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CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

ON THE

Livengood Gold Project

Feasibility Study

Livengood, Alaska

Prepared for:
International Tower Hill Mines Ltd.

by

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Common Acronyms and Symbols	
Acronym	**Description**
A	Ampere
a	Annum (year)
AARL	Anglo American Research Laboratories
ADEC	Alaska Department of Environment Conservation
ADF&G	Alaska Department of Fish & Game
Ag	Silver
AMHT	Alaska Mental Health Trust
amsl	Above Mean Sea Level
ANFO	Ammonium Nitrate Fuel Oil
ANILCA	Alaska National Interest Lands Conservation Act
ARD	Acid Rock Drainage
Au	Gold
BLM	United States Bureau of Land Management
CFR	Congressional Federal Register
CIL	Carbon In Leach
CIM	Canadian Institute of Mining
CN	Cyanide
CND	Cyanide Detoxification
CN_T	Cyanide (Total)
CN_{WAD}	Cyanide Wad
Cu	Copper
EPA	United States Environmental Protection Agency
EPCM	Engineering, Procurement, Construction Management
FWR	Fresh Water Reservoir
G&A	General and Administration
ICP	Inductively Coupled Plasma
IRR	Internal Rate of Return
ITH	International Tower Hill Mines, Ltd.
LLC	Limited Liability Company
LLDPE	Linear Low Density Polyethylene
LOM	Life of Mine
LPI	Livengood Placers, Inc.
MIK	Multiple Indicator Kriging
ML	Metal Leaching
MSHA	Mine Safety and Health Administration
MWMP	Meteoric Water Mobility Potential
My	Million Years
NAD	North American Datum (Topographical Surveying)

Common Acronyms and Symbols	
Acronym	**Description**
NEPA	National Environmental Policy Act
NPV	Net Present Value
PAG	Potentially Acid Generating
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
QA/QC	Quality Assurance/Quality Control
RMR	Rock Mass Rating
ROM	Run Of Mine
RQD	Rock Quality Designation
SAG	Semi-Autogenous Grinding
SE	Samuel Engineering
SMU	Smallest Mining Unit
SVC	Static VAR Compensator
SWPPP	Storm Water Pollution Protection Plan
TAPS	Trans-Alaska Pipeline
THM	Tower Hill Mines, Inc.
TMF	Tailings Management Facility
US	United States
USACE	United States Army Corps of Engineers
USGS	United States Geological Survey
UTM	Universal Transverse Mercator Coordinate System
WBS	Work Breakdown Structure
WSR	Water Storage Reservoir
XRF	X-ray Fluorescence

Units of Measure - Abbreviations			
Imperial		Metric	
Units	Description	Units	Description
%	Percent	%	Percent
% solids	Percent solids by weight	% solids	Percent solids by weight
$	United States dollar		
$/t	Dollars per ton	$/mt	Dollars per metric tonne
°F	degrees fahrenheit	°C	degrees celsius
B	Billion	B	Billion
Btu	British thermal units	g-Cal	gram - calories
cfm	cubic feet per minute	m^3/m	cubic meters per minute
cP	centipoise (viscosity)	cP	centipoise (viscosity)
d	day (24 hours)	d	day (24 hours)
deg or °	angular degree	deg or °	angular degree
F_{100}	100% passing - Feed	F_{100}	100% passing - Feed
F_{80}	80% passing - Feed	F_{80}	80% passing - Feed
ft	feet (12 inches)	m	meter
ft/d	feet per day	m/d	meters per day
ft/s	feet per second	m/s	meters per second
ft/s^2	feet per second squared	m/s^2	meters per second squared
ft^2	square feet	m^2	square meter
ft^2	square feet	cm^2	square centimeter
ft^2/d	square feet per day	cm^2/d	square centimeter per day
ft^3	cubic feet	m^3	cubic meter
ft^3/h	cubic feet per hour	m^3/h	cubic meters per hour
gal	gallons	L	Liter
gal/h	gallon per hour	L/h	Liters per hour
lb/lb	pound per pound	g/g	grams per grams
gpm	gallons per minute	L/m	Liter per minute
hrs	hour (60 minutes)	hrs	hour (60 minutes)
hp	horsepower	kW	kilowatt
Hz	Hertz	Hz	Hertz
in	inches	mm	millimeter
in	inches	μm	microns
in Hg	inches of mercury	mm Hg	millimeters of mercury
in WC	inches Water Column	mm WC	millimeters Water Column
in^2	square inch	mm^2	square millimeters
K	Thousand (000)	K	Thousand (000)
k	kips (1,000 pounds)	kg	kilograms
k/ft^2	kips per square foot	kg/m^2	kilograms per square meter
kWh/t	kilowatt hour per ton	kWh/mt	kilowatt hour per tonne
lb	pound	kg	kilogram

Units of Measure - Abbreviations			
Imperial		Metric	
Units	Description	Units	Description
lb/ft^3	pounds per cubic foot	kg/m^3	kilograms per cubic meter
lb/gal	pounds per gallon	g/L	grams per Liter
lb/h	pounds per hour	kg/h	kilograms per hour
lb/min	pounds per minute	kg/min	kilograms per minute
lb/t	pounds per ton	kg/mt	kilograms per tonne
M	Million	M	Million
MBtu	Million British thermal units	kj	kilojoules
mesh	US Mesh	micron	microns
Mgal/d	Million gallons per day	ML/d	Million Liters per day
mi	miles	km	kilometers
mil	one thousandth of an inch	mm	millimeter
min	minute (60 seconds)	min	minute (60 seconds)
mph	miles per hour	km/h	kilometers per hour
MW	Megawatts	MW	Megawatts
oz	Troy ounces	oz	Troy ounces
oz Av	ounce avoirdupois	g	gram
oz/t	Troy ounces per ton	g/mt	grams per tonne
oz/y	Troy ounces per year	g/y	grams per year
P_{100}	100% passing - Product	P_{100}	100% passing - Product
P_{80}	80% passing - Product	P_{80}	80% passing - Product
ppm	parts per million	ppm	parts per million
psf	pounds per square foot	kg/m^2	kilograms per square meter
psi	pounds per square inch	kPa	kilopascal
psia	pounds per square inch - absolute	kPaa	kilopascal - absolute
psig	pounds per square inch - gauge	kPag	kilopascal - gauge
rpm	revolutions per minute	rpm	revolutions per minute
s	seconds	s	seconds
scfm	standard cubic feet per minute	m^3/min	cubic meters per minute
sg	specific gravity	sg	specific gravity
t	ton (2,000 lbs)	mt	tonne (1,000 kg)
t/d	ton per day	mt/d	tonnes per day
t/h	tons per hour	mt/h	tonnes per hour
V	Volt	V	Volt
W	Watt	W	Watt
wt%	weight percent	wt%	weight percent
y	year (365 days)	y	year (365 days)
yd^3	cubic yard	m^3	cubic meters
yd	yard (36 inches)	m	meter

1.0 Summary

Samuel Engineering, Inc. (SE) was retained by Tower Hill Mines, Inc. (THM), a wholly owned subsidiary of International Tower Hill Mines Ltd (ITH), to support the preparation of a Canadian National Instrument (NI) 43-101 Technical Report for the Livengood Gold Project, located 70 miles (113 km) northwest of Fairbanks Alaska based on a Feasibility Study (FS) that was substantially complete as of July 23, 2013, with the final version to be completed shortly with no material changes anticipated.

The FS was prepared with input from the companies listed in Table 1.1

Table 1.1 Feasibility Study Participants	
Tower Hill Mines, Inc. (THM)	Owner, Property Description, History, Infrastructure, Geology, Environmental Baseline, Permitting, Socioeconomic Conditions, Exploration and Drilling
MTB Project Management Professionals Inc.(MTB)	Owner's Project Manager, Project Schedule, Financial Model & Project Economics
Roscoe Postle Associates USA Ltd. (RPA)	Owner's Project Metallurgist, Mineral Processing and Metallurgical Testing
Reserva International LLC (Reserva)	Geology, Resource Estimate and Mine Block Model
Mine Development Associates, Inc. (MDA)	Mine Design Engineer, Mine Production Schedule, Mineral Reserve Estimate
AMEC Environment and Infrastructure (AMEC)	Geotechnical Engineer, Site Water Balance, Access Roads, Tailings and Overburden Rock Management Facilities
SGS Mineral Services (SGS)	Metallurgical Testwork
SRK Consulting (U.S.) Inc. (SRK)	Fresh Water Supply, Site Water Balance
Dryden & LaRue, Inc., Electric Power Systems Inc. (EPS)	Incoming Power Line
Samuel Engineering, Inc. (SE)	Document Preparation, Coordination, Recovery Methods, and Capital and Operating Costs

The FS assumed that the Livengood Gold Project will be constructed using imperial units. Therefore, to the maximum extent practicable, all design work and equipment descriptions were completed and reported in imperial units, with metric units shown in parentheses. This convention was mirrored in this Technical Report.

However, it is important to note that both the Livengood Gold Project drill-hole database and the block model were originally created in metric units and have been consistently maintained in metric units. Therefore, in order to minimize the risk for error, all resource and reserve estimations as well as pit shell optimizations and production schedules, presented in this Technical Report, are presented in metric units.

For financial modeling, ore tonnage is reported in short tons, with all costs reported in $/ton.

Certain other testwork, such as comminution results and unconfined compressive strength tests, are reported in metric units.

1.1 Property Description, Location and Access

The Livengood property is located approximately 70 miles (113 km) by road (47 miles (75 km) by air) northwest of Fairbanks, Alaska in the Tolovana Mining District within the Tintina Gold Belt. The deposit area is centered near Money Knob, a local topographic high point. This feature and the adjoining ridge lines are the probable lode gold source for the Livengood placer deposits that lie in the adjacent valleys that have been actively mined since 1914 and produced more than 500,000 oz of gold.

The property lies in numerous sections of Fairbanks Meridian Township 8N and Ranges 4W and 5W. Money Knob, the principal geographic feature within the known deposit, is located at 65°30'16"N, 148°31'33"W.

The property straddles Highway 2 (also known as the Elliott Highway), a paved, all-weather highway linking the North Slope oil fields to Fairbanks, and adjoins the Trans-Alaska Pipeline (TAPS) corridor, which transports crude oil from the North Slope south and contains the fiber-optic communications cable that may be used at the Project site (see Figure 1.1). Locally, a number of unpaved roads lead from the Elliott Highway into and across the deposit. A 3,000-foot (914 m) runway is located 3.7 miles (6 km) to the southwest of the project and is suitable for light aircraft.

The plant is designed to process 36,500,000 t (33,112,240 mt) annually. The nominal plant throughput is 100,000 t/d (90,718 mt) (dry). The project life-of-mine (LOM) is 14 years. The plant will employ crushing, grinding, gravity separation, cyanide leaching, adsorption/desorption/recovery, and smelting techniques to produce a gold doré product and will operate 24 hours per day, seven days per week year round.

The site is approximately 40 mi (64 km) south of the Arctic Circle. The climate in this part of Alaska is continental with temperate and mild conditions in summer with average lows and highs in the range of 44° to 72°F (7 to 22°C). Winter is cold with average lows and highs for December through March in the range of -17°F to 23°F (-27°C to -5°C). The lowest temperatures are in the -40°F (-40°C), range. Annual precipitation is on the order of 15.7 in (400 mm) water equivalent. Winter snow accumulation snow pack depth is approximately 26 in (66 cm).

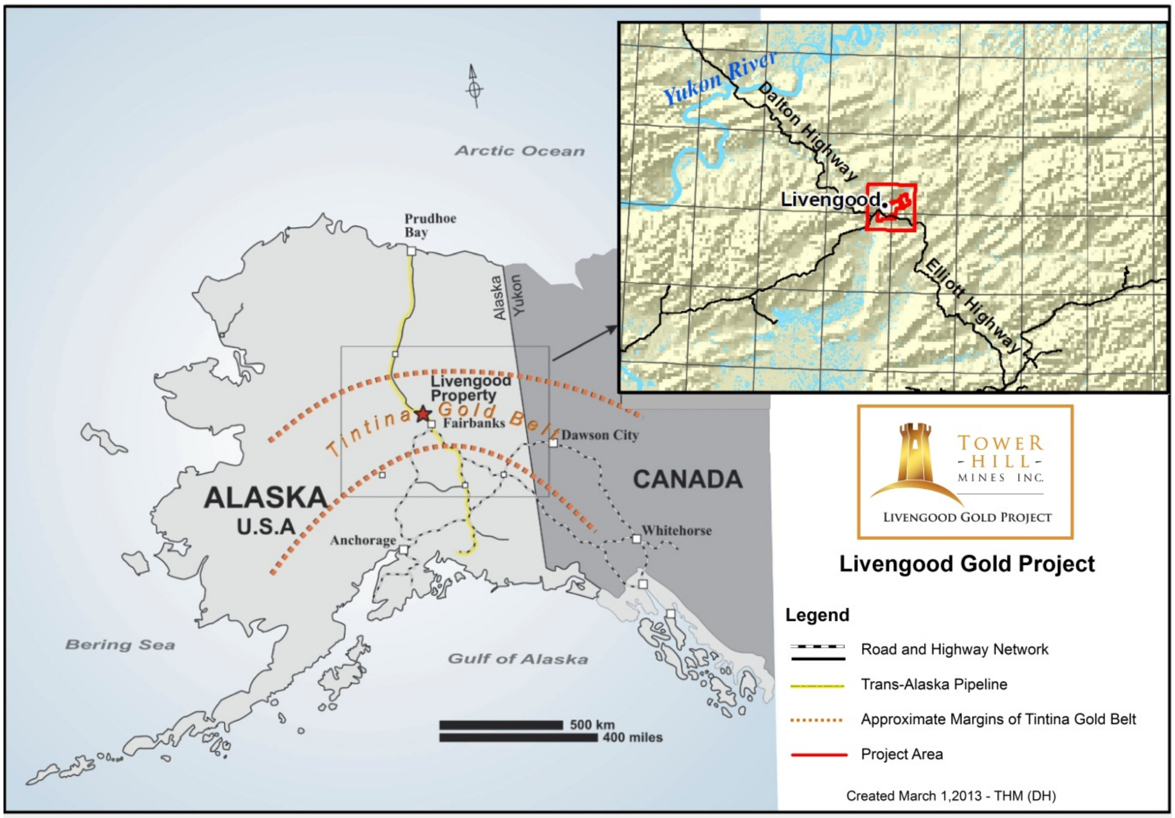


Figure 1.1 Project Location Map

1.2 Land Tenure

The Livengood Gold Project property covers approximately 48,300 acres (19,500 hectares), all of which is controlled by ITH through its wholly-owned subsidiaries, THM and Livengood Placers, Inc. (LPI). The Livengood Gold Project is comprised of multiple land parcels: 100% owned patented mining claims, 100% owned State of Alaska mining claims, and 100% owned federal unpatented placer claims; land leased from the Alaska Mental Health Trust (AMHT); land leased from holders of state and federal patented and unpatented lode and placer mining claims, and undivided interests in patented mining claims. The property and claims controlled through ownership, leases or agreements are shown in Figure 1.2.

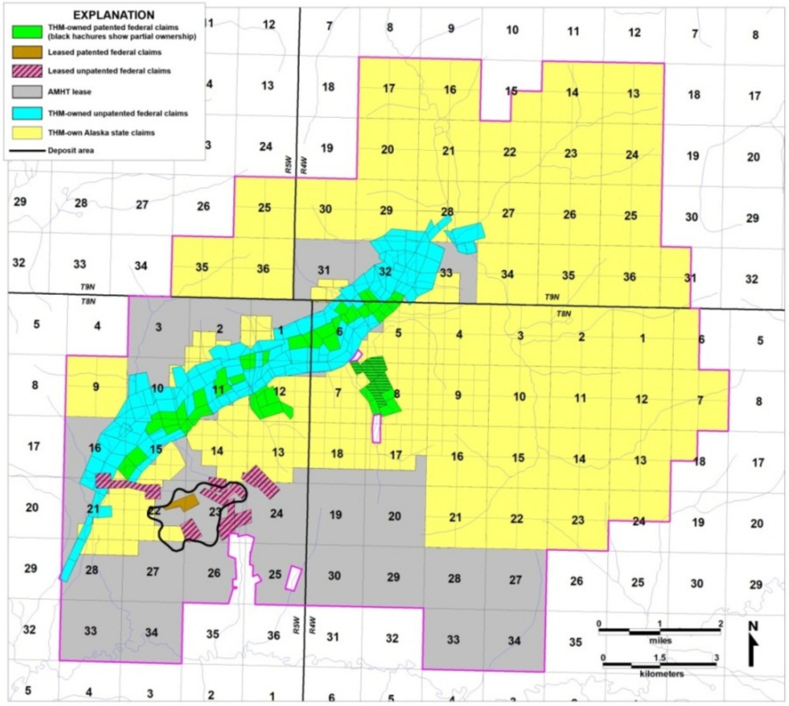


Figure 1.2 Livengood Land Holdings

1.3 Property History

Gold was first discovered in the gravels of Livengood Creek in 1914 (Brooks, 1916) and led to the founding of the town of Livengood. Subsequently, more than 500,000 oz of placer gold have been produced. From 1914 through the 1970's, the primary focus of prospecting activity was placer deposits. Historically, prospectors considered Money Knob and the associated ridgeline the source of the placer gold. Prospecting, in the form of dozer trenches, was carried out for lode type mineralization in the vicinity of Money Knob, primarily in the 1950's. However, no significant lode production has occurred to date.

Since the 1970's the property has been prospected and explored by several companies. Geochemical surveys by Cambior Inc. in 2000 and AngloGold Ashanti (U.S.A.) Exploration Inc. (AGA) in 2003 and 2004 outlined a 1.0 x 0.5 mile area (1.6 x 0.8 km) with anomalous gold in soil. Scattered anomalous samples continue along strike for an additional 1.2 miles (2 km) to the northeast and 1 mile (1.6 km) to the southwest. Eight reverse circulation holes were drilled by AGA in 2003 and a further four diamond core holes were drilled in 2004 to evaluate this anomaly. Favorable results from these holes revealed wide intervals of gold mineralization (BAF-7: 455 ft (138.7 m) @ 1.07 g/mt Au; MK-04-03: 181.4 ft (55.3 m) @ 0.51 g/mt Au) along with lesser intervals over a broad area. In 2006, AGA sold the Livengood Gold Project to THM. In the same year, THM drilled a 4,189 foot (1,227 m), seven-hole core program. The success of that program led to the drilling of an additional 14,432 ft (4,400 m) in fifteen diamond core holes in 2007 to test surface anomalies, expand the area of previously intersected mineralization, and advance geologic and structural understanding of the deposit. Subsequent programs have continued to expand the resource, leading to consideration of development of the deposit and concomitant geotechnical, engineering, and metallurgical work, along with the collection of initial environmental baseline data. Through completion of the delineation drilling at the end of the 2011 season, ITH completed a total of 712,994 ft (217,321 m) of exploration and delineation drilling, of which 575,078 ft (175,284 m) were RC drilling and 137,915 ft (42,037 m) were core drilling.

Beginning in 2009, technical studies were performed to generate metallurgical data for process definition, to generate preliminary surface mine designs, and to develop pre-conceptual information on the location and capacities of potential tailings management, overburden management, water reservoir and mill process facilities. A Pre-Feasibility study was begun in 2011 but was not completed as advancing technical studies indicated major changes to the flowsheet and project configuration warranted a shift to the Feasibility Study.

Detailed project configurations have now been generated in the Feasibility Study, which has been used as the basis for projected operating and capital cost estimation.

1.4 Geology and Mineralization

1.4.1 Geology

Rocks in the Livengood mining district are part of the Livengood Terrain, an east-west belt, approximately 240 km long, consisting of tectonically interleaved assemblages of various ages. These assemblages include the Amy Creek Assemblage, which is a sequence of latest Proterozoic and early Paleozoic basalt, mudstone, chert, dolomite, and limestone. In thrust contact above the Amy Creek Assemblage lies an early Cambrian ophiolite sequence of mafic and ultramafic seafloor rocks. Structurally above these rocks lies a sequence of Devonian shale, siltstone, conglomerate, volcanic, and volcaniclastic rocks which are the dominant host to the mineralization currently under exploration at Livengood. The Devonian assemblage is overthrust by more Cambrian ophiolite rocks. All of these rocks are intruded by Cretaceous multiphase monzonite, diorite, and syenite stocks, dikes, and sills. Gold mineralization is believed to be related to this intrusive event.

1.4.2 Mineralization

Gold mineralization is associated with disseminated arsenopyrite and pyrite in volcanic, sedimentary and intrusive rocks, and in quartz veins cutting the more competent lithologies, primarily volcanic rocks, sandstones, and, to a lesser degree, ultramafic rocks. Three principal stages of alteration are currently recognized; in order from oldest to youngest these are characterized by biotite, albite, and sericite. Carbonate was introduced with and subsequent to these stages. Arsenopyrite and pyrite were introduced primarily during the albite and sericite stages. Gold correlates strongly with arsenic and occurs primarily within and on the margins of arsenopyrite and pyrite.

Mineralization is interpreted to be intrusion-related, consistent with other gold deposits of the Tintina Gold Belt, and has a similar arsenic-antimony (As-Sb) geochemical association. Mineralization is controlled partly by stratigraphic units, but thrust-fold architecture is apparently key to providing pathways for magma (dikes and sills) and hydrothermal fluid.

Local fault and contact limits to mineralization have been identified, but overall the deposit has not been closed off in any direction. The current resource and area drilled covers the most significant portion of the area with anomalous gold in surface soil samples, but still represents only about 25% of the total gold anomaly.

1.5 Status of Exploration

Cambior was chiefly responsible for outlining the sizeable area of anomalous gold in soil samples, which ITH expanded between 2006 and 2010, improving definition of the extent of anomalous gold in soil to the southwest and northeast of the deposit outlined by drilling to date. The currently known deposit is defined by the most coherent and strongest gold anomaly, but represents detailed evaluation of only about 25% of the total gold-anomalous area.

During 2011, ITH completed an IP/Resistivity survey covering the deposit and gold-anomalous soil geochemistry to the northeast, where loess and frozen ground have prevented complete geochemical coverage. The objective of the survey was to establish the geophysical signature of the deposit and identify similar signatures elsewhere in the district to prioritize exploration drilling. When evaluation of the data is complete it should help guide exploration outside of the known deposit.

1.6 Mineral Resource Estimate

The global mineral resource estimate has been updated from that published in August, 2011 to include drilling in the deposit since that time. The resource model was constructed using Gemcom GEMS® and the Stanford GSLIB (Geostatistical Software Library) Multiple Indicator Kriging (MIK) post processing routine. The resource was estimated using MIK techniques.

A three-dimensionally defined stratigraphic model, based on interpretations by THM geologists, was used to code the rock type block model. A three-dimensionally defined probability grade shell (0.1 g/mt) was used to constrain the gold estimation. Gold contained within each block was estimated using nine indicator thresholds. The block model was tagged with the geologic model using a block majority coding method. Because there are significant grade discontinuities at stratigraphic contacts, hard boundaries were used between each of the stratigraphic units so that data for each stratigraphic unit was used only for that unit.

A summary of the estimated global (in-situ) mineral resource is presented below in Table 1.2 for cutoff grades of 0.2, 0.3, 0.5, and 0.7 g/mt gold.

Model validation checks include global bias check, visual validation, and swath plots. In all cases, the model appears to be unbiased and fairly represent the drilling data.

Table 1.2 Global Resource Estimation Summary – July 2013				
Classification	**Gold Cutoff (g/mt)**	**Tonnes (millions)**	**Gold (g/mt)**	**Million oz Gold**
Measured	0.20	994	0.52	16.4
Indicated	0.20	112	0.45	1.6
Total M&I	0.20	1106	0.51	18.0
Inferred	0.20	438	0.41	5.8
Measured	0.30	731	0.61	14.4
Indicated	0.30	71	0.56	1.3
Total M&I	**0.30**	**802**	**0.61**	**15.7**
Inferred	0.30	266	0.52	4.4
Measured	0.50	370	0.82	9.8
Indicated	0.50	31	0.80	0.8
Total M&I	0.50	401	0.82	10.6
Inferred	0.50	92	0.76	2.3
Measured	0.70	179	1.08	6.2
Indicated	0.70	13	1.09	0.5
Total M&I	0.70	192	1.08	6.7
Inferred	0.70	34	1.08	1.2

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to measured and indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

In 2011 an economic surface mine was generated using Whittle mine optimization software to define the Mineral Resources using an assumed long-term gold price of $1,400/oz (Brechtel, et al., 2011). Based on that study and by analogy to the nearby operating Fort Knox Mine, delineated mineralization of the Livengood Deposit is classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2010):

"In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended."

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when 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. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project.

An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of the geology and controls of the mineral deposit.

1.7 Mineral Reserve Estimate

The proven and probable reserves at Livengood which are contained in the final pit are summarized in Table 1.3 and match the production schedule.

Table 1.3
Reserve

Rock Type	Tonnes 000's	g Au/mt	Au oz 000's
RT4 Cambrian	58,247.3	0.639	1,196.6
RT5 Sunshine Upper Sediments	126,592.2	0.576	2,344.6
RT6 Upper Sediments	80,912.3	0.733	1,906.0
RT7 Lower Sediments-Bleached	51,020.0	0.772	1,266.3
RT8 Sunshine Volcanics	6,707.4	0.659	142.1
RT9 Volcanics	111,013.9	0.775	2,766.0
Proven Totals	**434,493.0**	**0.689**	**9,621.5**
RT4 Cambrian	5,129.8	0.720	118.7
RT5 Sunshine Upper Sediments	1,503.4	0.535	25.8
RT6 Upper Sediments	2,754.6	0.637	56.4
RT7 Lower Sediments-Bleached	4,005.3	0.726	93.5
RT8 Sunshine Volcanics	2,321.2	0.669	49.9
RT9 Volcanics	4,416.4	0.773	109.7
Probable Totals	**20,130.8**	**0.702**	**454.0**
RT4 Cambrian	63,377.1	0.645	1,315.2
RT5 Sunshine Upper Sediments	128,095.6	0.576	2,370.4
RT6 Upper Sediments	83,666.9	0.730	1,962.4
RT7 Lower Sediments-Bleached	55,025.3	0.769	1,359.8
RT8 Sunshine Volcanics	9,028.6	0.662	192.0
RT9 Volcanics	115,430.3	0.775	2,875.7
Proven + Probable Totals	**454,623.8**	**0.689**	**10,075.6**

1.8 Mining

The FS is based on a plan to mine and process 100,000 t (90,718 mt) of ore daily. The mining method uses conventional drill-blast-load-haul. The mine mobile fleet includes three 47 yd^3 front shovels, a maximum of twenty-five 320 ton haul trucks in year 3, and six blast hole drills. A fleet of support equipment includes two 40 yd^3 front end loaders, track and rubber tire dozers, graders, cranes, explosive storage and loading equipment, as well as maintenance support equipment.

Mining is planned on 33 ft (10 m) benches. The pit design uses an inter-ramp slope of 43° and an overall slope of 37° to 42° including the ramp, with a catch bench every 66 ft (20 m).

The production schedule is based on processing 100,000 t/d (90,718 mt/d), or 36.5 Mt (33.1 Mmt) annually. The ramp up schedule is 80% of design during year 1.

Within the constraints of the maximum annual mining capacity, the schedule maximizes the ore grade in the initial years by stockpiling lower grade materials. The maximum stockpile size reaches 93.2 Mt. This stockpiled material will be processed after direct feed from the pit is exhausted.

A pre-production period of two years will be needed to remove overburden and to produce rock to complete the construction of facilities.

The production schedule is shown below.

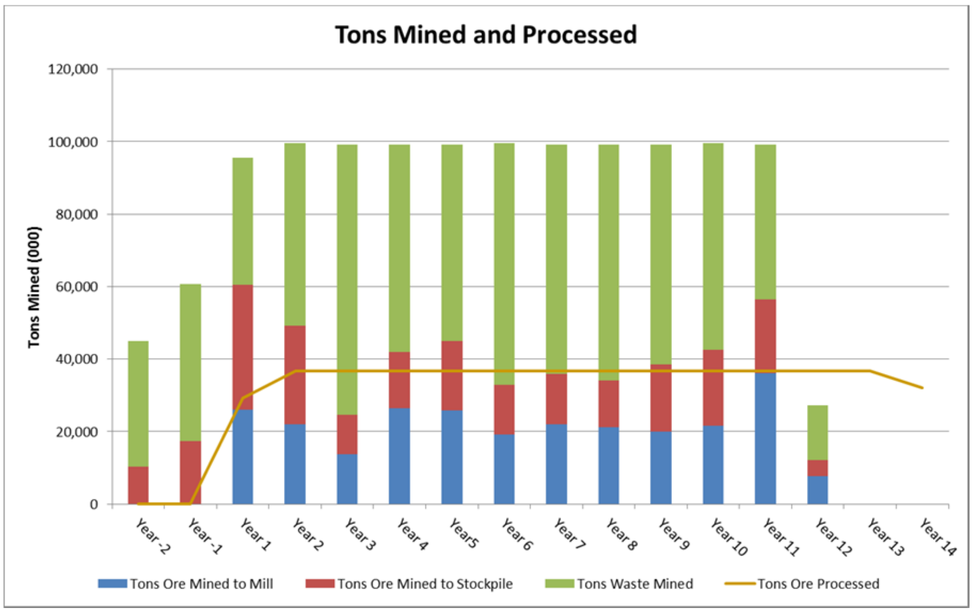


Figure 1.3 Tons Mined and Processed Production Schedule

1.9 Mineral Process and Metallurgical Testing

Metallurgical testing was performed during 2012/2013 by SGS Vancouver, Canada on mineralized core samples to provide data for the design of the Livengood Gold Project. The samples were composited into different rock types to represent the resource to be mined over the course of the mine life. The test program began with the initial three phases: comminution, optimization and variability. Optimization composites were designed to be representative of each rock type by choosing individual members of the composite samples that matched the deposit grade histograms of each rock type. These composite rock type samples produced metallurgical recoveries typical of that rock type. The variability samples were selected to test the geologic extremes of each rock type. The testwork was focused on the five rock types that made up the majority of the material that would be processed over the life of the mine.

Crusher Work Index, Grinding Bond Ball Mill, Rod Mill, and Abrasion Index data (CWi, BMi, RMi and Ai) were determined along with JKTech SMC breakage and abrasion parameters for input into the JKTech SimMet grinding simulation program. Comminution data was produced for both samples that comprised portions of the optimization composites and variability samples.

Gravity tests were performed on each optimization composite to determine the gravity recoverable gold for each rock type. The gravity recovery was optimized by testing various grind sizes. Ultimately all of the rock types were first processed by gravity, to remove the coarser gold particles, prior to subsequent flotation or cyanidation tests.

Flotation tests were run on optimization composites after gravity separation to determine the best operating conditions including grind size, reagent suite, and flotation retention time for each of the composites. Cyanide leach tests were conducted on the flotation concentrates from each rock type after regrinding the concentrates to achieve better liberation.

Sodium cyanide leach tests were run on each optimization composite after gravity separation to determine the optimal operating conditions including grind size, the benefit of air sparging and retention time for gold recovery.

Mineralogical examinations of the individual head samples, bulk concentrate products and tailings were performed for each optimization composite. Sophisticated mineralogical examination was conducted by QemScan to determine mineral liberation and mineral associations.

Variability tests were conducted utilizing the optimal process conditions determined in the optimization phase. This phase examined the extremes of each rock type, considering grade variations, spatial location, and anomalous concentrations of deleterious elements, such as arsenic and antimony.

The metallurgical response of the material tested indicated an increase in gold recovery by whole ore cyanidation leaching after gravity separation of the coarser gold particles compared to leaching the flotation concentrate after gravity separation. The overall gold recovery including both gravity and gravity tail cyanidation is shown below in Table 1.4.

Table 1.4
Gold Recovery by Rock Type

Rock Type	Gravity + CIL Rec of Gravity Tailings, Gold Recovery %	Gravity + Flotation of Gravity Tailings + CIL of Flotation, Gold Recovery %
RT4 Cambrian	84.2	-
RT5 Sunshine Upper Sediments	87.7	76.1
RT6 Upper Sediments	76.7	67.4
RT7 Lower Sediments Bleached	58.5	-
RT9 Volcanics	84.8	74.4

The following process block flow diagram, Figure 1.4, describes the optimal process flow from the ore delivery to the crusher through to doré production and tailings management.

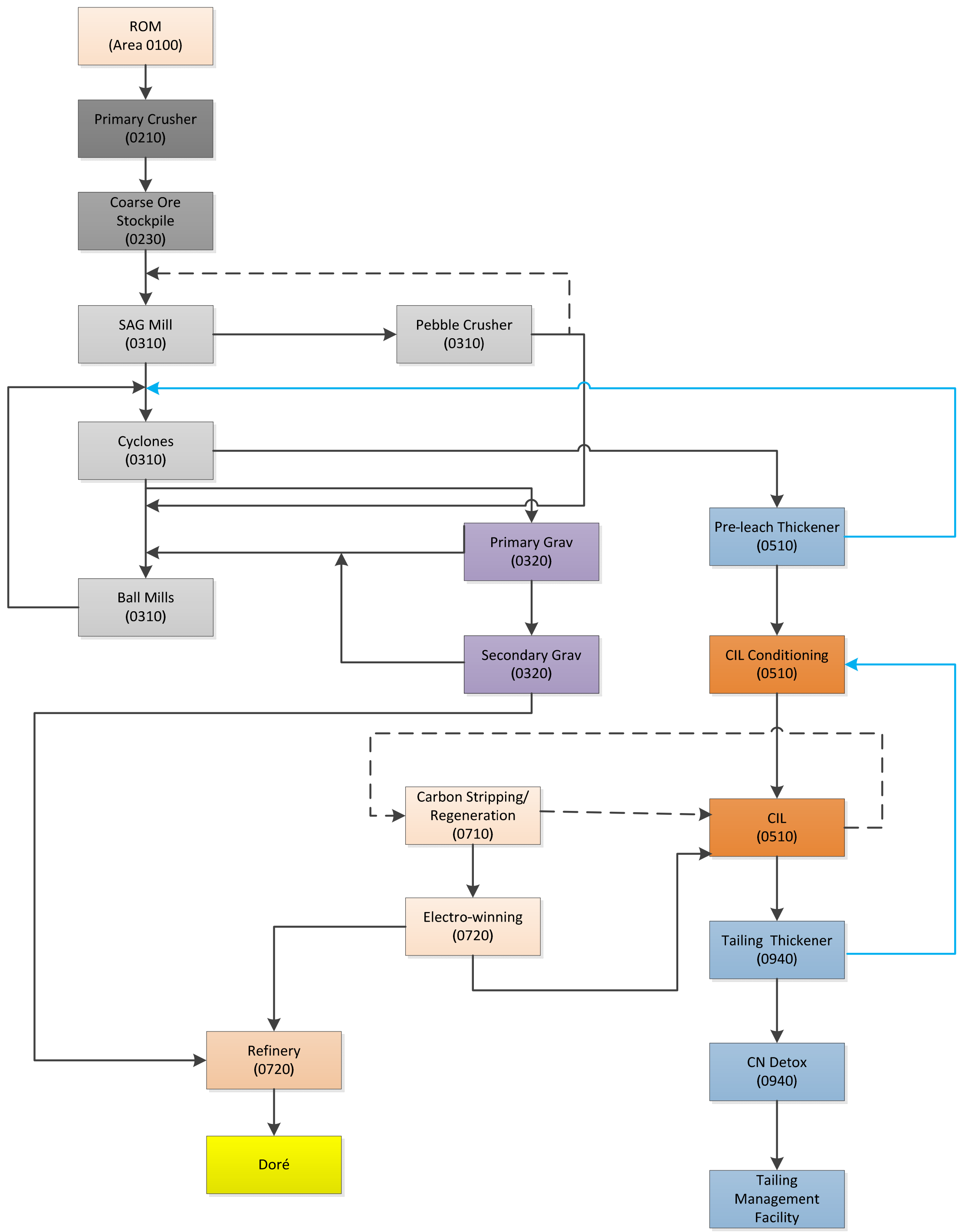


Figure 1.4 Process Block Flow Diagram

The following Figure 1.5 shows the gold oz production from the mine per year.

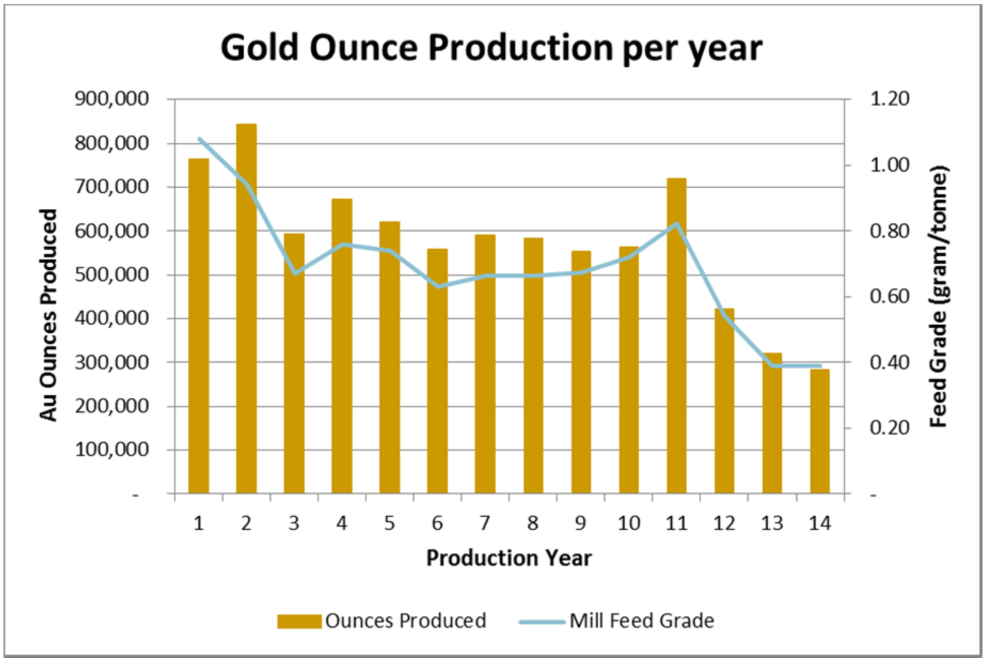


Figure 1.5 Gold oz Production Schedule

1.10 Local Resources and Infrastructure

Alaska infrastructure has been developed in a north-south corridor between ports on the south coast (Anchorage, Valdez and others) and Fairbanks in the center of the State. The Fairbanks North Star Borough, which in 2012 had a population of 100,343 people, contains a hospital, government offices, businesses, military bases, and the University of Alaska, Fairbanks. Fairbanks is linked to southern Alaska by a north-south transportation and utility corridor that includes 2 paved highways, a railroad, an interlinked electrical grid, and communications infrastructure. The city has an international airport serviced by up to 3 major airlines and has demonstrated capacity to serve as the primary employment and service base for the project.

The paved, all weather State Highway 2 (Elliott Highway) runs north from Fairbanks to the North Slope oilfields at Prudhoe Bay, and passes within one mile of the Money Knob deposit. Communications infrastructure (fiber optic) has been extended to the North Slope along the TAPS, which parallels the Elliott Highway and passes just west of the Livengood project site.

A study completed by Electric Power Systems has determined that the local utility in Fairbanks (Golden Valley Electric Association) can provide the 100 MW of power required for the project. The Project would be connected to the local grid by building a 50 mi (80 km) 230-kVa transmission line along the pipeline corridor.

1.11 Environmental and Permitting

THM has been conducting environmental baseline studies at the Livengood Gold Project since 2008 as part of their overall goal of providing environmentally relevant and supportable data for environmental permitting, engineering design, and a basis for permit-required monitoring during construction, mining, and closure of the project. These studies include surface water, hydrology, hydrogeology, wetlands & vegetation, meteorology & air quality, aquatic resources, rock characterization, wildlife, and cultural resources.

Table 1.5 Environmental Baseline Studies (2008-2012)					
Baseline Study	**2008**	**2009**	**2010**	**2011**	**2012**
Surface Water					
Surface Water Quality		●	●	●	●
Hydrology					
Hydrogeology			●	●	●
Groundwater Quality			●	●	●
Hydrogeological Modeling			●	●	●
Permafrost Studies			●	●	●
Wetlands					
Wetlands Delineations		●	●	●	●
Meteorology & Air Quality					
Meteorological Data			●	●	●
Precipitation			●	●	●
Ambient Air				●	
Aquatic Resources					
Bio-monitoring		●	●	●	●
Resident Fish Surveys		●	●	●	●
Rock Characterization					
Static ML/ARD Testing			●	●	●
Kinetic ML/ARD Testing				●	●
On-Site Kinetic Testing					●
Wildlife Studies					
Habitat Mapping				●	
Mammal Surveys				●	
Avian Surveys				●	●
Cultural Resources					
Cultural Site Surveys	●	●	●	●	●
Socioeconomics (Section 11.0)				●	●
Noise Studies					
Noise Surveys					●

In early 2011, project engineers identified a 50-mile power transmission corridor with a terminus at Livengood. Baseline investigations along this corridor have included: surface water quality, wetlands & vegetation, wildlife, aquatic resources, and cultural resources. The results of these programs are being used, in part, to select the transmission alignment.

Based on review of the studies completed to date, there are no known environmental issues that are anticipated to materially impact the Project's ability to extract the gold resource.

Since development of the Project will require a number of Federal permits, the National Environmental Policy Act (NEPA) and Council of Environmental Quality (CEQ) Regulations 40 CFR parts 1500-1508 will govern the federal permitting portion of the Project. The NEPA process requires that all elements of a project and their direct, indirect, and cumulative impacts be considered. A reasonable range of alternatives are evaluated to assess their comparative environmental impacts, including consideration of feasibility and practicality. In fulfillment of the NEPA requirements, it is anticipated that the Project will be required to prepare an Environmental Impact Statement (EIS). Upon completion of the EIS and the associated Record of Decision by the lead federal agency, the federal and state agencies will then complete their own permitting actions and decisions. Although at this time it is unknown which department will become the lead federal agency, the State of Alaska is expected to take a cooperating role to coordinate the NEPA review with the State permit process.

Actual permitting timelines are controlled by the Federal NEPA review and Federal and State agency decisions. There have been no permit applications submitted for Project construction.

1.12 Socioeconomic Conditions

Livengood lies within the Yukon-Koyukuk Census Area, which encompasses a very large swath of Interior Alaska from the Canadian border to the lower Yukon River. In 2012, the Census Area held a total population of 5,682 widely dispersed residents in 20 communities, of which 71% were Alaska Natives. Minto, which is approximately 40 miles (64 km) from Livengood and Manley Hot Springs, 80 miles (129 km) away, have road access to Fairbanks.

The Fairbanks area is the service and supply hub for Interior and Northern Alaska. Construction of the TAPS resulted in an economic boom in Fairbanks from 1975-77. The oil industry remains an important part of the local economy, with Fairbanks providing logistical support for the North Slope activity, the two local refinery operations, and the operation and maintenance of TAPS. Today, the University of Alaska, the Fairbanks Hospital, and the Fort Knox and Pogo gold mines are some of the Fairbanks area's largest employers. The Fairbanks North Star Borough (FNSB) economy included 39,400 non-agricultural wage and salary jobs in 2012. In 2011, average employment of 39,018 wage and salary jobs, accounted for $1.81 billion in annual payroll. Most of the small communities in rural interior Alaska live a subsistence lifestyle. Seventy-five percent of the Native families in Alaska's smaller villages acquire 50% of their food through subsistence activities (Federal Subsistence Board, 1992). For families who do not participate in a cash economy, subsistence can be the primary direct means of support; for others, it contributes indirectly to income by replacing household food purchases.

The Feasibility Study estimates a total of 6,974,000 man-hours during project construction with a peak construction workforce of 814. The average hourly wage of those workers is estimated at $42/h. During the two years of pre-production mine development, owner's crew will be approximately 200 employees. During operation, the peak employee count is estimated at 425 and an annual average wage of approximately $97,000/y. Total annual wages paid during operation is estimated to be $41.6M.

The labor force in the communities nearest the mine is very small. The total population of Minto, Manley Hot Springs and the Livengood area combined was just over 350 residents in 2012. Skilled and unskilled labor to support mine development and operations will come primarily from the Fairbanks area, with a total labor force of over 40,000 workers.

1.13 Capital Cost and Operating Cost Estimates

1.13.1 Capital Costs

The capital cost estimate for the Livengood Gold Project addresses the development, construction, and start-up of a mine and plant capable of processing 100,000 t/d (90,718 mt/d) of gold bearing material.

THM has engaged other Consultants to provide estimate support for various cost portions of the project that fall within their specialized scope of work (see Table 1.6). Summary data was supplied for inclusion and use in this capital cost estimate.

Table 1.6
Capital Cost Estimate Contributors

Scope / Responsibility	Consultant
Mine Costs	Mine Development Associates
Haul Roads and Access Roads	AMEC
Tailings Management Facility (TMF)	AMEC
Mine Power Supply Line	Dryden & LaRue, Inc., EPS and GVEA
Process & Ancillary Facilities	Samuel Engineering
Indirect Cost	All
Owner's Cost	THM & MTB
Contingency	All

Note 1 - Independent third party commercial contracting firms with the expertise to execute the construction plan were retained to confirm the cost estimates of all major facilities

The key objectives of the capital cost study are to:

- Support the economic evaluation and assessment of the project;
- Identify and minimize areas of excessive cost;
- Establish a budget for financing, control, forecasting and completion.

General exclusions from the capital estimate are as follows:

- Sunk costs (costs prior to start of detailed design)
- Disposal/clean-up of hazardous materials (none have been identified)
- Allowance for special incentives (schedule, safety, etc.)
- Interest and financing cost
- Escalation beyond First Quarter 2013
- Sales Taxes and import duties
- Risk due to labor disputes, permitting delays, weather delays or any other force majeure occurrences.

The total estimated cost to design, procure, construct and commission the facilities described in this section is \$2.79B and sustaining capital of \$893M. Table 1.7 summarizes the initial capital and sustaining capital costs by major area. The estimate is expressed in nominal First Quarter 2013 United States dollars. No provision has been included to offset future escalation.

