

Otis Gold maintains standard chain of custody procedures during all segments of sample transport. Samples are bagged and labelled in a manner sufficient to prevent tampering, and the samples remain in Otis Gold custody from the time they are collected until released to the lab. Upon receipt by the laboratory, samples are tracked by a blind sample number assigned and recorded by Otis Gold.

All whole and retained half core samples are securely stored in Otis Gold's St. Anthony core storage facility, with the exception of a few select intervals, which are stored at Otis Gold's office facility in Spokane. The core is neatly stored in labelled core boxes, which are arranged according to drill hole on sturdy shelving units. Coarse rejects were discarded, and pulp samples are stored at the Otis Gold Spokane office.

11.5 Opinion on Adequacy

GRE finds the sample preparation, analytical procedures, and security measures employed by Otis Gold to be reasonable and adequate to ensure the validity and integrity of the data derived from Otis Gold's sampling programs to date.

12.0 DATA VERIFICATION

Data verification efforts carried out by GRE included an on-site inspection of the Project site and core storage facility, visual examination of selected core intervals in comparison with drill hole logs and assay data, and mechanical and manual auditing of the Project database.

12.1 2018 Site Visit and Drill Core Inspection

GRE representative and qualified person (QP) J.J. Brown, P.G., conducted an on-site inspection of the Kilgore Project area and St. Anthony core storage facility (Photo 12-1) on August 4 and 5, 2018, accompanied by Otis Gold V.P. of Exploration, Alan Roberts. While on site, Ms. Brown conducted general geologic field reconnaissance, including inspection of surficial geologic features and ground-truthing of reported drill collar locations. Ms. Brown also reviewed plans and sections on which the conceptual geologic model is based, discussed Otis Gold's rationale for future exploration, and reviewed past and present drilling and sampling methods and protocols. The full second day of the site visit was spent at the core storage facility in St. Anthony, where select drill core intervals were visually inspected and compared with associated hand-written drill hole logs and original assay certificates.

Photo 12-1: Core Storage in St. Anthony

Observations during the site visit generally confirm previous reports on the geology and mineralization within the Project area, and specifically within the primary resource area. Bedrock lithologies, alteration types, and significant structural features appear consistent with descriptions provided in existing Project reports, and the author did not see any evidence in the field or in drill core that might significantly alter or refute the current interpretation of the local geologic setting as described in Section 7.0 of this report. The author did note some discrepancy between lithology and mineral assemblages observed in select core intervals and those described in the logs, with specific regard to sulfide mineralization, which was observed in a few the core intervals examined but was either not noted or was under-reported on the associated drill hole logs.

Geographic coordinates for 46 individual drill hole collar locations were recorded in the field using a hand-held GPS unit. In many instances, a single location was recorded for two to seven holes, all located within feet of each other (Photo 12-2), for a total of 109 collars checked for at least reasonable correlation with the coordinate locations listed in the Project database.

Photo 12-2: Closely Spaced, Variably Marked Drill Hole Collars

The difference between field collar coordinates and those contained in the Project database is quite variable, though generally within the expected, fairly wide margin of error given the rough manner of record. A number of drill hole collars are not well marked in the field, and many have no marker at all. The QP recommends that Otis Gold clearly identify existing drill holes in the field, where possible and practical, by installing semi-permanent markers such as a survey cap or labelled and grouted-in lathe. The marked collars should be professionally surveyed to confirm location and elevation, and the survey data should then be tied in to the digital topographic surface used for geologic and resource modeling. Future drill holes can be located using standard GPS instrumentation, provided that the GPS coordinates and elevations are reasonably similar to those reported for the same locations within the digital topographic surface.

12.2 Database Audit

GRE audited the database by generating graphic sections, plan views, checked assays, and downhole information. The database audit work completed to date indicates that occasional inconsistencies and/or erroneous entries are likely inherent or inevitable in the data entry process. The QP recommends that Otis Gold establish a routine, internal mechanical audit procedure to check for overlaps, gaps, total drill hole length inconsistencies, non-numeric assay values, and negative numbers. The internal mechanical audit should be carried out after any significant update to the database, and the results of each audit, including any corrective actions taken, should be documented and stored for future use in database validation.

12.3 2012 Data Verification

A vigorous data verification effort of gold assays was carried out during preparation of a previously reported mineral resource estimate for the Kilgore Project. That work is described in detail in the subsequent NI 43-101 technical report by Cameron (2012), and is summarized in the following paragraphs. The following is a summary of the work completed by Mr. Cameron in 2012.

During preparation of the 2012 mineral resource estimate, independent QP and professional geologist Don Cameron completed a variety of tasks in order to validate the technical data provided to him by Otis Gold. A random check of assay certificates stored by Otis Gold in Spokane, Washington, covered the 17 drilling programs conducted on the property between 1984 and 2011. Assay data for drill holes comprising approximately 10% of the resource database, 21 drill holes in all, was compared to the corresponding original assay certificates (2,486). This comparison revealed an error rate of 1.9%, almost all minor rounding errors apparently related to conversions from assays originally generated in units of opt to g/T, or vice-versa. Other errors included improper entry of detection limit assays.

Mr. Cameron reviewed hard copy drill hole information stored in Spokane and noted whether the physical information included the written log, core photographs, a map or reference section, original certificate, a summary or assay listing, survey data, recovery log, and rock quality designation (RQD) log. This data was not complete for all holes; e. g., some holes are reverse circulation drill holes and thus do not have photographs, recovery, and RQD. Two files contained the downhole survey records, eight included photographs, five included maps and/or sections showing the drill hole geology and assays, 11 files had recovery measurements, and 10 had RQD measurements. All drill holes had written logs, and all of the certificates were found in the files. A final check included comparing an interval or two of high assays in the database with the drill log to see if the assays corresponded to the geologic notes. In some cases, there was no correspondence due to a lack of detail in the log, or in others, because gold is not visible, and the controls are not apparent. In a few cases, higher gold appeared to have a definite association with silicified or tourmalinized intervals, faulting, and oxidation noted in the logs.

Assisted by Otis Gold geologists, Mr. Cameron conducted comprehensive checking of the electronic database provided by Otis Gold. The database comprises tables for collar information, downhole survey, assay, lithology and alteration. The collars were checked against topography to make sure they plotted properly, and the hole traces checked to look for kinks and corkscrews, which indicate survey errors. A Micromine software database validation was the principal checking tool used to detect overlapping intervals, intervals extending beyond the depth of the drill hole, anomalous downhole changes in azimuth and dip, duplicate intervals, interval information out of sequence, information with no corresponding collar, and other checks. A secondary check technique was use of filters in EXCEL® to detect spurious codes, handling inconsistencies for detection-limit assays, inconsistent hole naming, and other tests. Errors were corrected as a result of these checks in several sessions at Otis Gold's office.

Mr. Cameron visited the Kilgore Project area in July 2012, where he collected samples from locally derived float to test stockwork-veined and silicified rhyolite and rhyolite autobreccia near a caved adit along one of the drill roads to test for the presence of gold, with positive results. The site visit included an inspection of core preparation and storage facilities in Otis Gold's St. Anthony field office and an inspection of drill cores. Selected intervals from three holes were sampled for independent assay. Cores from three drill holes stored at Otis Gold's Spokane core storage facility were also examined, and one remaining half-core of an original sample from each of the three holes was submitted for check assay. The results of these check assays were variable, as is generally expected with alternate core halves. In total, the check assay data collected were insufficient to make conclusions about the adequacy, accuracy, and precision of assaying over the history of the project but were deemed acceptable to confirm the presence of gold, though with a potential persistent low bias.

The geologic and gold assay databases were considered by Cameron to have an industry-standard degree of content, organization, continuity, and documentation for a project at the exploration stage. The assay results between laboratories showed a high degree of variance, both Echo Bay data comparing Cone to Chemex, and 2008 to 2011 Otis Gold data comparing Inspectorate and Acme to ALS Chemex. The variance was addressed in the 2012 study by capping of metal-at-risk in the estimation, and Mr. Cameron concluded that taken as a whole, the database was sufficient for resource estimation.

12.4 Opinion on Adequacy

Based on the results of the site investigations related to this current study, visual examination of selected core intervals and the results of GRE's database audit, as well as the data verification work completed in 2012, GRE considers the data contained in the Project data base to be reasonably accurate and suitable for use in estimating mineral resources and reserves.

Comparison of field and database elevations indicates that additional or improved ground survey may be necessary to increase confidence in the accuracy of the drill hole location data contained within the database. GRE recommends that Otis Gold clearly identify all existing drill holes in the field. The existing drill collars should then be professionally re-surveyed and tied in to the digital topographic surface used for geologic and resource modelling.

GRE recommends re-assaying all existing pulps for silver and including silver assays on all future exploration campaigns.

12.5 2017 Site Visit

In accordance with NI 43-101 guidelines, Mr. Rowe conducted site visits for the Kilgore Project from the 9th to the 14th of August 2017. The site visits included visits to the Otis Gold Spokane, Washington, office and core shed, to the St. Anthony, Idaho, core shed/sample preparation facility, and to the Kilgore Project site. Mr. Rowe was accompanied on this visit by Senior Geologists Mitch Bernardi and John Carden of Otis Gold.

During the site visits, Mr. Rowe reviewed the following technical aspects for the property:

- How historic drill hole data was collected, stored, and compiled into the electronic drill hole database
- Property geology, gold mineralization, alteration and controlling structures
- Drill hole logging and sampling procedures
- QA/QC data verification program for project sampling
- Chain of custody procedure for sample handling and transport to ALS in Elko, Nevada
- Representative drill core from primary geologic setting for gold in all rock types
- Database management and data entry
- Drill hole collar locations.

At the Kilgore Project site, active core drilling sites were observed, and core handling was reviewed. Several drill hole collar locations were verified by GPS and compared to the Otis Gold collar database. Mr. Rowe reviewed the property geology and controls of gold mineralization on site and in representative drill core. Rowearth was provided access to all relevant data and interviewed Otis Gold geologic staff to understand the exploration history and the procedures to compile and store all project data.

The site visit included inspection of the core storage and sample preparation facilities in Spokane, Washington, and St. Anthony, Idaho. During these visits, Mr. Rowe requested and inspected 12 separate core sample intervals from eight drill holes that were stored in both Spokane and St. Anthony. Mr. Rowe collected 12 umpire core samples that consisted of ½ core samples that were previously sampled by Otis Gold, and these samples were given directly to ALS Labs for analysis. Two separate analytical samples were split from these core samples for analysis (Umpire samples A and B), and the analytical sample pulp for the Umpire B sample was also split and run as a pulp duplicate. The umpire sample gold values show agreement with the original Otis Gold gold samples. The results from this study are shown in Table 12-1 and Figure 12-1 and Figure 12-2.

Table 12-1: Umpire Samples Collected from Kilgore Core Samples

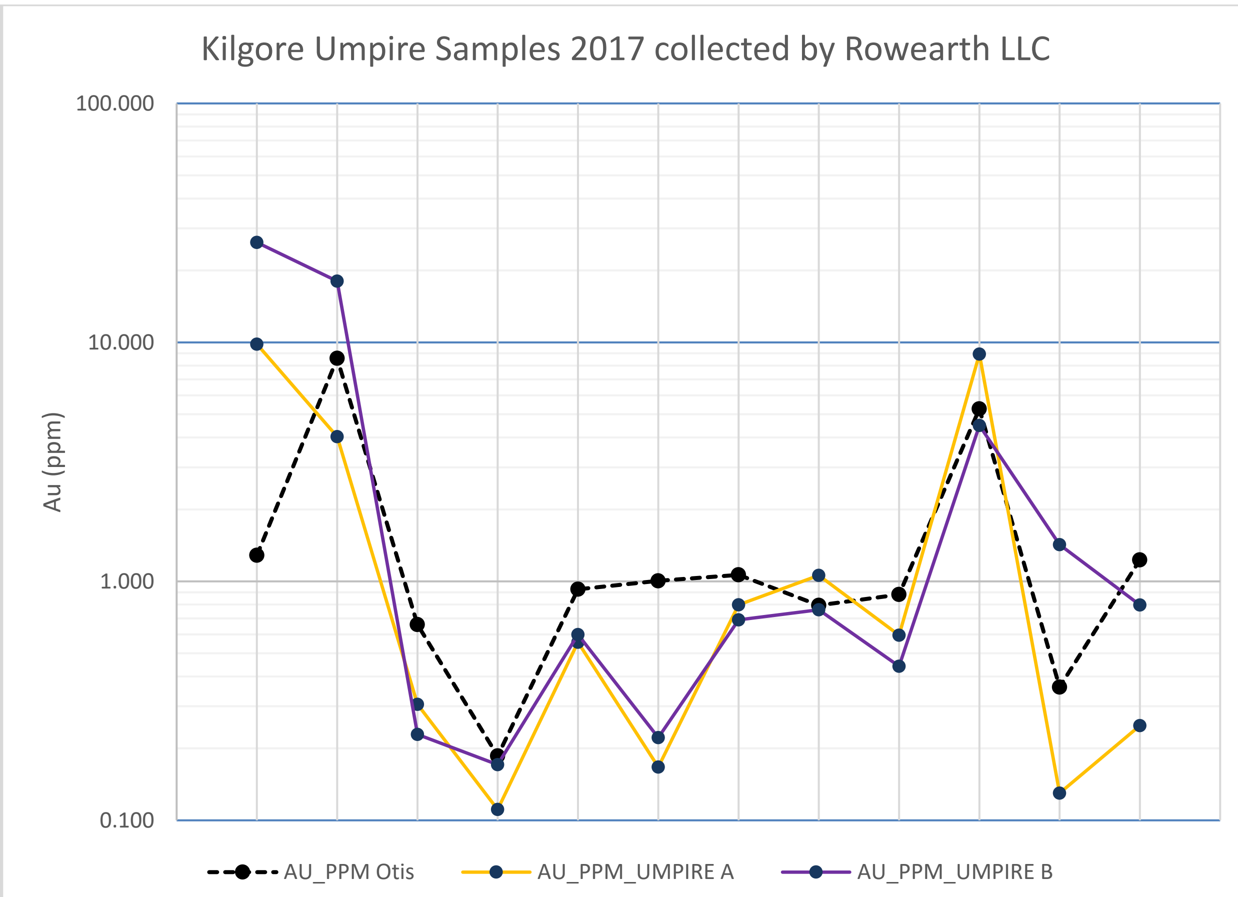
Hole_ID	From (ft)	To (ft)	Au (ppm) Otis Gold Sample	Au (ppm) Umpire A	Au (ppm) Umpire B	Au (ppm) *Pulp Dup* Umpire B
09OKC-195	462	467	1.29	9.83	26.20	36.40
11OKC-258	190	195	8.57	4.03	18.05	5.94
11OKC-265	435	440	0.66	0.31	0.23	0.74
11OKC-265	535	540	0.19	0.11	0.17	0.17
11OKC-253	248	253.5	0.93	0.56	0.60	0.60
11OKC-253	741	746	1.01	0.17	0.22	0.32
11OKC-259	305	310	1.07	0.80	0.69	0.84
10OKC-227	308	313	0.79	1.06	0.76	0.66
10OKC-227	358	363	0.88	0.60	0.44	0.41

Hole_ID	From (ft)	To (ft)	Au (ppm) Otis Gold Sample	Au (ppm) Umpire A	Au (ppm) Umpire B	Au (ppm) *Pulp Dup* Umpire B
10OKC-228	252	257	5.27	8.94	4.49	7.57
11OKC-260	585	590	0.36	0.13	1.43	0.52
11OKC-260	605	610	1.23	0.25	0.80	0.31

Samples are ½ Core Splits (all remaining material from original core)

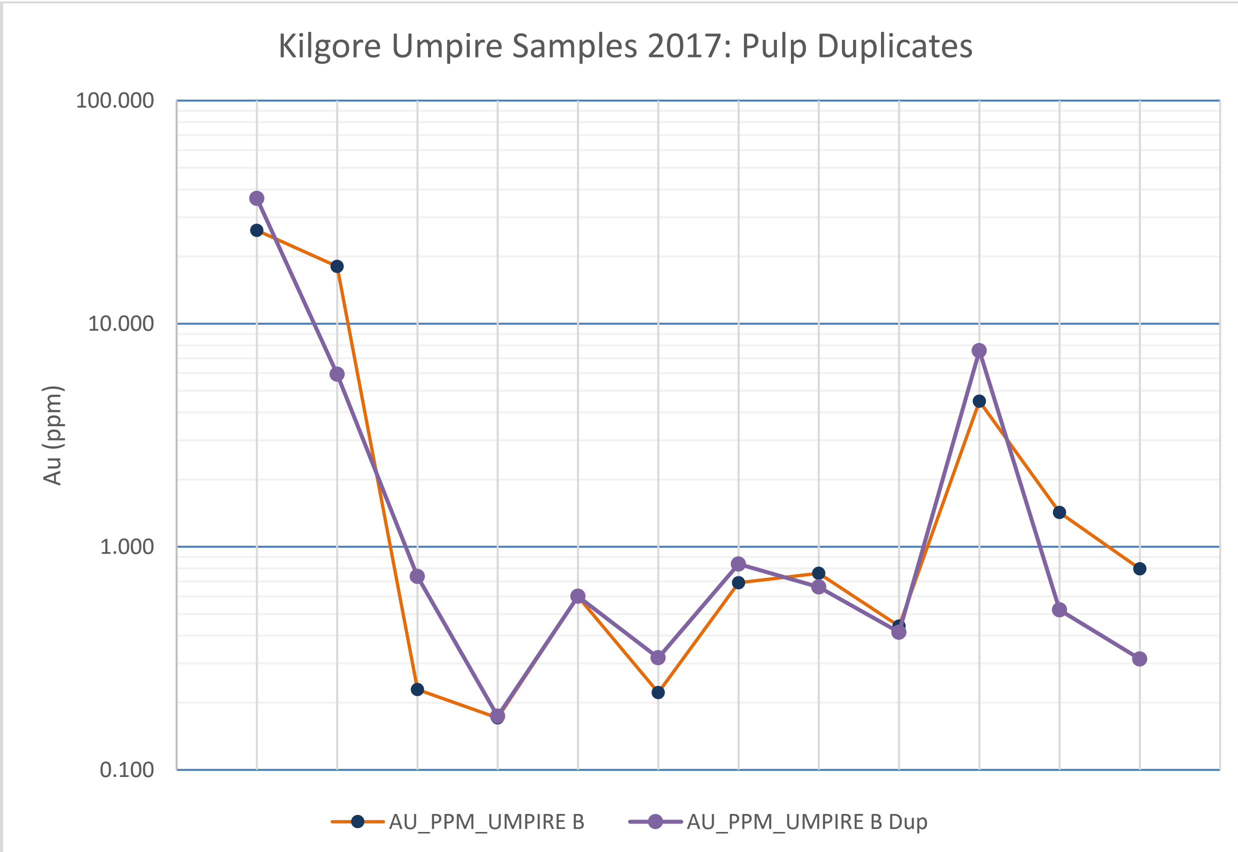
No significant issues were identified by Rowearth during the 2017 site visit, and the Otis Gold procedures for the Kilgore Project meet NI 43-101 operational standards.

Figure 12-1: Plot of Umpire Sample Gold Values



½ Core Sample Umpire sample values collected by Rowearth compared to original Otis Gold sample values.

Figure 12-2: Plot of Umpire Sample Gold Values: Duplicates from Same Pulp



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

A variety of metallurgical test work has been conducted on the Kilgore project by multiple parties, including Placer Dome, Echo Bay, and Otis Gold. Metallurgical testing of the Kilgore Project prior to 2012 is described in detail in the 2012 NI 43-101 Technical Report prepared by Cameron (2012). No details of the test work conducted by Placer Dome could be located for inclusion in this report.

13.1 Echo Bay Test Work

Prior to Otis Gold, the only company to do any significant metallurgical testing was Echo Bay. The samples employed in the testing are listed in Table 13-1.

Table 13-1: Echo Bay Samples

Drill Hole	Oxidation State	Grade - Au	
		opt	g/T
94EKR-78, 79, 80	Oxide	0.083	2.85
94EKR-81	Oxide Dominant	0.032	1.10
94EKR-84	Sulfide Dominant	0.028	0.96
94EKR-85	Sulfide	0.037	1.27
94EKC-128	Mixed Oxide-Sulfide	0.023	0.79
95EKM-6	Mixed	0.023	0.79

The intervals for the drill core were not provided in the metallurgical test report taken from 1995 Summary Report for Placer Dome U. S. Inc. (Echo Bay Exploration, Inc., 1995) or from the 1996 Summary Report: Kilgore Project (Echo Bay Exploration Inc., 1996). Placer Dome and Echo Bay were partners in the project during this period. The test work included a series of bottle roll and column leach tests as well as preliminary mineralogical analysis.

Rayner and Associates and Van Brunt (2002) discussed the Echo Bay programs which included:

- Mineralogical characterization and bottle roll tests performed on RC cuttings grouped by oxidation type (1995)
- Column leach tests, on drill core averaging between 0.03 and 0.05 opt Au (1.0 – 1.7 g/T Au) and categorized by geologists as oxide, mixed, and sulfide, crushed to P_{80} of ½-inch (12.5 mm)
- Column leach tests in 1996 on oxidized and unoxidized drill core with a grade of 0.023 opt Au (0.8 g/T Au) from a single hole at P_{80} 1-inch (25 mm)
- Coarse bottle roll testing on 15 composite samples from a core hole to test the amenability of rock to cyanide leaching with depth.

Echo Bay bottle roll leaching studies of RC drill cuttings from drill holes (94EKR-78, 79, 80, 94EKR-81, 94EKR-84 and 94EKR-85) show that gold in the Kilgore deposit is readily extracted by cyanide leaching. The extractions ranged from 83% to 95% (Table 13-2). A mineral examination by Hazen Labs of the "client-identified" sulfide ore found little or no sulfide mineralization in the sample (Boles, 1995).

Table 13-2: Echo Bay 96-Hour Bottle Roll Testing

Test	Sample Description	Consumption		% Au Extraction
		NaCN (lbs./short ton)	Lime (lbs./short ton)	
1	Oxide	2.7	2.1	88.8
2	Oxide	2.0	2.0	87.2
3	Oxide Dominant	0.1	2.2	92.3
4	Oxide Dominant	0.3	2.2	92.2
5	Sulfide Dominant	0.6	2.8	94.8
6	Sulfide Dominant	0.6	2.6	82.9
7	Sulfide	0.6	1.6	93.3
8	Sulfide	0.7	1.6	94.0

Source: Boles (1995)

Echo Bay submitted three ore types collected to characterize the orebody. The ore types were identified as "Oxide," "Mixed," and "Sulfide." Samples of drill core were crushed to 80% passing -½ inch (12.5 mm) to be tested in columns. The columns were six inches in diameter and 6 feet high. Approximately 100 pounds of ore were used in each column. Lime was blended with the ore prior to loading into the columns. One gram/liter of cyanide (CN) solution was applied to the ore in each column at a rate of 0.004 gallons per minute/square foot. The column effluent samples were collected and measured for volume, cyanide concentration, and precious metal concentration. Initially, the effluent samples were collected daily. After 14 days of leaching, effluent samples were collected 3 days per week.

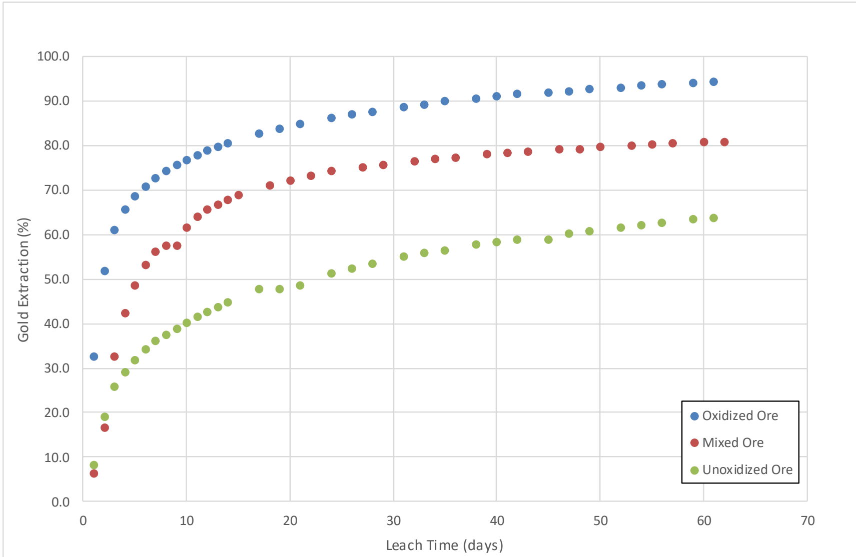
Column leaching was ceased after 60 days. The ore in the column was then rinsed with water to remove residual CN, and the columns were emptied. Leach residues were blended, and samples were split for gold assay. Results of the column data are shown in Table 13-3 below and is graphically illustrated in Figure 13-1.

Table 13-3: Echo Bay Column Leach Results for Three Samples of Differing Oxide Types

Test	Sample Description	Consumption	Assay oz Au/ton		% Au Extraction
		lbs NaCN/short ton	Head	Residue	
1	Oxide Ore	1.94	0.04	0.004	94.3
2	Mixed Ore	1.99	0.05	0.013	80.8
3	Sulfide Ore	2.08	0.03	0.102	63.8

Source: Boles (1995)

Figure 13-1: Echo Bay Column Leach on Various Rock Types at a P_{80} of ½-inch (12.5-mm)

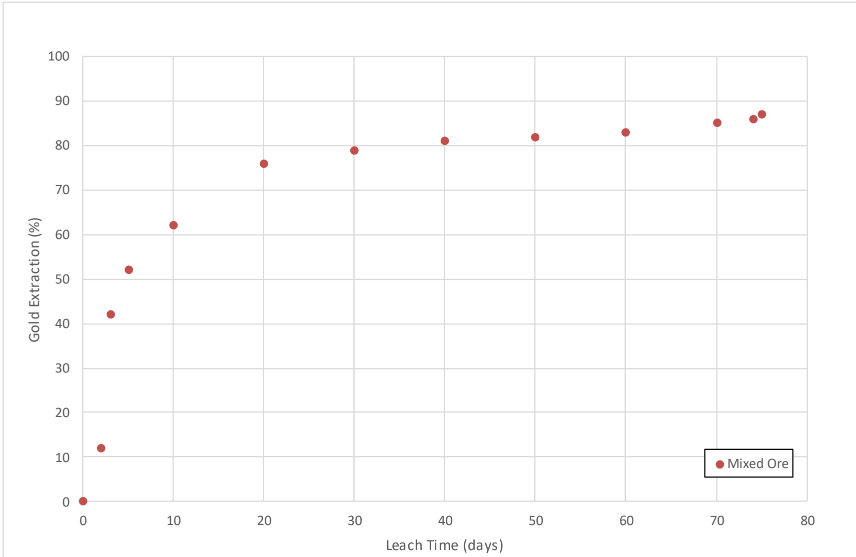


Source: Boyles, 1995

The results from this testing demonstrate a potential relationship between the oxidation type and recovery rates, although it should be emphasized that the "sulfide ore" is a misnomer, as there was little to no sulfide in the sample. The sample description should best be described as "unoxidized." Also, the results from the sulfide or un-oxidized sample show lower gold extraction than from either the mixed oxide/sulfide or the oxide ore. Gold recoveries obtained in the column after 60 days were 94.0%, 81.0%, and 64.0% for materials identified as oxide, mixed, and unoxidized, respectively. There was a large variation in the head grade that could also account for the differences in leach extraction (Oxide - 0.04 opt, Mixed – 0.055 opt and Sulfide 0.0.027 opt). The residue grades were similar for the Mixed and Sulfide ores (0.013 and 0.012 opt) and lower for the Oxide ore at 0.004 opt).

In 1996, Echo Bay submitted 80 kilograms (kg) of split core from hole 95EKC-128 that was mixed oxidized and un-oxidized material. The sample was crushed to 80% minus one inch (25 mm). Six duplicate samples assays for gold averaged 0.7 g/T. A column leach of six-inch diameter by 10-foot column was conducted. After 75 days, gold extraction was 87% (Figure 13-2). This result showed that for this ore type, the ore was not extremely crush-size sensitive. Unfortunately, no data exists as to the reagent consumptions for this work. (Echo Bay Exploration Inc., 1996).

Figure 13-2: Echo Bay Column Leach – Mixed Rock Type at a P_{80} of 1 Inch (25 mm)



Source: Mosher, 1996

In 1996, a series of 15 bottle roll tests was conducted on samples from 95EKM-6 to examine gold recovery as a function of depth. The gold recoveries from these tests ranged from 46 to 97%, with an average of 85% at a gold feed grade of 0.035 opt. It has been reported that the results did not show a relationship between extraction and depth, but there was a potential correlation between gold grade and recovery. The upper sections of the sample had the lowest grade (approximately 0.012 opt), and this result could be indicative of a constant tail effect (Echo Bay Exploration Inc., 1996).

13.2 Preliminary Otis Gold Test Work

In 2010, Otis Gold investigated the heap leach characteristics of each host rock separately to provide information for mine design and to confirm the heap leach scenario (Table 13-4).

Table 13-4: Otis Gold 2010 Test Work Samples

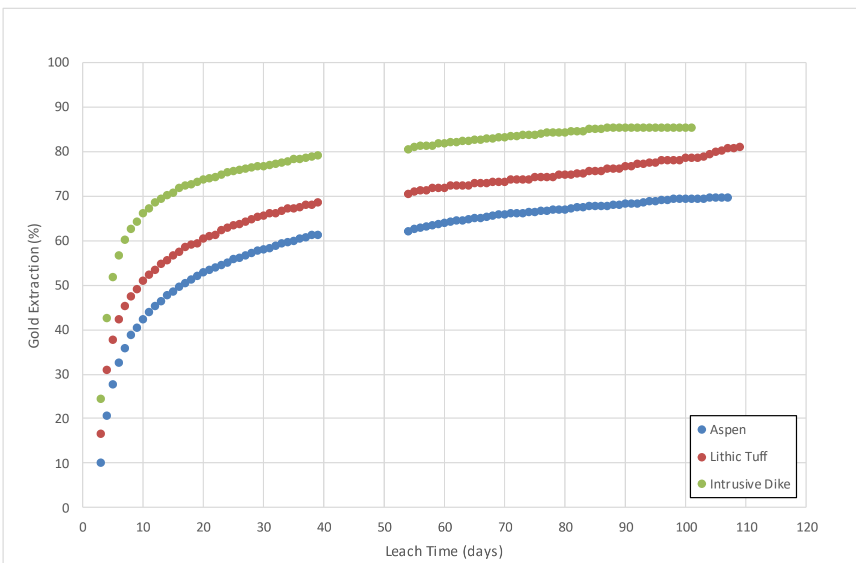
Type Composite	Drill Hole	Sample Number	Oxidation State	Footage		Interval	Grade - Au	
				From	To	(feet)	opt	g/T
AS-1 Aspen Sandstone	OKC-195	71	mixed	587	592	5.0	0.034	1.16
	OKC-195	128-130	non-oxidized	622	637	15.0	0.022	0.77
	OKC-197	80-85	mixed	561.3	587	25.7	0.052	1.78
	OKC-197	87-90	mixed	590.4	610.2	19.8	0.037	1.26
	OKC-205	151-155	non-oxidized	736	761	25.0	0.085	2.92
	OKC-205	162-165	non-oxidized	792	811	19.0	0.035	1.19
	OKC-205	176-177	non-oxidized	861	871	10.0	0.115	3.94
	Total					119.5	0.055	1.87
Tlt-3 Lithic Tuff	OKC-205	50-52	weak oxidation	241	256	15.0	0.050	1.70
	OKC-205	57-62	v. weak oxidation	276	296	20.0	0.034	1.16
	OKC-206	49-50	oxidized	247	257	10.0	0.037	1.26
	OKC-206	65-77	mixed	328	392	64.0	0.031	1.08
	Total					109.0	0.035	1.20

Type Composite	Drill Hole	Sample Number	Oxidation State	Footage		Interval	Grade - Au	
				From	To	(feet)	opt	g/T
TD-4 Felsic Dike	OKC-197	56-61	oxidized	457	487	30.0	0.028	0.97
	OKC-197	73-77	non-oxidized	536	555.8	19.8	0.055	1.90
	OKC-205	114-115	ox. on fractures	546	561	15.0	0.028	0.97
	OKC-205	131-136	v. minor ox.	636	656	20.0	0.031	1.07
	OKC-205	148-150	mixed	721	736	15.0	0.039	1.35
	Total					99.8	0.036	1.23

This material composed the feed for separate column leach tests of the three main host rock types identified at the time: Ka (Aspen sandstone), Tlt (lithic tuff), and Tpr (felsic dike), collected from four holes in the deposit area. The column tests were performed at a P_{80} of ½-inch (12.5-mm) feed size.

Tlt and Tpr lithologies host most of the gold mineralization in the Kilgore deposit. For the composites of these rocks, the tests show that approximately 77% of the gold extracted occurs in the first 30 days of leaching. The leach curves for Tpr and Ka flattened after about 90 days, whereas the leach curve for Tlt was still positive and climbing after 109 days, suggesting slightly more than 81% can be expected with a longer leach time (Figure 13-3). The Aspen ore also leached at a good rate, achieving almost 70% gold extraction in 109 days. Recovery results generally agree with the earlier Echo Bay tests.

Figure 13-3: Otis Gold Column LEACH - Various Rock Types at a P_{80} of ½-inch (12.5 mm)



In 2011, Otis Gold commissioned additional testing at McClelland Laboratories Inc. (McClelland Laboratories, Inc., 2011). Four drill core composites from the Kilgore Project were received for heap leach cyanidation testing. The composites were designated MAS-1, MTLT-2, MTLT-3 and MTD-4, and contained between 0.016 and 0.049 opt Au and between 0.11 and 0.22 opt Ag (Table 13-5).

Table 13-5: Otis Gold 2011 Test Work Samples

Sample	Drill Hole	Oxidation State	Intervals	Grade - Au		Grade - Ag	
				opt	g/T	opt	g/T
MAS-1	195 195 197 197 205 205 205	Mixed	587 – 592 622 – 637 561.3 – 587 590.4 – 610.2 736 – 761 792 – 811 861 – 871	0.042	1.44	0.099	3.39
MTLT-2	195 202	Mixed	97.7 – 164.6 327 – 387	0.016	0.55	0.17	5.83
MTLT-3	205 205 206 206	Mixed	241 – 256 276 – 296 247 – 257 328 – 392	0.02	0.69	0.203	6.96
MTD-4	197 197 205 205 205	Mixed	457 – 487 536 – 555.8 546 – 561 636 – 656 721 – 736	0.049	1.68	0.104	3.57

A direct cyanidation (bottle roll) test was conducted on each composite, at a P_{80} of 10-mesh feed size, to obtain preliminary information concerning heap leach amenability. Results showed that all four composites were amenable to direct agitated cyanidation at the 10-mesh feed size. Gold recoveries obtained in 96 hours of leaching ranged from 50.0% to 78.9%. Silver recoveries ranged from 37.5% to 59.1%. Reagent consumptions were low. See Table 13-6.

Table 13-6: 2011 Otis Gold Bottle Roll Results

	MAS-1		MTLT-2		MTLT-3		MTD-4	
Extraction: pct of total	Au	Ag	Au	Ag	Au	Ag	Au	Ag
in 2 hours	10.5	14.2	26.3	29.5	29.2	27.6	10.7	26.6
in 6 hours	20.7	19.5	42.6	39.3	43.3	40.1	22.1	35.2
in 24 hours	44.8	31.1	60	48.9	63.1	53	45.4	47.5
in 48 hours	49.5	37.5	63.7	50.7	72	56.7	63	51.4
in 72 hours	50	37.5	64.5	52.2	72.2	59.1	75.1	54.7
in 96 hours	50	37.5	66.7	53.3	72.2	59.1	78.9	57.1
Extracted, opt ore	0.023	0.03	0.01	0.08	0.013	0.13	0.071	0.08
Tail Assay, opt ore*	0.023	0.05	0.005	0.07	0.005	0.09	0.019	0.06
Calc'd. Head, opt ore	0.046	0.08	0.015	0.15	0.018	0.22	0.09	0.14
Average Head, opt ore*	0.042	0.12	0.03	0.18	0.021	0.2	0.082	0.11
NaCN Consumed, lb/ton ore	0.35		0.52		0.68		0.46	
Lime Added, lb/ton ore	3.1		2.6		4.4		2.2	
Final Solution pH	10.9		11.0		10.8		10.8	
Natural pH (40% solids)	7.4		6.2		5.9		6.1	

*Average of triplicate assays

Overall metallurgical results show that the Kilgore drill core composites were amenable to direct agitated cyanidation treatment at an 80% passing 10 mesh feed size. Gold recovery obtained from the MAS-1, MTLT-2, MTLT-3, and MTD-4 composites were 50.0%, 66.7%, 72.2% and 78.9%, respectively, in 96 hours of leaching.

A column leach test was conducted on each composite at a P_{80} of ½-inch feed size to determine gold and silver recovery, recovery rate, and reagent requirements under simulated heap leaching conditions. Results showed that all four composites were amenable to simulated heap leach cyanidation treatment at the ½-inch feed size. Column test gold recoveries obtained from the MAS-1, MTLT-2, MTLT-3, and MTD-4 composites were 69.8%, 57.1%, 81.0% and 85.3%, respectively, in 80 to 109 days of leaching and rinsing, as shown in Figure 13-4 through Figure 13-7 and in Table 13-7.

The ore charges did not require agglomeration pretreatment. Lime (2.0 - 4.5 lb/ton ore) was mixed with the dry ore charges before column loading procedures. Ore charges were placed into the 6-inch diameter x 10-foot high PVC leaching columns in a manner to minimize particle segregation and compaction.

Figure 13-4: Otis Gold Column Leach Tests - Composite MAS-1, P_{80} at ½-Inch

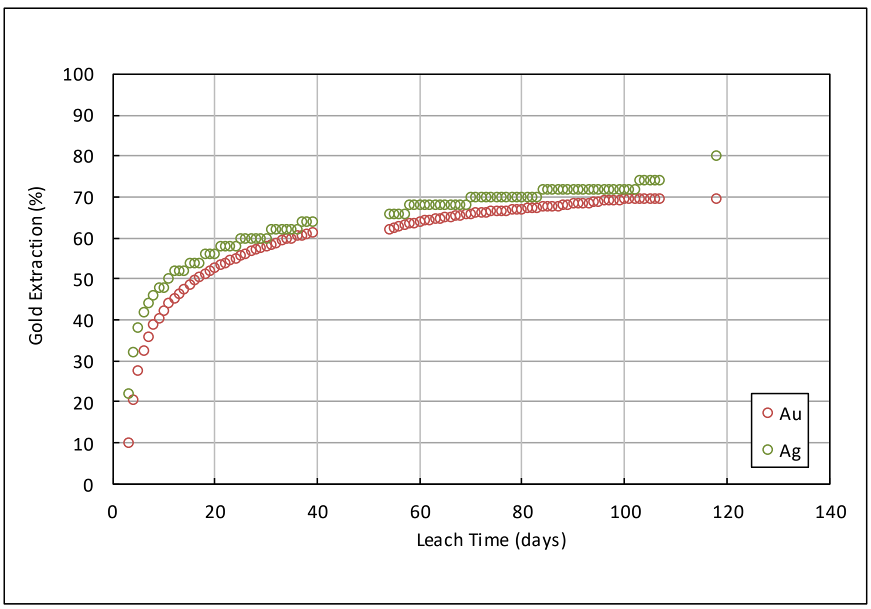


Figure 13-5: Otis Gold Column Leach Tests - Composite MTLT-2, P_{80} at ½-Inch

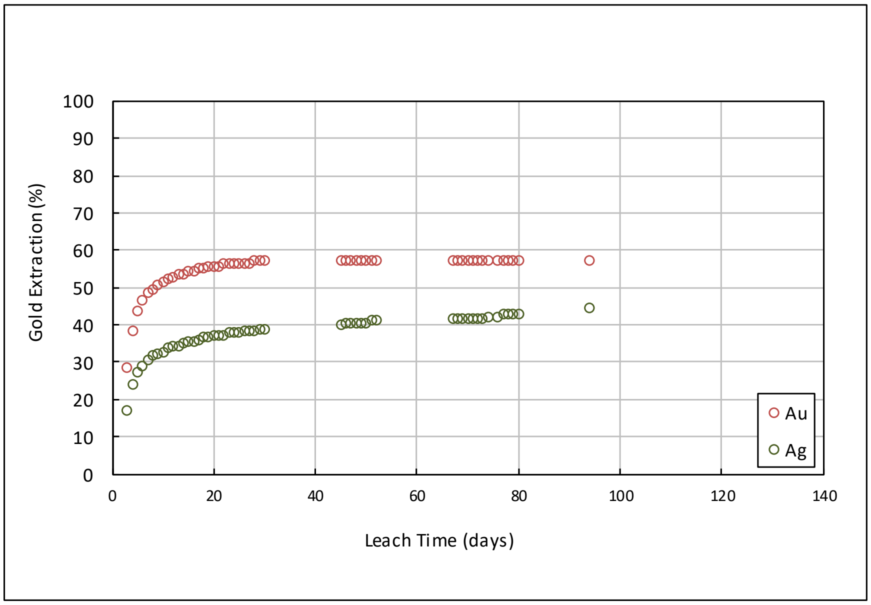


Figure 13-6: Otis Gold Column Leach Tests - Composite MTLT-3, P_{80} at ½-Inch

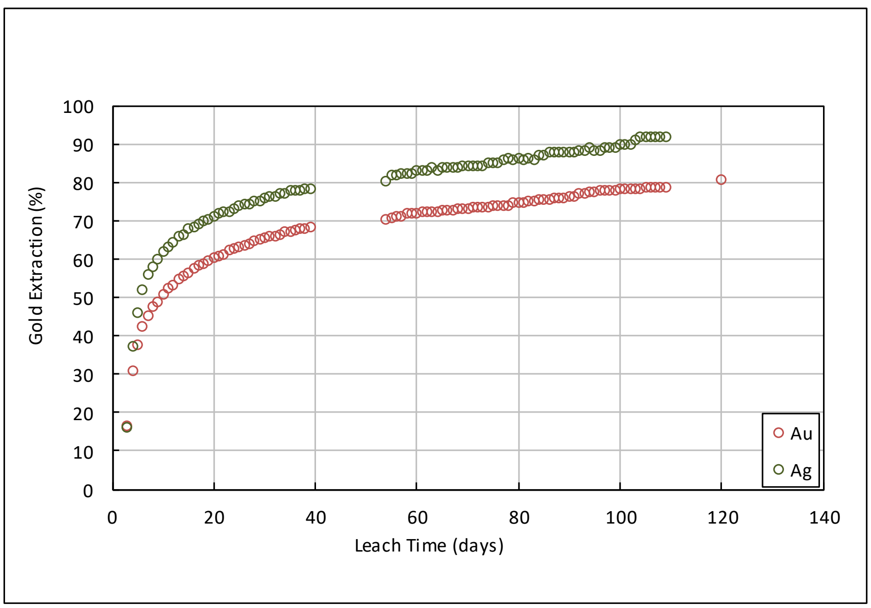


Figure 13-7: Otis Gold Column Leach Tests - Composite MTD-4, P_{80} at ½-Inch

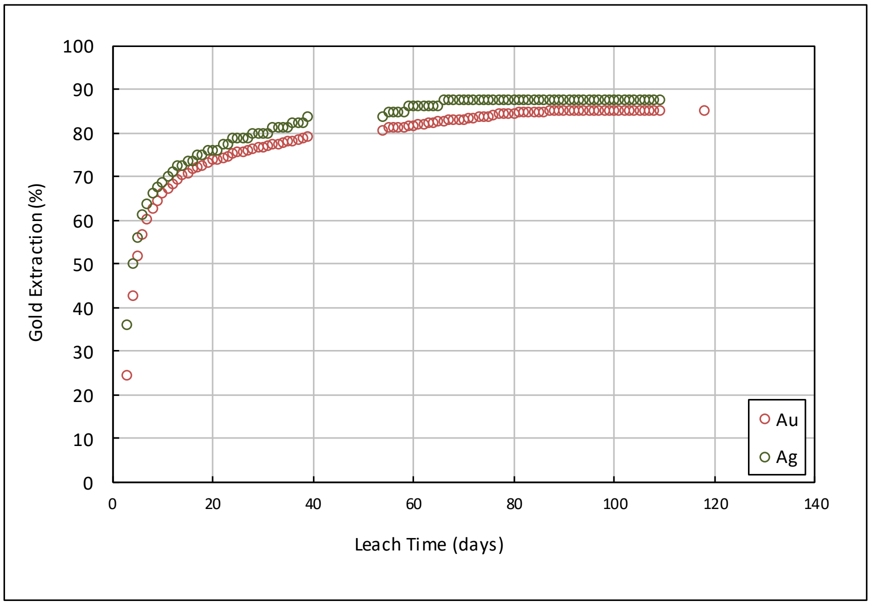


Table 13-7: 2011 Otis Gold Column Leach Results Summary

Sample I.D.	Leach/Rinse Time (days)	Solution Applied (ton/ton ore)		Au Rec. (%)	Au opt		Ag Rec. (%)	oz Ag/Ton ore		NaCN Consumed	Lime Added
		Leaching	Rinsing		Tail Assay	Calc'd. Head		Tail Assay	Calc'd. Head	lb/t	Lb/t
MAS-1	107	6.4	0.4	69.8	0.013	0.043	80.0	0.01	0.05	3.15	3.0
MTLT-2	80	3.3	0.3	57.1	0.006	0.014	44.4	0.10	0.18	2.00	2.5
MTLT-3	109	6.5	0.6	81.0	0.004	0.021	93.3	0.01	0.15	3.36	4.5
MTD-4	109	6.9	0.6	85.3	0.010	0.034	87.5	0.01	0.08	3.91	2.0

Leaching was conducted by applying cyanide solution (2.0 lb sodium cyanide [NaCN]/ton of solution) over the ore charges at a rate of 0.005 gallons per minutes (gpm)/square foot of column cross-sectional area. Pregnant effluent solutions were collected each 24-hour period. Pregnant solution volumes were measured by weighing, and samples were taken for gold and silver analysis using conventional A.A. methods. Cyanide concentration and pH were determined for each pregnant solution. Pregnant solutions were pumped through a three-stage carbon circuit for adsorption of dissolved gold and silver values. Barren solution, with appropriate make-up reagent, was applied to the ore charges daily. After leaching, fresh water rinsing was conducted to remove residual cyanide and to recover dissolved gold and silver values. Moisture required to saturate the ore charges (in process solution inventory) and retained moistures were determined. Apparent ore bulk densities were measured before and after leaching.

As a result of these positive column leach results, Otis Gold decided to perform leach tests on material from new drill holes and at a coarser size fraction. Otis Gold mobilized a drill rig to obtain PQ core (83-mm diameter) for metallurgy. Hole 11 OKC-285 was a twin of hole OKC-258, and hole 11 OKC-287 was a PQ twin of hole 10 OKC-228. Both holes were logged but not assayed or split. Twins of three mineralized intervals, weighing about 1,000 lbs each, were sent to McClelland Labs in Reno, Nevada, for column leach tests. Samples to be tested were segregated by rock type and oxidation state. Otis Gold segregated the samples by rock type to comprise three new composites: MTF-1 - oxidized bulk sample of Tlt, MDO-2 -

oxidized bulk sample of Tpr, MDS-3 - unoxidized bulk sample of Tpr. Details of the samples can be found in Table 13-8. (McLelland Laboratories, Inc., 2012).

Table 13-8: 2012 Otis Gold Test Work Samples

Sample	Drill Hole	Oxidation State	Interval (feet)		Grade - Au		Grade - Ag	
			From	To	opt	g/T	opt	g/T
11-MTF-1	11-OKC-285	Lithic Tuff (oxidized)	20	40				
	11-OKC-285		60	83.5				
	11-OKC-287		39	74				
	11-OKC-287		117	127				
	11-OKC-287		201	237				
				124.5	0.013	0.46	0.058	2.0
11-MDO-2	11-OKC-285	Felsic Dike (oxidized)	112	176				
	11-OKC-287		237	298				
				125.0	0.011	0.38	0.029	1.0
11-MDS-3	11-OKC-285	Felsic Dike (unoxidized)	180	288				
				108.0	0.034	1.15	0.029	1.0

Column leach test were conducted at a P_{80} of ½-inch and 1 ½-inch crush size on all three samples. The MTF-1 sample achieved similar recoveries of 84.9% and 85.5% after 91 days for the 1 ½-inch and ½-inch crush sizes, respectively. The MDO-2 sample exhibited a lower recovery for the coarse size fraction of 71.2% compared to 83.3% for the ½-inch crush size, both after 78 days of leaching. It appears that these recoveries would have equalized with the extension of the leaching time. The MDS-3 sample achieved similar recoveries of 78.5% and 74.5% after 78 days for the 1 ½-inch and ½-inch crush sizes, respectively. The results are summarized in Table 13-9.

Table 13-9: 2012 Otis Gold Column Leach Results

Sample Number	Crush Size (P_{80} in)	Leach Time (days)	Calculated Head Grade (g/T Au)	Gold Recovery (%)	Reagents	
					CN (lb/t)	Lime (lb/t)
11-MTF-1	1 ½-inch	91	0.53	84.9	0.60	1.88
	½-inch	91	0.69	85.5	0.87	1.80
11-MDO-2	1 ½-inch	78	0.52	71.2	0.69	1.00
	½-inch	78	0.42	83.3	0.76	1.00
11-MDS-3	1 ½-inch	78	1.21	78.5	1.00	1.10
	½-inch	78	1.45	74.5	0.82	1.80

The column leach curves are shown in Figure 13-8 through Figure 13-10.