Table 1.7 Initial Capital and Sustaining Capital Costs by Major Area (Millions)		
Description	Initial	Sustaining
Process Facilities	$1,119	$26
Infrastructure Facilities	708	506
Power Supply	129	0
Mine Equipment	189	126
Mine Development	177	0
Other Owners Costs	166	9
Contingency	271	0
Subtotal Before Reclamation	**2,758**	**667**
Funding of Reclamation Trust Fund (1)	32	226
Total	**$2,790**	**$893**

Rounding of some figures may lead to minor discrepancies in totals.
(1) Includes initial funding and trust fund contributions, total $353 M estimated costs. The difference of $95 M is assumed trust fund earnings.

1.13.2 Operating Cost Estimate

The total and unit operating cost estimate summaries are shown below in Table 1.8 & Table 1.9, respectively. The three major operating cost areas are mining, processing and general & administrative (G&A). The unit costs areas are shown in terms of total cost life of mine (LOM) per ore ton processed and total cost per troy oz of gold produced. Details of the Mining Costs provided by MDA can be found in Section 21.3.2. Details of the Processing Costs provided by SE can be found in Section 21.3.3. Details of the General and Administrative Costs provided by THM and MTB can be found in Section 21.3.4.

Table 1.8 Total Operating Costs	
Total Operating Costs	Total Cost LOM
Mining	$1,861,590,070
Processing	$5,236,646,116
General & Administration	$444,737,330
Project Total Operating Cost	$7,542,973,515

Table 1.9 Unit Operating Costs		
Unit Operating Costs	Units	Average LOM
Mining	$/t ore Processed(1)	3.93
Processing	$/t ore Processed	10.45
General & Administration	$/t ore Processed	0.89
Project Unit Operating Cost	$/t ore Processed	15.27

(1) Average LOM mining cost per ton excludes mining costs associated with the 27.6 million ore tons excavated during the preproduction period and capitalized. Total LOM ore tones mined 473.5 million, total LOM ore tons processed 501.1 million.

1.14 Project Economics

Table 1.10 presents the model inputs used in the economic analysis. MDA, AMEC, and SE developed execution plans describing how the Project would be built and operated. The pre-production period and construction period financial inputs flow from the execution plan. Furthermore, the mine plan provided additional financial model inputs: mine life, ore tons mined, head grade and average annual gold production rate. The financial model applies metal pricing of $1,500/oz based on the London P.M. cose three-year trailing average price of gold reported by Kitco.com, which equaled $1,549.03/oz on June 30, 2013. First quarter 2013 US dollars form the financial model currency basis. No inflation or escalation exists in the economic model. The model calculates pre-tax and after-tax returns, and includes Alaska state taxes and Federal taxes based on the June 2013 federal and state income tax regulations. The model applies 3% royalties on net smelter returns across the life of mine based on an average royalty calculation. The model includes provisions for doré transportation, insurance, refining and payable charges. These technical and economic parameters used in the model are summarized in the following sections.

Table 1.10
Model Inputs

Execution Plan	
Pre-production Period	27 months
Construction Period	29 months
Mine Life (after pre-production)	13.87 years
LOM Ore Tons (millions)	501
LOM Gold Grade (g/mt Au)	0.69
Average Annual Process Gold Production Rate (oz)	577,598
Metal Pricing	
Gold Price ($/oz)	1,500
Cost and Tax Criteria	
Estimate Basis	Q1 2013
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Income Tax	AK State, Federal
Royalties	
Royalty on Net Smelter Return (NSR)	3%
Gold Transportation and Insurance, Refining, and Payable Charges	
Gold ($/oz)	9.30
Payable Terms	
Gold	99.50%

Table 1.11 below presents the results of the Feasibility Study.

Table 1.11
Summary of Feasibility Results

OPERATING METRICS		
Mill Throughput	100,000	Dry tons/day
Head Grade – LOM	0.69	g/mt
Head Grade – Year 1-5	0.83	g/mt
Gold Recovery	80.3	%
Mine Life	14	Years
Total oz Produced	8,086,400	oz
Average Annual Production – LOM	577,600	oz
Average Annual Production – Year 1-5	698,500	oz
Total Ore Processed	501	Million tons
Total Overburden	720	Million tons
Annual Mining Rate	98	Million tons
Overburden Rock to Mill Ore Ratio – Year 1-14	1.34:1	Overburden to Ore
Low Grade Stockpile Maximum Size	93	Million tons
FINANCIAL METRICS		
CAPEX – Initial	2.790	$Billion
CAPEX – Sustaining	667	$Million

Table 1.11 Summary of Feasibility Results		
Reclamation & Closure	353	$Million
OPEX – Mining	1.67	$/ton material
OPEX – Processing	10.45	$/ton ore
OPEX – G&A	0.89	$/ton ore
OPEX – Operating Cost – LOM	1,030	$/oz
OPEX – Operating Cost – Year 1-5	885	$/oz
All-In Cost Pre-Tax (CAPEX+OPEX) – LOM	1,447	$/oz
All-In Cost Pre-Tax (CAPEX+OPEX) – Year 1-5	1,272	$/oz
All-In Cost After-Tax (CAPEX+OPEX) – LOM	1,474	$/oz
All-In Cost After-Tax (CAPEX+OPEX) – Year 1-5	1,292	$/oz

The financial model uses the inputs from the entire Feasibility Study as its basis. The resulting revenue compared to capital and operating cost estimates summarized above yields a minimal positive return. The after-tax payback period is 10.8 years.

The pre-tax internal rate of return (IRR) is 2.8% and the pre-tax net present value (NPV) using a 5% discount rate over the mine life is a loss of $300,286,677.

The after-tax IRR is 1.7%. The after-tax NPV at a discount rate of 5% over the mine life is a loss of $439,714,744. Table 1.12 presents pre-tax and after-tax NPVs at discount rates from 0% to 10%.

Table 1.12
Base Case Analysis

Discount Rate	0.0%	5.0%	7.5%	10.0%
Pre-Tax NPV ($000)	523,726,552	(300,286,677)	(551,788,498)	(734,636,264)
After-Tax NPV ($000)	303,579,667	(439,714,744)	(665,341,341)	(828,460,873)

The results of the sensitivity analysis performed are summarized in Table 1.13 and show gold price and recovery variation cause the greatest impact on project value. A 20% increase in gold price would yield an 8.0% increase in IRR. A 20% decrease in gold price would yield an 17.8% reduction in IRR. The next most pronounced project sensitivity is to capital cost. Capital changes would drive marginally larger project returns than operating cost changes, meaning reducing capital expense would benefit the Project more than reducing operating costs by the same percentage.

Table 1.13 Livengood Sensitivity Analysis – After-Tax IRR and NPV(5%)									
Base Case Variance	-20%	-15%	-10%	-5%	Base	5%	10%	15%	20%
Recovery		68%	72%	76%	80.3%	84.3%	88%	92%	
After-Tax IRR		-8.9%	-4.2%	-0.9%	1.7%	3.9%	6.0%	7.9%	
After-Tax NPV @ 5%		($1,459,434,684)	($1,087,021,372)	($740,683,532)	($439,714,744)	($147,194,638)	$143,565,855	$432,005,660	
Price of Gold	$1,200	$1,275	$1,350	$1,425	$1,500	$1,575	$1,650	$1,725	$1,800
After-Tax IRR	-16.1%	-8.9%	-4.2%	-0.9%	1.7%	3.9%	6.0%	7.9%	9.7%
After-Tax NPV @ 5%	($1,835,098,612)	($1,460,814,760)	($1,087,918,481)	($741,117,959)	($439,714,744)	($146,862,491)	$144,265,658	$433,034,632	$722,957,063
Annual Operating Cost	6,034,378,812	6,411,527,488	6,788,676,164	7,165,824,840	$7,542,973,515	7,920,122,191	8,297,270,867	8,674,419,543	9,051,568,218
After-Tax IRR	7.0%	5.8%	4.5%	3.1%	1.7%	0.1%	-1.8%	-4.2%	-7.2%
After-Tax NPV @ 5%	$297,868,649	$112,477,162	($71,327,795)	($254,948,240)	($439,714,744)	($627,190,205)	($833,572,824)	($1,055,440,019)	($1,288,808,774)
Capital Cost	1,790,698,318	2,019,812,036	2,262,713,409	2,519,402,438	$2,789,879,122	3,074,143,462	3,372,195,458	3,684,035,109	4,009,662,416
After-Tax IRR	10.0%	7.5%	5.3%	3.4%	1.7%	0.1%	-1.3%	-2.5%	-3.7%
After-Tax NPV @ 5%	$460,265,969	$253,898,100	$35,112,025	($196,092,257)	($439,714,744)	($695,755,438)	($964,214,339)	($1,245,091,445)	($1,538,386,758)

Table 1.14 summarizes the after tax IRR and NPV5 for $100 incremental change in gold price from $1,200 to $2,200.

Table 1.14 Gold Price Sensitivity			
Gold Price ($/Oz)	NPV 5% ($M)	IRR (%)	Payback (Years)
$1200	(1,835)	-16.1	N/A
$1300	(1,336)	-7.2	N/A
$1400	(854)	-1.9	N/A
$1500	**(440)**	**1.7**	**10.8**
$1600	(50)	4.6	8.8
$1700	336	7.3	7.2
$1800	723	9.7	6.1
$1900	1,109	12.0	5.2
$2000	1,493	14.1	4.6
$2100	1,869	16.1	4.2
$2200	2,219	17.8	3.8

1.15 Qualified Persons Conclusions and Recommendations

1.15.1 Conclusions

- The Livengood Gold Project mineral resource is estimated at 731 million measured tonnes at an average grade of 0.61 g/mt (14.4 million oz at 0.3 g/mt cut-off) and 71 million indicated tonnes at an average grade of 0.56 g/mt (1.3 million oz at 0.3 g/mt cut-off), for a total of 802 million tonnes at an average grade of 0.61 g/mt (15.7 million ounces at 0.3 g/mt cut-off).
- The FS has converted a portion of these mineral resources into proven reserves of 434 million tonnes at an average grade of 0.69 g/mt (9,621,000 oz) and probable reserves of 20 million tonnes at an average grade of 0.70 g/mt (454,000 oz), for a total of 454 million tonnes at an average grade of 0.69 g/mt (10,075,000 oz).
- The FS mine plan would provide sufficient ore to support an annual production rate of approximately 577,600 ounces per year over an estimated 14 year mine life, producing approximately 8 million ounces.
- Metallurgical testwork has identified the preferred flowsheet of gravity recovery followed by whole ore leaching of the gravity tailing for an overall LOM recovery of 80.3%.
- The initial capital cost of a 100,000 t/d mill and associated 234,000 t/d mine is estimated at $2.79 billion.
- The mining cost is estimated at $1.67/t mined, process operating cost is estimated at an average of $10.45/t ore processed, and general and administrative costs of $0.89/ton ore processed.
- Using the trailing three year gold price of $1,500 per ounce, the project generates a minimal positive return.

1.15.2 Recommendations

- The optimized final pit contains over 44 Mt of inferred material that is above cutoff grade. Additional drilling may improve the classification of this material.
- The optimized final pit extends to the bottom of the current grade model. It is apparent that deeper drilling is warranted to develop material below the current grade model bottom.
- Metallurgical testing has consistently shown higher calculated head grades compared to the average assay obtained from composited drill core assays that make up the metallurgical test samples. This result is consistent with the bi-modal size distribution of the gold in the Livengood deposit. An extensive check assay program by metallic screen assays did also show small gains after adjusting for sample distribution; however the average grade of the metallic screen was about the same as the average fire assay. In this series of tests the metallic assay sample may have been too small, and contained more samples of higher grade materials than the average distribution in the database. More follow up is suggested as there is a significant amount of information that suggests the drill hole assays may be 10-15% lower than the actual grade.

- After the flow sheet was fixed for the purposes of the feasibility study, additional analysis by FLSmidth Knelson suggested that a 1-3% improvement in overall gold recovery may be achievable if an intensive cyanide leach reactor is used in place of the shaking tables contemplated in the study. There are potential space savings and operational improvements associated with use of a reactor, which together with the potential recovery improvement, warrant further study.
- Pursue mill throughput and capital cost studies to evaluate the optimum scale for the project.
- There is an opportunity to enhance mill head grades in early years by a more aggressive stockpile management strategy than is assumed in the feasibility study.

2.0 Introduction

2.1 Purpose of the Technical Report

Samuel Engineering, Inc. (SE) was commissioned by Tower Hill Mines, Inc. (THM) to complete a Feasibility Study on the Livengood Gold Project located in Alaska and to provide support to the Qualified Persons identified in Table 2.1 in their preparation of a Technical Report summarizing the results of the Study. This Technical Report is intended to conform to the standards and reporting requirements set forth in National Instrument 43-101 Standards of Disclosure for Mineral Projects, including Companion Policy 43-101CP and Form 43-101F1. The Technical Report supports the ITH July 23, 2013 news release announcing the results of the study.

2.2 Sources of Information

The authors of this Technical Report have utilized published and unpublished reports and literature for the information incorporated herein. The documentation reviewed and sources of information referenced are listed in Item 27 of this Technical Report.

Table 2.1 summarizes the section responsibilities of Qualified Persons contributing to this Technical Report. All persons and their respective companies listed are independent of ITH and THM, as defined by NI43-101.

Table 2.1
Qualified Persons Section Responsibilities

Qualified Person	Site Visit	Consultant	Section Responsibility
Neil Prenn	October 9 – 10, 2012	Mine Development Associates	15, 16.1, 16.2, 16.4 through 16.10 as well as the relevant portions of 1, 2, 25 and 26.
Charles Rehn	October 8 -11, 2012	AMEC Environment and Infrastructure	18, 20, 21, as well as the relevant portions of 1, 2, 25 and 26.
Tim Carew	May, 2012	Reserva International LLC	4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 19 and 23, as well as the relevant portions of 1, 2, 25 and 26.
Mike Levy	June 20 – 22, 2012	SRK Consulting (U.S.) Inc.	16.3
Richard Kunter	October 9 – 10, 2012	Samuel Engineering Inc.	3, 13, 17, 22 and 24, as well as the relevant portions of 1, 2, 25, & 26

2.3 Personal Inspection of the Livengood Property

The qualified persons inspected the Livengood Property on the dates shown in the above Table 2.1.

3.0 Reliance on Other Experts

For the purpose of this Technical Report, the Qualified Persons relied upon legal, political, environmental, or tax matters relevant to the Technical Report as identified below.

Tim Carew, QP, relied on information as to the ownership and legal status of the mineral tenures comprising the Livengood Gold Project provided by THM as of May 17, 2013 as set forth in Section 4.1, Section 30, and the relevant portions of Section 1.

Charles Rehn, QP, relied upon information with respect to the environmental status of the project and required permits for project development as provided by Denise Herzog, Environmental Manager for THM, as of May 17, 2013 as set forth in Section 20.1, 20.3 and the relevant portions of Section 1.

Charles Rehn, QP, relied upon information regarding the socioeconomic conditions in the project area and the anticipated results of the project thereon as provided by Rick Solie, Manager of Community and Government Relations for THM, as of May 17, 2013 as set forth in Section 20.6 and the relevant portions of Section 1.

Richard Kunter, QP, relied upon THM for the tax information relevant to and incorporated in the financial model developed as of July 19, 2013, as summarized in Section 22 and the relevant portions of Section 1.

This report is intended to be used by International Tower Hill Mines Ltd. subject to the terms and conditions of its agreements with Samuel Engineering, Inc. and the relevant Qualified Persons. Such agreements permit International Tower Hill Mines Ltd. to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

4.0 Property Description and Location

4.1 Property Description

The Livengood Gold Project property (Fig. 4.1) covers approximately 48,300 acres (19,500 hectares), all of which is controlled by ITH through its wholly-owned subsidiaries, THM and LPI. The Livengood Gold Project is comprised of multiple land parcels: 100% owned patented mining claims, 100% owned State of Alaska mining claims, and 100% owned federal unpatented placer claims; land leased from the Alaska Mental Health Trust (AMHT); land leased from holders of state and federal patented and unpatented mining and placer claims, and undivided interests in patented mining claims. The property and claims controlled through ownership, leases or agreements are summarized below.

4.1.1 100% owned patented mining claims

- U.S Mineral Survey 2447, located on lower Livengood Creek, subject to the December 2011 land purchase agreement described below and further subject to an agreement to allow Larry Nelson as agent for Heflinger to operate a placer mine on MS 2477 through December 31, 2014.
- U.S. Mineral Survey 1956, located on lower Gertrude Creek, subject to a reserved royalty of 5% of gross value held by Key Trust Company on behalf of the Luther Hess Trust, and further subject to an agreement to allow Mammoth Mining LLC to operate a placer mine on MS 1956 and F61249, F61256, F61257, and F61259 on lower Livengood Creek through December 31, 2015.
- With respect to portions of U.S. Mineral Survey 1626, located on Lower Amy Creek: 100% of No. 2 Above Discovery Any Creek, 100% of No. 3 Above Discovery Amy Creek, and 100% of Up Grade Association Bench.

4.1.2 100% owned State of Alaska mining claims

- 169 state claims acquired by purchase. (Appendix 30.1)
- 157 state claims acquired by location. (Appendix 30.2)

4.1.3 100% owned federal unpatented placer claims

- 29 federal unpatented placer claims, subject to the December 2011 land purchase agreement described below. (Appendix 30.3)

4.1.4 100% owned by Livengood Placers, Inc.

Livengood Placers, Inc. (LPI), a private Nevada corporation that is 100% owned by THM, is the record owner of the following:

- 29 patented claims, subject to the December 2011 land purchase agreement described below. (Appendix 30.4)
- 108 federal unpatented placer claims, subject to the December 2011 land purchase agreement described below. (Appendix 30.5)

- 24 State of Alaska mining claims, subject to the December 2011 land purchase agreement described below. (Appendix 30.6)

4.1.5 Leased property

Alaska Mental Health Trust Lease. A lease of the AMHT mineral rights having a term beginning July 1, 2004 and extending 19 years until June 30, 2023, subject to further extensions beyond June 30, 2023 by either commercial production or payment of an advance minimum royalty equal to $125% of the amount paid in Year 19 and diligent pursuit of development. The lease requires minimum work expenditures and advance minimum royalties which escalate annually with inflation. All advance minimum royalties are recoverable from production royalties. An NSR production royalty of between 2.5% and 5.0% (depending upon the price of gold) is payable to AMHT with respect to the lands subject to this lease. In addition, an NSR production royalty of 1.0% is payable to AMHT with respect to the unpatented federal mining claims subject to the Hudson/Geraghty lease described below and an NSR production royalty of between 0.5% and 1.0% (depending upon the price of gold) is payable to AMHT with respect to the lands acquired by THM as a result of the purchase of LPI pursuant to the December 2011 land purchase agreement described below. As of December 31, 2012, there were 9,970 acres included in the AMHT lease.

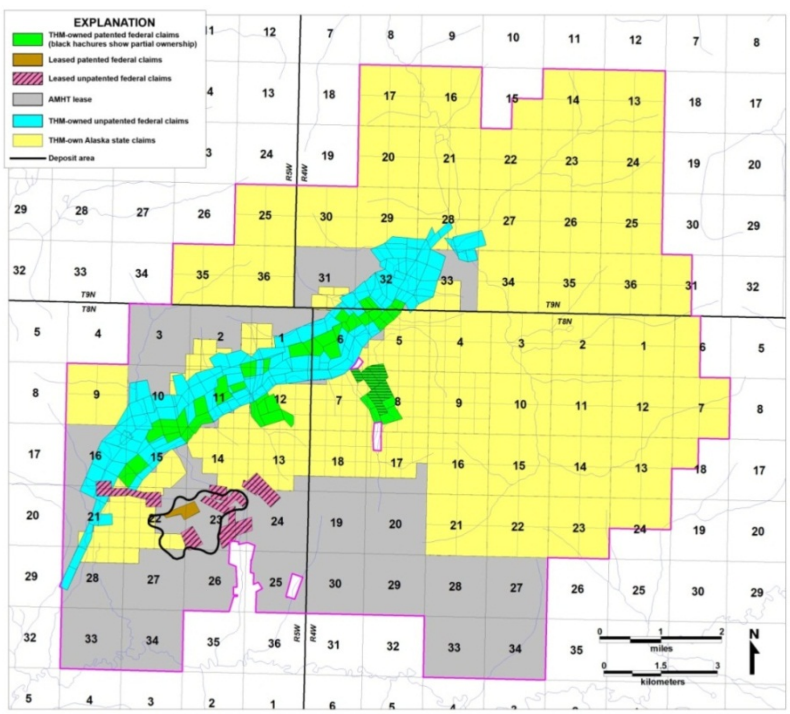


Figure 4.1 Map illustrating the Company's Livengood Gold Project land holdings
(as of February 13, 2013 by tenure type, referenced to the Fairbanks Meridian township, range and section grid).

- Hudson/Geraghty Lease. A lease of 20 unpatented federal lode mining claims having an initial term of ten years commencing on April 21, 2003 and continuing for so long thereafter as advance minimum royalties are paid and mining related activities, including exploration, continue on the property or on adjacent properties controlled by THM. The lease requires an advance minimum royalty of $50,000 on or before each anniversary date, (all of which minimum royalties are recoverable from production royalties). An NSR production royalty of between 2% and 3% (depending on the price of gold) is payable to the lessors. THM may purchase 1% of the royalty for $1,000,000. (Appendix 30.7)
- Griffin Lease. A lease of U.S. Mineral Survey 1990 having an initial term of ten years commencing January 18, 2007, and continuing for so long thereafter as advance minimum royalties are paid. The lease requires an advance minimum royalty of $20,000 on or before each anniversary date through January 18, 2017 and $25,000 on or before each subsequent anniversary (all of which minimum royalties are recoverable from production royalties). An NSR production royalty of 3% is payable to the lessors. THM may purchase all interests of the lessors in the leased property (including the production royalty) for $1,000,000 (less all minimum and production royalties paid to the date of purchase), of which $500,000 is payable in cash over four years following the closing of the purchase and the balance of $500,000 is payable by way of the 3% NSR production royalty.
- Tucker Lease. A lease of two unpatented federal lode mining claims and four federal unpatented placer claims having an initial term of ten years commencing on March 28, 2007, and continuing for so long thereafter as advance minimum royalties are paid and mining related activities, including exploration, continue on the property or on adjacent properties controlled by THM. The lease requires an advance minimum royalty of $15,000 on or before each anniversary date (all of which minimum royalties are recoverable from production royalties). THM is required to pay the lessor the sum of $250,000 upon making a positive production decision, payable $125,000 within 120 days of the decision and $125,000 within a year of the decision (all of which are recoverable from production royalties). An NSR production royalty of 2% is payable to the lessor. THM may purchase all of the interest of the lessor in the leased property (including the production royalty) for $1,000,000. (Appendix 30.8)

4.1.6 Patented claims (undivided interests less than 100%)

- An undivided 83.33% interest in that certain patented placer mining claim known as the “Kinney Bench” claim, included within U.S. Mineral Survey No. 1626 on lower Amy Creek.
- An undivided 2/9th interest in that certain patented placer mining claim known as the “Union Bench Association” claim, included within U.S. Mineral Survey No. 1626 on lower Amy Creek.
- An undivided 1/6th interest in that certain patented placer mining claim known as the “Bessie Bench” claim, included within U.S. Mineral Survey No. 1626 on lower Amy Creek.
- An undivided 1/3rd interest in those certain patented placer mining claims known as the “War Association” claim, the “Mutual Association” claim, and the “O.K. Fraction” claim, all included within U.S. Mineral Survey No. 2033 on lower Amy Creek.

On State of Alaska lands, the state holds both the surface and the subsurface rights. State of Alaska 40-acre mining claims require an annual rental payment of $35/claim to be paid to the state (by November

30th of each year), for the first five years, $70 per year for the second five years, and $170 per year thereafter. These rental rates are multiplied by 4 for each 160 acre claim. As a consequence of the annual rentals due, all Alaska State Mining Claims have an expiry date of November 30th each year. In addition, there is a minimum annual work expenditure requirement of $100 per 40-acre claim and $400 per 160-acre claim (due on or before noon on September 1 in each year) or cash-in-lieu, and an affidavit evidencing that such work has been performed is required to be filed on or before November 30th in each year. Excess work can be carried forward for up to four years. If the rental is paid and the work requirements are met, the claims can be held indefinitely. The work completed by THM during the 2012 field season was filed as assessment work, and the value of that work is sufficient to meet the assessment work requirements through September 1, 2016 on all State of Alaska mining claims.

Holders of State of Alaska mining claims are also required to pay a production royalty on all revenue received from minerals produced on state land during each calendar year. The production royalty rate is 3% of net income.

Holders of federal unpatented mining claims are required to pay an annual claim maintenance fee of $140 per 20 acres payable in advance on or before August 31 of each year.

All of the foregoing agreements are in good standing and are transferable. THM has taken reasonable steps to verify title to mineral properties in which it has an interest. Except for the patented claims, none of the properties have been surveyed.

Holders of Federal and Alaska State unpatented mining claims have the right to use the land or water included within mining claims only when necessary for mineral prospecting, development, extraction, or basic processing, or for storage of mining equipment. However, the exercise of such rights is subject to the appropriate permits being obtained.

December 2011 Land Purchase Agreement

In December 2011, ITH completed a transaction to acquire certain mining claims and related rights in the vicinity of the Livengood Gold Project. This acquisition included both mining claims and all of the shares of LPI. The aggregate consideration was $13,500,000 in cash plus an additional contingent payment based on the five-year average daily gold price (“Average Gold Price”) from the date of the acquisition. The contingent payment will equal $23,148 for every dollar that the Average Gold Price exceeds $720/oz. If the Average Gold Price is less than $720, there will be no additional contingent payment.

At initial recognition on December 13, 2011, the derivative liability was valued at $23,100,000. The key assumption used in the valuation of the derivative is the estimate of the future Average Gold Price. The estimate of the future Average Gold Price was determined using a forward curve on future gold prices as published by the Chicago Mercantile Exchange (CME) Group. The CME Group represents the merger of the CME, the Chicago Board of Trade (CBOT), the New York Mercantile Exchange (NYMEX) and its commodity exchange division, Commodity Exchange, Inc. (COMEX). Using this forward curve, ITH estimated an Average Gold Price based on actual gold prices to June 30, 2013 and projected gold prices from June 30, 2013 to the end of the five year period in December 2016 of $1,441 per ounce of gold. The amount payable in December 2016 of $16,700,000 represents the fair value of ITH’s derivative liability as at June 30, 2013 and will be revalued at each subsequent reporting period. No placer mineral reserves or mineral resources have been established on the ground subject to this agreement. However, records exist for 2,370 placer drill holes that have been completed on the subject ground between 1933 and 2011. Of these, the 945 holes completed between 1933 and 1984 were primarily 6 in churn drill holes. The 1,425 drill holes completed between 1984 and 2000 were 8 in RC rotary drill holes utilizing a center return tri-cone bit. All lands controlled by ITH, including the lands acquired pursuant to this agreement, are being evaluated as appropriate for integration into the Feasibility Study for the Livengood Gold Project.

4.1.7 Permits

THM has all of the necessary permits for exploration, geotechnical, and baseline data collection activities at the project. These permits are active and include Alaska Department of Natural Resources (hardrock exploration, temporary water use), U.S. Bureau of Land Management (plan of operations), U.S. Corps of Engineers (404 and nationwide wetlands), Alaska Department of Environmental Conservation (Section 401, stormwater), and Alaska Department of Fish and Game (fish habitat) authorizations. Permits required to support project development are discussed in Section 20.

4.1.8 Environmental Liabilities

With over 90 years of placer mining activity and sporadic prospecting and exploration in the region, there is moderate to considerable historic disturbance on the property. Some of the historic placer workings are now overgrown with willow and alder. The old mining town of Livengood is now abandoned except for more modern road maintenance buildings at the town site. ITH does not anticipate any significant obligations for recovery and reclamation of historic disturbance and there are no known significant existing environmental liabilities.

4.2 Location

The Livengood property is located approximately 70 miles (113 km) northwest of Fairbanks, Alaska in the Tolovana Mining District within the Tintina Gold Belt. The deposit area is centered near Money Knob, a local topographic high point. This feature and the adjoining ridge lines are the probable lode gold source for the Livengood placer deposits that lie in the adjacent valleys that have been actively mined since 1914 and produced more than 500,000 oz of gold.

The property lies in numerous sections of Fairbanks Meridian Township 8N and Ranges 4W and 5W. Money Knob, the principal geographic feature within the known deposit, is located at 65°30’16”N, 148°31’33”W.

The property straddles Highway 2 (also known as the Elliott Highway), a paved, all-weather highway linking the North Slope oil fields to Fairbanks, and adjoins the TAPS corridor, which transports crude oil from the North Slope south and contains the fiber-optic communications cable that may be used at the Livengood site (see Figure 4.2).

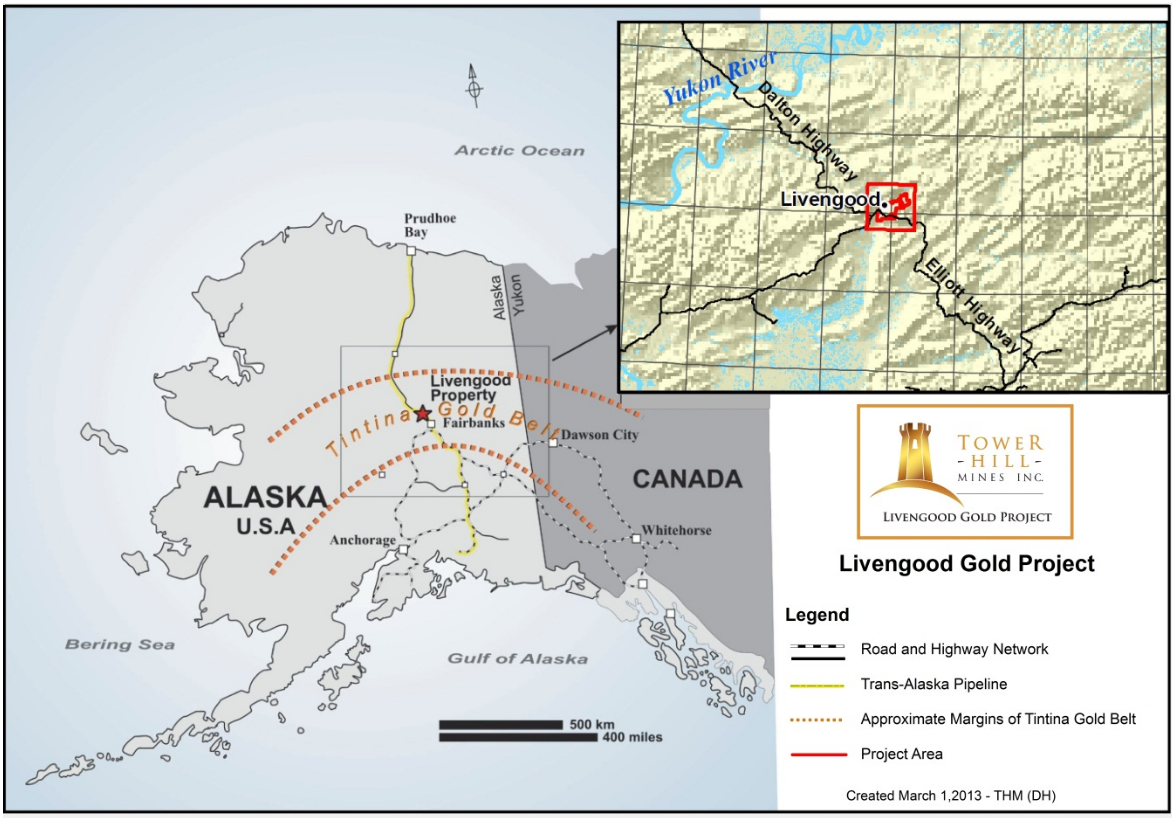


Figure 4.2 Project Location Map

5.0 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility

The Livengood property is located approximately 70 mi (113 km) northwest of Fairbanks, Alaska in the Tolovana Mining District within the Tintina Gold Belt. The property straddles Highway 2, a paved, all-weather highway linking the North Slope oil fields to Fairbanks, and adjoins the TAPS corridor. Locally, a number of unpaved roads lead from the highway into and across the deposit. A 3,000-foot (914 m) runway is located 3.73 mi (6 km) to the southwest near the former TAPS Livengood Camp and is suitable for light aircraft.

5.2 Climate

The site is approximately 40 miles (64 km) south of the Arctic Circle. The climate in this part of Alaska is continental with temperate and mild conditions in summer with average lows and highs in the range of 44° to 72°F (7° to 22°C). Winter is cold with average lows and highs for December through March in the range of -17°F to 23°F (-27°C to -5°C). The lowest lows are in the -40°F (-40°C), range. Annual precipitation is on the order of 15.7 in (400 mm) water equivalent. Winter snow accumulation snow pack depth is approximately 26 in (660 mm).

5.3 Local Resources and Infrastructure

5.3.1 Local Resources

The community of Minto (2012 pop. 223) is approximately 40 mi (64 km) southwest of the project, and Manley Hot Springs (2012 pop. 116) is approximately 80 mi (129 km) southwest of the project area at the western terminus of the Elliott Highway. The Fairbanks metropolitan area has a population of approximately 100,000 people, and comprises the regional center with hospitals, government offices, businesses and the University of Alaska, Fairbanks. The city is linked to southern Alaska by a north-south transportation and utility corridor that includes 2 paved highways, a railroad, an interlinked electrical grid, and communications infrastructure. The city has an international airport serviced by major airlines. Fairbanks services both the Fort Knox and Pogo gold mines, which operate year round. Skilled and unskilled labor to support mine development and operations will come primarily from the Fairbanks area, with a total labor force of over 40,000 workers.

5.3.2 Infrastructure

A study completed by Electric Power Systems has determined that the local utility in Fairbanks (Golden Valley Electric Association) can provide the 100 MW of power required for the project. The Project would be connected to the local grid by building a 50 mi (80 km) 230-kVa transmission line along the pipeline corridor.

SRK completed a regional hydrology study and determined that the average annual precipitation at the Livengood site at project elevation of 1,400 ft (427 m) amsl is 15.7 in (400 mm). Water balance studies completed by AMEC have concluded based on available and collected data that the site has an adequate water supply for the project as designed.

Two independent fiber-optic communications cables currently extend from Fairbanks to the North Slope, one along the TAPS, the other parallel to the Elliott Highway, both of which pass less than 2 mi (3.2 km) west of the project.

Project Area

The 48,300 acres (19,500 hectares) Livengood Gold Project property has sufficient area to support the required project facilities, including tailings and rock storage facilities and processing plant sites.

5.4 Physiography

The project area consists of rolling terrain of the Yukon-Tanana Uplands with a maximum elevation of 2,622 ft (800 m) at Livengood Dome. Upper and mid slopes are occupied by mature black spruce (*Picea mariana*), white spruce (*P. glauca*), paper birch (*Betula neoalaskana*), and quaking aspen (*Populus tremuloides*) forests. Low-lying areas and floodplains are dominated by poorly drained shrub and black spruce woodland communities often underlain by permafrost. Few lakes or ponds occur in the project area. Land disturbance from previous mining activity is highly conspicuous, particularly in Livengood and lower Goldstream Creeks.

6.0 History

6.1 General History

Gold was first discovered in the gravels of Livengood Creek in 1914 (Brooks, 1916) and led to the founding of the town of Livengood. Subsequently, over 500,000 oz of placer gold were produced and the former town of Livengood was established. From 1914 through the 1970's, the primary focus of prospecting activity was placer deposits. Historically, prospectors considered Money Knob, a topographic high within the currently known gold deposit, and the associated ridgeline to be the source of placer gold. Prospecting, in the form of dozer trenches, was carried out for lode mineralization in the vicinity of Money Knob, primarily in the 1950's. However, no significant lode production has occurred to date

Modern corporate exploration for lode gold mineralization under and around the Livengood placers on the adjacent hillsides, including the Money Knob area, was initiated in 1976, continued intermittently though 1999, and included extensive soil sampling, trenching, and 25 shallow drill holes. The most recent round of exploration of the Money Knob area began when AngloGold Ashanti (AGA) acquired property in 2003 and undertook an 8-hole RC program. The results from this program were encouraging and AGA followed up with an expanded soil geochemical survey which identified gold-anomalous zones in the Money Knob area. Based on the results of this and prior soil surveys and geological concepts, 4 diamond core holes were drilled in late 2004. The two drill programs intersected broad, apparently areally extensive zones of gold mineralization, but no further work was executed due to financial constraints and a shift in corporate strategy. In 2006, AGA sold the Livengood Gold Project to ITH. In the same year, THM drilled a 4,026-ft (1,227 m), 7-hole core program. The success of that program led to the drilling of an additional 14,436 ft (4,400 m) in 15 core holes in 2007 to test surface anomalies, expand the area of previously intersected mineralization, and advance geologic and structural understanding of the deposit. Subsequent programs have continued to expand the resource, leading to consideration of development of the deposit and concomitant geotechnical, engineering, and metallurgical work, along with the collection of initial environmental baseline data. As of the end of 2012, AGA and THM completed exploration and delineation drilling totaling 575,078 ft (175,284 m) in 604 RC holes and 137,915 ft (42,037 m) in 147 core drill holes.

6.2 Historical Mineral Resource Estimates

A historical mineral resource estimate for portions of the property (230,000 oz of placer gold) is available as described in ITH press release Feb. 27, 2012.

There are no known historical mineral resource estimates for hardrock minerals.

7.0 Geological Setting and Mineralization

7.1 Regional Geology

The Livengood deposit is hosted by rocks of the Livengood Terrane (Figure. 7.1), an east–west belt, approximately 150 mi (240 km) long, consisting of tectonically interleaved assemblages, which include: i) the Amy Creek Assemblage, a sequence of latest Proterozoic and/or early Paleozoic basalt, mudstone, chert, dolomite, and limestone; ii) a Cambrian ophiolite sequence of mafic and ultramafic sea floor rocks thrust over the Amy Creek Assemblage, in turn overthrust by iii) a sequence of Devonian clastic sedimentary, volcanic, and volcaniclastic rocks (Athey, et al., 2004). The Devonian rocks are the dominant host to the mineralization at Livengood and have been informally subdivided into "Upper Sediments" and "Lower Sediments" stratigraphic units, separated by volcanic rocks ("Volcanics" or "Main Volcanics"; Figure 7.2). The Devonian assemblage was overthrust by a second klippe of Cambrian ophiolite and structurally intercalated cherty sedimentary rocks ("Money Knob", Figure. 7.2). All of these rocks are intruded by post-thrusting, Cretaceous (91.7 – 93.2 My, Athey, Layer, and Drake, 2004) multiphase monzonitic and syenitic dikes; gold mineralization is spatially and temporally associated with these intrusive rocks.

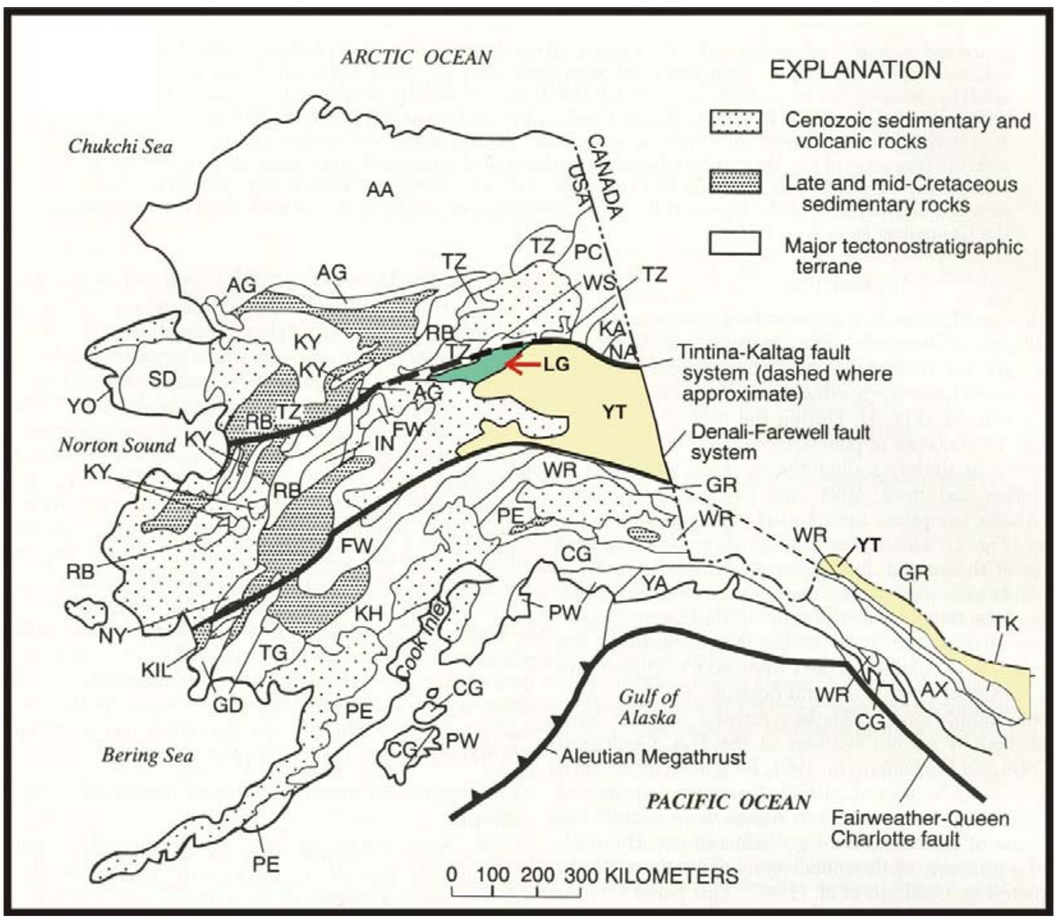


Figure 7.1 Terrane Map of Alaska showing Livengood Terrane (LG; red arrow).

The heavy black line north of the Livengood Terrane is the Tintina Fault. The heavy black line to the south of the Livengood and Yukon-Tanana Terrane (YT) is the Denali Fault. The Tintina Gold Belt lies between these two faults (after Goldfarb, 1997).

7.2 Mineralization and Alteration

Gold mineralization is associated with disseminated arsenopyrite and pyrite in volcanic, sedimentary and intrusive rocks, and in quartz veins cutting the more competent lithologies, primarily volcanic rocks, sandstones, and to a lesser degree, ultramafic rocks. Mineralization appears to be contiguous over a map area approximately 2.5 km^2 (Figure 7.2); a 0.1 g/mt grade shell averages 280 m thick and drilling has not closed off the deposit at depth. The stronger zones of mineralization are associated with areas of more abundant dikes. South of the Lillian Fault (Figures 7.2 and 7.3) individual mineralized envelopes are tabular and follow stratigraphic units, particularly the Devonian volcanics, or lie in envelopes that dip up to 45° to the south, mimicking the structural architecture and attitude of the diking. On the north side of the Lillian fault mineralization is similar in style and orientation and hosted primarily in steeply dipping Upper Sediments. Three principal stages of alteration are currently recognized; in order from oldest to youngest, these are characterized by biotite, albite, and sericite. Arsenopyrite and pyrite were introduced primarily during the albite and sericite stages. Gold correlates strongly with arsenic and occurs primarily within and on the margins of arsenopyrite and pyrite grains. Carbonate was introduced with and subsequent to these stages. Dating of the sericite alteration (Athey, Layer, and Drake, 2004) indicates that mineralization and alteration were contemporaneous with the emplacement of the dikes.

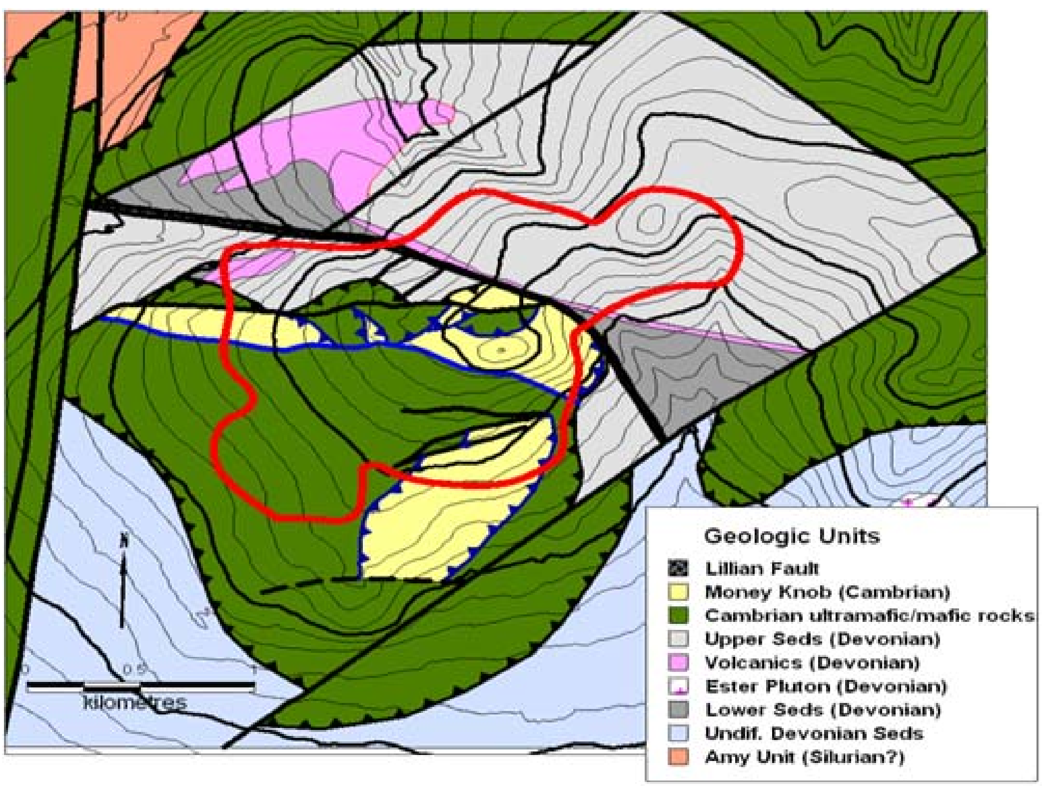


Figure 7.2 Generalized geologic map of the Money Knob area based on geologic work by THM.
(Red outline is the surface projection of the gold deposit)

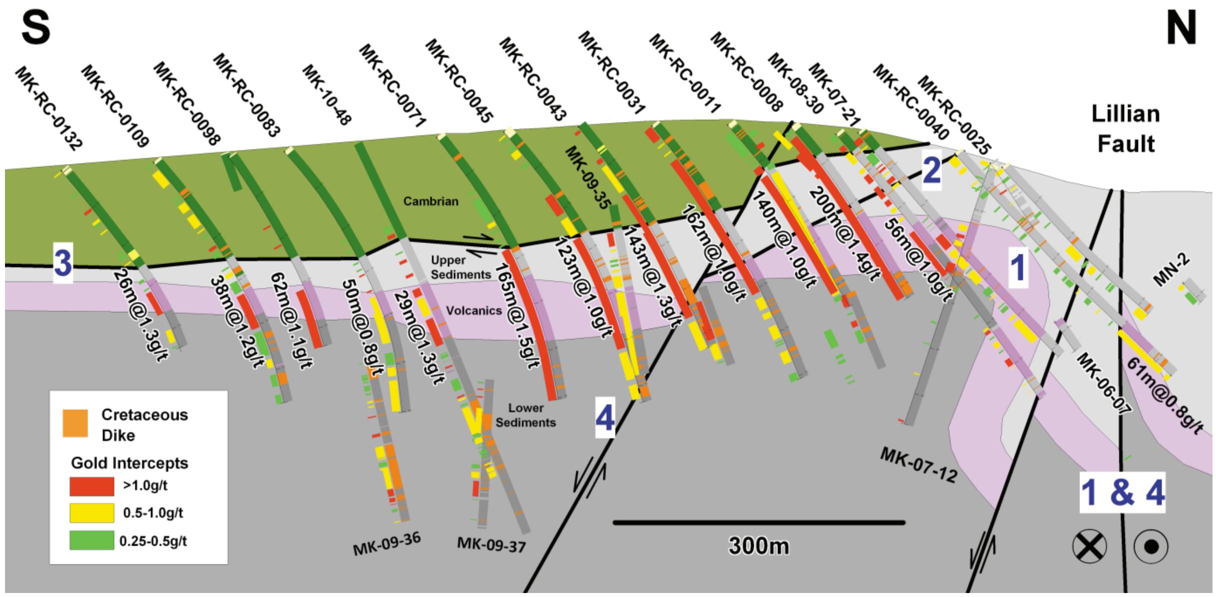


Figure 7.3 Cross Section through the Deposit

(Blue numbers indicate possible sequence of structural events: 1. Fold thrust development in the Permian(?). 2. NE-trending cross faults. 3. Thrust emplacement of Cambrian sheet. 4. Extensional collapse, all of which pre-date dike emplacement and coeval mineralization)

8.0 Deposit Types

Among gold deposits of the Tintina Gold Belt, Livengood mineralization is most similar to the dike and sill-hosted mineralization at the Donlin Creek deposit, where gold occurs in narrow quartz veins associated with dikes of similar composition (Ebert, et al., 2000). The age of the intrusions and the coincidence of mineralization and intrusive rocks are typical of those of other nearby gold deposits of the Tintina Gold Belt, which have been characterized as intrusion-related gold systems (Newberry and others, 1995; McCoy and others, 1997). For these reasons Livengood is best classified with these deposits.

9.0 Exploration

9.1 Exploration History

Multiple companies have explored the Livengood area as outlined above (Section 6). Among them Cambior Inc. was chiefly responsible for outlining the sizeable area of anomalous gold in soil samples, which THM expanded between 2006 and 2010 (Figure 9.1), collecting an additional 843 samples. These samples helped improve definition of anomalous gold in soil on the southwest side of Money Knob and to the northeast from Money Knob). The THM and Cambior samples were collected where C- horizon material was available; the -80 mesh fraction was analyzed for gold and a multi-element package. The currently known deposit is defined by the most coherent and strongest gold anomaly, but represents detailed evaluation of only about 25% of the total gold-anomalous area.

During 2011, THM completed an IP/Resistivity survey covering the deposit and gold-anomalous soil geochemistry to the northeast, where loess and frozen ground have prevented complete geochemical coverage. The objective of the survey was to establish the geophysical signature of the deposit and identify similar signatures elsewhere in the district to prioritize exploration drilling. When evaluation of the data is complete it should help guide exploration outside of the known deposit.

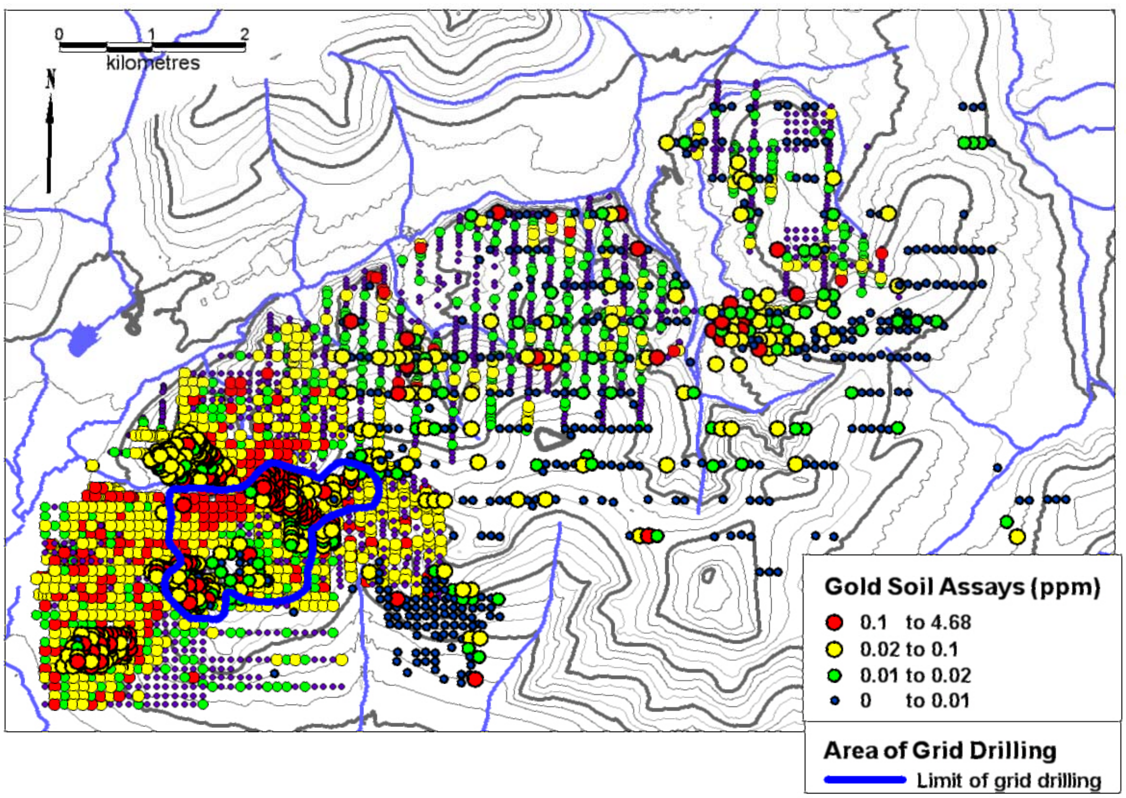


Figure 9.1 Plot of Gold Values in Soil Samples
(The surface projection of the known deposit is outlined in blue in the lower left corner of the figure)

10.0 Drilling

THM conducted drilling programs on the Livengood property from 2006 through 2012 (Figure 10.1) utilizing both core and reverse circulation (RC) drilling. These programs initially outlined mineralization in the Core Zone south of the Lillian fault in 2006 and subsequently in the Sunshine Zone area north of the fault beginning in 2009 through step-out drilling and drill testing of areas with anomalous values in surface soil samples. Through completion of the delineation drilling at the end of the 2011 season THM completed a total of 712,994 ft (217,321 m) of exploration and delineation drilling, of which 575,078 ft (175,284 m) was RC drilling and 137,915 ft (42,037 m) was core drilling.

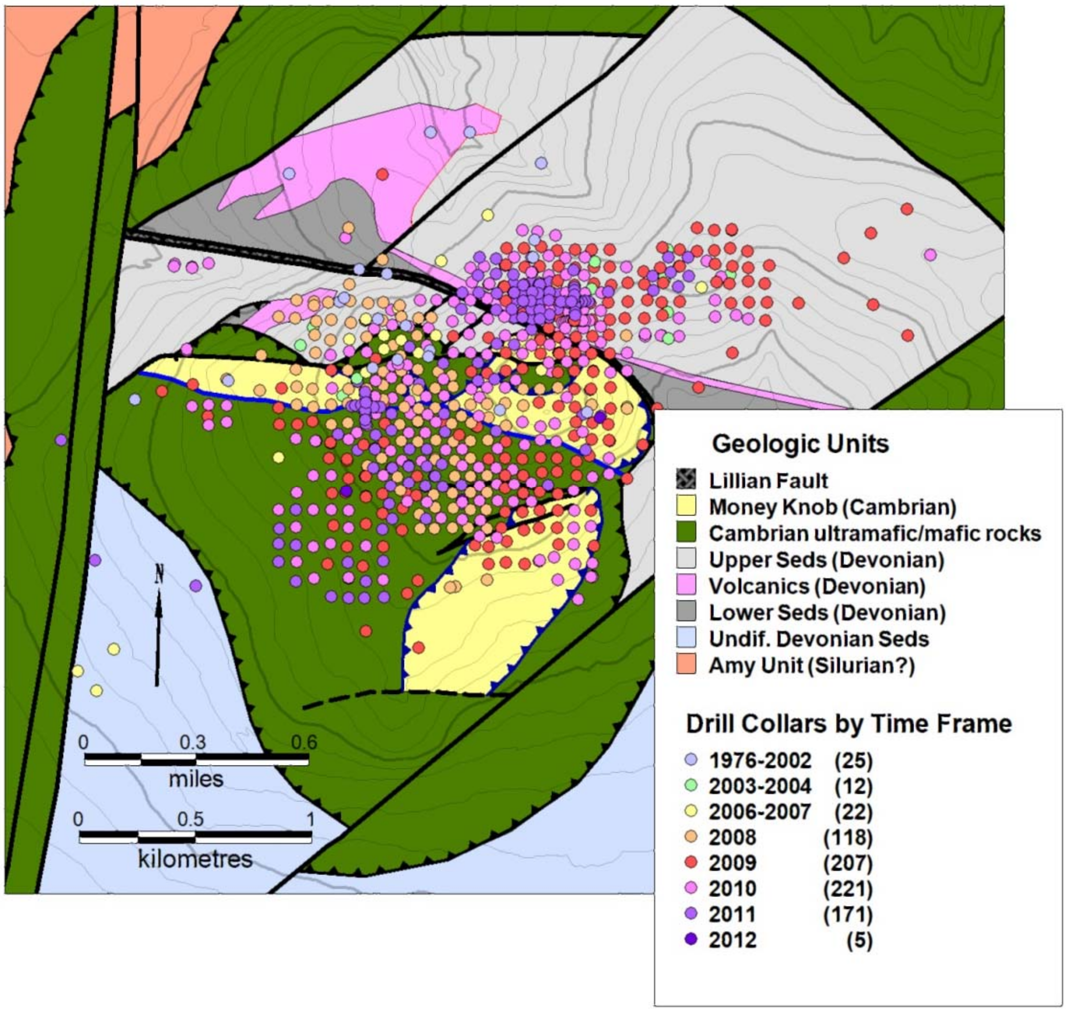


Figure 10.1 Distribution of Resource / Delineation Drill Holes in Money Knob area Over Time

(All holes completed after 2004 were drilled by THM. Drilling illustrated through 2011 dedicated to exploration and delineation; 2012 holes shown are geotechnical.)

Nearly all resource drill holes at Livengood have been drilled in a northerly direction at an inclination of -50° (RC) and -60° (core) in order to best intercept the south-dipping structures and mineralized zones as close to perpendicular as possible. A few holes have been drilled in other directions to test other features and aspects of mineralization. Initial grid drill holes were spaced at 246 ft (75 m) along lines 246 ft (75

m) apart; subsequent infill drilling in the center of the 246 ft (75m) squares brings the nominal drill spacing to 164 ft (50 m) for a significant portion of the deposit.

Reverse circulation holes are bored and cased for the upper 0-100 ft (0-30 m) to prevent down hole contamination and to help keep the hole open for ease of drilling at greater depths. Recovery of sample material from RC holes is done via a cyclone and dry or wet splitter, according to conditions. Drill cuttings are collected over the course of each 5 ft (1.52 m) interval and captured for a primary sample, an equivalent secondary sample ("met" sample) and a third batch of chips for logging purposes.

Diamond core holes represent 24% of the footage (meterage) drilled. Core is recovered using triple tube techniques to ensure good recovery (>92%) and confidence in core orientation. The core is oriented using either the ACT™ or the EZMark™ tools.

In the deposit, drill hole locations are determined by sub-meter differential GPS surveys at the drill collar. The initial azimuth of drill holes is measured using a tripod mounted transit compass in conjunction with a laser alignment device mounted on the hole collar. Down hole surveys of RC drill holes and most core holes are completed using a gyroscopic survey instrument manufactured by Icefield Tools Corporation. Some core holes have been surveyed using the Reflex EZ Shot™ system. Results of surveys and duplicate tests show normal minor deviation in azimuth and inclination for drill holes (Brechtel, et al., 2011).

Factors potentially affecting the validity of results are, for core drilling, core recovery, and, for RC drilling, cyclicity and down hole contamination; these are addressed in the section on data verification (Section 12).

11.0 Sample Preparation, Analyses and Security

11.1 Sample Collection, Procedures and Security

THM samples all holes from surface to total depth. Since 2009, core from the deposit is quick-logged in the split tube at the drill site, then boxed and transported by the geologist to the core logging facility in camp for detailed logging and sample markup. Samples lengths, based on geologic criteria, range from 1 ft (0.3 m) to 5 ft (1.52 m). After logging, the core is sawn in half longitudinally and sampled on the specified intervals into bags. Past procedures, largely similar, are documented in Brechtel, et al., 2011.

RC samples (an "original" and a duplicate) are collected at the rig as described above directly into bar-coded bags, which are printed and coded with the hole number and sample interval. The samples are transported by project personnel from the drill site to camp, where they are logged in using a bar code reader slaved to a portable Thermo Fisher Scientific NITON™ XRF analyzer (used to collect geochemical data on all the RC samples).

When all samples for a drill hole are accounted for, a sample shipment is assembled by adding control samples for quality assurance and quality control (QA/QC). One standard (certified gold content) purchased from RockLabs or Geostats and one blank (below detection limit for gold) are added for every 18 drill samples in the shipment. Shipment paperwork is prepared for the lab and includes instructions for the preparation of prep duplicates (1 per 20 drill samples). All core samples are weighed and the weights recorded. The shipment is bagged in sealed containers and the seal numbers are recorded on the sample submittal form. The shipments are picked up at the project site by USA, Inc. (ALS) lab personnel, who acknowledge receipt and custody of the samples by signing a copy of the submittal form, which is retained in the project files.

11.2 Lab Procedures

Per THM instructions, all drill samples are weighed on receipt at the ALS prep lab in Fairbanks. RC samples are then dried and re-weighed. The samples are crushed (-10 mesh) and a 1 kg fraction is pulverized. Aliquots for analysis and the coarse rejects are also weighed. The tracking of weights from the field through the sample preparation process permits the detection of sample switches and/or number transcription errors. ALS forwards pulps from the Fairbanks prep lab to Vancouver or Nevada for analysis. Samples are analyzed by standard 50 g fire assay/AA finish for the gold determinations. All core samples and select RC drilling samples are also submitted for multi-element ICP-MS analyses using a 4-acid digestion technique. These are standard analyses for the exploration industry and are performed to a high standard. ALS is accredited by the Standards Council of Canada, NATA (Australia) and also has ISO 17025 and 9001 accreditation.

11.3 QA/QC Procedures and Results

ALS analytical reports are reviewed when received to: i) verify shipped vs. received weights for core and dry weights vs. coarse rejects plus sample aliquots for all samples to check for weight loss or gain that indicates sample mixing, switches or transcription errors, and ii) blanks and standards with out of range values (±10% for standards and 3x detection limit for blanks). Errors are flagged and reported to ALS for resolution. If required, samples with questioned results and the surrounding 10 samples are re-analyzed. Upon satisfactory resolution of any discrepancies, new analytical certificates are issued by ALS.

In addition, duplicate gold pulp analyses and check assays with a second lab are requested on an annual basis. These analyses, and those for field duplicates and prep duplicates, are examined to evaluate the laboratory prep and analytical process. These data indicate no systematic bias introduced in the prep or gold assaying procedures, but do show scatter in the gold data, particularly at higher grades, which is interpreted as the product of nugget effect, typical for deposits with free gold. Results and detailed analysis of the data for 5,466 prep duplicates, 5,173 pulp duplicates, standard materials, and check assays are reported in Brechtel, et al., 2011.

As a further check on the integrity of gold assaying, 2,096 samples were selected for 1kg screen fire assays for comparison to the standard 50 g fire assay/AA finish results routinely used by THM (Brechtel, et al., 2011). The mean gold grade for the samples is very similar for both data sets (within 0.1%). In detail, the data suggest the standard fire assays are lower or equal to the screen fires at gold grades up to 9 g/mt. At grades over 9 g/mt the 50 g assays may over-represent the gold grade, but at Livengood the number of samples at these grades is very small, <0.2% of the sample population.

11.4 Data Collection, Entry and Maintenance

Two master project databases are maintained in Microsoft™ Access by THM: i) a drill hole database containing all the data collected in the field, including drill hole locations, down hole surveys, geologic logging, NITON™XRF geochemistry, and sample interval data; and ii) an assay database that is the repository of all laboratory generated analytical data.

Data gathered electronically in the field is uploaded daily to the drill hole database utilizing custom queries. These data include RC drill logs and NITON™XRF geochemistry, collar locations, and gyroscopic down hole survey data. Core logging and sampling information is collected on paper and hand entered. Once data is entered, database internal subroutines check the data for errors (i.e. gaps and overlaps in logging or sampling intervals) and data format consistency. Analytical data from ALS is received electronically, uploaded to the assay database, and merged with the sample interval data read from the drill hole database. Customized queries check blank and standard analyses and flag out of range values.

The databases and all raw data are stored on a hard drive in the field office which is copied automatically daily to the server in the Fairbanks office, where tape backup of the server is conducted nightly with rotation_of tapes into offsite storage.

11.5 Adequacy of Procedures

Reserva International has witnessed and reviewed sample and data collection in the field, inspected the ALS's Fairbanks prep lab, reviewed the QA/QC procedures and analysis, and completed a data validation check on a random sample (10%) of the subset of the resource drill hole data. Reserva International is satisfied that the THM data collection, management and verification procedures are adequate and were diligently followed.

12.0 Data Verification

12.1 Third Party Confirmation

In addition to the reviews described in Section 11, Reserva International has examined outcrop and core during site visits (Carew, et al., 2010) and his observations are consistent with those reported in THM documents. Drill logs, sections, and maps are of high quality.

From 2006 through 2009 Dr. Paul Klipfel annually, and independently, collected a total of 80 samples from outcrop (2006), and both RC and core drill holes for gold analysis. Comparison of the results to THM's original gold assays indicates a scatter due to the nugget effect, but no systematic bias in the data (detailed discussion in Brechtel, et al., 2011). Reserva International has reviewed the results of the 2009 verification sampling and agrees with the conclusions regarding accuracy, precision and lack of bias. Additionally, in 2010, 39 drill samples were collected for verification. The 2010 samples show a good overall correlation with the results reported by THM, with precision similar to or better than the analyses reported by THM (Brechtel, et al, 2011). Reserva International has not verified all sample types or material reported, but to the best their knowledge, THM has been diligent in their sampling procedures and efforts to maintain accurate and reliable results.

12.2 RC vs. Core Drilling

The use of RC drilling beneath the water table on other projects has resulted in inaccurate assay data due to cyclicity and/or down hole contamination. As THM has used both RC and core drilling above and below the water table, THM has conducted a detailed evaluation of the RC data and comparison of the gold data for the two drilling techniques to check the accuracy of the RC data and evaluate any potential bias between the two drilling methods.

During RC drilling cyclic contamination can occur if the driller fails to clean the drill hole prior to the addition of drill rods and can be detected by grade spikes that occur with the addition of rods. Examination of the RC database indicated potential cyclic contamination in portions of 6 holes and one entire drill hole (Brechtel et al., 2011). The data for the affected intervals have been removed from the database used for resource calculation.

Detectable migration of mineralized material down hole when drilling beneath the water table can occur following penetration of a high grade intersection and is manifested by a monotonic grade decrease for samples below the intersection. The frequency of monotonic decreases beneath high grade intersections in both core and RC drill holes is statistically comparable; significant down hole contamination is not indicated for the RC drilling (Brechtel, et al., 2011).

Early in 2011 THM modeled the distribution and mean of gold grades for both types of drilling (Brechtel, et al., 2011). Table 12.1 compares the mean values by stratigraphic unit. The data suggest that, on average for the deposit, core gold grades are 4% lower than RC grades. The most notable contrast occurs in the Sunshine Zone above the water table, where the core grade is 20% lower than the RC grade.

Based on this work, an area in the Sunshine Zone (Area 50, Figure 12.1) and above the water table was selected for detailed drilling to further evaluate the relationship between core and RC results where the discrepancy was the greatest. Area 50 was drilled out to nominal 123 ft (37.5 m) spacing to the water table (approximately 492 ft (150 m) below surface). The drilling included a mix of HQ core (7 drill holes sawn in half for sampling), PQ core (23 holes sampled whole), and RC drilling (28 holes), providing the opportunity to re-examine the difference between core and RC sampling. All Area 50 samples were composited to 5-meter lengths and grades modeled; the results are illustrated in Figure 12.2. For Area 50, the modeled mean PQ grade is 92% of that calculated for RC drilling, and the modeled HQ grade is 71% of the RC grade and 77% of the PQ grade, indicating that sawn HQ core recovers significantly less gold than either whole PQ core or RC sampling; PQ sampling is closer to RC sampling, but still lower. Ordinary kriging of the resource within the Area 50 volume by sample type bears out this relative relationship (contained gold based on PQ core is 94% of that based on RC; for HQ the contained gold is 80% of that calculated using RC; Table 12.2).

Mineralization in the Sunshine Zone (Area 50) is characterized by quartz-carbonate-sulfide veinlets that have a significantly higher proportion of associated coarse gold relative to the remainder of the deposit. Where the mineralized material is oxidized, the carbonate and sulfide is leached out, rendering the veinlets friable with the core often breaking along them. The most probable explanation for the discrepancies in grade are: i) loss of gold due to less than 100% core recovery (average 92%), and ii) progressive loss of gold with increased handling of the sample material, e.g. the HQ core was boxed, then taken from the boxes and sawn in half lengthwise then bagged (most handling), the PQ core was boxed, then transferred whole directly into sample bags (less handling), and the RC samples were bagged directly on the rig (no handling). This effect would be most pronounced in oxidized zones of the deposit but could also occur in unoxidized rocks if they are badly broken and core recovery is less than 100%. Because the gold at Livengood is relatively coarse, another contributing factor may be the relative sample volume (e.g. RC with a 5 in (127 mm) diameter, whole PQ3 core with an 3.3 in (83 mm) diameter, and HQ3 core with a 2.4 in (61 mm) diameter that has been halved). HQ core comprised 13% of the composites used to calculate the August 2011 resource. Based on the results above, it can be concluded that the resource is not significantly overstated and may be slightly understated.

Table 12.1
Comparison of Modeled Gold Grades Between Core and RC Drilling by Stratigraphic Unit

Unit	Core vs. RC Difference
Kint (dikes)	-6%
Cambrian	-3%
Main Volcanics	-3%
Sunshine Zone Upper Sediments above water table	-20%
Sunshine Zone Upper Sediments below water table	+6%
All Data	-4%

Table 12.2 Calculated Resources for Area 50 by Drill Sample Type (ordinary Kriging of 10 m Composites, 0.25 g/mt cutoff)						
Drill Sample Type	**Tonnes (millions)**	**Tonnage Ratio**	**Gold Grade (g/mt)**	**Grade Ratio**	**Gold (oz)**	**Gold Ratio**
RC drilling	16.73		0.575		309,114	
PQ drilling, PQ/RC ratios	15.95	0.953	0.566	0.984	289,981	0.938
HQ drilling, HQ/RC ratios	15.14	0.905	0.510	0.887	248,061	0.802
HQ/PQ ratios		0.949		0.901		0.855

12.3 Resource Verification Drilling

Two areas of the deposit, the Core and Sunshine crosses, were selected for 49 ft (15 m-spaced reverse circulation (RC) in-fill drilling on crosses with north-south and east-west legs 492 ft (150 m) in length (Figure. 12.1) to demonstrate continuity of grade and, thereby, confidence in the resource based on the wider spaced grid drilling defining the resource. A third area, Area 50, measuring 195 m by 240 m at the surface, was drilled on a 123 ft (37.5 m) grid with alternating core and RC drilling. Two resources were generated for each volume using ordinary kriging on samples composited to 33 ft (10 m) lengths: the first including those portions of the 164 ft (50 m) grid drilling (May 2011 resource) within the volume; and a second using both the grid and close-spaced drilling within the same volume. On average, the effect of the increased drilling density on tonnage, grade, and contained oz of gold is negligible (less than 1%; see Table 12.3), indicating that current grid spacing adequately defines the resource.

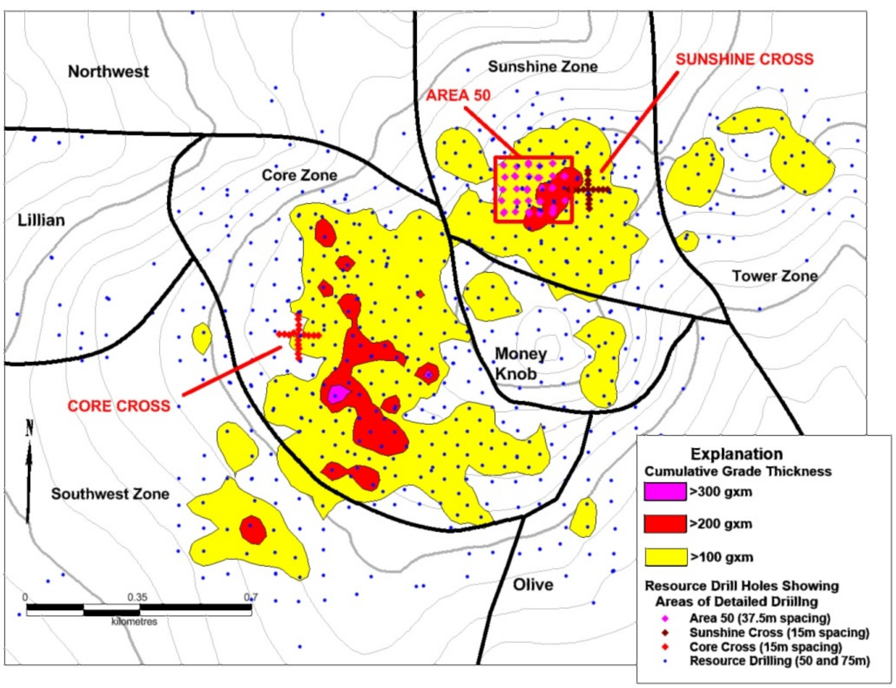


Figure 12.1 Map Showing Location of Areas of Detailed Drilling
(Area 50, Sunshine Cross and Core Cross)

Figure 12.2 is based on 869 RC Composites, 753 PQ Core Composites, and 203 HQ Core Composites (all composited to 16.4 ft (5 m)). The modeled grade means for the RC, PQ and HQ composites in Area 50 are 0.597, 0.549 and 0.424 g/mt gold respectively.

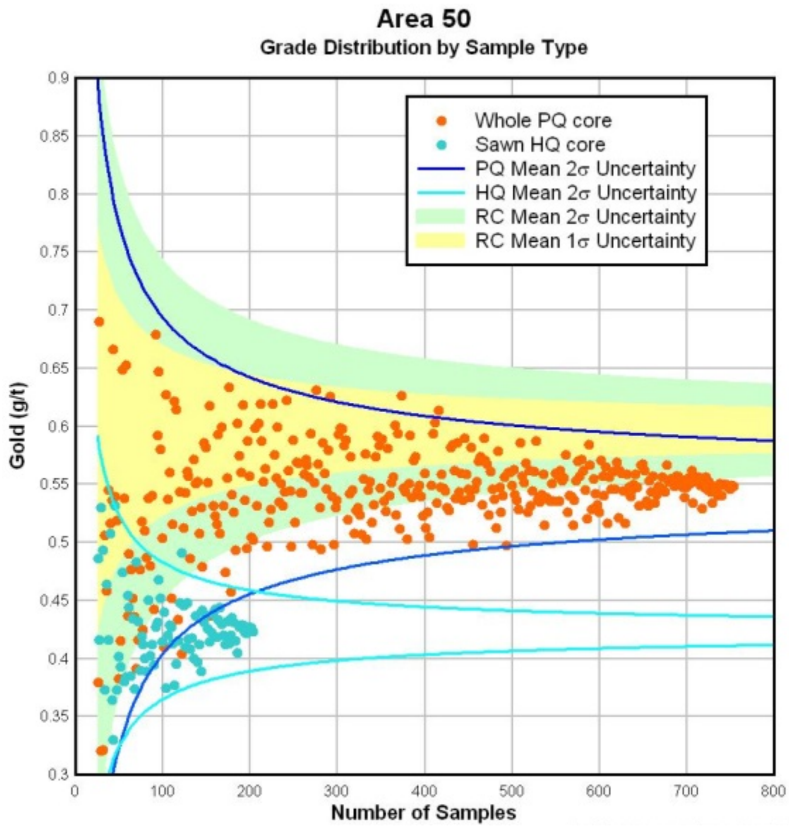


Figure 12.2 Models for RC, Whole PQ, and Sawn HQ from Area 50

Table 12.3 Calculated Resources for the Core Cross, Sunshine Cross and Area 50 (ordinary kriging, 0.25 g/mt cut-off)						
Area, Drill Hole Spacing	**Tonnes (millions)**	**Tonnage Ratio (all/grid)**	**Gold Grade (g/mt)**	**Grade Ratio**	**Gold (toz)**	**Gold Ratio (all/grid)**
Core Cross, 50m grid & 15m infill	15.67		0.481		242,401	
Core Cross, 50m grid drilling only	15.37	1.020	0.477	1.008	235,715	1.028
Sunshine Cross, 50m grid & 15m	9.82		0.553		174,647	
Sunshine Cross, 50m grid drilling	9.81	1.001	0.566	0.977	178,556	0.978
Area 50, all drilling (37.5m)	16.04		0.562		289,685	
Area 50, 50m grid drilling	16.13	0.994	0.550	1.022	285,136	1.016
All areas (averages)		**1.005**		**1.002**		**1.007**

13.0 Mineral Processing and Metallurgical Testing

13.1 Metallurgical Testwork

13.1.1 Introduction

Prior to the commencement of this Feasibility Study in January 2012, in late 2010 THM engaged FLSmidth to undertake a pre-feasibility study for the Livengood Gold Project. As that engineering work advanced and the project concepts evolved, in mid-2012 THM decided to not complete the pre-feasibility study and to instead redirect its efforts to focus on this Feasibility Study. Although the pre-feasibility study was halted prior to its completion, much of the data obtained during that work was considered during the development of this Feasibility Study. This included metallurgical testwork completed by Hazen Research, Kappes, Cassidy & Associates, AMTEL, Resource Development (RDi) and Pocock Industrial Inc.

13.1.2 Feasibility Study Metallurgical Testwork

A test program was completed at SGS, Vancouver on representative mineralized core samples from the Livengood Gold Project. The goal of the program was to develop the optimum flowsheet for the different ore types expected. The testing consisted of sample preparation, mineralogy, comminution, gravity separation, CIL, flotation and CIL, cyanide detoxification and rheology.

The test program was split into several phases; Optimization, Variability, Comminution, Cyanide Detoxification, and Solid Liquid Separation. Optimization samples were designed to be large composites that represent the average grades of each ore type. The variability samples were selected to test the geologic extremes of each ore type and to establish how the metallurgical response changes for each of the rock types based on variations in feed grade and other parameters. Comminution testing was conducted on portions of the samples that comprised the optimization samples, as well as variability samples. Cyanide detoxification and solid liquid separation tests were performed on testwork tailings.

<u>Sample Selection</u>

In order to begin sample selection for FS metallurgical testwork, the ore from the project was classified into rock types representing the 912 million ton resource developed as part of the prefeasibility study. The objective was to focus on the rock types and grades that would be most prevalent in the processing of Livengood ore. A composite of each rock type to be tested was carefully constructed to be both representative of the average grade of each ore type and also representative of the histogram of the number of samples comprising each grade range in making up the average grade.

To select the samples, the first step was to evaluate the preliminary mine production schedule to assess the tonnage and average annual grade distribution for each rock type over the life of the mine, see Table 13.1 below. Although RT4 and RT6 accounted for smaller proportions of the ore than some of the other ore types, these rock types will be a large proportion of the ore processed in the early years of the operation. During these early years, the plant operators will be adjusting the plant operation and it will be important to understand how the feed to the plant will vary and what the metallurgical responses will be. Conversely, although RT7 Bleached accounts for a significant amount of the plant feed, the majority of it will be processed later in the mine life. Optimization of this ore type was initially deemed to be less important than the four rock types tested. Based on the results from previous metallurgical testing, an understanding of the geology, and the proportions of rock types to be processed based on the production statistics, the optimization testing was conducted on four composite samples. For the optimization phase, these included Rock Types RT4, RT5, RT6 and RT9. The variability tests included these four rock types as well as Rock Type RT7. A mini optimization of RT7 was completed after the initial determination from the variability testing indicated that the unbleached RT7 experienced poor recovery and the RT7 Bleached material was selected for separate optimization testing.

The proportions of the six rock types which comprise the 500 million ton life of mine (LOM) reserve are displayed below in Table 13.1. Five of these, RT4, RT5, RT6, RT7 Bleached and RT9, comprising 98% of the reserve, were tested as part of this FS.

Table 13.1
Livengood LOM Summary by Rock Type

Rock Type		**LOM (Tonnes)**	**%**
RT4	Cambrian	63,377,103	13.9
RT5	Upper Sediments-Sunshine	128,095,489	28.2
RT6	Upper Sediments	83,666,788	18.4
RT7 Bleached	Lower Sediments-South of Lillian Fault	55,025,328	12.1
RT8	Volcanics-North of the Lillian Fault	9,028,604	2.0
RT9	Volcanics-South of the Lillian Fault	115,430,336	25.4
Total		**454,623,648**	**100.0**

Table 13.2 displays the percentage of each rock type to be processed by year. Figure 13.1 displays a graphical depiction of the rock type to be processed at various periods.

Table 13.2
Livengood LOM Summary by Rock Type

			Year				
	RockType		**1-3**	**4-6**	**7-10**	**11-14**	**LOM**
Primary	4	Tonnes	23,291,219	16,912,297	10,426,483	12,747,104	63,377,103
		%	25.0%	17.00%	7.80%	9.90%	13.90%
	5	Tonnes	28,616,276	23,207,054	35,977,249	40,294,910	128,095,489
		%	30.70%	23.30%	27.10%	31.30%	28.20%
	6	Tonnes	20,007,083	20,751,051	21,936,569	20,972,086	83,666,788
		%	21.50%	20.80%	16.50%	16.30%	18.40%
	9	Tonnes	19,721,782	29,337,057	39,194,134	27,177,363	115,430,336
		%	21.20%	29.40%	29.50%	21.10%	25.40%
Secondary	3	Tonnes	0	0	0	0	0
		%	0.000096	0.00%	0.00%	0.00%	0.00%
	7	Tonnes	874,821	6,709,979	22,086,354	25,354,173	55,025,328
		%	0.939%	6.728%	16.612%	19.683%	12.103%
	8	Tonnes	613,662	2,818,571	3,330,370	2,265,999	9,028,604
		%	0.659%	2.826%	2.50%	1.76%	1.99%
	Total	Tonnes	93,124,844	99,736,010	132,951,159	128,811,636	454,623,648
		%	100.00%	100.00%	100.00%	100.00%	100.00%

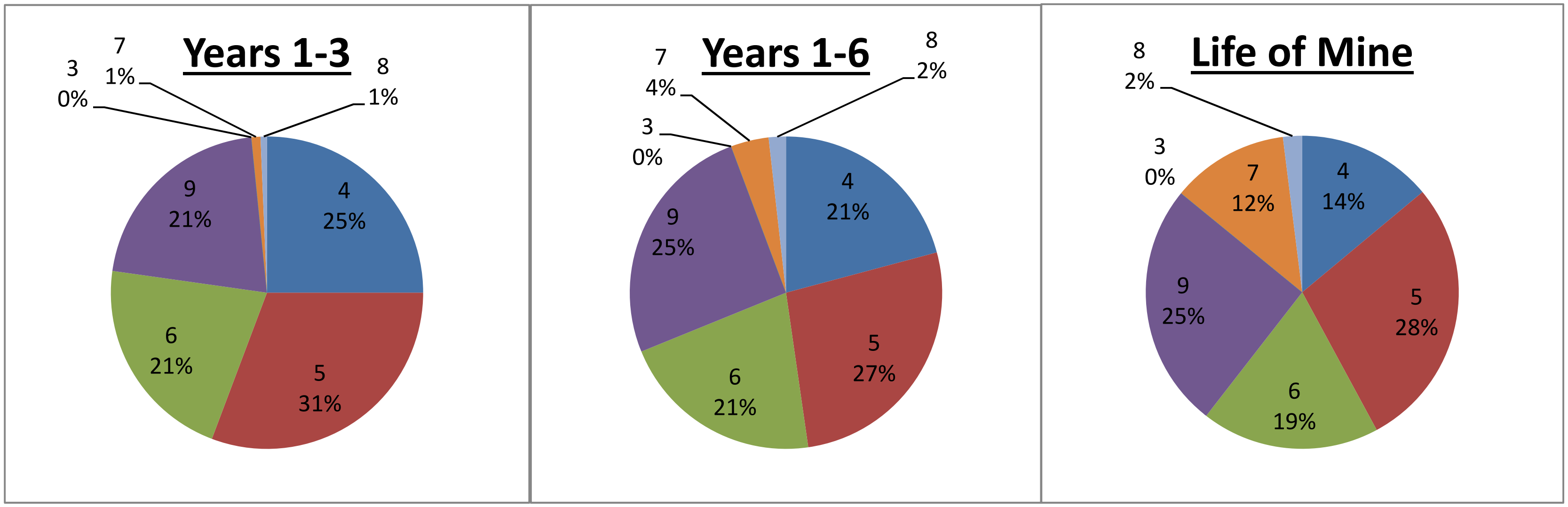


Figure 13.1 Progressive Mine Plan by Rock Type

Sample Selection Discussion

The metallurgical sample compositing was done by the proportions of the grade ranges in the pre-feasibility resource model above a cut-off grade of 0.2 g/t Au. The samples were selected from as diverse an area as possible and utilized 91 drill holes within the resource. The final FS pit utilized a higher cutoff grade than the resource and established the ore tonnage at 500 million tons. As a result of the reduction in tonnage from resource to reserve, 2 of the 91 holes utilized in the metallurgical sampling lie outside of the final FS pit boundary. Thus, 45 out of a total of 2,969 intervals (1.5%) used for the composites in the metallurgical test program lie outside the final FS pit boundary, which is considered insignificant for this feasibility level study.

The histograms of grades from the individual samples that went into the composite samples were compared with the histograms from the resource for the four main rock types. A substantial proportion of each composite came from the lower grade range of 0.2 to 0.3 g/t Au, which will not be part of the reserve in the FS pit. The lower grade material comprised 27% to 29% for RT4, RT5 and RT6 and about 12% for RT9 of the composites, representing a part of the 412 million tons of lower grade resource which is not in the FS reserve. As a result, there is some metallurgical skewing of the composite material. The exclusion of the lower grade material from the economic resource is not expected to negatively impact the recoveries obtained in the metallurgical test program. Considering the broad distribution of lower grade material in the Livengood resource optimization samples and in examining the more diverse variability samples in the metallurgical test program, there was not any consistent correlation between grade and recovery across all ore types.

Based on the grades of the samples utilized in the metallurgical test program versus the average grades in the mine plan, the samples are considered to be reasonably representative of the ore body.

In general the metallurgical test samples had higher grades than had been predicted using the assays from the drill core samples. Presumably this was due to better assays using the larger screen fire assay size that was determined to be necessary at the outset of the metallurgical program, which included the effect of coarser gold in the larger bulk samples. The resultant, somewhat higher grades of the test composites, brings their grades closer to, although lower than, the grades projected for the FS economic resource with its higher cutoff grade.

The impact of a higher grade resource on the derived reagent consumptions and tested energy requirements for milling the ore contained in the economic resource compared to the tested composites is expected to be within the stated limits of the accuracy of this feasibility study.

13.1.3 Optimization Testing

Optimization composites (major rock types) were prepared as;

- Optimization Composite 1 (RT5),
- Optimization Composite 2 (RT4),
- Optimization Composite 3 (RT6),
- Optimization Composite 4 (RT9),
- Mini optimization composite (RT7).

The gold head grades for these samples are summarized below.

Table 13.3 Optimization Sample Head Grade	
Rock Type	**Au (g/t)**
RT4	1.21
RT5	0.89
RT6	0.98
RT9	1.09
RT7	1.43

The optimization testwork examined various process test conditions to determine the ideal flow sheet. To improve assay repeatability, all samples had gravity recoverable gold removed by a combination of a centrifugal concentrator and a subsequent gravity table with the gravity tailings going to flotation or to CIL leaching. Grinding was investigated to determine the coarsest grind size that would sufficiently liberate gold to enable gravity concentration prior to subsequent treatment. It was determined that a primary grind size of P_{80} 180 µm would sufficiently liberate the gravity recoverable gold particles. During the testing, the gravity concentration was achieved utilizing a Knelson concentrator with the Knelson concentrate subsequently concentrated on a Mozley gravity table. The table tailings were combined with the Knelson tailings for subsequent processing by flotation followed by cyanidation or direct cyanidation of the gravity tailing.

Various grinds were tested to optimize the grind for gravity recovery from each ore type. These grind recovery data represented below in Figure 13.2 indicated that a primary grind of P_{80} 180 µm was suitable for all the ore types tested.

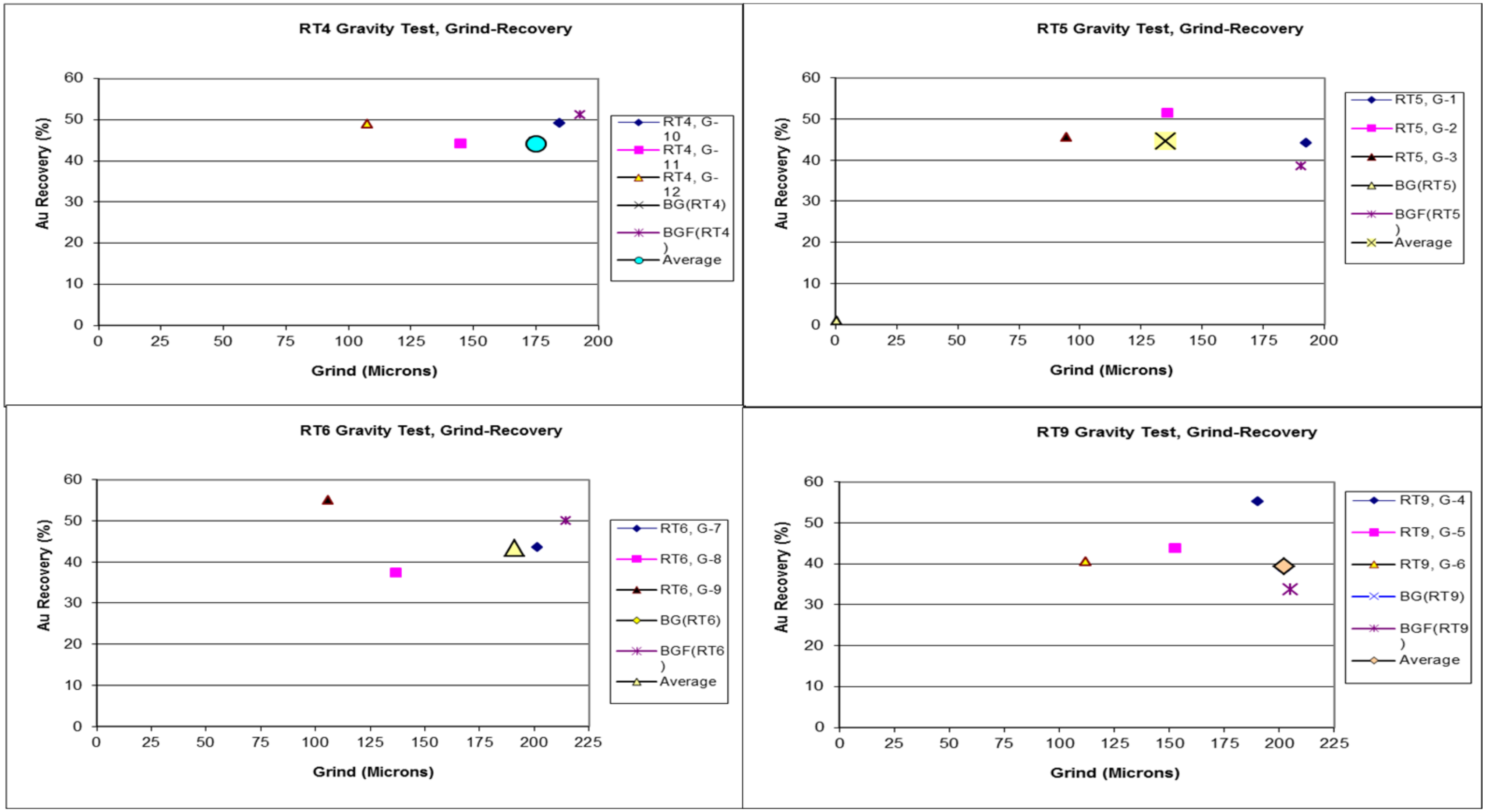


Figure 13.2 Au Gravity Concentration Grind-Recovery Relationships for RT4, RT5, RT6, and RT9

Table 13.4
Comparison of Gravity Test Results for Different Rock Types

Test	Rock Type	Optimization Composite	Product	Mass	Grade, g/t	Recovery, %	Gravity Tail K80
				%	Au	Au	µm
G 1 10 kg	RT5 Sunshine Upper Sediments	Opt Comp 1	Mozley Concentrate Final Tails	0.04 99.96	860 0.48	44.1 55.9	193
			Calculated Head Direct Head		0.86 0.89		
G 4 10 kg	RT9 Volcanics	Opt Comp 4	Mozley Concentrate Final Tails	0.04 99.96	1816 0.61	55.3 44.7	190
			Calculated Head Direct Head		1.36 1.09		
G 7 10 kg	RT6 Upper Sediments	Opt Comp 3	Mozley Concentrate Final Tails	0.06 99.94	710 0.52	43.5 56.5	202
			Calculated Head Direct Head		0.92 0.98		
G 10 10 kg	RT4 Cambrian	Opt Comp 2	Mozley Concentrate Final Tails	0.06 99.94	745 0.46	49.0 51.0	185
			Calculated Head **Direct Head**		**0.90** **1.21**		

Flotation testing examined the effect of grind, reagent dosage, and reagent selection. Optimization of the cyanidation of the flotation concentrate and of the gravity tailings required the examination of the effects of grind, cyanide concentration, and time. Rock type RT4 contained significant quantities of talc, which was difficult to separate and which would increase the bulk of potential flotation concentrate. Talc flotation cells were considered as a process option. The decision to go to direct cyanidation leaching of the gravity tails, based on test results on all four rock types, eliminated the need to test RT4 flotation concentrate in CIL.