Figure 13-8: Otis Gold Column Leach Tests - Composites MDS-3, MTF-1, MDO-2 at a P_{80} of ½-inch

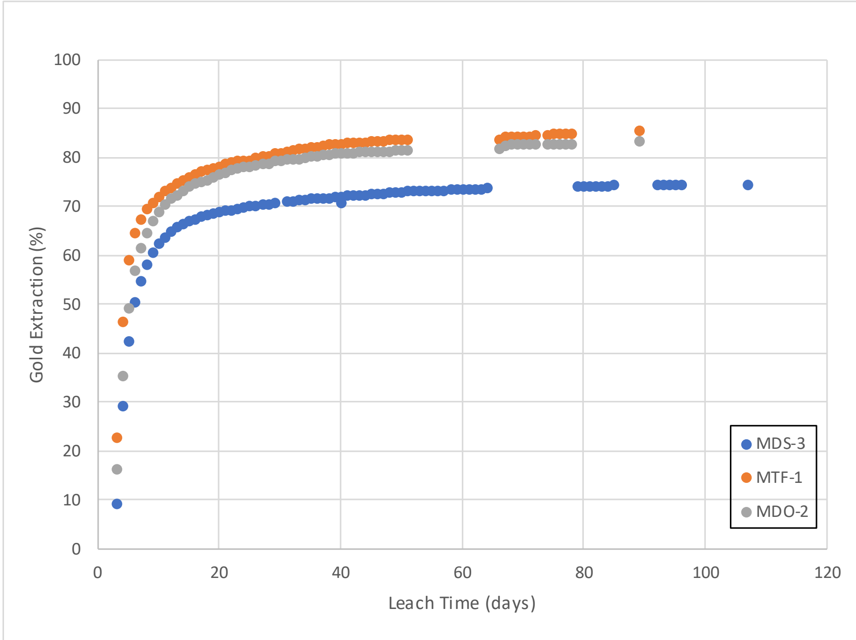


Figure 13-9: Otis Gold Column Leach Tests - Composites MDS-3, MTF-1, MDO-2 at a P_{80} of 1 ½-inch

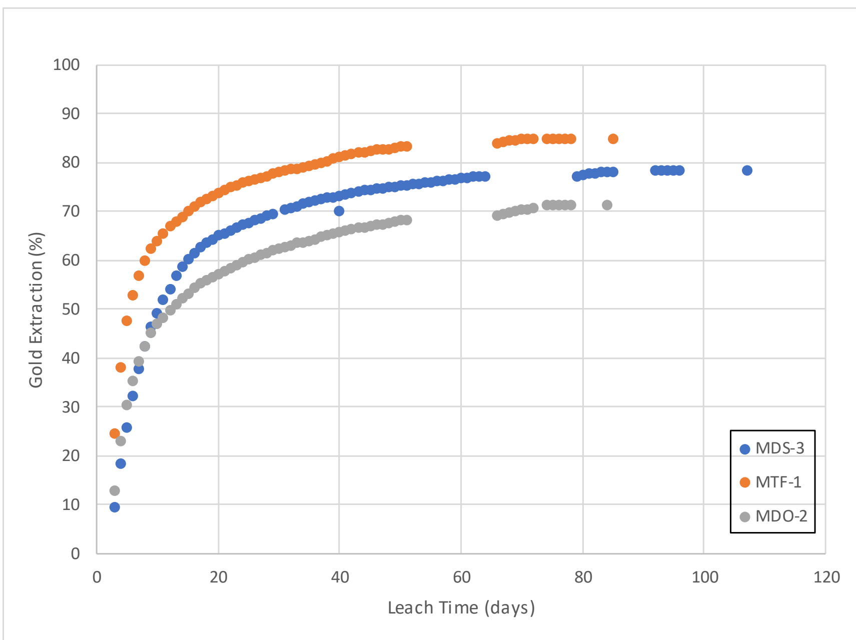


Figure 13-10: Otis Gold Column Leach Tests - Composites MDS-3, MTF-1, MDO-2 at a P_{80} of 1 ½-inch and ½-inch

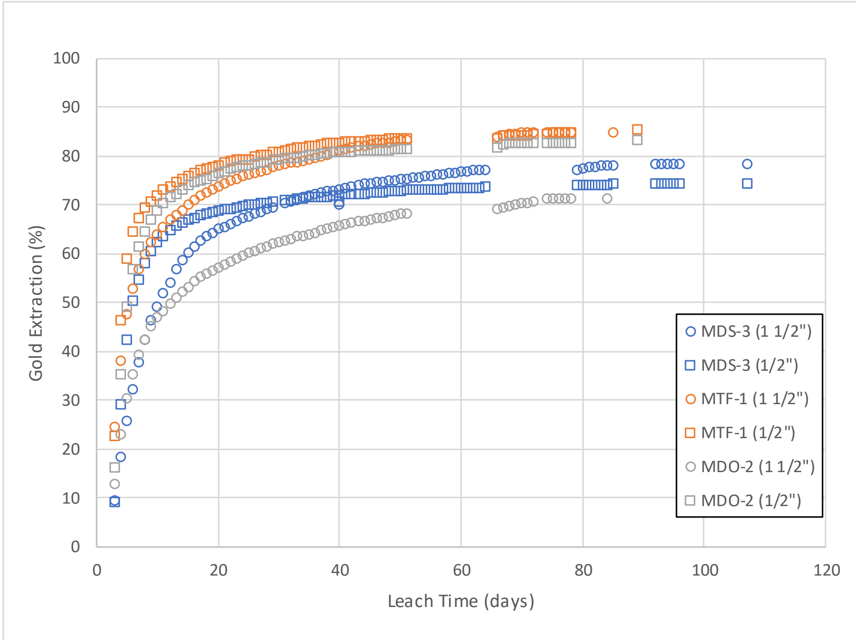


Figure 13-10 shows the impact of crush size on the column leach extraction. It is evident from this series of tests that finer crushing can improve gold extraction on some samples, such as 11MDO-2, while the other samples were relatively insensitive to crush size. Depending on the make-up of the deposit, a coarser crush size may provide adequate gold extractions, but further definition is required. The amenability of the Kilgore deposit ore to ROM leaching needs to be investigated by further metallurgical testing.

The amenability of the gold mineralization to cyanide extraction is important information to determine how responsive the ore is to processing. The main difference between fire assay and CN-soluble gold is that fire assay determines the total Au content of the rock, whereas CN-soluble gold only determines that part that is easily extractable by quick cyanidation. The ratio of the two methods (Au by CN leach/Au by fire assay) is usually less than 1:1 but can show if a particular lithology or portion of the deposit may be less amenable to extraction by cyanide. Reasons for particularly poor recovery are many but could include silica encapsulation, pregnant solution-robbing carbon, or mineralogy.

Column leach tests, as described above, usually take several months to perform, are expensive, and test only one or two holes within the deposit. To more fully understand if larger parts of the deposit are amenable to leaching or how a specific lithology responds to cyanide, it is a common practice to look at gold assay by cyanide leach, Chemex's AA13 method. Their assay technique is to prepare a sample that is weighed into a closed 100-milliliter (ml) vessel to which a sodium cyanide solution (0.25% NaCN/0.05% NaOH) is added. The sample is immediately shaken until homogenized and rolled for an additional hour. An aliquot of the final leach solution is centrifuged and analyzed by atomic absorption spectrometry. Otis Gold undertook a study where it looked at 271 samples of core and compared the original fire assay values to assays derived by cyanide soluble gold testing. Table 13-10 summarizes the results of the study, and Figure 13-11 is a histogram of the total population of how the mineralized rock responded to cyanidation.

Again, this test is not as rigorous as a column leach but provides a quick and inexpensive method to see if a particular lithology or area of the deposit is problematic.

Table 13-10: Average Gold Extraction by CN Leach Techniques for Three Major Rock Types Hosting Mineralization at Kilgore

Rock Type	Number of Samples	Number of Holes	Average Au Extraction (%)
Lithic Lapilli Tuff	162	12	59%
Rhyolite Dike	80	7	61%
Aspen Sandstone	29	3	47%
Total	**271**		

Note that the lithic lapilli tuff and dike are nearly identical in gold extraction by cyanide leaching, whereas the Aspen Sandstone shows lower amenability to CN. This is also reflected by the -1/2-inch inch column leach study performed by Otis Gold in 2010 and illustrated in Figure 13-11. The histogram (Figure 13-11) shows a slightly skewed bell-shaped curve for the range of CN soluble gold from the Mine Ridge deposit. Note that the average extraction by this method for all samples is about 58%. To analyze this information by rock type, Figure 13-12 is a scatter plot of the g/T gold from each sample determined by the two methods, fire assay vs. CN extraction. Upon review, several items are noted: 1) nearly all CN-soluble gold assays fall below the 1:1 line; 2) the data for the three rock fields overlap at levels below about 0.8 g/T Au; and 3) at grades greater than 0.8 g/T Au, the tuff seems to be more amenable to cyanide extraction than the dike rocks, with the Aspen Sandstone showing a much lower recovery at higher grades.

Figure 13-11: Histogram of % CN Soluble Gold Derived with Respect to the Gold Reported in the Original Fire Assay in Kilgore Deposit

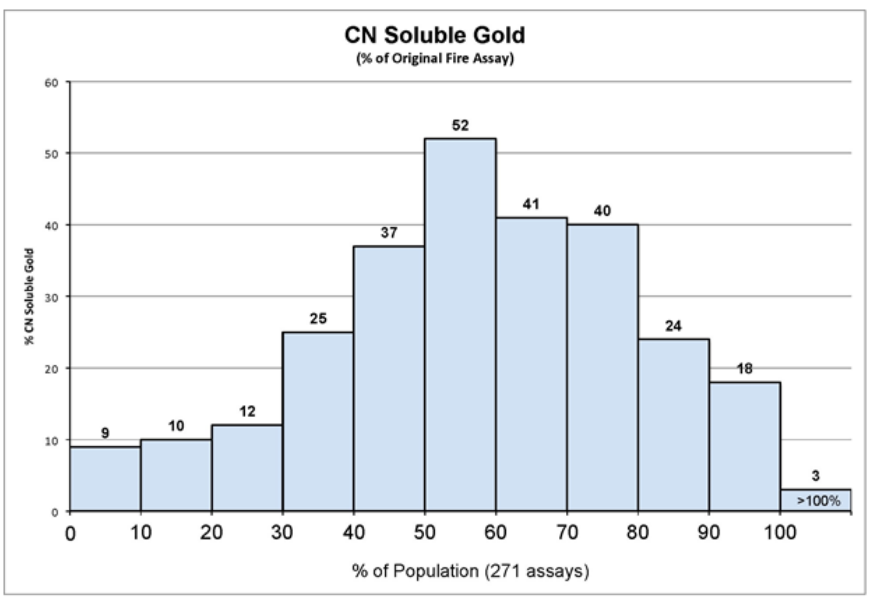
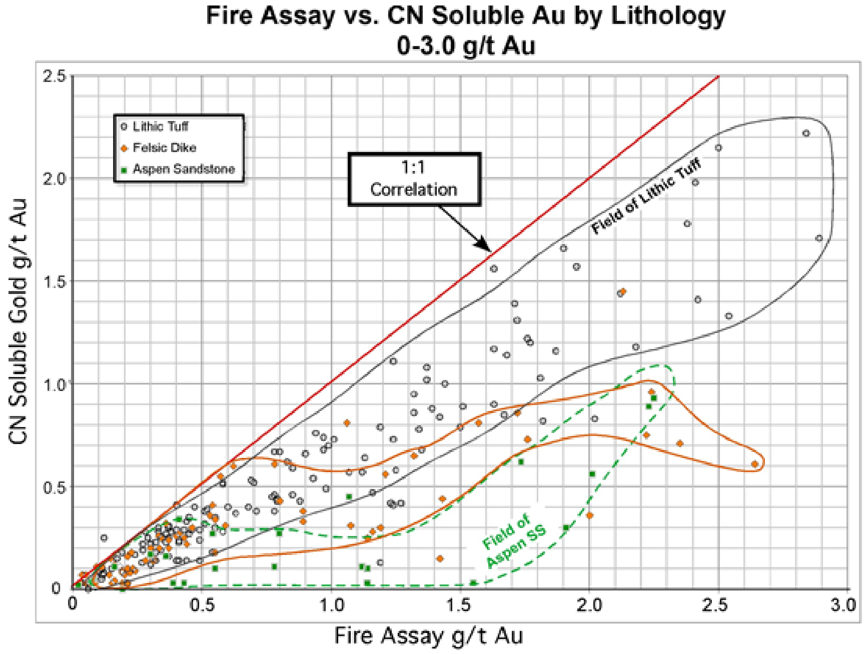


Figure 13-12: Scatter Plot Showing Fire Assay Values Compared to Corresponding CN Soluble Gold Assays from the Same Rock



Physical ore characteristic data show that very little to no “slumping” of ore charges occurred during leaching. Ore apparent bulk densities were essentially the same before and after leaching. No solution percolation, fines migration, or solution channeling problems were encountered during leaching. Table 13-11 shows the material characteristics. Load/permeability type geotechnical testing was performed on the column leached residues of the ½-inch feed size.

The results showed hydraulic conductivities for 11-MDS-3, MTF-1 and MDO-2 composites of 9.79 x 10^2, 5.72 X 10^2 and 5.47 x 10^2 cm/sec, respectively, under simulated heap stack height compressive loadings equivalent to approximately 40 meters. These rates are significantly higher than the equivalent solution application rates typically employed during commercial heap leaching.

Table 13-11: Column Characteristics – Composites 11-MDS-3, 11-MTF-1, 11-MDO-2

Sample	Feed Size P_{80}	Charge kg	Moisture wt%			Apparent Bulk Density (mt ore/m^3)	
			As Received	To Saturate	Retained	Before	After
11-MDS-3	1 1/2-inch	185.00	0.8	13.9	8.8	1.34	1.34
11-MDS-3	1/2-inch	67.43	0.2	17.3	8.2	1.44	1.45
11-MTF-1	1 1/2-inch	172.45	0.6	12.3	8.0	1.37	1.38
11-MTF-1	1/2-inch	66.43	0.2	16.6	9.5	1.46	1.47
11-MDO-2	1 1/2-inch	172.72	0.4	13.4	9.1	1.43	1.42
11-MDO-2	1/2-inch	66.52	0.2	16.7	9.1	1.48	1.48

The crushing characteristics of the samples was also analyzed and is reported in Table 13-12.

Table 13-12: Crushing Characteristics – Composites 11-MDS-3, 11-MTF-1, 11-MDO-2

Sample	Abrasion Index	CWi (kW-hr/st)	CWi (kW-hr/mt)
11-MTF-1	0.1556	10.17	11.21
11-MDO-2	0.3003	11.73	12.93
11-MDS-3	0.1482	10.10	11.14

The samples tested would be classified as moderately hard and moderately abrasive.

13.3 2018 Otis Gold Test Work

In mid-2018, Otis Gold delivered 14 plastic drums of PQ drill core to RDi for analysis. The core was derived from drill hole 17OKC-379 and categorized into three lithologies as shown in Table 13-13.

Table 13-13: Otis Gold Aspen Samples – 17OKC-379

<table>
<tr><th rowspan="2">Sample</th><th rowspan="2">Drill Hole</th><th rowspan="2">Oxidation State</th><th colspan="2">Intervals (ft)</th><th colspan="2">Grade - Au</th><th colspan="2">Grade - Ag</th></tr>
<tr><th>From</th><th>To</th><th>opt</th><th>g/T</th><th>opt</th><th>g/T</th></tr>
<tr><td>Aspen Top – arkosic turbidite sandstone</td><td>17OKC-379</td><td>Unoxidized, hydrothermally altered</td><td>510</td><td>595</td><td>0.227</td><td>7.80</td><td>0.104</td><td>3.55</td></tr>
<tr><td rowspan="2">Aspen Bottom – arkosic turbidite sandstone - siltsone</td><td rowspan="2">17OKC-379</td><td rowspan="2">Unoxidized, hydrothermally altered</td><td>670</td><td>785</td><td rowspan="2">0.044</td><td rowspan="2">1.52</td><td rowspan="2">0.078</td><td rowspan="2">2.68</td></tr>
<tr><td>825</td><td>935</td></tr>
<tr><td>Tertiary - Andesite porphyry flow</td><td>17OKC-379</td><td>Unoxidized, hydrothermally altered</td><td>614.5</td><td>653.5</td><td>0.018</td><td>0.61</td><td>0.155</td><td>5.30</td></tr>
</table>

The goal of the test work was to determine if the Aspen material had similar metallurgical performance to the other areas of the deposit, specifically with respect to heap leach amenability. Details of the test work can be found in the report titled Leach Testing of Kilgore Aspen Samples Otis Gold Corp (Resource Development Inc., 2019). The test work was conducted by RDi in Wheat Ridge Colorado.

The 14 drums representing three different domains were prepared for testing. Each drum was weighed and individually jaw crushed to a particle size of minus 2-inch. Crusher work index samples were taken from each of the three domains, and bulk density samples were taken from each drum. Each drum was thoroughly blended, and approximately 5 to 7 kilograms of material was split out for head assay, moisture determination, mineralogy, and bottle roll leaches. The remaining material from each drum was then combined by domain. Each domain sample was thoroughly blended, and material was split out for column testing, acid-base accounting, and Quantitative Evaluation of Materials by Scanning Electron Microscopy (QEMSCAN). The column testing splits were jaw crushed to the appropriate sizes designated for each column test. The moisture and bulk density results are summarized in Table 13-14.

Table 13-14: Aspen Samples Bulk Density and Moisture

Sample	Bulk Density (t/m3)	Moisture %
Aspen Bottom Drum 1	2.525	0.6
Aspen Bottom Drum 2	2.619	0.1
Aspen Bottom Drum 3	2.593	0.2
Aspen Bottom Drum 4	2.554	0.1
Aspen Bottom Drum 5	2.586	0.3
Aspen Bottom Drum 6	2.647	0.1
Aspen Bottom Drum 7	2.435	0.5
Aspen Bottom Drum 8	2.6	0.1
Aspen Bottom - Average	**2.57**	**0.3**
Aspen Top Drum 9	2.422	3.1
Aspen Top Drum 10	2.298	3
Aspen Top Drum 11	2.636	1.3
Aspen Top Drum 12	2.573	0.6
Aspen Top - Average	**2.543**	**0.8**
Sill Drum 13	2.461	0.1
Sill Drum 14	2.479	0.1
Average Sill - Average	**2.47**	**0.1**

Sample splits from each individual drum were submitted for assay of gold, silver, forms of carbon, forms of sulfur, and ICP analysis. The assay results are summarized in Table 13-15. The gold grades varied significantly, from 0.2 g/T to 16.9 g/T Au. Based on the weighted average of each drum that was used when creating the domain composites, the average gold grade of the Bottom domain would be 1.5 g/T Au, the Top domain would be 7.8 g/T Au, and the Sill domain would be 0.6 g/T Au.

Table 13-15: Otis Gold Samples Head Assays and ICP Analysis – 17OKC-379

Assay	Aspen Bottom								Aspen Top				Sill	
	Drum 1	Drum 2	Drum 3	Drum 4	Drum 5	Drum 6	Drum 7	Drum 8	Drum 9	Drum 10	Drum 11	Drum 12	Drum 13	Drum 14
Au, g/mt	0.795	1.118	3.137	0.274	0.226	0.305	3.281	3.037	1.039	1.652	8.125	16.884	0.339	0.884
Ag, g/mt	0.8	1.4	5.2	4.2	3.6	0.4	4.8	1	0.8	6.6	2.4	4.4	6.8	3.8
Organic C %	0.07	0.04	0.08	0.01	0.05	0.08	0.06	0.06	0.15	0.53	0.07	0.2	<0.01	<0.01
Inorganic C %	0.87	1.09	1.1	1.1	0.89	0.62	1.11	0.96	0.34	0.25	0.54	0.8	<0.01	<0.01
Total C %	0.94	1.13	1.18	1.11	0.94	0.69	1.18	1.01	0.5	0.78	0.61	1	0.02	0.02
Sulfide %	<0.01	0.06	<0.01	<0.01	<0.01	<0.01	<0.01	0.06	0.06	<0.01	0.06	<0.01	<0.01	<0.01
Sulfate %	<0.01	0.02	<0.01	<0.01	<0.01	<0.01	<0.01	0.02	0.02	0.05	0.01	<0.01	<0.01	<0.01
Total S %	<0.01	0.08	<0.01	<0.01	<0.01	<0.01	<0.01	0.08	0.08	0.05	0.07	<0.01	0.13	0.05
ICP %														
Al	4.44	4.89	5.11	5.5	4.87	4.33	5.02	5.35	3.74	5.8	4.89	5.19	1.93	2.21
Ca	5.77	5.23	7.56	5.98	5.06	4.36	8.2	4.77	1.53	0.94	2.84	5.01	0.03	0.03
Fe	1.33	2.11	2.12	2.02	1.69	1.54	2.02	1.84	1.6	3.91	1.63	2.43	1.32	1.32
K	2.92	2.93	2.63	3.71	2.95	2.34	2.95	3.65	2.28	4.87	3.31	4.44	1.58	1.73
Mg	1.22	1.5	1.88	1.77	1.31	1.06	2.17	1.5	1.03	1.07	1.36	2.08	0.56	0.6
Na	0.07	0.06	0.13	0.07	0.11	0.04	0.11	0.05	<0.01	0.06	0.03	0.05	<0.01	<0.01
Ti	0.15	0.13	0.2	0.21	0.15	0.13	0.22	0.14	0.04	0.1	0.07	0.14	<0.01	<0.01
ICP ppm														
As	<10	34	<10	63	87	17	12	30	23	42	25	19	220	134
Ba	703	551	475	534	610	503	505	477	395	433	533	516	129	158
Bi	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Cd	1	3	3	3	4	2	3	2	2	5	2	3	2	2
Co	6	6	8	8	6	5	10	6	4	10	4	8	1	1
Cr	137	126	156	109	116	133	113	105	103	124	117	90	84	73
Cu	12	26	31	34	45	9	33	33	<2	5	5	4	12	55
Mn	390	628	574	450	314	304	444	469	728	2680	458	1210	774	972
Mo	<1	2	2	<1	<1	<1	<1	<1	<1	<1	<1	<1	3	4
Ni	16	21	44	30	24	23	52	20	14	29	16	29	<5	14
Pb	<10	<10	<10	21	<10	<10	<10	27	<10	<10	<10	<10	14	<10
Sr	160	148	195	229	160	126	215	126	31	142	67	164	17	20
V	50	64	70	79	56	50	75	62	37	70	52	60	4	2
W	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Zn	35	54	51	46	78	37	49	24	17	210	62	83	124	155

Samples from each domain were submitted for crusher work index (CWi) and abrasion index testing. A summary of the results is given in Table 13-16.

Table 13-16: Aspen Samples Crusher Work Index

Sample	CWi (kWh/metric ton)	Classification
Aspen Bottom	20.3	Very Hard
Aspen Top	15.5	Medium Hard
Sill	14.0	Medium

Bottle roll leach tests were completed with samples from each of the 14 drums at a particle size of P_{80} 10-mesh. This test is designed to provide an indication of heap leach amenability by using a finer particle size than a heap leach but over a much shorter leach duration of 96 hours. Table 13-17 shows the coarse particle bottle roll cyanidation test results.

Figure 13-13 shows the graph of the leach extractions versus time for the bottle roll cyanidation tests on the Aspen samples.

The leach results indicate the following:

- The maximum gold extractions were observed from the Sill samples. Each drum sample achieved gold extractions of 87.8% and 84.9%.
- The Bottom samples had a wide range of gold extractions, varying from 7.5% to 50.0%, with an average of 27%. Similarly, the gold extraction of the Top samples ranged from 20.2% to 68.0%, with an average of 38%.
- Silver extractions were similar for all tests, ranging from 34.5% to 58.4%, with an average of 52%.
- In general, the calculated head grade from each leach test agreed with the individual head assay.

As a further investigation, the gold extraction was plotted against the feed grade in an attempt to identify a correlation between grade and recovery. This relationship often exists in gold processing due to an unliberated constant tail. Figure 13-14 shows the relationship between grade and recovery for the Aspen samples.

Based on these results, a series of hot cyanide leach tests were conducted to determine if there was a specific mineralogical issue related to the lower gold recoveries in certain areas of the Top and Bottom Aspen formations.

The hot cyanide leach tests utilized 15.0 grams of pulverized sample, 30 milliliters of water at 0.10% sodium cyanide (NaCN) with a 2-hour hot cyanide shake. Gold and silver were assayed by AAS Analysis.

Table 13-18 shows the results of the hot cyanide leach tests.

The hot cyanidation results were inconclusive and did not provide much additional insight into any potential mineralogical issue related to leaching. Based on these results and the previous bottle roll tests, a mineralogical examination was undertaken on select samples from each domain along with a CIL test. The CIL test was conducted because the presence of carbon in the Aspen materials and the previous bottle

Table 13-17: Otis Gold Aspen Samples Coarse Bottle Roll Cyanidation (10 mesh, 96 hours) – 2018-169

	1 - Bot		2 -Bot		3 -Bot		4 -Bot		5 - Bot		6 - Bot		7 - Bot	
Time	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**
2	1.8	7.4	2.3	9.4	1.1	8.4	1.6	4.2	3.2	12.5	3.3	6.6	1.5	8.7
6	1.8	10.3	8.5	18.1	3.2	13.4	3.3	5.7	6.5	17.5	6.7	10	3.7	14
24	3.7	25.3	28.9	38.1	10.7	28.2	5	17	16.3	31.2	13.5	18.4	13.9	30.8
48	5.5	35.8	43.4	50	20.5	39.9	6.7	35.4	26.3	39.4	20.4	26.9	20	41.2
72	5.6	44.8	48.1	55.5	25.1	47.6	10.1	43	30	44.8	27.3	30.7	24	47.7
96	7.5	51	50	56.9	26.6	53.5	11.9	45.1	30.5	49.4	31.1	34.5	25.1	52.1
Residue, g/T	0.78	0.8	1.36	1.4	1.05	3.2	0.82	0.6	0.33	0.8	0.31	0.6	1.56	2
Cal. Feed, g/T	0.84	1.6	2.71	3.2	1.42	6.9	0.93	1.1	0.47	1.6	0.45	0.9	2.08	4.2
NaCN kg/T	0.661		0.718		0.84		0.781		0.658		0.6		0.901	
Lime kg/T	0.723		0.748		0.897		0.718		0.564		0.572		0.594	
	8 - Bot		**9 - Top**		**10 -Top**		**11 -Top**		**12 -Top**		**13 - Sill**		**14 - Sill**	
Time	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**	**Au %**	**Ag %**
2	2.4	6.6	6.4	20.2	3.6	10.9	2.8	13	5.4	7.1	14.9	13.1	19.3	9
6	6	14.6	10.7	26	8.8	18.1	8.4	20.8	13.4	14.5	35.9	19.2	37.1	16.7
24	16.1	32.5	15.1	39.9	21.3	35.5	18.3	38.3	35.4	34.6	63.2	31.3	62.3	32.9
48	23.5	44.9	19.6	44.6	29	46.4	26	51.9	51	45.3	79.1	38.4	66.8	44.1
72	27.4	48.8	19.9	46.6	33	56.2	29.6	56	64.3	51.4	95.3	47.7	81.9	53.4
96	34.3	56.9	20.2	47.3	33.6	58.1	32.4	58	68.0	56.1	87.8	50.6	84.9	58.4
Residue, g/T	1.69	1	0.58	0.6	1.4	3.2	2.25	1.2	6.46	5.8	0.06	2.4	0.13	3
Cal. Feed, g/T	2.57	2.3	0.73	1.1	2.11	7.6	3.32	2.9	20.17	13.2	0.51	4.9	0.86	7.2
NaCN kg/T	1.085		1.634		2.854		1.27		1.569		0.901		0.66	
Lime kg/T	0.717		1.593		2.791		1.23		1.437		0.502		0.512	

Figure 13-13: Otis Gold Sample Bottle Roll Cyanidation Results – 17OKC-379

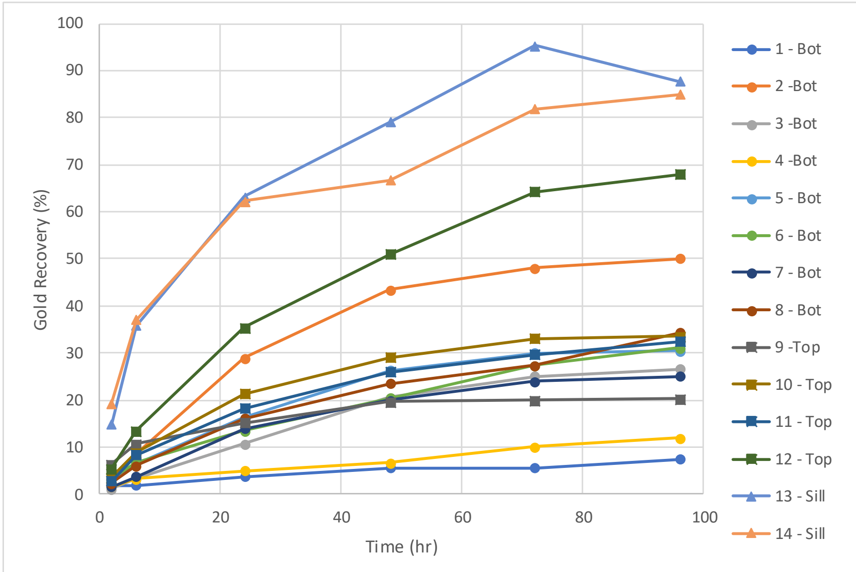


Figure 13-14: Otis Gold Samples Grade versus Gold Recovery for the Aspen Sample Bottle Roll Tests – 17OKC-379

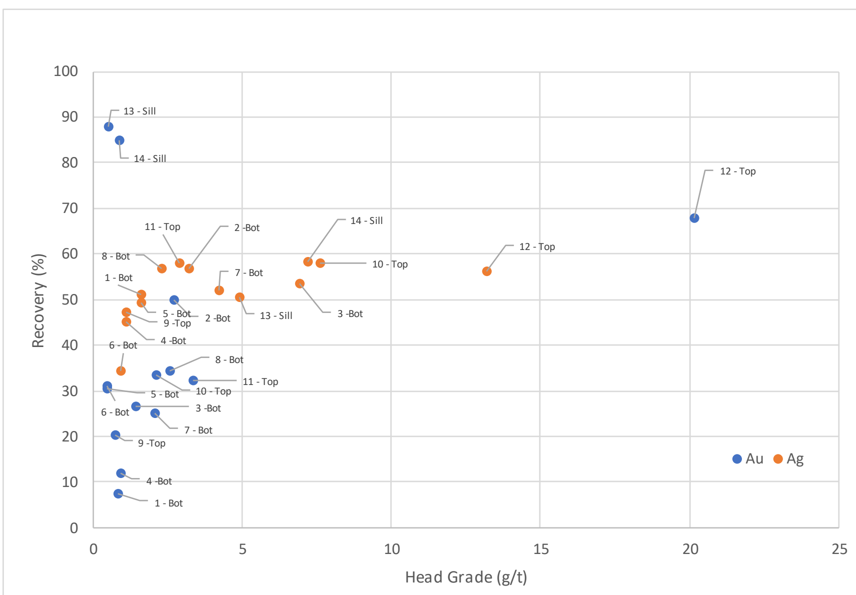


Table 13-18: Otis Gold Aspen Samples Hot Cyanide Leach Results – 17OKC-379

	1 - Bot		2 -Bot		3 -Bot		4 -Bot		5 - Bot		6 - Bot		7 - Bot	
	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %
Grade g/mt	0.16	0.54	0.26	0.96	0.54	2.34	0.04	1.4	0.04	1.5	0.08	0.22	0.3	1.64
Extraction (%)	19.0	33.8	9.6	30.0	38.0	33.9	4.3	127.3	8.5	93.8	17.8	24.4	14.4	39.0

	8 - Bot		9 - Top		10 -Top		11 -Top		12 -Top		13 - Sill		14 - Sill	
	Au %	Au %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %
Grade g/mt	0.28	0.28	0.1	0.52	0.22	4.32	0.78	0.86	1.24	2.46	0.12	4.16	0.22	1.74
Extraction (%)	10.9	10.9	13.7	47.3	10.4	56.8	23.5	29.7	6.1	18.6	23.5	84.9	25.6	24.2

roll leach tests showed the potential for pregnant solution-robbing. The following samples were submitted for CIL testing:

- Aspen Bottom - Drum 2
- Aspen Bottom - Drum 8
- Aspen Top - Drum 10

The samples were ground to a P_{80} of 75 µm. The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to approximately 11 with hydrated lime, and sodium cyanide was added to a calculated level of 1.0 gram per liter (g/L). At six, 24, 48, and 72 hours, the pH and free cyanide were determined. Sodium cyanide was added to return the level to 1.0 g/L, and the pH was adjusted to 10 with hydrated lime if needed. After 96 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents. Table 13-19 shows the results of the CIL testing.

Table 13-19: Otis Gold Aspen Samples CIL Test Results – 17OKC-379

	Head Assay		Extraction		Reagent Consumption	
Sample	Au g/T	Ag g/T	Au %	Ag %	NaCN kg/T	Lime kg/T
Aspen Bottom - Drum 2	3.60	6.70	87.2	55.3	1.953	1.406
Aspen Bottom - Drum 8	3.78	3.70	85.7	56.7	2.133	1.515
Aspen Top - Drum 10	2.55	14.50	87.9	47.4	2.857	3.571

The results from the CIL tests clearly show that good gold extractions can be achieved from the Aspen samples when ground and treated with cyanide in the presence of carbon. A standard CIL process would likely be suitable to treat this material. The Sill samples were not tested as they did not show any gold extraction issues and would be suitable for treatment in a heap leach format.

13.4 2018 Otis Gold Mineralogy – 17OKC-379

To further validate the presence of carbon in the Aspen Bottom and Top materials, samples were submitted by RDi for mineralogical examination to DCM Science Laboratory, Inc. The following samples were submitted for CIL testing:

- Aspen Bottom Drum 2
- Aspen Bottom Drum 8
- Aspen Top Drum 10
- Sill Drum 13

13.4.1 2018 Otis Gold Mineralogy Aspen Bottom Drum 2

Mineralogy: Quartz 60% K-spar/Adularia 18% Calcite 11% Chlorite 2% Clay (undifferentiated) 5% Epidote 3% Carbon 1%

Trace Mineralogy: Pyrite, Chalcopyrite, Sphalerite, Rutile, Plagioclase, Zircon, Iron Oxide, Au

In thin section, this sample contains rock fragments that represent a silicified sediment. The clasts contained in the altered sediment are primarily quartz with lesser amounts of primary igneous potassium feldspar represented by orthoclase/sanidine and grid twinned microcline. The quartz occurs as angular to well-rounded grains with measurements in the 5-µm to 300-µm range. Many of the quartz grains show corroded grain boundaries. The K-spar is generally angular with a grain size that varies from 10 µm up to 150 µm and shows a cloudy appearance from weathering. Angular grains of plagioclase are present as a trace with a grain size up to 50 µm. The quartz/feldspar clasts are firmly cemented by fine grained secondary quartz showing a microcrystalline habit. Large fragments of chert-like quartz with no included clasts are also common. Intermixed with the secondary silica are pockets of brown clay and fine grained, water clear adularia in the 5-µm to 15-µm size range. Secondary adularia is also associated with large 2-mm fragments of coarse vein quartz as attachments. The coarse quartz commonly carries euhedral prisms of epidote displaying anomalous blue colors. Epidote is also seen as prisms and granular inclusions in coarse calcite and silicified sedimentary rock, where it is associated with green chlorite and yellow rutile. Opaque carbon/graphite is present in low amounts and occurs as a fine-grained dust and fragments up to 50 µm. Sulfides are present as a trace but represented by several types. Pyrite is the primary sulfide and occurs as euhedral cubes, thin strings, and anhedral grains in quartz and carbonate. Pyrite grain size varies from 5 µm to approximately 100 µm. Some grains show mild alteration to goethite. Chalcopyrite occurs as small anhedral grains with a maximum size of around 20 µm. Some of the larger pyrite grains carry minute inclusions of chalcopyrite. One large spongy looking grain of yellow sphalerite measuring greater than 1 mm was identified in a silicified rock fragment and carries small inclusions of pyrite. An extensive search of the sample identified one 2-µm grain of Au situated in a small quartz pit. See Photo 13-1 through Photo 13-8.

Photo 13-1: Aspen Bottom Drum 2 - Coarse Quartz With Numerous Inclusions of Epidote Showing Anomalous Blue Colors – 200X PL

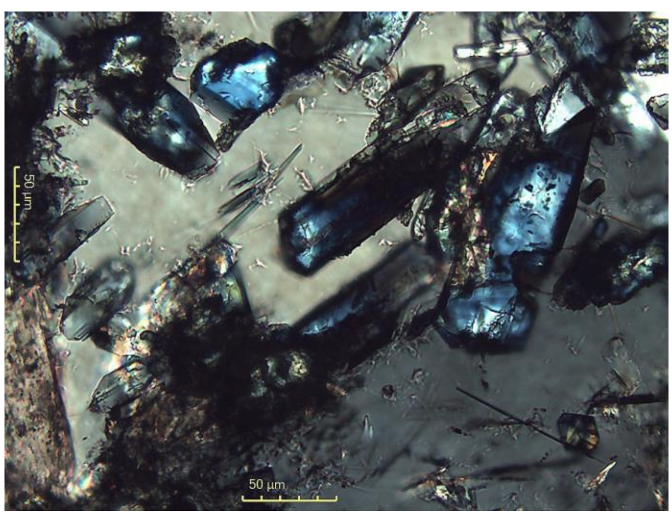


Photo 13-2: Aspen Bottom Drum 2 - Dark Blocky Carbon/Graphite in Secondary Silica – 200X RL

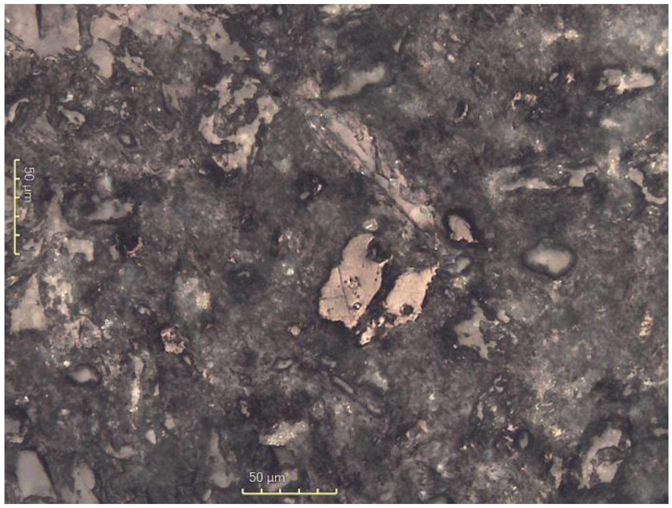


Photo 13-3: Aspen Bottom Drum 2 - A Large Mass of Spongy Looking Sphalerite in a Fragment of Silicified Sediment – 200X RL

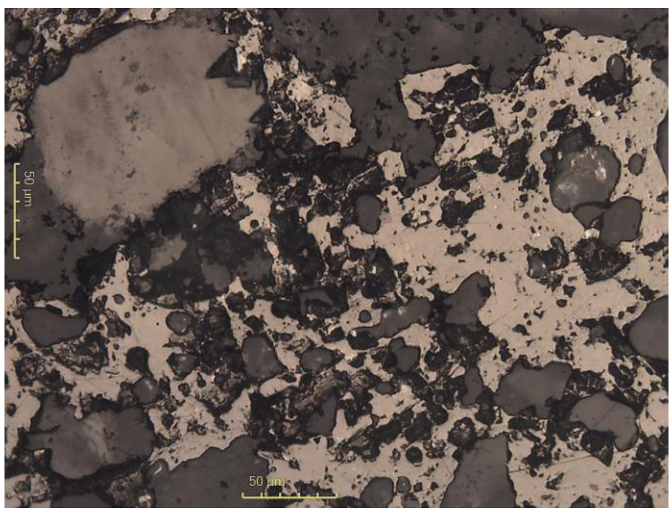


Photo 13-4: Aspen Bottom Drum 2 - Bright 2-µm Au Grain in Secondary Silica – 500X RL

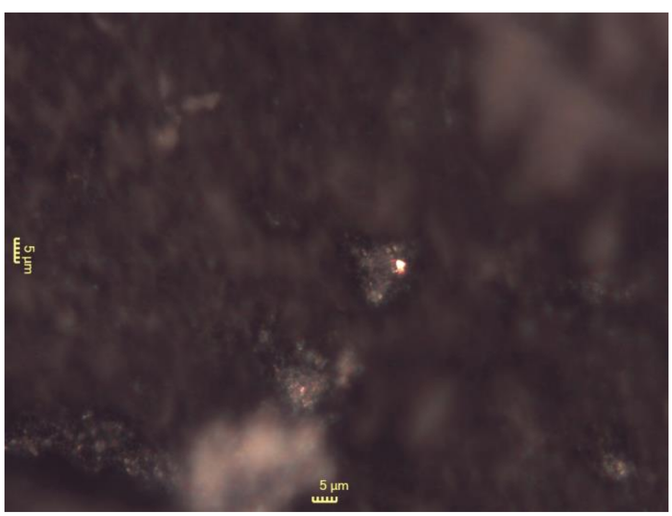


Photo 13-5: Aspen Bottom Drum 2 - Odd Shaped Grain of Pyrite in Coarse Calcite – 200X RL

Photo 13-6: Aspen Bottom Drum 2 - Brown Clay Mass With Opaque Carbon/Graphite And Chlorite – 200X PL

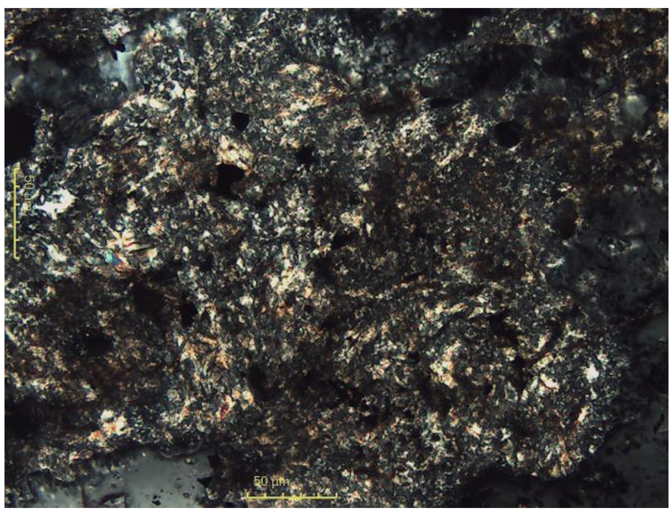


Photo 13-7: Aspen Bottom Drum 2 - Quartz Clasts Cemented By Secondary Silica And Adularia – 200X PL

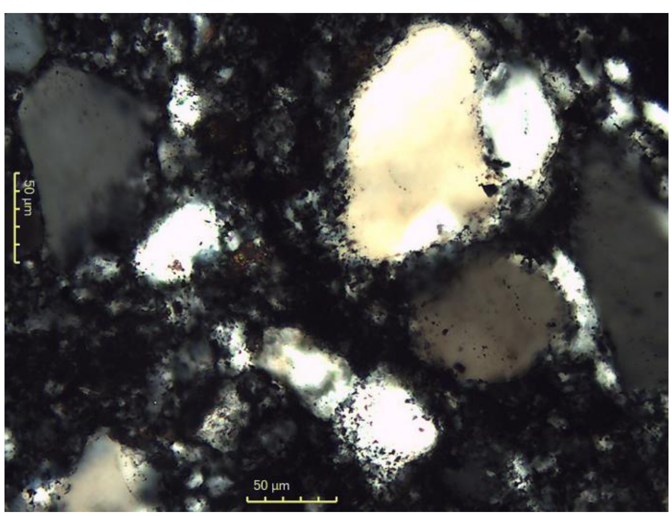
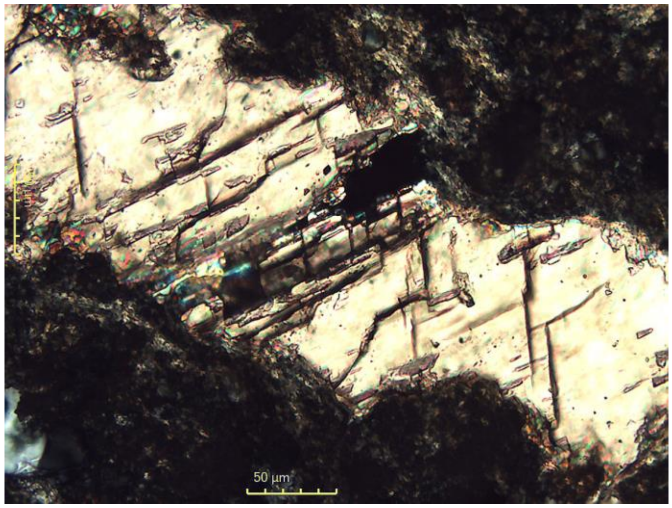


Photo 13-8: Aspen Bottom Drum 2 - Coarse Calcite Flanked By Brown Clay And Chlorite – 200X PL



13.4.2 2018 Otis Gold Mineralogy Aspen Bottom Drum 8

Mineralogy: Quartz 60% K-spar/Adularia 17% Calcite 13% Clay (undifferentiated) 6% Epidote 2% Chlorite 1% Carbon/Graphite 1%

Trace Mineralogy: Pyrite, Chalcopyrite, Rutile, Zircon, Plagioclase, Iron Oxide, Au

In thin section, this sample is essentially the same as Aspen Bottom Drum 2 and represents a silicified clastic sediment. The original sediment is a fine to medium grained arkosic sand subsequently cemented by fine grained secondary hydrothermal silica. Some of the silicified fragments are very fine grained and have the appearance of silt. Individual clasts of angular to well-rounded quartz vary significantly in size from 2 µm up to 400 µm. Many of the grains show mild to moderate corrosion along grain boundaries. Clasts of primarily K-spar and lesser amounts of plagioclase occur as angular fragments in the 2-µm to 150-µm size range and show pitting from dissolution. A brown clay showing a sinuous habit is present in appreciable amounts and occurs as irregularly shaped patches and thin strings cutting most of the clastic fragments. Associated with the clay is green chlorite and small rusty grains of iron oxide. Most of the clastic fragments are dark in color due to carbon/graphite that occurs as thin discontinuous strings, dust and fragments up to 50 µm in size. Adularia is well represented and occurs as fine-grained aggregates and fairly large water clear grains showing sharp crystal faces and zoning. The coarsest grains measure up to 300 µm are generally associated with coarse calcite and coarse secondary vein quartz. The vein quartz commonly carries euhedral prisms of epidote. Sulfides are present as a trace, with pyrite as the main type. Pyrite occurs as small cubes and irregularly shaped grains in carbonate and quartz with a grain size of 2 µm to 75 µm. Chalcopyrite occurs as small 2-µm to 25-µm grains primarily confined to quartz. The sulfides show minor decay to goethite. An extensive search of the sample identified one grain of Au measuring approximately 15 µm. The Au is situated between quartz/feldspar grains in secondary silica. Although other small bright grains were identified in the sample that strongly suggest Au, they are too small for positive identification by light microscopy. See Photo 13-9 through Photo 13-15.

Photo 13-9: Aspen Bottom Drum 8 - String of Opaque Carbon Particles in Silicified Sediment Fragment – 200X RL

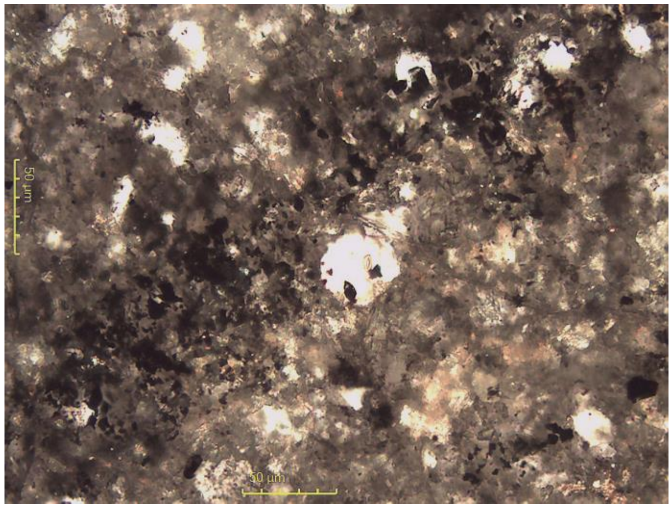


Photo 13-10: Aspen Bottom Drum 8 - Large Carbon/Graphite Grain with Quartz and Calcite – 200X RL

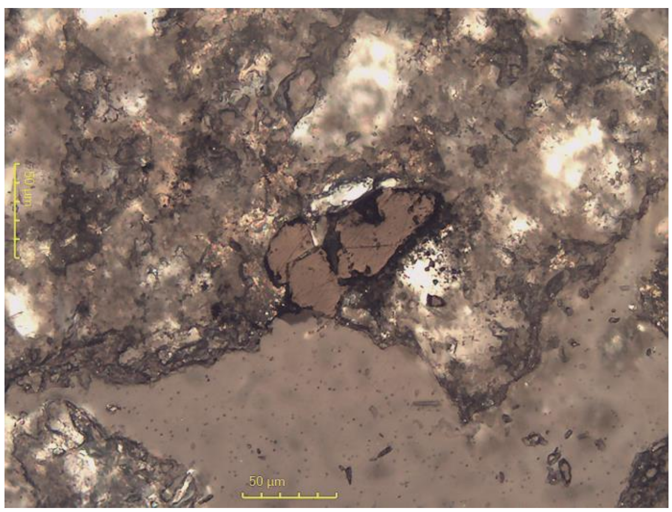


Photo 13-11: Aspen Bottom Drum 8 - Large Grain of Pyrite and Yellow Chalcopyrite with Quartz Grains – 200X RL

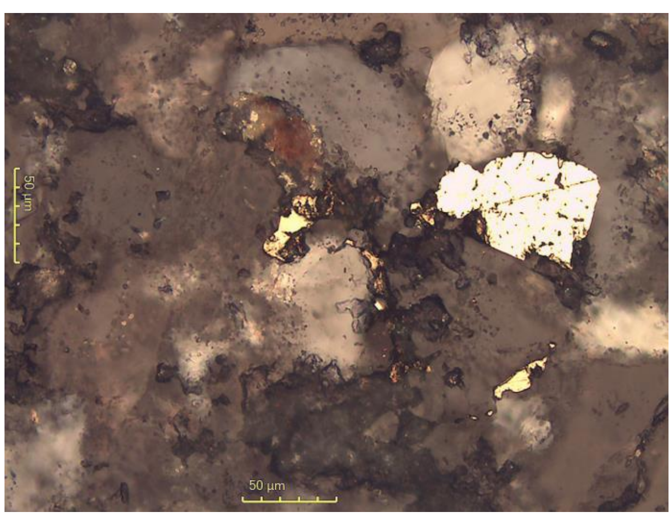


Photo 13-12: Aspen Bottom Drum 8 - A 15-µm Grain of Au Between Quartz Grains – 500X RL

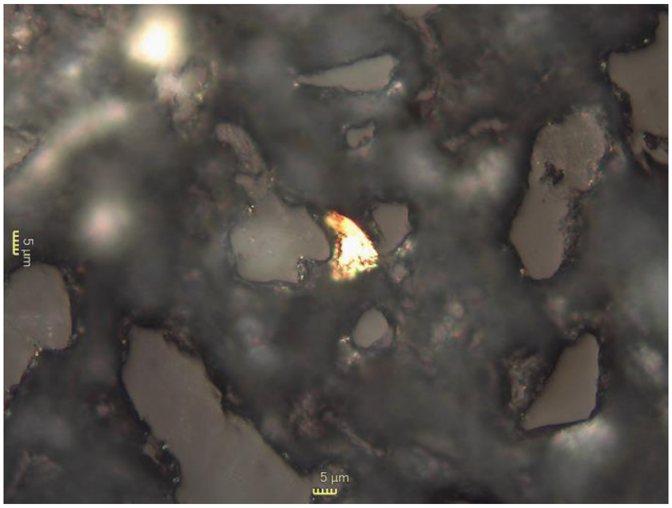


Photo 13-13: Aspen Bottom Drum 8 - Prisms of Anomalous Blue Epidote in Coarse Secondary Quartz – 200X PL

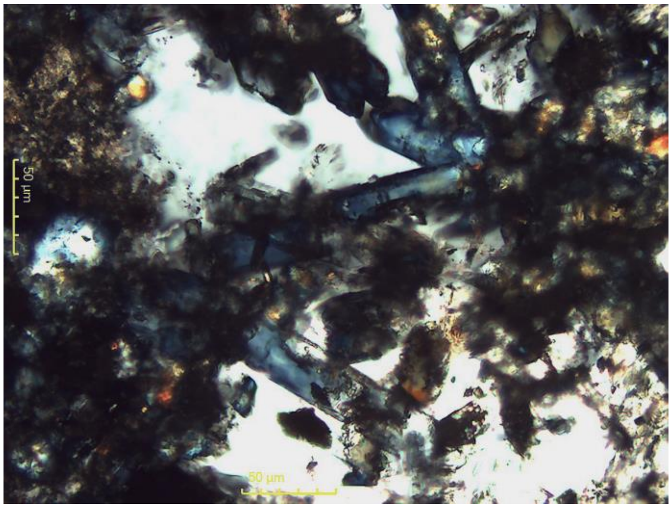


Photo 13-14: Aspen Bottom Drum 8 – Adularia With Inclusions of Carbon Showing Crystal Faces and Zoning – 200X PL

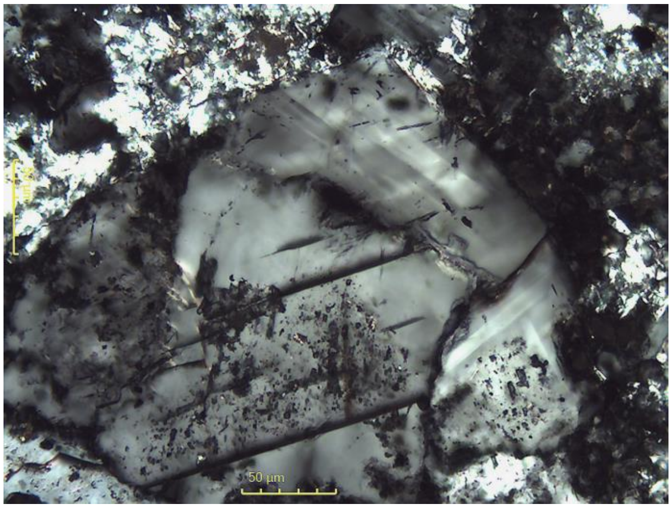
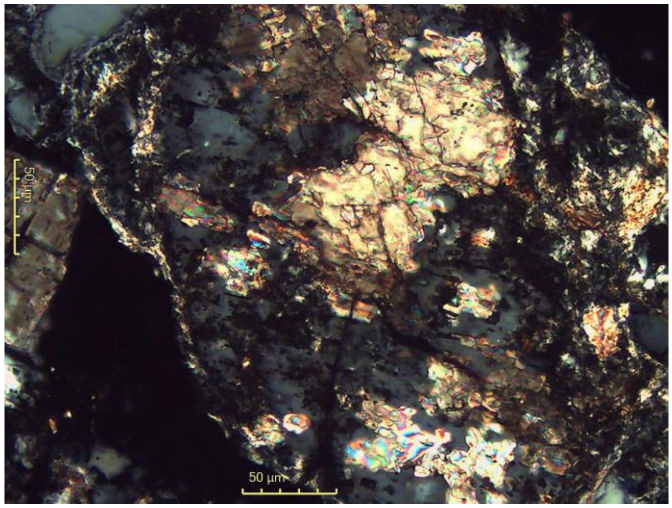


Photo 13-15: Aspen Bottom Drum 8 – Calcite and Grey Adularia – 200X PL



13.4.3 2018 Otis Gold Mineralogy Aspen Top Drum 10

Mineralogy: Quartz 58% K-spar/Adularia 28% Siderite 7% Clay (undifferentiated) 5% Chlorite 1% Carbon/Graphite 1%

Trace Mineralogy: Pyrite, Chalcopyrite, Rutile, Zircon, Epidote, Plagioclase, Iron Oxide, Au

Although there are some subtle differences, this sample is similar to Aspen Bottom Drum 2 and Aspen Bottom Drum 8 and represents a silicified, fine to medium grained clastic sediment. Fragments of microcrystalline chert-like fragments that carry little or no clastic material are common. Like the previous samples, individual clasts of quartz are angular to well-rounded and measure from 2 µm to around 300 µm in size and show corroded grain boundaries. Clasts of K-spar and plagioclase feldspar are generally angular with measurements up to 200 µm. Secondary silica cementing the fragments carry thin strings, blocky grains and fine dust sized particles of carbon/graphite. Interstitial patches and thin veins of brown clay showing a sinuous habit are common in most of the silicified fragments. The clay is commonly associated with green chlorite, rutile, and iron oxide. Coarse grained vein quartz commonly carries numerous prisms of epidote and opaque carbon and is associated with euhedral crystals of secondary, water clear adularia. Adularia also occurs as small interstitial patches in some of the silicified sediment fragments. In contrast to the previous samples, the carbonate contained in this material is siderite. The siderite has a distinct yellow color and generally occurs as fine-grained aggregates with individual grains measuring up to 100 µm. Much of the siderite is iron stained and carries inclusions of sulfides. Pyrite is the dominant sulfide and occurs as small cubes and anhedral grains with measurements that vary from 2 µm up to 50 µm. Chalcopyrite occurs as small anhedral grains up to 10 µm. The sulfides show mild to strong decay to goethite and, in some cases, smaller grains show compete to nearly complete replacement. Au is present but difficult to locate. One 15-µm grain was identified in an aggregate of siderite and quartz. See Photo 13-16 through Photo 13-20.

Photo 13-16: Aspen Top Drum 10 - Bright Pyrite Grain in Yellow Siderite – 200X RL

Photo 13-17: Aspen Top Drum 10 - Aggregate of Colorful Siderite – 200XPL

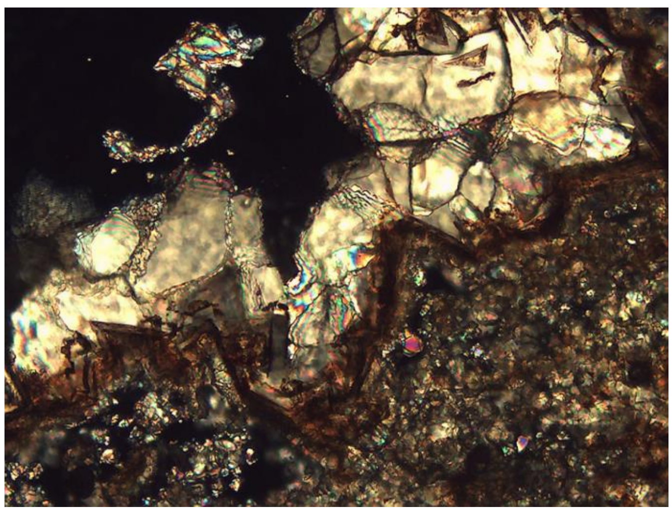


Photo 13-18: Aspen Top Drum 10 - Aggregate of Secondary Quartz and Adularia – 200X PL

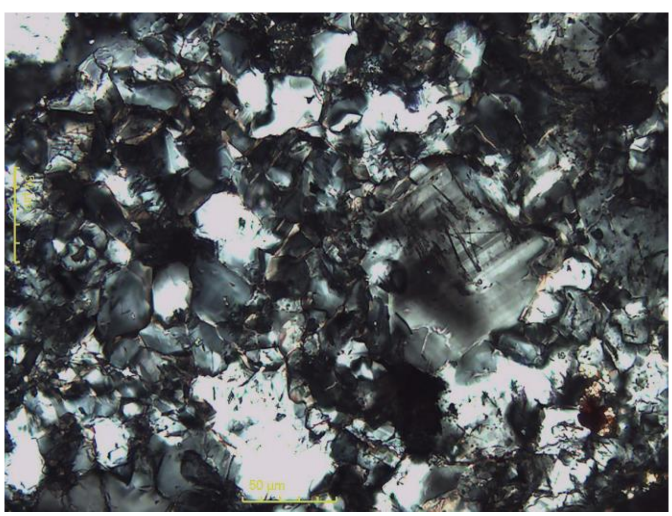
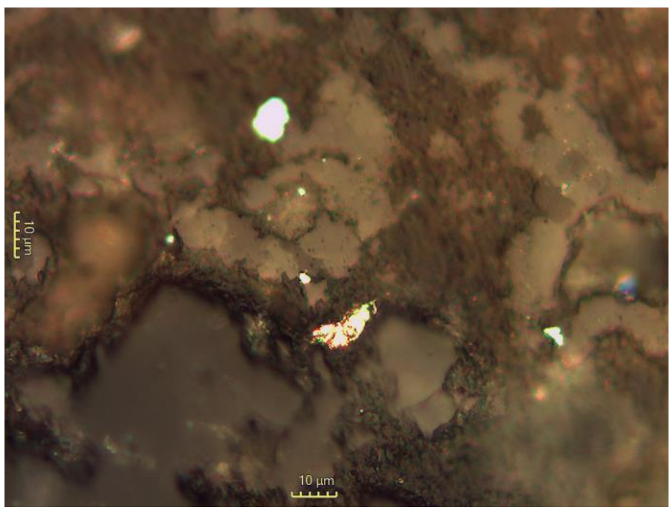


Photo 13-19: Aspen Top Drum 10 - Quartz/Feldspar Clasts Cemented by Secondary Silica – 200X PL

Photo 13-20: Aspen Top Drum 10 - Bright 15-µm Grain of Au with Quartz and Siderite – 500X RL



13.4.4 2018 Otis Gold Mineralogy Sill Drum 13

Mineralogy: Quartz 62% K-spar/Adularia 36% Chlorite 2% Trace Mineralogy: Pyrite, Marcasite, Chalcopyrite, Rutile, Zircon, Epidote, Iron Oxide, Au, Sphalerite, Kaolinite

In thin section, this sample is markedly different than the previous samples. Although this material shows fairly strong alteration, there is no evidence of any silicified clastic material. This sample represents a porphyritic igneous rock composed of quartz and potassium feldspar set in a fine grain microlitic matrix. Quartz occurs as anhedral to euhedral phenocrysts and some bipyramidal forms with a grain size up to 300 µm. Some of the quartz shows embayed margins and all phenocrysts tend to wear ragged, secondary quartz overgrowths. K-spar occurs a euhedral phenocrysts with measurements up to 1 mm. The phenocrysts show moderate to strong alteration to the point where only ghost outlines remain. Some of the phenocrysts show complete replacement by fine grained green chlorite. Due to strong alteration, optical measurements for determination of K-spar type is difficult. However, measurements that could be made indicate the feldspar is likely sanidine. Cross cutting many of the porphyry fragments are thick fractures and micro seams filled with secondary quartz. Although the vein quartz generally has a mosaic texture, some prismatic forms are present. Some of the quartz carries small inclusions of epidote, attachments of clear secondary adularia and small vugs filled with kaolinite. Pyrite is the primary sulfide and occurs as euhedral cubes and anhedral grains with measurements up to 50 µm. Aggregates of pyrite mixed with minor marcasite measure up to 300 µm. Chalcopyrite and yellow colored sphalerite are present as a trace. The sphalerite carries minute exsolution bodies of chalcopyrite. Some of the pyrite/chalcopyrite shows mild to moderate decay to iron oxide. An extensive search of the material identified one 4-µm grain of Au in the matrix of a porphyry fragment. See Photo 13-21 through Photo 13-26.

Photo 13-21: Sill Drum 13 - Bright 4-µm Grain of Au in Porphyry Fragment – 500X RL

Photo 13-22: Sill Drum 13 - Liberated Grain of Yellow Sphalerite with Exsolution Bodies of Yellow Chalcopyrite – 500X RL

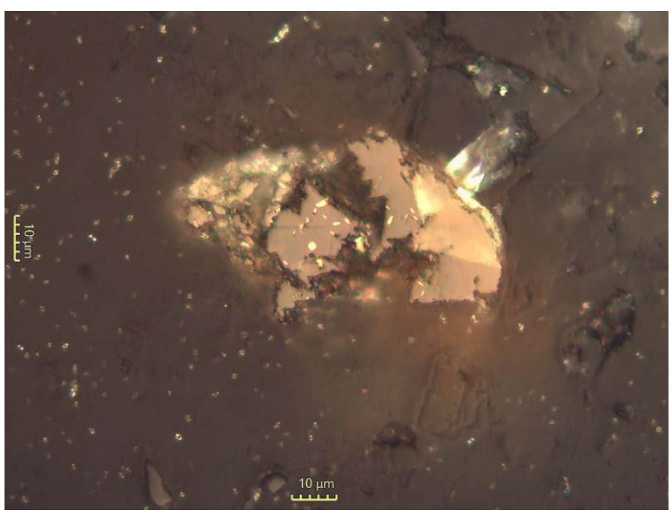


Photo 13-23: Sill Drum 13 – Aggregate of Pyrite and Marcasite with Adularia – 200X RL

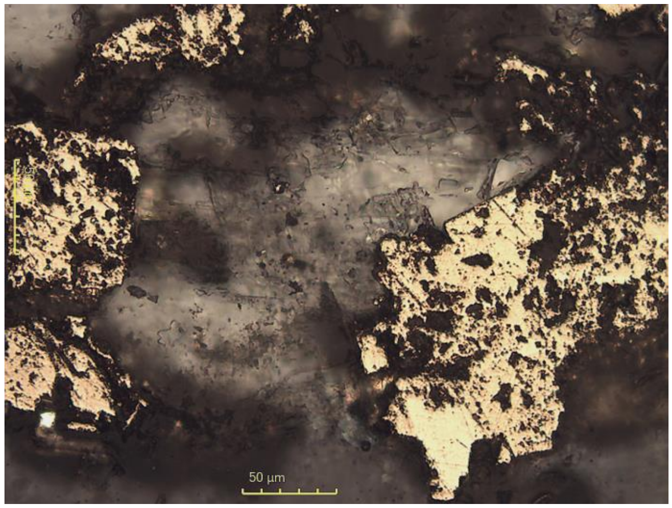


Photo 13-24: Sill Drum 13 – Aggregate of Adularia Attached to Vein Quartz – 200X PL

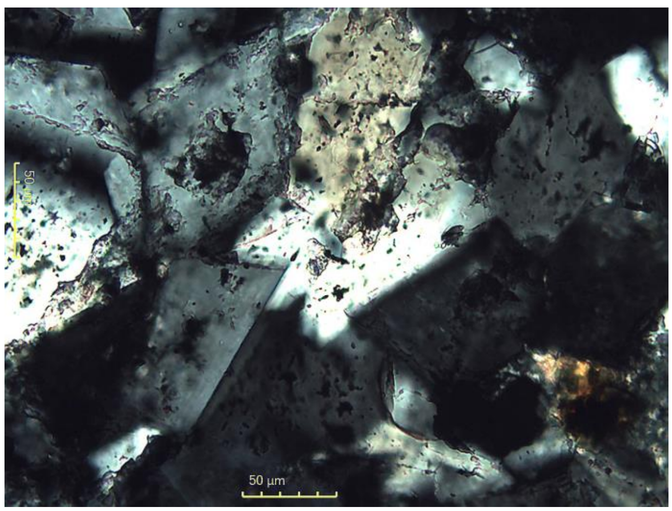


Photo 13-25: Sill Drum 13 – K-Spar Phenocryst Strongly Altered to Chlorite in a Microlitic Groundmass – 100X PL

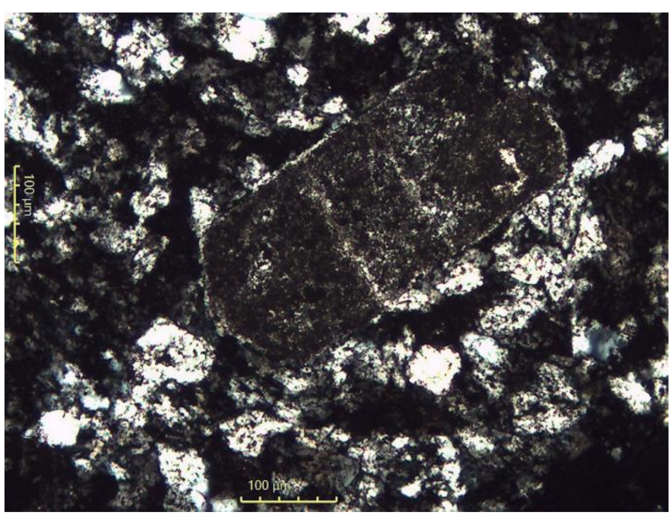
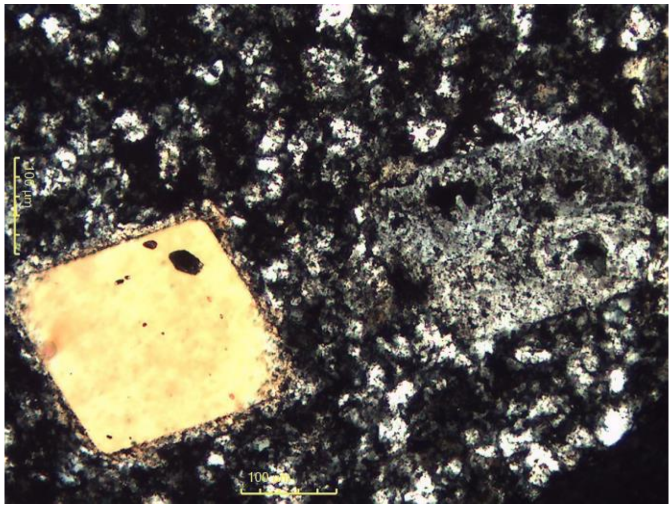


Photo 13-26: Sill Drum 13 – Phenocrysts of Yellow Quartz and Altered K-Spar in a Microlitic Groundmass – 100X PL



13.5 2018 Otis Gold Column Leach Tests – 17OKC-379

Column leach tests were completed with the Sill Domain sample to determine metal extractions at various crush sizes. A total of two columns were completed, one at ½-inch crush and one at 1 ½-inch crush. The column feed was blended with 0.5 kg/mt of lime and loaded in columns, covered, and allowed to cure for 24 hours before leaching. The tests utilized four-and eight-inch diameter columns based on the crush size. Each column was loaded with ore to a height of approximately 6 feet with non-agglomerated feed and leached in open-circuit. The leach solution at 0.5 g/L NaCN and pH of 10.5 was applied at a rate of 0.004-gpm/square foot to each column. The columns were leached for 135 to140 days before rinsing with water, with one rest period. The metal extraction of each column was tracked on a daily basis throughout the leaching process. In addition, the pregnant solution pH and free cyanide were checked on a daily basis.

Once the leaching was completed, each column was detoxified to 10 parts per million (ppm) free cyanide with fresh water before unloading. The leach residue from each column was air dried. Each residue was thoroughly blended and split in half. One-half of the column residue was crushed to minus 10 mesh and a 2-kg sample was split out for assay. The entire 2-kg sample was pulverized to minus 200 mesh before splitting out a sample for fire assay of gold and silver. The column leach test results are summarized in Table 13-20. The metal extractions for each column versus solution addition are given in Figure 13-15 and Figure 13-16.

Table 13-20: Otis Gold Aspen Column Leach Summary – 17OKC-379 Sill Sample

Sample	Feed Size P_{80}	Extraction		Calc Head		Tails Assay		NaCN	Lime
		Au %	Ag %	Au g/T	Ag g/T	Au g/T	Ag g/T	kg/T	kg/T
2018-169 Sill	1/2-inch	74.8	42.8	1.38	8.7	0.348	5.00	2.784	1.035
2018-169 Sill	1 1/2-inch	83.5	29.4	1.42	12.0	0.234	8.50	2.517	0.604

Figure 13-15: Otis Gold Aspen Column Leach Gold Extraction Results P_{80} 1 ½-inch and ½-inch – 17OKC-379 Sill Sample

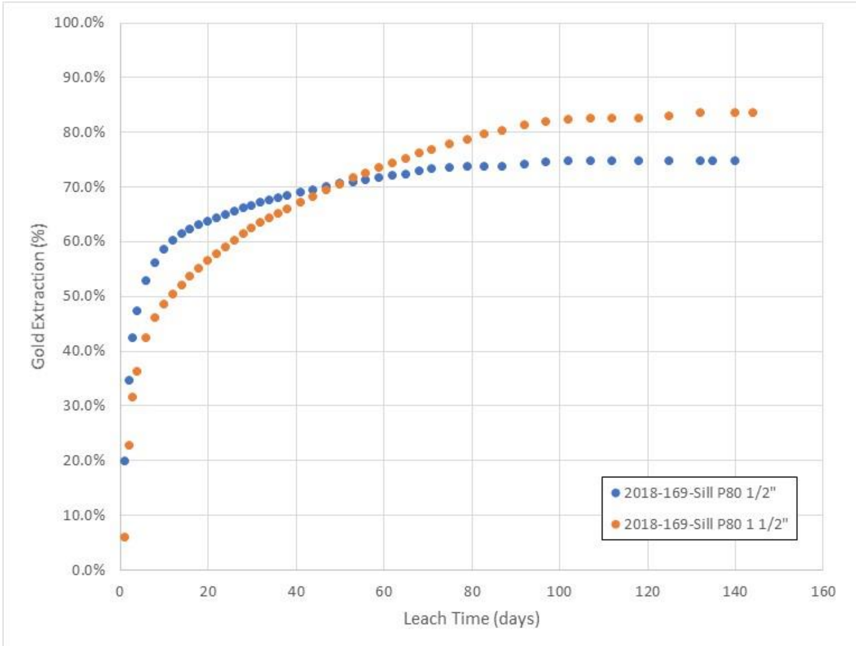
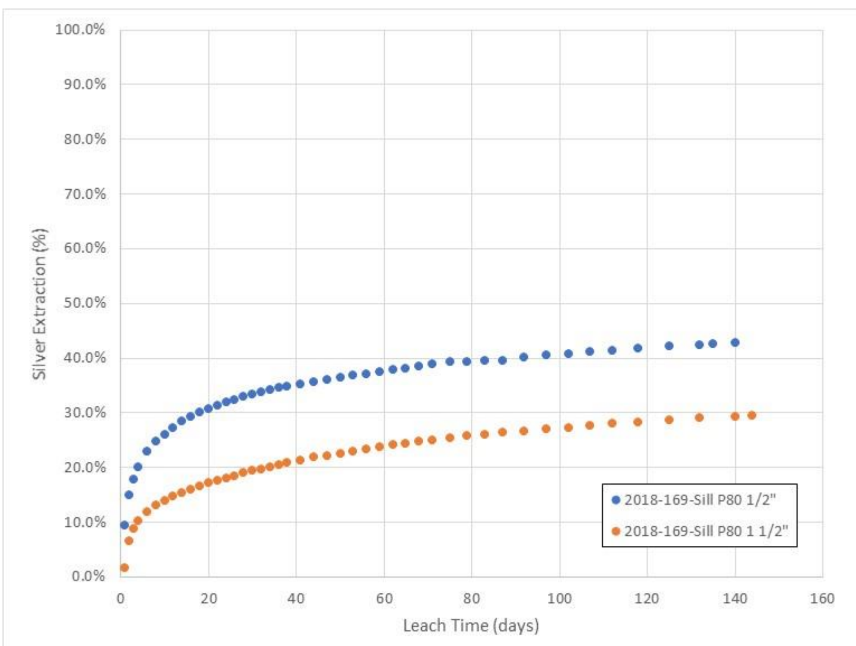


Figure 13-16: Otis Gold Aspen Column Leach Silver Extraction Results P_{80} 1 ½-inch and ½-inch – 17OKC-379 Sill



The height of the ore in the column tests was measured before and after leaching. None of the column tests exhibited significant slumping. Both tests exhibited 0.5% slump or less. Percolation tests were conducted on each column after the rinsing was completed. The ore height and compaction had stabilized before the columns were tested. The percolation test procedure was as follows:

- The column was flooded to a level of approximately two inches above the surface of the material.
- The water flow rate was adjusted to maintain the level above the material surface.
- The amount of solution exiting the bottom of the column was measured to evaluate the flow rate once the solution level was determined to be stable.

A summary of the compaction and percolation test results are reported in Table 13-21.

Table 13-21: Otis Gold Aspen Column Leach Permeability – 17OKC-379 Sill Sample

Sample	Feed Size (P_{80})	Column Density (lb/ft3)	Column Compaction (%)	Percolation Rate Liters/ min	Percolation Rate gpm/ sq. ft.	Percolation Rate Multiple of Application Rate
2018-169 Sill	1/2-inch	95.1	0.2	0.81	2.45	613
2018-169 Sill	1 1/2-inch	93.1	0.5	2.33	1.76	440

Both columns exhibited reasonable permeability. The percolation rate for the ½-inch crush column was 2.5 gpm/square foot, while 1 ½-inch crush column was 1.76 gpm/square foot.

13.6 2019 Otis Gold Test Work - Ongoing

In early-2019, Otis Gold delivered 27 samples of PQ drill core to RDi for analysis. The core was derived from drill hole 17OKC-379, which is a twin of hole 17OKC-356. Details of the sample are shown in Table 13-22.

Table 13-22: Otis Gold Samples – 17OKC-379

Sample	Drill Hole	Oxidation State	Intervals ft From	Intervals ft To	Grade - Au opt	Grade - Au g/T	Grade - Ag opt	Grade - Ag g/T
Volcanic – tertiary lithic tuff	17-OKC-379	Mixed, hydrothermally altered	13	133	0.043	1.47	0.052	1.79

The grades shown in Table 13-22 are a weighted average of assays taken from each lot. The gold grade ranged from 0.058 g/T to 42.68 g/T. The silver grade ranged from less than 1 g/T to 36 g/T. Table 13-23 shows the details of each interval assay.

The goal of the test work was to provide additional information regarding variability of the deposit with respect to leach amenability and to investigate crush size for heap leaching. The test work is currently underway, and only preliminary information on the column leach tests is available. The test work was conducted by RDi in Wheat Ridge, Colorado.

The 27 samples representing an array of domains were prepared for testing. Each sample was weighed and broken to a nominal particle size of 3-inch. The goal was to maintain large particle sizes for a run of mine leach test (+/- 3-inch). A sample from each interval was taken by diamond sawing the core and removing a small section for assay.

Table 13-23: Otis Gold Sample Analysis 17OKC-379

Sample id	Interval	Wt (kg)	Au (g/mt)	Total Carbon (%)	Organic Carbon (%)	Inorganic Carbon (%)	Total Sulfur (%)	Sulfide Sulfur (%)	Sulfate Sulfur (%)	Ag (ppm)	Al (%)	As (ppm)	Ba (ppm)	Bi (ppm)	Ca (%)	Cd (ppm)	Co (ppm)
495054	13-16.6	11.4	0.209	0.03	0.02	0.01	0.24	0.09	0.15	<1	4.09	192	543	<10	0.03	<1	4
495055	16.6-21.7	17.7	0.154	0.04	0.02	0.01	0.76	0.64	0.12	<1	5.08	104	867	<10	0.02	<1	4
495056	21.7-26.6	18.4	0.295	0.04	0.03	0.01	0.25	0.12	0.13	<1	4.94	64	676	<10	0.03	<1	4
495057	26.6-31.7	19.8	0.096	0.04	0.02	0.02	0.76	0.62	0.14	<1	4.59	85	710	<10	0.03	<1	5
495058	31.7-37.2	20.7	0.675	0.04	0.03	0.02	0.49	0.36	0.13	<1	4.44	68	785	<10	0.03	<1	4
495059	37.2-39.7	8.6	0.305	0.04	0.03	0.01	1.47	0.88	0.59	<1	4.17	241	707	<10	0.03	<1	4
495060	39.7-42	9.5	0.240	0.06	0.03	0.03	0.48	0.26	0.22	<1	3.50	592	606	<10	0.03	<1	3
495061	42-47.2	19.8	0.288	0.05	0.03	0.01	1.12	0.92	0.21	<1	3.79	402	677	<10	0.02	<1	3
495062	47.2-49.5	9.3	0.137	0.04	0.02	0.02	2.53	2.23	0.30	<1	3.49	305	663	<10	0.07	<1	4
495063	49.5-54	15.5	0.120	0.06	0.04	0.02	1.85	1.65	0.20	<1	4.10	467	723	<10	0.03	<1	3
495064	54-59	20.5	0.134	0.05	0.03	0.02	1.15	1.06	0.09	<1	4.17	1120	629	<10	0.03	<1	3
495065	59-64	19.3	0.381	0.05	0.03	0.02	0.68	0.56	0.12	<1	3.79	1570	497	<10	0.03	<1	2
495066	64-69	19.3	0.171	0.04	0.03	0.01	0.96	0.55	0.40	<1	3.52	1490	722	<10	0.04	<1	2
495067	69-72.7	14.8	0.391	0.04	0.03	0.01	1.28	0.47	0.81	36	3.50	4800	724	1080	0.03	<1	2
495068	72.7-73.7	3.2	0.154	0.05	0.05	<0.01	0.60	0.34	0.26	2	3.96	353	705	<10	0.03	<1	1
495069	73.5-78.7	20.7	0.137	0.04	0.03	0.01	0.49	0.33	0.16	<1	3.99	831	721	<10	0.03	<1	<1
495070	78.7-84	20.0	0.127	0.04	0.02	0.02	1.04	0.93	0.11	<1	4.02	551	609	<10	0.09	<1	2
495071	84-89	18.9	0.278	0.04	0.02	0.02	0.98	0.90	0.08	<1	3.64	3320	531	<10	0.03	<1	3
495072	89-94	20.0	0.257	0.04	0.03	0.01	2.35	2.24	0.11	<1	3.52	3880	536	<10	0.03	<1	4
495073	94-99	19.3	0.123	0.04	0.03	0.01	1.40	1.31	0.09	<1	3.57	673	552	<10	0.03	<1	4
495074	99-104	19.8	0.182	0.04	0.03	0.02	2.72	2.60	0.12	<1	3.76	360	460	<10	0.03	<1	4
495075	104-109	16.6	1.776	0.04	0.03	0.01	1.76	0.91	0.85	4	4.27	1490	516	<10	0.03	<1	6
495076	109-114	12.3	42.681	0.08	0.06	0.02	1.58	0.62	0.96	15	3.46	2630	339	<10	0.02	<1	2
495077	114-119	17.5	0.319	0.05	0.04	0.01	0.93	0.58	0.35	<1	1.90	462	291	<10	0.03	<1	<1
495078	119-124	18.2	0.219	0.03	0.02	0.01	0.42	0.29	0.13	<1	2.32	429	297	<10	0.03	<1	3
495079	124-129	18.6	0.075	0.05	0.03	0.01	0.29	0.20	0.09	<1	3.23	175	454	<10	0.04	<1	3
495080	129-133	11.6	0.058	0.04	0.03	0.02	0.31	0.24	0.08	<1	2.38	148	280	<10	0.04	<1	3

Sample id	Interval	Cr (ppm)	Cu (ppm)	Fe (%)	K (%)	Mg (%)	Mn (ppm)	Mo (ppm)	Na (%)	Ni (ppm)	Pb (ppm)	Sr (ppm)	Ti (%)	V (ppm)	W (ppm)	Zn (ppm)
495054	13-16.6	90	22	1.85	3.41	0.14	108	5	0.02	6	68	14	0.05	13	<10	7
495055	16.6-21.7	67	18	1.57	5.2	0.11	80	4	0.03	8	58	15	0.07	18	<10	<2
495056	21.7-26.6	80	14	1.04	4.52	0.17	98	4	0.03	7	50	12	0.08	21	<10	<2
495057	26.6-31.7	75	17	1.47	4.74	0.14	92	3	0.03	9	46	16	0.12	20	<10	<2
495058	31.7-37.2	94	13	1.02	4.86	0.1	100	3	0.03	12	55	18	0.11	17	<10	2
495059	37.2-39.7	103	11	3.77	4.59	0.14	109	9	0.04	7	53	22	0.08	19	<10	6
495060	39.7-42	108	16	1.61	3.79	0.06	108	4	0.04	6	92	14	0.07	8	<10	2
495061	42-47.2	111	11	2.11	4.15	0.07	155	3	0.05	5	44	18	0.11	11	<10	13
495062	47.2-49.5	106	31	3.49	3.97	0.07	227	4	0.04	6	78	26	0.12	11	<10	82
495063	49.5-54	93	18	2.64	3.91	0.14	146	2	0.03	7	71	26	0.11	15	<10	8
495064	54-59	85	34	1.43	3.88	0.14	124	2	0.03	6	50	13	0.11	15	<10	42
495065	59-64	107	33	1.39	3.47	0.09	136	3	0.04	5	48	13	0.11	14	<10	37
495066	64-69	117	14	3.18	4.02	0.09	107	4	0.03	<5	83	40	0.11	13	<10	2
495067	69-72.7	103	15	5.21	4.21	0.06	82	12	0.03	<5	350	33	0.12	19	<10	5
495068	72.7-73.7	143	15	2.04	4.41	0.07	96	8	0.04	<5	35	30	0.08	18	<10	<2
495069	73.5-78.7	121	9	1.51	4.10	0.06	104	5	0.05	<5	32	14	0.12	11	<10	<2
495070	78.7-84	100	19	1.43	4.27	0.06	117	4	0.06	6	36	14	0.13	12	<10	4
495071	84-89	128	15	1.41	3.24	0.08	120	6	0.04	8	42	20	0.10	12	<10	22
495072	89-94	154	17	2.62	3.05	0.08	117	12	0.04	10	45	14	0.10	13	<10	11
495073	94-99	145	9	1.82	3.42	0.07	132	6	0.07	10	29	20	0.10	12	<10	<2
495074	99-104	177	13	2.90	3.96	0.07	121	5	0.06	10	25	13	0.11	13	<10	<2
495075	104-109	159	27	4.68	4.81	0.12	88	16	0.04	7	69	26	0.11	20	<10	22
495076	109-114	128	35	5.76	2.18	0.18	67	27	0.04	<5	170	34	0.06	65	<10	<2
495077	114-119	168	14	2.84	1.39	0.07	61	6	0.04	<5	34	36	0.03	17	<10	<2
495078	119-124	164	19	1.35	2.00	0.07	72	5	0.04	<5	36	29	0.04	18	<10	<2
495079	124-129	156	15	0.78	2.40	0.07	71	6	0.04	<5	30	48	0.06	22	<10	<2
495080	129-133	170	15	0.69	1.62	0.10	62	6	0.04	6	22	24	0.05	17	<10	<2

Each domain sample was thoroughly blended, and the material was split out for column testing and mineralogy. The column testing splits were jaw crushed as required to the appropriate sizes designated for each column test. Three column leach tests are underway on this material, including crush sizes of P_{80} ½-inch, 1 ½-inch and 3-inch. The column diameters and heights vary; the ½-inch crush column is 4-inch diameter x 6-feet tall, containing approximately 22 kg of material, the 1 ½-inch crush column is 8-inch diameter x 8-feet tall, containing approximately 88 kg of material, and the 3-inch ROM column is 12-inch diameter x 4-feet tall, containing approximately 256 kg of material. The columns are being irrigated with 0.005 gpm/square feet of 0.5 g/L NaCN solution. The preliminary column results are shown in Figure 13-17 and Figure 13-18.

Figure 13-17: Otis Gold Column Leach Tests Gold Extraction – 17OKC-379 – P_{80} ½-inch, 1 ½-inch, 3-inch

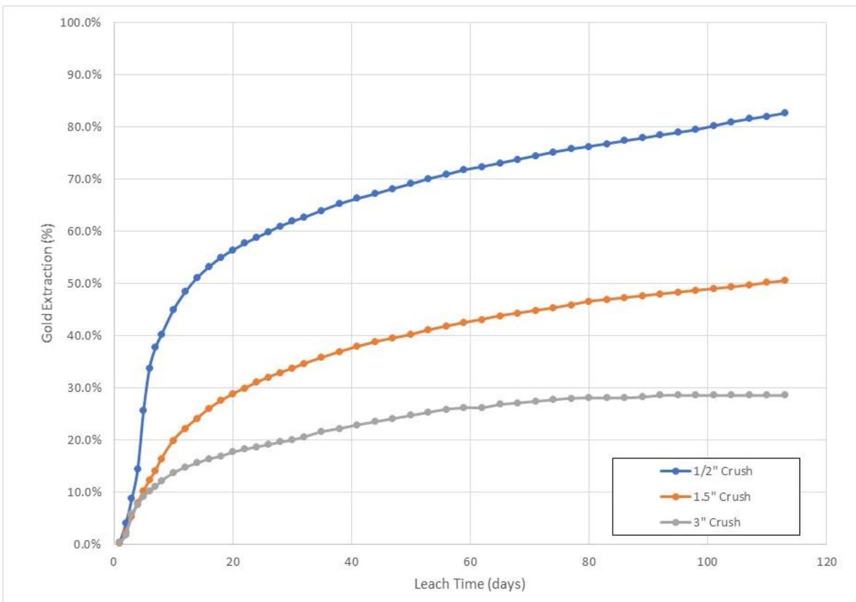
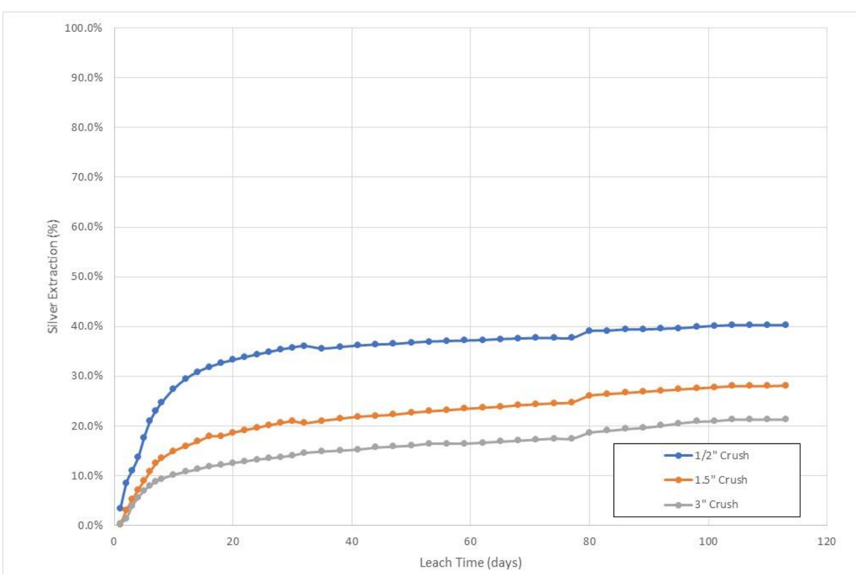


Figure 13-18: Otis Gold Column Leach Tests Silver Extraction – 17-OKC-369 – P_{80} ½-inch, 1 ½-inch, 3-inch



Preliminary projections estimate a gold recovery of 92%, 62%, and 35% at 90 days of leaching for the P_{80} of ½-inch, 1 ½-inch and 3-inch, respectively. This material appears to be sensitive to crush size, unlike some other materials tested.

13.6.1 2019 Mineralogy 17OKC-379

Three samples from the 17-OKC-369 core (OK 495057, OK 495061 and OK 495076) were submitted for mineralogical examination.

Client Sample No.: OK 495057

Mineralogy: Quartz 60% K-feldspar 34% Tourmaline 3% Pyrite 1% Jarosite 1% Clay/Sericite 1%

Trace Mineralogy: Chalcopyrite, Iron Oxide, Rutile, Zircon

In thin section, the sample is primarily composed of rock fragments representing a quartz/feldspar porphyry. Quartz is the primary component of the sample and occurs in several modes. In the porphyry quartz occurs as embayed phenocrysts and small fragments floating in a microcrystalline groundmass showing some alteration to clay/sericite. Some of the quartz phenocrysts have crystal faces and a grain size that varies from 50 µm up to 600 µm. Larger phenocrysts often show ragged quartz overgrowths from secondary silicification. Quartz also occurs as mosaic aggregates, some prismatic forms and chert like microcrystalline fragments. Feldspar occurs as euhedral, water clear phenocrysts locked in the porphyry's groundmass and as liberated fragments. Some of the feldspar shows dissolution and partial replacement by secondary quartz. Optical measurements indicate the majority of K-spar is sanidine with a grain size that varies from 100 µm up to 2.5 mm. The porphyry also carries fragments of syenite with a well-defined trachytic texture. The fragments are composed of turbid orthoclase microlites in a subparallel arrangement. Green tourmaline is well represented and occurs as sprays and individual prisms included in rock fragments, quartz and feldspar. Pyrite is present in low amounts and occurs as euhedral cubes and anhedral grains. Individual grains measure from <2 µm up to 120 µm and show minor oxidation. Although rare, chalcopyrite is present as small 2-µm to 5-µm grains associated with pyrite. Jarosite occurs as large masses and small individual grains in rock fragments. Larger masses measure up to several millimeters and are commonly mixed with iron oxide. An extensive search of this sample failed to identify Au. (See Photo 13-27 through Photo 13-31)

Photo 13-27: Client Sample No.: OK 495057 Irregularly Shaped Pyrite Grains in Quartz – 200X RL

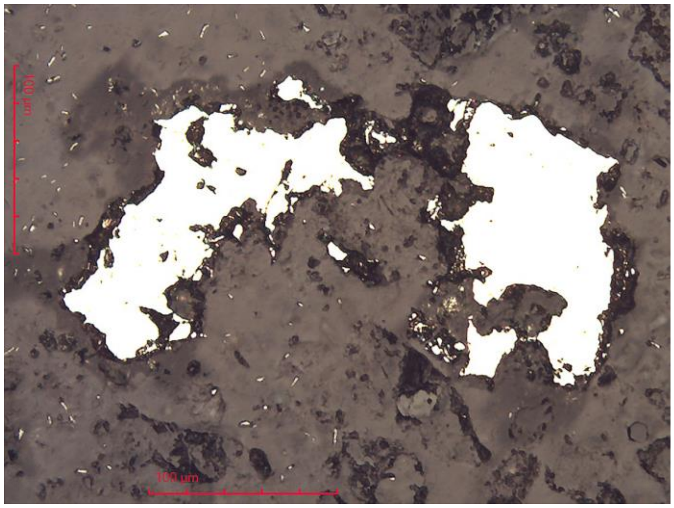


Photo 13-28: Client Sample No.: OK 495057 Large Aggregate of Yellow Jarosite – 100X PL

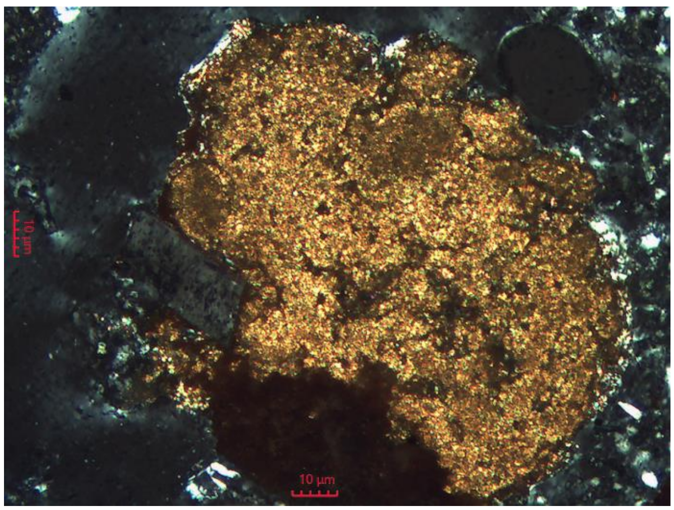


Photo 13-29: Client Sample No.: OK 495057 Phenocryst of K-spar with Inclusions of Tourmaline – 100X PL

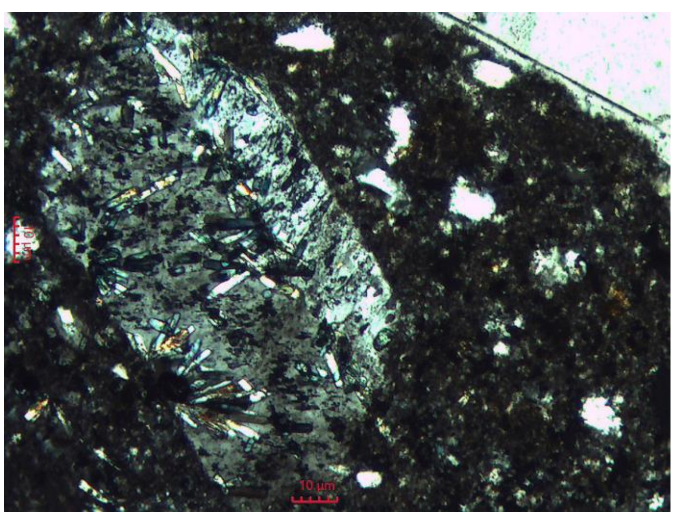


Photo 13-30: Client Sample No.: OK 495057 Large Aggregate of Iron Oxide, Jarosite and Quartz – 100X PL

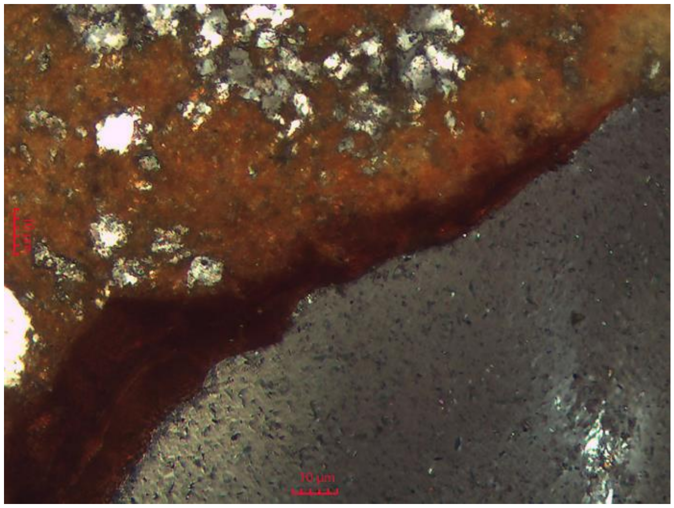
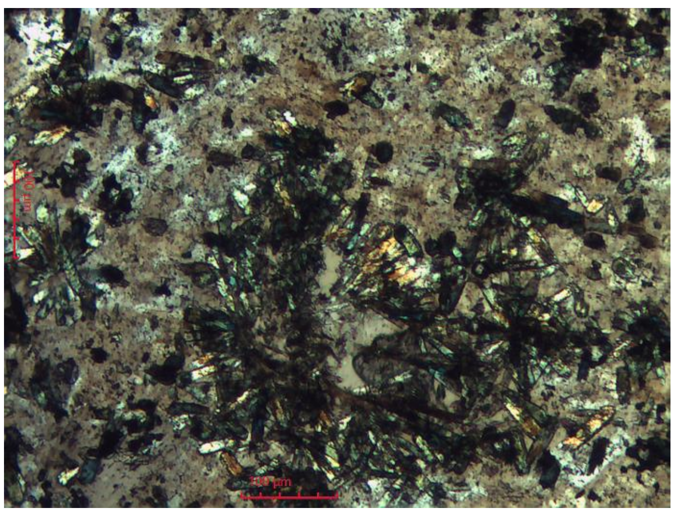


Photo 13-31: Client Sample No.: OK 495057 Green tourmaline prisms in a fine grained groundmass – 100X PL



<u>Client Sample No.: OK 495061</u>

Mineralogy: Quartz 63% K-feldspar 28% Jarosite 4% Tourmaline 3% Pyrite 1% Clay/Sericite 1%

Trace Mineralogy: Chalcopyrite, Iron Oxide, Rutile, Zircon, Au

In thin section, this sample is essentially the same as OK 495057 in terms of mineralogy. The primary difference is this material seems to have a greater degree of secondary silicification. There are more aggregates of mosaic quartz and microcrystalline chert like quartz and the groundmass of porphyry fragments shows a coarser texture and more seritization. It is not uncommon to see fractures in rock fragments filled with secondary quartz. Some vugs and fractures are lined with prismatic quartz showing sharp crystal faces. The feldspar tends to show greater dissolution and cloudiness. Tourmaline content and its prismatic occurrence are generally the same in this material. Pyrite content and grain size measurements are the same. Individual grains/cubes are generally confined to secondary quartz aggregates. Jarosite shows an increase in volume and occurs as large liberated aggregates measuring several millimeters and as aggregates filling interstitial areas between secondary quartz. Small aggregates and individual jarosite grains are also seen scattered throughout larger rock fragments. An extensive search of this sample identified one fairly large 20-µm grain of Au included in a mass of yellow jarosite. (See Photo 13-32 through Photo 13-35)

Photo 13-32: Client Sample No.: OK 495061 Numerous Grains of Pyrite in Quartz – 100X RL

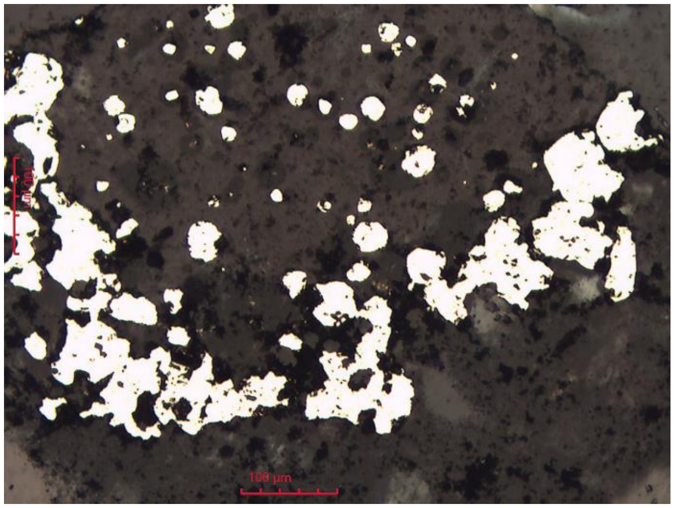


Photo 13-33: Client Sample No.: OK 495061 Gold Grain Measuring 20µm in a Mass of Jarosite – 500X RL

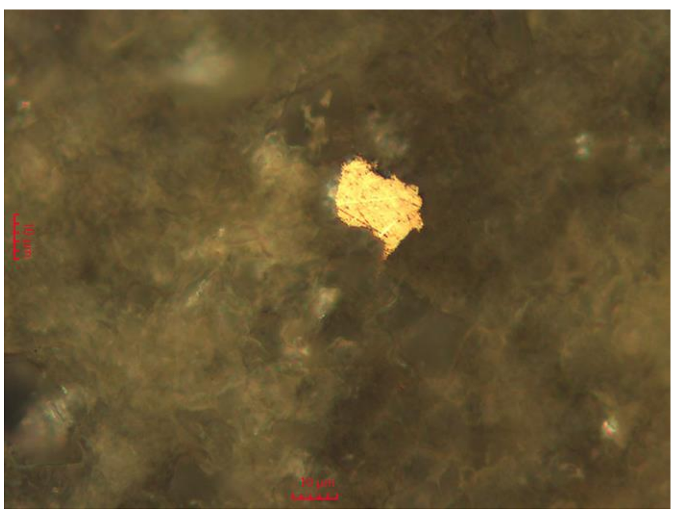


Photo 13-34: Client Sample No.: OK 495061 Phenocryst of K-spar Surrounded by Iron Oxide and Jarosite – 100X PL

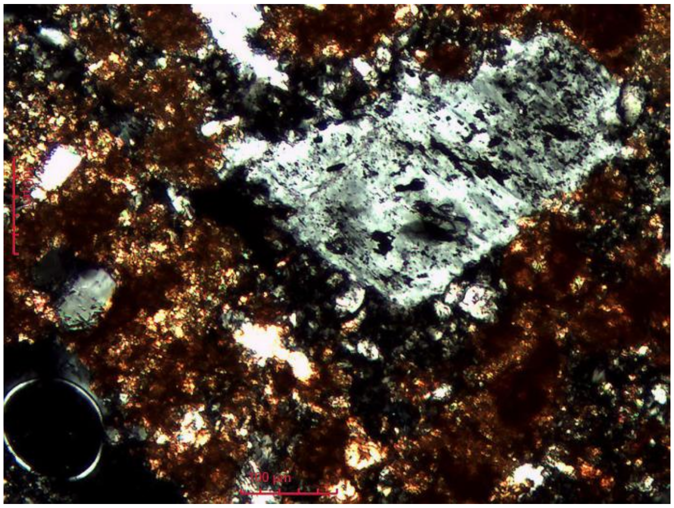
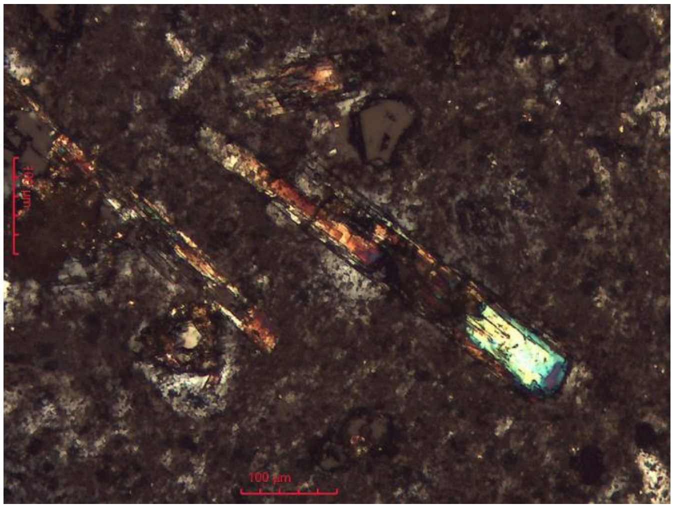


Photo 13-35: Client Sample No.: OK 495061 Prisms of Tourmaline in Fine-grained Groundmass – 100X PL



<u>Client Sample No.: OK 495076</u>

Mineralogy: Quartz 68% Jarosite 20% Illite/Smectite 6% Opaline Silica 3% Tourmaline 1% Kaolinite 1% Sericite 1%

Trace Mineralogy: Pyrite, Arsenopyrite, Rutile, Zircon, Iron Oxide, Au

In contrast to samples OK 495057 and OK 495061, this material is primarily composed of hydrothermal silica and no apparent feldspar. Quartz occurs in several modes and varies greatly in grain size. The majority of quartz is in the form of large aggregates with a very fine to coarse grained mosaic texture with grain size measurements that vary from 5 µm to several millimeters. Intermixed with granular quartz are patches and seams of microcrystalline chert/chalcedony and prismatic forms that occupy small vugs. Also cutting quartz aggregates are thin seams of dark colored opaline silica. Opaline silica also occurs as large fragments several millimeters in size that carry numerous inclusions of small yellow jarosite crystals.