Various grinds were tested to optimize the grind for rougher recovery from each ore type. These grind recovery data represented below in Figure 13.3 indicated that a grind of P_{80} 90 µm was suitable for all the ore types tested. Rock type RT4 did not respond well to flotation.

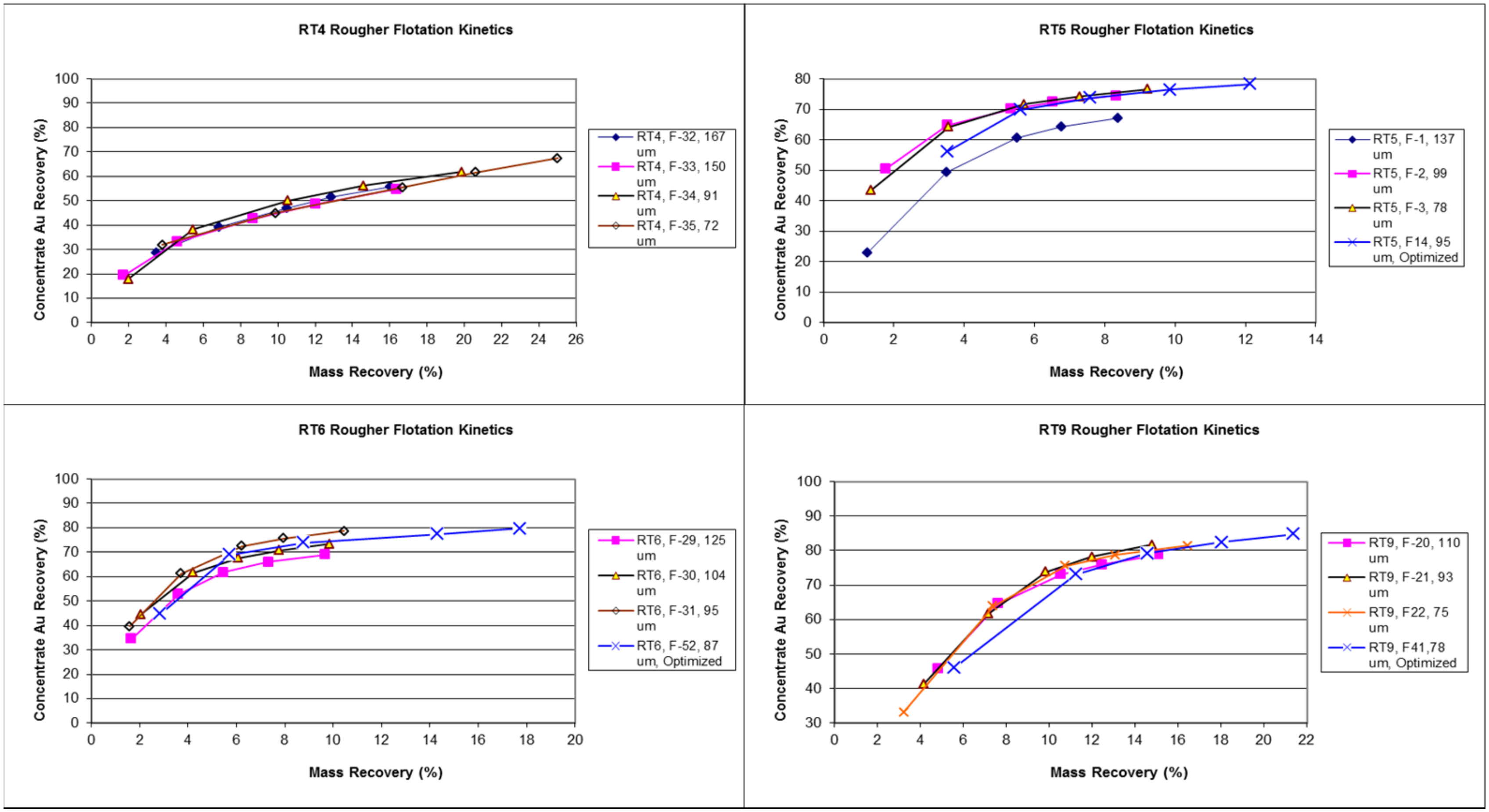


Figure 13.3 Effect of Primary Grind on Au Rougher Flotation Test Kinetics for RT4, RT5, RT6 and RT9

At 12% mass pull for the rougher flotation process for all rock types, the projected rougher recoveries were 78%, 74%, 75% and 60% for RT5, RT9, RT6 and RT4, respectively.

Flotation concentrate was then leached (CIL) to determine recoveries for RT4, RT6 and RT9.

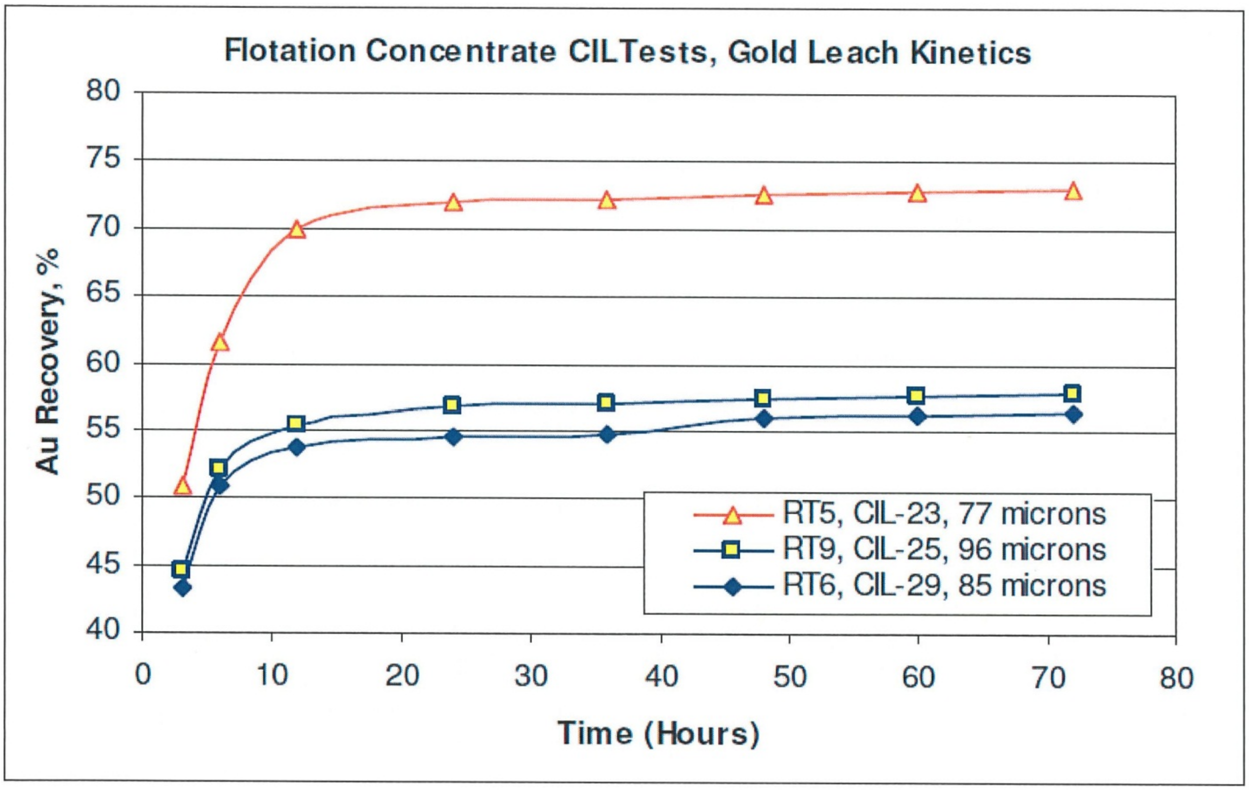


Figure 13.4 Flotation Concentrates CIL Test Kinetics for Different Rock Types

Based on the analysis of the metallurgical results it was determined that the recovery of gold was higher using CIL on the gravity tailings. Therefore, it was decided not to conduct any further flotation and CIL tests on flotation concentrate for RT4.

The results seen for each rock type in this case is summarized in Table 13.5 below.

Table 13.5
Gold Recovery Resulting from Combination of Flotation, Gravity and CIL

	Au Recovery (%)			
Rock Type	**Gravity**	**Flotation**	**CIL**	**Total**
RT4	49.0%	60.0%	-	-
RT5	44.1%	78.3%	73.0%	76.1%
RT6	43.5%	75.0%	56.3%	67.4%
RT9	55.3%	74.0%	57.8%	74.4%
Arithmetic AVG	48.0%	71.8%	62.4%	71.3%

Note: RT7 was not tested through flotation as it was tested after the determination to go to whole gravity tail leaching.

The "whole ore leach" option was also investigated in which the Livengood process would consist of gravity and CIL leach of the gravity tails. Various grinds were tested to optimize the grind for the CIL leach recovery from each ore type. These grind recovery data represented below in Figure 13.5 indicated that a grind of P80 90-100 µm was suitable for CIL leaching of all the ore types.

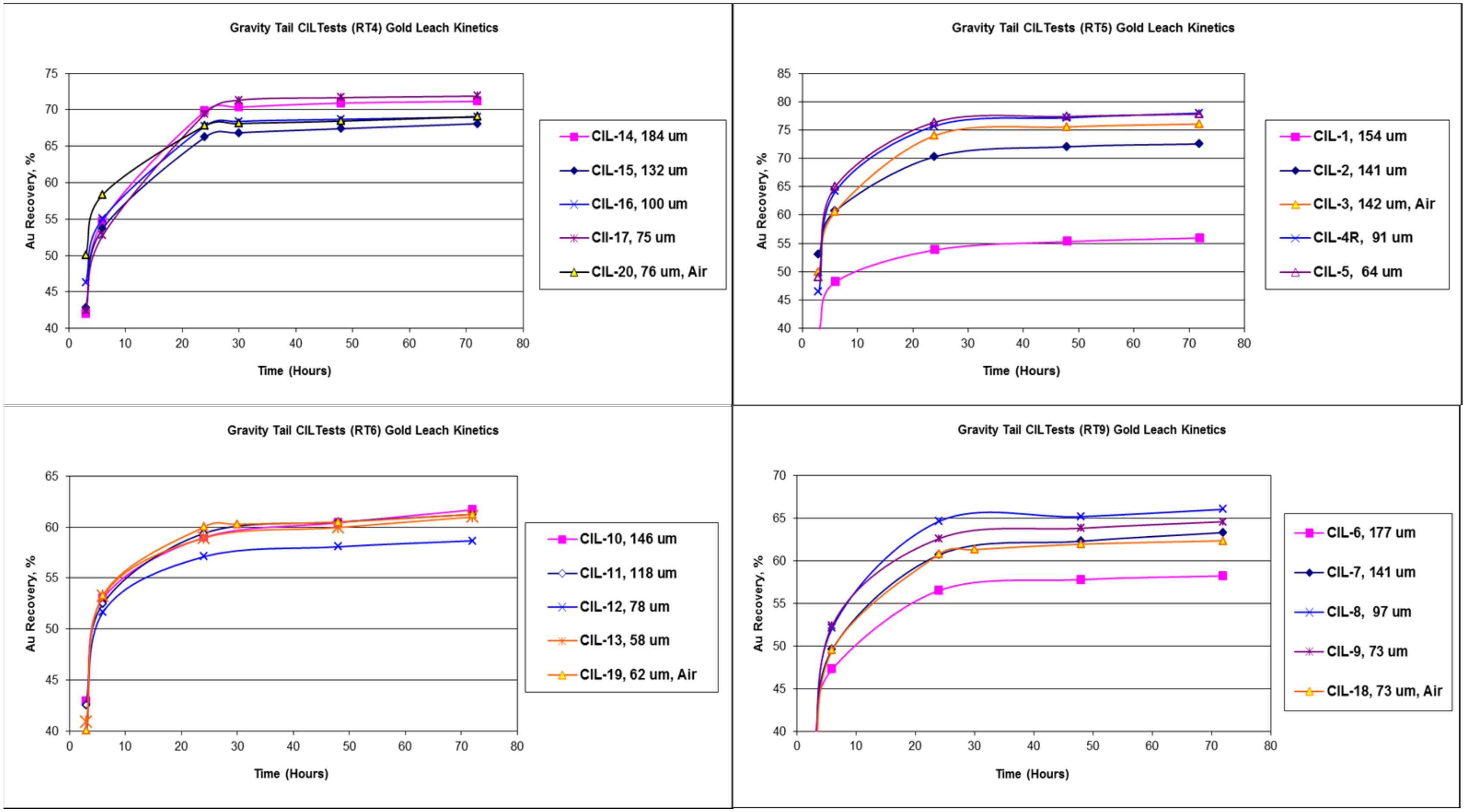


Figure 13.5 Effect of Grind on Gold Extraction Kinetics for RT4, RT5, RT6, and RT9

CIL tests using Vancouver and mine water and air sparging were conducted using Mozley gravity tailings to compare the extractable gold at similar reagent conditions. Kinetics for the tests using mine water are compared in Figure 13.6.

The gold leach kinetics, evaluated within the range of 24-36 hours, reached a plateau at 68-72% Au recovery for different rock types between 24-36 hours. The gold dissolution for the tests was achieved in the grind range of P_{80} of 74-109 µm.

Gold recoveries were almost identical for RT9 and RT4 using Vancouver and mine waters. RT5 achieved higher gold recovery using Vancouver water, while RT6 achieved higher gold recovery using mine water.

The cyanide consumption at 36 hours was low at 2.0 lbs/ton (0.9 kg/mt), which is lower than the cyanide consumption at 72 hours using Vancouver water. The lime consumptions were in the range of 4.4-6.6 lbs/ton (2-3 kg/mt) and considerably high at 9.2 lbs/ton (4.2 kg/mt) for RT4. Lime consumption was higher using mine water as compared to Vancouver water.

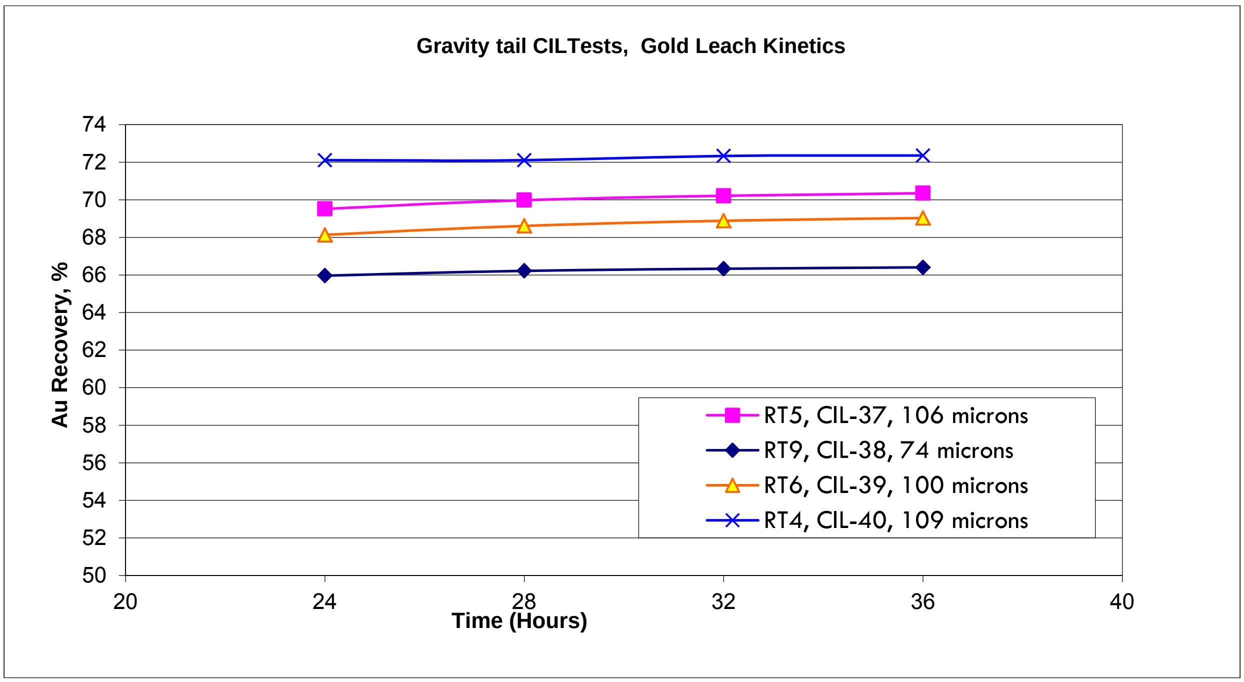


Figure 13.6 Mozley Gravity Tailings CIL Test Kinetics for Different Rock Types (Mine Water)

The above graph demonstrates the very flat leach recovery curves for the gravity tail leach indicating little if any increased extraction over 32 hours of leach time.

The evaluation completed with the optimization samples demonstrating that the preferred flow sheet is gravity followed by CIL of the gravity tailings. The gravity plus CIL leaching of the gravity tails offers 9-12% improved gold recovery for all rock types compared to gravity plus flotation plus CIL leaching of flotation concentrate. The overall results of whole ore leaching can be seen below in Table 13.6.

Table 13.6 Gold Recovery Resulting from Whole Ore Leaching			
	Au Recovery (%)		
Rock Type	**Gravity**	**CIL**	**Total**
RT4	49.0%	69.0%	84.2%
RT5	44.1%	78.0%	87.7%
RT6	43.5%	58.7%	76.7%
RT9	55.3%	66.0%	84.8%
Arth. AVG (RT4 to RT9 only)	48.0%	67.9%	83.3%
RT7 (bleached)*	24.3%	44.8%	58.2%

* Note: RT7 (bleached) was tested in a mini-program after the other rock types.

During the CIL testwork the data demonstrated that the cyanide consumption is not overly sensitive to grind. On a weighted average by rock type over the mine life, the ore required 5.75 lbs/t (2.88kg/mt) of lime and 1.74 lbs/t (0.87kg/mt) of sodium cyanide in the gold leach.

The overall gold recoveries achieved by both process options are summarized in Table 13.7 below.

Table 13.7 Overall Gold Recovery of Optimization Samples for Both Process Options		
Rock Type	**Gravity + CIL**	**Gravity + Flotation + CIL**
RT4	84.2%	-
RT5	87.7%	76.1%
RT6	76.7%	67.4%
RT9	84.8%	74.4%
RT7	58.2%	

Important conclusions from the optimization testing include:

- All rock types responded well to gravity separation, with 44 to 55% of the gold recoverable in the gravity circuit. At the grind of approximately P80 180 µm, these gravity recoveries were achieved at a 1% mass pull.
- Rougher flotation works reasonably well for RT5, RT6 and RT9 although the mass recovery was variable. Rougher flotation does not work well for RT4 due to the presence of talc.
- Rock type RT4 is quite different from the other rock types. It is softer, contains significantly more talc than the other samples, more total carbon, more Total Organic Carbon (TOC), and more carbonate.
- Overall gold extraction was increased 9-12% by the leaching of gravity tailings as compared to the leaching of flotation concentrate.

13.1.4 Variability Testing

Upon completion of the optimization testing, the test program moved to the variability phase. The goal was to determine the variation that exists in the ore and to test the geological extremes of each rock type. In addition to the samples tested in the optimization phase, RT7 as well as rock type Stibnite was included in the variability phase of testing. Rock type RT7 was not tested in the initial optimization testing as it does not have a large presence in the early period of the mine life and only represents 12.1% (by tons) of the LOM reserve. Rock type RT7 was split into two types, bleached and unbleached, as these types exhibited different metallurgical responses. Rock type Stibnite represents a very small fraction of the mine ore, but has very high head grade.

For variability testing, the most favorable process conditions determined in the optimization phase were used. The variability test results showed an overall lower average gold recovery than what was experienced in the optimization phase, reflecting the extremes of the deposit rather than the more representative optimization samples. The average overall gold recovery resulting from multiple tests for each rock type is summarized below.

Table 13.8
Variability Sample Gold Recovery

Var. Sample	Rock Type	Gravity Rec.%	CIL Rec.%	Overall Rec.,%
1	5	36.8	90.7	94.1
2	5	39.7	79.3	87.5
3	5	27.8	90.5	93.1
4	5	39.9	96.2	97.7
5	5	38.3	54.8	72.1
6	5	58.4	83.4	93.1
7	5	30.3	57.1	70.1
8	5	49.3	40.9	70.0
9	5	53.7	75.0	88.4
10	5	19.4	89.8	91.8
11	5	62.7	92.4	97.2
12	5	44.8	83.8	91.1
Average		41.8	77.8	87.2
Minimum		19.4	40.9	70.0
Maximum		62.7	96.2	97.7
76	9	17.2	40.4	50.7
77	9	20.0	53.6	62.9
78	9	11.9	50.1	56.0
79	9	24.8	37.7	53.2
81	9	56.3	32.0	70.3
82	9	36.5	73.7	83.3
83	9	21.3	74.9	80.2
84	9	26.2	39.2	55.1
85	9	8.40	17.8	24.7
86	9	34.2	50.4	67.4
87	9	41.4	70.2	82.5
88	9	53.8	42.4	73.4
89	9	40.5	58.7	75.4
Average		30.2	49.3	64.2
Minimum		8.40	17.8	24.7
Maximum		56.3	74.9	83.3
31	6	35.8	76.8	85.1
32	6	29.4	26.5	48.1
33	6	38.1	76.7	85.6
34	6	41.6	87.7	92.8
35	6	44.5	94.1	96.7
36	6	51.8	62.2	81.8
37	6	21.9	63.7	71.6
Average		37.6	69.7	80.3
Minimum		21.9	26.5	48.1
Maximum		51.8	94.1	96.7
Var. Sample	Rock Type	Gravity Rec.%	CIL Rec.%	Overall Rec.
16	4	71.0	77.1	93.4
17	4	71.3	95.4	98.7
18	4	59.1	60.5	83.8
19	4	20.5	69.4	75.7
20	4	17.1	32.4	44.0
21	4	9.90	44.9	50.4
22	4	39.6	58.3	74.8
23	4	28.7	42.3	58.9
Average		39.7	60.0	72.4
Minimum		9.90	32.4	44.0
Maximum		71.3	95.4	98.7

Var. Sample	Rock Type	Gravity Rec.%	CIL Rec.%	Overall Rec.,%
46	7 bleached	19.0	26.0	40.1
47	7 bleached	45.7	23.1	58.2
48	7 bleached	22.7	41.4	54.7
49	7 bleached	35.1	10.3	41.8
50	7 bleached	13.9	25.5	35.9
51	7 bleached	26.0	16.9	38.5
52	7 bleached	21.8	13.4	32.3
53	7 bleached	46.6	69.0	83.4
54	7 bleached	59.5	79.7	91.8
55	7 bleached	14.7	13.8	26.5
Average		30.5	31.9	50.3
Minimum		13.9	10.3	26.5
Maximum		59.5	79.7	91.8
61	7 unbleached	18.6	28.3	41.6
62	7 unbleached	26.9	39.9	56.1
63	7 unbleached	33.5	12.7	41.9
64	7 unbleached	6.60	3.90	10.2
65	7 unbleached	50.7	35.2	68.1
66	7 unbleached	19.0	12.1	28.8
67	7 unbleached	15.3	12.5	25.9
68	7 unbleached	14.7	3.80	17.9
69	7 unbleached	21.9	11.2	30.6
70	7 unbleached	63.0	60.3	85.3
Average		27.0	22.0	40.7
Minimum		6.60	3.80	10.2
Maximum		63.0	60.3	85.3
90	stibnite	2.00	7.90	9.74
91	stibnite	1.90	0.20	2.10
92	stibnite	1.90	0.60	2.49
93	stibnite	0.80	24.2	24.8
94	stibnite	3.80	70.7	71.8
95	stibnite	2.50	2.00	4.45
96	stibnite	2.00	0.30	2.29
97	stibnite	1.30	0.40	1.69
98	stibnite	1.00	2.50	3.48
Average		1.91	12.1	13.7
Minimum		0.80	0.20	1.69
Maximum		3.80	70.7	71.8

13.1.5 Comminution Testing

Comminution testing was performed on samples that comprised part of the optimization samples, as well as the variability samples. Comminution testing included determination of Bond Work Index (BWi), Rod Work Index (RWi), Crusher Work Index (CWi) and Abrasion Index (Ai). These indexes are utilized in crusher and mill calculations including sizing and consumables such as balls and liners. The overall average values for each of these rock types are displayed in Table 13.9.

Table 13.9
Average Comminution Data for each Rock Type

	Work Index (kWh/mt) – Metric			
RockType	**BWi**	**RWi**	**CWi**	**Ai**
RT4	12.30	13.10	14.55	0.14
RT5	11.87	15.73	14.14	0.15
Rt6	14.36	17.33	14.34	0.12
RT7	14.07		7.69	0.17
RT9	14.20	16.25	7.35	0.35

JK Drop-Weight tests were conducted on a selected rock type samples. The data obtained was analyzed to determine the JKSimmet comminution parameters. These parameters are combined with equipment details and operating conditions to analyze and/or predict grinding circuit performance. The A and b values are not independent and cannot be used directly for comparisons between ore types. However, Axb provides a good parameter for comparison. Lower Axb values indicate a higher resistance to abrasion breakage and also a greater resistance to impact breakage. The table below shows the average A and b values for each rock type, indicating that RT4 and RT7 would require less comminution energy than the other rock types. The numbers are reflective of a medium hard rock type.

Table 13.10
JK Drop-Weight Parameters

Rock Type	A	b	Axb
RT4	62.10	0.83	51.54
RT5	67.60	.50	33.80
RT6	50.70	0.64	32.45
RT7	55.35	0.89	49.26
RT9	60.50	0.58	35.39

Analysis of the JK Drop weight parameters was performed using JKSimmet to analyze the grinding circuit, which was a 40' by 25' SAG mill followed by two 28' by 45' ball mills with a pebble crusher in closed circuit with the SAG mill. The JKSimmet results concluded, after optimization, that the selected circuit would produce about 84,000 dmt/d.

It should be noted that one vendor recommended the use of a 42' SAG mill in order to achieve the target throughput. Consideration was given to this size of mill, but it was decided that a "first of its kind" (42' SAG) was not warranted due to the lack of operating experience in the industry at this time.

Following further consultation, FLSmidth ran the model again using the new parameters:

- Circuit P_{80} target grind of 90 µm
- Daily dry metric tonnage of 90,718 @ 92% availability (100,000 t/d)
- Bond ball mill work index of 14.3 kWh/t (corresponding to the 75th percentile of LOM hardness).

The simulation resulted in a 15% circulating load through the pebble crusher and the ball mill circuit running at 350% circulating load producing a 90 µm product. The circuits use a single 40' by 26' SAG mill with 27 MW installed power and two 28' by 46' ball mills with 29.5 MW installed power each.

The decision was made to go with the FLS recommendation, but to install a bypass after the pebble crusher to allow the option to shift some of the SAG load downstream to the ball mill circuit to balance the power draw in the circuits.

13.1.6 Solid Liquid Separation Testwork

Livengood ore samples were submitted to Pocock Industrial, Inc. for solid liquid separation (SLS) testing. The current flowsheet contains a total of 4 thickeners; 2 pre-leach thickeners, and 2 tailings thickeners. Pre-leached and leached samples from the optimized testwork were submitted to Pocock for each of the primary rock types (RT4, RT5, RT6, RT7 Bleached and RT9).

The Livengood design criteria use a high rate thickening rise rate of 1.64 gpm/ft^2 (4.0 m^3/m^{2}h) for both the pre-leach thickener and the tailings thickener.

13.1.7 Cyanide Detoxification Tests

Following the optimization and variability phases of the SGS test program, the cyanide detoxification testing was initiated. The testing used the SO2/Air INCO process to remove cyanide and base metal complexes from the CIL tailings for each rock type. The objective of this phase of testing was to optimize cyanide detoxification (CND) of CIL tailings. The "Interim Test Program" used a 10 kg sample of each rock type (RT4, RT5, RT6, RT7 and RT9).

The average feed pulp density to cyanide detoxification was between 31-39 percent pulp density. The results showed that it was possible to treat the CIL tailings using the INCO process to have both weak acid dissociable cyanide (CN_{WAD}) and total cyanide (CNT) levels below 1 mg/L. The test conditions indicate that a pH of 8.5-8.6 is ideal with a retention time of 94-147 minutes. The reagent consumptions from the Phase 1 testing are 8.2-14.7 g/g CN_{WAD} of equivalent SO2, 4.9-8.9 g/g CN_{WAD} of lime, and 0.27-0.57 g/g CN_{WAD} of Cu.

The design application rates are expected to be:

- Lime = 0.82 lb/t
- Copper Sulfate = 0.08 lb/t
- Sodium Metabisulfite = 1.65 lb/t.

13.1.8 Gold Deportment Studies

SGS undertook a high definition mineralogical examination of Livengood samples utilized for the metallurgical testwork. Examination of four samples identified as RT5, RT4, RT6, and RT9 was carried out with X-ray diffraction (XRD), QEMSCAN, Electron Microprobe Analysis (EMPA), optical microscopy, and chemical analysis. The purpose of this test program was to determine the overall mineral assemblage, the liberation/association of the Fe Sulfides and Au-bearing minerals and mass balance of microscopic gold.

The RT5, RT6 and RT9 samples consist of quartz (33.0-40.2%), micas (11.8-16.9%) feldspars (21.7%-27.7%), carbonates (3.7-7.2%), and oxides (1.5-2.1%), along with trace (<1%) talc, apatite and other minerals. Pyrite accounts for 2.9-10.5%, arsenopyrite (1.0-1.4%). Gold minerals are tentatively quantified in the samples at less than 0.001%.

The RT4 sample consists of carbonates (22.5%), talc (18.6%), quartz (16%), feldspars (13.1%), chlorite (11.0%), micas (6.4%), and other silicates (mainly amphibole, pyroxene, garnet and epidote) (4.7%), clays (2.5%), oxides (1.8%), along with trace (<1%) apatite and other minerals. Arsenopyrite accounts for (1.9%), and pyrite for 0.9%. Gold minerals are tentatively quantified in the sample at less than 0.001%.

In the four samples, Au occurs mainly as native gold (defined as Au 75-100%), and carries an average of 90.8-93.5 wt% Au, all other elements are less than 1.0 wt%.

The results of gold deportment characterization demonstrated that RT5, RT6 and RT9 all exhibited broadly similar characteristics. Rock types RT6 and RT9 demonstrated poor correlation with chemical assays, suggesting that the contribution of finer gold populations may be more significant in these ore domains. Rock type RT4 showed significant variation in both mineralogical composition and identified gold populations. It would be anticipated that RT4 may cause difficulties in recovery for a process tailored to the other ore domains.

Rock types RT5 and RT6 have pyrite as the dominant sulfide mineral over arsenopyrite. Rock type RT9 maintains this trend but with <10% arsenopyrite present. Generally, solid solution gold could be expected to be hosted with arsenopyrite and consequently the potential contribution of solid solution gold to the overall gold balance should not be expected to be significant in these rock types.

Rock type RT4 does show arsenopyrite to be the dominant sulfide mineral. However, the abundance of sulfide minerals was lower in this rock type, once again suggesting that solid solution gold should not be a major factor in process development.

Comparison of the four rock types examined for the Project demonstrated a consistent trend for the major population of gold to be present as free gold within the gravity concentration size range. The majority of gold grains that were not within the gravity recoverable range were identified as fine exposed gold grains and should be readily amenable to recovery by CIL leaching of a gravity tails.

13.2 Trade-off Studies

Samuel Engineering conducted two formal trade-off studies during the Feasibility Study for the Livengood Project. The first study was a trade-off between a gravity/flotation/CIL of flotation concentrate vs. a gravity/whole gravity tail CIL leach circuit. Based on preliminary testwork, the trade-off study used an 8% increase in gold recovery for the whole ore leach. The whole ore leach circuit would eliminate the flotation circuit; however now all of the gravity tailings would go to cyanidation rather than only the flotation concentrates. The CIL tank volume would increase substantially along with the reagents required for cyanidation but eliminate the need for flotation reagents. Additionally, a greater amount of slurry would need to go through thickening and detoxification prior to going to the tailings management facility.

Balancing this cost was the reduction of capital cost required for the regrind circuit and flotation cells and the energy to operate them.

Final results of the testwork and the trade-off study supported the decision to select gravity followed by CIL of the gravity tailings as the design process.

WHOLE ORE LEACH

13.2.1 Equipment Cost Estimate

An equipment cost estimate was prepared to evaluate the alternatives of gravity/flotation/CIL of flotation concentrate and gravity/whole gravity tail CIL leach. The estimate showed a total increase of $1,917,000 in equipment cost by switching to the whole ore leaching option. The main difference in costs arise from 1) the elimination of the flotation circuit and all equipment associated with it and 2) increasing the size of the CIL, thickening and CN detoxification circuits. All equipment costs for the flotation alternative were determined using data prepared by FLS.

13.2.2 Operating Cost Estimate

Operating cost on a per day basis was prepared for both alternatives. The whole ore leach option shows a slight increase in operating cost over the flotation option. The trade-off study estimated the whole ore leaching option would cost $143,000/d more primarily due to the increased cost for sodium cyanide.

Reagent consumption estimates for both the whole ore and flotation options were based on SGS data wherever possible. Flotation reagent consumption was based on the consumption rates seen in optimized sample for RT5. Flocculant consumption was determined using Pocock data and is assumed to be the same for both whole ore and the flotation option. A scale factor of 65% was used to determine plant reagent consumption rate for lime and sodium cyanide relative to the laboratory determined consumption rates.

Reagents utilized in the CN detoxification circuit as well as in the ADR circuit required some assumptions. In general, reagent consumption increased in the whole ore option according to how it was used. Reagent consumption was determined by their function. The increase of reagents can be calculated based on the additional gold that would be recovered. The determination of lime, caustic soda, sulfur dioxide and copper sulfate consumption was given a larger allowance that related to maintaining specific conditions within the whole slurry mass (i.e. pH modifiers). Though the carbon inventory will increase in the whole ore option, the consumption of carbon is not anticipated to increase.

13.2.3 Gold Recovery Comparison

By eliminating the flotation circuit, it is expected that the daily gold recovery will increase. This is due to the increased recovery of gold that would otherwise be lost in the flotation tailings. For the purpose of the trade-off study, SE estimated that gold recovery would increase by a total of 8%. This is based on the gold recovery data received from SGS at the time of the study. Therefore, the tradeoff study determined that whole ore leaching would increase the total gold revenue by $191,000/d using a price of $1250/oz.

INTENSE LEACH vs. DIRECT SMELTING

Two options were considered for the treatment of the gravity concentrate:

- Intensive cyanidation
- Direct smelting

The direct smelting option necessitates the inclusion of a secondary gravity circuit to reduce the volume to be treated. The secondary gravity tail will join the CIL leach of the primary gravity tail and will contain only the finest gold particles recovered in the primary gravity circuit, which will be amenable to leach in the CIL circuit.

The intensive cyanidation option would treat the total gravity concentrate in pressurized leach reactors and would require subsequent liquid/solid separation of the leached residue from the pregnant solution and recovery of gold from solution in electrowinning cells. The sludge from the cells would require smelting to produce doré.

13.3 Trade-off Results

The equipment and operating costs were not found to differ largely from one option to the other. The Direct Smelting option reduces the overall equipment cost by an estimated $2,487,000. This is due to the elimination of the intense leach reactors. The change in operating cost is negligible because the decrease due to lower reagent demands is offset by the increase in labor costs. The benefits of this trade off lay mainly in the minimization of handling the gold concentration. In the Intensive Leach option, a gold concentrate is produced through the primary gravity and intense leach circuits. There will undoubtedly be some gold losses associated with the intense leach reactors. Sending the gold concentrate through a secondary gravity circuit and then directly to smelting reduces the risk of gold loss.

The downside of the direct smelt option involves the increased demand for personnel as well as security. The secondary gravity circuit would require two operators 24h/d. There are also increased security demands that would exist.

SE recommended the adoption of the direct smelting of the gravity concentrate. While difficult to conclusively prove until there is substantial production, SE believes that removing the coarsest gold from the mill circuit quickly once in hand is technically and physically a preferable option. Intensive leaching will not leach 100% of the contained gold allowing some potentially unrecoverable losses to process tailings. Direct smelting will, properly executed, have a superior gold extraction to metal. Any slag loses can be recovered by including the slag with the primary ball mill feed where it will be ground and any free gold entrainment will be recovered in the gravity circuit.

13.4 Flowsheet Development

Livengood ore has demonstrated it is amenable to gravity concentration, as a substantial proportion of the gold is free and liberated at a reasonably coarse grind. Gold deportment studies indicated that a substantial amount of the finer gold had least a 25% or greater exposure allowing it to be recovered by cyanidation. Some of the gold with exposure however was not contained in sulfide aggregates and was therefore less amenable to sulfide flotation. A considerable amount of testing of flotation with cyanidation of the flotation concentrate compared to direct cyanidation verified the mineralogical observations. Direct cyanidation of the gravity tails provided 9-12% better gold extraction than from the cyanidation of the flotation concentrate. The difference was the loss of gold to the flotation tailing. There was insufficient value in this flotation tailing however to separately cyanide the tailing in parallel with the flotation concentrate.

The fact that Livengood ores contain coarser gold particles makes analytical measurement of samples more difficult. Ultimately on the basis of mineralogical observation and of practical assaying knowledge, larger sample sizes were chosen (1 kg) and the coarser gold particles screened out and weight averaged back into the undersize assays to smooth the effect of the erratic gold dispersion in the low grade deposit. The effect of these erratic assays made initial metallurgical results difficult to interpret in part because the mass balances were often further apart than the effect of the test changes. Under these circumstances it was difficult to determine if test changes were making improvements to the process. This difficulty was noticeable in the RDI testing. The program at SGS in Vancouver made the initial choice to go with screen fire assays allowing better gold averages for samples and improving gold mass balances.

On the basis of the substantial testwork conducted on the major rock types, the results warranted the selection of directly leaching the gravity tails vs. the leaching of the flotation concentrate.

The incorporation of activated carbon in the cyanide leach was utilized to obviate the gold robbing presence of some organics in the ore at Livengood. The activated carbon removes solubilized gold prior to the ability of the naturally occurring organics to rob it from the leach solutions. The daily tonnage proposed for milling at Livengood is large and the resulting amount of carbon in the leach circuit will also be large. The mineralogical studies indicating that silver is only a minor contributor to the precious metals at Livengood further justified the choice of carbon. Livengood ore contains some soluble copper minerals. The copper that does solubilize will load onto carbon in the CIL leach and as a result will increase the required amount and advance frequency of carbon. The copper is removed from the carbon in the desorption process by using a cold strip prior to stripping the gold from the carbon. The copper removed is further utilized to reduce the copper requirements for the cyanide destruction process prior to the final tailings reporting to the tailings management facility.

Based on the metallurgical testwork results from SGS and Pocock, SE developed a Process Design Criteria and Process Flow Diagrams as described in Section 17.

14.0 Mineral Resource Estimates

14.1 Global Mineral Resource Estimate

The global mineral resource estimate has been updated from that published in August, 2011 to include drilling in the deposit since that time. The resource model was constructed using Gemcom GEMS® and the Stanford Geostatistical Software Library (GSLIB) Multiple Indicator Kriging (MIK) post processing routine. The resource was estimated using MIK techniques.

A three-dimensionally defined stratigraphic model, based on interpretations by THM geologists, was used to code the rock type block model. A three-dimensionally defined probability grade shell (0.1 g/mt) was used to constrain the gold estimation. Gold contained within each block was estimated using nine indicator thresholds. The block model was tagged with the geologic model using a block majority coding method. Because there are significant grade discontinuities at stratigraphic contacts, hard boundaries were used between each of the stratigraphic units so that data for each stratigraphic unit was used only for that unit.

A summary of the estimated global (in-situ) mineral resource is presented below in Table 14.1 for cutoff grades of 0.2, 0.3, 0.5, and 0.7 g/mt Au.

Model validation checks include global bias check, visual validation, and swath plots. In all cases, the model appears to be unbiased and fairly represent the drilling data.

Table 14.1
Global Resource Estimation Summary – July 2013

Classification	Gold Cutoff (g/mt)	Tonnes (millions)	Gold (g/mt)	Million oz Gold
Measured	0.20	994	0.52	16.4
Indicated	0.20	112	0.45	1.6
Total M&I	0.20	1106	0.51	18.0
Inferred	0.20	438	0.41	5.8
Measured	0.30	731	0.61	14.4
Indicated	0.30	71	0.56	1.3
Total M&I	**0.30**	**802**	**0.61**	**15.7**
Inferred	0.30	266	0.52	4.4
Measured	0.50	370	0.82	9.8
Indicated	0.50	31	0.80	0.8
Total M&I	0.50	401	0.82	10.6
Inferred	0.50	92	0.76	2.3
Measured	0.70	179	1.08	6.2
Indicated	0.70	13	1.09	0.5
Total M&I	0.70	192	1.08	6.7
Inferred	0.70	34	1.08	1.2

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to measured and indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

"In this Instrument, the terms 'mineral resource,' 'inferred mineral resource,' 'indicated mineral resource,' and 'measured mineral resource' have the meanings ascribed to those terms by the Canadian Institute of Mining,

Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when 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. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions."

Since the August 2011 global resource estimate (Brechtel et al., 2011), the total measured and indicated (M &I) oz Au estimated has increased 2% for cutoff grades of 0.30 g/mt Au.

14.2 Mineral Resource Defined by Surface Mine Optimization

In 2011, an economic surface mine was generated using Whittle mine optimization software to define the Mineral Resources using an assumed long-term gold price of $1,400/oz (Brechtel, et al., 2011).

Based on the 2011 study and by analogy to the nearby operating Fort Knox Mine delineated mineralization of the Livengood Deposit is classified as a resource according to the definitions from National Instrument 43-101 and from CIM (2010).

14.3 Data Used

The THM data available for the FS model comprised 714,026 ft (217,635 m) of core and RC drilling, plus trench data. Historical drilling and sampling is shown in Table 14.2. Drilling performed by THM is shown in Table 14.3. The historical data represent about 2% of the total information used. The use of historic data is based on its statistical consistency with current data and the small portion of the total data represented as shown in past technical reports (Klipfel and Giroux, 2008a and b, 2009; Klipfel et al., 2009a and 2009b). For data validation purposes, in 2011 Reserva International checked the assay data for a sample of drill holes (10%) used for the resource estimate in GEMS against the original assay certificates (Secure PDF). The error rate of less than 1% is well within acceptable standards. These minor errors arose exclusively from mismatches with samples re-assayed for QA/QC purposes, and were corrected by revising the GEMS database update procedure.

The topographic surface used is based on a 4 m Digital Elevation Model derived from 2008 aerial photography.

Densities used in the resource are based on 98 determinations from core and RC chip samples and are shown in Table 14.4.