Although some of the opal shows minor devitrification, the majority is isotropic and has a glassy appearance. Brown colored clay with a sinuous habit occurs as large mats and thin seams in quartz associated with sericite. The clay is thought to be a mix of illite and swelling smectite, however, x-ray diffraction would be needed to speciate the types. Clear vermicular kaolinite is seen filling small vugs and interstitial space between silica grains in some quartz aggregates. Green tourmaline occurs as individual prisms and spheroidal sprays confined to quartz and occasionally jarosite. Individual tourmaline prisms measure up to 120 µm in length. Sulfides are present as a trace with pyrite as the primary type. Pyrite grains occur as euhedral cubes in quartz with measurements between 2 µm to 125 µm. Although pyrite generally shows no alteration, a few grains show complete replacement by jarosite. Arsenopyrite is rare, however, a few prisms and rhomb shaped grains up to 6 µm were identified as inclusions in quartz. Jarosite is present in significant amounts and occurs as small individual grains and large continuous masses measuring several millimeters where it is mixed with minor iron oxide. Gold is relatively easy to find in this sample. Although a 20-µm grain was identified with quartz/sericite, jarosite appears to be the primary host for Au. Several grains of Au were identified as inclusions in jarosite where it occurs as thin strings and irregularly shaped grains measuring from <2 µm up to 70 µm in size. (See Photo 13-36 through Photo 13-46)

Photo 13-36: Client Sample No.: OK 495076 Jarosite Pseudomorph after Pyrite in Quartz – 100X RL

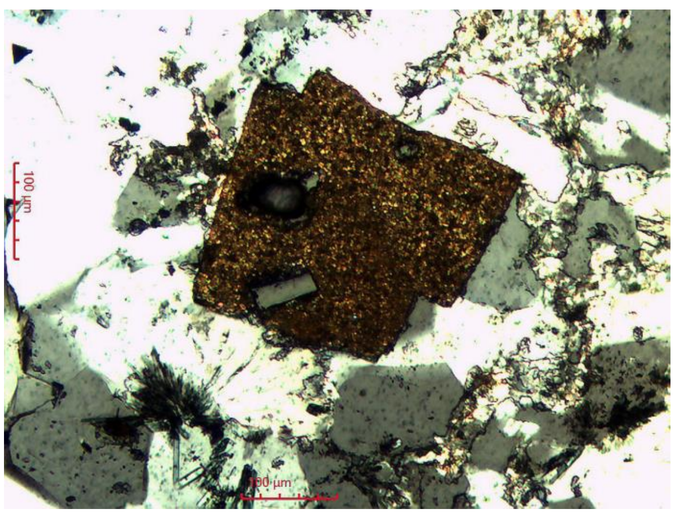


Photo 13-37: Client Sample No.: OK 495076 Small Vug Filled with Vermicular Kaolinite – 200X PL

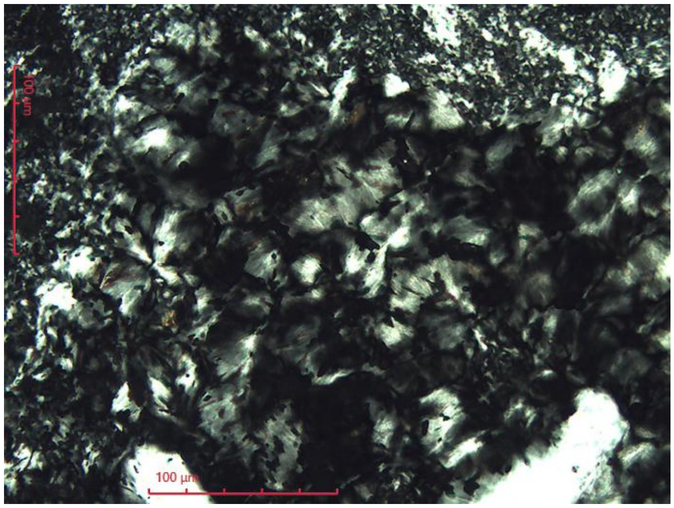


Photo 13-38: Client Sample No.: OK 495076 Euhedral Pyrite in an Aggregate of Quartz – 200X RL

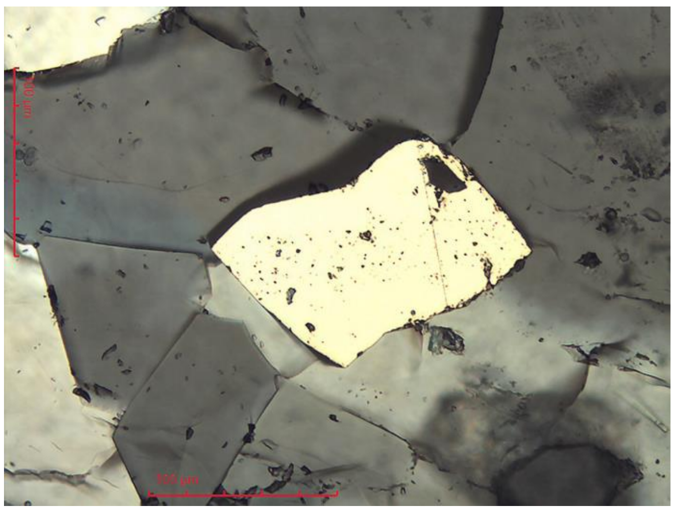


Photo 13-39: Client Sample No.: OK 495076 Several Grains of Bright Au in Jarosite with Prismatic Quartz – 200X RL

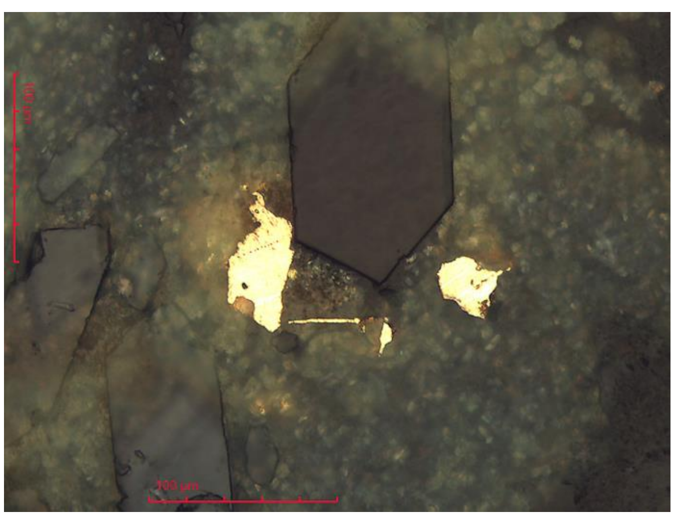


Photo 13-40: Client Sample No.: OK 495076 Small Grain of Bright Au Between Sericite and Quartz – 200X RL

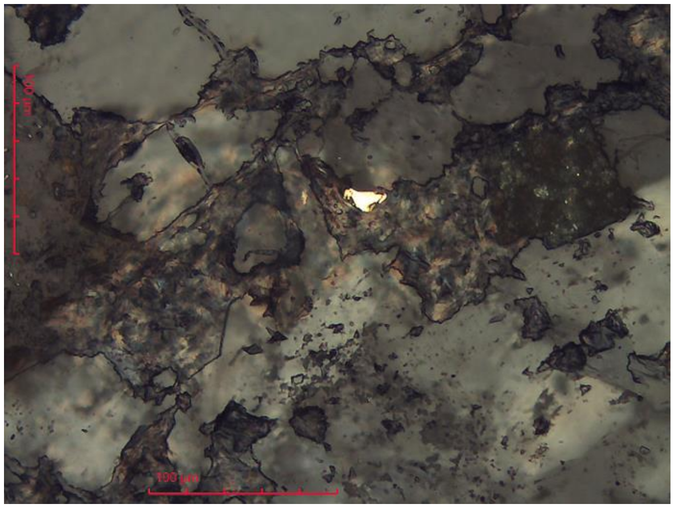


Photo 13-41: Client Sample No.: OK 495076 Cluster of Bright Au in a Matrix of Jarosite – 200X RL

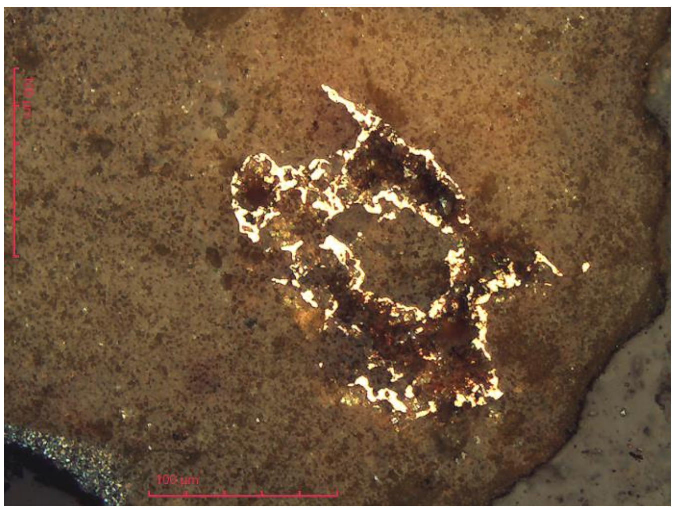


Photo 13-42: Client Sample No.: OK 495076 Yellow Jarosite and Colorful Prisms of Tourmaline – 200X PL

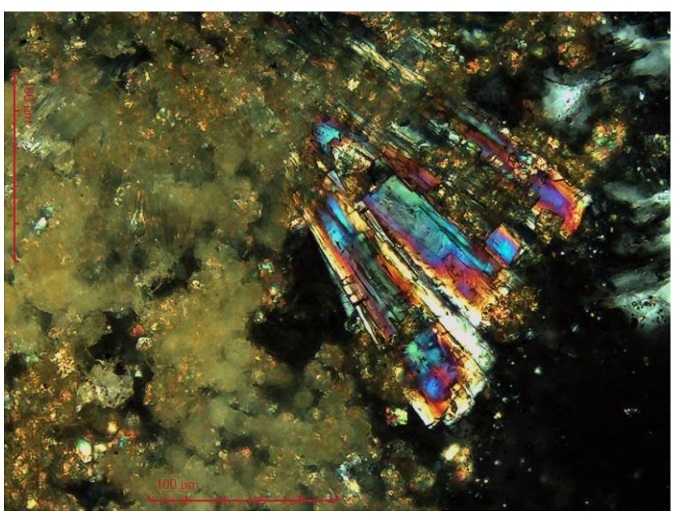


Photo 13-43: Client Sample No.: OK 495076 Corroded Quartz Bordered by Brown Opal with Inclusions of Jarosite – 200X PL

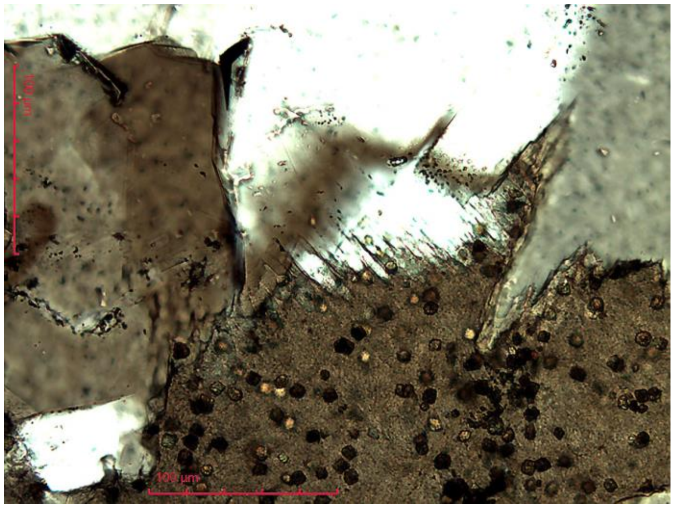


Photo 13-44: Client Sample No.: OK 495076 Small Grains of Arsenopyrite Locked in Quartz – 500X RL

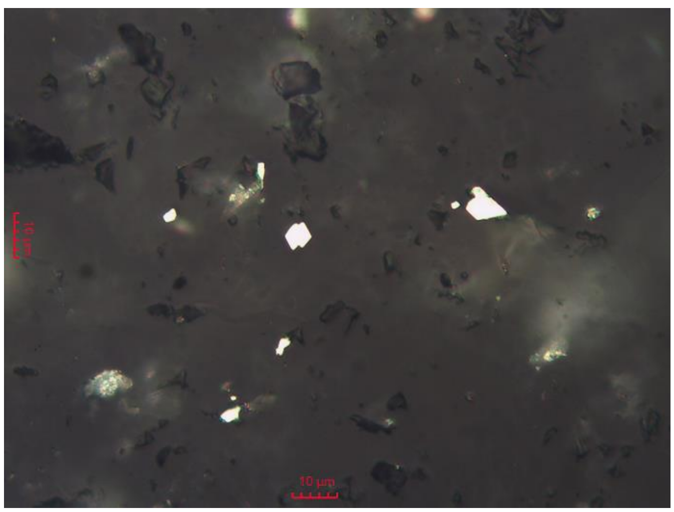


Photo 13-45: Client Sample No.: OK 495076 Large Mass of Opaline Silica Peppered with Inclusions of Jarosite – 100X PL

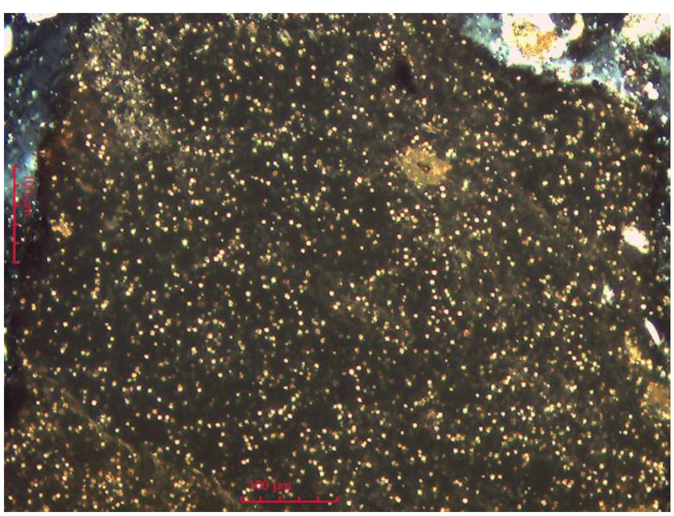
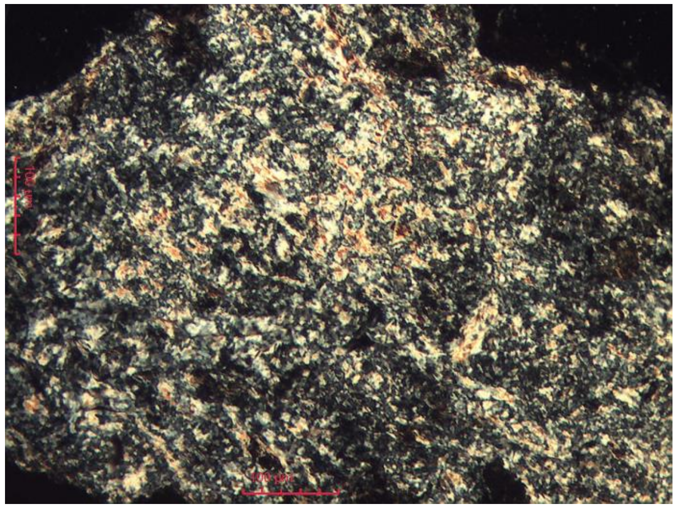


Photo 13-46: Client Sample No.: OK 495076 Large Mass of Brown Clay Assumed to be Illite/Sericite – 100X PL



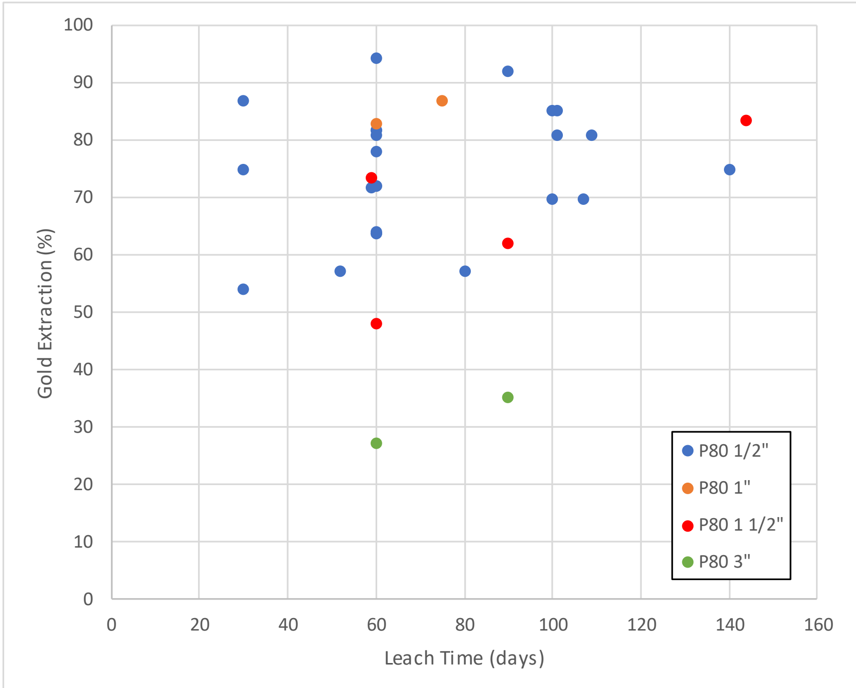
13.7 Test Work Summary

In general, the Otis Gold deposit is characterized by materials that are highly amenable to heap leaching. In many cases, the ore benefits from a finer crush size both in terms of gold recovery and extraction kinetics. One area of the deposit, identified as Aspen (2018-169), had significantly higher grades but also the presence of “pregnant solution-robbing” carbon. This material when treated by CIL had excellent gold extractions. This material is not amenable to heap leaching in its current form.

The Kilgore deposit samples had a moderate abrasion index and a moderate crushing index.

Figure 13-19 shows the summary of all the gold extractions for all column leach tests to date. Both the mid-point (generally 60 days) and final leach period have been plotted.

Figure 13-19: Summary of Column Leach Gold Extractions by Leach Time and Crush Size



As shown by Figure 13-19, the ore shows a fairly high degree of variability at coarser crush sizes but less at the finer crush size of P_{80} ½-inch. Gold extractions tend to be good in most cases at periods of 90 days or more for the ½-inch material.

Based on the metallurgical review, a conservative approach has been taken to ensure maximum gold and silver recovery is obtained.

- A crush size P_{80} of ½-inch has been selected for the heap design.
- A primary leach period of 90 days should be employed. Based on these parameters, gold and silver extractions of 82% and 40%, respectively, should be achievable.
- Cyanide and lime consumptions are moderate. The cyanide consumption has been scaled from the average for all column tests of 2.16 pounds per ton (lb/t) to a projected heap consumption of 0.5 lb/t. The average lime consumption from the column tests has been employed (with the removal of one outlier) to provide an expected consumption of 2.0 lb/t.
- No agglomeration is necessary as the column tests all exhibited excellent permeability.
- The silver grade does not appear to be high enough to warrant the use of a Merrill-Crowe recovery system. A standard carbon adsorption circuit should be acceptable.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The Mineral resource statement presented herein represents the second mineral resource estimate reported by Otis Gold for the Kilgore Project in accordance with the Canadian Securities Administrators' National Instrument (NI) 43-101. The mineral resource evaluation reported was originally published in the report titled "Independent Technical Report and Mineral Resource Estimate for the Kilgore Project, Clark County, Idaho, U.S.A." prepared by GRE and Rowearth (2018). This section re-iterates the Mineral Resource Estimate methodology and results presented in the 2018 Technical Report.

The mineral resources were estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian National Instrument 43-101. Mineral resource estimates do not account for mine-ability, selectivity, mining loss and dilution. This mineral resource estimate includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that the inferred mineral resources will be converted to the measured or indicated categories through further drilling, or into mineral reserves, once economic considerations are applied. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve. The project presently has no mineral reserves.

The mineral resource estimate for the Kilgore Project was completed by David Rowe, CPG, of Rowearth LLC, who is an independent Qualified Person as defined in NI 43-101. The effective date of the resource statement is August 14, 2018. Vulcan's Pit optimization was applied to the resource estimate to assess the reasonable prospects for economic extraction for the resource, and was completed by Kelsey Stark, Mining Engineer, under the direction of Terre Lane, Principal Mining Engineer, of GRE. Peer review of the mineral resource estimate was completed by Terre Lane.

In the opinion of Rowearth, the mineral resource estimate reported here is a reasonable representation of the global mineral resources found in the Kilgore Project at the current level of sampling.

14.2 Resource Estimation Procedures

The gold resource estimate is based on the current drill hole database, interpreted geology and fault structures, and topographic data. Three-dimensional (3D) geologic modelling was completed with Leapfrog Geo, and estimation of mineral resources was completed using Leapfrog EDGE (Version 4.3).

Geostatistical analysis, semi-variogram analysis, and block model validation were completed using both Snowden Supervisor™ Version 8.8 and Leapfrog EDGE.

The resource evaluation methodology involved the following procedures:

- Database compilation and verification

- Construction of 3D geologic models for lithology, alteration, oxidation state, and gold mineralization in Leapfrog Geo
- Definition of the resource estimation domains for use in the gold and specific gravity estimations
- Sample data preparation (compositing and capping) for geostatistical analysis, variography, and block model estimation
- Block modeling and grade estimation in Leapfrog EDGE
- Resource validation and classification
- Assessment of "reasonable prospects for eventual economic extraction" and selection of appropriate cutoff grades with Vulcan pit optimization
- Preparation of the Mineral Resource Statement and Grade Sensitivity Analysis.

14.3 Drill Hole Database for the Resource

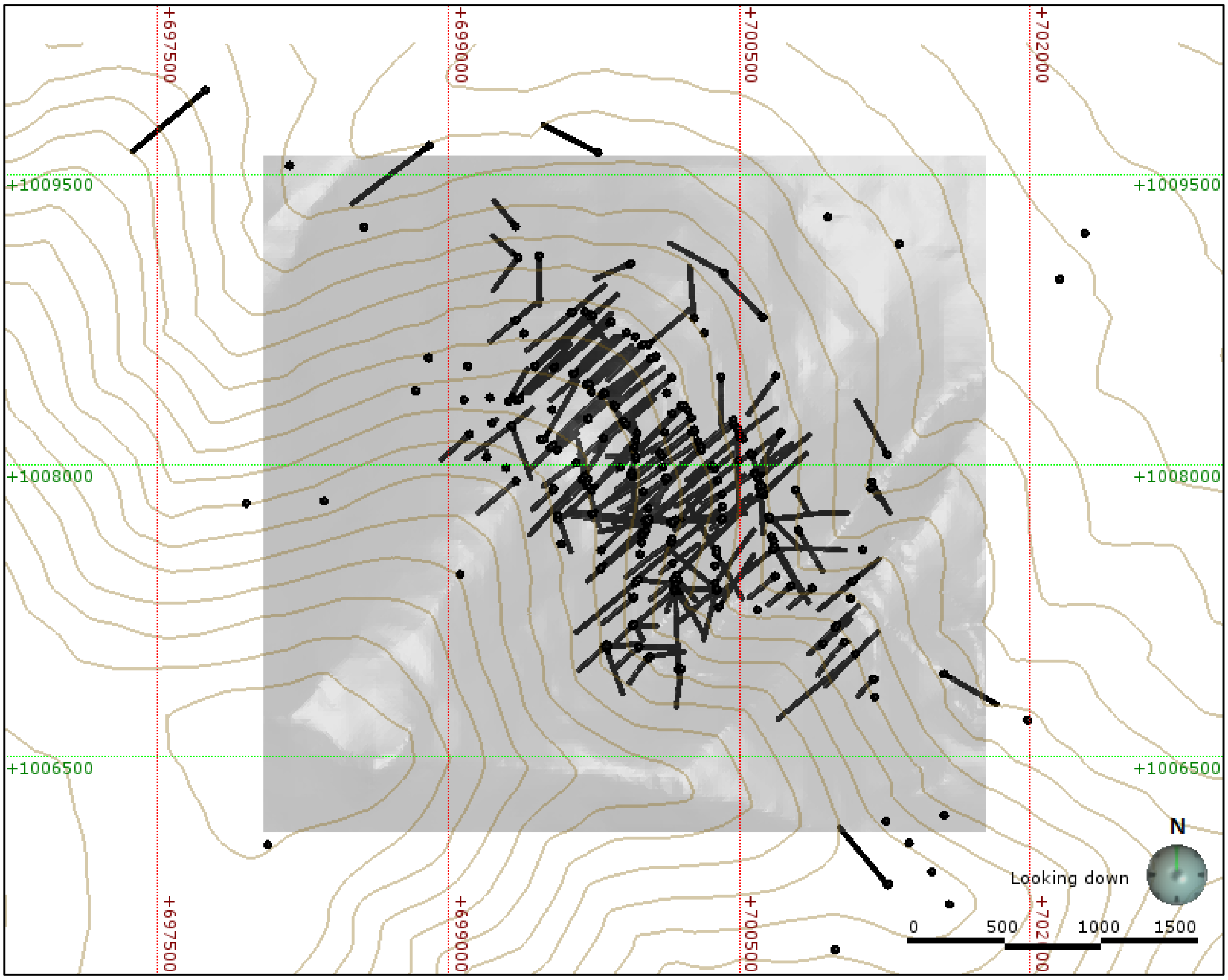
Otis Gold prepared the drill hole database for Kilgore and is determined to be of good quality. The drill hole data for the Kilgore Project was delivered as a Microsoft Access database that contains collar locations, drill hole survey orientations, sample intervals with gold assays in ppm, geologic intervals with rock types, alteration, and oxidation state, and specific gravity values. The collar locations are projected in State Plane Coordinate System, Zone Idaho East, NAD83 datum, with planar and elevation units in feet. All downhole intervals are captured in feet. Rowearth believes the drill hole assay data are sufficiently reliable to support the estimation of gold mineral resources.

The exploration database for Kilgore contains drill hole information from numerous companies that begin in 1984 and end in 2017. The cutoff date for drill hole assay data pertaining to this resource estimate is February 20, 2018, ending with drill hole 17OKC-379. The drill holes were validated during import with minor corrections made, and certain drill holes were excluded for use in the mineral resource estimation if they were judged to be of insufficient quality. The complete drill hole database delivered by Otis Gold contains 377 separate drill holes. After filtering out holes that were well outside the main Kilgore resource model and flagging six drill holes for exclusion, the resulting drill hole dataset contains 323 holes and is summarized in Table 14-1. Figure 14-1 shows the position of the holes relative to the geologic model boundary for the resource.

Table 14-1: Drill Hole Data with Assays for the Kilgore Project

Years	Company	Drill Holes	Samples	Interval Length (ft)	Percent of Total (ft)
1984-1985	Kennecott	6	1,341	6,720	3%
1990-1992	Placer Dome US	33	2,865	17,700	8%
1993	Pegasus Gold	19	1,637	8,245	4%
1994-1996	Echo Bay Mines	87	11,611	58,874	26%
2008-2017	Otis Gold	178	26,490	134,173	59%
	Grand Total	**323**	**43,944**	**225,711**	**100%**

Figure 14-1: Kilgore Drill Holes Used in Geologic Models and Resource Estimation

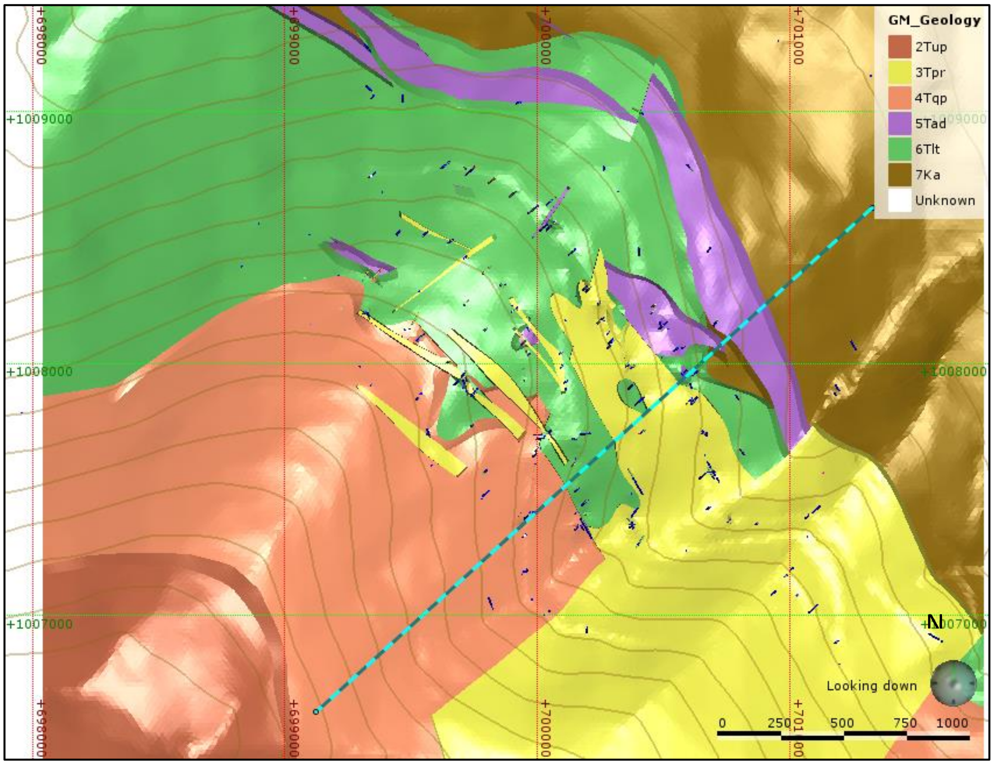


Gray rectangle outlines the boundary of the geologic and block models. Plan view.

14.4 Geologic Modelling

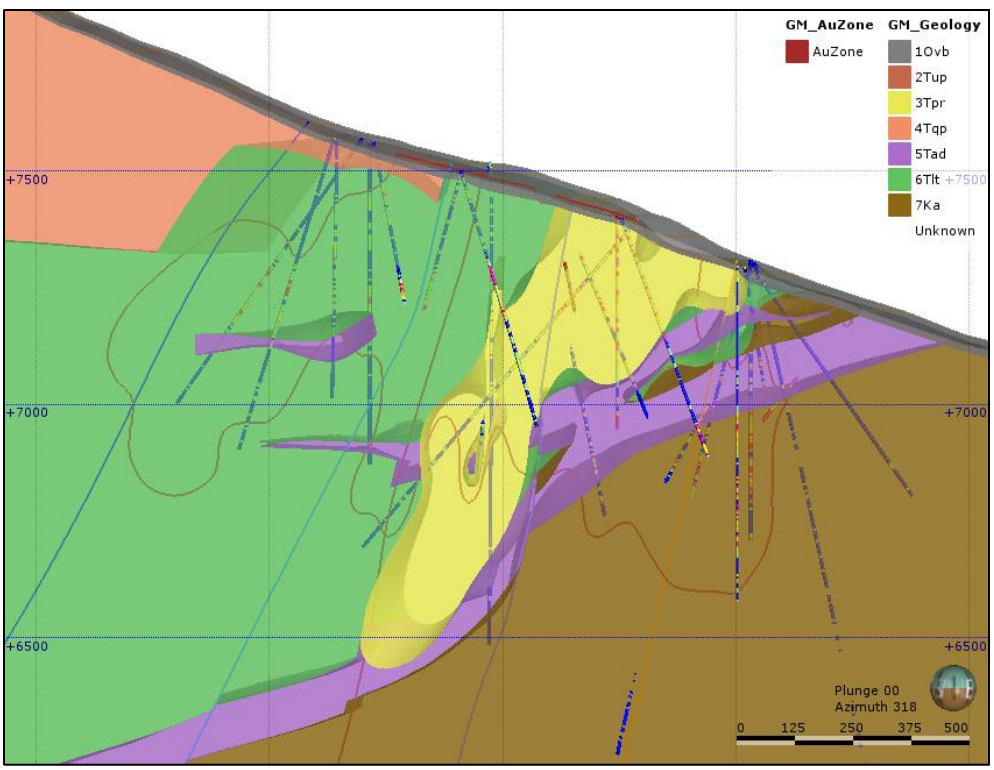
The resource estimation was constrained by a 3D geologic model consisting of multiple rock types, including the basal Aspen formation, volcaniclastic lithic tuff, granodioritic and granitic intrusive rocks, a minor involvement of overlying porphyritic rhyolite volcanic rocks, and a layer of overburden material. The principal gold mineralization is disseminated and structurally controlled mineralization with quartz veins, quartz vein stockwork, and fault related rock types. The gold mineralization and associated alteration are controlled by steeply dipping northwest trending fault zones and a cross-cutting northeast trending fault. The gold mineralization is also controlled by the host rock lithologies cut by the fault zones. The geologic rock type model is displayed in Figure 14-2 and Figure 14-3.

Figure 14-2: Geologic Model for the Kilgore Deposit, Plan View



Showing top of bedrock with overburden removed. North is up. Vertical section line shown. Aspen formation (Ka), lithic tuff (Tlt), andesitic sills (Tad), rhyolite flow (Tqp), rhyolite intrusive rock (Tpr).

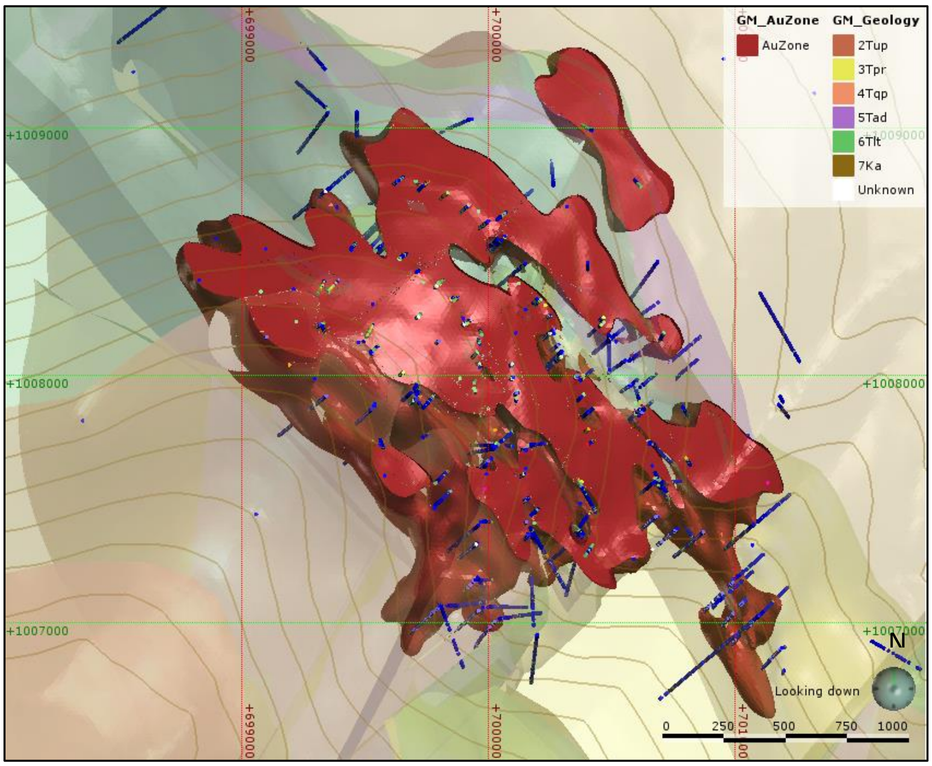
Figure 14-3: Geologic Model for the Kilgore Deposit, Vertical Section



Looking NW parallel the northwest fault system. Gold zone > 0.1 g/T Au outlined. Aspen formation (Ka), lithic tuff (Tlt), andesitic sills (Tad), rhyolite flow (Tqp), rhyolite intrusive rock (Tpr), overburden (Ovb). Steeply dipping fault zones in blue.

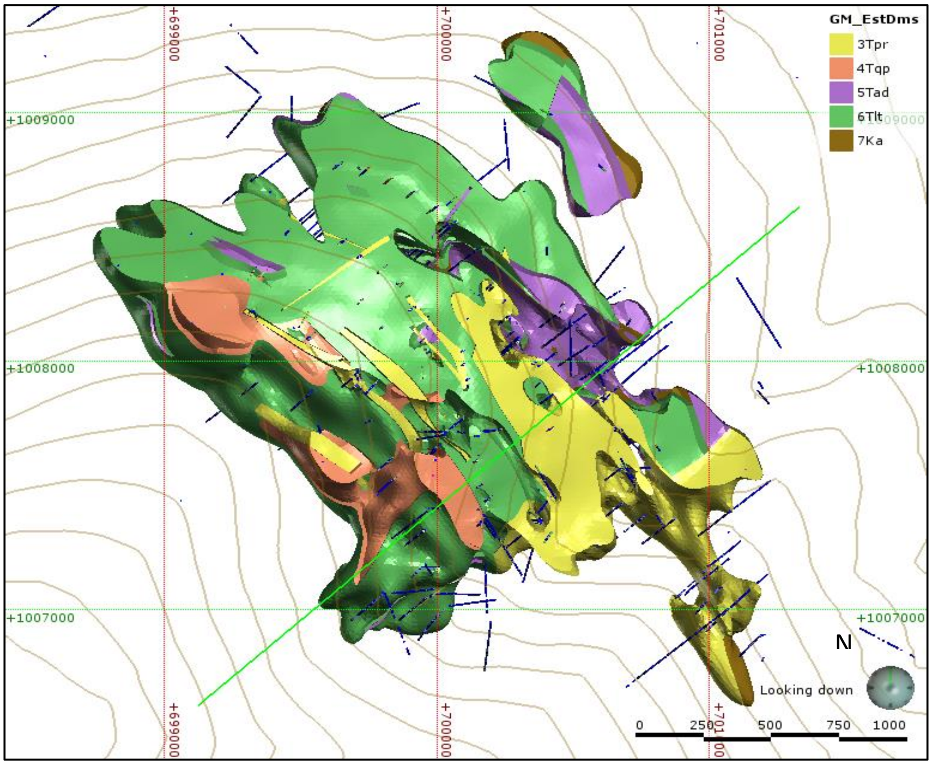
The Kilgore deposit is subdivided into five estimation domains based on host rock types and the modeled extents of the gold mineralization using a 0.1 g/T Au threshold. Modeling of the gold zone was controlled by the gold grade values from drilling while respecting the geologic and structural trends identified for the deposit, the northwest trending fault system. The gold zone boundary was treated as a hard boundary during block model estimation, and samples outside the boundary were excluded from the gold grade estimation. Rock type contacts are also treated as hard boundaries during the grade estimation. The estimation domains used in the resource evaluation are presented in Figure 14-4, Figure 14-5, and Figure 14-6.

Figure 14-4: Gold Zone for the Kilgore Deposit, Plan View



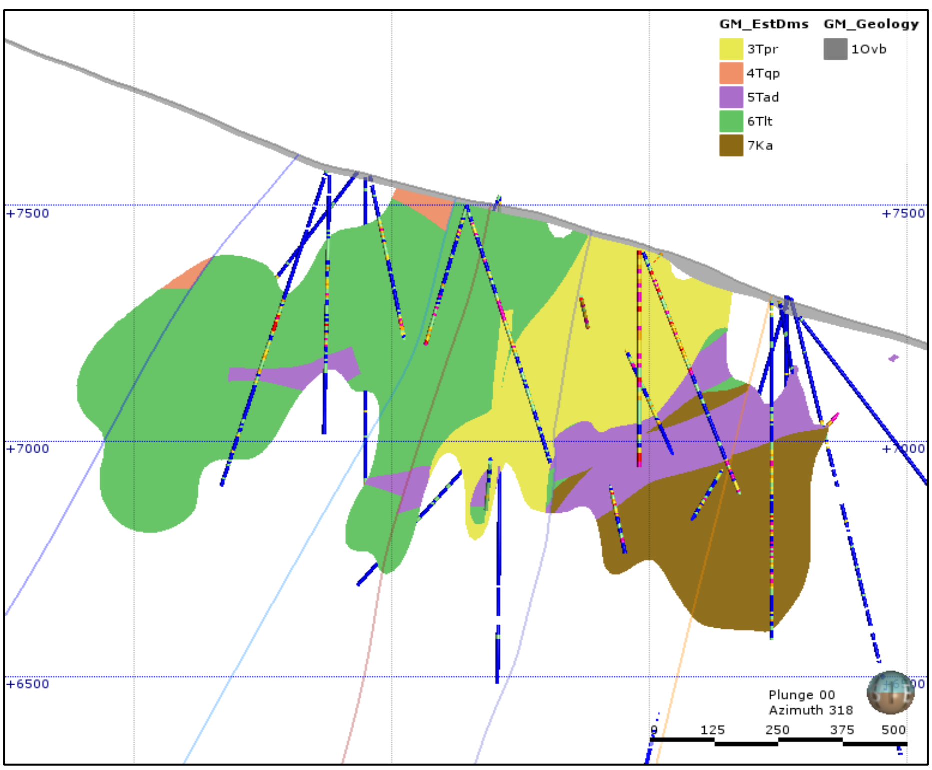
Showing top of bedrock with the 0.1 g/T Au zone shown in red. Fully projected, north is up.

Figure 14-5: Five estimation domains for the Kilgore Deposit, plan view



Rock types clipped by the limits of the 0.1 g/T Au zone. Fully projected, north is up. Vertical section line shown. Aspen formation (Ka), lithic tuff (Tlt), andesitic sills (Tad), rhyolite flow (Tqp), rhyolite intrusive rock (Tpr). Overburden removed

Figure 14-6: Five Estimation Domains for the Kilgore Deposit, Vertical Section



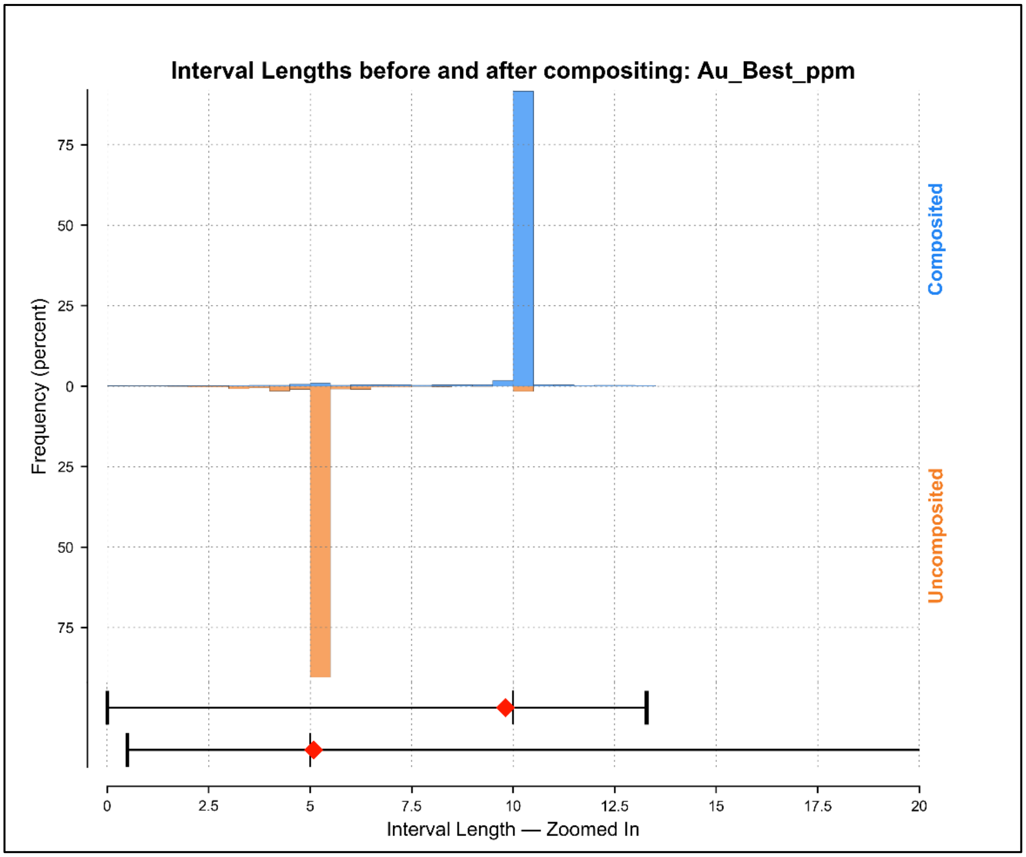
Looking NW. Rock types clipped by the limits of the 0.1 g/T Au zone. Steeply dipping fault zones in blue. Aspen formation (Ka), lithic tuff (Tlt), andesitic sills (Tad), rhyolite flow (Tqp), rhyolite intrusive rock (Tpr), overburden (Ovb)

14.5 Sample Compositing

Prior to constructing the composite samples for gold estimation, assay values were compared to corresponding sample interval lengths. Higher gold grades are not observed to correlate with sample lengths shorter than the most common sample length of 5 feet, and evaluation of outlier values was completed on the composite samples (Figure 14-7).

Ten-foot composite assay intervals were constructed from the original assay samples. The composite samples were constructed within the boundary limits for each of the five estimation domains and tagged with the associated rock code for each domain. Any residual end composite samples less than 3.3 feet in length were added to the previous composite sample interval.

Figure 14-7: Sample Interval Lengths, Uncomposited and Composited



14.6 Evaluation of Outliers

Very high-grade assay values can bias block grade estimates. Therefore, the drill hole composite samples were evaluated for extreme high-grade outliers and reduced or "capped" to values appropriate for the estimation. The capped values were identified from inflection points of cumulative probability plots, at the highest end of the grade distributions. Grades above these inflection points were selected for capping.

Capping of assay gold values for the Kilgore deposit was limited to a select few extreme values. To reduce bias from a larger set of high-grade gold samples, those outlier values are range restricted by a method known in Leapfrog EDGE as "clamping." In clamping, samples above a specified high-grade threshold value are used at full value out to a specified distance from the sample. Beyond the specified distance, the samples are reduced in value or "clamped" to a stated high-grade threshold value (Table 14-2).

To quantify the impact of sample value capping in combination with the clamping of other high-grade samples, the resource was evaluated using uncapped and unclamped sample grades. The process of capping and clamping of sample values reduced the total estimated metal content in the Kilgore deposit by 3%.

Table 14-2: Kilgore Gold Value Capping and High-Grade Clamping Values by Estimation Domain

Estimation Domain	Number Comps	Capping g/T Au	Number Capped	% Capped	Clamp g/T Au	Number Clamped	% Clamped
Tpr	2758	21	2	0.07%	12	9	0.3%
Tqp	71	0.60	1	1.41%	0.35	3	4.2%
Tad	2585	None	1	0.04%	9	8	0.3%
Tlt	4690	48	0	0.00%	14	13	0.3%
Ka	2040	None	0	0.00%	8	11	0.5%
ALL	**12144**		**4**	**0.03%**		**44**	**0.4%**

Ndat = number of samples, HG Threshold is the value above which samples are clamped

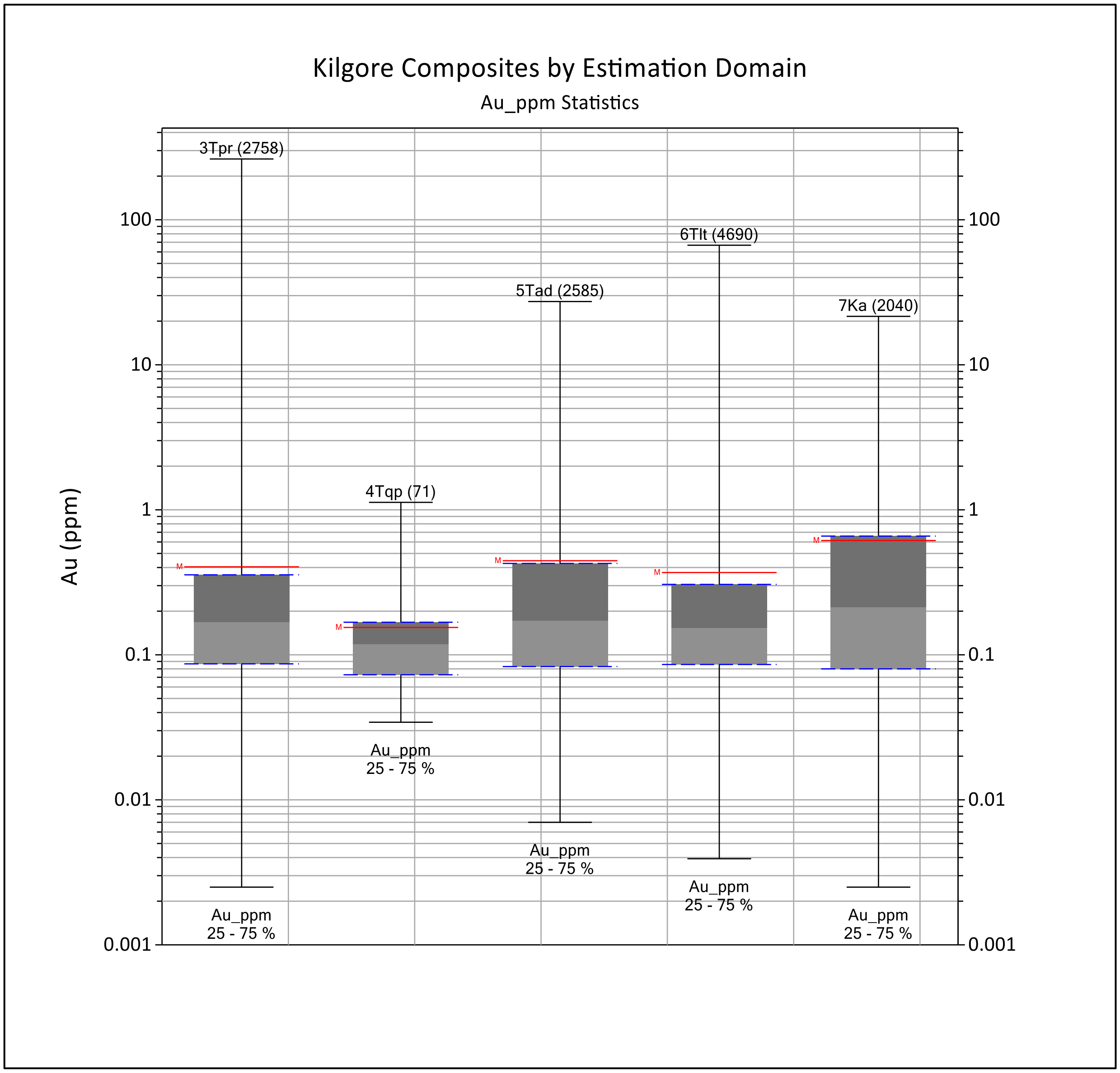
14.7 Statistical Analysis and Variography

For the following statistics and for the composite assay data used in the resource estimation; assay data listed as missing in the original database were omitted from the composite samples.

14.7.1 Statistics of Composited Data

To assess the global, unbiased characteristics of the gold composite samples within each of the Kilgore estimation domains, the data were declustered by a cell declustering method. The declustered composite data statistics reveal that gold grades observed vary according to the host rock estimation domain. The declustered statistics of composite samples for all estimation domains in the Kilgore deposit are presented in Figure 14-8.

Figure 14-8: Kilgore, Au Box-Plot and Composite Sample Declustered Statistics by Estimation Domain



Domain	3Tpr	4Tqp	5Tad	6Tlt	7Ka	Domain
Samples	2758	71	2585	4690	2040	Samples
Minimum	0.003	0.034	0.007	0.004	0.003	Minimum
Maximum	262.569	1.125	27.305	66.755	21.583	Maximum
Mean	0.404	0.154	0.446	0.369	0.614	Mean
CV	7.723	1.089	2.566	3.766	1.997	CV
Median	0.168	0.118	0.172	0.153	0.214	Median

14.7.2 Boundary Conditions

Composite data statistics establish that gold values observed within the estimation rock-type-based domains are variable, with higher mean grades found in the Aspen formation, 7Ka, and the andesitic sills, 5Tad. Inspection of drilling intersections across intrusive rock /lithic tuff contacts reveal that higher grades commonly occur in the lithic tuff (6Tlt) adjacent to the intrusive rocks. In general, metal grades across the domains change substantially at distances shorter than the average drill hole spacing. Accordingly, hard boundary conditions were applied during resource estimation between all five of the estimation domains developed for the resource. Hard boundary conditions restrict sample values to the domain in which they are located, and a domain's resource block grade estimates are not affected by samples outside the domain.

14.7.3 Variography

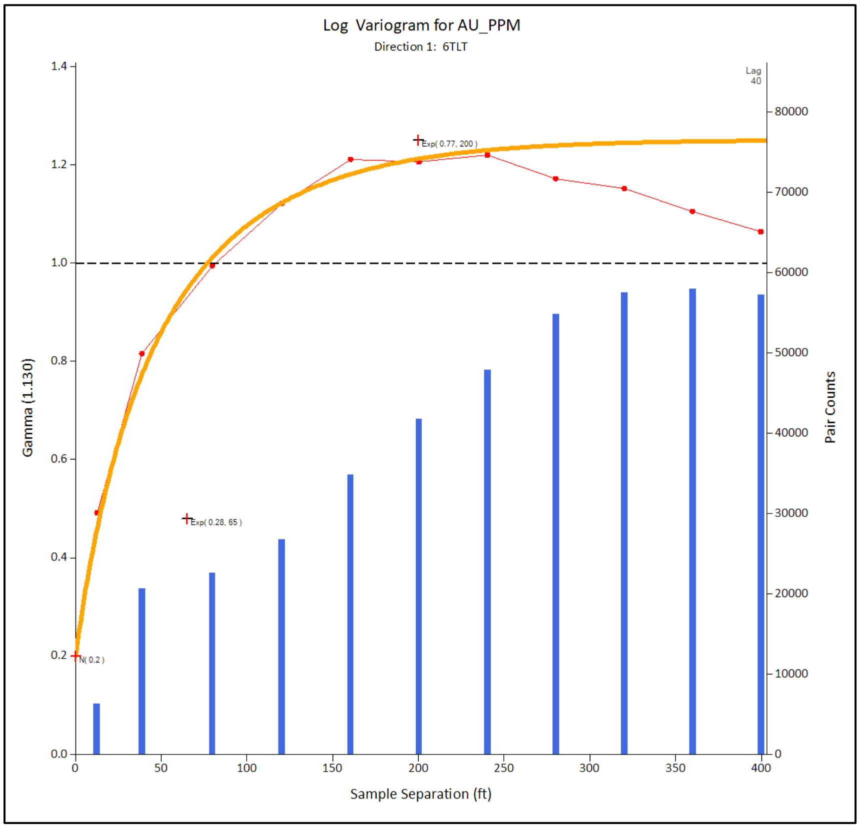
Variogram models were developed with Snowden Supervisor from the composite samples for gold within each estimation domain separately, and the nugget values were established from downhole variograms. The variogram parameters were transferred from Supervisor to Leapfrog Edge. The experimental variograms are log transformed to handle the strongly skewed data set. The variogram model parameters used for gold grade estimation are summarized in Table 14-3. An example of the gold variography for the 6Tlt domain (lithic tuff) is presented in Figure 14-9, and variogram plots for all domains are included in Appendix A.

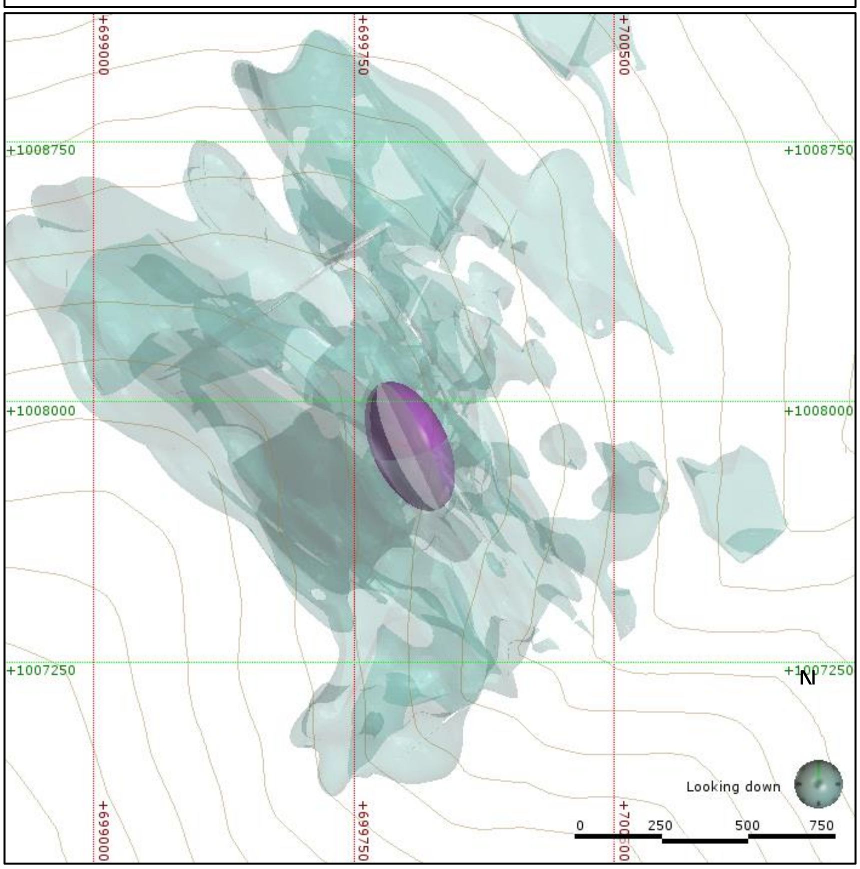
Table 14-3: Kilgore Variogram Models by Estimation Domain

Estimation Domain	Leapfrog Trend			Nugget C_0	Sill C_1 and C_2	Range (ft)	Model
	Dip	Dip Az	Pitch				
3Tpr	75	240	15	0.22	0.46	70	Spheroidal
					0.33	200	Spheroidal
4Tqp	75	245	100	0.20	0.28	65	Spheroidal
					0.79	200	Spheroidal
5Tad	75	255	90	0.25	0.16	70	Spheroidal
					0.6	210	Spheroidal
6Tlt	75	245	100	0.20	0.28	65	Spheroidal
					0.77	200	Spheroidal
7Ka	75	245	80	0.25	0.24	50	Spheroidal
					0.45	225	Spheroidal

Leapfrog trend orientation. Variography from the 6Tlt domain was applied to the 4Tqp domain

Figure 14-9: Variography for the 6Tlt Domain



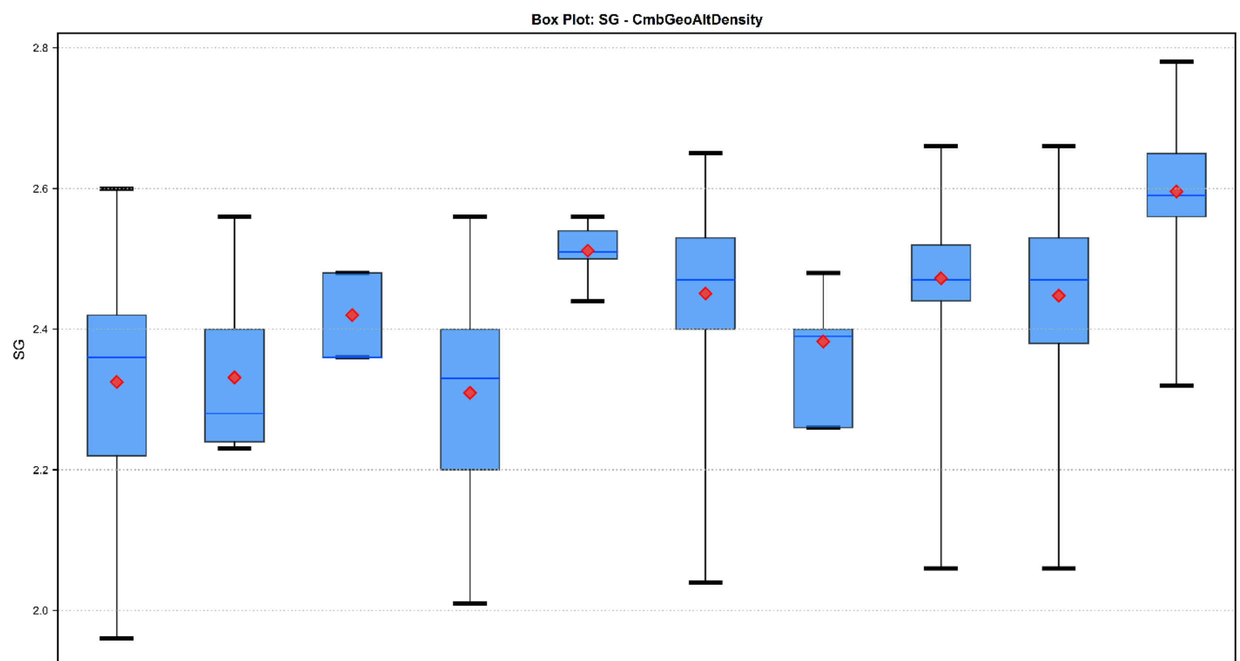


Top: major axis variogram model; Bottom: ellipsoid of variogram trend within Tlt domain

14.7.4 Specific Gravity

Otis Gold has measured specific gravity (SG) for 671 drill core samples from the Kilgore deposit at the time of this resource estimation. The SG sampling program was designed to collect representative specimens from all rock types present at Kilgore and from all alteration types impacting the host rock types. The SG values range from 1.96 to 2.78 across the deposit, with a mean value of 2.45, and it was observed that clay-altered zones tend to have the lowest mean SG values observed. Accordingly, the five host rock types at Kilgore were sub-divided by modeled clay alteration zones vs. all other alteration types. This resulted in 10 SG sub-domains that best outline the density variation at Kilgore. The SG statistics for the combined domains are displayed in Figure 14-10.

Figure 14-10: Statistics for Specific Gravity at Kilgore



	Clay, 3Tpr	Clay, 4Tqp	Clay, 5Tad	Clay, 6Tlt	Clay, 7Ka	Non_..3Tpr	Non_..4Tqp	Non_..5Tad	Non_..6Tlt	Non_.. 7Ka
Min	1.96	2.23	2.36	2.01	2.44	2.04	2.26	2.06	2.06	2.32
Q1	2.22	2.24	2.36	2.2	2.5	2.4	2.26	2.44	2.38	2.56
Median	2.36	2.28	2.36	2.33	2.51	2.47	2.39	2.47	2.47	2.59
Q3	2.42	2.4	2.48	2.4	2.54	2.53	2.4	2.52	2.53	2.65
Max	2.6	2.56	2.48	2.56	2.56	2.65	2.48	2.66	2.66	2.78
Mean	2.3248	2.3312	2.42	2.3095	2.5117	2.4507	2.3825	2.4723	2.4477	2.5958
SD	0.13234	0.11382	0.084853	0.13827	0.041191	0.11117	0.091059	0.079177	0.1155	0.07726
N	62	8	2	57	6	151	4	91	167	120

Box plot of SG values for 10 combined domains: rock type sub-divided by clay and non-clay alteration zones.

14.8 Resource Estimation Procedure

The block model resource estimate for the Kilgore deposit was completed using Leapfrog EDGE version 4.3. In Leapfrog EDGE, a sub-blocked model was created that consists of primary parent blocks that are sub-divided into smaller sub-blocks whenever triggering surfaces intersect the parent blocks. For the Kilgore sub-block model, the five estimation domain boundaries were used as triggers to produce sub-blocks. Gold grades are estimated for the primary parent blocks, and all categorical data such as estimation domains, oxidation, resource class, or optimized pits are evaluated into the sub-blocks. Optimal parent block size was identified by kriging neighborhood analysis and considers possible future selective mining units. The block model parameters for Kilgore are presented in Table 14-4.

Table 14-4: Kilgore Block Model Parameters

Description	X Axis	Y Axis	Z Axis
Block Model Origin (min X and Y, max Z)	698,060	1,006,120	8400
Size by number of Parent Blocks	187	173	226
Parent Block Dimension	20	20	10
Sub-block Dimension	10	10	10

Block model rotation is dip of 0° and rotation of 0°

The Kilgore block model resource estimate was completed for Au and was constrained by the five estimation domains, which are the Au zone subdivided by the host rock types. All block grades were estimated from 10-foot composite sample values captured within the respective domains.

Block grades were estimated by ordinary kriging using the variogram models observed for composite sample Au values within the estimation domains. Hard boundary conditions were applied during the block estimation for all estimation domains. An exception was made for small regions of the Tqp domain where a modified soft boundary was used that allow samples outside the domain if within five feet of the boundary.

The blocks were estimated with two successive interpolation passes for all Au. The shorter first pass was designed to interpolate gold grades for blocks that are well-informed by drill hole composite samples. The second pass was designed to estimate most of the remaining blocks within the geologic domains, including extrapolated estimates.

Within select estimation domains, extreme composite Au assay values were capped, and other samples above a specified high-grade threshold value were used at full value out to a specified range from the sample. Beyond the specified range, the samples are reduced in value or "clamped" to a specified high-grade threshold value.

Block specific gravity was estimated within ten estimation domains: the five host rock types subdivided by clay or non-clay alteration. Specific gravity was estimated using inverse distance cubed methodology (ID^3) and using hard boundary conditions between most domains. The blocks were estimated by a single search pass within each domain, and all blocks not estimated by the single pass were assigned the mean specific gravity value for the respective domain. A summary of estimation parameters is presented in Table 14-5 and Table 14-6.

Table 14-5: Summary of Au Estimation Parameters for the Kilgore Block Model Estimation

	Tpr		Tqp		Tad		Tlt		Ka	
Estimation Domain	**Pass 1**	**Pass 2**	**Pass 1**	**Pass 2**	**Pass 1**	**Pass 2**	**Pass 1**	**Pass 2**	**Pass 1**	**Pass 2**
Value Clipping Lower (g/T Au)										
Value Clipping Upper (g/T Au)	21	21	0.6	0.6			48	48		
Search Ellipsoid Orientation										
dip	75	75	75	75	75	75	75	75	75	75
dip-azimuth	240	240	245	245	255	255	245	245	245	245
pitch	15	15	100	100	90	90	100	100	80	80
Search Ellipsoid Lengths ft										
max	240	400	240	400	240	400	240	400	240	400
Interm.	240	400	240	400	240	400	240	400	240	400
min	120	200	120	200	120	200	120	200	120	200
Minimum Samples	6	5	6	5	6	5	6	5	6	5
Maximum Samples	17	17	17	17	17	17	17	17	17	17
Outlier Restriction										
Enabled (True or False)	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
Discard or Clamp	Clamp	Clamp	Clamp	Clamp	Clamp	Clamp	Clamp	Clamp	Clamp	Clamp
Range Restriction (% of search range)	25%	15%	25%	15%	30%	18%	33%	20%	42%	25%
High-grade threshold (g/T Au)	12	12	0.35	0.35	9	9	14	14	8	8
Octant Search										
Enabled (True or False)	TRUE	FALSE	TRUE	FALSE	TRUE	FALSE	TRUE	FALSE	TRUE	FALSE
Max samples per octant	5		5		5		5		5	
max number of empty octants	5		5		5		5		5	
Drill hole Limit										
Enabled (True or False)	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE	TRUE
max samples per drill hole	5	5	5	5	5	5	5	5	5	5

Table 14-6: Summary of SG Estimation Parameters for the Kilgore Block Model Estimation

	Clay, Tpr	Clay, Tqp	Clay, Tad	Clay, Tlt	Clay, Ka	Tpr	Tqp	Tad	Tlt	Ka
SG Estimation Domain	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**	**Pass 1**
Value Clipping Lower (SG) Value Clipping Upper (SG)										
Search Ellipsoid Orientation										
dip	77	75	75	75	75	75	75	75	75	75
dip-azimuth	233	240	240	240	240	240	240	240	240	240
pitch	61	15	15	15	15	15	15	15	15	15
Search Ellipsoid Lengths ft										
max	240	240	240	240	240	240	240	240	240	240
interm	200	200	200	200	200	200	200	200	200	200
min	140	140	140	140	140	140	140	160	140	160
Minimum Samples	2	2	1	1	1	2	2	2	2	2
Maximum Samples	6	6	6	6	6	6	6	6	6	6
Outlier Restriction Enabled (True or False)	False	False	False	False	False	False	False	False	False	False
Octant Search Enabled (True or False)	False	False	False	False	False	False	False	False	False	False
Drill hole Limit Enabled (True or False)	False	False	False	False	False	False	False	False	False	False

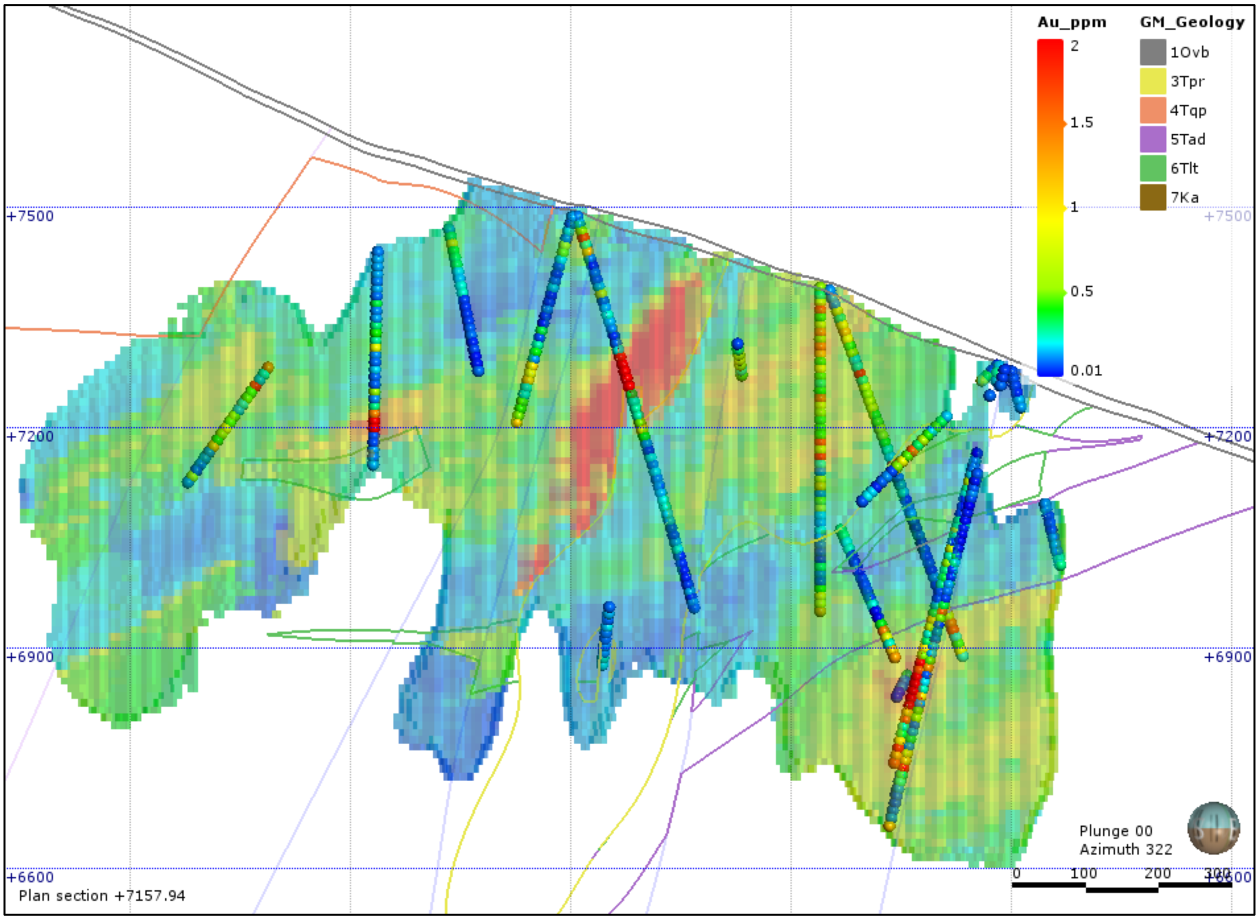
14.9 Block Model Validation

Validation of the estimated block grades for the Kilgore deposit was completed for each of the geologic domains. The resource block model estimate was validated by:

- Completing a series of visual inspections by comparisons of composite sample grades to estimated block values across the deposit. This was done for gold and SG.
- Comparison of “well informed” block gold grades with the average of composite sample values contained within those blocks using both scatter and cumulative probability plots.
- Comparing average composite sample values with average estimated block grades along east, north, and elevation orientations - Swath grade trend plots.

Estimated Au and SG block grades were visually inspected in a series of detailed vertical sections across the deposit. This review confirmed that the supporting composite sample grades closely match the estimated block values. Figure 14-11 displays a representative vertical section of the estimated gold block grades and the composite sample values used in the estimation.

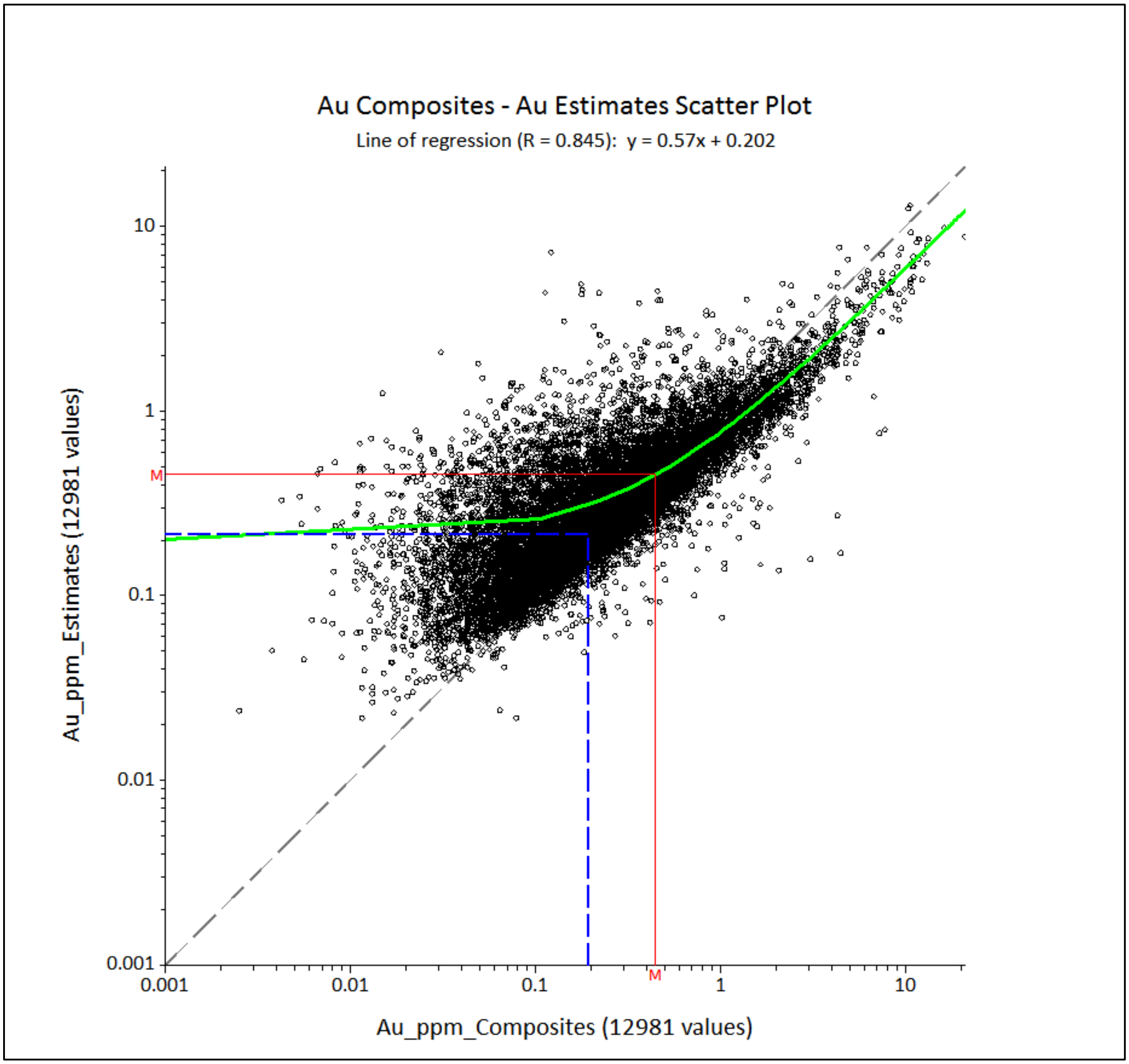
Figure 14-11: Kilgore Block Model and Composite Sample Gold Values, Vertical Section



Vertical section looking NW. 50 ft thick displaying estimated gold block grades and composite gold values in g/T. Geologic contacts shown.

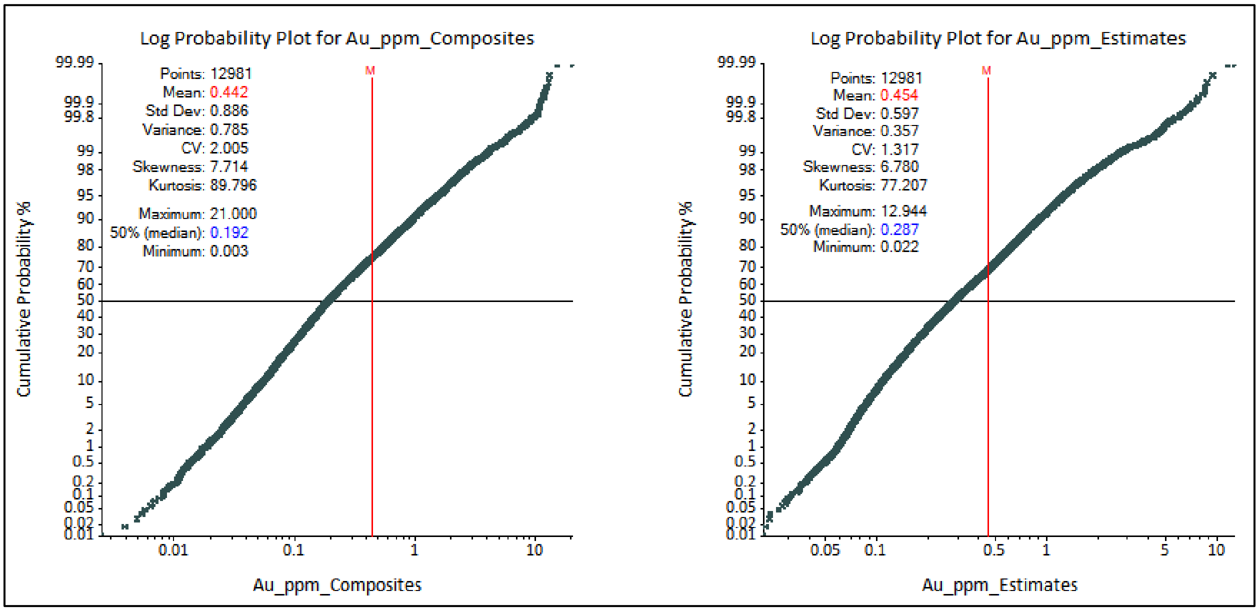
Figure 14-12 and Figure 14-13 compare estimated block grade distribution with drill hole composite sample grades for Au using the average of composite samples within the blocks. Well-informed parent blocks are selected that have composite samples within 14 feet of the block centroid. The scatter plot demonstrates that the estimated block grades correlate well to the composite sample value mean, with scatter around the x = y line. This indicates that the estimated block grades are quite variable and not over-smoothed. The probability plots reveal similar sample distribution between estimated block and composite sample values, and the mean estimated block grades and composite sample averages for the blocks are nearly identical.

Figure 14-12: Scatter Plot Comparison of Gold Composites with Estimated Block Values



Parent blocks selected with composite samples within 14 feet of block centroid. Composite grades developed as the average of samples within a 5-foot distance tolerance of the distance to the closest point.

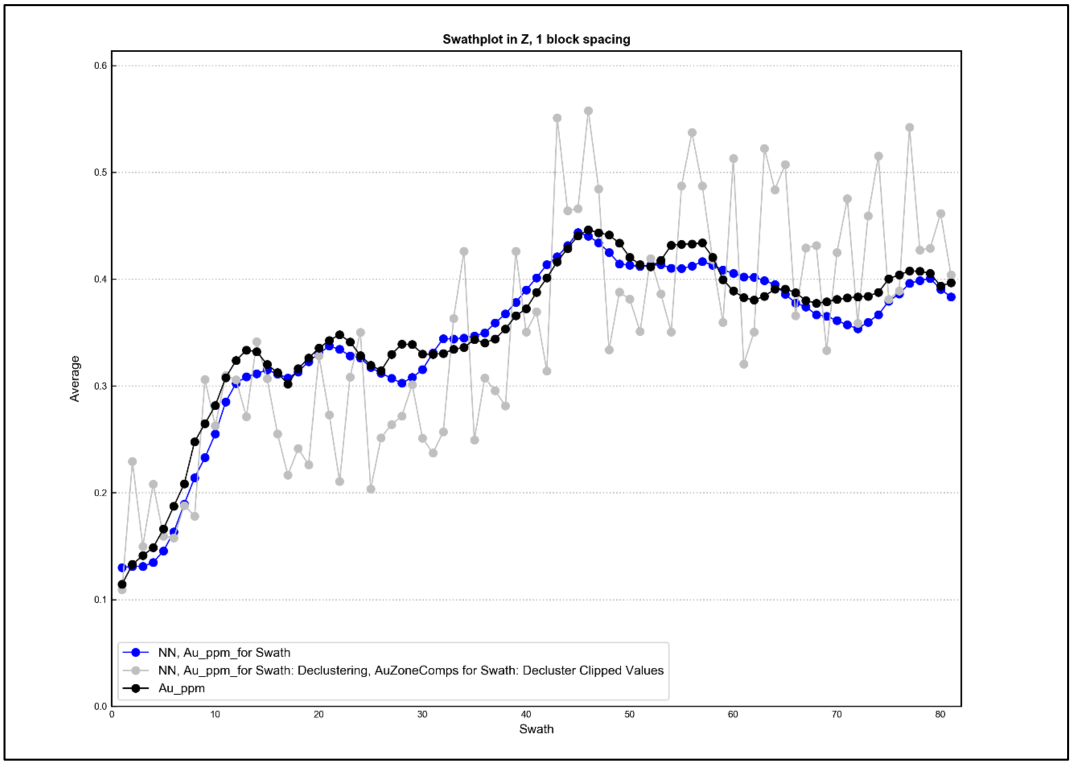
Figure 14-13: Probability Plot Comparison of Gold Composites with Estimated Block Values



Parent blocks selected with composite samples within 14 feet of block centroid. Composite grades developed as the average of samples within a 5-foot distance tolerance of the distance to the closest point.

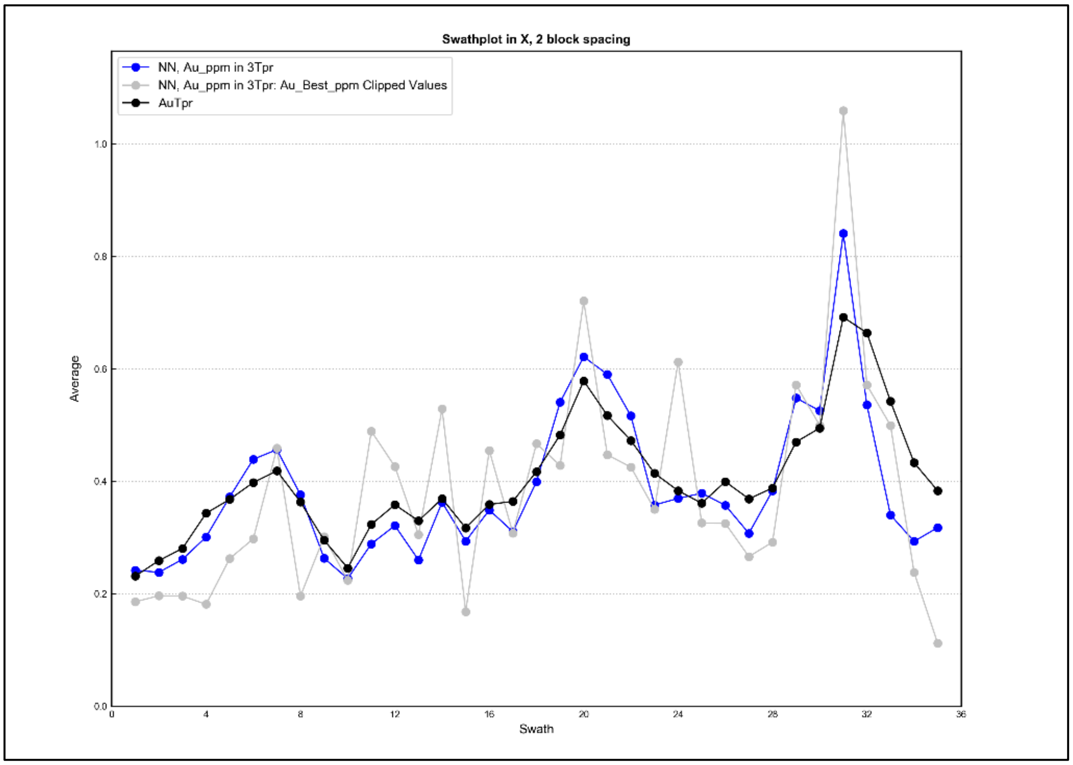
The block estimates were further validated by comparing the estimated block gold grades to nearest neighbor block estimates and to the de-clustered composite sample data within a series of slices through the Kilgore deposit (swath plots). The slices are in the X and Z (Elevation) directions. The swath plots created for Au across the entire Au zone and within four important estimation domains are shown in Figure 14-14 to Figure 14-18. The estimated block grades, the nearest neighbor block grades, and the composite sample values are similar in all directions for all estimation domains. Overall, the validation shows that current resource grade estimates are a good representation of drill hole assay data.

Figure 14-14: Swath Plot Across the Kilgore Au Deposit



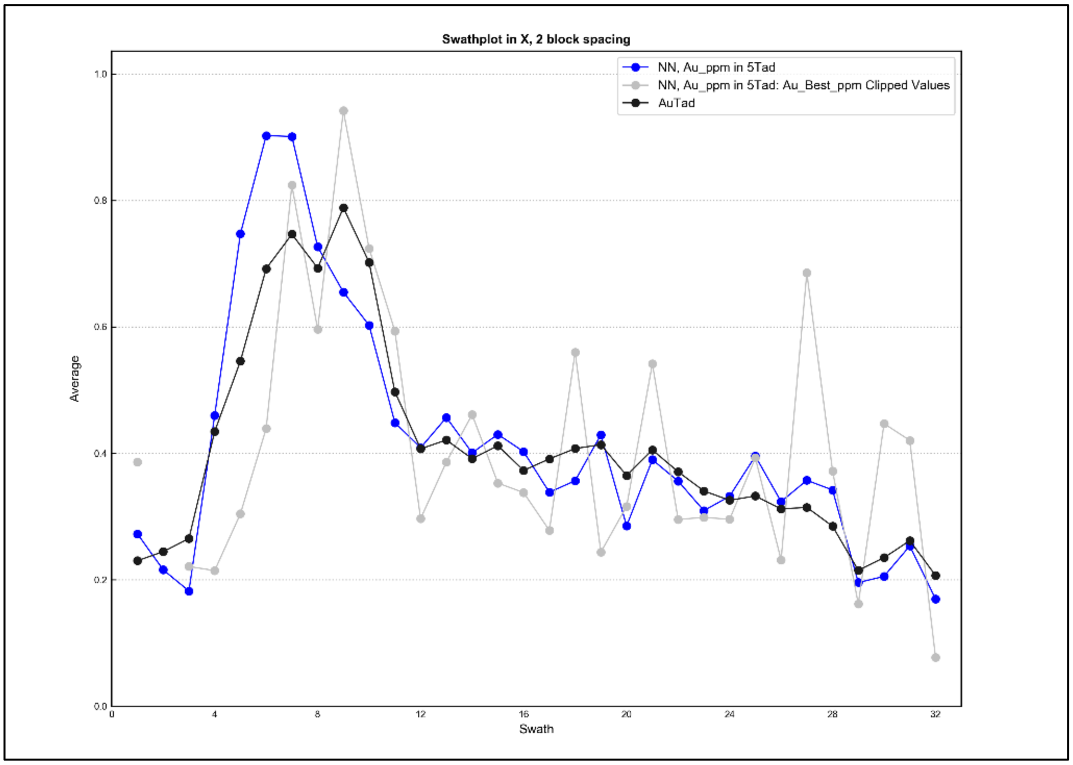
Swath plot of gold values (ppm) comparing declustered composite sample grades (gray) to block estimates (black) and NN block estimates (blue). Swaths along Z.

Figure 14-15: Swath Plot of the Tpr Domain



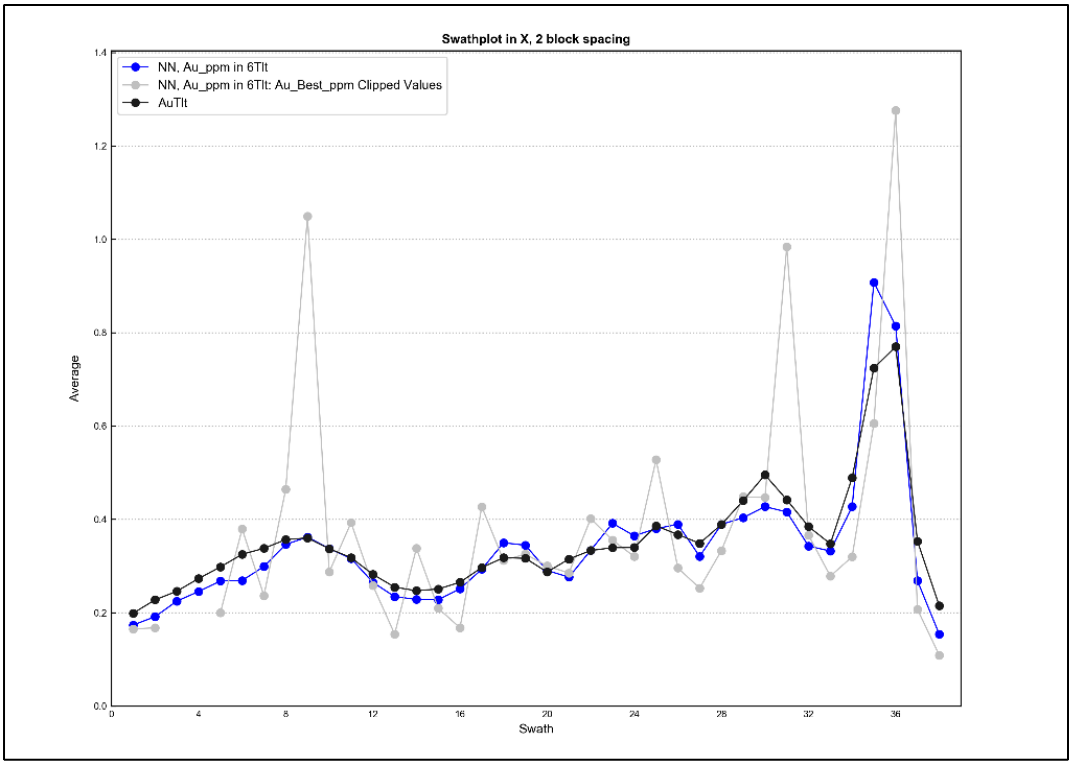
Swath plot of gold values (ppm) comparing clipped composite sample grades (gray) to block estimates (black) and NN block estimates (blue). Swaths along X

Figure 14-16: Swath Plot of the Tad Domain



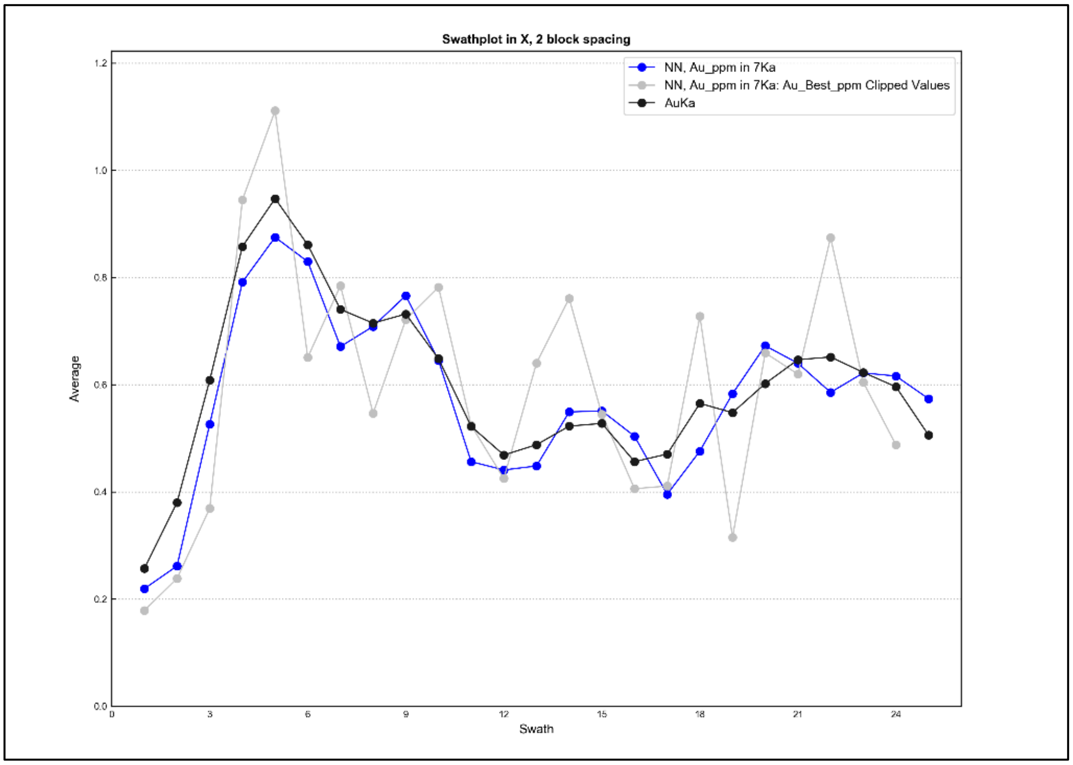
Swath plot of gold values (ppm) comparing clipped composite sample grades (gray) to block estimates (black) and NN block estimates (blue). Swaths along X

Figure 14-17: Swath Plot of the Tlt Domain



Swath plot of gold values (ppm) comparing clipped composite sample grades (gray) to block estimates (black) and NN block estimates (blue). Swaths along X

Figure 14-18: Swath Plot of the Ka Domain



Swath plot of gold values (ppm) comparing clipped composite sample grades (gray) to block estimates (black) and NN block estimates (blue). Swaths along X

14.10 Mineral Resource Classification

Block model quantities and grade estimates for the Kilgore deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines.

Generally, most of the factors influencing the mineral resource classification at Kilgore are positive. Rowearth is satisfied that the geologic modeling for the deposit honors the current geologic information and knowledge available. The location of the samples and the assay data are sufficiently reliable to support resource evaluation.

Mineral resources are classified as Measured, Indicated, or Inferred. To classify mineralization as a Measured mineral resource, "the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit" (CIM, 2014). No blocks were classified as a Measured mineral resource at Kilgore.

To classify mineralization as an Indicated Mineral Resource, "the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization" (CIM, 2014).