Table 14.2 Historical Drilling and Sampling				
Year	**Company**	**Drill Type**	**Number of Holes**	**Feet (Meters)**
1976	Homestake	Percussion	6	994 (303)
1981	Occidental	Percussion	7	1,017 (310)
1989	AMAX	Trench	2	525 (160)
1990	AMAX	RC	3	1,050 (320)
1997	Placer Dome	Core	9	3,612 (1,101)
2003	AngloGold	RC	8	4,968 (1,514)
2004	AngloGold	Trench	8	906 (276)
2004	AngloGold	Core	4	2,500 (762)
Total			**47**	**15,571 (4,746)**

Table 14.3 THM Resource Drilling and Sampling			
Year	**Drill Type**	**Number of Holes**	**Feet (Meters)**
2006	Core	7	4,027 (1,227)
2007	Core	15	14,471 (4,411)
2008	Core	9	7,175 (2,187)
2008	Trench	4	262 (80)
2008	RC	109	93,402 (28,469)
2009	Core	12	15,004 (4,573)
2009	RC	195	196,243 (59,815)
2010	Core	40	44,723 (13,631)
2010	RC	181	185,988 (56,689)
2011	RC	111	94,478 (28,796)
2011	Core	67	58,257 (17,757)
Total		**750**	**714,026 (217,635)**

Table 14.4 Density Determinations					
Lithology Unit	**N**	**Mean**	**Std.Dev.**	**Max**	**Min**
Amy Sequence	4	2.67	0.04	2.72	2.65
Cambrian	12	2.82	0.07	2.95	2.69
Combined Cambrian-Amy		2.78			
Kint	3	2.56	0.18	2.76	2.44
Lower Sediments	21	2.74	0.05	2.84	2.62
Main Volcanics	36	2.72	0.13	2.86	2.11
Upper Sediments	22	2.68	0.13	2.79	2.23
Total N/ Average	98	2.72			

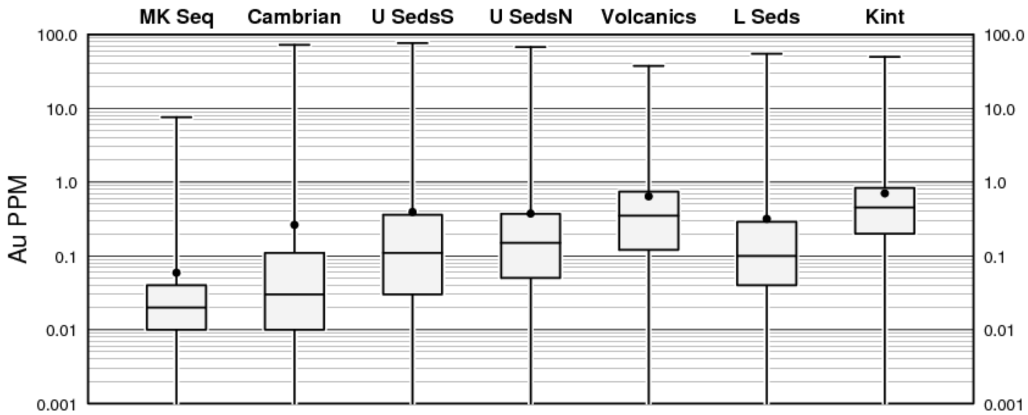
14.4 Data Analysis

Multi-element assay information is available for nearly 50% of the samples. A statistical summary of this data from a previous report (Klipfel, et al., 2009a) is shown in Table 14.5. The only element of economic significance is gold, which was the only element modeled in the resource model. No significant correlations were found between the various elements. There were numerous weak-to-moderate correlations, but nothing that could be exploited to improve the gold estimate. Based on the lack of significant correlations previously determined, the exercise was not updated for this estimate.

Table 14.5
Statistical Summary of Assay Data

Element	Units	N	Mean	Maximum	Std.Dev.	C.V.
Au	ppm	34786	0.40	56.2	1.22	3.0
Ag	ppm	12969	0.41	440	4.07	10.0
Cu	ppm	12969	42	1120	34	0.8
Pb	ppm	12969	19	9240	128	6.7
As	ppm	12971	2169	137000	4181	1.9
Sb	ppm	12969	221	138000	2394	10.8
Zn	ppm	12969	186	3440	221	1.2
Fe	%	12708	4.3	21.3	1.4	0.3
Mo	ppm	12969	5.5	74.0	6.9	1.3
S	%	12081	1.4	18.4	1.4	1.0
Te	ppm	12063	0.16	25.1	0.5	3.0

Each of the assay intervals were also logged for lithology, stratigraphy, alteration and mineralization. Of all of the available qualitative data, the stratigraphic unit appears to exert the most influence on the gold mineralization (Figure 14.1). It is still a matter of geological debate as to exactly why this is so, but the volcanic unit is preferentially mineralized relative to the units above and below it. Also, the Kint dikes, which appear to be the conduits for much of the mineralization, are also well mineralized. Not only are the volcanics and Kint dikes higher grade, they are uniformly well mineralized as shown by the relatively low coefficient of variation (C.V.) of each unit.



	MK Seq	Cambrian	U SedsS	U SedsN	Volcanics	L Seds	Kint	
Number of data	6860	20000	20000	20000	16604	20000	8019	Number of data
Mean	0.059	0.263	0.393	0.374	0.64	0.317	0.703	Mean
Maximum	7.55	72.0	76.5	67.3	37.4	54.5	49.5	Maximum
Upper quartile	0.04	0.11	0.36	0.37	0.74	0.29	0.83	Upper quartile
Median	0.02	0.03	0.11	0.15	0.35	0.1	0.45	Median
Lower quartile	0.01	0.01	0.03	0.05	0.12	0.04	0.2	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum
Variance	0.0472	2.6662	1.4878	1.166	1.5071	0.7829	1.5912	Variance
CV	3.698	6.202	3.104	2.89	1.917	2.795	1.796	CV
Skewness	17.425	22.768	20.693	27.487	9.41	20.538	14.437	Skewness

Figure 14.1 Gold grade distribution by stratigraphic unit

14.5 Geologic Model

THM geologists provided a three-dimensional, wire-framed geological model of the major stratigraphic units and major fault structures. South of the Lillian Fault, the stratigraphic units modeled were the Cambrian, Money Knob, Upper Sediments, Main Volcanics and Lower Sediments. North of the Lillian Fault, most of the material is undifferentiated Upper Sediments, with a small amount of Volcanics and Lower Sediments modeled. These represent the major stratigraphic units that host the mineralization. No other geologic features with possible controls were modeled.

14.6 Composite Statistics

All of the available drilling was composited into fixed length 10 m composites. Composite residuals <4 m in length were added to the previous composite. These composites were back-tagged with the stratigraphic unit using the rock type block model developed from the defined geological three-dimensional wire frames.

The composite data was declustered by estimating a nearest-neighbor value into each block. The declustered composite statistics are tabulated below (Table 14.6).

Table 14.6 Gold Composite Statistics	
Mean:	0.36
Variance:	0.32
C. of V.:	1.57
Min:	0.00
Q1:	0.06
Median:	0.20
Q3:	0.46
Max:	20.69

The composite data was used to set the gold indicator thresholds. Since the coefficient of variation of the composite data is relatively low, only nine indicator thresholds were needed to fully define the gold distributions. The indicator thresholds were chosen at the low end to have approximately 20% of the data per class and at the high end to have 10% – 11% of the metal per class (Table 14.7). With MIK, top cutting of the assays is not necessary. In this case all composite values higher than 2.0 g/tm Au (the highest threshold) are treated the same as "high grade."

Table 14.7
Gold Indicator Statistics

		Data		Metal		
Rock Type	Threshold	%	Cum%	%	Cum%	Median
1	0.08	18.9	20.8	2.5	2.5	0.05
2	0.18	24.2	43.0	8.8	11.3	0.13
3	0.33	22.6	65.6	16.1	27.4	0.25
4	0.45	10.4	76.0	11.5	38.8	0.39
5	0.60	8.5	84.5	12.5	51.4	0.51
6	0.72	4.6	89.1	8.5	59.9	0.65
7	0.90	3.8	92.9	8.8	68.7	0.80
8	1.20	3.4	96.3	10.1	78.9	1.04
9	2.00	2.7	98.9	11.2	90.0	1.43
Max	20.69	1.1	100.0	10.0	100.0	2.74

Because significant grade contrasts were noted between the different stratigraphic units from the assay statistics, contact analysis was performed in the previous study (Klipfel, et al., 2009b) using the composite data to evaluate grade discontinuities at the stratigraphic contacts. Wherever a contact was crossed with a drill hole, the grade profile was examined on either side of the contact. Contacts were evaluated from the Cambrian to the Upper Sediments, from the Upper Sediments into the Main Volcanics, and from the Main Volcanics into the Lower Sediments.

The grade contrast is fairly significant between the Cambrian and Upper Sediments. In the vicinity of the contact, the average grade of the Cambrian is 0.30 g/mt Au while the Upper Sediments is 0.45 g/mt Au (Figure 14.2).

The grade contrast is also fairly significant between the Upper Sediments and the Main Volcanics. The contact between the Main Volcanics and the Lower Sediments is the most significant with the grade in the Main Volcanics being 0.63 g/mt Au and the Lower Sediments at 0.43 g/mt Au. The additional data available for this update did not appear to alter these relationships, and the contact analysis was not repeated.

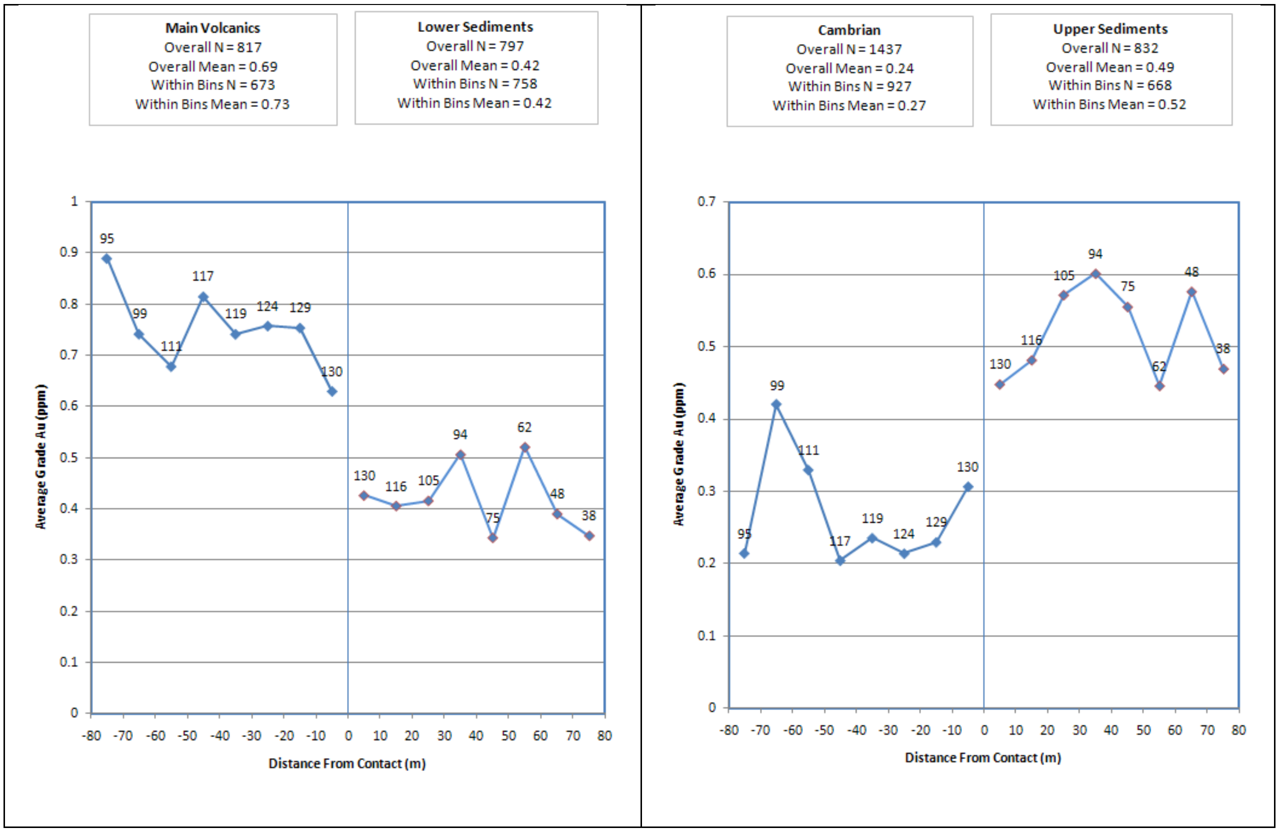


Figure 14.2 Contact Plots

Because of the sharp contrasts in gold grade between the different units, it was decided to treat the boundaries between the different units as hard boundaries. That is, the blocks of a given unit were estimated using only the composite data that fell within the same unit. The Main Volcanics are significantly more mineralized than the surrounding units for reasons not fully understood. It is not geologically unreasonable to see grade discontinuities at contacts. The use of hard boundaries will have an impact on the local estimates because the data has been partitioned. Overall, however, the use of either hard boundaries or soft boundaries would have a minimal effect on the global estimate.

14.7 Spatial Statistics

Analysis of the additional data available for the update indicated that there were no significant changes in the spatial statistics. The variography from the November 2010 update, also used for the August 2011 update, was therefore retained for gold.

Indicator variograms were calculated for each of the indicator thresholds within each of the stratigraphic domains. Variogram models were fitted for each. Because the data was so heavily partitioned, the results from the individual domains were generally unsatisfactory. Many units are relatively thin, especially in the Main Volcanics, making it very difficult to infer a model of vertical continuity. For this reason, the use of the partitioned data for variogram calculations was abandoned and all of the data was used to calculate a set of average indicator variograms that were used over all domains. The average indicator variograms that were used for estimation of the gold indicators in all domains are shown in Table 14.8.

In prior work, indicators were generated for oxidation and dikes (Kint) so that related metallurgical recoveries could be assigned in the block model. Because metallurgical recoveries for the present study are based on representative (alteration and mineralization type, oxidation, and Kint%) bulk samples for the major stratigraphic units, these indicators are not utilized in the present study.

A portion of the mineralization associated with little or no alteration in the Lower Sediments stratigraphic unit is associated with reduced gold recovery. MinIndex, a measure of alteration based on geologic logging of alteration and arsenic content, was used to divide mineralized Lower Sediments into "Bleached (altered)" and "Unbleached" populations, which were assigned differing metallurgical recoveries based on representative bulk composites. Composited (10 m) MinIndex values were assigned to blocks in the model by inverse distance interpolation, using similar search ranges and orientation as was used for gold estimation.

Block values were also estimated for a number of minor elements that are of interest in terms of environmental chemistry, namely sulfur, calcium and magnesium. Assay data for these elements was composited in 10 m fixed length composites, and interpolated using ordinary kriging based on three-dimensional variographic analysis using the 10 m composite data. Hard boundaries were not used for interpolation of the minor elements.

Table 14.8
Average Gold Indicator Variograms

Indicator	Sill	Range X	Range Y	Range Z
1	0.50			
	0.39	90	62	67
	0.11	570	303	188
2	0.48			
	0.35	69	116	61
	0.17	208	399	390
3	0.48			
	0.36	77	115	57
	0.16	190	386	375
4	0.54			
	0.32	58	104	99
	0.14	324	405	158
5	0.55			
	0.33	61	82	61
	0.12	191	442	253
6	0.60			
	0.30	59	72	64
	0.10	183	562	242

Table 14.8 Average Gold Indicator Variograms				
Indicator	**Sill**	**Range X**	**Range Y**	**Range Z**
7	0.61			
	0.31	16	50	46
	0.08	159	525	205
8 & 9	0.61			
	0.33	23	42	30
	0.06	106	518	158

14.8 Resource Model

The resource model was constructed to encompass the drilling data and the defined geological model. The resource model for the project was constructed using the UTM NAD27 Alaska coordinate system. The model extents are shown in Table 14.9.

The block size was selected based on the drill hole spacing of 50m (164 ft) to 75m (246 ft)

Table 14.9 Model Extents					
	Minimum (m)	**Maximum (m)**	**Extent (m)**	**Block Size (m)**	**No. of Blocks**
East	427,500	430,800	3,300	15	220
North	7,264,300	7,266,700	2,400	15	160
Elevation	50	560	510	10	51

The gold contained within each block was estimated using MIK with nine indicator thresholds. The block model was tagged with the geological model using a block majority coding method. The contact analysis indicated that there are significant grade discontinuities at the major stratigraphic boundaries. Hard boundaries were used between each of the units. That is, each unit was estimated using only data that also fell within the same unit. There was no potentially economic mineralization outside of the geological model and it was not estimated. The estimation was done in three passes, with progressively larger search distances and varying interpolation parameters. The gold kriging plan is shown in Table 14.10 for all units.

An octant search was used. The kriging plan forces data to be available from a minimum of two octants and from two separate drill holes for an estimate to be made. Each of the gold indicators was estimated independently.

Table 14.10
Gold Kriging Plan

Pass 1	
Minimum No. of Composites	12
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	100 (Maj.), 100 (Semi-Maj.), 60 (Min.)
Search Rotation	Maj. -5° →190°, Semi-Maj. 100°

Pass 2	
Minimum No. of Composites	12
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	200 (Maj.), 160 (Semi-Maj.), 120 (Min.)
Search Rotation	Maj. -5° →190°, Semi-Maj. 100°

Pass 3	
Minimum No. of Composites	8
Maximum No. of Composites	48
Maximum Composites per Octant	6
Maximum No. of Composites per Hole	4
Block Discretization	4 x 4 x 1
Search Distances (m)	300 (Maj.), 250 (Semi-Maj.), 200 (Min.)
Search Rotation	Maj. -5° →190°, Semi-Maj. 100°

14.9 Model Validation

Various forms of model validation were undertaken and are shown below. In all cases, the model appears to be unbiased and fairly represent the drilling data. The composite data was declustered by estimating a nearest-neighbor value into each block.

The global average of the declustered composite values for Measured and Indicated material is 0.392 g/mt Au and the corresponding average block value (e-type estimate or block average calculated from MIK bins) is 0.397 g/mt. The estimated block values are within 2.0% of the composite values. This is reasonable and within the expectations of the model.

The model was visually compared to the composite gold data in both N-S and E-W sections. The estimates were checked to see that they appeared to be consistent with the data and that they were geologically reasonable. In all cases everything appeared reasonable.

Swaths were taken through the model and the averaged block values (e-type MIK estimates) and the averaged declustered composite values (nearest-neighbor estimates) were compared on E-W, N-S and vertical swaths (Figure 14.3). The kriged values have a small amount of spatial smoothing, but generally compare quite favorable to the composite values, with areas of some divergence corresponding to swaths with a low number of samples.

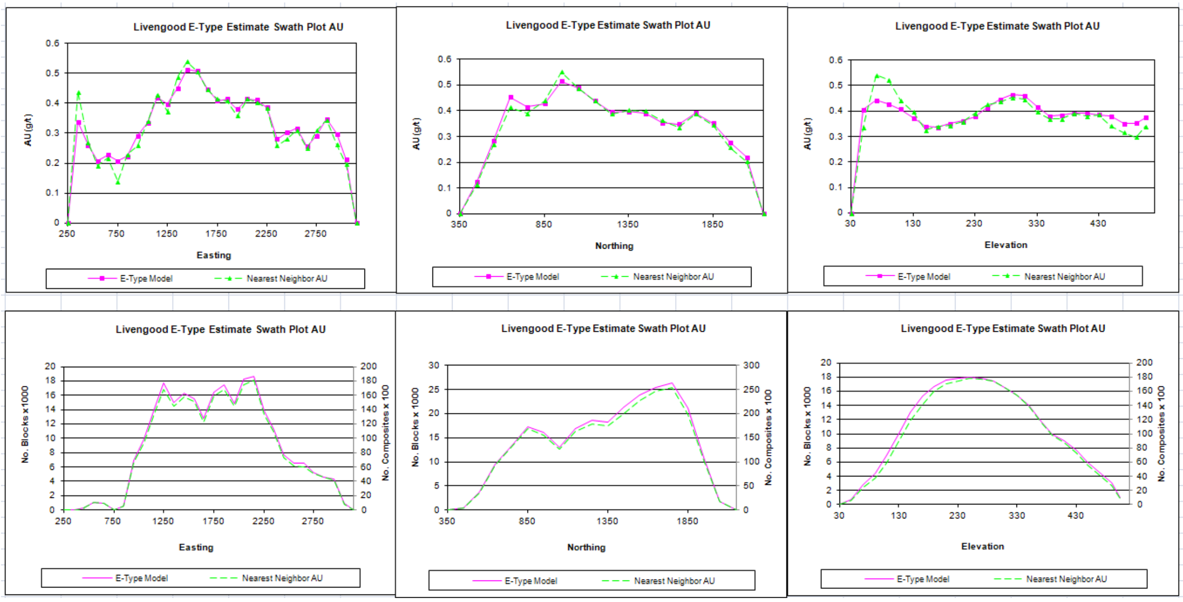


Figure 14.3 Swath plots of E-Type estimate vs. nearest neighbor.

THM commissioned an independent review of the resource estimation methodology as part of its quality assurance program (Schofield, 2010). The review concluded that Multiple Indicator Kriging (MIK) was the appropriate estimation method for the deposit. The MIK approach to recoverable resource estimation has been found to be more useful than Ordinary Kriging (OK) where the size of the ore selection unit is small compared to the spacing of the drill holes, and/or when sensitivity to extreme sample grades exists.

The review made suggestions for adjusting block size, composite lengths, and search radii, but the tonnage, grade, and contained metal of the volumes common to both calculations are quite similar (Brechtel, et al. 2011). The current Livengood resource estimate is larger than would be produced using the alternate (Schofield) assumptions, with the main difference relating to material that is projected below the drill holes when using the current, larger neighborhood search parameters. The location of this material is illustrated by the cross section showing drill-hole data and model blocks (Fig. 14.4), where resource blocks are extrapolated beyond the base of the drill data due to the larger search neighborhood used in the current Livengood resource calculation methodology. This material and that extrapolated laterally are predominantly classified as inferred resource and, therefore, not subject to economic analysis in the current study.

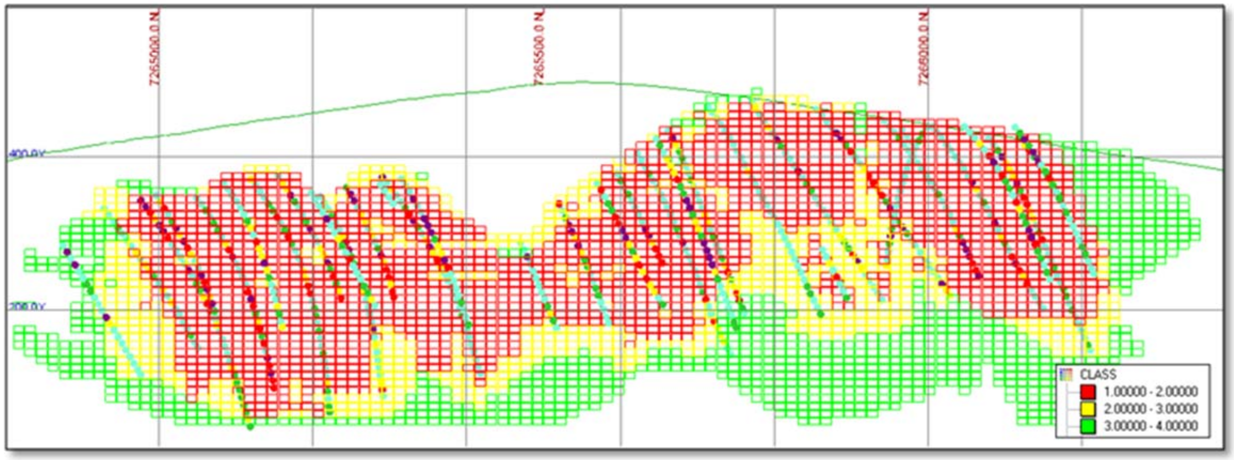


Figure 14.4 Cross-section through the global resource block model from Brechtel et al., 2011
(Blocks in red are classified as measured, yellow as indicated and green blocks as inferred).

14.10 Post-processing of MIK Model

The post-processing of the indicator kriging was done with the GSLIB post processing routine. It is necessary to provide a maximum grade of the distribution. This grade can be calculated as:

$Z_{max} = Z_{cn} + 3(Z_n - Z_{cn})$

In this formula, Z_{cn} is the uppermost indicator threshold and Z_n is the mean of values > Z_{cn}. Considering a mean of 3.45 ppm (raw composites), the maximum grade used in the post-processing was calculated to be 6.35 ppm.

The multiple indicator kriging produces an estimate of the distribution of grade within a block rather than just a single average grade of a block. The distribution produced is the distribution of composite sized units within the block not minable units. It is therefore necessary to correct the distribution so that the distribution represents selective mining units (SMUs), not composite sized units. This correction is called a change of support correction. Since the average grade of the block is the same whether mined in one scoop or mined by a core drill, the correction does not change the average grade of the block, but only reduces the variance of the distribution.

The variance reduction factor is the ratio of the variance of an SMU within a block to the variance of a composite within a block. This is calculated using average variogram values. The variance of the SMU within the block is the variance of a composite within a block minus the variance of a composite within an SMU. Since the estimated blocks are small relative to the data spacing, the effective block size was taken to be 131 ft x 131 ft (40 m x 40 m) approximately half the drill spacing.

The method used for the change of support was an indirect lognormal correction. This correction uses the ratio of standard deviations rather than the ratio of variances. This is just the square root of the ratio of variances.

For the purposes of the change in support calculation and global resource estimation, the mining SMU was assumed to be 16.4 ft x 16.4 ft x 32.8 ft (5 m x 5 m x 10 m) selectivity. Because the projected size of the operation indicates a larger SMU, the estimation of the "Economic" resource is, accordingly, based on a larger SMU size of 24.6 ft x 24.6 ft x32.8 ft (7.5 x 7.5 m x 10m).

A correction for change of support was applied on a block-by-block basis with a global reduction target based on the overall gold variography. This is done on a trial and error basis to find a block reduction factor that will achieve the calculated target global variance reduction.

14.11 Resource Classification

The resource is broken down into three categories: Measured, Indicated, and Inferred. As mentioned, the MIK interpolation was done in three passes, with the search distances and other relevant interpolation parameters varying from pass to pass. The interpolation parameters include the distance and orientation of the search neighborhood, the minimum and maximum number of samples, and the minimum number of holes and octants informed for each pass. These parameters were selected to reflect levels of confidence commensurate with classification into Measured, Indicated, and Inferred categories. Blocks are therefore classified with respect to the pass in which they are interpolated, with pass 1 corresponding to the Measured category, pass 2 corresponding to the Indicated category and pass 3 corresponding to the Inferred category. The estimation variance from the estimation of the third indicator (median indicator), along with the number of composites used, number of drill holes used, and the distance to the nearest composite, was also saved for each block estimated for possible use in refining the classification. The estimation variance provides a good measure of the confidence in the estimate, remaining relatively low when data is near and evenly spaced around the block being estimated, and rising rapidly with extrapolation.

On average, Indicated blocks are within 111.5 ft (34 m) of the nearest composite, and are indicated by 27 composites from at least eight drill holes. On average, Inferred blocks are within 276 ft (84 m) of the nearest composite, and are informed by 20 composites from at least six drill holes.

15.0 Mineral Reserves Estimate

15.1 Resource Check

MDA compared the resource reported by THM with the resource contained in the model received by MDA from Reserva International in order to evaluate whether there were any inconsistencies in the ways different software reports the material contained in the same grade model. The model used to report the resource was a 49.2 ft x 49.2 ft x 32.8 ft, (15 m x 15 m x 10 m) indicator kriged model. The model used to plan the mine was the same model but based on 24.6 ft x 24.6 ft x 32.8 ft, (7.5 m x 7.5 m x 10 m) parcels of this model. Table 15.1 illustrates that the model's reported resource compares closely to the model summary generated by MDA.

Table 15.1
Livengood Model Check

Item	Classification	Gold Cutoff (g/mt)	Tonnes (Million)	Gold (g/mt)	Million oz Gold
Reported	Measured	0.20	994	0.52	16.4
Reported	Indicated	0.20	112	0.45	1.6
Reported	Total M & I	0.20	1,106	0.51	18.0
Reported	Inferred	0.20	438	0.41	5.8
MDA	Measured	0.20	980	0.53	16.7
MDA	Indicated	0.20	111	0.48	1.7
MDA	Total M & I	0.20	1,091	0.53	18.4
MDA	Inferred	0.20	427	0.43	5.9

15.2 Assumptions and Methods

The Livengood deposit is planned to be mined by surface mining methods with the ore-grade material crushed and processed by a gravity-whole ore carbon in leach (CIL) plant designed for 100,000 tons (90,718 tonnes) of feed per day, 365 days per year. The mine must supply 36.5 million tons (33.2 million tonnes) of ore to the plant annually. The production is planned to be lower during the first year of operation before full capacity is achieved. Mining is planned on 32.8 ft (10 m) bench intervals.

15.2.1 Pit Optimization

The pit optimization was based on a $1,250 gold price. Table 15.2 shows the results of pit optimization for gold prices between $400 and $2,500/oz of gold. It should be noted that at fairly low gold prices ($638/oz Au), the bottom of the THM model is contained in the optimized pit. Future studies should extend the grade model deeper. Pit 36, the optimized pit at a gold price of $1,250/ oz, is the basis for final pit design. The production schedule utilizes cutoff grades based on a $1,500/oz gold price to determine if the material will be processed or placed in an overburden stockpile area.

Table 15.2 Optimization Results									
Pit	Gold Price $/oz Au	Total Tonnes 000's	Waste Tonnes 000's	Ore Tonnes 000's	Gold Grade g Au/t	Gold oz 000's	Strip Ratio	Max Bench	Min Bench
2	400	702.0	531.9	170.1	2.05	11,194	3.13	47	29
4	450	1,303.7	989.1	314.6	1.91	19,274	3.14	48	28
6	500	2,385.7	1,810.8	574.9	1.75	32,352	3.15	48	28
8	550	4,340.5	3,263.8	1,076.7	1.57	54,428	3.03	48	26
10	600	12,102.5	9,221.4	2,881.1	1.41	130,308	3.20	48	25
12	650	57,604.7	40,908.1	16,696.6	1.16	619,922	2.45	48	24
14	700	97,086.9	67,937.3	29,149.6	1.11	1,036,795	2.33	48	17
16	750	192,494.1	137,600.3	54,893.8	1.07	1,892,042	2.51	48	13
17	775	213,671.0	151,201.5	62,469.5	1.05	2,112,947	2.42	48	12
18	800	236,440.9	165,099.1	71,341.8	1.03	2,350,688	2.31	48	12
20	850	335,758.9	230,303.2	105,455.7	0.96	3,254,345	2.18	48	10
21	875	390,328.0	267,023.5	123,304.5	0.94	3,719,442	2.17	50	9
22	900	493,529.4	340,120.3	153,409.1	0.92	4,512,533	2.22	50	9
24	950	699,918.1	489,017.7	210,900.4	0.89	6,010,904	2.32	50	5
25	975	719,568.7	495,824.4	223,744.3	0.87	6,276,774	2.22	50	5
26	1000	765,671.4	523,652.3	242,019.1	0.86	6,686,326	2.16	50	5
28	1050	796,413.8	529,741.3	266,672.5	0.83	7,150,146	1.99	50	3
30	1100	844,939.4	549,618.5	295,320.9	0.81	7,701,421	1.86	50	3
32	1150	912,273.3	582,841.7	329,431.6	0.79	8,311,854	1.77	51	3
34	1200	960,819.7	602,658.8	358,160.9	0.77	8,804,155	1.68	51	2
36	1250	1,009,422.1	622,661.9	386,760.2	0.75	9,278,623	1.61	51	2
38	1300	1,063,715.9	645,972.8	417,743.1	0.73	9,763,106	1.55	51	1
40	1350	1,124,921.1	675,053.0	449,868.1	0.71	10,260,025	1.50	51	1
42	1400	1,151,323.6	677,639.7	473,683.9	0.70	10,592,310	1.43	51	1
44	1450	1,197,855.9	694,921.9	502,934.0	0.68	11,010,884	1.38	51	1
46	1500	1,235,859.9	705,611.2	530,248.6	0.67	11,386,726	1.33	51	1
48	1550	1,277,286.2	720,137.7	557,148.4	0.66	11,751,364	1.29	51	1
50	1600	1,336,144.9	747,564.8	588,580.1	0.65	12,210,097	1.27	51	1
52	1650	1,390,474.0	773,054.3	617,419.7	0.63	12,583,624	1.25	51	1
54	1700	1,441,542.1	795,423.7	646,118.4	0.62	12,956,158	1.23	51	1
56	1750	1,477,442.5	807,783.7	669,658.8	0.62	13,245,794	1.21	51	1
58	1800	1,530,197.9	832,787.6	697,410.3	0.61	13,577,395	1.19	51	1
60	1850	1,559,032.4	840,617.2	718,415.2	0.60	13,813,891	1.17	51	1
62	1900	1,582,698.1	843,776.5	738,921.7	0.59	14,035,661	1.14	51	1
64	1950	1,612,484.6	852,864.1	759,620.5	0.58	14,254,296	1.12	51	1
66	2000	1,637,197.5	858,023.4	779,174.1	0.58	14,454,979	1.10	51	1
68	2050	1,669,805.4	870,300.0	799,505.5	0.57	14,671,771	1.09	51	1
70	2100	1,685,598.8	868,532.6	817,066.3	0.57	14,841,896	1.06	51	1
72	2150	1,768,472.3	924,557.2	843,915.1	0.56	15,186,775	1.10	51	1
74	2200	1,792,279.7	930,403.4	861,876.4	0.55	15,357,981	1.08	51	1
76	2250	1,820,234.4	940,295.6	879,938.8	0.55	15,525,936	1.07	51	1
78	2300	1,836,481.9	941,037.4	895,444.5	0.54	15,662,927	1.05	51	1
80	2350	1,869,093.7	955,685.5	913,408.2	0.54	15,828,746	1.05	51	1
82	2400	1,890,210.5	960,955.3	929,255.2	0.53	15,964,320	1.03	51	1
84	2450	1,904,906.6	961,625.0	943,281.6	0.53	16,084,148	1.02	51	1
86	2500	1,918,019.9	962,963.6	955,056.3	0.53	16,179,977	1.01	51	1

Note: Bench in the table refers to the model bench, counted up from the bottom of the model (50m= bench1).

The optimized pit at the base case gold price of $1,250/oz is shown in Figure 15.1.

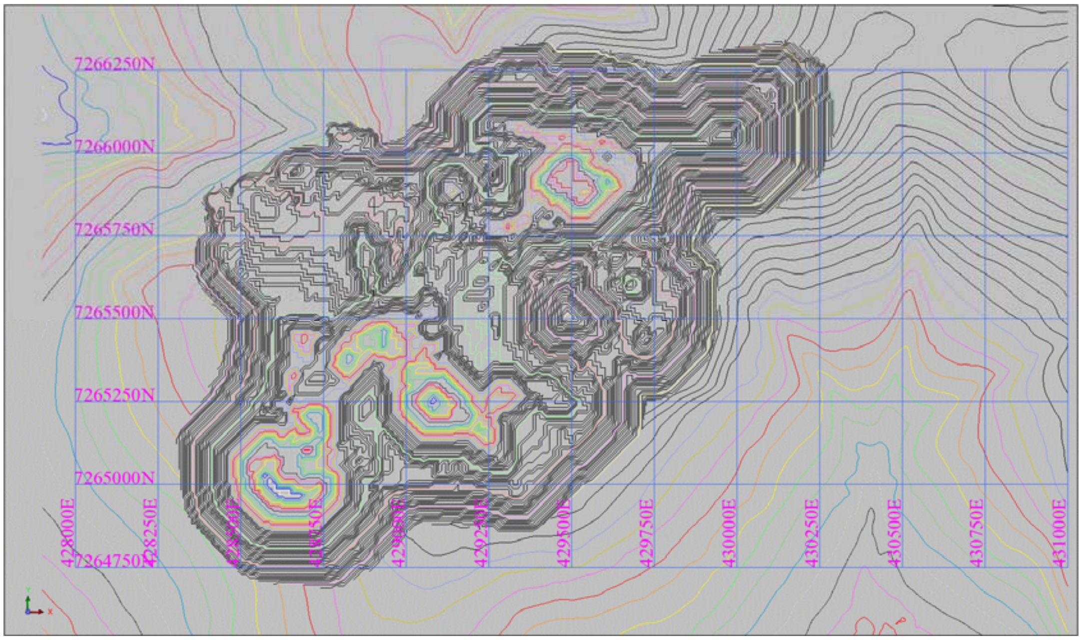


Figure 15.1 Optimized Base Case Final Pit

The Livengood grade model was prepared by Reserva International using multiple indicator kriging (MIK) methodology to estimate grades into 49.2 ft x 49.2 ft x 39.8 ft, (15m x 15m x 10m) "master" blocks. These full blocks were used to estimate grades into selective mining unit (SMU) size (7.5m x 7.5m x 10m) parcels of the master block by Reserva International prior to pit optimization. The model also contained estimated process recovery and cost information for each block parcel. Table 15.3 illustrates the recovery information and process costs assigned by rock type for pit optimization. Note that the cost used in the pit optimization for G & A and transportation and refining was $0.89/mt of ore and is included in process + G & A column. A mining cost of $1.80/mt of material and a 2.5% royalty were also used in the pit optimization. The pit optimization did not vary mining cost by bench and was higher than the final estimated cost of $1.61 per tonne. The base case gold price for the optimization was $1,250/oz of gold.

Table 15.3
Process Recovery and Cost by Rock Type

Material	Process	G&A	Process + G&A	% Recovery	Internal Cutoff Grade
	$/tonne ore	$/tonne ore	$/tonne ore		g Au/t ($1500/oz AU)
Rock Type 4 - Cambrian	$11.17	$0.92	$12.08	84.2%	0.300
Rock Type 5 - Upper Seds - Sunshine	$11.81	$0.92	$12.72	87.7%	0.303
Rock Type 6 - Upper Seds - Core	$11.56	$0.92	$12.48	76.7%	0.340
Rock Type 7b - Lower Seds - Bleached	$12.02	$0.92	$12.93	58.5%	0.461
Rock Type 8; 9 - Vocanics	$12.68	$0.92	$13.59	84.8%	0.335
*Rock Type 4 - Final	$10.76	$0.98	$11.74	84.2%	0.289
*Rock Type 5 - Final	$11.38	$0.98	$12.36	87.7%	0.292
*Rock Type 6 - Final	$11.11	$0.98	$12.09	76.7%	0.327
*Rock Type 7b - Final	$11.57	$0.98	$12.55	58.5%	0.445
*Rock Type 8.9 - Final	$12.23	$0.98	$13.21	84.8%	0.323

*Note: The final costs were revised late in the study. It was decided that optimization and cutoff grades that used the costs prior to the final numbers did not need to be revised, as the revision would only add an insignificant amount of low grade material

15.3 Summary of Reserves from Detailed Surface Mine Design

The proven and probable reserves which are contained in the final pit are summarized in Table 15.4 and match the production schedule. The indicator kriged grade model contained four SMU parcels in each 49.2 ft x 49.2 ft x 39.8 ft (15 m x 15 m x 10 m) block which are noted in the table as parcels 1- 4.

Table 15.4
Livengood Reserves

		Parcel 1		Parcel 2		Parcel 3		Parcel 4		Totals		
Class	**Rock Type**	**000's Tonnes**	**g Au/mt**	**000's Tonnes**	**g Au/mt**	**000's Tonnes**	**g Au/mt**	**000's Tonnes**	**g Au/mt**	**000's Tonnes**	**g Au/mt**	**000's Au oz**
Proven	4	1,330.9	0.360	8,753.9	0.408	19,144.7	0.488	29,017.8	0.821	58,247.3	0.639	1,196.6
Proven	5	14,022.7	0.404	27,122.1	0.471	37,433.8	0.532	48,013.6	0.720	126,592.2	0.576	2,344.6
Proven	6	5,661.0	0.517	14,757.9	0.578	23,645.4	0.642	36,847.9	0.886	80,912.3	0.733	1,906.0
Proven	7b	2,185.5	0.567	9,023.0	0.614	16,471.9	0.676	23,339.7	0.920	51,020.0	0.772	1,266.3
Proven	8	409.2	0.471	1,454.8	0.523	2,104.4	0.602	2,738.9	0.803	6,707.4	0.659	142.1
Proven	9	17,042.5	0.484	28,831.8	0.604	31,992.5	0.740	33,147.1	1.107	111,013.9	0.775	2,766.0
Proven	**Totals**	**40,651.8**	**0.461**	**89,943.6**	**0.540**	**130,792.7**	**0.616**	**173,105.0**	**0.875**	**434,493.0**	**0.689**	**9,621.5**
Probable	4	112.0	0.398	758.9	0.437	1,750.5	0.521	2,508.5	0.958	5,129.8	0.720	118.7
Probable	5	99.2	0.363	240.2	0.469	477.9	0.491	686.0	0.613	1,503.4	0.535	25.8
Probable	6	69.2	0.402	478.6	0.446	861.0	0.530	1,345.9	0.785	2,754.6	0.637	56.4
Probable	7b	103.3	0.597	623.9	0.593	1,310.8	0.643	1,967.3	0.830	4,005.3	0.726	93.5
Probable	8	237.2	0.446	546.2	0.529	713.6	0.620	824.4	0.868	2,321.2	0.669	49.9
Probable	9	391.7	0.423	1,149.7	0.524	1,356.4	0.677	1,518.6	1.137	4,416.4	0.773	109.7
Probable	**Totals**	**1,012.5**	**0.436**	**3,797.6**	**0.505**	**6,470.0**	**0.588**	**8,850.7**	**0.899**	**20,130.8**	**0.702**	**454.0**
P + P	4	1,442.9	0.363	9,512.9	0.4103136	20,895.1	0.491	31,526.3	0.832	63,377.1	0.645	1,315.2
P + P	5	14,121.9	0.404	27,362.3	0.4709824	37,911.7	0.531	48,699.6	0.718	128,095.6	0.576	2,370.4
P + P	6	5,730.2	0.516	15,236.5	0.5738538	24,506.4	0.638	38,193.8	0.882	83,666.9	0.730	1,962.4
P + P	7b	2,288.8	0.568	9,646.9	0.6126418	17,782.6	0.674	25,307.1	0.913	55,025.3	0.769	1,359.8
P + P	8	646.4	0.462	2,001.0	0.5246377	2,818.0	0.607	3,563.3	0.818	9,028.6	0.662	192.0
P + P	9	17,434.2	0.483	29,981.6	0.6009321	33,348.9	0.737	34,665.7	1.108	115,430.3	0.775	2,875.7
P + P	**Totals**	**41,664.3**	**0.461**	**93,741.2**	**0.539**	**137,262.7**	**0.614**	**181,955.6**	**0.876**	**454,623.8**	**0.689**	**10,075.6**

16.0 Mining Methods

16.1 Introduction

This Feasibility Study is based on a plan to mine and process 100,000 t (90,718 mt) of ore daily. A production schedule was developed based on pit optimization results and designed pit phases to accomplish this. Pit optimization indicated that a mine life of about 14 years could be achieved. During the first several years of production, low-grade materials are to be stockpiled to allow processing of higher grade materials.

A two year and three month pre-production period is required. Initially, a contractor will remove 7.6 Mt (6.9 Mmt) of material preparing the site for larger mine equipment. The overburden materials removed during pre-production stripping will be used for construction of facilities (tailings embankment and basin, Gertrude Creek embankment, out-of-pit haul and access roads). Ore will be stockpiled during preproduction for later plant feed.

16.2 Mining Methods

The Livengood deposit is planned to be mined by surface mining methods with the ore-grade material crushed and processed by a gravity – whole ore carbon in leach (CIL) plant designed for 100,000 t (90,718 mt) of feed per day, 365 d/y. The mine must supply 36.5 Mt (33.1 Mmt) of ore to the plant annually. The production is planned to be lower during the first year of operation before full capacity is achieved. Mining is planned on 32.8 ft (10 m) bench intervals.

16.3 Pit Slope Geotechnical Evaluation

The following information summarizes the findings of the SRK report "Feasibility Pit Slope Evaluation, Livengood Project" dated July, 2013.

16.3.1 Data Collection

A field data collection program was designed and carried out for the project with the primary objective of rock mass characterization and discontinuity orientation to serve as the basis of geotechnical model development. Field data collection consisted of geotechnical core logging and discontinuity orientation, point load testing and laboratory strength testing. The Livengood site has very minimal outcrop exposure and, consequently, geotechnical mapping was not able to be carried out to a significant degree.

THM technicians logged geotechnical data for all of the 2010 resource drillholes providing the first geotechnical data for mine design; 17 of these 2010 holes (totaling 22,227 ft (6,470 m)) were located within the proposed open pit area and were considered in the development of the geotechnical model. Based on the 2010 information, two supplemental geotechnical specific drilling campaigns were undertaken in 2011 (three holes totaling 2,700 ft (823 m)) and 2012 (four holes totaling 4,508 ft (1,374 m)). Core from these holes was logged by SRK personnel at the drill rig on a 24-hour basis in order to orient the core and observe the core in its most undisturbed state. The distribution of 24 geotechnical drillholes used in the analysis is shown on Figure 16.1.

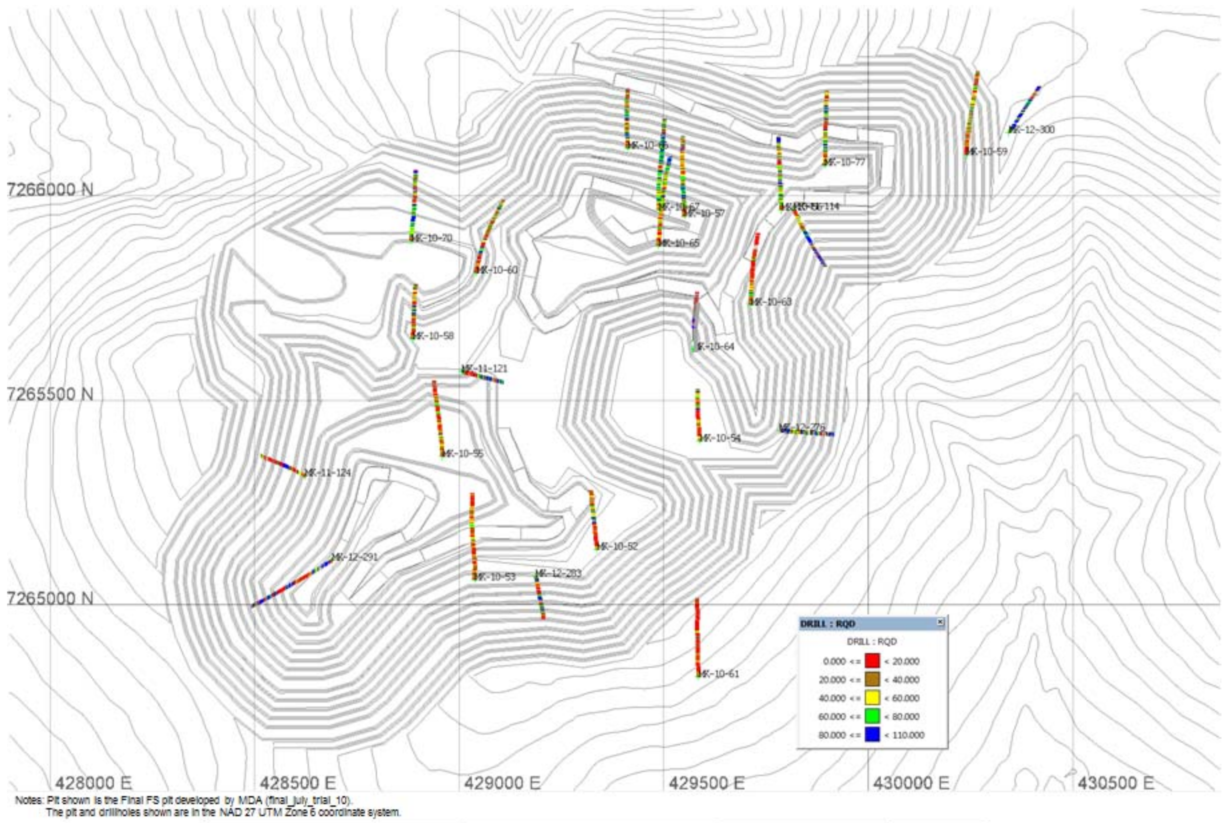


Figure 16.1 Location of Drillholes Used for Geotechnical Analysis

A total of 107 core samples were selected for laboratory testing in the course of the drill programs. The overall laboratory test program included 68 uniaxial compressive strength, 15 triaxial compressive strength, 19 Brazilian tensile strength and 29 direct shear tests. The geomechanical testing was conducted at the University of Arizona Mining Rock Mechanics Laboratory in Tucson, Arizona and at the Agapito Associates Inc. laboratory in Grand Junction, Colorado.