Estimated blocks were classified as either Indicated or Inferred according to:

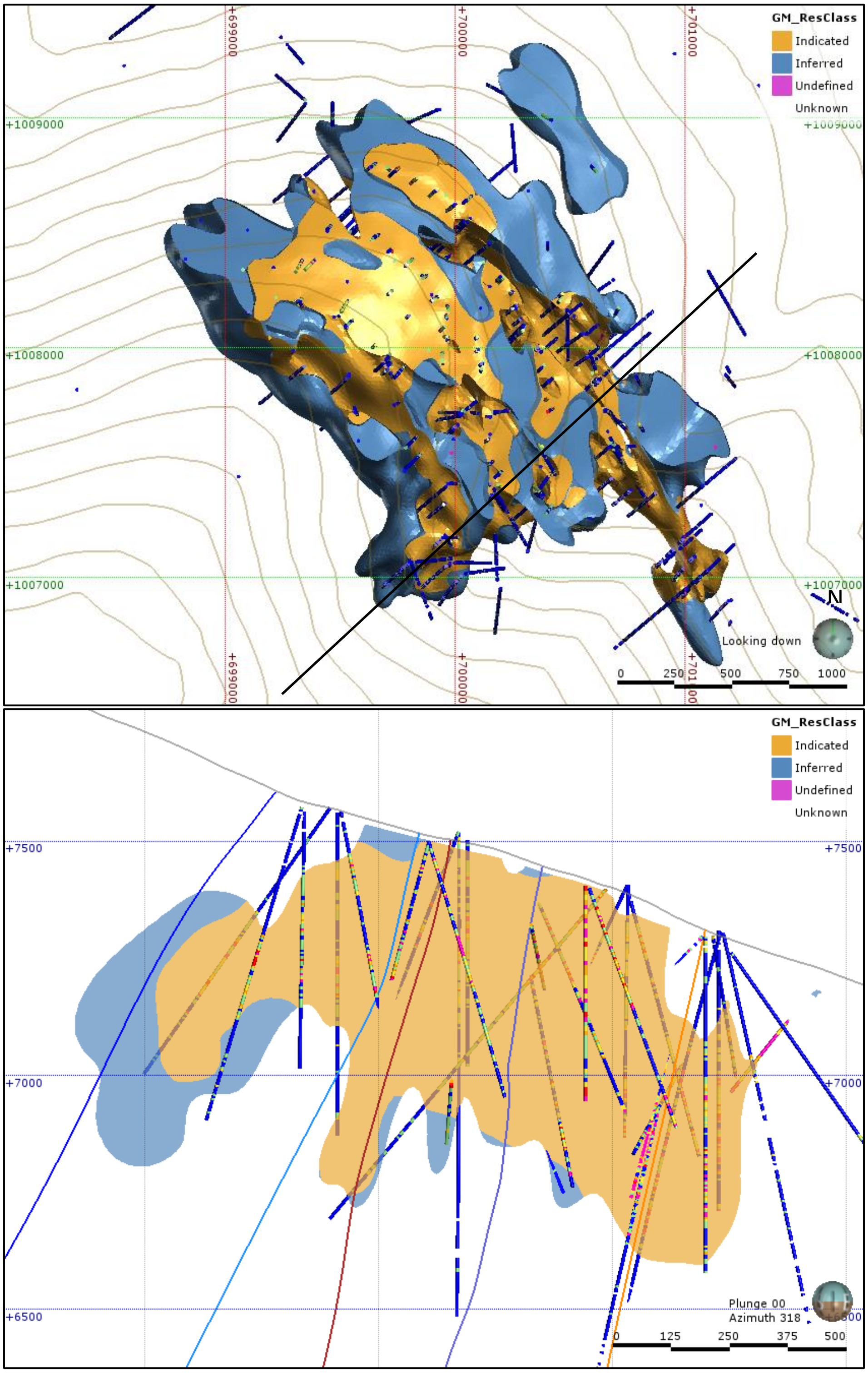
- Confidence in interpretation of the mineralized zones.
- Continuity of gold grades defined from variogram models.
- Number of samples used to estimate a block.
- Number of drill holes used to estimate a block.
- Number of Octants required to estimate a block.

To identify blocks to consider for Indicated classification:

1. Blocks were flagged by a classification search pass that required:
   a. Au composite values for each estimation domain.
   b. A 130 ft x 130 ft x 100 ft search volume obtained from the Au variography.
   c. At least 6 samples.
   d. At least 2 Drill Holes.
   e. At least 3 Octants. Octants demonstrate that minimal spatial support exists from drilling to allow regions into the Indicated Class.
2. Final broad areas of flagged blocks were outlined by constructing a classification wireframe designed to encompass zones predominantly flagged by the search pass used. This process allows review of the geologic confidence on the deposit along with drill hole support and expands certain areas but excludes others from Indicated. The number of blocks flagged for Indicated class was increased by the wire-framing process.
3. Blocks were finally selected as Indicated if the centroid of the block falls inside the classification wireframe.

For blocks classified as Inferred, the confidence in the estimate was insufficient to allow for the meaningful application of technical and economic parameters or to enable an evaluation of economic viability. All estimated blocks not assigned to the Indicated class are classified as Inferred. Figure 14-19 displays the distribution of Indicated and Inferred resources at Kilgore.

Figure 14-19: Indicated and Inferred Resources Classified at Kilgore with Composite Samples



Top: Plan view fully projected. Bottom: Vertical section across the resource looking NW, 130 ft thick projection

14.11 Pit Constrained Mineral Resource

CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) defines a mineral resource as: "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling." The mineral resources may be impacted by further infill and exploration drilling that may result in increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic, and other factors. Mineral resources are not mineral reserves and do not have demonstrated economic viability. Mineral reserves can only be estimated based on the results of an economic evaluation as part of a Preliminary Feasibility Study or Feasibility Study. As a result, no mineral reserves have been estimated as part of this study. There is no certainty that all or any part of the mineral resources will be converted into a mineral reserve.

The requirement, "reasonable prospects for eventual economic extraction," generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at a cutoff grade considering appropriate extraction scenarios and processing recoveries. To meet this requirement, Rowearth considered that major portions of the Kilgore deposit are amenable for open pit extraction.

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by an open pit, GRE constructed open pit scenarios developed from the resource block model estimate using Vulcan's Lerchs-Grossman miner "Pit Optimizer" software. Reasonable mining assumptions were applied to evaluate the portions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit. The optimization parameters presented in Table 14-7 were selected based on experience and benchmarking against similar projects. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cutoff grade. Rowearth considers that the blocks located within the resulting conceptual pit envelope show "reasonable prospects for economic extraction" and can be reported as a mineral resource.

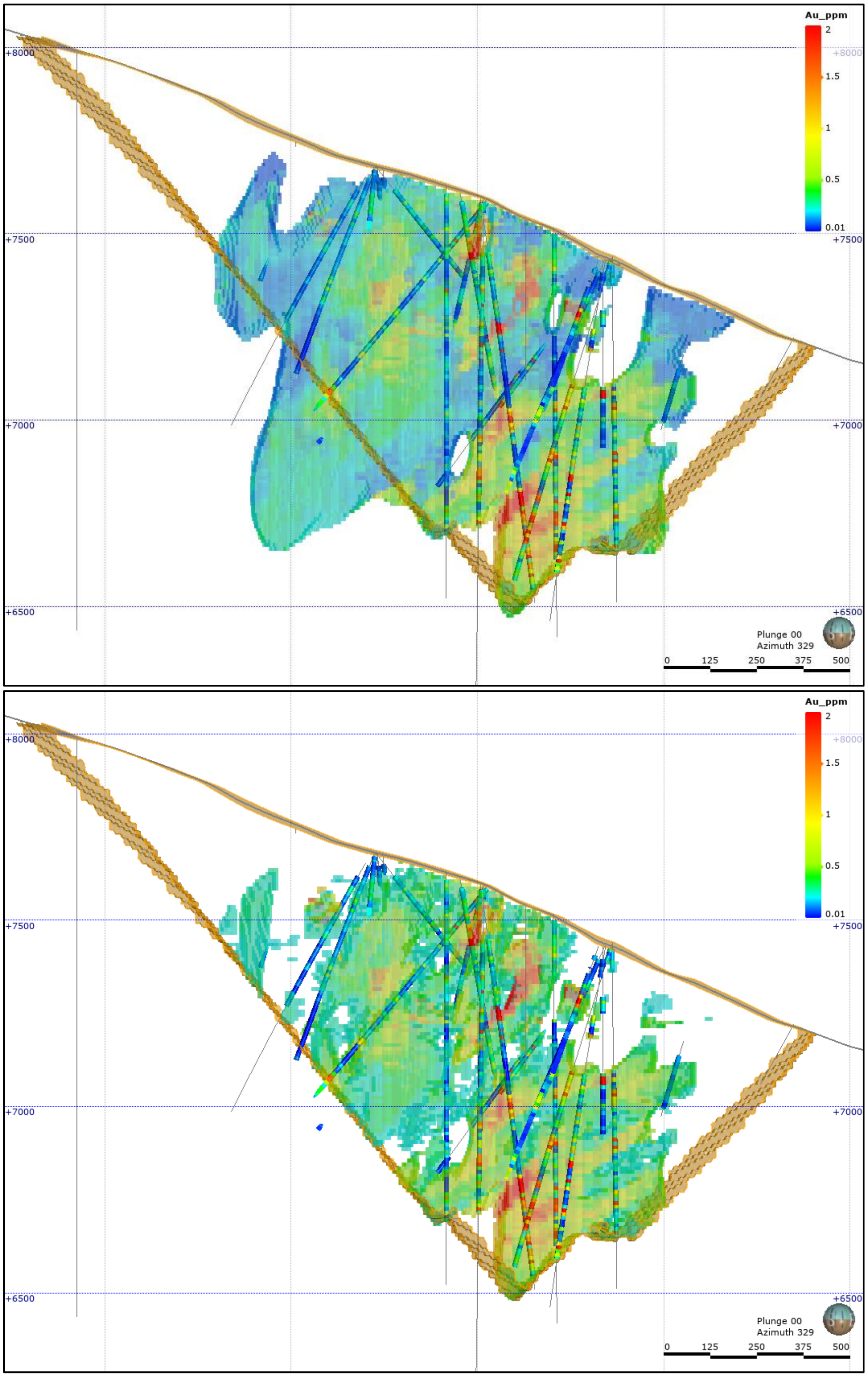
Table 14-7: Kilgore Resource Parameters for Conceptual Open Pit Optimization

Parameter	Unit	Values
Metal Price	US$/oz gold	$1,300.00
Selling cost	US$/oz gold	$2.20
Gold Recovery	%	80.00%
Mining cost	US$/short ton	$2.00
Process cost	US$/short ton includes $1.00 G&A	$4.00
Pit slope	degrees	50

The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for eventual economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are presently no mineral reserves on the project.

The Kilgore mineral resource within the pit is shown in Figure 14-20, and the mineral resources for the Kilgore deposit are reported in Table 14-8.

Figure 14-20: Representative Vertical Section Displaying Au Block Model Resource Looking NW



Top: Au block model estimate with pit shell and gold composite sample values.
Bottom: Reported gold mineral resources above 0.21 Au g/T cutoff grade within the optimized pit shell.

Table 14-8: Mineral Resource Statement for the Kilgore Deposit

Category	Cutoff (Au opt)	Short tons	Au Grade (opt)	Cutoff (Au g/T)	Metric Tonnes	Au Grade (g/T)	Au Ounces
Indicated	0.006	49,106,000	0.017	0.21	44,556,000	0.58	825,000
Inferred	0.006	10,354,700	0.013	0.21	9,399,000	0.45	136,000

Mineral resources have been classified in accordance with the CIM Definition Standards on Mineral Resources
Gold resources are reported above a 0.21 g/T Au cutoff
Mineral resources reported here are constrained within an optimized pit shell.
Pit shell input parameters: Gold price $1,300, Selling price $2.20/oz, Recovery 80%, Mining cost $2/ton, Process cost + G&A $4/ton, Pit slope 50°

14.12 Grade Sensitivity to Gold Cutoff

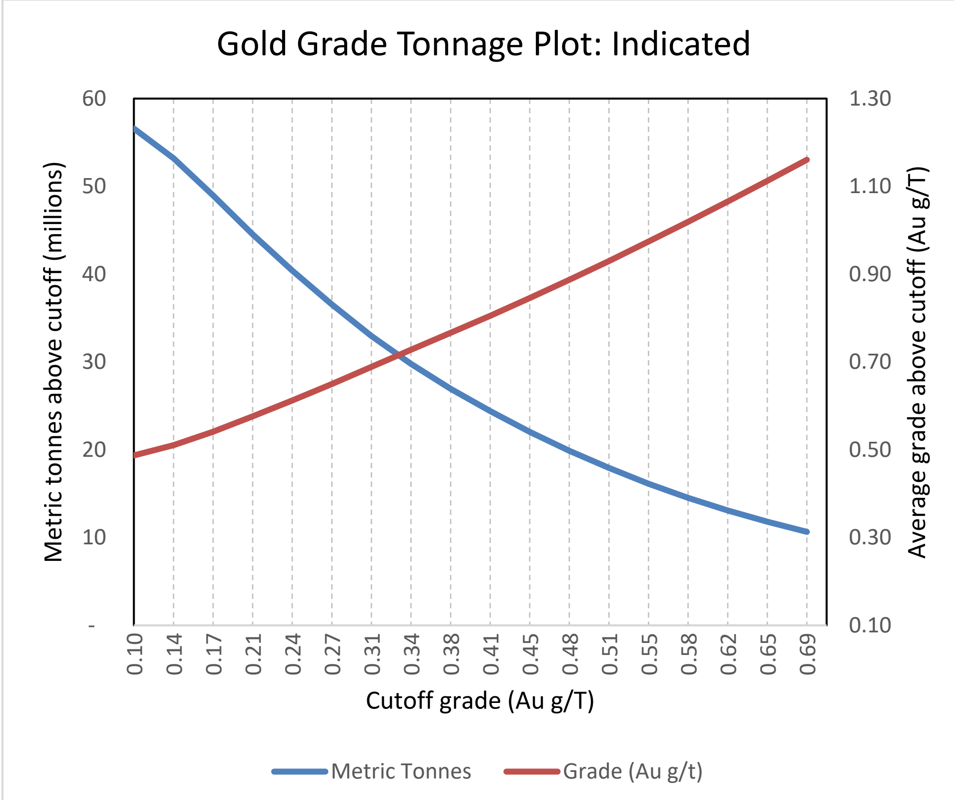
The mineral resources reported for the Kilgore deposit are sensitive to the selection of the reporting gold cutoff grade. To illustrate this sensitivity, the block model gold quantities and grade estimates are presented at different cutoff grades within the conceptual pit used to constrain the mineral resources (Table 14-9). The sensitivity to gold cutoff grade is also presented as grade tonnage curves for the Kilgore deposit (Figure 14-21). The reader is cautioned that the information presented in the table should not be misconstrued as a Mineral Resource Statement.

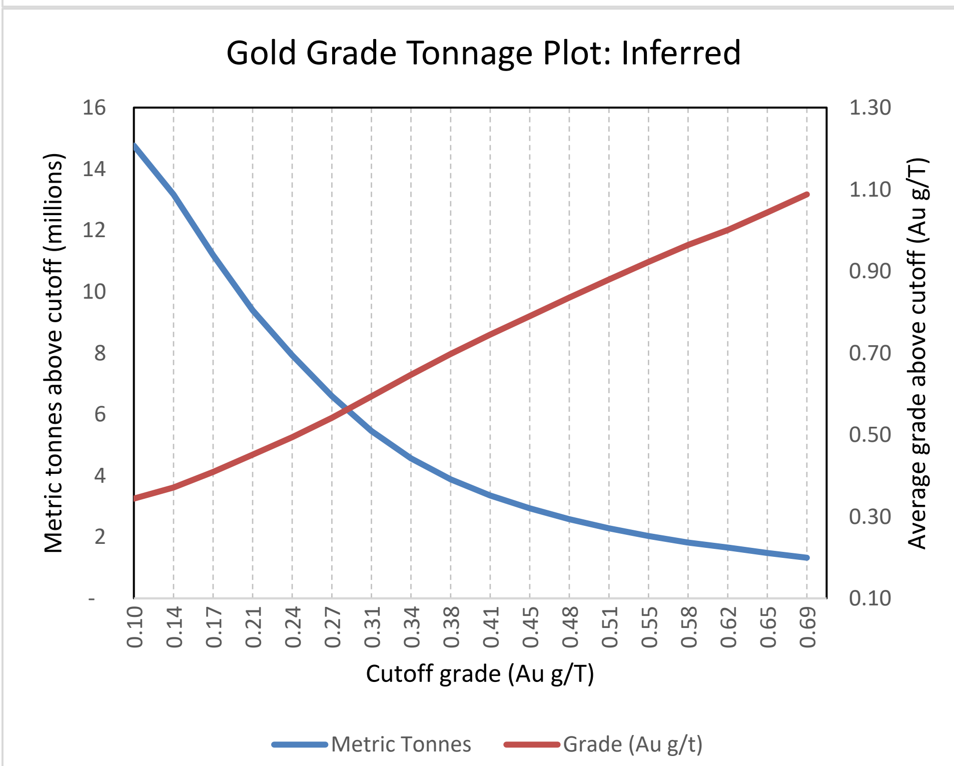
Table 14-9: Mineral Resource Sensitivity

Classification	Cutoff (Au opt)	Short tons	Au Grade (opt)	Cutoff (Au g/T)	Metric Tonnes	Au Grade (g/T)	Au Ounces
Indicated	0.003	62,382,000	0.014	0.10	56,592,000	0.49	886,000
	0.004	58,647,000	0.015	0.14	53,206,000	0.51	873,000
	0.005	53,976,000	0.016	0.17	48,966,000	0.54	852,000
	0.006	**49,106,000**	**0.017**	**0.21**	**44,556,000**	**0.58**	**825,000**
	0.007	44,549,000	0.018	0.24	40,414,000	0.61	796,000
	0.008	40,294,000	0.019	0.27	36,559,000	0.64	764,000
	0.009	36,343,000	0.020	0.31	32,970,000	0.69	730,000
	0.010	32,830,000	0.021	0.34	29,786,000	0.73	697,000
Inferred	0.003	16,271,700	0.010	0.10	14,761,000	0.34	163,000
	0.004	14,511,400	0.011	0.14	13,168,000	0.37	157,000
	0.005	12,336,900	0.012	0.17	11,192,000	0.41	147,000
	0.006	**10,354,700**	**0.013**	**0.21**	**9,399,000**	**0.45**	**136,000**
	0.007	8,736,180	0.014	0.24	7,925,000	0.49	126,000
	0.008	7,272,060	0.016	0.27	6,600,000	0.54	115,000
	0.009	6,017,710	0.017	0.31	5,459,000	0.59	104,000
	0.010	5,030,820	0.019	0.34	4,567,000	0.65	95,000

Au block model metal quantities reported at various Au cutoff grades for the Kilgore deposit.

Figure 14-21: Kilgore Deposit Grade-Tonnage Curves for Au Reported Above a Cutoff Grade





Rowearth is of the understanding that Otis Gold Corp is unaware of any factors that may potentially affect the resource estimate reported here.

15.0 MINERAL RESERVE ESTIMATE

There are no mineral reserve estimates for the Kilgore Project.

16.0 MINING METHODS

Conventional open pit mining methods using drill, blast, load, and haul mining are applicable to the Kilgore deposit. Mine scheduling and optimization were conducted as described in the following subsections.

16.1 Pit Size Options

As stated in Section 14.11, GRE generated pit shells using Vulcan's Lerchs-Grossman miner "Pit Optimizer" software using reasonable mining assumptions as identified in Table 14-7.

Initially, 21 pit shells were generated using gold prices ranging from $500/Au ounce to $1,500/Au ounce, in increments of $50/Au ounce. These pit shells included the Aspen rock type (which includes the Aspen top and bottom, but not the sill, which is categorized differently in the block model) as a potential source of leachable material. After review of the pit shell data and cumulative cash flow generated by the pit shells, GRE focused on the $800/Au ounce and $900/Au ounce pit shells as they captured nearly all of the cash flow and eliminated high strip, low profit tonnage.

Subsequently, because of the low recoveries reported for the Aspen top and bottom rock types (see Section 13.3), GRE generated two additional pit shells with the assumption that all Aspen top and bottom rock type material was waste. These two subsequent pit shells were also generated at gold prices of $800/Au ounce and $900/Au ounce.

The four ultimate pit designs were created for the following Lerchs-Grossman pit shells:

- $800/oz Au no Aspen included in pit
- $800/oz Au Aspen included in pit
- $900/oz Au no Aspen included in pit
- $900/oz Au Aspen included in pit

The four pit shells were imported into Geovia GEMS software, where the pits were designed with haul roads. Two intermediate size pits were also designed to provide pit phasing. Figure 16-1 shows the four ultimate pits, and Figure 16-2 shows the phase 1 and 2 pits.

16.2 Leaching Options

GRE evaluated two leaching options:

- crushing of all material above the mining cutoff grade
- a combination of ROM and crushing:
    - ROM of all material above a cutoff grade of 0.004 opt (0.14 g/T) up to the mining cutover grade
    - crushing of all material above the mining cutover grade.

16.3 Cutoff/CutoverGrade Options

For this PEA, each scenario was evaluated at five mining cutoff/cutover grades: 0.006 opt (0.2057 g/T), 0.007 opt (0.24 g/T), 0.008 opt (0.274 g/T), and 0.009 opt (0.309 g/T), and 0.010 opt (0.343 g/T).

Figure 16-1: Four Ultimate Pits

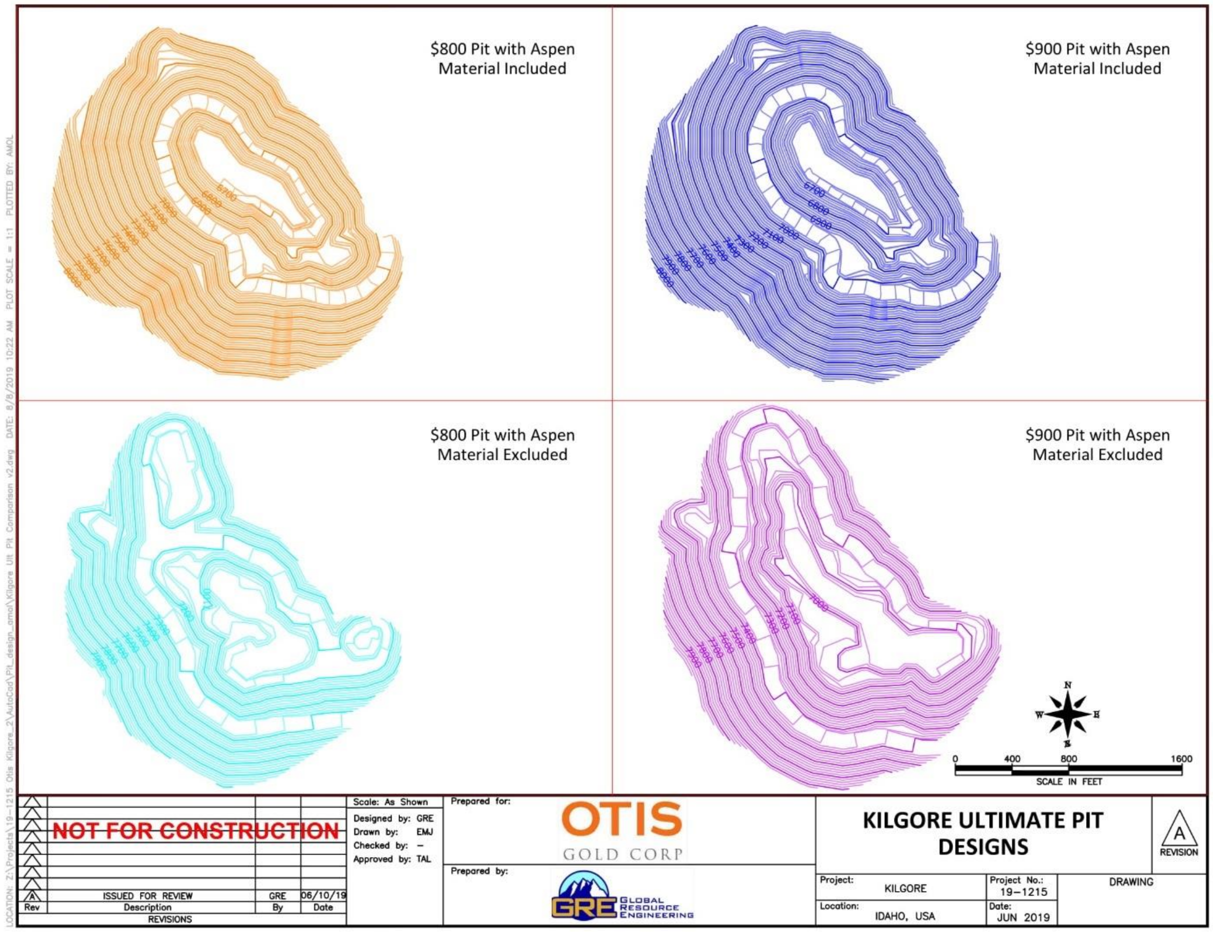
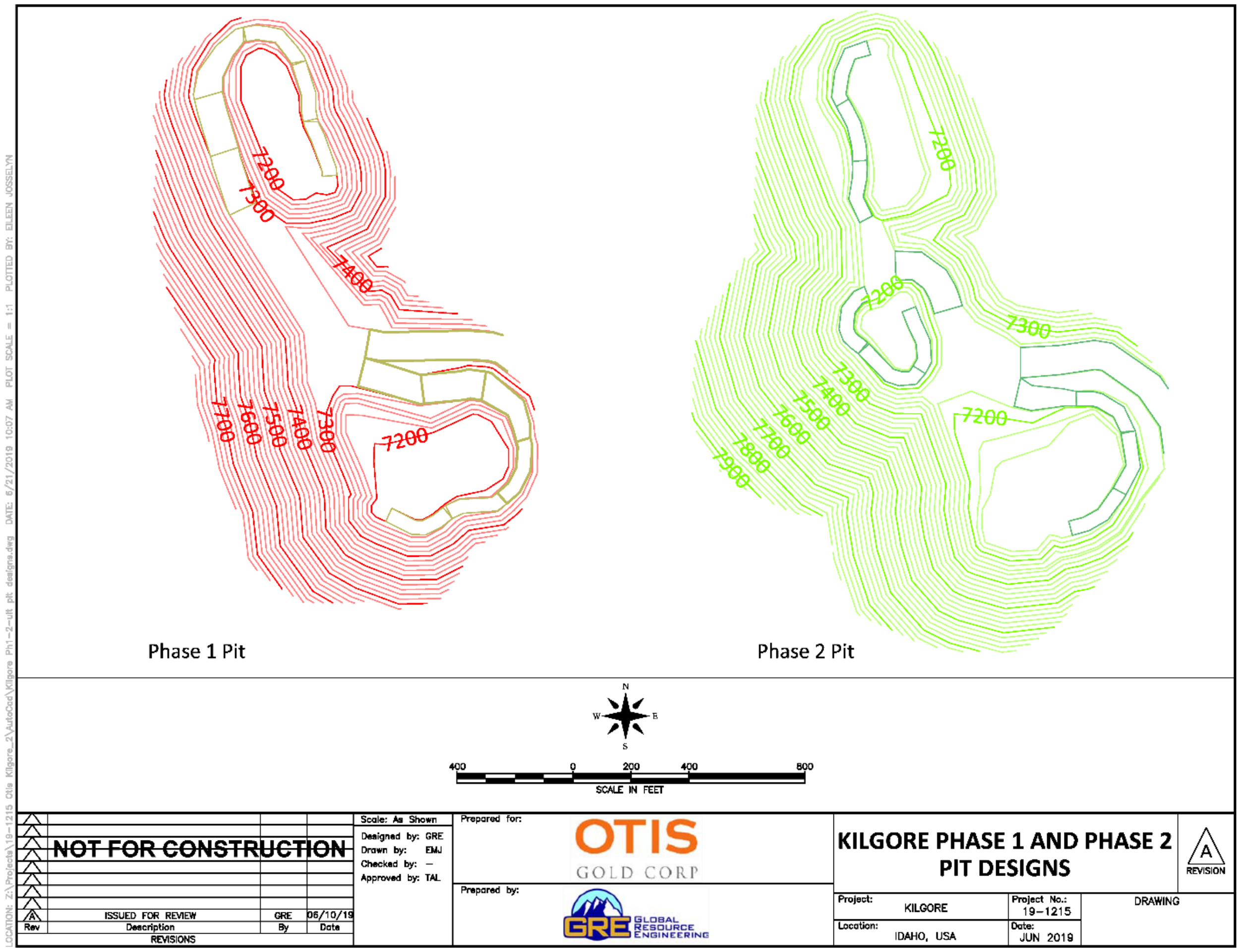


Figure 16-2: Phase 1 and Phase 2 Pits



16.4 Reported Resources

Resources for each designed pit were reported out of GEMS by bench and with codes identifying rock type, average grade of leachable material, and resource category (i.e., Indicated and Inferred). Both resource categories (Indicated and Inferred) were treated equally for the purposes of this PEA. For the two pits that included the Aspen material, 60% of that material was considered leachable; the other 40% was considered waste. Similarly, 60% of the reported gold ounces within the Aspen rock type were included as contained ounces and the other 40% were considered waste.

16.5 Evaluation

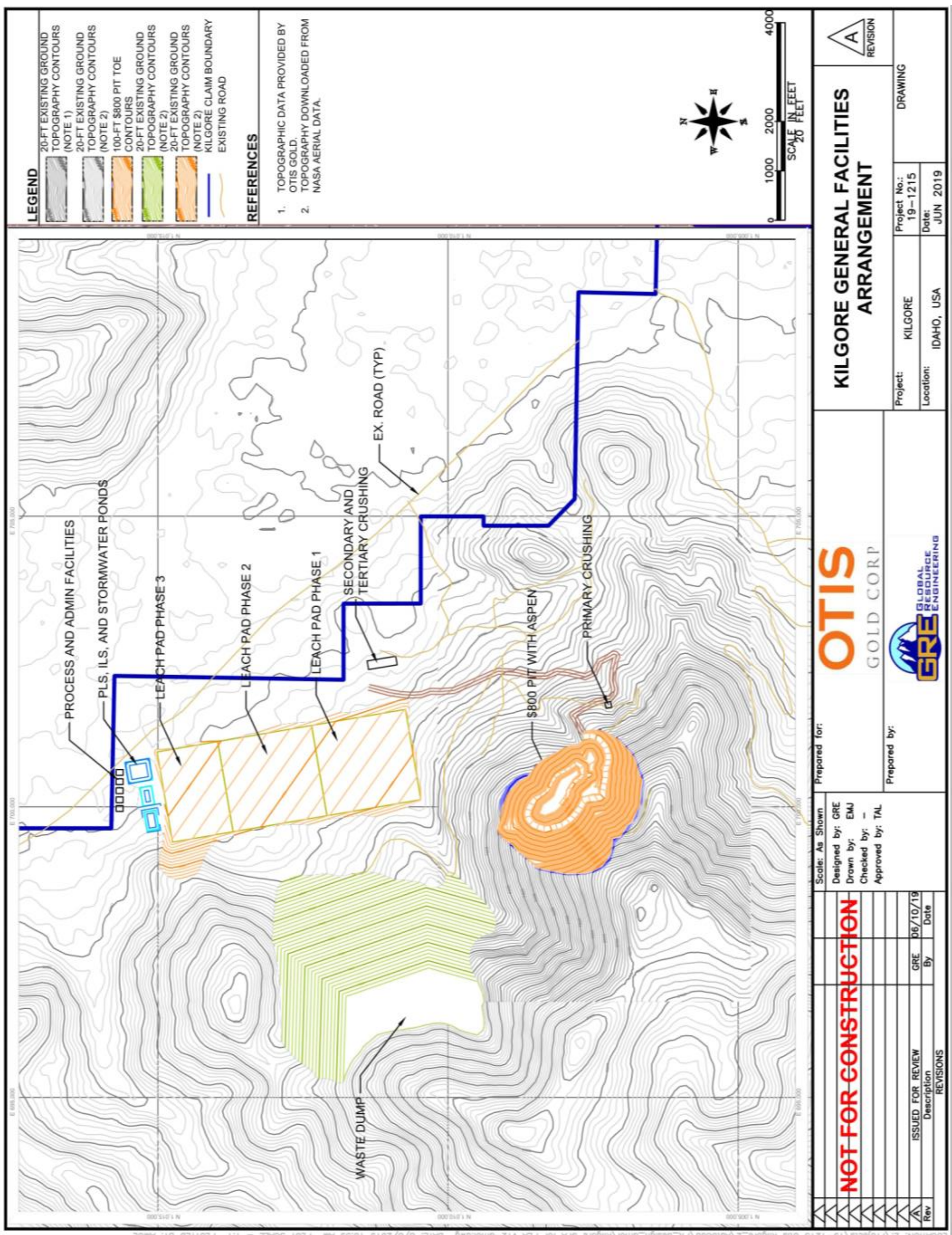
GRE's economic model for the mine evaluated the four designed pits with two crushing options at five cutoff/cutover grades (a total of 40 cases) to optimize the mine planning and design. Based on the economic analysis of all 40 cases, GRE selected the $800/oz Au pit with Aspen at a cutoff grade of 0.010 opt (0.34 g/T) with ROM and crush processing. All further references to "the ultimate pit" or the "base case" in this report are referring to the $800 pit with Aspen at a cutoff grade of 0.010 opt (0.34 g/T) with ROM and crush processing.

The resources contained within the base case designed pit are shown in Table 16-1, and the layout of the pit is shown in Figure 16-3.

Table 16-1: Kilgore Base Case Designed Pit Resource

Phase	Ore Tons Crush (millions)	Ore Tons ROM (millions)	Waste Tons (millions)	Total Tons (millions)	Au Ounces Crush (1000s)	Au Grade Crush (opt)	Au Grade Crush (g/T)	Au Ounces ROM (1000s)	Au Grade ROM (opt)	Au Grade ROM (g/T)	Stripping Ratio
1	6.73	6.57	8.85	22.16	155.04	0.0230	0.79	0.00	0.0000	0.00	0.67
2	3.94	3.84	6.88	14.66	77.82	0.0198	0.68	0.00	0.0000	0.00	0.88
3	16.42	16.56	44.22	77.20	337.86	0.0206	0.71	181.49	0.0024	0.08	2.69
Total	**27.09**	**26.98**	**59.95**	**114.02**	**570.72**	**0.0211**	**0.72**	**181.49**	**0.0016**	**0.05**	**1.11**

Figure 16-3: Kilgore Mine General Facility Arrangement



16.6 Mine Scheduling

A preliminary mining schedule was generated from the base case pit resource estimate. GRE used the following assumptions to generate the schedule:

- Leachable/Crush Material Production Rate: 15,000 tons per day (tpd)
- Mine Operating Days per Week: 7
- Mine Operating Weeks per Year: 52
- Mine Operating Shifts per Day: 2
- Mine Operating Hours per Shift: 10

The schedule was broken out into Phase 1, Phase 2, and Phase 3. Phase 1 included all material within the originally generated $550 pit shell, which was designed in GEMS with haul roads; Phase 2 included all material within an intermediate pit; and Phase 3 included all remaining material, approximating a phased pit design. Additional design work will likely better balance the annual leach and waste stripping and metal production.

Pre-stripping of waste was included if either of the following criteria were true: 1) waste occurred on a bench that had no corresponding leachable material or 2) the tonnage of waste on a bench exceeded 10 times the tonnage of leachable material on that bench. The production rate for pre-strip benches was set to three times the leach material production rate, or 45,000 tpd.

For all other benches, all waste on a bench was scheduled to be mined over the same duration as the crush material on that bench. For scenarios with ROM material, the ROM was scheduled to be mined over the same duration as the crush material on that bench. Average ROM production was approximately 15,000 tpd. This scheduling method resulted in some years with high waste quantities relative to the crush material quantity mined. GRE used pre-stripping and phasing, as described above, as much as possible to smooth out the production, but the limitations of the scheduling program resulted in some inefficiencies.

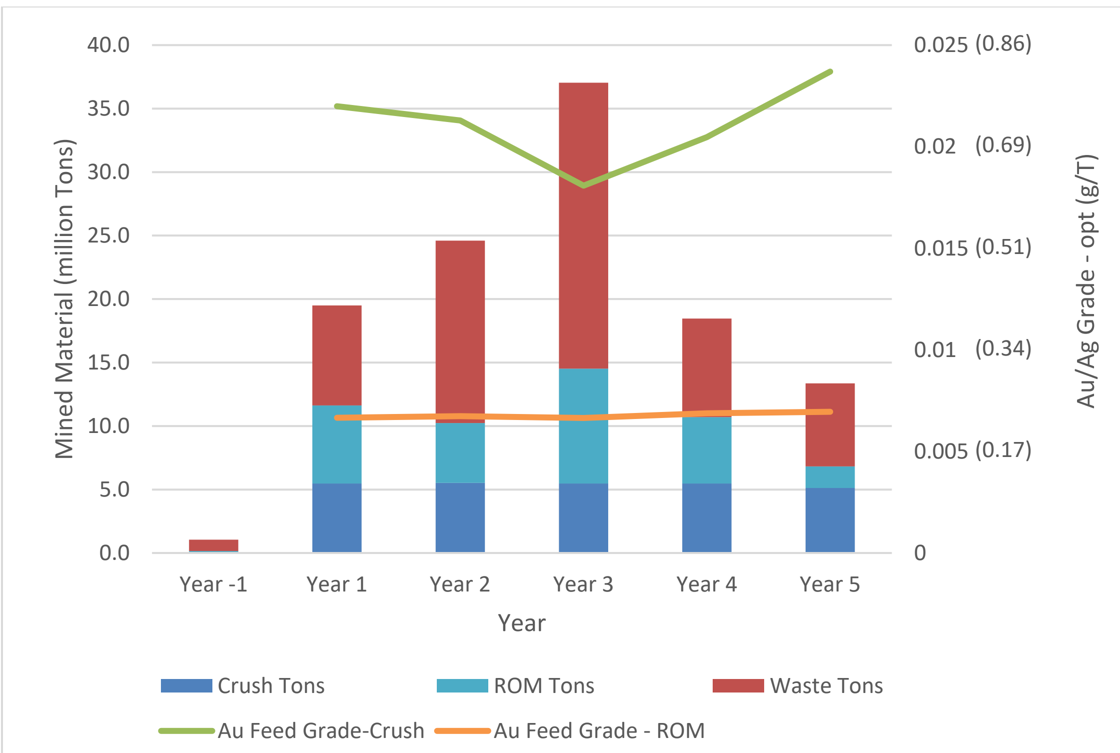
The mining schedule is summarized in Table 16-2 and Figure 16-4.

Table 16-2: Mine Schedule Summary

Pit Phase	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Crush Tons							
Phase 1	0.021	5.475	1.238	0.000	0.000	0.000	6.734
Phase 2	0.000	0.000	3.937	0.000	0.000	0.000	3.937
Phase 3	0.000	0.000	0.345	5.475	5.475	5.124	16.419
Total	0.021	5.475	5.520	5.475	5.475	5.124	27.090
ROM Tons							
Phase 1	0.138	6.143	0.291	0.000	0.000	0.000	6.572
Phase 2	0.000	0.000	3.842	0.000	0.000	0.000	3.842
Phase 3	0.000	0.000	0.582	9.046	5.233	1.701	16.562
Total	0.138	6.143	4.715	9.046	5.233	1.701	26.976
Waste Tons							
Phase 1	0.886	7.884	0.082	0.000	0.000	0.000	8.852
Phase 2	0.000	0.000	6.879	0.000	0.000	0.000	6.879

Pit Phase	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Phase 3	0.000	0.000	7.408	22.520	7.752	6.539	44.220
Total	0.886	7.884	14.369	22.520	7.752	6.539	59.950
Au Ounces - Crush							
Phase 1	0.3	120.4	34.4	0.0	0.0	0.0	155.0
Phase 2	0.0	0.0	77.8	0.0	0.0	0.0	77.8
Phase 3	0.0	0.0	5.3	99.0	112.1	121.4	337.9
Total	0.3	120.4	117.5	99.0	112.1	121.4	570.7
Au Ounces - ROM							
Phase 1	0.8	40.9	2.2	0.0	0.0	0.0	44.0
Phase 2	0.0	0.0	25.9	0.0	0.0	0.0	25.9
Phase 3	0.0	0.0	3.7	60.1	36.0	11.8	111.6
Total	0.8	40.9	31.8	60.1	36.0	11.8	181.5

Figure 16-4: Mine Production Summary



16.7 Mine Operation and Layout

The general mine arrangement would include a crushing site, leach pad and ponds, plant site, and waste dump. The leach material and waste would be drilled and blasted using a rotary crawl driller and ammonium nitrate fuel oil (ANFO) and transported in dump trucks to the primary crusher, which would be located near the pit.

Initially, leachable crush material would be trucked from the pit to the primary crushing site, while leachable ROM material would be trucked directly to the leach pad, and waste rock would be trucked to the waste dump. After crushing, leachable crush material would be transported to the leach pad by conveyor, where it would undergo leaching with sodium cyanide, with gold recovered from the pregnant solution using the ADR process.

The proposed general mine arrangement is shown in Figure 16-3.

16.8 Mine Equipment Productivity

GRE estimated cycle times determining the equipment size and numbers of trucks and loaders that would be required to meet the project schedule. A simplified approach to cycle calculations was used; it considered productivity variables such as average daily production of leachable material and waste, average truck haul distance and travel speed, hours per shift and shifts per day, availability variables such as breaks during the day, and truck and loader/shovel capacities. Hourly production rates and truck and loader wait times were calculated in order to optimize the design. The final analysis uses Caterpillar 777G size trucks, with a heaped capacity of 84 cubic yards, and Caterpillar 992K size loaders, with a bucket capacity of 16 cubic yards.

17.0 RECOVERY METHODS

17.1 Process Description

The Kilgore project would employ open pit mining with a conventional heap leach system on a 365 day per year 24 hour per day basis. A conventional heap leach process has been proposed for the Kilgore deposit. Depending on the grade of the material it will either be crushed prior to being placed on the heap leach or treated directly as ROM. Both ROM and crushed material would have lime added prior to pad placement for pH control. The ROM material would be trucked to a separate area of the pad and dumped and ripped with a dozer after each lift is complete. The crushed material would be stacked into lifts on the lined pad by means of a radial stacker. The stacker would be fed by a series of jump or grasshopper conveyors that would be fed from the main overland conveyor from the crusher. The crushing circuit has been designed to minimize truck haulage by placing the primary crusher closer to the pit and using an overland conveyor to transport material to the secondary crushing facility. This system takes advantage of the topography to significantly reduce crushed material haulage cost. Crushing is conducted in an open circuit manner to minimize the number of screens and conveyors.

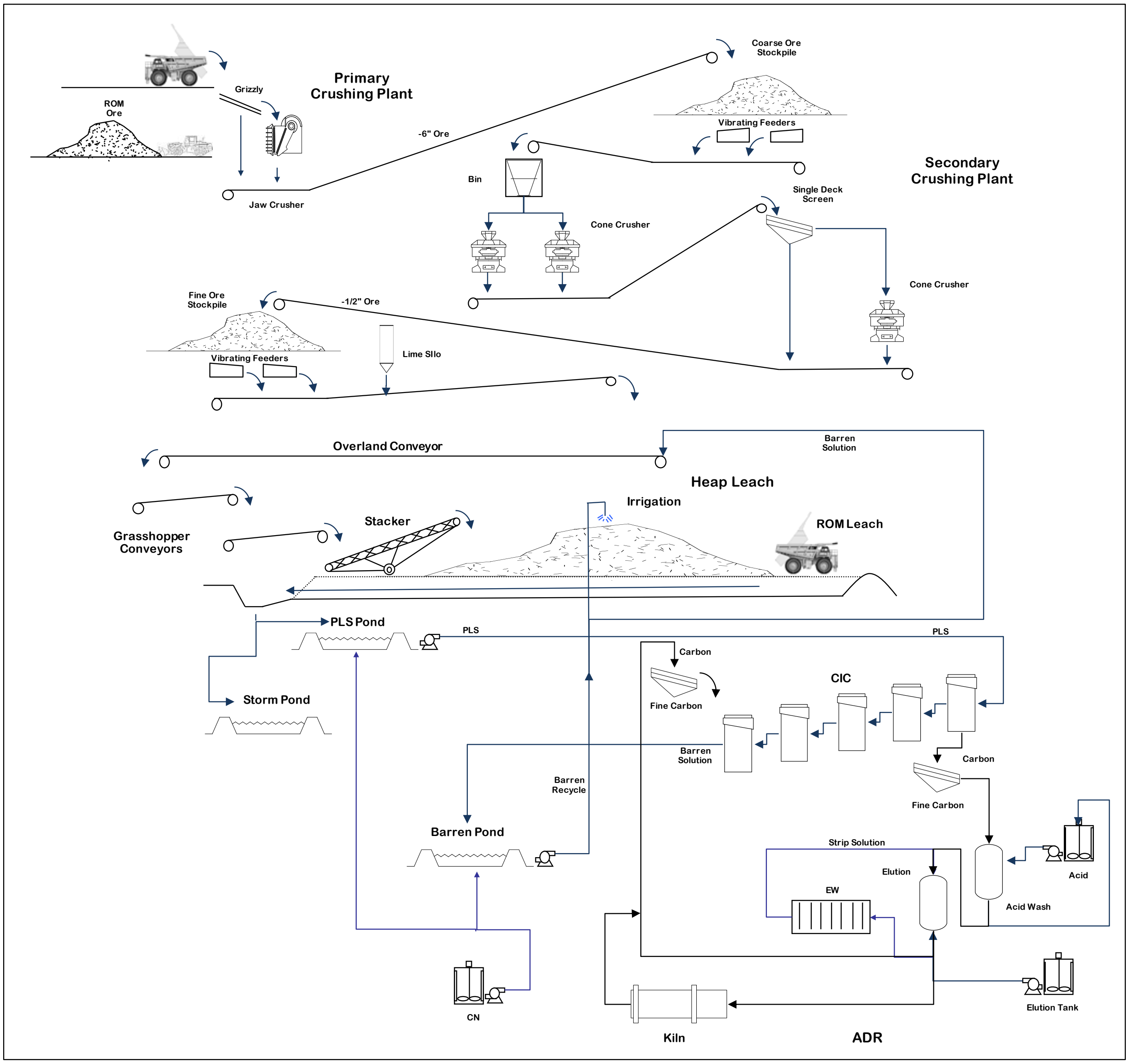
The heap leach would consist of a suitable area lined with a containment system, typically a linear low-density polyethylene (LLDPE) liner with an over liner of sized material to facilitate drainage. Within this over liner would be placed drainage pipes to conduct the leach solution to the centralized collection ponds. The crushed material would be stacked by a radial stacker and the ROM material direct end-dumped into heap lifts ranging from 5 to 10 meters in height. After a suitable area has been stacked, the heap would be irrigated with dilute cyanide solution. Stacking would continue, and the areas irrigated with solution as the heap advances. The solution leaches gold from the heap materials and is transported to a series of storage ponds.

This pregnant leach solution would be collected in a dedicated pond and either recirculated or processed in the ADR plant. The gold in the solution would be collected on activated carbon in a series of carbon-in-column (CIC) vessels (from 4 to 8 columns is typical). The depleted “barren” solution would report back to the heap leach barren pond and have the reagent levels adjusted prior to being recirculated back to the heap.

Once the gold level on the carbon in the CIC circuit reaches a specific setpoint (e.g., 3,000 g/T in the lead column), the carbon is advanced, and a set amount removed for gold recovery. Gold recovery takes place through stripping the activated carbon using a specifically designed process (ZADRA or Anglo American Research Laboratory [AARL] are typical). The gold is stripped from the carbon into an enriched solution that reports to an electrowinning circuit where the gold is recovered as a sludge that is ultimately smelted into high purity doré bars.

The heap leach is typically designed to have multiple lifts installed. Each new lift goes on top of the last lift until the heap reaches its ultimate height. Heap leaches often utilize 10 or more lifts to reach an ultimate height of 328 to 492 feet (100 to 150 meters). The configuration of the heap leach is heavily dependent on the permeability characteristics of the material, the terrain available, and the geotechnical aspects of the site. Figure 17-1 shows the complete conceptual flowsheet.

Figure 17-1: Conceptual Heap Leach Flowsheet



17.1.1 Crushing Circuit

The run of mine feed passes over a vibrating grizzly with a 2.5-inch (64-mm) opening. The undersize reports directly to the jaw crusher discharge conveyor while the oversize feeds the jaw crusher. The jaw crusher would crush to a nominal 5-inch (100-mm), with the crushed product reporting by conveyor to a coarse ore stockpile. The crusher would be located near the mine uphill from the secondary crushing plant approximately 4,000 feet (1,219 meters) away with about a 600-foot (183-meter) drop in elevation. This will reduce haul time to and from the crusher plant. The crusher would process approximately 15,000 short tons per day (tpd) on a 24-hour basis with an availability of 85%. The design crushing rate is 765 tph.

Three vibrating feeders beneath the stockpile would feed a conveyor belt that discharges to the secondary crusher bin. This bin feeds two standard cone 6-foot crushers with a closed side setting of 0.87 inches (22 mm). The discharge from the crushers falls onto a common conveyor belt feeding a single deck screen.

The screen is equipped with a 0.79-inch (20-mm) screen. The secondary crushing circuit is operated in open circuit.

The screen oversize would be fed to the secondary 6-foot short-head cone crusher (CSS 17 mm). Screen undersize (< 0.79-inch [< 20-mm]) would report to the fine ore stockpile. Tertiary crusher discharge reports directly to the fine ore stockpile in open circuit on the common conveyor. This crushing circuit would be capable of achieving a P_{80} of ½-inch.

Ore would be reclaimed from the fine ore stockpile by a series of three vibrating feeders. Crusher material would be reclaimed from the stockpile and fed to overland conveyor. Barren solution and lime would be added to the material. The target is to deliver approximately 50% of the total cyanide demand to the ore while not exceeding 8 to 10% moisture by weight. The ore would be conveyed via overland and grasshopper conveyors to a prepared permanent leach pad. The ore is stacked using a slewing radial stacker to lift heights of 26 feet (eight meters). Stacking would be conducted in retreat mode during the creation of each leach cell.

17.1.2 Heap Leach Circuit

Ore would be allowed to cure for up to five days prior to irrigation being introduced; this also allows operation personnel to be a safe distance from active irrigation areas. Irrigation is provided by an emitter-type irrigation system designed to deliver 0.005 gallons per minute per square foot (gpm/ft^2) (12 liters per hour per square meter [lph/m^2]). Emitter layout is designed to provide suitable ore wetting. The heap would be placed under primary irrigation for a period of approximately 90 days. After the primary leach, irrigation would be discontinued. No rinse phase is included because of the multiple lift system employed. The subsequent lift would be placed on top up to a total of 10 lifts. Rinsing would only occur before closure or once the heap reaches its ultimate height.

High concentration gold leach solutions or pregnant leach solutions (PLS) flow from the pad to the PLS pond by gravity. Solution is collected from each heap cell by a series of drain pipes under the heap that transport the solution to perimeter ditches. The solution can be placed in either the PLS, Barren, or storm water ditch. PLS solutions flow in the ditch by gravity to the respective pond. Storm water collected from the pad during heavy precipitation events can be diverted to a storm water pond. The storm water can be used as fresh make up water to the circuit.

17.1.3 Adsorption, Desorption, Recovery (ADR)

During normal operations, PLS solution is pumped to the CIC tanks. The CIC circuit consist of two trains of five CIC vessels, each containing four tons of carbon. Carbon is advanced counter current to the PLS flow as the first tank in the series reaches its loading limit. The target carbon loading is 3,000 g/T of gold. Carbon is advanced by recessed impeller pumps.

The loaded carbon from the first tank is pumped across the loaded carbon screen to the acid wash column. The screen under flow is returned to the PLS flow. Fine carbon from the screen underflow is stockpiled and sent for separate off-site recovery.

The barren solution exiting the last of the CIC would be returned to the heap leach barren solution pond after passing through a carbon safety screen. Loaded carbon would be acid washed with dilute nitric acid to remove calcium and adsorbed metals. Spent acid would be neutralized and disposed. After acid washing, the carbon would be passed to an elution column. Elution would be conducted by the modified ZADRA system. A solution of caustic and cyanide would be passed through the elution column to remove the adsorbed gold. The rich electrolyte would be pumped to electrowinning cells, where the gold and silver are recovered on the cathodes. The cathodes are washed, and the recovered sludge would be refined in a conventional induction furnace after drying. The circuit would be designed to conduct two strip cycles per day. The doré produced would be assayed and stored in a vault before being shipped off-site for payment.

Barren carbon from the elution column would be returned to the CIC circuit after passing across a carbon sizing screen. Fine carbon from the screen underflow would be stockpiled and sent for separate off-site recovery. Approximately 50% of the barren carbon reports to an indirect fired kiln for thermal regeneration. The regenerated carbon reports to a quench tank before being pumped to the carbon sizing screen. Fresh makeup carbon would be first sent to an attrition tank for fines removal before being pumped to the carbon sizing screen. The fine carbon from the screen underflow would be captured in a plate and frame filter.

17.2 Optional Mill Processing Flowsheet

Portions of the Kilgore material appear better suited to carbon-in-leach (CIL) processing due to the presence of active carbon and higher head grade. At the present time, the process flow sheet does not include CIL processing of the carbon-rich material; however, this process can be evaluated in subsequent studies. Often two processing systems are developed for projects with variations in grade and mineralogy. Typically, a heap leach is employed for the lower grade materials and a mill system for the higher-grade material. The determination of which process is utilized for which material is defined by the cutover grade. The cutover grade defines the material treatment path: grades higher than the cutover grade would report to the mill, and grades lower would report to the heap leach.

For Kilgore, a typical milling process would include primary crushing – likely shared with the heap leach plant - followed by semi-autogenous grinding (SAG) and ball milling to achieve a suitable size distribution. The test work indicates that a grind size of 80% passing 75 µm should be sufficient to achieve the desired gold extraction.

The ground material would report to a CIL circuit where cyanide would be added to leach the gold, while activated carbon simultaneously adsorbs the gold from solution. This leaching takes place in a series of stirred tanks equipped with air injection and carbon screens to retain the carbon in the tank. This system would be specifically designed to combat "pregnant solution-robbing" type materials.

The gold-loaded carbon would be advanced counter-current to the leach slurry flow as it becomes loaded with gold. The carbon advancement would be achieved by pumping the tank slurry containing the carbon to the next upstream tank. The slurry flows back to the original tank, and the carbon would be retained by the screens. Loaded carbon would be removed from the circuit once it reaches the desired gold loading, just as in the CIC circuit of the heap leach. Similarly, the carbon would be stripped of its adsorbed gold via a dedicated system such as the ZADRA or AARL system. The stripped carbon would be regenerated with steam in a kiln, as required, before being returned to the leach circuit. In combination plants such as is envisioned here, many of the systems can be shared, including the carbon elution, regeneration, and the gold refining unit operations.

17.3 Conceptual Heap Leach Pad and Pond Design

The HLF consists of the following system components:

- Heap leach pad
- Liner system
- Leachate (solution) collection system
- Storm pond
- Stormwater management system
- Freshwater supply

To minimize capital expenditure, the heap leach pad has been designed in phases, with each phase requiring advanced expansion of the engineered pad. The HLF would be constructed in two phases, with the pad foundation preparation, liner installation, and collection piping advanced as the leach pad expands. The capacity of each stacking stage includes an initial three-year period with one additional three-year period.

The initial HLF development (Phase 1) would also include the full development of the solution handling system, storm pond, and perimeter diversion ditches prior to commencing ore stacking and leaching. Table 17-1 shows the development phases and the lift capacity in ore volume and duration. Design details for each of the HLF components are discussed further in the following sections.

Table 17-1: Heap Capacity

Development Phase	Elevation (abs m)	Lift Capacity (days)	Mine Life (years)	Ore Volume	
				(m^3)	(cum m^3)
1	8	212	0.6	2,105,419	2,105,419
	16	393	1.1	1,798,253	3,903,672
	24	545	1.5	1,515,236	5,418,907
	32	672	1.8	1,256,368	6,675,275
	40	775	2.1	1,021,650	7,696,925
	48	856	2.3	811,081	8,508,006
	56	919	2.5	624,662	9,132,668
	64	966	2.6	462,391	9,595,059
	72	998	2.7	324,269	9,919,327
	80	1019	2.8	210,294	10,129,621
2	8	1231	3.4	2,105,419	12,235,040
	16	1412	3.9	1,798,253	14,033,293
	24	1565	4.3	1,515,236	15,548,528
	32	1691	4.6	1,256,368	16,804,896
	40	1794	4.9	1,021,650	17,826,546
	48	1876	5.1	811,081	18,637,627
	56	1939	5.3	624,662	19,262,289

Development Phase	Elevation (abs m)	Lift Capacity (days)	Mine Life (years)	Ore Volume	
				(m^3)	(cum m^3)
	64	1985	5.4	462,391	19,724,680
	72	2018	5.5	324,269	20,048,948
	80	2039	5.6	210,294	20,259,242
3	8	2202	6.0	1,617,053	21,876,295
	16	2368	6.5	1,655,669	23,531,964
	24	2536	6.9	1,670,132	25,202,096
	32	2704	7.4	1,660,442	26,862,538
	40	2867	7.9	1,626,596	28,489,134
	48	3025	8.3	1,568,594	30,057,728
	56	3175	8.7	1,486,433	31,544,162
	64	3314	9.1	1,380,109	32,924,271
	72	3439	9.4	1,249,615	34,173,885
	80	3550	9.7	1,094,939	35,268,825

17.3.1 Heap Leach Pad

The heap leach pad consists of a perimeter berm, pad liner system, and leachate collection system to collect and convey the leachate solution to the ADR plant, which should be located adjacent to the heap leach facility. The leach pad has an approximate final footprint area of 5,267,500 square feet (489,400 square meters). The heap leach pad is designed to be operated as a fully drained system with no leachate storage within the HLF. Prior to the start of each of the development stages, the pad foundation must be prepared. Foundation preparation involves stripping the topsoil and vegetation and the removal of any rocks. The topsoil would be stockpiled at a convenient location and used for reclamation of the HLF at closure. The underlying soils would be excavated down to a competent, stable bedrock foundation to provide a uniform and graded surface for the pad liner. Grading and backfill would be used to level the bedrock surface and to ensure that the pad grading will promote leachate flow towards the collection piping system and sump. A minimum pad grade of 2% is required.

17.3.2 Liner System

A liner system is planned to maximize solution recovery and minimize environmental impacts by minimizing leachate losses through the bottom of the leach heap pad. The liner system consists of both barrier and drainage layers using a combination of synthetic and natural materials to provide leachate solution containment that meets the accepted standards for leach pad design. The pad is designed to operate with minimal solution storage within the pad structure during normal operating conditions. The liner system is designed to meet the required performance standards assuming fully saturated solution storage conditions.

17.3.2.1 Liner Design

A liner system has been developed for the pad using an engineered composite double liner design. The double liner system is designed to be installed as the primary liner system under the entirety of the HLF. The double liner system consists of the following components:

- 1.6-foot-thick (0.5-meter-thick) over liner (1.5-inch [38-mm] minus with less than 10% fines content) using ore as the material
- 80-mil (2-mm) LLDPE geomembrane
- 1-foot-thick (0.3-meter-thick) compacted low permeability soil liner
- Non-woven, needle punched geotextile layer
- Leak Detection and Recovery System (LDRS)
- 60-mil (1.5-mm) LLDPE geomembrane.
- LLDPE was proposed for the geomembrane liner systems for the heap leach pad because it has the following benefits (Lupo, 2005):
    - Generally higher interface friction values, compared to other geomembrane materials
    - Ease of installation in cold climates due to added flexibility,
    - Good performance under high confining stresses (large heap height)
    - Higher allowable strain for projects where moderate settlement may become an issue.

17.3.2.2 Construction

Development of the heap leach liner would be constructed in two phases, with pad expansions proposed every three years to meet ore stacking requirements. The liner system would be constructed with both the synthetic and natural layers extending to the top of the perimeter berms to provide full containment. The synthetic liners and geotextiles would be anchored and backfilled in a trench along the heap leach pad perimeter and perimeter berms to ensure that ore loading does not compromise the liner coverage of the heap leach pad footprint by pulling the liner into the pad. Along the pad toe, all liners would be tied into their corresponding liner layer along the foundation of the pad to provide a continuous seal and drainage connection.

The perimeter berm would be constructed as part of the liner tie-in around the perimeter of the pad footprint to ensure that heap solution is contained within the pad and to prevent surface runoff entering the pad collection system. A 1-foot-thick (0.3-meter-thick) bedding sand layer would be placed on the face of the confining embankment directly underneath the second (bottom) geomembrane liner to provide additional integrity protection to the liner.

17.3.2.3 Over Liner

A protective layer of approximately 1.5 feet (½ meter) of coarse crushed ore/waste would be placed over the entire liner system footprint to protect the liner’s integrity from damage during ore placement. The over liner acts as the drainage layer, allowing solution drainage into the pipe collection system. The over liner material must be competent and be free from fines.

17.3.3 Solution Collection System

Collection and recovery of the leach solution is facilitated by the solution collection system in conjunction with the heap leach liner, over liner, and LDRS. The collection system consists of the following pipe and sump components:

- Lateral collection pipes

- Collection header pipes
- Main header collection pipes
- Leachate collection sumps

The solution collection system would be designed to facilitate quick and efficient solution conveyance off the pad to reduce the potential risk of solution losses through liner system. The entire piping system would be constructed from perforated corrugated plastic tubing (CPT), which is embedded within the over liner layer.

The lateral collection pipes, which would be spaced approximately 16 feet (five meters) apart under the entire pad footprint, feed directly into the collection header pipes, which then flow into the main header. The main header pipes would be positioned along the centerline of each heap leach pad cell and terminate at the upstream toe of the perimeter berm at the leachate collection ditch. Two leachate collection ditches allow solution to flow by gravity to the required storage pond. The collection pipes would be fitted with gate valves to allow solution to be directed to one of the three perimeter collection ditches – PLS, Barren, or Storm.

17.3.3.1 Leak Detection and Recovery System

The LDRS would be designed to capture and convey any solution that may leak through the overlying geomembrane and low permeability soil layers. The LDRS consists of a 1-foot-thick (0.3-meter-thick) sand layer embedded with 4-inch (100-mm) diameter perforated CPT collection pipes. A non-woven needle punched geotextile overlies the LDRS sand layer to prevent particles from the above soil layer from entering the LDRS. Any leakage recovered by the LDRS would be conveyed into the LDRS sump at the downstream toe of the HLF. A level-switch controlled submersible sump pump would transfer the recovered solution via a pipe installed within the LDRS sand layer and connect into the main solution recovery line for processing. Monitoring of the leakage recovery would be undertaken by recording pump operating hours.

17.3.3.2 Leakage Detection Cells

To facilitate more accurate leak identification, the entire pad solution collection system is typically subdivided into multiple independently monitored areas (cells) separated by small berms. Each of these cells has a dedicated leakage detection collection system comprising a drain gravel layer beneath the inner composite liner system which conveys the leakage to a 4-inch (100-mm) diameter perforated collection pipe within the LDRS collection trench. The LDRS ditches flow by gravity at a minimum 0.5 % slope towards the LDRS collection sump, located along the sides of the leach pad. The flow rates from the dedicated collection pipes are continuously monitored and measured prior to discharging into a sump.

17.3.4 Ponds

17.3.4.1 Storm Pond

The Storm Pond is designed to provide storage for excess leachate and runoff generated as a result of rainfall events. The pond is situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping and ditches. The Storm Pond is designed to meet the following design criteria:

- Storage capacity to contain the excess HLF leachate and surface runoff from the 1 in 100-year 24-hour storm event without discharge
- Overflow designed to discharge the 1 in 200-year 24-hour storm event

The storage requirements for the Storm Pond were established based on containment of the entire estimated surface runoff generated from the HLF (at the Phase 2 footprint) during the 1 in 100-year 24-hour storm event. Based on the surface runoff estimates, the following storage requirements for the events pond were identified:

- Total runoff estimate for 1 in 100-year 24-hour storm event 2,038,000 cubic feet (57,700 cubic meters)
- 10% additional factor of safety 203,800 cubic feet (5,770 cubic meters)
- Total pond storage capacity 2,241,800 cubic feet (63,470 cubic meters)

Solution stored in the Storm Pond would be pumped back to the heap leach pad using the Storm Pond pump station. The pump station is designed to be able to drain the storm volume over a period of approximately ten days.

17.3.4.2 PLS and Barren Ponds

The PLS and Barren ponds are designed to provide storage for leachate and CIC return solutions. The ponds are situated immediately down gradient of the HLF, and pond flows are conveyed via solution collection piping and ditches. The PLS and Barren ponds are designed to meet the following design criteria:

- Storage capacity to contain sufficient solution volumes to maintain irrigation and feed to the CIC circuits
- The PLS and Barren Ponds are designed to contain up to 24 hours of solution assuming a maximum irrigation rate of 15 lph/m^2
- PLS and Barren Ponds are designed with a capacity of approximately 997,000 cubic feet (28,000 cubic meters).

Excess solution flows to any of these ponds would be diverted to the Storm Pond for recycle back to the heap.

17.3.4.3 Pond Liner System

The engineered double liner system designed for the ponds uses the same design principles as the HLF pad liner system. The liner design consists of the following layer configuration:

- 60-mil (1.5 mm) high-density polyethylene (HDPE) geomembrane
- 1-foot-thick (0.3-meter-thick) low permeability soil liner
- Geosynthetic “geonet” drainage layer
- 60-mil HDPE geomembrane.

The liner system installed on the upslope of the pond embankment would have an additional 1-foot-thick (0.3-meter-thick) bedding sand layer that would interface with the lower geomembrane layer to protect the integrity of the liner.

Installation of a LDRS is not required for the Storm Pond as the pond is operated as a dry facility and would only receive and store runoff water during significant storm events. In the event that leakage does occur through the double liner system, this water would be conveyed via the geonet layer to a 3-foot-thick (1-meter-thick) drainage blanket that underlies the Storm Pond embankment. This drainage blanket discharges to a sump for solution return to the pond.

It is recommended that HDPE geomembrane be used for the pond liner system rather than LLDPE. Unlike the heap leach pad, the pond liner system would not be subjected to high confining stresses from ore stacking, and HDPE has a higher ultraviolet resistance, which is critical for exposed surfaces like that of the ponds.

17.3.5 Runoff Collection and Diversion

The surface water management system proposed for the site consists of a series of ditches constructed around the perimeter of the HLF to intercept overland surface runoff around the HLF pad and to convey surface water away from the active site. The ditches are designed to meet the following design criteria:

- Conveys the 1 in 100-year 24-hour duration storm event
- Minimum freeboard = 1-foot (0.3 meters)
- Minimum ditch grade = 0.01 foot/foot (meter/meter)
- Side slopes = 2H:1V
- Channel shape = trapezoidal.

Lining and protection of the ditch channels from erosion and scouring may be required for all permanent ditches. Temporary ditches would be constructed between heap phases.

18.0 PROJECT INFRASTRUCTURE

A limited amount of infrastructure is currently available on site. Power, water, and all other systems necessary for a mining and processing operation will be required.

Sufficient water appears to be available on the Kilgore property. Groundwater wells would be developed to meet the project water requirements.

Power is available near the mine site from the Idaho Public Service grid through a 33kV power line. There are no electrical substations at the site. Local labor for mining is available.

18.1 Water Supply

Modeling of the heap operation on a monthly basis over the projected mine life indicates that operation of the HLF requires a water supply with an approximate average flowrate of 259 gpm (59 m^3/hr). An additional 88 gpm (20 m^3/hr) is required for mine, shop, and office water consumption.

18.2 Water Balance

A preliminary operational average monthly water balance model was developed for the HLF. The intent of the modeling was to estimate the magnitude and extent of any water surplus or deficit conditions in the HLF based on annual average climatic conditions. The modeling timeline was for 6 years of HLF operations.

The model incorporates the following major project components:

- Heap Leach Pad
- Mine Usage
- Shop Usage
- General Usage
- Fresh Water Supply
- Pond Storage – PLS, ILS, Raffinate, and Storm

The findings of the water balance were that the HLF would operate in a water deficit. The deficit is most pronounced in the early years and is reduced as water stored within the ore is released from the earlier leaching stages. The total make-up required by the HLF is estimated at 2,000 acre-feet (2.5 million cubic meters) over the life of the facility. The HLF water requirement ranges from 370 acre-feet (457,000 cubic meters) to 485 acre-feet (598,000 cubic meters) annually. The project requires a significant amount of water at start up due to the initial ore wetting requirements and the solution retention in the heap. GRE estimates that approximately 101 acre-feet (125,000 cubic meters) of fresh water would be necessary at the start of heap operations.

The water balance was based on assumed moisture content values for the stacked ore and climatic conditions for the site. The model is sensitive to these values and they should be reviewed and confirmed for future design studies. The following criteria were employed in the water balance:

- Natural Moisture Content – Ore 5%

- Field Moisture Content – Ore 15%
- Drain-Down Final Moisture Content 10%
- Evaporation Losses – Irrigation 5%
- Pan Evaporation for pond base on Dubois Idaho.
- Average Irrigation Rate 0.005 gpm/ft^2 (12 lph/ m^2)
- Pad Area – Phase 1 – 2,583,000 square feet (240,000 square meters), Phase 2 – 2,583,000 square feet (240,000 square meters), Phase 3 – 1,669,000 square feet (155,040 square meters)
- Climate Conditions monthly temperature, precipitation and evaporation

18.3 Mine Facilities

GRE has provided conceptual design of facilities required for mine operations. These include access roads, offices, warehouses, shops, leach pad and ponds, and waste dumps (see Figure 16-3).

19.0 MARKET STUDIES AND CONTRACTS

No market studies or contracts were created for the project. Third party refiners are common in the international gold market, and gold doré is a readily marketable commodity. The study assumes a selling cost of $2.20 per ounce of gold, which is a value on the conservative side.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

In December of 2010, Golder Associates prepared a Preliminary Environmental Report (Golder Associates, 2010) to provide Otis Gold with an overview of potential ecological and environmental issues that may be encountered in developing a new mine, identify potential roadblocks to development, and outline the environmental permit process, and additional baseline studies that will be necessary to develop the project. The report identifies the various permits that will be required on the local, state, and Federal level if this project goes to a mining stage. The Golder report states that, in general, the company did not identify any issues that they consider to be fatal flaws. They state that, as project development and design continue, and specific studies are completed for the project facilities, it is possible that issues may surface that are currently difficult to identify. The Golder Report recommends that general baseline studies for the immediate project area be initiated. Otis Gold plans to initiate these various baseline studies when funding permits.

20.2 Cultural Inventory

In 2010, and in connection with permitting a Plan of Operation (PoO), North Wind Engineering of Idaho Falls, Idaho, performed a cultural inventory of the Mine Ridge area. No material items were identified, and the PoO was approved. Subsequent PoOs have been approved after the completion of two separate Environmental Assessments in 2015 and 2018, and no cultural issues were identified.

20.3 Permitting

As covered in Section 4.5 of this report, the Kilgore Project is located on federal ground administered by the USFS. The local headquarters for this portion of the Caribou-Targhee National Forest is located in Dubois, Idaho. In 2018, the USFS issued a Decision Notice authorizing a 5-year exploration PoO, which approved drilling on 140 drill sites throughout the Kilgore Project, including step-out and infill drilling at the existing Kilgore deposit and exploration drilling at emerging targets including Gold Ridge, Prospect Ridge, and Dog Bone Ridge. The PoO was approved by the USFS after completion of an Environmental Assessment, a process which included input and feedback from the local community and others. As required by the current PoO, Otis Gold is currently increasing its reclamation bond from \$121,275 to \$370,600 and meeting other USFS stipulations in the Decision Notice to enable the commencement of exploration activities, including road construction and drilling.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The following mining, processing, and G&A items were included in the capital cost estimate. The majority of the capital costs occur during year -1, i.e., during the year preceding the start of operations. Leach pad expansions, however, are scheduled to occur as needed throughout the mine life.

- The project plans to use contractor mining, and mining production equipment would be provided by the contractor; therefore, no capital costs are included for mining production equipment.
- Mining support equipment would be provided by the contractor; therefore, no mining support equipment capital costs are included.
- Mining facilities would be provided by the contractor; therefore, no mining facilities capital costs are included.
- The Process Plant includes an ADR recovery system, leach pad, ponds, crushing plant and stacking system, a laboratory, and mobile equipment.
    - The ADR Plant includes the following items:
        - Carbon Columns - Train A/B
        - CIC Feed Pumps
        - Loaded Carbon Screen
        - Loaded Carbon Pump
        - ZADRA Package - Acid wash, Strip
        - Barren Carbon Tank
        - Barren Carbon Pump
        - Regeneration Kiln
        - Carbon Quench Tank
        - Regeneration Carbon Pump
        - Fresh Carbon Attrition Tank
        - Fresh Carbon Attrition Agitator
        - Fresh Carbon Pump
        - Fresh/Reg Carbon Screen
        - Strip Solution Mix Tank
        - ADR Building
        - Heat Exchangers
        - EW Cells
        - Rectifiers
        - Induction Furnace
        - Sump Pump
    - The leach pad was sized with 10 26-foot (8-meter) lifts and a maximum height of 260 feet (80 meters and includes the following items:
        - Pad Liner (2-layer LLDPE + leak detection)
        - Geotextile underdrain
        - Drain Rock
        - Interlift Liners

- PLS/Barren Liner
- PLS/Barren Geonet
- Storm Pond Liner
- Storm Pond Liner Geonet
- PLS Pump
- Barren Pump
- Leach Underdrain Piping
- Leach Underdrain Piping
- Irrigation Headers
- Earth Works

- The primary Crushing Plant includes the following items:
  - Coarse Ore Bin
  - Vibrating Grizzly
  - Rock Breaker - MB432
  - Jaw Crusher - C150
  - Discharger Conveyors + Magnets
  - Stockpile Reclaim Feeders
  - Weightometer
- The secondary Crushing Plant includes the following equipment items:
  - Cone Crusher - std medium
  - Feed Bin
  - Vibrating Feeder
  - Conveyors
- The tertiary Crushing Plant includes the following items:
  - Cone Crusher - SH fine
  - Vibrating Screen
  - Conveyors
  - Building
- The stockpile area includes the following items:
  - Reclaim Feeders - Stacking
  - Feed Conveyor
  - Weightometer
  - Overland Conveyor
- The following utilities were included:
  - Compressed Air
  - Process Water Tank
  - Process Water Pump
  - Sub-Station
  - Forklift
  - Miscellaneous Equip
- The following construction activities/items were included:
  - Installation labor
  - Concrete
  - Piping
  - Structural steel

    - Instrumentation
    - Insulation
    - Electrical
    - Coatings and sealants
    - Spares and first fills
    - Engineering/management
- G&A items include:
    - Survey
    - Guard house/security
    - Startup training
    - Emergency vehicle/supplies
    - Office
    - Warehouse
    - Engineering
    - Fire Protection
    - Water supply
    - Power line to the site
    - Electrical and switchgear
    - Reclamation bond
- Development included pioneering, clearing, grubbing, access road improvements, and haul road construction, assumed to be 20,000 feet of new haul roads.

Capital cost estimates for were obtained from budgetary pricing from equipment suppliers, GRE internal database, and InfoMine USA, Inc. *Mine and Mill Equipment Costs, An Estimator's Guide* (InfoMine, 2018).

Working capital was estimated to be 2 months' operating costs. The working capital was estimated to be recovered the year after production ends. Sustaining capital was estimated as 10% of the mobile equipment cost per year. Capital contingency was set at 20%.

Owner's costs such as permitting, land, exploration, metallurgical testing, and feasibility studies were not included in the cash flow.

The capital costs are summarized in Table 21-1.

Table 21-1: Summary of Kilgore Capital Costs (millions)

Capital Cost Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mine	$3.90	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$3.90
Process	$41.12	$0.00	$0.00	$7.87	$0.00	$6.03	$0.00	$0.00	$0.00	$55.03
G&A	$7.21	$1.52	$1.52	$1.52	$1.52	$1.42	$5.00	$0.00	$0.00	$19.71
Sustaining	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Working	$15.46	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	($15.46)	$0.00
Contingency	$13.54	$0.30	$0.30	$1.88	$0.30	$1.49	$1.00	$0.00	$0.00	$18.82
Total	**$81.23**	**$1.52**	**$1.52**	**$9.39**	**$1.52**	**$7.45**	**$5.00**	**$0.00**	**($15.46)**	**$97.46**

21.2 Operating Costs

Operating costs include manpower, mining equipment costs, processing equipment costs, consumables, and G&A operating costs.

- Mining production equipment hours were estimated from the equipment productivity estimates (see Table 16-2), the scheduled leach and ROM material and waste tonnage, and the number of pieces of equipment required.
- Mining production equipment costs were assumed to be recovered by the contractor at a rate of every 2 ½ years.
- Mining support equipment hours were calculated from the number of pieces of equipment times the operating hours per day, assuming utilization of 90% and availability of 95%, times the operating days per year.
- Mining support equipment costs were assumed to be recovered by the contractor at a rate of every 2 ½ years.
- Blasting materials requirements were determined in a drilling and blasting schedule that used the parameters and assumptions detailed in Table 21-2.

Table 21-2: Parameters for Drilling and Blasting

Constants		
ANFO Density	0.028093	tons/ft3
ANFO powder factor - ore	0.453593	lb/ton
ANFO powder factor - waste	0.362874	lb/tonne
bench height	20	ft
rock density	0.076	tons/ft3
drilling rate	1.968504	meters/min
drill availability %	0.9	
minutes used/hr	50	
available hours per shift	9	
blasthole depth	19.68504	ft
blasthole diameter	9.88	inch
rig type	rotary crawler	
rod length	45	ft
ANFO thickness	16.4042	ft
ANFO setup min/hole	15	min
Calculated		
blasthole diameter	0.823333	ft
blasthole volume	10.48039	ft3
ANFO volume	8.733661	ft3
ANFO weight	0.245351	tons
ANFO weight	490.7011	lb
tons of rock blasted per hole	1081.81	tons/hole
vol of rock blasted per hole	14234.34	ft3/hole
drillhole grid spacing	26.67802	ft

- Operating hours for the crushing plant, ADR plant, and leach pads were assumed to be 24 hours per day, 7 days per week, for 52 weeks per year.
- Reagent usage was estimated from metallurgical testwork, InfoMine estimates, and GRE experience. Table 21-3 summarizes the estimated annual reagent consumption.

Table 21-3: Summary of Estimated Reagent Usage

Item	Unit Consumption	Units
Crusher		
Jaws	4.00	set/yr
Mantles	12.00	set/yr
Screens	4.00	set/yr
Leach		
Cyanide	0.500	lb/ton ore
Lime	2.604	lb/ton ore
ADR		
Cyanide	0.1876016	lb/ton ore
Carbon	0.01876015	lb/ton ore
Caustic	0.1876015	lb/ton ore
Consumables		
Diesel	2.64172	gal/hr

- Manpower for the mine and processing facility includes hourly-rate employees and salaried employees, who are generally superintendents and professional personnel. The number of required equipment operators was estimated using the quantities of equipment required, the quantity of personnel per piece of equipment, and the number of shifts per day. Numbers of required processing and salaried personnel were estimated based on GRE's experience. A burden factor of 40% was added to all labor for fringe benefits, holidays, vacation and sick leave, insurances, etc. A summary of the manpower requirements is provided in Table 21-4.

Table 21-4: Summary of Manpower Requirements

Labor	Quantity	Units	Annual Cost per Person ($)
Mine			
Hourly			
Drillers	4	per machine	$113,724
Blasters	2	per machine	$113,724
Excavator/Loader Operators	4	per machine	$104,976
Truck Drivers	4	per machine	$104,976
Grader/Dozer Operators	2	per machine	$104,976
Water Truck Operators	2	per machine	$104,976
Mechanics	2.5	per machine	$113,724
Laborers/Maintenance	4	per day	$78,732
Salaried Personnel			
Mine Superintendent	1		$202,500

Labor	Quantity	Units	Annual Cost per Person ($)
Foreman	3		$145,800
Engineer	3		$129,600
Geologist	1		$129,600
Technician	1		$97,200
Processing			
Hourly			
Plant Operations			
Process Technician	1		$67,500
Instrument Technician	1		$60,750
Crusher			
Operator	4		$81,000
FEL Operator	4		$81,000
Maintenance	2		$87,750
Electrical	1		$87,750
Leach Pad			
Stacking	4		$81,000
Irrigation Operator	4		$81,000
Reagent Operator	4		$81,000
Dozer/FEL Operator	4		$60,750
Assayers	4		$81,000
Samplers	4		$81,000
Mechanic	1		$60,750
Electrician	1		$87,750
ADR			
Carbon Handling	4		$81,000
EW Operators	4		$81,000
Cathode Stripping	4		$81,000
Refiners	2		$81,000
Samplers	4		$60,750
Reagent Operator	4		$60,750
Salaried			
Plant Superintendent	1		$168,750
Plant General Foreman	4		$121,500
Maintenance Foreman	1		$121,500
Shift Foreman	4		$94,500
Chief Assay Chemist	1		$94,500
Senior Metallurgist	1		$108,000
Metallurgist	1		$81,000

- Administrative operating costs include labor and services and supplies.

Annual and unit operating costs for equipment and facilities were obtained from recent quotes from suppliers, GRE internal database, and InfoMine (2018).

A summary of the project operating costs is provided in Table 21-5, and Table 21-6 provides the project operating unit costs**.**

Table 21-5: Summary of Kilgore Estimated Operating Costs (millions)

Operating Cost Item	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mine	$1.95	$43.71	$47.14	$81.65	$54.86	$35.68	$264.99
Process	$0.08	$33.12	$30.41	$37.90	$31.31	$24.01	$156.83
G&A	$0.15	$2.76	$2.76	$2.76	$2.76	$2.58	$13.75
Total	**$2.18**	**$79.59**	**$80.31**	**$122.31**	**$88.93**	**$62.26**	**$435.57**

Table 21-6: Summary of Kilgore Estimated Operating Unit Costs

Item		Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Mine	($/mined ton)	$1.86	$2.24	$1.92	$2.20	$2.97	$2.67	$2.32
Process	($/process ton)	$3.73	$2.85	$2.97	$2.61	$2.92	$3.52	$2.90
G&A	($/mined ton)		$0.47	$0.54	$0.38	$0.51	$0.76	$0.51

The cash operating costs per ounce of Au is $780, and the all-in cost per ounce of Au is $954.

22.0 ECONOMIC ANALYSIS

Readers are advised that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability under National Instrument 43-101. This PEA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under National Instrument 43-101. Readers are advised that there is no certainty that the results projected in this Preliminary Economic Assessment will be realized.

The 3-year trailing averages for gold prices through July 2019 is $1,274.89/oz. In line with this data, GRE used a gold price of $1,300/oz for the base case analyses.

The projected metal recovery rates were:

- Crushed material: 82%
- ROM: 50%

Recovery was factored as 70% during the first year after placement on the leach pad, 25% during the second year after placement on the leach pad, and 5% during the third year after placement on the leach pad. Base case gold revenues are summarized in Table 22-1.

Table 22-1: Summary of Gold Revenues (millions)

Revenue	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Gold Crush	$0.00	$89.84	$119.07	$111.35	$116.24	$126.68	$38.67	$6.54	$608.39
Gold ROM	$0.00	$16.28	$20.54	$32.36	$27.83	$15.54	$4.65	$0.77	$117.97

The project does not have overriding royalties.

GRE included depreciation and depletion deductions from the income before taxes to obtain taxable income. Depreciation for the base case was assumed over 5 years, capturing most of the depreciable mining capital. Depletion allowance was calculated as 15% of revenues up to a maximum of 50% of before-tax income minus depreciation. Federal tax at 21% was applied to the taxable income, Idaho corporate tax at 6.925% was applied to the taxable income, Idaho License tax at 1% was applied to net revenue, and Idaho property tax at 0.78% was applied to annual net profit. The taxes were deducted from the taxable income, then the depreciation and depletion allowance were added back from taxable income to obtain net cash flows after taxes. After-tax net present value (NPV) @5%, NPV@7%, NPV@9%, and internal rate of return (IRR) were calculated from the net after-tax cash flow. Table 22-2 summarizes the economic model, and Table 22-3 shows the after-tax NPV and IRR results.

Table 22-2: Summary of Economic Model

Description	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Net Revenue	$0.00	$106.12	$139.60	$143.72	$144.07	$142.22	$43.31	$7.31	$0.00	$726.36
Total Operating Costs	$2.18	$79.59	$80.31	$122.31	$88.93	$62.26	$0.00	$0.00	$0.00	$435.57
Before Tax Cash Flow	($2.18)	$26.54	$59.29	$21.41	$55.15	$79.96	$43.31	$7.31	$0.00	$290.79
Depreciation	$0.00	$12.75	$13.52	$13.88	$16.14	$16.50	$5.14	$0.00	$0.00	$77.93
Loss Carry Forward	$0.00	($2.18)	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00

Description	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Depletion Allowance	$0.00	$6.89	$20.94	$3.76	$19.51	$21.33	$6.50	$1.10	$0.00	$80.03
Taxable Income	$0.00	$4.71	$24.83	$3.76	$19.51	$42.12	$31.68	$6.21	$0.00	$132.83
Total Taxes	$0.00	$1.72	$7.78	$1.39	$6.23	$12.97	$9.55	$1.85	$0.00	$41.51
After Tax Cash Flow	($2.18)	$24.82	$51.51	$20.02	$48.92	$66.99	$33.76	$5.45	$0.00	$249.28
Total Capital Costs	$81.23	$1.82	$1.82	$11.27	$1.82	$8.94	$6.00	$0.00	($15.46)	$97.46
Net Cash Flow	($83.41)	$22.99	$49.69	$8.74	$47.09	$58.04	$27.76	$5.45	$15.46	$151.82

Table 22-3: Summary of Economic Results

After-tax NPV@5% (millions)	After-tax NPV@7% (millions)	After-tax NPV@9% (millions)	After-tax Cash Flow (millions)	After-tax IRR	Payback Years
$110.39	$96.82	$84.64	$151.82	34.04%	3.0

The economic model indicates favorable after-tax NPV@5% of $110.39 million with an IRR of 34.0%, at a gold price of $1,300/oz.

The tax calculation should be considered conservative, without consideration for exploration costs incurred, corporate tax philosophy, losses carried forward, and other factors.

Key economic measurements include the following:

Royalties:	0%
Undiscounted Operating Pre-Tax Cash Flow:	$193.3 million
Pre-tax NPV@5%:	$144.0 million
Pre-tax NPV@7%:	$127.9 million
Pre-tax NPV@9%:	$113.4 million
Pre-tax IRR:	40.6%
After-tax NPV@5%:	$110.4 million
After-tax NPV@7%:	$96.8 million
After-tax NPV@9%:	$84.6 million
After-tax IRR:	34.0%
Undiscounted Operating After-tax Cash Flow:	$151.8 million
After-tax Payback Period:	3.0 years
All-in Sustaining Costs:	$832/Au ounce
All-in Costs:	$954/Au ounce
Total Operating Costs:	$780/Au ounce

22.1 Sensitivity Analyses

GRE evaluated the after-tax NPV@7% and IRR sensitivity to changes in gold price, capital costs, and operating costs. The results are shown in Table 22-4, Figure 22-1, Table 22-5, and Figure 22-2, respectively.

Table 22-4: Kilgore Project NPV@7% Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

Variable	NPV@7% at % Of Base Case (millions)				
	80%	90%	100%	110%	120%
Capital Cost	$114.35	$105.71	$96.82	$87.68	$78.28

Operating Cost	$152.78	$125.03	$96.82	$67.05	$39.72
Gold Price	$4.31	$48.34	$96.82	$143.00	$188.71

Figure 22-1: NPV@7% Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

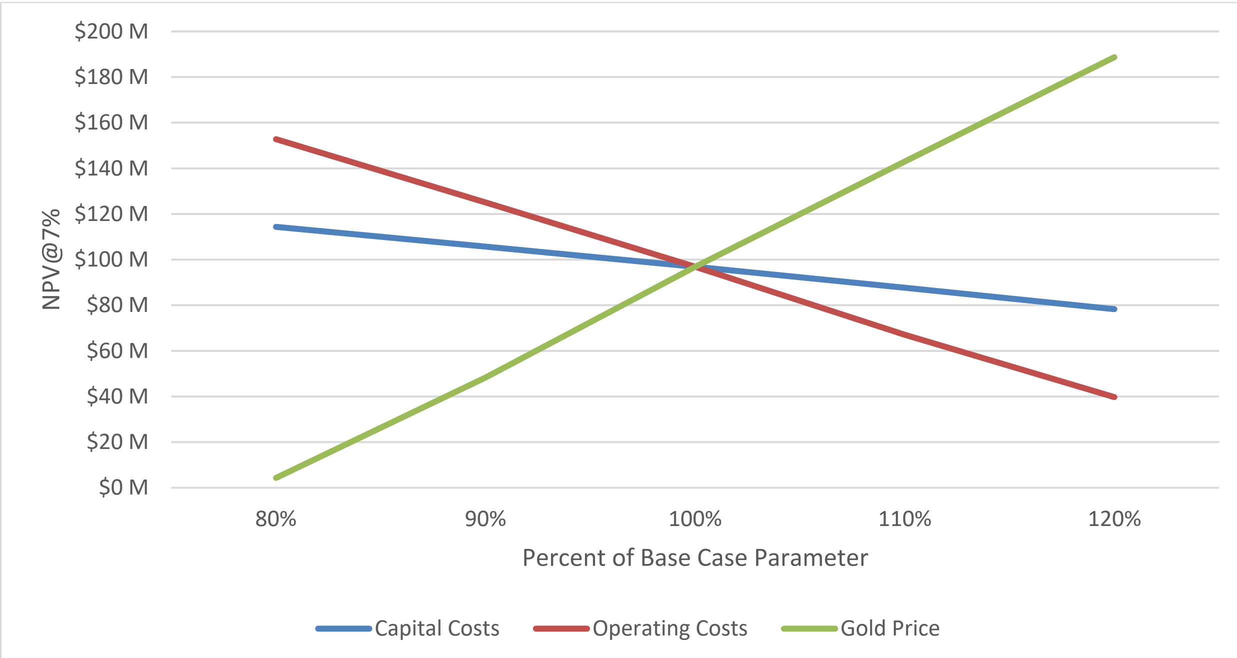
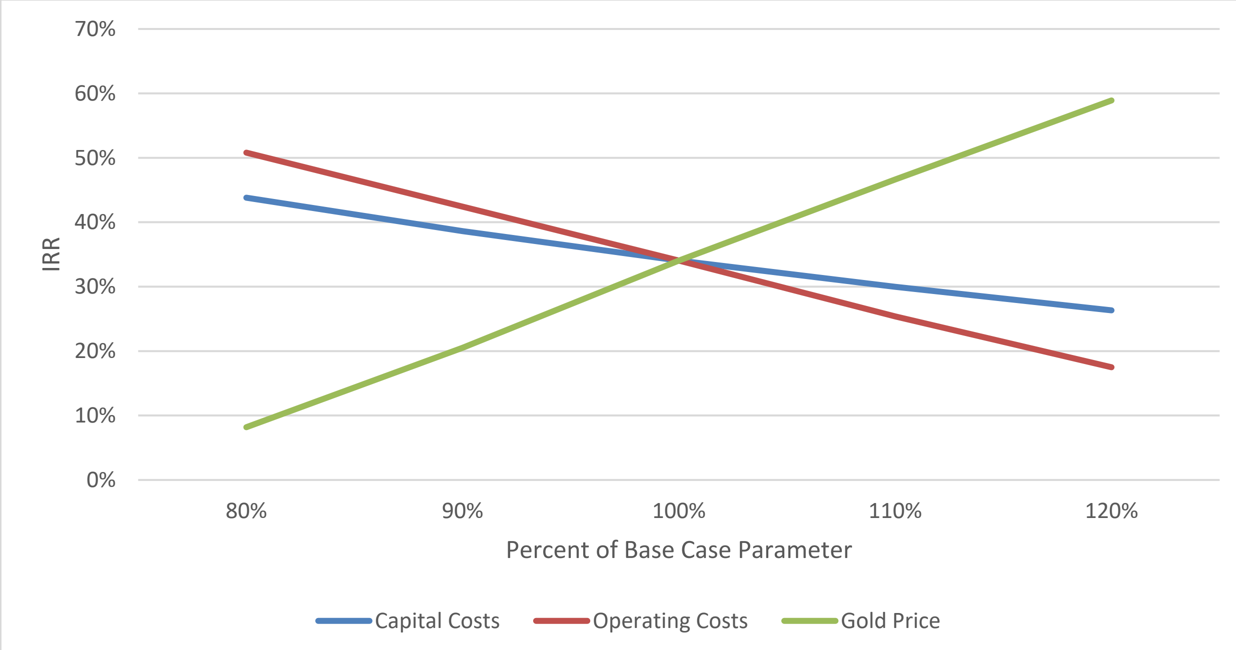


Table 22-5: Kilgore Project IRR Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs

Variable	IRR at % Of Base Case				
	80%	90%	100%	110%	120%
Capital Cost	43.81%	38.61%	34.04%	29.98%	26.33%
Operating Cost	50.80%	42.42%	34.04%	25.38%	17.50%
Gold Price	8.19%	20.53%	34.04%	46.65%	58.90%

Figure 22-2: IRR Sensitivity to Varying Gold Price, Capital Costs, and Operating Costs



The results in Figure 22-1 indicate:

- After-tax NPV@7% is least sensitive to changes in capital costs and stays positive for the full range of capital costs examined.
- After-tax NPV@7% is moderately sensitive to changes in operating costs and stays positive for the full range of operating costs examined.
- After-tax NPV@7% is most sensitive to changes in gold price and stays positive for the full range of gold prices examined.

The results in Figure 22-2 show that:

- After-tax IRR is least sensitive to changes in capital costs.
- After-tax IRR is moderately sensitive to changes in operating costs.
- After-tax IRR is most sensitive to changes in gold price.

A positive valuation is maintained across a wide range of sensitivities on key assumptions.

22.2 CONCLUSIONS OF ECONOMIC MODEL

The project economics shown in this PEA are favorable, providing positive NPV at varying gold prices, capital costs, and operating costs.

23.0 ADJACENT PROPERTIES

There are no surrounding mineral properties adjacent to Kilgore.

24.0 OTHER RELEVANT DATA AND INFORMATION

Section 27, References, provides a list of documents that were consulted in support of this report. No further data or information is necessary, in the opinion of the authors, to make the Report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Interpretation

The Kilgore Project is located in the northeastern portion of the ESRP, locally situated to the south of the Centennial Mountains, and regionally along the northern margin of the Miocene-Pliocene Heise Volcanic Field. The ESRP is an arcuate depression of low topographic relief that extends more than 500 km across southern Idaho (Figure 7-1). The plain is distinguished from the surrounding terrain by lower elevation and surface relief and by a complete cover of Cenozoic sedimentary and volcanic rocks. Geologic relationships and recent radiometric dating have demonstrated that since middle Miocene time the ESRP-Yellowstone Plateau province has been characterized by voluminous bimodal rhyolite and basalt volcanism that has progressed eastward with time and is now focused at Yellowstone National Park. The development of this eastward younging bimodal volcanism is attributed to west-southwestward movement of the North American plate over a stationary melting anomaly, or plume-like zone of hot and molten magma rooted at least several hundred km below the surface (Leeman, 1982), commonly referred to as the Yellowstone hotspot.

Gold mineralization at Kilgore occurs in two genetically related but distinct hosts of mineralization. The near surface mineralization is hosted within rocks of volcanic or subvolcanic origin, including the Tlt and the sub-vertical felsic dikes, dike swarms, and granodioritic bodies that intrude it. Locally concentrated mineralization is known to occur within the Tlt in association with sub-vertical fissures and fault zones and along lithologic contacts of flows, dikes, and sills within the Tlt, and between the Tlt and the Aspen Formation. The sedimentary rocks of the Aspen Formation host additional and potentially significant mineralization, one which is characterized by a low-grade, bulk-mineable type distribution with an overall higher average grade than the volcanic hosted mineralization. It's relationship to fault and or feeder structures is not clearly understood but has been demonstrated by drilling to be found at significant depths below the volcanic – sedimentary rock contact.

Gold mineralization in the volcanic and related intrusive rocks is generally higher grade as a result of weak to moderate vein development and open space fracture-fill, together within a broad, low grade halo of disseminated gold within variably silicified and argillically altered rocks. Gold content appears to decrease rapidly to lower grades (less than 50 to 100 ppb Au), with corresponding decrease in quartz or quartz-adularia as silicification and increase in argillic alteration. Exceptions occur in strongly oxidized rock near the topographic surface where strong to pervasive iron-oxide, yellow-orange to brown staining is accompanied by high gold grades. Mineralization in the volcanic and associated intrusive rocks accounts for an estimated 70% of the known mineral resource, with the remaining 30% occurring in the underlying Aspen Formation.

The 2017 drilling results extended mineralization in the Aspen Formation up to 300 meters deeper than was previously known, largely in the central part of the deposit southeast of the Mine Ridge Fault (Figure 10-5 through Figure 10-7) and north of the Cabin Fault. Average grades in this area are generally higher than the overall average grade of the Kilgore deposit as reported in this report, and mineralization appears to be fairly continuous between holes within sections and from section to adjacent section. The addition to depth of the mineralization within the Aspen Formation sediments represents an opportunity to significantly expand the size of the resource at Kilgore.

The Kilgore deposit is subdivided into five estimation domains based on host rock types and the modeled extents of the gold mineralization using a 0.1 g/T Au threshold. Modeling of the gold zone was controlled by the gold grade values from drilling while respecting the geologic and structural trends identified for the deposit, the northwest trending fault system. The gold zone boundary was treated as a hard boundary during block model estimation, and samples outside the boundary were excluded from the gold grade estimation. Rock type contacts are also treated as hard boundaries during the grade estimation.

The estimation domains used in the resource evaluation are:

- 3Tpr Biotite Rhyolite
- 4Tqp Rhyolite Quartz Porphyry
- 5Tad Sills and Dikes of Intermediate Composition
- 6Tlt Undifferentiated Tuff
- 7Ka Aspen Formation

These are modeled separately because they appear to deport differently with geostatistical analysis.

The majority of the deposit is amenable to open pit mining with heap leach recovery of gold and silver.

25.2 Conclusions

The authors have reviewed the information supplied by Otis Gold for the Kilgore Project and have found it to be reasonable in the context in which it is used herein. The authors have modified geological interpretations to some degree where deemed appropriate and necessary to complete the resource estimation and have applied their independent judgment in the application of Otis Gold information to the resource estimate and PEA.

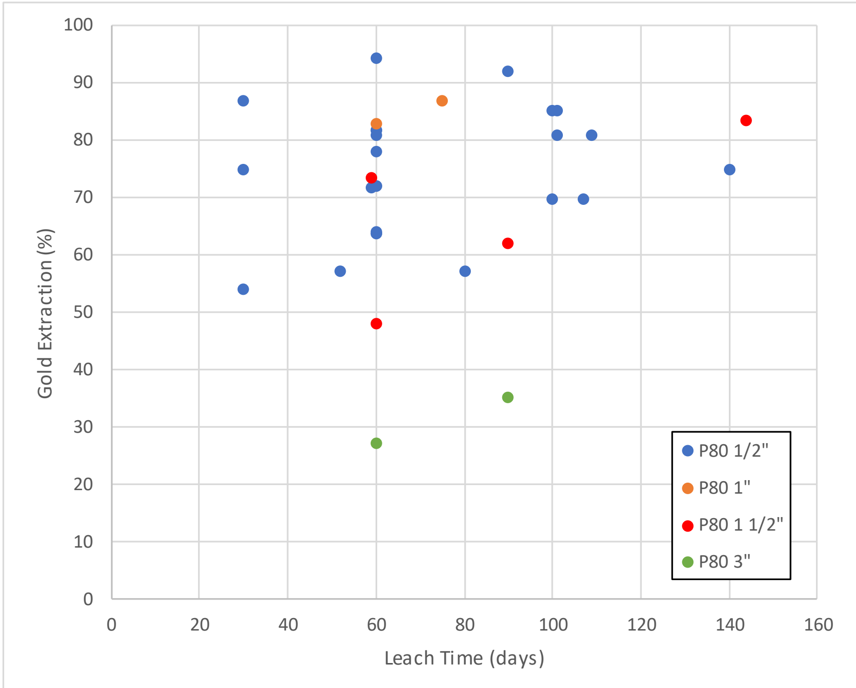
Kilgore is an epithermal, volcanic- and sediment-hosted vein-fracture to disseminated, structure-controlled gold mineralized system in a caldera environment. The deposit is emplaced beneath a silicified cap that was at, or close to, the paleosurface at the time of formation of the gold deposit. Near surface gold was deposited in the cover of Tertiary volcanic and subvolcanic rocks that were emplaced unconformably on Cretaceous sedimentary rocks. Drilling has indicated that the mineralization is potentially significant and extensive within the Aspen Formation sediments; these are now known to extend up to 300 meters below the unconformable contact with the overlying volcanic rocks.