Evaluation of the field and laboratory data indicates a high degree of variability in rock strength and geologic structure at the Livengood project. This natural variation in rock strength and structure suggests that a probability-based method of analysis is most appropriate, thereby yielding a higher confidence in the design than would strictly deterministic analyses. Probabilistic methods differ from deterministic methods in that each model parameter is characterized by a statistical distribution of values having a central tendency and some variation around that central tendency, rather than by a single unique value which could lead to overly conservative designs. SRK used statistical modeling techniques for both the bench scale and overall slope stability analyses.

16.3.2 Geotechnical Model

The Livengood project is located within a geologically complex environment composed of interlayered sediments and volcanics that have undergone intense thrusting and faulting. Results of the data collection programs support this, showing heavily fractured, weak to moderate strength rock with various types of alteration.

The field and laboratory data was used to calculate rock mass rating (RMR) values according to the Bieniawski (1989) system for each core run. This data was used as the primary means of evaluating the overall quality of the various rock types and stratigraphies encountered. It was determined from data analysis that the Money Knob Sequence, Upper Sediments, Main Volcanics, Lower Sediments (including the Lower Sand) and Cambrian rock types are each mechanically similar such that they can be grouped to form their own individual engineering units for pit slope analysis and that further subdivisions within each stratigraphic unit is not warranted. Given that nearly all of the Sunshine area geologic materials are believed to be the Upper Sediments and demonstrated similar geotechnical characteristics, the materials were classified together as one engineering unit, i.e., Sunshine Upper Sediments. Statistical values for each engineering unit are summarized Table 16.1.

Table 16.1
Distributions of RMR (1989) per Engineering Unit

Engineering Unit	No.	Mean	Std. Dev.
Money Knob	106	54	10
Cambrian	166	55	14
Main Volcanics	64	52	13
Upper Seds (Core Zone)	211	56	14
Lower Seds	190	53	13
Upper Seds (Sunshine)	193	62	14

In addition to the RMR value, the intact rock strength, described in terms of uniaxial compressive strength (UCS), is an important indicator of overall rock mass quality. In order to develop a large population of UCS data for statistical analysis, all 1,923 valid point load tests (PLT) taken during the core logging program were multiplied by correlation factors to estimate UCS values for each test. A correlation factor was developed for each engineering unit according to ASTM standards by pairing each laboratory UCS test with one or more adjacent PLTs which generally resulted in linear relationships between the two variables. Table 16.2 contains a statistical summary of the overall UCS data per engineering unit.

Table 16.2
Distributions of UCS per Engineering Unit

Engineering Unit	No.	Mean	Std. Dev.
Money Knob	65	20	15
Cambrian	227	88	172
Main Volcanics	106	69	47
Upper Seds (Core Zone)	249	32	34
Lower Seds	290	36	26
Upper Seds (Sunshine)	808	59	42

16.3.3 Slope Stability Analyses

SRK evaluated both global and bench scale stability for the proposed Livengood project open pit. Global failure is defined as one that occurs relatively deep through the rock mass, is pseudo-rotational and is of sufficient scale to impact interramp and/or overall slopes. Bench scale failures typically involve only one or two bench levels and can be described as block type failures involving the translation of a block delineated by one or more discontinuities.

Representative overall slope models were constructed for a total of six critical design sections as shown on Figure 16.2 to confirm the stability of overall and high interramp slopes. The critical sections were selected to represent the anticipated most adverse stability conditions. The current (2012) Livengood three-dimensional stratigraphic and structural models were used to generate the two–dimensional cross sections for modeling. The overall slopes were analyzed with limit equilibrium methods using the Hoek-Brown (2002) rock mass shear strength criteria and the end of mining groundwater surface exported from the SRK (2012) hydrogeologic model.

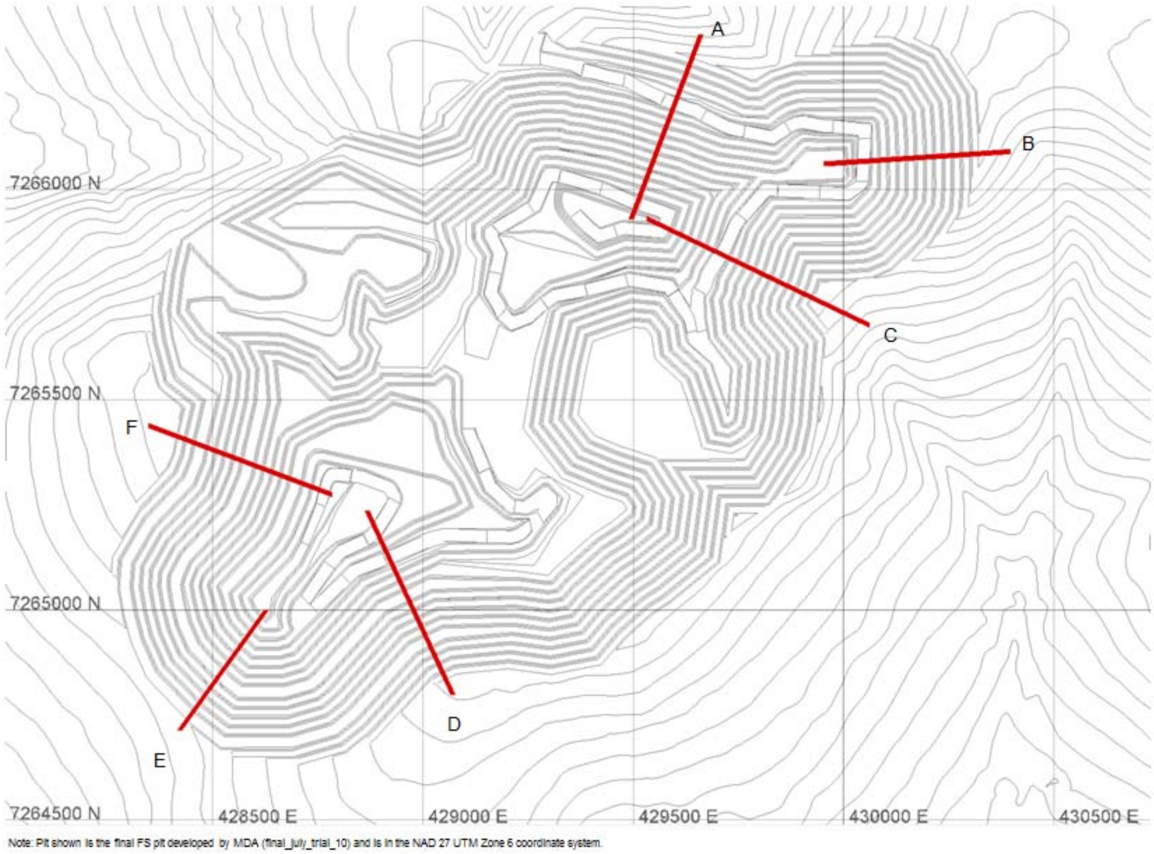


Figure 16.2 Critical Slope Stability Sections

Based on accepted engineering experience, interramp/overall slope designs subject to probabilities of failure ranging from 20% to 30% for slopes with low failure consequences and approximately 5% to 10% for high failure consequences are considered appropriate by SRK for most open pit mines. Slopes of high failure consequence are generally those slopes that are critical to mine operations, such as those on which major haul roads are established, those providing ingress or egress points to the pit, or those underlying infrastructure such as processing facilities or structures. Given the relatively high variability in rock quality and groundwater levels, a maximum probability of failure of 20% was considered to be an appropriate target for the non-critical slopes of the proposed Livengood pit.

The initial final pit provided to SRK with continuous interramp slopes of 42 degrees was analyzed first and then sections where probabilities of failure exceeded acceptable criteria (Sections D, E and F) were modified by reducing the overall slope angle with the addition of geotechnical berms until acceptable criteria was achieved. Geotechnical berms are defined as extra wide catch benches designed to break-up high interramp bench stacks and to provide a wide catchment should an unexpected instability occur above. For Livengood, the geotechnical berms are designed at widths of 25 m, including the normally designed bench width. Results of the analyses are summarized in Table 16.3.

Table 16.3 Overall Slope Stability Analysis Results					
Section	Original Final Pit		Recommended Final Pit		Recommended Revisions to Original Final Pit
	Probability of Failure	Mean Factor of Safety	Probability of Failure	Mean Factor of Safety	
A	3%	1.7	-	-	NA
B	3%	1.9	-	-	NA
C	22%	1.2	-	-	NA
D	26%	1.2	8%	1.4	Geotechnical Berms at 220, 320
E	24%	1.2	14%	1.3	Geotechnical Berms at 220
F	39%	1.1	18%	1.3	Geotechnical Berms at 120, 220, 320

Geotechnical cross sections A and B demonstrated acceptable probabilities of failure for the original final pit with 42 degree interramp slopes. Sections D, E and F, however, indicated higher than acceptable probabilities ranging from 24% to 39%. While the original pit Section C demonstrated a slightly higher probability of failure than targeted, it was considered acceptable due the very narrow extent of the slope in that area and the flexibility to re-design the ramp, should an instability occur.

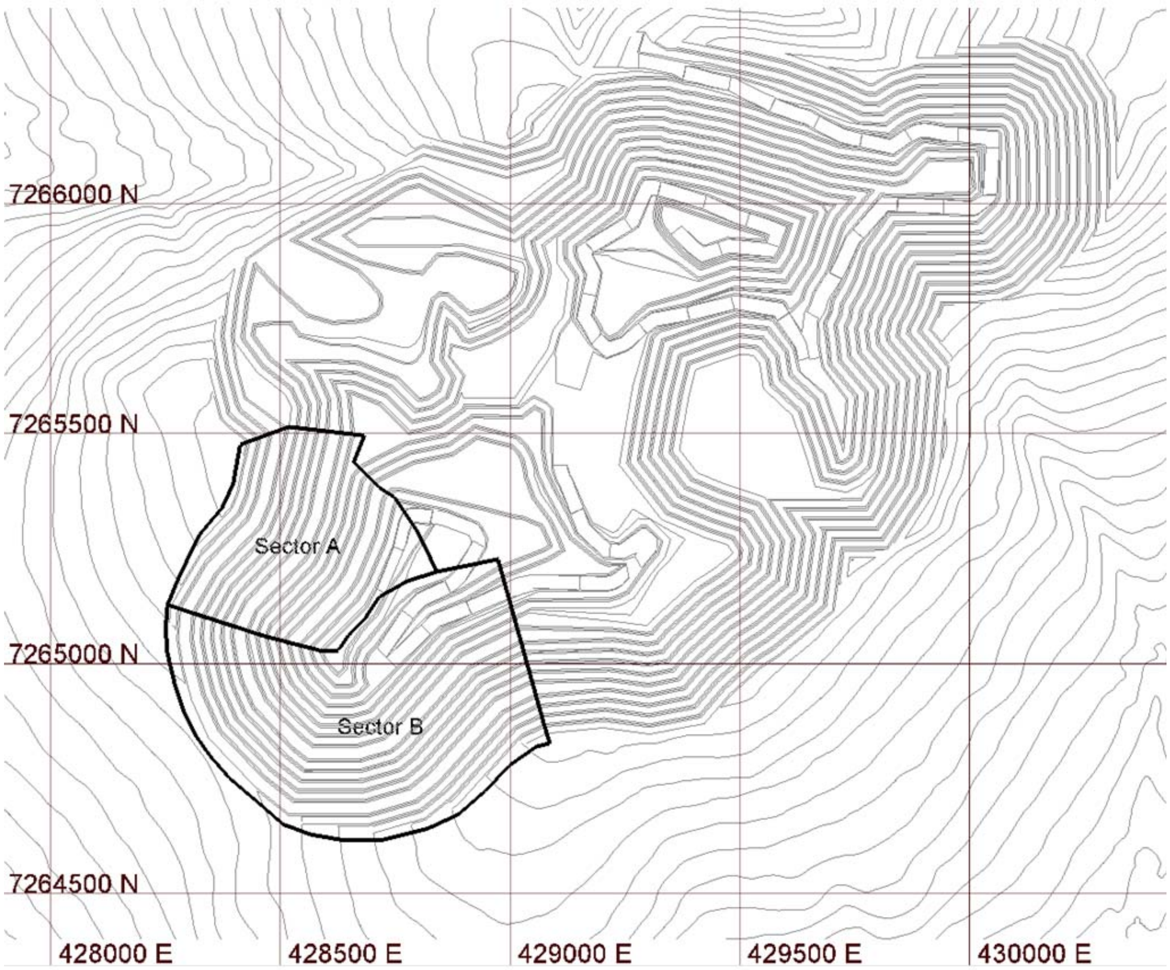
Although it was determined that the performance of the overall and higher interramp slopes at Livengood would best be predicted and subsequently examined using rock mass failure models, an assessment of bench stability was also made to verify that the interramp slope angles recommended could be safely achieved with appropriately dimensioned catch benches.

16.3.4 Pit Slope Design Recommendations

The final pit slope design recommendations are summarized in Table 16.4 with corresponding sectors shown on Figure 16.3.

Table 16.4 Pit Slope Design Recommendations						
Pit Sector	Max. Overall Slope Angle	Max. Interramp Slope Angle	25 m Geotech. Berms (Elev.)	Bench Height (m)	*Bench Width (m)	*Bench Face Angle
A	40	42	120, 220, 320	20	12/14.9	63/70
B	41	42	220, 320	20	12/14.9	63/70
Remaining Areas	42	42	N/A	20	12/14.9	63/70

*The 42° interramp may achieved by either 14.9 m width with 70° bench face angles or 12 m width with 63° bench face angles



Note: Pit shown is final FS pit developed by MDA (final_july_trial_10) and is in the NAD 27 UTM Zone 6 coordinate system..

Figure 16.3 Pit Slope Design Sectors

Recommendations are given for both 63° and 70° bench faces angle configurations. The 63° bench face angle represents the lowest risk of local bench instabilities, particularly for the Sunshine pit north wall, where bedding will dip shallowly into the pit; however, depending on the mining equipment selected and on operational considerations, excavation of a 70° bench face angle may be more practical. Considering the relatively wide catch benches (14.9 m) that would be required to achieve the 42° interramp angle, localized bench sloughing that may occur is expected to be retained by the catch bench beneath. Regardless of which bench configuration is selected, interramp slope angles should not be increased over 42°.

16.4 Mine Design

Designed pits were completed using a bench face angle of 70˚ and a 47.6 ft (14.5 m) wide catch bench every 65.6 ft (20 m). This resulted in an inter-ramp slope of about 43°. This slope angle is generally in accord with the SRK recommendations or has a slightly shallower slope angle, however, in a few places the slopes are slightly steeper than the SRK recommendations. This is due to the SRK recommendations being based on the azimuth of the pit wall. After a review of the final pit by SRK, the catch benches in the southern bulge of the pit on the 120, 220, and 320 elevations were increased to 82 ft, (25 m) from 47.6

ft, (14.5 m). Pit ramps were designed at a 10% gradient and 100 ft (30.5 m) wide. A production schedule was developed utilizing optimized pits 17, 21 and 30 as templates for phase design and optimized pit 36 as the final pit.

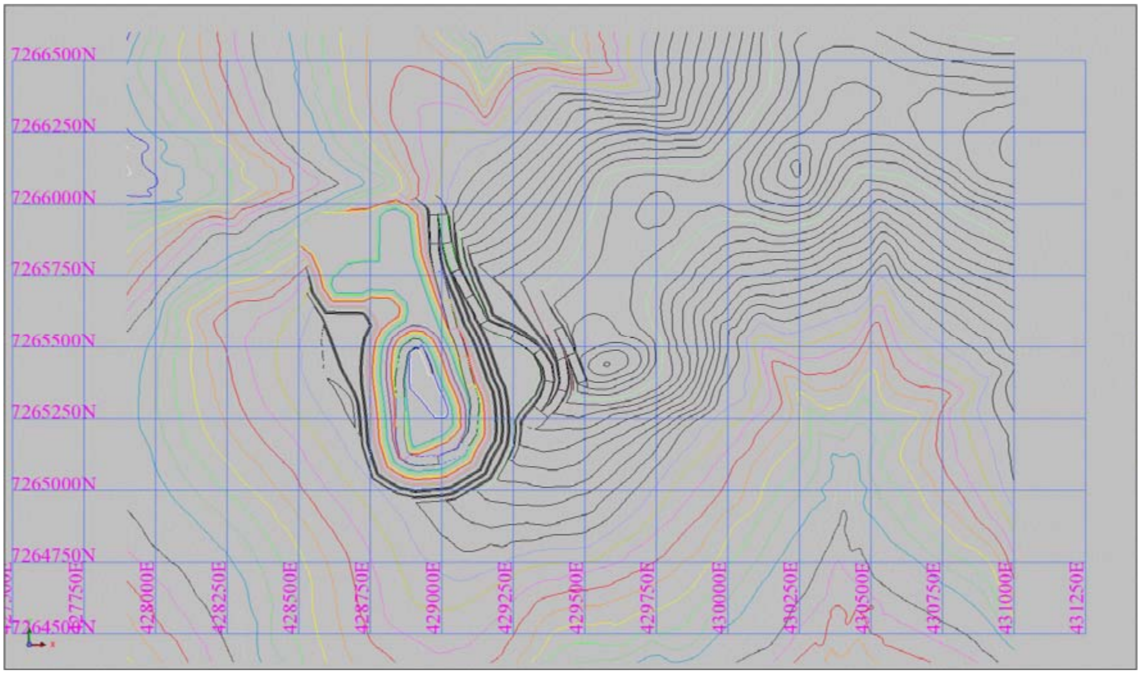
Phases 1 and 2 were based on the two pits contained in optimized pit 17, while Phase 3 was based on the pit contained in optimized pit 21. Phases 4 and 5 were based on optimized pits 30 and 36.

Table 16.5 summarizes the material contained in the phase designs. The phase designs are shown in Figures 16.4 through 16.8.

Table 16.5
Material Contained in the Phase Designs Tabulated For Scheduling

Material	Item	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Totals
HG*	000's Tonnes	34,673.7	18,878.2	30,904.2	50,906.1	31,407.6	166,769.8
	g Au/t	1.12	0.89	1.06	1.00	0.98	1.02
	000's oz Au	1,243.9	540.3	1,052.4	1,631.0	985.9	5,453.6
	Rec %	81.7%	87.5%	81.1%	80.2%	81.5%	81.7%
	000's Rec oz Au	1,015.9	472.9	853.2	1,307.6	803.4	4,453.1
MHG*	000's Tonnes	15,811.5	12,768.0	19,732.7	38,386.8	26,251.5	112,950.4
	g Au/t	0.60	0.53	0.61	0.62	0.59	0.60
	000's oz Au	303.5	215.8	386.0	765.3	496.1	2,166.7
	Rec %	80.4%	87.3%	79.2%	77.1%	80.6%	79.8%
	000's Rec oz Au	244.2	188.5	305.9	590.1	400.1	1,728.8
MLG*	000's Tonnes	11,526.3	9,240.0	17,859.0	33,952.8	25,042.7	97,620.8
	g Au/t	0.46	0.41	0.48	0.50	0.46	0.47
	000's oz Au	169.9	121.5	276.2	545.1	369.5	1,482.0
	Rec %	80.2%	87.3%	77.6%	74.4%	79.8%	78.0%
	000's Rec oz Au	136.3	106.0	214.3	405.4	294.7	1,156.7
LG*	000's Tonnes	8,138.8	5,962.8	14,513.6	27,330.3	21,337.4	77,282.8
	g Au/t	0.37	0.33	0.40	0.42	0.37	0.39
	000's oz Au	97.7	64.0	186.3	367.1	255.8	970.9
	Rec %	80.2%	87.2%	76.8%	72.6%	79.8%	77.0%
	000's Rec oz Au	78.4	55.8	143.0	266.5	204.2	747.9
Total Ore	000's Tonnes	70,150.3	46,849.0	83,009.5	150,575.9	104,039.1	454,623.8
	g Au/t	0.80	0.63	0.71	0.68	0.63	0.69
	000's oz Au	1,815.1	941.6	1,900.9	3,308.5	2,107.3	10,073.3
	Rec %	81.2%	87.4%	79.8%	77.7%	80.8%	80.3%
	000's Rec oz Au	1,474.7	823.2	1,516.4	2,569.7	1,702.4	8,086.4
SG*_Wst	000's Tonnes	5,148.6	2,942.8	9,534.7	17,590.0	13,914.2	49,130.4
	g Au/t	0.32	0.29	0.34	0.36	0.32	0.34
	000's oz Au	52.7	27.1	104.2	205.1	143.5	532.6
	Rec %	80.4%	86.9%	76.9%	71.8%	79.4%	76.4%
	000's Rec oz Au	42.4	23.6	80.1	147.2	113.9	407.1
W*_Wst	000's Tonnes	93,349.1	11,182.4	137,723.8	193,667.3	168,332.9	604,255.4
Total Waste	000's Tonnes	98,497.7	14,125.2	147,258.6	211,257.3	182,247.1	653,385.8
Total	000's Tonnes	168,648.0	60,974.1	230,268.0	361,833.2	286,286.2	1,108,009.6
Strip Ratio	W:O	1.40	0.30	1.77	1.40	1.75	1.44

* HG > $750 Au cutoff
* MHG $750-$1000 Au cutoff
*MLG $1000-$1250 Au cutoff
*LG $1250-$1500 Au cutoff
*SG Waste $1500-$1750 Au cutoff
W* Waste <$1750 Au cutoff
Note 1.1023 tons in 1 tonne; 31.103486 grams in 1 oz



Note that Phase 1 was expanded to obtain more waste materials for construction requirements.

Figure 16.4 Design Phase 1

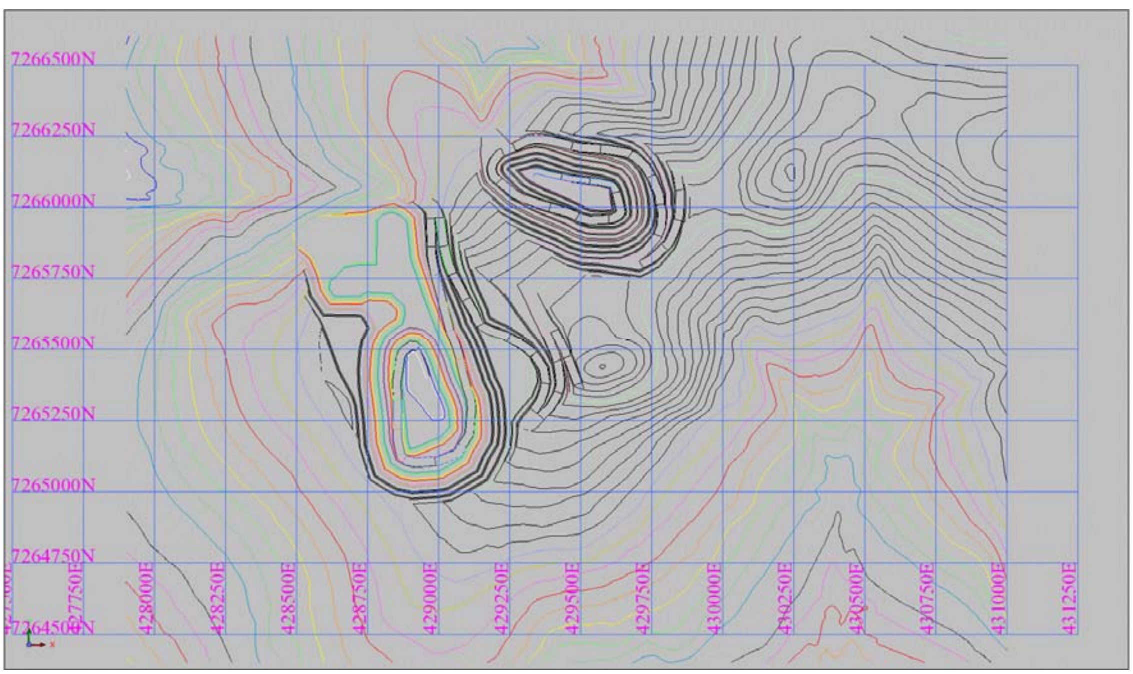


Figure 16.5 Design Phase 1 and 2

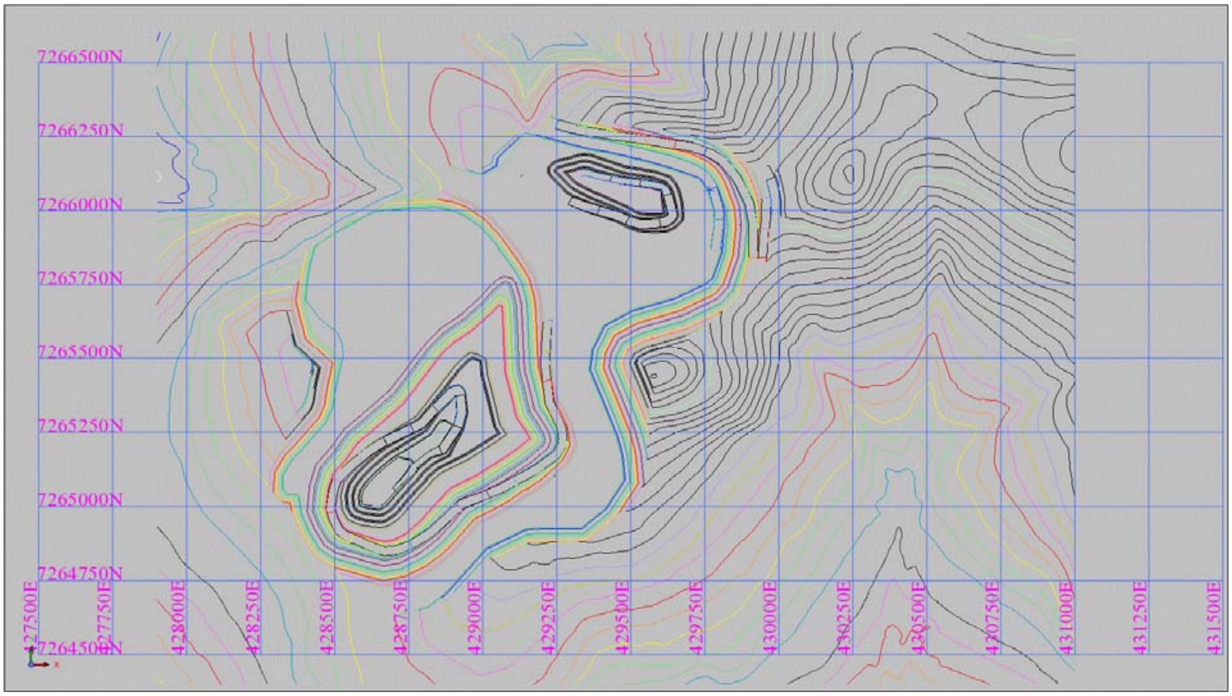


Figure 16.6 Design Phase 2 and 3

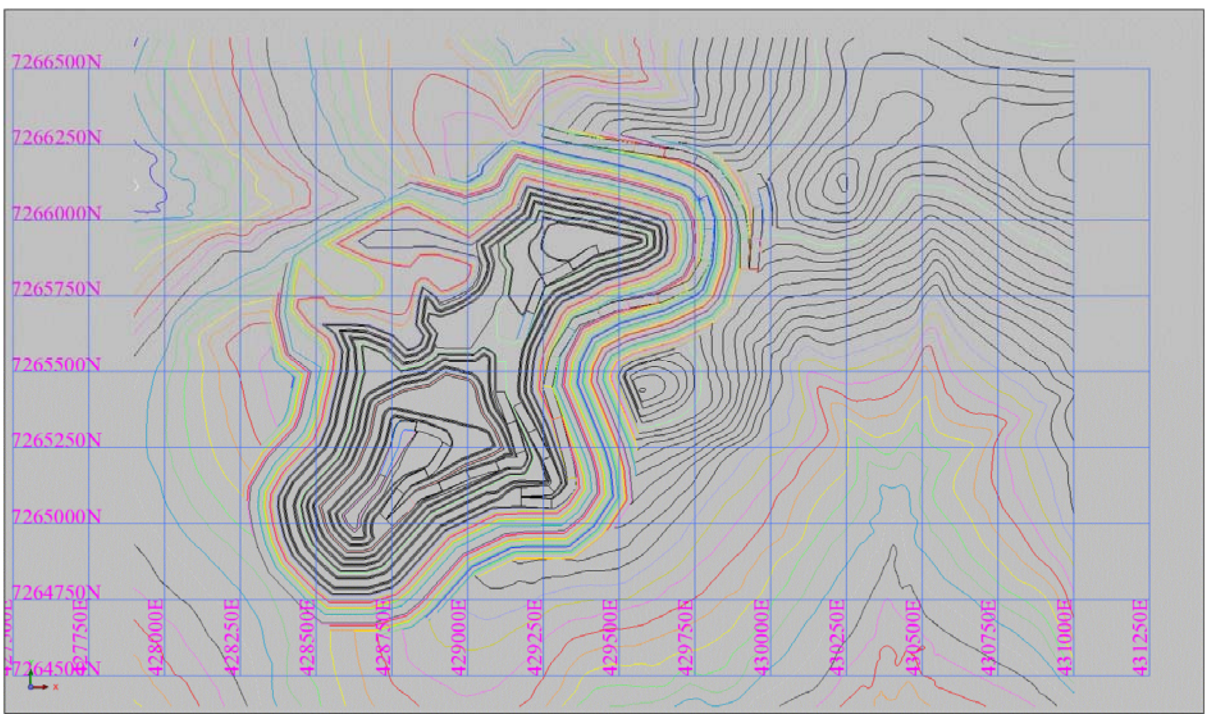


Figure 16.7 Design Phase 4

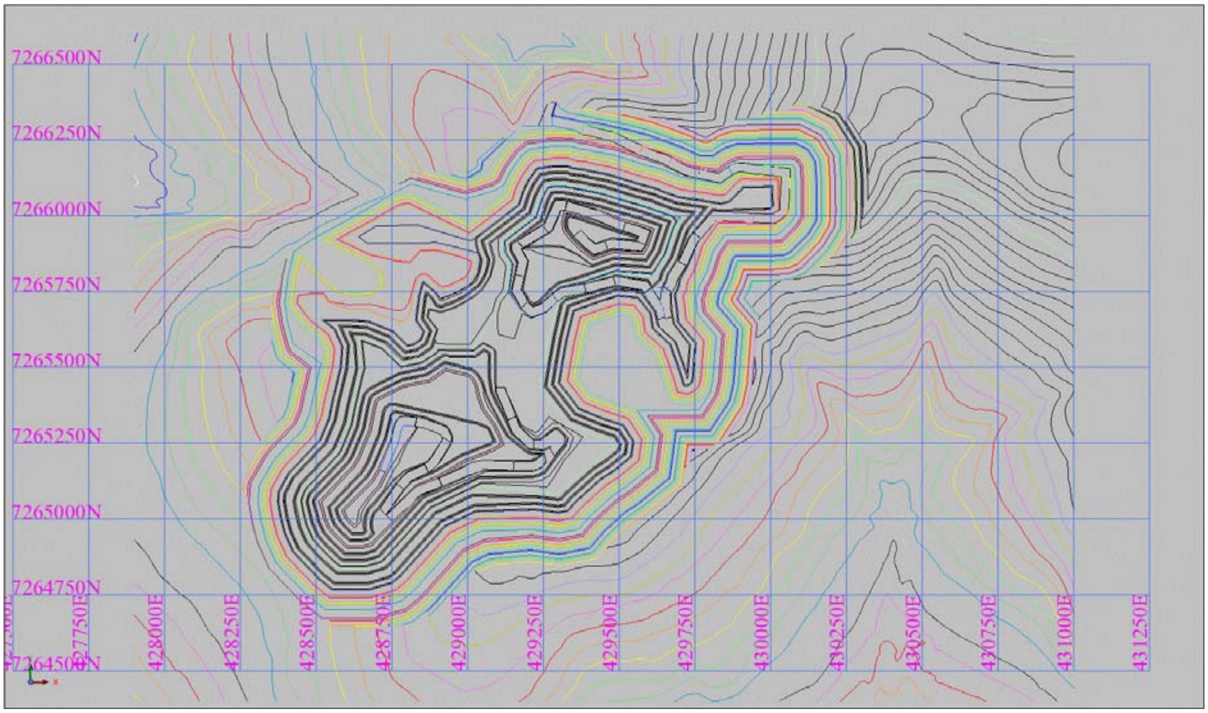


Figure 16.8 Design Final Pit (Phase 5)

16.5 Development Rock, Stockpiles, and Overburden

The mine will commence overburden removal in year -3 (month -27) with a contractor removing about 7.6 Mt (6.9 Mmt) of overburden materials from the top of Money Knob to establish a road system and areas where the large mining equipment fleet could work.

A considerable amount of construction material is required from pit overburden materials to construct:

- The crusher platform. Cut material during the crusher site excavation and mine waste will be used to construct the crusher platform.
- The Tailings Management Facility (TMF) embankment and basin. The TMF embankment and basin fill will be constructed with excess material from mine facility excavations and surface mine waste. Surface mine waste will be used during pre-production and to raise the TMF embankment during the mine life.
- Out-of-pit haul roads. Some overburden material from the surface mine is used to construct out-of-pit haul roads required for the operation.
- Gertrude Creek embankment. The embankment will be constructed with material cut from the south access road and surface mine waste.

Table 16.6 summarizes the construction material supplied from the mine.

Table 16.6 Construction Materials Schedule							
Material Units	**Tailings Embankment 000's Tonnes**	**Tailings Basin 000's**	**Gertrude Creek 000's**	**Access and Haul Roads 000's**	**Crusher Platform 000's**	**Other 000's**	**Totals Tonnes 000's**
Year -2	20,000.0		6,000.0	3,000.0	2,299.6		31,299.6
Year -1	15,676.2		8,709.3	4,788.1		11,375.8	40,549.4
Year 1	11,924.2	4,869.8		2,851.3		2,590.9	22,236.2
Year 2							
Year 3	13,480.9	3,574.3		587.0		2,287.0	19,929.2
Year 4							
Year 5	12,095.6	2,331.5		718.5		2,019.4	17,165.0
Year 6							
Year 7	12,969.2	2,119.2		616.0		1,532.3	17,236.7
Year 8							
Year 9							
Year 10	6,524.5	3,079.9		1,584.9		2,345.1	13,534.4
Totals	**92,670.7**	**15,974.8**	**14,709.3**	**14,145.7**	**2,299.6**	**22,150.5**	**161,950.6**

16.6 Haul Roads

Haul roads were designed to be 3.5 times the truck width of 28.5 ft, (8.7 m) or 100 ft (30.5 m). The maximum grade of the haul roads is 10%. There will be a considerable amount of downhill hauling.

16.7 Schedule

A production schedule was developed to mine the deposit. The schedule was based on producing 100,000 ore t (90,718 mt) per day or 36.5 Mt (33.1Mmt) annually. The first year of production was reduced to produce about 80% of normal due to start-up conditions. A quarterly schedule was developed for the pre-production period and the first two years of operation.

The schedule was developed to maximize the ore grade in the initial years by stockpiling lower grade materials. During pre-production, 27.6 Mt (25 Mmt) of ore-grade material is mined, of which 3.2 Mt (2.9 Mmt) of the higher grade material used for mill feed during year 1. During the pre-production and the first two years of production a total of about 71.2 Mt (64.6 Mmt) of ore will be stockpiled. The stockpile balance is increased to a maximum of 93.2 Mt (84.6 Mmt) at the end of year 11. The higher grade stockpiled material will be used to supplement mill feed during operations, with the lower grade material processed after mining in the pit ceases in year 12.

A pre-production period of two years and three months will be needed to strip mine overburden and to complete the construction of facilities requiring mine fill overburden material prior to mill start-up. A swell factor of 40% was used in the calculations to determine the amount of fill material. The loose density was 1.9 tonnes per cubic meter. The production schedule is shown in Table 16.7 and 16.8, while stockpile movement is shown in Table 16.9. Table 16.10 shows the bottom bench mined by phase and year.

Table 16.7
Livengood Production Schedule - Material Mined

Material	Item	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Totals
HG*	000's Tonnes	3,122.2	6,115.9	27,108.4	18,941.9	7,155.8	14,763.6	12,822.3	9,828.4	11,906.5	11,847.5	9,709.2	9,994.1	19,272.4	4,181.4	166,769.8
	g Au/t	0.93	1.08	1.09	0.97	1.04	1.02	1.06	0.95	0.96	0.95	1.02	1.07	1.00	1.07	1.02
	000's oz Au	93.3	211.8	949.1	590.2	238.6	485.7	435.0	300.2	366.2	361.0	317.7	344.1	616.9	143.9	5,453.6
	Rec %	82.5%	80.6%	82.7%	85.2%	83.4%	82.9%	78.3%	84.4%	84.5%	83.9%	77.4%	74.0%	82.5%	69.3%	81.7%
	000's Rec oz Au	77.0	170.8	785.1	502.9	199.1	402.7	340.5	253.3	309.6	302.7	246.0	254.6	509.1	99.7	4,453.1
MHG*	000's Tonnes	2,434.9	4,153.6	12,606.2	11,320.4	5,313.5	9,237.9	10,529.1	7,654.5	8,106.5	7,409.9	8,429.9	9,625.1	13,406.6	2,722.2	112,950.4
	g Au/t	0.57	0.59	0.58	0.56	0.57	0.59	0.62	0.58	0.57	0.59	0.64	0.66	0.59	0.70	0.60
	000's oz Au	44.6	78.6	233.9	203.4	98.1	174.7	208.9	142.3	149.6	141.2	172.2	203.8	253.8	61.7	2,166.7
	Rec %	81.2%	80.4%	83.0%	84.1%	81.7%	81.6%	77.8%	82.5%	83.1%	80.9%	75.0%	72.6%	80.9%	66.1%	79.8%
	000's Rec oz Au	36.2	63.2	194.0	171.1	80.1	142.6	162.5	117.3	124.4	114.2	129.1	148.0	205.3	40.8	1,728.8
MLG*	000's Tonnes	2,157.8	3,132.8	9,190.4	8,377.1	5,391.2	7,939.2	9,705.0	6,508.8	6,579.0	6,662.6	8,906.3	10,056.7	10,662.7	2,351.0	97,620.8
	g Au/t	0.44	0.45	0.44	0.44	0.45	0.46	0.50	0.46	0.45	0.46	0.49	0.52	0.48	0.56	0.47
	000's oz Au	30.7	45.5	130.4	118.1	78.1	117.5	155.1	96.6	94.5	98.6	141.6	168.7	163.9	42.7	1,482.0
	Rec %	81.3%	80.8%	83.2%	83.1%	81.1%	80.9%	75.0%	80.4%	82.1%	80.2%	74.9%	70.8%	77.1%	63.9%	78.0%
	000's Rec oz Au	25.0	36.8	108.5	98.2	63.3	95.0	116.3	77.7	77.6	79.1	106.0	119.5	126.3	27.3	1,156.7
LG*	000's Tonnes	1,549.7	2,334.0	6,025.2	5,971.1	4,449.5	6,105.0	7,709.7	5,784.8	6,049.9	5,029.7	7,845.8	8,849.6	7,790.4	1,788.2	77,282.8
	g Au/t	0.36	0.37	0.36	0.36	0.37	0.38	0.42	0.38	0.36	0.38	0.40	0.43	0.40	0.47	0.39
	000's oz Au	18.1	27.9	69.7	69.5	53.2	74.0	103.7	71.6	70.9	61.2	102.1	122.2	100.1	26.9	970.9
	Rec %	81.2%	80.6%	83.0%	82.8%	80.8%	80.6%	73.4%	79.3%	81.6%	79.1%	74.3%	70.2%	75.1%	63.1%	77.0%
	000's Rec oz Au	14.7	22.4	57.8	57.5	43.0	59.6	76.1	56.8	57.9	48.4	75.8	85.8	75.1	17.0	747.9
Total Ore	000's Tonnes	9,264.7	15,736.4	54,930.2	44,610.6	22,310.1	38,045.7	40,766.1	29,776.4	32,641.9	30,949.7	34,891.3	38,525.6	51,132.1	11,042.9	454,623.8
	g Au/t	0.63	0.72	0.78	0.68	0.65	0.70	0.69	0.64	0.65	0.67	0.65	0.68	0.69	0.78	0.69
	000's oz Au	186.6	363.8	1,383.1	981.2	467.9	851.8	902.8	610.7	681.2	662.0	733.7	838.7	1,134.6	275.2	10,073.3
	Rec %	81.9%	80.6%	82.8%	84.6%	82.4%	82.2%	77.0%	82.7%	83.6%	82.2%	75.9%	72.5%	80.7%	67.1%	80.3%
	000's Rec oz Au	152.8	293.2	1,145.5	829.7	385.5	699.9	695.4	505.1	569.4	544.4	557.0	607.9	915.8	184.7	8,086.4
SG*_Wst	000's Tonnes	1,030.9	1,665.9	3,373.2	3,189.8	3,188.7	3,842.2	4,774.4	3,557.0	3,922.1	3,528.4	5,407.3	5,847.1	4,664.7	1,138.8	49,130.4
	g Au/t	0.31	0.32	0.31	0.32	0.31	0.32	0.36	0.33	0.31	0.33	0.35	0.37	0.35	0.40	0.34
	000's oz Au	10.3	17.1	33.5	32.3	32.2	39.8	55.4	38.0	39.4	37.1	61.3	69.0	52.5	14.8	532.6
	Rec %	81.4%	80.5%	82.6%	82.1%	81.2%	80.6%	73.1%	79.3%	81.5%	78.5%	73.2%	70.2%	72.8%	62.7%	76.4%
	000's Rec oz Au	8.3	13.8	27.7	26.5	26.1	32.0	40.5	30.1	32.1	29.1	44.9	48.4	38.3	9.3	407.1
W*_Wst	000's Tonnes	30,486.4	43,266.8	22,687.6	42,469.5	64,516.6	48,127.1	44,474.6	56,936.4	53,451.1	55,536.9	49,716.5	45,897.5	34,218.1	12,470.5	604,255.4
Total Waste	000's Tonnes	31,517.3	44,932.7	26,060.8	45,659.2	67,705.3	51,969.3	49,249.0	60,493.4	57,373.2	59,065.3	55,123.8	51,744.6	38,882.8	13,609.3	653,385.8
Total	000's Tonnes	40,782.0	52,949.9	88,710.2	90,269.8	90,015.4	90,015.0	90,015.1	90,269.8	90,015.1	90,015.1	90,015.1	90,270.2	90,014.9	24,652.1	1,108,009.6
Strip Ratio	W:O	3.40	2.86	0.47	1.02	3.03	1.37	1.21	2.03	1.76	1.91	1.58	1.34	0.76	1.23	1.44

* HG > $750 Au cutoff
* MHG $750-$1000 Au cutoff
*MLG $1000-$1250 Au cutoff
*LG $1250-$1500 Au cutoff
*SG Waste $1500-$1750 Au cutoff
W* Waste <$1750 Au cutoff
Note 1.1023 tons in 1 tonne; 31.103486 grams in 1 oz

Note that most of the ore to the process plant from years 12-14 is from stockpile ore.