In general, the Otis Gold deposit is characterized by materials that are highly amenable to heap leaching. In many cases, the ore benefits from a finer crush size both in terms of gold recovery and extraction kinetics. One area of the deposit, identified as Aspen (2018-169), had significantly higher grades but also the presence of “pregnant solution-robbing” carbon. This material when treated by CIL had excellent gold extractions. This material is not amenable to heap leaching in its current form.

The Kilgore deposit samples had a moderate abrasion index and a moderate crushing index.

Figure 25-1 shows the summary of all the gold extractions for all column leach tests to date. Both the mid-point (generally 60 days) and final leach period have been plotted.

Figure 25-1: Summary of Column Leach Gold Extractions by Leach Time and Crush Size



As shown by Figure 25-1, the ore shows a fairly high degree of variability at coarser crush sizes but less at the finer crush size of P_{80} ½-inch. Gold extractions tend to be excellent in most cases at periods of 90 days or more for the ½-inch material.

RC and core assays do not compare well with paired data comparisons and separate estimates showing that RC assay samples are generally higher than core. Issues are identified with both types of data, which will not be fully resolved without collecting bulk samples. The various operators of the project have been alerted to recovery and sampling issues and appear to have taken measures to reduce sample bias, reflected in the core drilling techniques used. A bulk sample testing program should be designed, including potential development of an underground bulk sample operation.

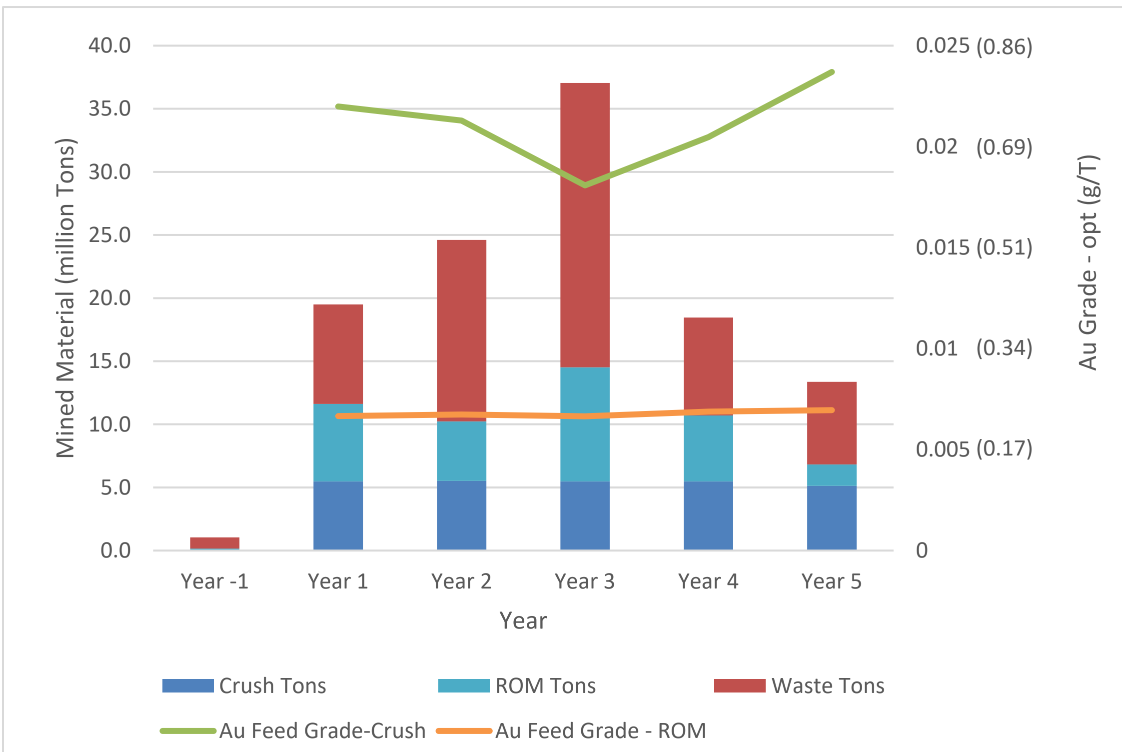
The individual domain resource estimates are generally contiguous and form a body of mineralization potentially amenable to bulk tonnage mining in an open pit setting. This appears to be supported by the metallurgical studies performed to date by previous companies and Otis Gold. The estimated mineral resources for the Kilgore Project (Table 14-8) conform to standards set forth in NI 43-101 for Indicated and Inferred mineral resources.

There is no assurance that mineral resources will be converted into mineral reserves. Mineral resources are subject to further dilution, recovery, lower metal price assumptions, and inclusion in a mine plan to demonstrate economics and feasibility of extraction.

Exploration in and around the Kilgore Project reveals a large area of hydrothermal alteration that resulted from the geothermal system generated by the magmatism and volcanism associated with the Heise Volcanic Field. Multiple super-volcanic eruptions created both the host rocks and the structural environment to allow precious metal-bearing fluids to be emplaced throughout the Kilgore Project area. Continuing exploration work conducted by Otis Gold has demonstrated gold occurrences over the entire land package currently held. Regional exploration including stream sediment and soil sampling in combination with surface geologic mapping are valuable in identifying further near surface precious metal epithermal style mineralization.

For purposes of costing and economic modeling, GRE considered two designed pits derived at gold prices of \$800/oz Au and \$900/oz Au, both with and without the Aspen material being mined. Subsequent cost modeling and economic analyses examined two processing options: processing of all material above the mining cutoff grade with crushing; and processing of material between 0.004 opt and the mining cutover grade as ROM and material above the mining cutover grade with crushing. Mining cutoff/cutover grades of 0.006, 0.007, 0.008, 0.009, and 0.010 opt were evaluated. The \$800/oz Au pit with Aspen material included at a cutofver grade of 0.010 opt (0.34 g/T) with a ROM cutoff grade of 0.004 opt (0.14 g/T) was selected as the base case for this study. This pit has life-of-mine leach material of 54.0 million tons at a strip ratio of 1.1:1. The mine production is summarized in Figure 25-2.

Figure 25-2: Mine Production Summary



The analysis assumed a heap leach process with an ADR plant to recover gold.

The economic model used a base gold price of \$1,300/oz, which is commensurate with the 3-year trailing average through June 2019, and results in total revenue of \$726.36 million.

The life of mine capital costs were \$97.46 million, with initial capital costs of \$81.23 million. Operating costs were \$8.06 per ton of leachable material.

Depreciation of capital costs and depletion allowance were deducted from the income before taxes, depreciation, and depletion to determine taxable income. Federal Income tax of 21%, Idaho corporate tax of 6.925%, Idaho license tax of 1%, and Idaho property tax of 0.78% were applied to the taxable income, generating income after tax. The depreciation and depletion allowance were then added back from taxable income to determine the final net cash flows after taxes. The total after-tax cash flow over the life of the project was estimated to be $151.8 million. The resulting NPV@5% was $110.4 million, the NPV@7% was $96.8 million, the NPV@9% was $84.6 million, the IRR was 34.0%, and the payback period was 3.0 years.

Key economic measurements include the following:

Royalties:	0%
Undiscounted Operating Pre-Tax Cash Flow:	$193.3 million
Pre-tax NPV@5%:	$144.0 million
Pre-tax NPV@7%:	$127.9 million
Pre-tax NPV@9%:	$113.4 million
Pre-tax IRR:	40.6%
After-tax NPV@5%:	$110.4 million
After-tax NPV@7%:	$96.8 million
After-tax NPV@9%:	$84.6 million
After-tax IRR:	34.0%
Undiscounted Operating After-tax Cash Flow:	$151.8 million
After-tax Payback Period:	3.0 years
All-in Sustaining Costs:	$832/Au ounce
All-in Costs:	$954/Au ounce
Total Operating Costs:	$780/Au ounce

A positive valuation is maintained across a wide range of sensitivities on key assumptions.

The project economics shown in this PEA are favorable; continued exploration and development is warranted.

25.3 Risk

This PEA is preliminary in nature and includes inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves under National Instrument 43-101. Readers are advised that there is no certainty that the results projected in this preliminary economic assessment will be realized. There is inherent engineering and metallurgical risk in all projects at this stage of development, which are reduced as metallurgical test work and engineering studies, including the prefeasibility and feasibility studies, progress. This project, as with others, would be negatively impacted should metallurgical recoveries prove to be less than that used in the study or positively impacted if recoveries are higher. Similarly, if further rock mechanics or other studies lead to changes in the inputs to this study, the project could be impacted positively or negatively. It is unlikely that changes will impact the NPV by more than 20%, with the largest likely impact being future changes in the price of gold.

26.0 RECOMMENDATIONS

The coordinate system for the project is a local Imperial coordinate system that may have served its purpose at the early stages; some surveying has been conducted in this local coordinate system while others have been performed in UTM NAD83. Some of the drill holes cannot be re-located and have been adjusted to local coordinates mathematically from UTM. The assays from most of the drilling campaigns are reported in ppm, ppb, or g/T units, but much work is performed using ounces per short ton because of the Imperial coordinate units. GRE recommends adding the UTM coordinates to the drill hole database and using that grid and metric measurements for all project work going forward, including the construction of the geologic models that support the resource estimation. This will reduce the possibility of conversion errors and facilitate reporting to international standards.

The test work conducted to date on the Kilgore Project has indicated that there are two distinct mineral hosts in the deposit: free milling gold and a more recalcitrant mineral host that is finer grained and contains active carbon. The free milling gold has been shown to respond well to heap leach testing, and the more recalcitrant material showed good gold extraction when subjected to grinding and CIL leaching. The more recalcitrant material tends to have a higher gold grade and should be able to support a conventional milling/CIL process provided the tonnage justifies this processing method.

Based on these findings the following recommendations have been presented:

- Continue drill testing the near surface potential of the deposit by drilling to north, south, and west where it remains open including fracture / fault studies to better define the relationship between mineralization and structure, and oriented and geotechnical drilling to assist in mine design studies.
- Assay existing pulps for silver and include silver assays in all new exploration sampling.
- Continue drill testing the lateral and vertical extent of the sediment hosted gold mineralization in the Aspen Formation.
- Drill 3-5 core holes for metallurgical test work including large diameter holes to test ROM potential in the lithic tuff and sill.
- Relog and analyze Aspen Formation intervals to identify good and poor-performing metallurgical domains.
- Ensure that all subsequent metallurgical analysis on new samples utilizes cyanide amenability tests (P_{80} of 10 mesh with a 96-hour bottle roll leach) to define the direction for subsequent testing. This will establish a database of amenability tests for future geometallurgy.
- Ensure that complete carbon assays are undertaken on all mineral domains.
- Quantify the ore types tested to determine the relative abundance of each domain and map the recoveries to those domains.
- Explore the crush size relationship in more detail to allow for the optimization of the ultimate heap leach design.
- A detailed analysis of the material tested and its representativity to the deposit should be conducted to ensure adequate grade, material type and spatial representativity.

- Increase the tracking of silver in subsequent metallurgical testing and inform the model with additional silver assay details.
- More geotechnical investigation should be undertaken to ensure heap permeability under a multiple lift scenario.
- Based on the metallurgical review, a conservative approach has been taken to ensure maximum gold and silver recovery is obtained.
    - A crush size P_{80} of ½-inch has been selected for the heap design.
    - A primary leach period of 90 days should be employed. Based on these parameters, gold extractions of 82% for crush and 50% for ROM should be achievable.
    - Cyanide and lime consumptions are moderate. The cyanide consumption has been scaled from the average for all column tests of 2.16 pounds per ton (lb/t) to a projected heap consumption of 0.5 lb/t. The average lime consumption from the column tests has been employed (with the removal of one outlier) to provide an expected consumption of 2.6 lb/t.
    - No agglomeration is necessary as the column tests all exhibited excellent permeability.
    - The silver grade does not appear to be high enough to warrant the use of a Merrill Crowe recovery system. A standard carbon adsorption circuit should be acceptable.
- A Lidar survey will be needed for mine, waste rock, heap leach, and plant facility designs.
- Base line surface water flows and water quality will be needed for design.
- Groundwater monitoring and testing wells will be needed to create a groundwater model and predict pit inflows and pit dewatering requirements.
- Along with the geologic description of core holes, they should be logged, and RQD and rock mass rating should be identified.
- Geotechnical testing of soils near the plant, waste rock, and heap leach sites.
- Geotechnical testing of rock for pit wall design.
- Continue Kilgore Project wide exploration to test for emerging targets both inside and outside the existing land position.
- The recommended budget is $4.4 million as shown in Table 26-1.

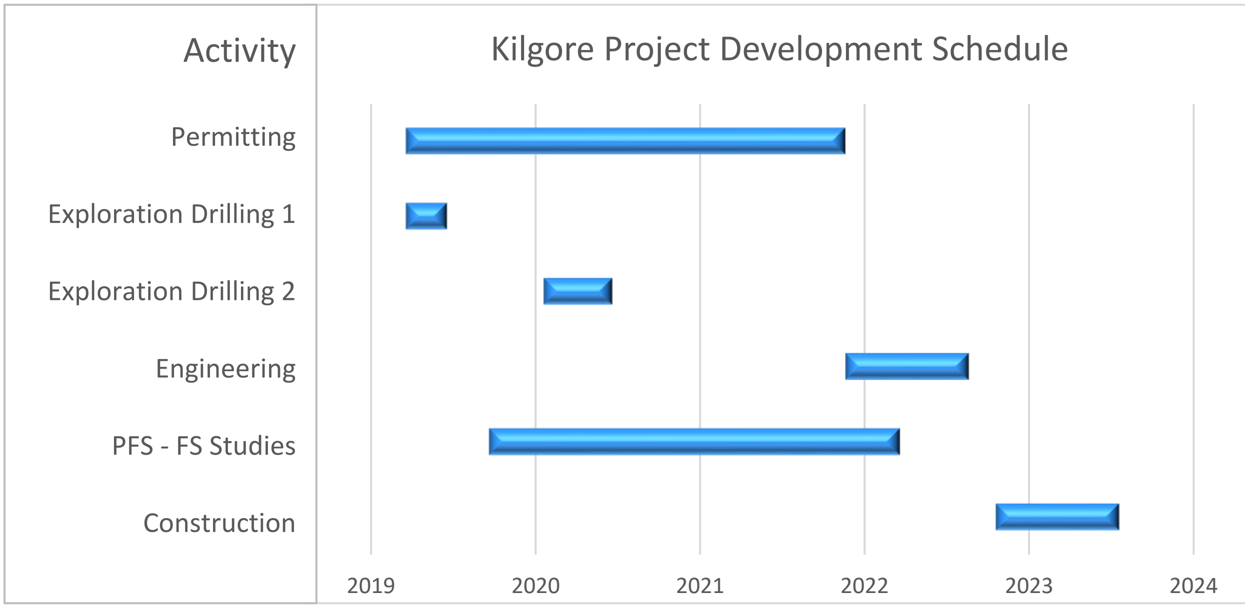
Table 26-1: Proposed Budget for Kilgore Project

Drilling – Exploration & development	7,500 m	$200/m	$1,500,000
Drilling – metallurgical and water monitoring	2,500 m	$200/m	$500,000
Large diameter bulk samples	600 m	$1,000/m	$600,000
Soils testing			$196,000
Geologic mapping			$140,000
Core studies			$50,000
LiDAR survey			$75,000
Baseline Studies			$250,000
Office Rent			$36,000
Bonding			$371,000
Annual Claim Maintenance Payments			$118,000

Bulk Sample Column Testing			$500,000
Data management			$105,000
Total			**$4,441,000**

The estimated schedule for completing development work on the project is shown in Figure 26-1.

Figure 26-1: Kilgore Project Development Schedule



27.0 REFERENCES

Allmendinger, R. (1982). Sequence of Late Cenozoic Deformation in the Blackfoot Mountains, Southeastern Idaho. (B. Bonnichsen, & R. Breckenridge, Eds.) *Cenozoic Geology of Idaho, Idaho Bur. Mines and Geol. Bull. 26*, pp. 505-516.

Armstong, R., Harakel, J., & Neill, W. (1980). *K-Ar Dating of the Snake River Plain Idaho Volcanic Rocks - New Results: Isochron West, no. 27.*

Benson, C. (1986). *Geology of the Kilgore Deposit, Clark County, Idaho: University of Idaho M.S. Thesis, Moscow, Idaho.*

Berger, B., & Eimon, P. (1982). Conceptual Models of Epithermal Precious Metal Deposits. *AIME Preprint No. 82-13, SME-AIME mtg.* Dallas, Texas.

Berger, C. (1985). Geology-Geochemical Features of Hot-Spring Precious-Metal Deposits. (E. Tooker, Ed.) *Geologic Characteristics of Sediment and Volcanic-HostedDisseminated Gold Deposits - Search for an Occurrence Model: U.S. Geol. Survey Bull. 1646*, pp. 47-54.

Buchanan, L. (1981). Precious Metal Deposits Associated with Volcanic Environments in the Southwest. (W. Dickinson, & W. Payne, Eds.) *Relations of Tectonics to Ore Deposits in the Southern Cordillera: Ariz. Geol. Soc. Digest, v. 14*, pp. 237-262.

Caddey, S. (2003). *Preliminary Structural Investigation and Identification of Exploration Target Areas, Kilgore Gold Project, Southeast, Idaho: Report for Kilgore Gold Ltd.*

Cameron, D. (2012). *Technical Report and Resource Estimate for the Kilgore Project, Clark County, Idaho, U.S.A.: NI 43-101 Technical Report, July 20.*

Campbell, A. (1937). *Thirty-ninth Annual Report of the Mining Industry of Idaho for the Year 1937: Idaho Bur. Mines and Geol.*

Campbell, A. (1937). Thirty-ninth Annual Report of the Mining Industry of Idaho for the Year 1937: Idaho Bureau of Mines and Geology. p. 146.

CIM. (2014). *Definition Standards for Mineral Resources and Mineral Reserves.* CIM Standing Committe on Reserve Definitions.

Echo Bay Exploration Inc. (1996). *1996 Summary Report: Kilgore Project, Clark County, Idaho.*

Echo Bay Exploration, Inc. (1995). *1995 Summary Report for Placer Dome U.S., Inc., Kilgore Gold Project, Clark County, Idaho (January 18).*

Embree, G., McBroome, L.A., & Doherty, D. (1982). Preliminary Stratigraphic Framework of the Pliocene and Miocene Rhyolite, Eastern Snake River Plan, Idaho. (B. Bonnichsen, & R. Breckenridge, Eds.) *Cenozoic Geology of Idaho: Idaho Bureau of Mines and Geology Bulletin 26*, pp. 333-343.

Golder Associates. (2010). *Kilgore Mine Preliminary Enviuronmental Scoping Report.*

GRE & Rowearth. (2018). *Independent Technical Report and Mineral Resource Estimate for the Kilgore Project, Clark County, Idaho, U.S.A., August 14.*

Hazen Research Inc. (1995). *Kilgore, Idaho, Gold Ore Characterization Study: Report to Echo Bay Mines.*

InfoMine. (2018). *Mine and Mill Equipment Costs, An Estimator's Guide.*

JBR Environmental Consultants. (1997). *Final Report Biological Baseline, Echo Bay Exploration Inc., Kilgore Exploration Project.*

Knopf, A. (1924). Geology and Ore Deposits of the Rochester Distric, Nevada. *U.S. Geol. Survey Bull. 762*.

Kuntz, M., & Dalrymple, G. (1979). *Geology, Geochronology, and Potential Volcanic Hazards of the Lava Ridge-Hills Half Acre Area, Eastern Snake River Plan, Idaho: U.S. Geological Survey Open-File Report 79.*

Larabee, B. (2012). *Identification and Analysis of Alteration Minerals Collected from Rock Cores from the Kilgore Mine, ID: Western Washington University Geol. Dept. Senior Thesis, Bellinham, WA.*

Leeman, W. (1982). Development of the Snake River Plain - Yellowston Plateau Province, Idaho and Wyoming: An Overview and Petrologic Model. (B. Bonnichsen, & R. Breckenridge, Eds.) *Cenozoic Geology of Idaho: Idaho Bur. Mines and Geol. Bull. 26*, pp. 155-177.

Love, T. (1986). *Geochemical Correlation of Salt Lake Equivalent Pyroclastic Deposits in Idaho and Wyoming: M.S. Thesis, New Orlean, U. New Orleans.*

Lupo, J. (2005). *Heap Leach Facility Liner Design.* Golder Associates, Inc., Lakewood, Colorado.

Mabey, D. (1982). Geophysics and Tectonics of the Snake River Plain, Idaho. (B. Bonnichsen, & R. Breckenridge, Eds.) *Cenozoic Geology of Idaho: Idaho Bur. Mines and Geol. Bull. 26*, pp. 139-153.

Mansfield, G. (1920). Coal in Eastern Idaho. *U.S. Geol. Survey Bull. 716-F*.

McClelland Laboratories, Inc. (2011). *Report on Heap Leach Cyanidation Testing - Kilgore Drill Core Composites (March 2).*

McKamy, R. (2011). *Quit Claim Deeds - Kilgore Gold Company to Otis Gold - County and BLM Filing.*

McKamy, R. (2012). *Opinion Concerning Upcoming Technical Report 43-101.*

McLelland Laboratories, Inc. (2012). *Report on Heap Leach Cyanidation Testing - Kilgore Drill Core Composites (August 9).*

Mitchell, V., & Bennett, E. (1979). Geologic Map of the Ashton Quadrangle, Idaho. *Idaho Bur. Mines and Geol., 2 Quadrangle Geologic Map Series, scale 1:250,000*.

Modroo, J. (2017). *Kilgore Ground Magnetic Survey & Exploration Synopsis, Kilgore, ID USA.*

Modroo, J. (2018). *2017 Kilgore Ground Magnetic Survey & Exploration Synopsis, Otis Gold Internal Report.*

Morgan, L. (1988). *Explosive Silicic Volcanism in the Eastern Snake River Plain, PhD Thesis, Univ. Hawaii-Manoa, Honolulu, HI.*

Morgan, L. (1992). Stratigraphic Relations and Paleomagnetic and Geochmical Correlations of Ignimbrites of the Heise Volcanic Field, Eastern Snake River Plain, Idaho and Western Wyoming. (P. Link, M. Kuntz, & L. Platt, Eds.) *Regional Geology of Eastern Idaho and Western Wyoming: Geol. Soc. of America Memoir 179*, pp. 215-226.

Morgan, L., & MacIntosh, W. (2005). Timing and Development of the Heise Volcanic Field, Snake River Plain, Idaho, U.S.A. *Geol. Soc. of American Bull., 117*(3/4), pp. 288-306.

Morgan, L., Doherty, D., & Leeman, W. (1984). Ignimbrites of the Eastern Snake River Plain: Evidence for Major Caldera-Forming Eruptions. *Jour. Geophys. Res., 89*(B10), pp. 8665-8678.

Morgan, L., Shanks, W., & Pierce, K. (2009). Hydrothermal Processes Above the Yellowstone Magma Chamber: Large Hydrothermal Systems and Large Hydrothermal Explosions. *Geological Society of America Special Paper 459*, 1-95.

Oriel, S., & Moore, D. (1985). *Geologic Map of the West and Easte Palisades Roadless Areas, Idaho and Wyoming: U.S. Geological Field Survey Miscellaneous Field Studies Map MF-1619-B, scale-1:50,000.*

Otis Gold Corp. (2012). *Otis Releases Complete Kilgore 2011 Drill Results; Mine Ridge Deposit Continues to Expand, News Release.* Retrieved from http://otisgold.com

Pancoast, L. (2004). *Summary - 2004 Kilgore Drill Program: unpublished report.*

Rayner, G.H. and Associates and Van Brunt, B.H. (2002). *Technical Report for the Kilgore Project (NI 43-101 Compliant Technical Report) .*

Resource Development Inc. (2019). *Leach Testing of Kilgore Aspen Samples, Otis Gold Corp (April 5).*

Robert, F., Brommecker, R., Bourner, B., Dobak, P., McEwan, C., Rowe, R., & Xhou, X. (2007). Models and Exploration Methods for Major Gold Deposit Types. (B. Milkeriet, Ed.) *Proceedings of Exploration 07: Fifth Decennial International Conference on mineral Exploration*, pp. 691-711.

Rytuba, J. (1994). Evolution of Volcanic and Tectonic Features in Caldera Settings and Their Importance in the Localization of Ore Deposits. *Econ. Geol., 89*, pp. 1687-1696.

Scholten, R., Keenmon, K., & Kupsch, W. (1955). Geology of the Lima Region, Southwestern Montana and Adjacent Idaho. *Gol. Soc. of America Bull., 66*, pp. 345-404.

Silberman, M. (1982). Hot Spring Type Large Tonnage, Low Grade Gold Deposits. *U.S. Geol. Surv. Open-File Report 82-795*, pp. 131-143.

Sillitoe, R., & Hedenquist, J. (2003). Linkages Between Volcanotectonic Settings, Ore Fluid Compositions,a nd Epithermal Precious Metal Deposits. *Soc. Econ. Geol. Special Pub. 10*, pp. 315-343.

Skipp, B., Prostka, H., & Schleicher, D. (1979). *Preliminary Geologic Map of the Edie Ranch Quadrangle, Clark County, Idaho and Beaverhead County, Monana: U.S. Geol. Surv. Open-File Report 79-845, scale 1:62,500.*

Watts, K., Bindeman, I., & Schmitt, A. (2011). Large-volume Rhyolite Genesis in Caldera Complexes of the Snake River Plain: Insights from the Kilgore Tuff of the Heise Volcanid Field, Idaho, with Comparison to Yellowstone and Bruneau-Jarbidge Rhyolites. *Journal of Petrology, 52*(5), pp. 857-890.

White, D. (1974). Diverse Origins of Hydrothermal Ore Fluids. *Econ. Geol., 69*, pp. 954-973.

Witkind, I., & Prostka, H. (1980). *Geologic Map of the Lower Red Rock Lake Quadrangle, Beaverhead and Madison Counties, Montana, and Clark County, Idaho: U.S. Geol. Surv. Mis. Geologic Inv. Map I-1216, scale 1:62,500.*

Woolham, R. (1996). *Report on a Combined Helicopter-borne Electromagnetic, Magnetic, and Radiometric Survey, Kilgore Gold Project, Clark County, Idaho.*

Wright, J. (2009). *Kilgore Gold Property CSAMT Survey.*

CERTIFICATE OF QUALIFIED PERSON

I, Terre A Lane, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled "Independent Technical Report and Preliminary Economic Assessment, Kilgore Project, Clark County, Idaho, USA" with an effective date of July 30, 2019 and an Issue date of August 26, 2019 (the "PEA"), DO HEREBY CERTIFY THAT:

1. I am a MMSA Qualified Professional in Ore Reserves and Mining, #01407QP and a Registered member of SME - 4053005.
2. I hold a degree of Bachelor of Science (1982) in Mining Engineering from Michigan Technological University.
3. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. My relevant experience for the purpose of this MRE is project management, mineral resource estimation, mine capital and operating costs estimation, and economic analysis with 25 or more years of experience in each area.
4. I have created or overseen the resource estimation, mine design, capital and operating cost estimation, and economic analysis of well over a hundred open pit projects.
5. I have been involved in or managed several hundred studies including scoping studies, prefeasibility studies, and feasibility studies.
6. I have been involved with the mine development, construction, startup, and operation of several mines.
7. I have read the definition of "Qualified Person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101.
8. I have not visited the property.
9. I am responsible for Sections 2, 3, 4, 5, 6, 10, 11, 12 15, 16, 18-24, and corresponding sections of the Summary, Other Relevant Data and Information, Interpretation and Conclusions, Recommendations and References that are related to these sections.
10. I am independent of Otis Gold as described in section 1.5 by National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1. The MRE has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.
12. As of the effective date of the Resource Estimate, to the best of my knowledge, information and belief, the Resource Estimate contains all scientific and technical information that is required to be disclosed to make the Resource Estimate not misleading.

Terre A. Lane

"Terre A. Lane"

Principal Mining Mining Engineer

Date of Signing: August 26, 2019

CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Todd Harvey, PhD, of 600 Grant St., Suite 975, Denver, Colorado, 80203, the co-author of the report entitled “Independent Technical Report and Preliminary Economic Assessment for the Kilgore Project, Clark County, Idaho, USA” with an effective date of July 30, 2019 and an Issue date of August 26, 2019 (the “PEA”), DO HEREBY CERTIFY THAT:

1. I am a Society of Mining Engineers (SME) Registered Member Qualified Professional in Mining/Metallurgy/Mineral Processing, #04144120.
2. I hold a degree of Doctor of Philosophy (PhD) (1994) in Mining and Mineral Process Engineering from Queen’s University at Kingston. As well as an MSc (1990) and BSc (1988) in Mining and Mineral Process Engineering from Queen’s University at Kingston.
3. I have practiced my profession since 1988 in capacities from metallurgical engineer to senior management positions for production, engineering, mill design and construction, research and development, and mining companies. My relevant experience for the purpose of this PEA is as the test work reviewer, process designer, process cost estimator, and economic modeler with 25 or more years of experience in each area.
4. I have taken classes in mineral processing, heap leach design, cost estimation and mineral economics in university, and have taken several short courses in process development subsequently.
5. I have worked in mineral processing, managed production and worked in process optimization, and I have been involved in or conducted the test work analysis and flowsheet design for many projects at locations in North America, South America, Africa, Australia, India, Russia and Europe for a wide variety of minerals and processes.
6. I have supervised and analyzed test work, developed flowsheets and estimated costs for many projects including International Gold Resources Bibiani Mine, Ashanti Goldfields Obuasi Mine, Equinox Gold Castle Mountain Mine, Cluff Resources Agnes Mine, and others, and have overseen the design and cost estimation of many other similar projects.
7. I have worked or overseen the development or optimization of mineral processing flowsheets for close to one hundred projects and operating mines, including gold heap leach and stirred tank gold leaching processes.
8. I have been involved in or managed many studies including scoping studies, prefeasibility studies, and feasibility studies.
9. I have been involved with the mine development, construction, startup, and operation of several mines.
10. I have read the definition of “Qualified Person” set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional organization (as defined in National Instrument 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of National Instrument 43-101.
11. I have not visited the project.
12. I am responsible for Sections 13 and 17 of the PEA and have contributed to Sections 1, 18, 25, 26, and 27.
13. I am independent of Otis Gold as described in section 1.5 by National Instrument 43-101.
14. I worked on the 2018 Technical Report for the Otis Gold Kilgore Project.
15. I have read National Instrument 43-101 and Form 43-101F1. The Resource Estimate has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1.

16. As of the effective date of the Resource Estimate, to the best of my knowledge, information and belief, the Resource Estimate contains all scientific and technical information that is required to be disclosed to make the Resource Estimate not misleading.

Jeffrey Todd Harvey, PhD
"Todd Harvey"
President and Director of Process Engineering
Global Resource Engineering, Ltd.
Denver, Colorado
Date of Signing: August 26, 2019

CERTIFICATE OF QUALIFIED PERSON

I, Jennifer J. Brown, P.G., of Hard Rock Consulting, LLC, 7114 W. Jefferson Ave., Ste. 313, Lakewood, Colorado, 80235, DO HEREBY CERTIFY THAT:

1. I am a graduate of the University of Montana and received a Bachelor of Arts degree in Geology in 1996.
2. I am a:
   - Licensed Professional Geologist in the State of Wyoming (PG-3719)
   - Registered Professional Geologist in the State of Idaho (PGL-1414)
   - Registered Member in good standing of the Society for Mining, Metallurgy, and Exploration, Inc. (4168244RM)
3. I have worked as a geologist for a total of 20 years since graduation from the University of Montana, as an employee of various engineering and consulting firms and the U.S.D.A. Forest Service. I have more than 10 collective years of experience directly related to mining and or economic and saleable minerals exploration and resource development, including geotechnical exploration, geologic analysis and interpretation, resource evaluation, and technical reporting.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I am a co-author of the report titled “Independent Technical Report and Preliminary Economic Assessment for the Kilgore Project, Clark County, Idaho, US” with an effective date of July 30, 2019 and an Issue date of August 26, 2019, with specific responsibility for Sections 7 through 9 and corresponding sections of the Summary, Other Relevant Data and Information, Interpretation and Conclusions, Recommendations and References that are related to these sections.
6. As of the date of this certificate and 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 required to be disclosed to make the report not misleading.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 26th day of August 2019.

Jennifer J. (J.J.) Brown, SME-RM
Printed name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, David Rowe, of PO Box 10087, Bainbridge Island, WA, 98110, a co-author of the report entitled "Independent Technical Report and Preliminary Economic Assessment Kilgore Project dated August 26, 2019, with an effective date of July 30, 2019 (the "Technical Report") prepared for Otis Gold, DO HEREBY CERTIFY THAT:

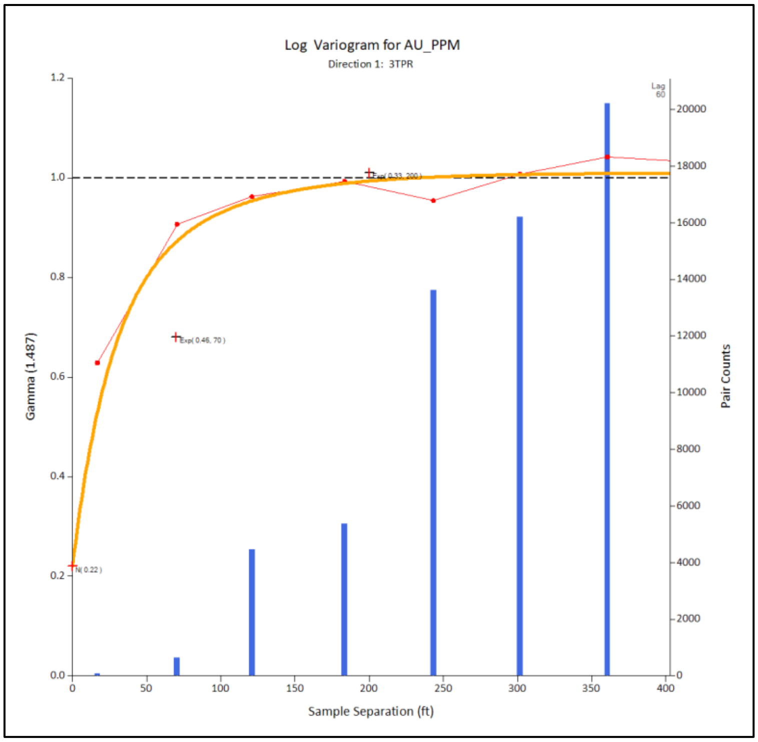
1. I am a Certified Professional Geologist registered with the American Institute of Professional Geologists, Certificate # 10953;
2. I hold a BA degree in Geology (1984) from the University of Montana and a Master of Science degree in Geology (1987) from the University of Wyoming;
3. I have practiced my profession continuously since 1987. I have been involved in mineral exploration and mineral resource estimation and consulting covering a wide range of mineral commodities in North America, Central America, the Caribbean, Africa, and Asia;
4. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience (as defined in National Instrument 43-101), I fulfill the requirements to be a "Qualified Person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101;
5. I personally inspected the subject property on August 9-14, 2017;
6. I am a co-author of this report and responsible for sections 12.5, and 14, which were extracted from the report entitled "NI 43-101 Technical Report on the Mineral Resource Estimate Technical Report of the Kilgore Project, Clark County, Idaho, USA" with an effective date of August 14, 2018 and an Issue date of September 28, 2018 (the "MRE"), and corresponding sections of the Summary and Conclusions that are related to these sections;
7. I did not review or compare the MRE block model used in this Technical Report to the one with an effective date of August 14, 2018. I was not involved in and I did not participate in the economic analysis or additional studies represented in this Technical Report;
8. I am independent of Otis Gold as defined in Section 1.5 of National Instrument 43-101;
9. I have no prior involvement with the subject property;
10. I have read National Instrument 43-101 and confirm that the MRE has been prepared in compliance with the National Instrument 43-101 and Form 43-101F1;
11. As of the effective date of the MRE, to the best of my knowledge, information and belief, the MRE contains all scientific and technical information that is required to be disclosed to make the MRE not misleading.

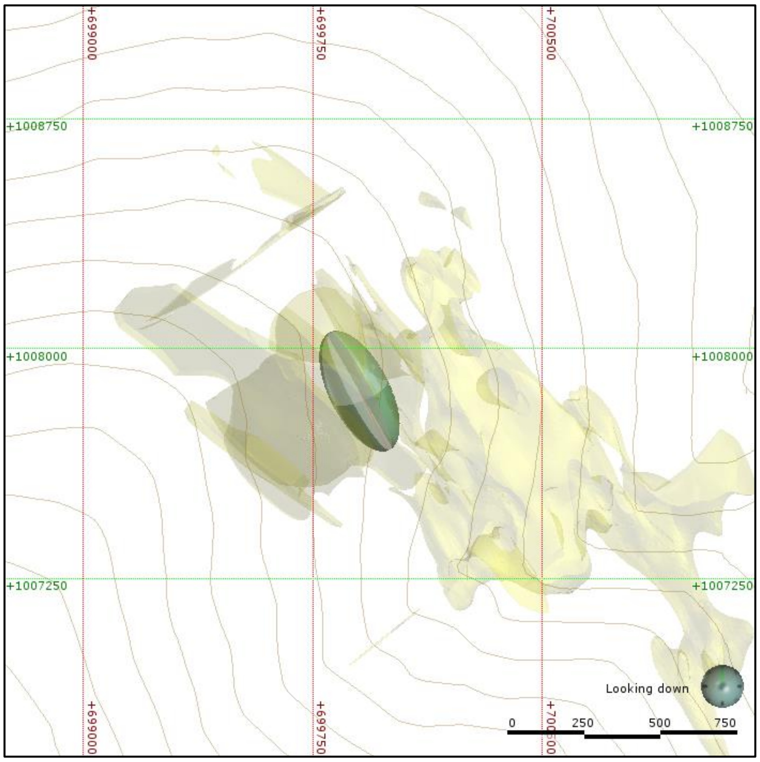
Original document dated August 26, 2019, *signed and sealed.*

David Rowe
Certified Professional Geologist, AIPG
Senior Resource Geologist, Rowearth LLC

AMERICAN INSTITUTE OF PROFESSIONAL GEOLOGISTS
CERTIFICATE NUMBER
10953
AIPG
DAVID ROWE
CERTIFIED PROFESSIONAL GEOLOGIST

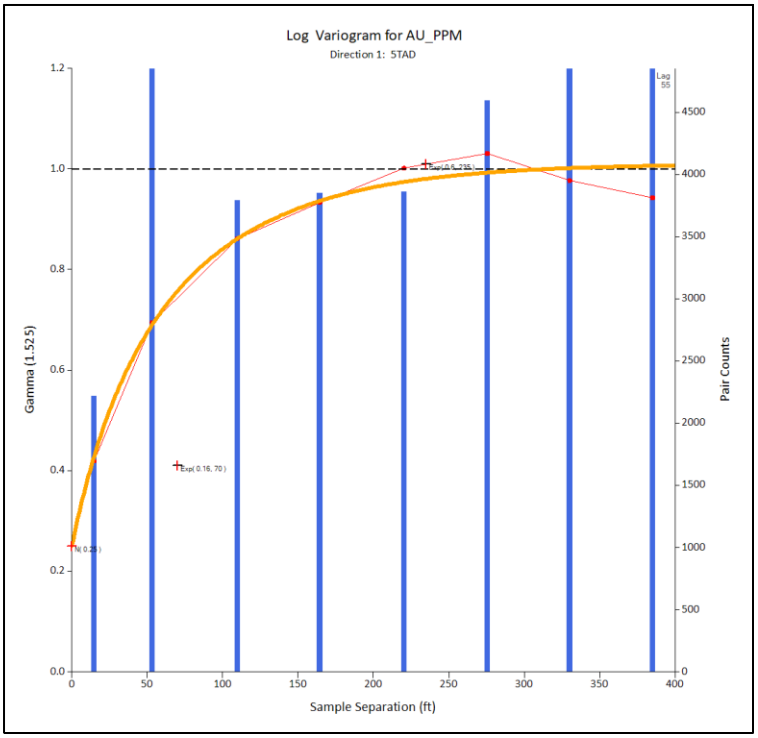
APPENDIX A VARIOGRAPHY FOR GOLD COMPOSITE SAMPLES

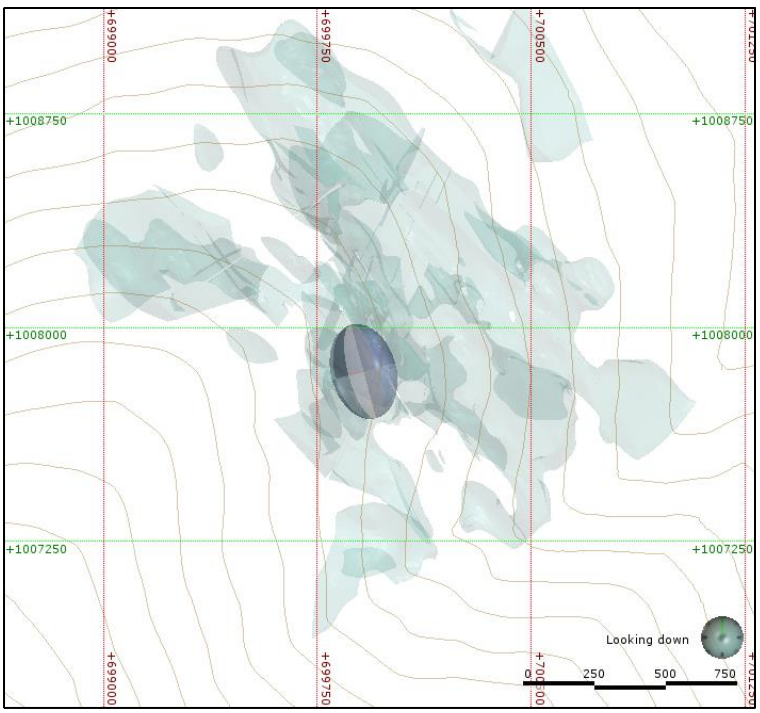




Top: major axis variogram model; Bottom: ellipsoid of variogram trend

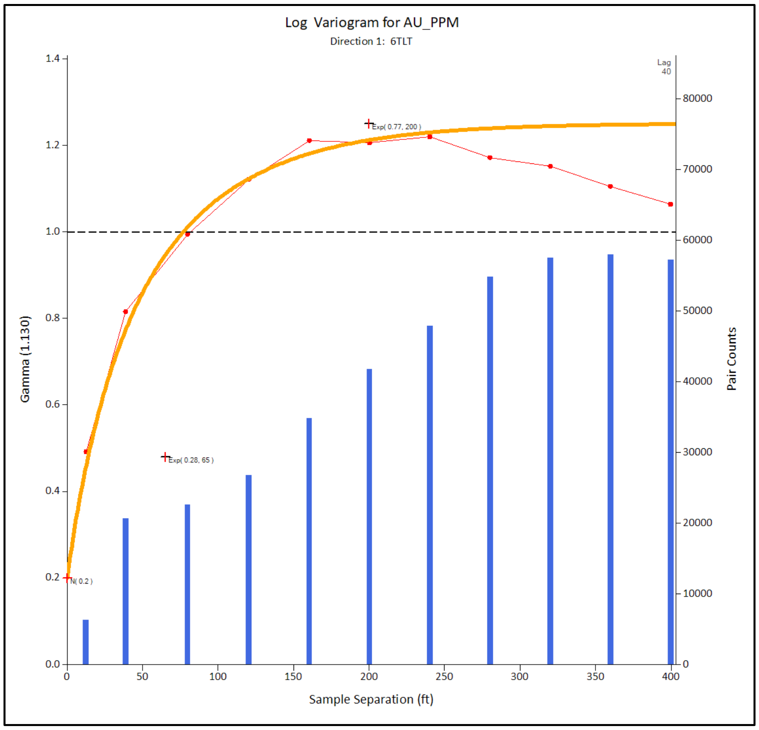
Variogram model for estimation Domain 3Tpr

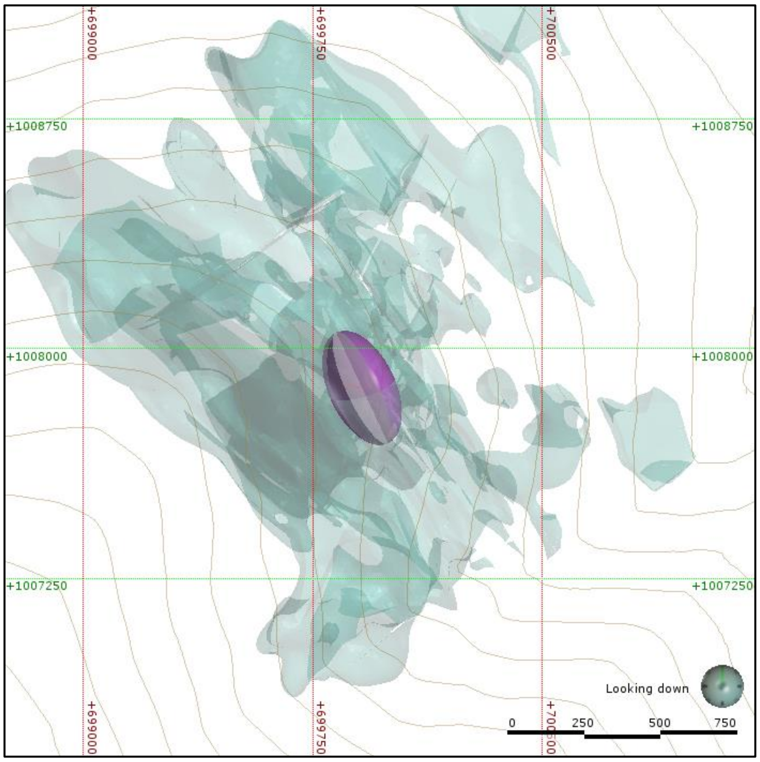




Top: major axis variogram model; Bottom: ellipsoid of variogram trend

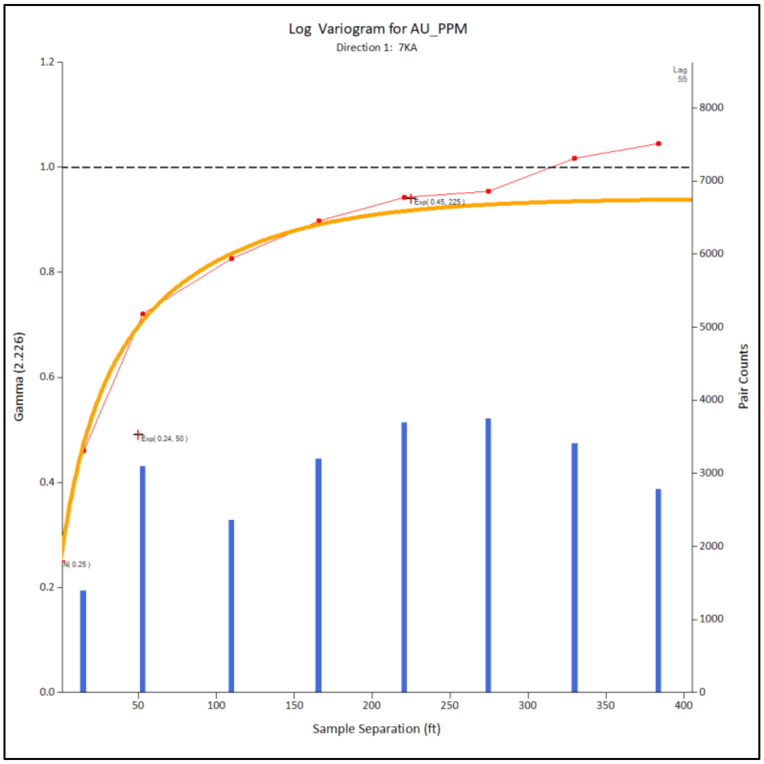
Variogram model for estimation Domain 5Tad

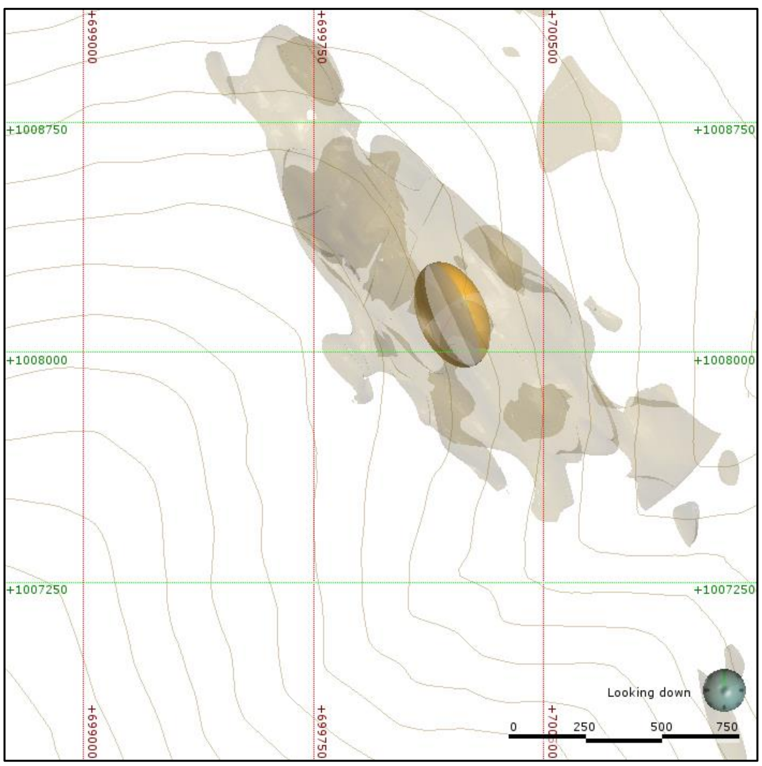




Top: major axis variogram model; Bottom: ellipsoid of variogram trend

Variogram model for estimation Domain 6Tlt





Top: major axis variogram model; Bottom: ellipsoid of variogram trend

Variogram model for estimation Domain 7Ka