Table 16.8
Livengood Production Schedule – Material Processed by Rock Type

Rocktype	Item	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Totals
Rocktype 4	000's Tonnes	-	-	6,066.5	5,616.1	11,608.6	7,760.4	3,538.2	5,613.7	3,646.8	2,834.3	2,183.1	1,762.3	1,574.0	2,241.5	4,755.7	4,176.0	63,377.1
	g Au/t	-	-	1.00	0.89	0.71	0.68	0.49	0.61	0.55	0.51	0.57	0.71	0.99	0.34	0.33	0.33	0.65
	000's oz Au	-	-	195.0	160.4	265.1	168.9	56.3	109.2	64.6	46.5	40.1	40.4	50.3	24.2	50.0	43.9	1,314.8
	Rec %	0.0%	0.0%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%	84.2%
	000's Rec oz Au	-	-	164.2	135.0	223.2	142.2	47.4	92.0	54.4	39.1	33.8	34.0	42.3	20.4	42.1	37.0	1,107.1
Rocktype 5	000's Tonnes	-	-	3,926.0	16,053.7	8,636.6	5,682.6	6,460.7	11,063.7	9,934.3	9,871.8	9,139.0	7,032.1	11,798.7	10,215.6	9,733.5	8,547.2	128,095.5
	g Au/t	-	-	0.86	0.86	0.52	0.56	0.52	0.57	0.57	0.59	0.57	0.54	0.69	0.44	0.33	0.33	0.58
	000's oz Au	-	-	108.6	444.5	145.1	101.7	108.1	202.8	181.8	186.4	166.1	122.4	262.3	145.7	103.2	90.6	2,369.5
	Rec %	0.0%	0.0%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%	87.7%
	000's Rec oz Au	-	-	95.2	389.9	127.2	89.2	94.8	177.9	159.4	163.5	145.6	107.4	230.1	127.8	90.5	79.5	2,078.0
Rocktype 6	000's Tonnes	-	-	6,519.4	6,450.8	7,036.9	7,252.2	6,798.3	6,700.6	4,488.7	4,124.3	5,806.4	7,517.2	2,567.1	4,316.8	7,501.2	6,587.0	83,666.8
	g Au/t	-	-	1.25	1.04	0.73	0.82	0.79	0.65	0.64	0.66	0.68	0.71	0.92	0.46	0.43	0.43	0.73
	000's oz Au	-	-	261.8	216.4	164.5	190.8	172.8	140.7	92.9	87.0	126.5	172.1	76.2	63.9	104.5	91.8	1,961.9
	Rec %	0.0%	0.0%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%
	000's Rec oz Au	-	-	200.8	166.0	126.1	146.3	132.5	107.9	71.3	66.7	97.1	132.0	58.5	49.0	80.2	70.4	1,504.8
Rocktype 7	000's Tonnes	-	-	33.0	534.5	307.3	591.5	4,520.3	1,598.2	1,504.3	2,441.4	7,140.5	11,000.2	2,928.8	10,620.0	6,285.7	5,519.6	55,025.3
	g Au/t	-	-	1.03	1.28	0.79	0.88	1.01	0.66	0.69	0.76	0.85	0.85	1.04	0.75	0.50	0.50	0.77
	000's oz Au	-	-	1.1	22.0	7.8	16.8	146.5	33.9	33.5	60.0	195.4	299.1	97.7	255.5	101.6	89.2	1,360.0
	Rec %	0.0%	0.0%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%	58.5%
	000's Rec oz Au	-	-	0.6	12.9	4.5	9.8	85.7	19.8	19.6	35.1	114.3	174.9	57.2	149.5	59.4	52.2	795.6
Rocktype 8	000's Tonnes	-	-	70.2	276.2	267.3	301.8	906.1	1,610.7	866.5	846.8	1,100.1	517.0	399.4	802.1	566.8	497.7	9,028.6
	g Au/t	-	-	0.74	0.94	0.58	0.62	0.77	0.66	0.67	0.69	0.73	0.68	0.76	0.68	0.37	0.37	0.66
	000's oz Au	-	-	1.7	8.4	5.0	6.0	22.6	33.9	18.6	18.9	25.9	11.3	9.8	17.6	6.7	5.9	192.1
	Rec %	0.0%	0.0%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%
	000's Rec oz Au	-	-	1.4	7.1	4.2	5.1	19.1	28.8	15.8	16.0	22.0	9.6	8.3	14.9	5.7	5.0	162.9
Rocktype 9	000's Tonnes	-	-	9,988.8	4,374.6	5,358.3	11,626.6	10,991.5	6,719.0	12,774.5	13,096.4	7,845.9	5,477.3	13,947.1	5,019.0	4,372.1	3,839.2	115,430.3
	g Au/t	-	-	1.10	1.12	0.74	0.88	0.80	0.72	0.77	0.73	0.66	0.70	0.85	0.44	0.37	0.37	0.77
	000's oz Au	-	-	354.8	157.2	128.2	328.7	283.2	155.5	318.2	308.7	166.7	123.9	382.6	70.9	51.4	45.1	2,875.1
	Rec %	0.0%	0.0%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%
	000's Rec oz Au	-	-	300.9	133.3	108.7	278.7	240.1	131.9	269.8	261.8	141.4	105.0	324.5	60.1	43.6	38.3	2,438.1
Total	**000's Tonnes**	**-**	**-**	**26,603.9**	**33,306.0**	**33,215.0**	**33,215.0**	**33,215.0**	**33,306.0**	**33,215.0**	**33,215.0**	**33,215.0**	**33,306.0**	**33,214.9**	**33,215.0**	**33,215.0**	**29,166.7**	**454,623.6**
	g Au/mt	**-**	**-**	**1.08**	**0.94**	**0.67**	**0.76**	**0.74**	**0.63**	**0.66**	**0.66**	**0.67**	**0.72**	**0.82**	**0.54**	**0.39**	**0.39**	**0.69**
	000's oz Au	**-**	**-**	**923.0**	**1,008.9**	**715.5**	**812.8**	**789.5**	**676.1**	**709.5**	**707.5**	**720.8**	**769.1**	**878.9**	**577.7**	**417.3**	**366.5**	**10,073.3**
	Rec %	**0.0%**	**0.0%**	**82.7%**	**83.7%**	**83.0%**	**82.6%**	**78.5%**	**82.6%**	**83.2%**	**82.3%**	**76.9%**	**73.2%**	**82.0%**	**73.0%**	**77.0%**	**77.0%**	**80.3%**
	000's Rec oz Au	**-**	**-**	**763.2**	**844.2**	**594.0**	**671.3**	**619.7**	**558.3**	**590.3**	**582.3**	**554.2**	**562.9**	**720.7**	**421.6**	**321.4**	**282.2**	**8,086.4**

*HG > $750 Au cutoff
*MHG $750-$1000 Au cutoff
*MLG $1000-$1250 Au cutoff
*LG $1250-$1500 Au cutoff
*SG Waste $1500-$1750 Au cutoff
W* Waste <$1750 Au cutoff
Note 1.1023 tons in 1 tonne; 31.103486 grams in 1 oz

Note that most of the ore to the process plant from years 12-14 is from stockpile ore.

Table 16.9
Livengood Production Schedule – Stockpile Movement (000's Tonnes)

Item	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Totals
Material Stockpiled	9,264.7	15,736.4	28,326.3	21,047.3	9,840.7	9,978.9	7,709.7	5,784.8	6,049.9	5,029.7	7,845.8	7,782.2	17,917.2	0.0	0.0	0.0	152,313.7
Material Reclaimed	0.0	0.0	0.0	9,742.7	20,745.7	5,148.2	158.6	9,314.3	6,623.1	7,295.0	6,169.5	2,562.7	0.0	22,172.1	33,215.0	29,166.7	152,313.6
Stockpile Balance	9,264.7	25,001.0	53,327.4	64,632.0	53,727.1	58,557.8	66,108.9	62,579.3	62,006.2	59,740.9	61,417.2	66,636.7	84,553.9	62,381.7	29,166.7	0.0	

Table 16.10 Phase Bench Elevation Schedule (m)					
Year	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5
Start	520	510	530	510	550
-2	410				
-1	350				
1	260	440			
2	210	290	370		
3			300		
4			240	420	
5			130	360	
6				280	510
7				260	430
8				230	390
9				190	350
10				120	310
11				60	210
12					90

16.8 Mine Equipment

Mine equipment requirements were developed using detailed equipment calculations for drills, shovels, loaders, and haul trucks.

16.8.1 Drilling Equipment

The primary production drill is a 59,500 lb. (27,000 kg) pull-down rotary blasthole drill capable of drilling 10 in (251 mm) blastholes on 33 ft (10 m) benches with 7.2 ft (2.2 m) of sub-grade. These drill holes are completed on a 26.2 ft x 26.2 ft pattern blasting about 1,900 t (2.72 SG) per blasthole. These drills are projected to complete 687 ft (209.4 m) of drilling per shift, or 17.2 holes per shift. Six of these drills will be needed to complete the required annual drilling demands. Two 6.75 in (171 mm) rotary blasthole drills are included in the mine equipment for back up and trim blasting. A summary of the drilling requirements is shown in Table 16.11.

Table 16.11 Drill Requirements						
	9 7/8 inch Drill			6 3/4 inch Drill		
Year	Annual Drill Hours	Drills Required	Rounded Drills	Annual Drill Hours	Drills Required	Rounded Drills
-2	11,213	2.1	3	1,957	0.4	1
-1	20,098	3.8	4	4,327	0.8	1
1	26,231	4.9	5	7,026	1.3	2
2	29,530	5.6	6	7,840	1.5	2
3	29,839	5.6	6	6,817	1.3	2
4	29,560	5.6	6	9,253	1.7	2
5	29,512	5.6	6	9,376	1.8	2
6	29,792	5.6	6	8,889	1.7	2
7	29,656	5.6	6	9,008	1.7	2
8	29,686	5.6	6	8,931	1.7	2
9	29,616	5.6	6	9,110	1.7	2
10	29,638	5.6	6	8,425	1.6	2
11	29,329	5.5	6	8,985	1.7	2
12	8,085	2.0	5	3,341	0.6	1
Total Hrs	**361,785**			**103,283**		

16.8.2 Blasting

A powder factor of 0.227 lbs/t, (0.104 kg/mt) was used to calculate explosives requirements. Most of the drill holes are expected to be dry, and if water is encountered in the blastholes, it is expected that most of the holes can be pumped dry so ANFO can be used. MDA assumed that 85% of the holes will use ANFO, while 15% of the holes will use an ANFO/emulsion mix designed for the wet holes. Silos will be constructed on site to house ANFO and emulsion. The explosives will be delivered to the hole in owner operated ANFO trucks. Table 16.12 shows the annual explosives requirements.

Table 16.12
Explosives Requirements (Tonnes)

Year	70/30 Blend Emulsion	ANFO	Totals	000's Liters Fuel Oil
-2	847	7,622	8,468	709.9
-1	1,517	13,651	15,167	1,271.5
1	2,025	18,223	20,248	1,697.4
2	2,257	20,311	22,567	1,891.9
3	2,250	20,253	22,504	1,886.5
4	2,250	20,253	22,504	1,886.5
5	2,250	20,253	22,504	1,886.5
6	2,257	20,311	22,567	1,891.9
7	2,250	20,253	22,504	1,886.5
8	2,250	20,253	22,504	1,886.5
9	2,250	20,253	22,504	1,886.5
10	2,257	20,311	22,568	1,891.9
11	2,250	20,253	22,504	1,886.5
12	616	5,547	6,163	516.7
Totals	**27,528**	**247,748**	**275,275**	**23,077.0**

16.8.3 Loading Equipment

Front shovels of 47 y^3 (36 m^3) capacity or 40 y^3 (30.6 m^3) front end loaders will load the 320 t (290 mt) trucks. The front end loader will be back-up for the three front shovels that are required. The front shovel is expected to load the truck in five passes, while the front end loader is expected to load the trucks in six passes. At the end of the mine life, the shovels are moved to the low-grade stockpile to feed the process plant. Table 16.13 shows the estimated shovel and loader hours required.

Table 16.13 Shovel and Loader Requirements						
	47 yd³ Shovel			40 yd³ Loader		
Year	Annual Shovel Hours	Shovels Required	Rounded Shovels	Annual Loader Hours	Loaders Required	Rounded Loaders
-2	6,501	1.1	2	898	0.2	0
-1	11,664	2.0	2	1,578	0.3	1
1	14,764	2.5	3	3,328	0.6	1
2	16,851	2.8	3	6,215	1.0	2
3	17,331	2.9	3	8,914	1.5	2
4	16,956	2.9	3	4,511	0.8	1
5	16,891	2.8	3	3,019	0.5	1
6	17,204	2.9	3	5,545	0.9	1
7	17,085	2.9	3	4,786	0.8	1
8	17,125	2.9	3	4,939	0.8	1
9	17,031	2.9	3	4,723	0.8	1
10	16,995	2.9	3	3,709	0.6	1
11	16,645	2.8	3	3,341	0.6	1
12	9,184	1.5	3	809	0.1	1
13	6,824	1.2	3	0	0.0	0
14	5,992	1.0	2	0	0.0	0
Total Hrs	**225,043**			**56,314**		

16.8.4 Hauling

All hauling is completed using 320 t (290 mt) trucks, except for the initial contractor pre-production to ready the site for large mining equipment. Haul distances and cycle times were estimated for all material destinations in all pit phases. Truck hours were calculated and are shown in Table 16.14.

Table 16.14 Truck Requirements									
Year	**Annual Truck Hrs**	**Trucks Required**	**Rounded Trucks**	**Cycle (Min) Ore Shovel**	**Cycle (Min) Ore Loader**	**Cycle (Min) Stockpile Shovel**	**Cycle (Min) Stockpile Loader**	**Cycle (Min) Waste Shovel**	**Cycle (Min) Waste Loader**
-2	58,363	9.0	9	27.6	29.7	19.0	20.6	31.6	33.7
-1	104,429	16.1	17	26.3	28.3	19.0	20.6	32.0	34.1
1	110,164	17.0	18	23.4	25.5	19.0	20.6	25.0	27.1
2	129,105	19.9	20	21.0	23.1	19.0	20.6	24.8	26.9
3	155,702	24.0	24	21.7	23.8	19.0	20.6	27.0	29.1
4	138,429	21.4	23	22.8	24.9	19.0	20.6	28.3	30.3
5	129,009	19.9	20	24.2	26.2	19.0	20.6	26.2	28.3
6	134,875	20.8	21	21.3	23.3	19.0	20.6	25.7	27.8
7	149,978	23.1	24	23.7	25.8	19.0	20.6	30.3	32.4
8	144,669	22.3	23	24.8	26.9	19.0	20.6	27.7	29.8
9	147,124	22.7	23	27.2	29.3	19.0	20.6	27.6	29.7
10	159,615	24.6	25	29.3	31.4	19.0	20.6	31.7	33.8
11	155,686	24.0	25	29.5	31.6	19.0	20.6	31.9	34.0
12	67,536	10.4	11	33.0	35.1	19.0	20.6	30.3	32.4
13	35,312	5.4	6			19.0			
14	31,008	4.8	5			19.0			
Total Hrs	**1,851,003**								

16.8.5 Support Equipment

Table 16.15 shows a summary of the annual equipment requirements for the mine. Support equipment will be needed to keep all haul roads watered and graded, shovel and loading sites level and clean, and drilling sites level and clean. Support equipment will also aid in the construction of the TMF, Gertrude embankment, roads, etc. A portable crushing plant is included to provide road base materials.

Table 16.15 Mine Equipment Summary																
Item	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Rotary Drill - 171 mm	1	1	2	2	2	2	2	2	2	2	2	2	2	1	0	0
Rotary Drill - 251 mm	3	4	5	6	6	6	6	6	6	6	6	6	6	2	0	0
30.6 cm Loader	1	1	1	2	2	1	1	1	1	1	1	1	1	1	1	1
Spare 30.6 cm Bucket	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
36 cm Front Shovel	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Spare 36 m^3 Bucket	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
15-17 m^3 Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Table 16.15
Mine Equipment Summary

Spare 15-17 m^3 Bucket	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
1.75 m^3 Front End Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Spare 1.75 m^3 Bucket	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
320 Tonne Truck	9	17	17	20	24	22	20	21	24	23	23	25	25	11	6	6
320 Tonne Truck Bed	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2
35 unit dispatch	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Handler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
20,000 gal H2O Truck			1	1	1	1	1	1	1	1	1	1	1	1	1	1
13,000 gal H2O Truck	1		2	2	2	2	2	2	2	2	2	2	2	2	2	2
Dozer 500-600 HP	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Dozer 800-900 HP	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Rubber Tire Dozer 800-1000 HP	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader 16M	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader 24M		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
ANFO Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Portable Crushing Plant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
4.1 cm Mass Excavator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Plant	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Stemming/Sanding Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Low Boy & Prime Mover			1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube and Fuel Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lube and Fuel Truck - Backup	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics Truck	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Welding Truck/Crane	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
30 T Crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
50 T Hydraulic Crane	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
100 T Crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Radios and Base Station	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
WiFi Communications	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
GPS & Technology	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
200 HP Integrated Tool Carrier		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Flatbed Truck with Crane	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Crew Vans	3	3	6	6	6	6	6	6	6	6	6	6	6	6	6	6

16.9 Manpower

The manpower requirements were estimated based on the equipment requirements. Maintenance personnel were estimated based on ratios of mine personnel to maintenance personnel from existing operations of similar size. Normal training of personnel is included in the estimate. However, startup training is included elsewhere under owner's costs. Table 16.16 shows the estimated annual mine personnel.

Table 16.16
Mine Personnel

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
MINE OPERATIONS																
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pit Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1			
General Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1			
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Trainer	2	2	2	2	1	1	1	1	1	1	1	1	1			
Load and Haul Foreman	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Load and Haul Operator	33	57	60	66	84	72	66	69	78	75	75	81	81	36	24	21
Drill and Blasting Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1			
Mine Foreman	6	6	6	6	6	6	6	6	6	6	6	6	6	1	1	1
Blasting Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
Blastman	2	2	2	2	2	2	2	2	2	2	2	2	2	2		
Blasting Helper	3	3	5	5	5	5	5	5	5	5	5	5	5	4		
Driller	9	12	18	18	18	18	18	18	18	18	18	18	18	3	0	0
Support Equipment Operators	19	19	22	22	22	22	22	22	22	22	22	22	22	13	13	13
Trainee	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	1
Subtotal Mine Operations	86	113	127	133	150	138	132	135	144	141	141	147	147	66	44	41
MINE MAINTENANCE																
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shop Shift Foreman	6	6	6	6	6	6	6	6	6	6	6	6	6	3	3	3
Planning Engineer	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Mechanic	21	30	34	36	41	37	36	37	39	38	38	40	40	18	13	12
Electrician	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Welder	3	3	4	5	5	5	5	5	5	5	5	5	5	4	2	2
Servicemen	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3

Table 16.16
Mine Personnel

Light Vehicle Mechanic	6	6	6	6	6	5	5	5	5	5	5	5	5	3	2	2
Tireman	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Mechanic Trainee	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Subtotal Mine Maintenance	47	56	61	64	69	64	63	64	66	65	65	67	67	36	28	27
Total Mine Operations	**133**	**169**	**188**	**197**	**219**	**202**	**195**	**199**	**210**	**206**	**206**	**214**	**214**	**102**	**72**	**68**
MINE ENGINEERING																
Chief Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Sr Mining Engineer	3	3	3	3	3	3	3	3	3	3	3	3	2	1		
Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor Assistant	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal Engineering	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**7**	**6**	**5**	
GEOLOGY AND GRADE CONTROL																
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1			
Ore Control Geologist	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Sampler	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	1
Subtotal Geology and Grade Control	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**3**	**3**	**3**
TOTAL MINE STAFF	**148**	**184**	**203**	**212**	**234**	**217**	**210**	**214**	**225**	**221**	**221**	**229**	**228**	**111**	**80**	**71**

16.10 Production

The mine is planned to produce ore and overburden during two 12-hour shifts per day, seven days per week. There will be two working crews onsite and one crew offsite to achieve this schedule.

Normal ore production is 100,000 t (90,718 mt) of ore per day or 36.5 Mt (33.1 Mmt) per year. Table 16.17 shows the ore and overburden daily production rates for each year. When ore is produced at a rate greater than 100,000 t/d, excess low-grade material is stockpiled to increase the grade of material processed.

Table 16.17
Daily Mine Production – Annual Averages (Tonnes per day)

Year	Pit Ore	Pit Waste	Stockpile	Total Material
-2	25,383	67,421		92,804
-1	43,113	123,103		166,217
1	150,494	71,399		221,893
2	122,221	125,094	26,692	274,007
3	61,124	185,494	56,837	303,455
4	104,235	142,382	14,105	260,721
5	111,688	134,929	435	247,051
6	81,579	165,735	25,519	272,833
7	89,430	157,187	18,145	264,762
8	84,794	161,823	19,986	266,603
9	95,593	151,024	16,903	263,520
10	105,550	141,766	7,021	254,336
11	140,088	106,528	0	246,616
12	90,515	111,551	60,746	262,812
13			91,000	91,000
14			79,909	79,909

17.0 Recovery Methods

17.1 Conceptualized Block Flow Diagram

The following process block flow diagram, Figure 17.1, describes the optimal process flow from the ore delivery to the crusher through to doré production and tailings management

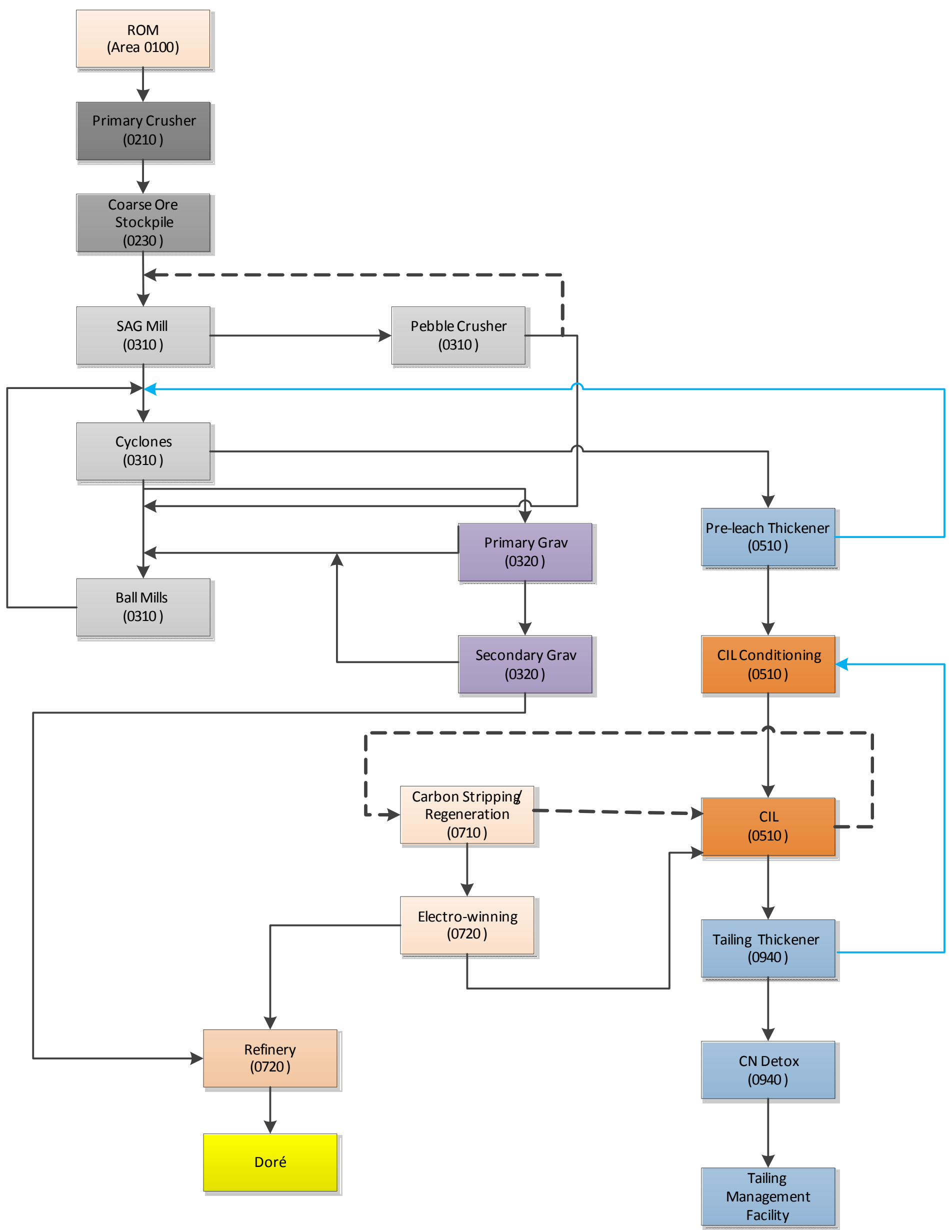


Figure 17.1 Process Block Flow Diagram

17.2 Plant Operating Design Parameters

The plant operating design parameters are presented in Table 17.1

Table 17.1
Plant Operating Design Parameters

Criterion	Imperial				Metric			
	Units	Value			Units	Value		
		Min	Nominal	Max		Min	Nominal	Max
Production Summary								
Operating Schedule:								
Mine Life	y	14	14	14	y	14	14	14
Operating Days/year	d	365	365	365	d	365	365	365
ROM Ore Haulage	t/d	100,000	100,000	100,000	mt/d	90,718	90,718	90,718
Primary Crushing								
Crushing Availability	%	80%	80%	80%	%	80%	80%	80%
Operating days	d/y	365	365	365	d/y	365	365	365
Shifts	Shift/day	2	2	2	Shift/day	2	2	2
Hours Per Shift	h	12	12	12	h	12	12	12
Available Hours/day	h/d	19	19.2	19	h/d	19	19	19
Available Hours /year	h/y	7,008	7,008	7,008	h/y	7,008	7,008	7,008
Mill								
Mill Schedule: Shifts /day	lb/d	2	2	2	lb/d	2	2	2
Hours Per Shift	h	12	12	12	h	12	12	12
Hours Per Year	h	8,760	8,760	8,760	h	8,760	8,760	8,760
Mill utilization	%	92%	92%	92%	%	92%	92%	92%
Mill operating hours / year	h	8,059	8,059	8,059	h	8,059	8,059	8,059
General Design Factor			1.20				1.20	
Production Rates								
Life of Mine	t ('000)	511,000	511,000	511,000	mt ('000)	463,572	463,572	463,572
Annual	t/y ('000)	36,500	36,500	36,500,000	mt/y ('000)	33,112	33,112	33,112
Daily	t/d	100,000	100,000	100,000	mt/d	90,718	90,718	90,718
Crusher Hourly Rate	**t/h**	**5,208**	**5,208**	**6,250**	**mt/h**	**4,725**	**4,725**	**5,670**
Mill hourly rate	**t/h**	**4,529**	**4,529**	**5,435**	**mt/h**	**4,109**	**4,109**	**4,930**

17.3 Process Plant

The Livengood process facilities will consist of a comminution circuit (one SAG and two ball mill circuits) followed by a gravity concentration circuit. The tailings from gravity concentration circuit will be fed to a carbon-in-leach (CIL) circuit. Gold will be recovered by an adsorption-desorption-recovery (ADR) circuit where the final product will be doré. Process tailings will be thickened, treated to detoxify cyanide, and discharged to the Tailings Management Facility (TMF). The gravity gold will be directly smelted from the gravity concentrate.

ROM will be dumped into the feed hopper of the primary gyratory crusher. The crushed ore will be conveyed to the coarse ore stockpile. The stockpile will be continuously reclaimed using three reclaim apron feeders and chutes with one additional apron feeder and chute installed and used as standby. Coarse ore and dust from these apron feeders and chutes will report to the semi-autogenous (SAG) grinding mill feed conveyor.

The SAG mill feed conveyor feeds the SAG mill. SAG mill discharge trommel screen undersize will flow by gravity to the cyclone feed sump, and the oversize conveyed to the pebble crusher circuit.

SAG mill screen undersize will also report to the cyclone feed sump. Slurry will be pumped to the two parallel hydrocyclone clusters. The hydrocyclone overflow will feed the downstream CIL circuit. The cyclone underflow slurry will be split between the gravity concentration circuit and the ball mill feed chute. Pebble lime will be added continuously at the ball mills to maintain ball mill discharge pH above 9.0 to promote the sodium cyanide leaching downstream and limit conditioning required prior to CIL.

The Livengood ore contains significant amounts of free gold which responds well to gravity concentration. A portion of the ground ore from the hydrocylone underflow slurry will be fed to the gravity concentrator screens. Screen oversize will discharge to the gravity circuit overflow pump boxes to be returned to the respective ball mill feed chute. The undersize is fed to the centrifugal gravity concentrators. Centrifugal concentrator tailings are returned to the ball mill feed chute.

Gold concentrate from the centrifugal concentrators will flow by gravity to the gravity tables to further concentrate the gold. The concentrate will discharge to the finishing gravity tables. The final concentrate from the finishing tables will be manually transferred to a final gravity table and concentrate calcining oven for downstream conversion to gold doré. Tailings from the gravity tables and gravity finishing table will flow to the gravity circuit overflow pump box for return to the cyclone feed sump.

Hydrocyclone overflow will be pumped to pre-leach thickening. These two high rate thickeners will thicken the slurry to 57% by weight in the thickener underflow streams. Thickener overflow from both thickeners will report to a common overflow tank. The underflow from thickener No. 1 will feed CIL circuits 1 and 2. The underflow from thickener No. 2 will feed CIL circuits 3 and 4.

The four CIL circuits each contain twelve CIL tanks. The slurry will flow from tank 1 through to tank 12 counter-currently to the carbon. Fresh carbon will be added to tank 12 and flow to tank 1, by way of the carbon advance pumps located in each CIL tank. Slurry will exit tank 12 over the carbon safety screens before heading to the tailings thickeners. Loaded carbon exiting tank 1 will report to the carbon stripping system for recovery of the adsorbed metals.

Loaded carbon from the CIL tanks reports to the loaded carbon stripping where gold will be stripped and the carbon reactivated for recycle to the four CIL circuits. The carbon is stripped using the Anglo American Research Laboratories (AARL) procedure. The stripping cycle will be two stages in which copper is stripped first followed by gold. The stripped copper is converted to copper sulfate for use in the cyanide detoxification circuit downstream.

The stripped carbon will flow to the carbon regeneration kiln. The regenerated carbon will be combined with fresh carbon making up for carbon losses that occur through the process. This regenerated/fresh carbon mixture will maintain an adequate supply to the CIL circuits.

The eluate from carbon stripping will report to the electrowinning circuit. Gold will be loaded from the solution onto stainless steel cathodes. After loading, the cathodes will be washed to remove the metals sludge. The cathode sludge will be combined with the gravity concentrate and sent to the smelting process to produce gold doré.

The slurry exiting the CIL circuits will report to the two tailings thickeners. Thickener overflow reports to the process water tank and will be used for water needs upstream. Thickener underflow slurry will be pumped to the cyanide detoxification system to reduce cyanide toxicity. It will be treated using the INCO process. Detoxified tailings will report to the Tailings Management Facility. Water recovered by the reclaim barge pumps from the settled tailings will be returned upstream to meet process water requirements.

Process plant ancillary facilities are comprised of the following:

- Electrical substation
- Electrical power distribution system (PDC buildings & overhead power distribution lines)
- Process water system
- Potable water treatment
- Sewage treatment plant
- Administration / Truck shop
- Operations camp
- Tailings distribution pipeline
- Reclaim water barge, pipeline and pumps
- Laboratory

Process plant support facilities are comprised of the following:

- Tailings management facilities
- Two fresh water reservoirs and pumps (Hess Creek Reservoir & Fresh Water Reservoir)
- Main electrical overhead power supply and associated substations.

18.0 Project Infrastructure

18.1 Access Roads

The property straddles the Elliott Highway, a paved, all-weather highway linking the North Slope oil fields to Fairbanks and adjoins the Alyeska Pipeline corridor, which transports crude oil from the North Slope south and contains the fiber-optic communications cable that may be used at the Project site. Locally, a number of unpaved roads lead from the Elliott Highway into and across the deposit. A 3,000 ft (914 m) runway is located 3.7 mi (6 km) to the southwest of the project and is suitable for light aircraft.

18.2 Mine Waste Management and Water Control

18.2.1 Overburden Rock Storage Area

Non-economic overburden rock produced by mining activities at the Livengood site will be hauled and stockpiled in the Gertrude Creek valley. The current design is for 730 Mt storage.

A rock filled embankment will be constructed in lower Gertrude Creek valley (Gertrude Creek embankment). The embankment is designed to enhance slope stability of the overburden storage facility, to collect seepage and runoff from the Gertrude Creek valley, and as a structure to support the TMF liner at the base of the overburden facility. The shear key starter embankment will be lined on the upslope side and pumps installed to collect runoff within the storage facility for discharge into the TMF.

18.2.2 Tailings Management Facility

The Tailings Management Facility (TMF) has been designed as a fully lined facility to provide safe and secure storage of approximately 501 million tons of mill tailings along with a supernatant pond for ore processing solutions.

The main TMF embankment is situated across the Livengood Creek valley. Both the TMF embankment and the impoundment area will be designed as geomembrane lined facilities. The TMF embankment requires the removal of some native materials within the embankment footprint to improve stability characteristics of the foundation. These materials will be excavated and transported to growth media stockpiles in the general area for use during reclamation of the project site. The embankment will then be constructed in phases beginning with a Starter Dam, followed by a succession of five raises to the final crest elevation. In addition to the phased embankment expansions, the basin of the TMF will also be expanded in phases. The embankment and basin expansions will be constructed concurrently approximately every other year during operations.

The TMF embankment will be the primary structure for the TMF impoundment. The TMF embankment will be constructed with earth and rock fill materials generated from the surface mine or borrowed from within the project limits. The design of the embankment includes a 60-mil linear low density polyethylene (LLDPE) geomembrane on the upstream slope, underlain with Transition Zones, Select Rockfill and Rockfill material zones. The Starter embankment also includes a geosynthetic clay liner (GCL) below the LLDPE geomembrane. The GCL will further reduce the potential for seepage through the embankment during the initial years of operation when the supernatant pond will be located adjacent to the embankment. The upstream slope of the starter embankment is proposed to be 3H:1V (horizontal : vertical) and slope of

2.5H:1V for all subsequent raises. Reclaim pipe benches are provided at each raise crest elevation. The downstream slope is designed at a 1.8H:1V.

A TMF underdrain system will be installed within the major drainages in the Livengood Creek valley and will be located below the 60-mil LLDPE TMF impoundment geomembrane. These drains are designed to capture near surface groundwater flow and seepage from the Fresh Water Reservoir and convey it through the TMF embankment to the underdrain collection sumps located immediately downstream of the TMF embankment. Toe drains located along the downstream toe of the TMF embankment will also be incorporated into this drain system. Water collected in the TMF underdrain system sumps will be pumped into the TMF impoundment for reclaim.

A tailings underdrain collection system will be provided above the entire impoundment geomembrane to reduce the hydraulic head on the geomembrane and improve consolidation of the tailings. This underdrain system will collect solution that drains from the tailings and convey it to a collection sump located near the TMF embankment south abutment. The collected solution will then be pumped into the TMF impoundment for reclaim. Mill tailings will flow by gravity to the TMF. The tailings pipeline will follow the road on the south side of the valley (road to access Gertrude Creek embankment) and on the dam to spigot tailings along the face of the dam to minimize seepage. A reclaim barge is designed to recycle reclaim water to the mill.

18.2.3 Low Grade Ore Stockpile

Ninety three million tons of low grade ore will be stockpiled in upper Gertrude Creek during the mine life at a facility with a design capacity of 140 million tons. Runoff from the low grade ore stockpile will be collected and discharged into the TMF.

18.2.4 Water Control

Surface water management structures consist of two water reservoir's (WSR) and two surface water diversion channels. These structures will be used to manage and divert surface water generated from precipitation events and the spring freshet. The WSR's are identified as the Hess Creek Water Storage Reservoir and the Fresh Water Reservoir. The surface water diversion channels will be constructed along the roads built to access the WSR's Access Roads.

The Hess Creek WSR and Fresh Water Reservoir will provide the fresh water needed for the operation of the processing and other project facilities. The Hess Creek WSR will be located in Goldstream and Hess Creek basins, northeast of the proposed Livengood TMF, and has been sized to store approximately 14,390 acre-feet of water. The Fresh Water Reservoir will be located west of the Overburden Stockpile at the southeast end of the proposed TMF impoundment and has been sized to store approximately 2,880 acre-feet of water. Fresh water collected in the Hess Creek WSR will be pumped to the Fresh Water Reservoir, as needed, to maintain a minimum operating pool. The water will then be pumped from the Fresh Water Reservoir to the processing facilities for use.

The second surface water diversion channel is located adjacent to the WSR Access Road and Pipe Corridor. This road and channel alignment is generally oriented northeast-southwest along the northern boundary of the TMF impoundment. The channel is planned to be 10 ft wide and varies in depth, along the alignment, from 3 to 6 ft. The erosion protection also varies along the channel alignment, from grass/vegetation lining near the upper limits of the channel, to a 4-in D50 riprap prior to the Myrtle Creek Drop Structure which will convey the water to the bottom of the valley west of the TMF embankment.

18.3 Logistics and Transportation

18.3.1 Introduction

SR International Logistics (SRIL) completed a logistics and transportation study to support the Livengood Feasibility Study. SRIL reviewed and compiled extensive data to plan a seamless and un-interrupted flow of materials and equipment from global suppliers to the project site. SRIL, with input from shippers Lynden Transport and Totem Ocean Express, created a comprehensive report detailing the logistics and transportation needs of the project. The report includes pricing detail for ocean freight, inland freight, air freight, heavy haul requirements, rail freight, consolidation and marshaling points and warehousing.

18.3.2 Freight Options Considered

The construction and commissioning of the Project will require effective front-end planning and a complete, schedule driven transportation and logistics plan. All freight forwarding activities will feature identification of critical path items. Expediting and inspection personnel will control, verify and facilitate the movement of goods to the project site.

Key project personnel and/or agents acting on behalf of the Project will be located at strategic points to ensure ocean freight and inland freight schedules are met and freight inspections/inventories and import customs documentation are compliant with USA government requirements.

Foreign shipments will be pre-inspected to verify quantities, purchase order engineer's compliance (EC) certification, customs documentation and completeness. The B-Harmonization classification number will be incorporated in all import documents to expedite customs clearance and delivery of goods to the project site. Duties & taxes will also be based on this number.

Designated key equipment will require pre-inspections to verify quality & quantities, EC certification and packing/handling compliance.

THM will set up a Primary Receiving Yard to hold and consolidate freight near the Project site. For this Feasibility Study, SRIL assumed the Primary Receiving Yard would be located on the northern outskirts of Fairbanks, near Highway 2. Alternatively, THM may decide to place the Primary Receiving Yard closer to site, near the current Alaska DOT station.

Ocean freight will be the dominant mode of transporting materials and equipment not readily available in Alaska. All methods of ocean freight may be utilized. Ships may take five days and barges ten days duration from Puget Sound to Anchorage.

Trucking will be the primary method to move materials and equipment to the project yards from Alaskan arrival ports. Freight will be consolidated at a Primary Receiving Yard assumed to be located near Fairbanks. The distances and drive time elements between Alaska ports and the prospective Fairbanks yard are given below:

- Anchorage Port to Fairbanks yard: 360 mi (6 hrs) via State Highways 1 and 3. The road has year round state road maintenance and regulations.
- Valdez Port to Fairbanks yard: 365 mi (7 hrs) via State Highway 4 with year round State maintenance and regulations.
- Seward Port to Fairbanks yard: 485 mi (8 hrs 30 min) via State Highways 1 and 3 with year round state maintenance and regulations. This route holds little benefit and should be avoided, but ocean shipping situations may dictate its use.
- Whittier Port to Fairbanks yard: 417 mi (7 hrs 30 min) via State Highways 1 and 3 with year round state maintenance and regulations. The primary size restriction is the Anton Anderson Tunnel which all road & rail must use. This is not a desirable location for on-forwarding freight by road. The port is primarily used for rail. Ocean alternatives may dictate this route.

Railroads have very detailed size-weight restrictions but are pound for pound the most cost effective method to move materials and equipment to Fairbanks. Regularly scheduled service connects with US and Canadian lines.

CAT, Komatsu and other mining and construction equipment dealers use rail as their primary method to move equipment to the Alaskan market. Rail should be considered for any producer with national rail contracts selling FOB Fairbanks. Also, any mining contractor moving equipment from the lower 48 states to Alaska should consider rail.

18.3.3 Recommended Base Routes and Costs

The preferred base route for most project equipment and materials contains four legs, and is shown in Figure 12.1. The legs are listed below with the typical cost per ton:

EX-works to Puget Sound, Project average cost is $438/t
Puget Sound to Anchorage, cost is $148/t
Anchorage to Fairbanks, cost is $167/t
Fairbanks to Livengood cost is $32/t
Total Base Case Freight Cost: $785/t

Figure 18.1 Primary Route, Livengood Logistics Plan (MTB, Google Earth)

18.4 Fresh Water Supply

Two Water Storage Reservoirs (WSRs) will be developed to provide the project with a sufficient supply of process make-up and fresh water. The WSRs are identified as the Hess Creek Water Storage Reservoir and the Fresh Water Reservoir. The Hess Creek WSR will be located approximately 11 miles (17 kilometers) northeast of the mill within Hess Creek. This facility consists of a cross valley, zoned earth-fill embankment and associated appurtenant structures. The Hess Creek WSR is designed to hold an estimated 14,390 acre-feet of water covering an area of 505 acres with an approximate maximum operating depth of 85 ft. (26 m).

The proposed embankment will be located immediately upstream of an abandoned placer mine water reservoir embankment. This abandoned reservoir embankment will be utilized as a buttress for the new facility. During initial construction activities, a coffer dam will be constructed immediately upstream of the construction area to allow diversion of stream flows around the proposed embankment footprint.

Water will be recovered from the Hess Creek WSR via a water intake structure for use as process water make-up and freshwater supply.

A reinforced concrete emergency spillway will be constructed across the embankment crest on the east side of the embankment to route excess inflow (up to and including runoff associated with the Probable Maximum Flood) without permitting the embankment to overtop. A 48-in diameter low-level outlet pipe will be provided that will be capable of draining the reservoir. Flows from the spillway and the low-level outlet will be discharged downstream of the embankment into Hess Creek.

The Fresh Water Reservoir is located east of the Overburden Stockpile at the southeast end of the proposed TMF facility, approximately 3 mi (4.8 km) East Northeast of the process plant. A fresh water pipeline will be constructed from the Hess Creek WSR to the Fresh Water Reservoir where a series of fresh water pumps will transfer the fresh water to the process facilities. The fresh Water Reservoir is designed to hold an estimated 2,880 acre-feet of water covering an area of 75 acres with an approximate maximum operating depth of 116 ft.

Water will be recovered from the FWR via a water intake structure for use as process water make-up and freshwater supply.

18.5 Power Supply

Golden Valley Electric Association (GVEA), a member-owned cooperative, provides the only regulated electrical service to customers connected to the rail belt power grid north of the Alaska Range. Historic peak winter demand on the GVEA system is approximately 210 MW. GVEA is connected to south central Alaska via a single 138 kV transmission line that has a capacity to import approximately 75 MW into the GVEA service area.

Electric Power Systems, Inc. (EPS) conducted a power supply study and determined that the GVEA system, with modifications, is capable of providing the Project with the estimated 100 MW of power required. The additions and modifications to the GVEA system that will be required include:

- 50 mi 230 kV transmission line and SVC
- 60 MW turbine generator
- O'Connor Creek Substation
- GVEA transmission system upgrades

18.5.1 50 Mile 230 kV Transmission Line and SVC

Dryden & Larue completed design for the 50 mi 230 kV transmission line. The transmission line would be permitted in conjunction with the Project, would be constructed by Tower Hill Mines, and operated by GVEA. EPS determined that a 25 MVAR SVC is required at the Livengood mine site substation to modulate the transient effects of the project to GVEA specifications.

18.5.2 60-MW Turbine Generator

GVEA operates a naphtha fired General Electric LM6000 PC gas turbine/steam combined cycle generation plant approximately 15 mi southeast of Fairbanks. This plant currently has a single turbine installed. The back end steam generation unit was originally designed to accommodate two gas turbines. The facility was permitted and constructed in 2006 with provisions for installation of a second LM6000 turbine. The power study completed by EPS determined that this second turbine is required to provide service to Livengood.

18.5.3 O'Connor Creek Substation

A new 138/230 kV substation (O'Connor Creek) will be required to connect the Livengood transmission line to the GVEA system. GVEA has obtained a lease from the Fairbanks North Star Borough for the land parcel required for the substation.

18.5.4 GVEA Transmission System Upgrades

Upgrades to the GVEA transmission system will be required to supply the Project, including 33 mi of new transmission line or double circuiting of existing line.

19.0 Market Studies and Contracts

19.1 Market Studies

The gold market is global in nature and is unlikely to be affected by production from the Livengood Gold Project.

19.2 Contracts

There are several large 3rd party gold refineries with well-established industry relationships in North America. Among the more notable ones are:

- Metalor Technologies USA; North Attleboro, Massachusetts
- Johnson Matthey; Salt Lake City, Utah
- Canadian Mint; Ottawa, Ontario

THM has not contacted any of the aforementioned companies for competitive treatment bids.

The Feasibility Study utilized a refining, transportation and insurance charge of $9.30/oz of doré and payable terms of 99.5% for gold.

20.0 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environmental

20.1.1 Historical Project Activities and Permitting

Livengood Creek and the creeks draining Money Knob are mineralized and have been placer mined for nearly 100 years. Parts of the resource area on Money Knob have also hosted intermittent hardrock mineral exploration activities. The project area contains federal mining claims (Bureau of Land Management), state mining claims (Department of Natural Resources), state leases (Alaska Mental Health Trust Land), and private land (as described in Section 4.0). THM has received all appropriate authorizations required to conduct exploration, geotechnical, and baseline data collection activities.

20.1.2 Baseline Studies

THM has been conducting environmental baseline studies at the Livengood Gold Project since 2008 as part of THM's overall goal of providing environmentally relevant and supportable data for environmental permitting, engineering design, and a basis for permit-required monitoring during construction, mining, and closure of the project. These investigations are summarized in Table 20.1 and Table 20.2.

Table 20.1
Environmental Baseline Studies (2008-2012)

Baseline Study	2008	2009	2010	**2011**	2012
Surface Water					
Surface Water Quality		●	●	●	●
Hydrology			●	●	●
Hydrogeology					
Groundwater Quality			●	●	●
Hydrogeological Modeling			●	●	●
Permafrost Studies			●	●	●
Wetlands					
Wetlands Delineations		●	●	●	●
Meteorology & Air Quality					
Meteorological Data			●	●	●
Precipitation			●	●	●
Ambient Air				●	
Aquatic Resources					
Bio-monitoring		●	●	●	●
Resident Fish Surveys		●	●	●	●
Rock Characterization					
Static ML/ARD Testing			●	●	●
Kinetic ML/ARD Testing				●	●
On-Site Kinetic Testing					●
Wildlife Studies					
Habitat Mapping				●	
Mammal Surveys				●	

Table 20.1 Environmental Baseline Studies (2008-2012)					
Baseline Study	2008	2009	2010	**2011**	2012
Avian Surveys				●	●
Cultural Resources					
Cultural Site Surveys	●	●	●	●	●
Noise Studies					
Noise Surveys					●

Table 20.2 Summary of Environmental Baseline Studies	
Baseline Study	**Program Summary**
Surface Water	
Surface Water Quality	Surface water quality samples have been collected since 2009 over a wide range of hydrologic conditions. The station network includes 19 stations in the project area and 4 stations along the power line corridor. All samples have been analyzed for a comprehensive suite of analytes and include QC sample collection. While there are apparent local and seasonal spikes among some analytes, these are deemed to be mostly natural and, in part, a reflection of placer mining activity.
Hydrology	The project region is characterized by large areas of permafrost that limit groundwater recharge into local streams. As a result, many streams are ephemeral during periods of low precipitation. The USGS currently maintains five stream gauges in the project area. Snow surveys were completed in a variety of aspects, elevations, and vegetation types in late spring 2010-2013. Regional data sources were also used to characterize average, extreme drought, and flood conditions at the project site, enabling the development of a long-term synthetic record of estimated monthly precipitation at the project site, and forming the basis of the water balance model.
Hydrogeology	
Groundwater Quality	THM has sampled 52 groundwater wells throughout the project area. Water chemistry data indicates that groundwater varies locally and is controlled by geology and permafrost. Groundwater is most mineralized in the vicinity of the deposit; groundwater distal to the deposit has the least mineralization.
Hydrogeological Modeling	Compilation of average well static water levels collected from the site piezometer network and pump tests indicates that the groundwater surface generally follows topography indicating groundwater flows from higher elevations to lower elevation areas. Groundwater recharge to the deposit area is from the ridge to the east from the pit. The hydraulic conductivities observed down-gradient from the proposed pit and in the rocks in the Livengood Valley are relatively high. The lowest hydraulic conductivity values were observed to the north and east of the deposit. Groundwater is confined under permafrost. Predictive numerical simulations for project groundwater have been conducted for passive pit inflow conditions and indicate that the pit will take several hundred years to fill.
Permafrost Studies	Thermal analysis has been performed to provide a site-wide understanding of permafrost conditions and a basis for engineering design. In general, the permafrost beneath the Livengood Gold Project area is extensive, but relatively warm (>-2°C) and discontinuous. Permafrost depths at the project reach nearly 600

Table 20.2 Summary of Environmental Baseline Studies	
Baseline Study	**Program Summary**
	feet (183 m) below ground surface.
Wetlands	A 62,000-acre preliminary wetlands map of the project area and power line corridor has been completed using field data collected from 2009-2012, and is being used for mine design. Slightly less than half of the mapped area has been delineated as wetlands, the majority of which are dominated by black spruce forests and near-surface permafrost. Despite the fairly wide distribution of eleven invasive species within the study area, most of the populations are relatively small. The control and containment of these species will be considered during development of project management and reclamation plans.
Meteorology & Air Quality	Two meteorological stations were installed in late 2010 for use in dispersion modeling, air quality permitting, facility design, and other baseline studies. One station is located on Gertrude Ridge, above and east of the resource area, and collects data including temperature, year-round precipitation, wind direction and speed, and relative humidity. The other station is located to the southwest of the resource area at a lower elevation and collects the same meteorological parameters as well as seasonal evaporation data. Two PM 2.5 meters were co-located with this station to monitor ambient air quality in 2011.
Aquatic Resources	
Bio-monitoring	As the most populous fish in the project area, young of year Arctic Grayling were targeted for full-body tissue analysis. Fish tissue, macro-invertebrate, and periphyton sampling was conducted in 2009-2012. Tissues of the resident fish in the area contain detectable metals concentrations, as do many regional steams in naturally mineralized areas. The project area supports a robust benthic population of less sensitive species, as would be expected in streams that have hosted long-term placer mining.
Resident Fish Surveys	In addition to bio-monitoring, the 2010 program included a summer fish presence/absence survey, a May Arctic Grayling spawning survey, May Northern pike metals analysis, and a fall Whitefish otolith study. In 2011, a fish overwintering investigation was completed as well as a data gap analysis along the power line corridor. Survey results indicate that there are grayling overwintering in the West Fork of the Tolovana River and the old placer pond located in the Livengood Creek Valley. No salmon species have been found in the project area. The three major drainages (Chatanika, Tatalina, and Tolovana Rivers) and their tributaries along the power line corridor are identified as fish-bearing.
Rock Characterization	In 2010, composites of various resource rock types, alterations, and oxidation were created and tested for metal content, sulfur speciation and Acid Rock Drainage (ARD) potential. This work has since been expanded to include static and kinetic testing on selected samples obtained from the entire resource area data package, the resource dataset screened for gold grades less than 0.3 g/mt, ore composites, tailing samples, regional rock types, and overburden. The sample selection process included screening for rock type as well as sulfur, arsenic, mercury, selenium, and antimony content. Currently, 75 humidity cell tests are underway. Samples from the datasets have also been tested for Meteoric Water Mobility Potential (MWMP) and sequential MWMP. Nineteen 250-kg barrels of resource area materials are also

Table 20.2 Summary of Environmental Baseline Studies	
Baseline Study	**Program Summary**
	undergoing on-site multi-year testing to establish scalability factors. The data indicates that certain stratigraphic units in the resource area are potentially acid generating (PAG) while other rock types are non-PAG. All resource rock types are potential metal leaching (ML), with arsenic, antimony, and selenium being of primary interest. Mineral content and ARD potential tends to decrease outside the resource area. Management of these materials is discussed in Section 20.1.4.
Wildlife Studies	
Habitat Mapping	Wildlife studies were initiated in 2011 and included a review and synthesis of existing data in the project area, GIS mapping of wildlife habitats, and field surveys for key wildlife species. There are currently no threatened and endangered wildlife species known in the project area. The majority of the wildlife habitats in the study area comprise black-spruce dominated upland open needle leaf forest.
Mammal Surveys	Aerial surveys of moose were conducted in the project area to determine the population density and late winter distribution. During the survey, a total of 51 moose within 13 surveyed sample units were sighted.
Avian Surveys	In the project area and the power line corridor, less than a third of the raptor nests were found to be occupied. Eight species of land birds that are considered high priority species for conservation were recorded in the project area in 2012, although none of these species were confirmed nesting.
Cultural Resources	Cultural resource surveys have been complete on nearly 16,000 acres of the project area and 5,000 acres of the power line corridor. To date, 124 historic features and 21 prehistoric sites have been newly identified. The majority of these historic features are remains of historic placer camps and workings; the majority of prehistoric sites contain surface and subsurface lithic materials. These sites are not currently expected to impact project construction and the archaeological consultants have provided recommendations which include a policy of feature avoidance to prevent damage to the condition or integrity of identified features. During the project permitting process, all features will be reviewed by the State Historic Preservation Office (SHPO) and federal agencies working under Section 106 of the National Historic Preservation Act (NHPA). Mitigation plans will be developed as needed.
Noise Studies	Winter session noise monitoring was completed in March 2013 with the summer session scheduled for July 2013. Five locations are being monitored during each session employing two different techniques (short term and 24 hour).

20.1.3 No Known Material Environmental Issues

Based on review of the studies completed to date, there are no known environmental issues that are anticipated to materially impact the Project's ability to extract the gold resource. THM is incorporating results of the baseline data collection into engineering and closure design plans.

20.1.4 Environmental Management Strategies

Tailings Management Facility - The TMF has been designed to safely contain tailings and fluids through the use of a geosynthetic liner and a cross-valley embankment on the west end of the Livengood Valley. A rock fill underdrain system will be constructed in the basin to collect near surface groundwater and any seepage that may occur from the overlying liner system. During operations seepage from the underdrain will be collected and pumped into the TMF. Modeling and pump tests suggest that permafrost underlying the basin isolates the TMF from deep groundwater.

Waste Rock Facility - In order to minimize ARD potential and achieve an ideal blend of PAG and non-PAG materials, the facility will be constructed in lifts to facilitate blending. If needed, rocks demonstrating high relative levels of ARD or metal leaching (ML) will be specifically managed within the waste rock facility. Underdrains will collect meteoric waters that infiltrate the waste rock and carry it to a lined sump at the up-gradient base of the embankment constructed along the bottom of the Gertrude Creek basin. From there the collected water will be pumped into the TMF. The Gertrude Creek basin is underlain by permafrost that restricts communication with the deep groundwater.

20.2 Closure Plan

A key to the successful closure of the Livengood Gold Project is to incorporate as many environmental considerations into the initial design process as possible. These considerations are reflected in the FS design and include the characterization studies of the overburden, tailings, and water that have been underway for several years.

The closure plan presented is conceptual and may not represent the executed closure plan should this project advance to an operational facility. The plan will extend over a 32 -year period starting in production Year 14 with the construction of a water treatment plant and ending in Year 45 with the decommissioning of the water treatment plant. The facility closure plan is divided into two main phases: closure and post-closure.

A reclamation and closure plan will be submitted to the agencies during the permitting process and will discuss the final outcome of the project, including a final land use plan, re-grading, long-term water quality monitoring and management, test vegetation plots, the closure design, removal of facility components, and financial assurances. In addition, the Livengood Gold Project will need to prepare a U.S. Corps of Engineers Compensatory Mitigation Plan for mitigating unavoidable wetlands impacts and, including the input from many reclamation and mitigation bank experts. It may require the setting up of mitigation banks with third parties.

20.2.1 CLOSURE ACTIVITIES

Closure will involve initial reclamation and salvage and will take approximately 5 years to complete.

<u>Water Treatment Plant</u>

A 6,000 gpm water treatment plant will be constructed during Mine Year 13 to treat water removed from the TMF supernatant pond and seepage from the TMF underdrain system and the Overburden Rock Storage sump. Geochemistry and groundwater sampling suggests that the arsenic, selenium, and antimony contained in pond, seepage, and sump water will be treatable. The water treatment plant will be of modular construction consisting of 500 gpm units, so over time, as the treatment requirements reduce, modules can be taken out of service.

Management Facility

A dry closure of the TMF has been incorporated into its design. The supernatant pond will be removed and treated. Three years will be required to place a 3-foot thick layer of overburden rock over the entire tailings surface. A 1.5-foot layer of Growth media will then be placed over the rock. The capped tailings surface will be seeded and fertilized. Diversion channels will be constructed along the perimeter of the tails basin; the flow will be diverted past the embankment through drop structures.

Surface Mine

At the end of mine life, active dewatering of the surface mine will cease and the pit will be allowed to naturally fill with groundwater. Groundwater modeling indicates that the pit will take several hundred years to fill.

Overburden and Ore Rock Facility

The Overburden Rock Facility has been designed to minimize the impacts from potentially acid-generating waste rock. During closure, the overburden will be contoured, covered with 1.5 feet of growth media, seeded and fertilized. The ore stockpile area will be ripped prior to placement of growth media, seed, and fertilizer. The interface area between the graded stockpile toe and the natural ground will be riprapped to prevent erosion of the stockpile toe in areas where there will be concentrated runoff flows. The flow will be directed to the TMF diversion channels. Once flows to the sump have decreased, the pumps and other equipment will be salvaged.

Roads, Foundations, Buildings, and Equipment

During closure, buildings will be removed from their foundations, with the exception of the Water Treatment Plant and other closure support buildings. All work pads and roads not needed for site access will be dozer ripped, covered with 1.5 feet of growth media, seeded and fertilized. Pre-construction drainage patterns will be restored or enhanced to minimize stormwater impacts. Safety berms will be dozed over the road slope or into road ditches to further enhance drainage.

Water Storage Reservoirs

The Hess Creek WSR will be drained and the embankment will be breached. Riprap will be installed in the breached area to protect against erosion. Mechanical and electrical structures will be removed.

The Fresh WSR will be backfilled with overburden rock to the height of the tailings and the embankment will be breached above the tailings level. All rock will be contoured prior to placement of 1.5 foot of growth media, seed, and fertilizer. A channel will be installed with riprap to allow drainage into Livengood valley.

20.2.2 POST CLOSURE ACTIVITIES

The post closure period includes 5 years of site stabilization and maintenance after closure is complete and a subsequent 20 years of water treatment and monitoring.

20.3 Permitting

20.3.1 Project Permitting Requirements

The Project will require numerous federal and state permits and authorizations. Table 20.3 lists the permits likely to be required based on the conditions at the time of this report.

Since development of the Project will require a number of Federal permits, the National Environmental Policy Act (NEPA) and Council of Environmental Quality (CEQ) Regulations will govern the federal permitting portion of the Project. The NEPA process requires that all elements of a project and their direct, indirect, and cumulative impacts be considered. A reasonable range of alternatives are evaluated to assess their comparative environmental impacts, including consideration of feasibility and practicality. In fulfillment of the NEPA requirements, it is anticipated that the Project will be required to prepare an Environmental Impact Statement (EIS). Upon completion of the EIS and the associated Record of Decision by the lead federal agency, the federal and state agencies will then complete their own permitting actions and decisions. The State of Alaska is expected to take a cooperating role to coordinate the NEPA review with the State permit process. Actual permitting timelines are controlled by the Federal NEPA review and Federal and State agency decisions.

Table 20.3
Project Permit Requirements

Agency	Authorization
Federal	
U.S. Army Corps of Engineers (USACE)	CWA Section 404 Permit (wetlands dredge and fill)
	Section 106 Historical and Cultural Resources Protection
U.S. Environmental Protection Agency	Storm Water Construction General Permit
(EPA)	Storm Water Discharge Multi-Sector General Permit for Industrial Activities
	Spill Prevention, Control and Countermeasure Plan (SPCC)
	EPA Air Quality Permit Review
	EPA Hazardous Waste Generator ID Resource Conservation and Recovery Act (RCRA)
National Marine Fisheries Service	Threatened and Endangered Species Act Applicability Consultation
U.S. Fish and Wildlife Service	Section 7 Threatened and Endangered Species Act Consultation
	Bald Eagle Protection Act Clearance
	Migratory Bird Protection
	Fish and Wildlife Coordination Act
U.S. Bureau of Land Management	Plan of Operations Approval
	Decision Record
	Bond Approvals
U.S. Bureau of Alcohol, Tobacco & Firearms	Permit & License for Use of Explosives
	License to Transport Explosives
Mine Safety and Health Administration	Notification of Legal Identity
	Training of Miners Plan
Federal Aviation Administration (FAA)	Notice of Controlled Firing Area (Blasting)
	Structure Warning Lights
Federal Communication Commission - WTB	Radio Station License
U.S. Department of Transportation	Approval to Transport Hazardous Materials
U.S. Regulatory Commission	Material License for geotechnical studies
State	
Alaska Department of Natural Resources Division of Mining, Land & Water	Miscellaneous Land Use Permits
	Plan of Operations

Table 20.3 Project Permit Requirements	
Agency	**Authorization**
	Reclamation Plan Approval
	Reclamation Bond
	Mining License
	Land Use Permits and Leases
	Certificate of Approval to Construct a Dam
	Certificate of Approval to Operate a Dam
	Dam Safety Certification
	Material Sale (for construction material borrow areas)
	Temporary Water Use Permit (if not acquiring water rights)
	Water Rights Permit (if not using Temporary Water Use Permit)
	Road Right of Way/Access
	Power Line ROW
	Cultural Resource Protection
	Archeology Study Permits
Alaska Department of Environmental Conservation	Alaska Pollution Discharge Elimination System (APDES)
	Section 401 Water Quality Certification (SWA 404 Permit)
	Overburden Management Permit (includes Solid Overburden and wastewater)
	Storm Water Pollution Prevention Plan (SWPPP)
	SPCC Plan Review Approval
	Approvals to Construct and Operate a Public Water Supply System
	Plan Review and Construction Approval for Domestic Sewage System
	Solid Waste Landfill Permit
	Food Sanitation Permit
	Air Quality Pre-Approved Limit – Diesel Engines
	Air Quality Construction Permit (first 12 months)
	Air Quality Control Major/Minor Permit to Operate
	Air Quality Permit to Open Burn
Alaska Department of Fish & Game	Fish Habitat and Fish Passage permits
Alaska Department of Transportation & Public Facilities	Notification of Blasting for Road Closure
	Controlled Firing Area for Blasting
	Right of Way/Access/Driveway
Alaska Department of Public Safety-FP	Communication Site Permit
	Fire Marshal Plan Review
Alaska Department of Labor and Workforce Development	Certificate of Inspection for Fired & Unfired Pressure Vessels
	Employer Registration
Alaska Department of Health and Social Services-CHEMS	Life Flight Service
Other Entities	
Alyeska Pipeline	TAPS ROW access/crossing approvals

20.3.2 Status of Permit Applications

There have been no permit applications submitted for Project construction.

20.4 Requirements for Performance or Reclamation Bonds

There are two State of Alaska agencies that require financial assurance in conjunction with approval and issuance of large mine permits. The Department of Natural Resources Division of Mining, Land and Water and the Department of Environmental Conservation require financial assurance both during and after operations, and to cover short and long-term water treatment if necessary, as well as reclamation and closure costs, monitoring, and maintenance needs. The financial assurance amounts will be estimated in conjunction with development of the Reclamation and Closure Plan.

20.5 Mine Closure Requirements and Costs

AMEC developed a mine closure plan featuring dry closure of the tailings management facility. Closure costs track reclamation and closure expenses from Year 14 through 45. The main reclamation and closure effort occurs from Year 15 through 19 and includes deconstruction of the facilities and closure of the tailing management facility, overburden rock facility, roads, and water storage reservoirs as described in Section 20.2. These costs total $256.3 M, including contractor indirect costs. Subsequent post-closure costs incurred during Years 19 through 45 include pumping, water treatment, maintenance and post-closure monitoring. These costs total $96.7 million. Year 45 is the last year with planned closure expenses.

The total closure cost is $353.0 M. This total closure cost is applied to the cash flow in Year 14. This cost includes closure of the overburden stockpile, tailings management facility, solid waste landfill, and ancillary facilities, including indirect costs.

Closure cost funding will flow from a closure trust fund financed by mine cash flow. Annual contributions to the closure trust fund are included in the cash flow model. The annual contribution is $16,174,000 during Years -2 through 14. The model includes trust fund earnings at 4.0% annual percentage rate (APR), applied to the fund balance until closure is complete Year 14 through 45. The main reclamation and closure effort occurs from Year 15 through 19 and includes deconstruction of the facilities and closure of the tailing management facility, overburden rock facility, roads, and water storage reservoirs as described in Section 20.2.

20.6 Socioeconomic Conditions

The Livengood Mining District has a history of cyclical employment and development dating back to 1914 when placer gold mining became the primary economic activity in the area and has produced over 500,000 oz of placer gold, with two-thirds of that production coming prior to World War II. In 2012, there were three small placer operations active in the Livengood area. Today, there are no year round residents in the town-site, with only a handful of abandoned structures still standing.

20.6.1 Regional Economy

Livengood lies within the Yukon-Koyukuk Census Area, which encompasses a very large swath of Interior Alaska from the Canadian border to the lower Yukon River. In 2012, the Census Area held a total population of 5,682 widely dispersed residents in 20 communities, of which 71% were Alaska Natives. Minto, which is approximately 40 mi (64 km) from Livengood and Manley Hot Springs, 80 miles (129 km) away, have road access to Fairbanks.

The Fairbanks area is the service and supply hub for Interior and Northern Alaska. Construction of the Trans-Alaska Pipeline (TAPS) resulted in an economic boom in Fairbanks from 1975-77. The oil industry remains an important part of the local economy, with Fairbanks providing logistical support for the North Slope activity, the two local refinery operations, and the operation and maintenance of TAPS. Today, the

University of Alaska, the Fairbanks Hospital, and the Fort Knox and Pogo gold mines are some of the Fairbanks area's largest employers. The Fairbanks North Star Borough (FNSB) economy included 39,400 non-agricultural wage and salary jobs in 2012. In 2011, average employment of 39,018 wage and salary jobs, accounted for $1.81B in annual payroll.

20.6.2 Recreational and Subsistence Resources

The State of Alaska Tanana Area Basin plan designates mining and the primary land use for the Livengood project area. The plan identifies recreation as a secondary use in the project area. It will be important to consider both the present and likely future recreational uses of the area and how mining projects can cohabitate successfully.

Most of the small communities in rural interior Alaska are largely dependent on subsistence. Seventy-five percent of the Native families in Alaska's smaller villages acquire 50% of their food through subsistence activities (Federal Subsistence Board, 1992). For families who do not participate in a cash economy, subsistence can be the primary direct means of support; for others, it contributes indirectly to income by replacing household food purchases.

20.6.3 Socioeconomic and Project Consequences

Developing the Livengood Gold Project into a mine would offer residents and families from the surrounding communities the opportunity of year round stable wage paying jobs. Continuing local hire efforts by THM will be a key focus of the project. Training programs such as the Drill Helper Training Program conducted in May of 2011, a partnership with the State Dept. of Labor, will be used to attract, train and retain an Alaskan workforce for the various construction and operating jobs available.

The feasibility study estimates a total of 6,974,000 man-hours during project construction with a peak construction workforce of 814. The average wages of those workers is estimated at $42/h. During the two years of pre-production mine development, owners crew will be approximately 200 employees. During operation, the peak employee count is estimated at 425 and an annual average wage of approximately $97,000/y. Total annual wages paid during operation is estimated to be $41.6 M.

20.6.4 Support Services

A 2011 study of the economic impact of the Fort Knox Mine on the Fairbanks North Star Borough determined that 62% of the mine's goods and services spending were with businesses located in the Borough. For purposes of this report, we have assumed a local purchase volume of 50% for the Project. Using that assumption, the result would be an annual local expenditure of approximately $250 M on consumables, supplies and purchases.

One-way travel time from Fairbanks to the Project site is approximately 1.5 hours in the summer, and 2 hours in the winter, making daily commuting a significant challenge. As a result, workers will be housed in a camp at Livengood both during construction and operations. There will be three crews. Schedules will generally be two crews on 12-hour shifts, using a two week on and one week off rotation.

20.6.5 Employment and Training

The labor force in the communities nearest the mine is very small. The total population of Minto, Manley Hot Springs and Livengood combined is just over 350 residents in 2012. Skilled and unskilled labor to support mine development and operations will come primarily from the Fairbanks area, with a total labor force of over 40,000 workers. The training plan for the Livengood Gold Project will be designed to promote safety, environmental stewardship, efficient production, and local hire.

21.0 Capital and Operating Costs

The capital cost estimate for the Livengood Gold Project addresses the development, construction, and start-up of a mine and plant capable of processing 100,000 t/d (90,718 mt/d) of gold bearing material.

THM has engaged various Consultants to provide estimate support for various cost portions of the project that fall within their specialized scope of work (see Table 21.1). Summary data was supplied for inclusion and use in this capital cost estimate.

Table 21.1
Capital Cost Estimate Contributors

Scope / Responsibility	Consultant
Mine Costs	Mine Development Associates
Haul Roads and Access Roads	AMEC
Tailings Management Facility	AMEC
Mine Power Supply Line and substations	Dryden & LaRue, EPS and GVEA
Process & Ancillary Facilities	Samuel Engineering
Indirect Cost	All
Owner's Cost	THM & MTB
Contingency	All

Note 1 Independent third party commercial contracting firms with the expertise to execute the construction plan were retained to confirm the cost estimates of all major facilities

21.1 Capital Costs

The total estimated cost to design, procure, construct and commission the facilities described in this section is $2.79B and sustaining capital of $893 M. Table 21.2 summarizes the initial capital and sustaining capital costs by major area. The estimate is expressed in nominal first quarter 2013 United States dollars. No provision has been included to offset future escalation.

Table 21.2
Initial Capital and Sustaining Capital Costs by Major Area
($ millions)

	Initial	Sustaining
Process Facilities	$1,119	$26
Infrastructure Facilities	708	506
Power Supply	129	0
Mine Equipment	189	126
Mine Development	177	0
Other Owners Costs	166	9
Contingency	271	0
Subtotal Before Reclamation	**2,758**	**667**
Funding of Reclamation Trust Fund (1)	32	226
Total	**$2,790**	**$893**

Rounding of some figures may lead to minor discrepancies in totals.

(1) Includes initial funding, total $353 Million estimated costs. The difference of $95 M is assumed trust fund earnings.

21.1.1 Accuracy

After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy in the range of plus or minus 15%. Estimate accuracy ranges are projections based upon cost estimating methods and are not a guarantee of actual project costs.

21.1.2 Exclusions and Assumptions

General exclusions from the capital estimate are as follows:

- Sunk costs (costs prior to start of detailed design)
- Allowance for special incentives (schedule, safety, etc.)
- Interest and financing cost
- Escalation beyond First-Quarter 2013
- Taxes and import duties
- Risk due to labor disputes, permitting delays, weather delays or any other force majeure occurrences.

General Assumptions are:

- Construction will be subcontracted
- Overburden from the surface mine will be available to construct geotechnical facilities
- Rotation for craft and supervision is 20 d on and 10 d off during construction
- Earthworks estimates are quantity based using engineering-derived material take off (MTO) quantities for the work elements

21.1.3 Estimating Methodology

The estimate is built up by cost centers as defined by the project work breakdown structure (WBS) and by prime commodity accounts, which include earthwork, concrete, structural steel, process building, mechanical equipment (including platework), piping, electrical and instrumentation.

Imperial units of measure have been used throughout the estimate. Metric units may be used in item descriptions to aid with clarification or if used in equipment model numbers.

Estimators derived costs through various sources including budgetary quotations, in-house historical data, published databases, factors and estimators' judgment (allowances).

Estimators assumed that equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages for lump sum or unit rate contracts.

MTB Project Management Professionals (MTB) conducted a detailed estimate review with Samuel Engineering, MDA and AMEC. MTB and the estimators reviewed each cost item with respect to its scope, how the engineer quantified the estimate, and the pricing used for the estimate. Any discrepancies or inconsistencies were discussed and resolved.

21.2 Operating Costs

The three major operating cost areas are mining, processing and general & administrative (G&A). The unit costs areas are shown in terms of total cost life of mine (LOM) per ore ton processed and total cost per oz of gold produced.

21.3 Mine and Operating Costs and Processing Production Schedule

Mine Development Associates (MDA) generated a production schedule for the life of mine. Key life of mine (LOM) schedule parameters are listed below:

- 105 Mt material mined during pre-production
- 1,116 Mt material mined during production
- Gold contained in ore totals 10,073,269 oz
- Average head grade is 0.69 g/mt
- Average recovery is 80.3%

The schedule accounts for four months of contactor mine development followed by pre-production stripping by the owner's mine forces to prepare the mine for full scale production. The mining schedule calls for the mine to operate in production through three months of year 12, a total of 11.27 years after pre-production. The production schedule specifies the plant runs through 10.5 months of year 14, or a total of 13.87 years. Ore stockpiles would feed the plant between years 11.27 and 13.87. Table 21.3 shows the mine and processing production schedules for life of mine.

Table 21.3
Mine and Processing Production Schedule

MDA Production Schedule Dated 03July13	Units	LOM	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
Mine																		
Total Ore Mined	Tons (000)	**501,137**	10,213	17,346	60,550	49,175	24,593	41,938	44,937	32,823	35,982	34,116	38,461	42,467	56,363	12,173	0	0
Mined Ore to Plant	Tons (000)	**261,841**	0	0	26,100	21,985	13,745	26,457	25,740	19,272	22,061	21,228	19,995	21,627	36,022	7,610	0	0
Mined Ore to Stockpile	Tons (000)	**239,295**	10,213	17,346	34,450	27,189	10,848	15,481	19,196	13,551	13,921	12,889	18,466	20,841	20,341	4,563	0	0
Stockpile																		
Stockpile Inventory	Tons (000)	**-**	10,213	27,559	58,783	71,245	59,224	64,549	72,873	68,982	68,350	65,853	67,701	73,454	93,205	68,764	32,151	0
Stockpile Ore to Plant	Tons (000)	**239,295**	-	-	3,226	14,728	22,868	10,156	10,873	17,442	14,553	15,386	16,618	15,087	591	29,003	36,613	32,151
Process Plant																		
Total Tons Processed	Tons (000)	**501,137**	-	-	29,326	36,714	36,613	36,613	36,613	36,714	36,613	36,613	36,613	36,714	36,613	36,613	36,613	32,151
Grade	g Au/t	**0.69**	-	-	1.08	0.94	0.67	0.76	0.74	0.63	0.66	0.66	0.67	0.72	0.82	0.54	0.39	0.39
Contained Metal	oz Au	**10,073,269**	-	-	922,990	1,008,896	715,548	812,825	789,472	676,137	709,550	707,543	720,793	769,098	878,887	577,724	417,337	366,471

21.3.1 Average LOM Operating Costs - Mine

The LOM operating costs include all expenses incurred to operate the mine from the start of Year 1 through Year 14. Mine pre-production costs are considered a capital expense. General mine expenses and engineering costs cover mine management and technical support. Drilling costs cover the expense of operating as many as four 171 mm drills and up to eleven 251 mm drills, including labor and materials over the life of mine. Blasting costs include the explosives materials and labor required to break the ore and overburden loose from the surface mine. Loading costs include labor and operating costs to operate 47 yd^3 front shovels or 40 yd^3 front end loaders and place the blasted rock into 320 t haul trucks. Hauling costs cover the labor, fuel and maintenance required to haul the overburden and ore to their respective destinations. Support costs define the cost to run equipment to keep in-mine and out-of-mine haul roads watered and graded, shovel and loading sites level and clean, and drilling sites level and clean. MDA based the mining operating costs on production hours required to perform required tasks.

Table 21.4 shows the life of mine and weighted average annual operating costs during the production period of Year 1 through Year 14. Weighted average annual costs are based on the tonnage mined. The total mine operating cost equals $3.93 per ore ton, which is $1.67 per ton of material mined.

Table 21.4
Average Annual LOM Operating Costs – Mining

Description	Life of Mine Cost $(000)	Weighted Average Annual Cost $(000)/y.	Cost per Ton Ore Mined
General Mine Expense & Engineering	104,462	8,356	$0.22
Drilling	151,849	13,259	$0.32
Blasting	287,598	25,068	$0.61
Loading	247,241	20,165	$0.52
Hauling	767,950	63,356	$1.62
Support	302,495	24,517	$0.64
Mining Totals	**1,861,590**	**154,722**	**$3.93**

21.3.2 Average LOM Operating Costs - Processing

Samuel Engineering (SE) estimated Livengood's process operating costs based on 100,000 t (90,718 mt) per day ore feed. SE estimated the operating cost to within plus or minus 15% accuracy. SE analyzed laboratory testing results by rock type to arrive at their overall operating cost estimate. SE categorized operating costs by labor, power, grinding steel, reagent chemicals, heating fuel, maintenance and operating supplies, and water and sewage treatment costs. Table 21.5 shows the resulting life of mine and average annual operating costs for processing. The total process operating cost equals $10.45 per ore ton.

Table 21.5 Average Annual LOM Operating Costs – Processing			
Description	**Life of Mine Cost $(000)**	**Weighted Average Annual Cost $**	**Cost per Ore Ton Processed**
Facility Power	1,570,439	112,568,754	$3.13
Reagent Chemicals	2,099,085	150,503,656	$4.19
Grinding Steel	1,042,706	74,739,604	$2.08
Hourly Labor	187,082	13,374,925	$0.37
Salaried Labor	54,279	3,880,556	$0.11
Ancillary Power (Camp)	9,428	673,996	$0.02
Facility Heating (LNG)	63,434	4,535,068	$0.13
Maintenance Supplies & Material	182,185	13,024,819	$0.36
Operations Supplies, Oil & Lube, Misc.	27,328	1,953,723	$0.06
Processing Totals(1)	**5,236,646**	**375,303,783**	**$10.45**

(1) Processing Total includes de minimus LOM costs for Potable Water Treatment and Waste Water Treatment of $49,000 and $631,000, respectively.

21.3.3 Average LOM Operating Costs – General and Administrative (G&A)

MTB and THM provided estimates of general and administrative (G&A) costs including general management, environmental management, community relations, human resources, accounting, etc. The total G&A operating cost equals $0.89 per ore ton processed. Table 21.6 shows life of mine and average annual operating costs for general and administrative expenses (G&A).

Table 21.6 Average Annual LOM Operating Costs – G&A			
	Life of Mine Cost $(000)	**Weighted Average Annual Cost $**	**Cost per Ore Ton Processed**
Gen Mgt. and Admin	13,104	936,864	$0.03
Environmental	34,915	2,496,178	$0.07
Community Relations	11,751	840,080	$0.02
Human Resources	15,985	1,142,838	$0.03
Health, Safety & Security	28,341	2,026,181	$0.06
Accounting	21,019	1,502,714	$0.04
Info Technology	16,100	1,151,007	$0.03
Warehouse	14,666	1,048,480	$0.03
Purchasing	9,565	683,837	$0.02
Camp	150,699	10,773,841	$0.30
Land	6,939	496,090	$0.01
Corp allocation	41,634	2,976,542	$0.08
Fuel, Tires & Main. Mobile Equip. & Process Plant Vehicles	24,506	1,751,985	$0.05
Insurance	55,512	3,968,722	$0.11
G&A Total	**$444,737**	**31,795,359**	**$0.89**

22.0 Economic Analysis

22.1 Introduction

Tower Hill Mines (THM) and MTB Project Management Professionals, Inc. (MTB) generated a Feasibility Study financial model for Livengood Gold Project. The model calculates revenues based on contained oz, head grade, recovery and a gold price of $1,500/oz. The model then subtracts costs to generate the project cash flow. The financial model provides the means to evaluate the Project's discounted cash flow and can guide future development decisions for the project.

22.2 Model Inputs

Table 22.1 presents the model inputs used in the economic analysis. MDA, AMEC and SE developed execution plans describing how the Project would be built and operated. The pre-production period and construction period financial inputs flow from the execution plan. Furthermore, the mine plan provided additional financial model inputs: mine life, ore tons mined, head grade and average annual gold production rate. The financial model applies metal pricing of $1,500/oz based on the three year trailing average price of gold reported by Kitco.com, which equaled $1,549.03/oz on June 30, 2013. First quarter 2013 US dollars form the financial model currency basis. No inflation or escalation exists in the model. The model operates on pre-tax and after-tax bases, and includes Alaska state taxes and Federal taxes according to 2013. The model applies 3% royalties on net smelter returns across the life of mine based on an average royalty calculation. The model includes provisions for gold transportation, insurance, refining and payable charges. These technical and economic parameters used in the model are summarized in the following sections.

Table 22.1
Model Inputs

Execution Plan	
Pre-production Period	27 months
Construction Period	29 months
Mine Life (after pre-production)	13.87 years
LOM Ore Tons (millions)	501
LOM Gold Grade (g/mt Au)	0.69
Average Annual Process Gold Production Rate (oz)	577,598
Metal Pricing	
Gold Price ($/oz)	1,500
Cost and Tax Criteria	
Estimate Basis	Q1 2013
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Income Tax	AK State, Federal
Royalties	
Royalty on Net Smelter Return (NSR)	3%
Gold Transportation and Insurance, Refining, and Payable Charges	
Gold ($/oz)	9.30
Payable Terms	
Doré Gold	99.50%

The total and unit operating cost estimate summary are shown below in Table 22.2 & Table 22.3 respectively.

Table 22.2 Total Operating Costs	
Total Operating Costs	**Total Cost LOM**
Mining	$1,861,590,070
Processing	$5,236,646,116
General & Administration	$444,737,330
Project Total Operating Cost	**$7,542,973,515**

Table 22.3 Unit Operating Costs		
Unit Operating Costs	**Units**	**Average LOM**
Mining	$/t ore Processed	3.93
Processing	$/t ore Processed	10.45
General & Administration	$/t ore Processed	0.89
Project Unit Operating Cost	**$/t ore Processed**	**15.27**

(1) Average LOM mining cost per ton excludes mining costs associated with the 27.6 M ore tons excavated during the preproduction period and capitalized.

(2) Total LOM ore tons mined 473.5 M.

(3) Total LOM ore tons processed 501.1 M.

22.3 Capital Cost Summary

Feasibility level designs defined the project to estimate the initial capital required to build the Project to an accuracy of plus or minus 15%. Capital estimates followed a work breakdown structure (WBS), defined to three levels. Mine Development Associates (MDA) designed the surface mine and developed a production schedule. MDA estimated the cost to prepare the surface mine for production and equip the mine to support the mill's throughput requirements. AMEC Environment and Infrastructure designed facilities for water management, tailings management, overburden and ore stockpiles, access roads and out-of-mine haul and access roads. Samuel Engineering designed the process plant based on the selected flow sheet and production schedule of 100,000 t/d, (90,718 mt/d). The process capital costs include provisions for plant and ancillary buildings, plant capital equipment, an operations camp to house employees, tailings and fresh water pipelines, contractor indirect costs, construction camp, EPCM and consultant costs, vendor representatives, start-up costs, spare parts and initial fills and freight. The model recaptures cost for spares and initial fills in Year 14. Dryden and LaRue, EPS and GVEA estimated the cost to provide power to site. MTB and THM estimated owner's costs including plant mobile equipment and light vehicles, communications systems, relocation of the Alaska DOT facilities, pre-production employment and training, corporate legal, environmental and permitting, and community development expenses. Each estimate contains contingency to account for unknown costs yielding an overall project contingency of 10.9%. The estimate assumes contingency will be spent. Table 22.4 summarizes the Livengood Gold Project's initial capital costs by WBS area.

Table 22.4
Summary of Initial Capital Costs (US $000)

WBS	Area	Cost
0100	Mine Area Facilities	122,614
0200	Crushing	83,523
0300	Grinding and Gravity Separation	199,663
0500	Leaching	187,971
0700	Carbon Stripping, Regeneration, Refining	12,138
0800	Reagents	23,284
0900	Thickening / Tailings	458,176
1000	Utilities	270,812
1100	Buildings	49,513
1200	Site Development and Roads	70,580
1300	Common Distributables	234,628
2000	Contracted Indirects	247,280
3000	Owner's Direct Cost	448,673
4000	Owner's Indirect Cost	78,138
9000	Contingency	270,538
	Total Cost	**2,757,531**

22.4 Sustaining Capital Costs

Sustaining capital expenditures result from acquiring assets, increasing facility capacities, or replacing assets during production (Years 1-14), totaling $666,850,000.

Processing sustaining capital costs include replacements for worn out equipment and costs for relocating tailings and other utility pipelines.

Geotechnical sustaining capital costs include expansions of the following facilities:

- Tailings Management Facility
- Gertrude Embankment, Overburden Storage Facility and Ore Stockpile
- Site Access and Mine Haul Roads
- Growth Media Stockpile

Geotechnical sustaining capital costs also include Year 14 closure costs, geotechnical construction indirect cost and geotechnical construction management.

Mining sustaining capital costs include purchasing additional capital equipment as the need arises, rolling stock, and other equipment over the life of mine.

Mobile and light vehicle sustaining capital costs provide for replacement of vehicles at the end of their useful life.

Table 22.5 summarizes the sustaining capital requirements over life of mine.

Table 22.5
Sustaining Capital

Description	Units	LOM
Process & Tailings		
Pipeline and Process Equipment Sustaining Capital	$(000)	26,401
Geotechnical Facilities		
Geotechnical Facilities	$(000)	505,604
Owner's Cost		
Mobile Equipment and Light Vehicles	$(000)	9,028
Mining		
Mine Rolling Stock	$(000)	125,817
Total(1)	**$(000)**	**666,850**

(1) Does not include Funding of Reclamation Trust Fund $226 million.

22.5 Closure Costs

AMEC developed a dry closure plan for the tailings management facility. Closure costs track reclamation and closure expenses from Year 15 through 45; the closure expenditure plans for a water treatment plant to be constructed in Year 14 and is classified as sustaining capital because it occurs during the mine life. The main closure construction effort occurs from Year 15 through 19, containing 70% of the overall closure costs. Costs for pumping and management operations are included in Years 19 through 45. Year 45 is the last year with planned closure expenses.

The total closure cost is $352,952,000. This total closure cost is applied to the cash flow in Year 14. This cost includes closure of the overburden stockpile, tailings management facility, solid waste landfill, and ancillary facilities, including indirect costs.

Closure cost funding will flow from a closure trust fund financed by mine cash flow. Annual contributions to the closure trust fund are included in the cash flow model. The annual contribution is $16,174,000 during Years -2 through 14. The model includes trust fund earnings at 4.0% annual percentage rate (APR), applied to the fund balance until closure is complete.

22.6 Working Capital

Working capital is the maximum funding required during the initial operating period to offset expenses prior to the cumulative revenue offsetting the cumulative expenses; that is, when the operation becomes self-sustaining in its cash flow. Working Capital is recovered at the end of the project.

The revenue was calculated on a weekly basis, using the amount of saleable product produced and the price, allowing for the following ramp-up which corresponds to the mine production schedule:

Quarter 1:	11.6% of 1st year production
Quarter 2:	26.7% of 1st year production
Quarter 3:	30.0% of 1st year production
Quarter 4:	31.7% of 1st year production
Total:	100.0% of 1st year production (80% of design capacity)

Revenue receipt was projected based on shipping and receipt of 85% of funds four weeks after the shipping date and the balance of 15% of funds received eight weeks after shipping doré.

Average weekly expenditure rates were calculated from the operating costs for Year 1. The average weekly expenditure of funds starts immediately in week one of Year 1.

The maximum cash flow deficiency would occur in week twelve, totaling $77,030,478. The model contains this working capital cost in Year 1, and recovers the equivalent amount in Year 14.

22.7 All-in Costs

The all-in costs to produce gold at Livengood total $1,030/oz before capital and $1,447/oz including capital. Taxes add another $27/oz for a total all-in cost of $1,474/oz. Production costs before capital represent 70% of the all-in cost while the capital expense represents 28% and taxes represent 2%.

The table below, Table 22.6, highlights the all-in operating cost of production over the life of the Project:

Table 22.6
All-in Cost of Production

	$/oz	LOM ($Million)
On-Site Mine Operating Costs	$933	$7,543
Royalties	45	362
Third-Party Smelting, Refining and Transport Costs	9	75
Sub-Total	987	7,980
Reclamation & Remediation	43	353
Sub-Total Production Cost Before Capital	1,030	8,333
Capital Expenditures (initial and sustaining) (1)	416	3,367
All-In Costs – Pre-Tax	1,447	11,700
Mining and Income Taxes	27	220
All-In Costs – After-Tax	$1,474	$11,920

Rounding of some figures may lead to minor discrepancies in totals.

(1) Excludes $32M upfront funding included in reclamation and remediation above and $57M of recoverable initial stores inventory.

22.8 Financial Analysis

The financial model uses the inputs from the entire Feasibility Study as its basis. The resulting revenue compared to capital and operating cost estimates summarized above yields a minimal positive return. The after tax payback period is 10.8 years.

The pre-tax internal rate of return (IRR) is 2.8% and the pre-tax net present value (NPV) using a 5% discount rate over the mine life is a negative $300,286,677.

The after tax IRR is 1.7%. The after tax NPV at a discount rate of 5% over the mine life is a negative $439,714,744. Table 22.7 presents pre-tax and after-tax NPVs at discount rates from 0% to 10%.

Table 22.7
Base Case Analysis

Discount Rate	0.0%	5.0%	7.5%	10.0%
Pre-Tax NPV ($000)	523,726,552	(300,286,677)	(551,788,498)	(734,636,264)
After Tax NPV ($000)	303,579,667	(439,714,744)	(665,341,341)	(828,460,873)

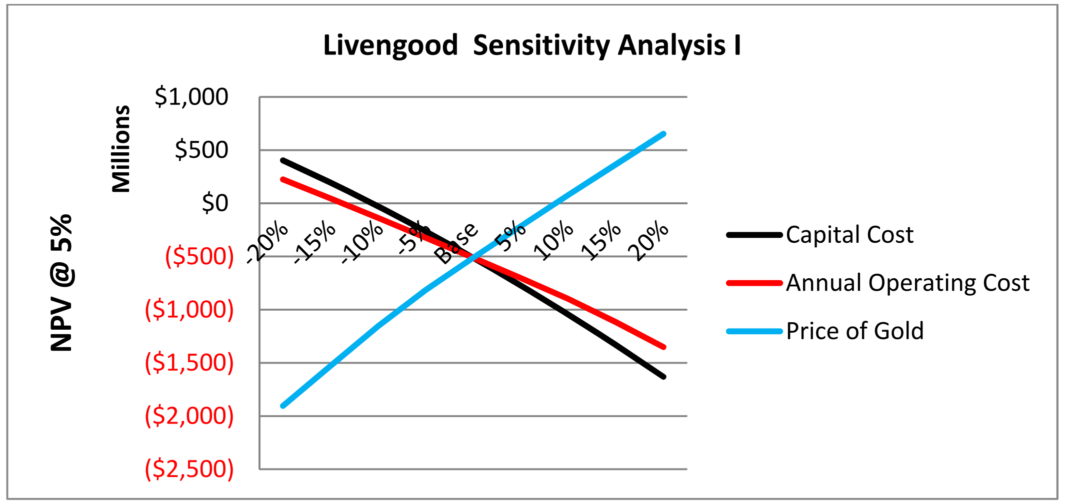
22.8.1 Sensitivity Analysis

The economic evaluation includes an analysis of the Project's sensitivity to key financial parameters. Sensitivity measures how much impact a change in a given parameter has on the base project value, all other factors remaining constant. Table 22.8 presents the after tax IRR and NPV(5) sensitivity results for varying recovery, gold price, total operating cost, and initial capital cost. Figures 22.1 to 22.4 present each sensitivity analysis graphically; steeper curves represent greater sensitivity.

This sensitivity analysis shows gold price and recovery variation cause the greatest impact on project value. A 20% increase in gold price would yield an 8.0% increase in IRR. A 20% decrease in gold price would yield an 17.8% reduction in IRR. The next most pronounced project sensitivity is to capital cost. Capital changes would drive marginally larger project returns than operating cost changes, meaning reducing capital expense would benefit the Project more than reducing operating costs by the same percentage.

Table 22.8
Livengood Sensitivity Analysis – After Tax IRR and NPV(5)

Base Case Variance	-20%	-15%	-10%	-5%	Base	5%	10%	15%	20%
Recovery		68%	72%	76%	80.3%	84.3%	88%	92%	
After Tax IRR		-8.9%	-4.2%	-0.9%	1.7%	3.9%	6.0%	7.9%	
After Tax NPV @ 5%		($1,459,434,684)	($1,087,021,372)	($740,683,532)	($439,714,744)	($147,194,638)	$143,565,855	$432,005,660	
Price of Gold	$1,200	$1,275	$1,350	$1,425	$1,500	$1,575	$1,650	$1,725	$1,800
After Tax IRR	-16.1%	-8.9%	-4.2%	-0.9%	1.7%	3.9%	6.0%	7.9%	9.7%
After Tax NPV @ 5%	($1,835,098,612)	($1,460,814,760)	($1,087,918,481)	($741,117,959)	($439,714,744)	($146,862,491)	$144,265,658	$433,034,632	$722,957,063
Annual Operating Cost	6,034,378,812	6,411,527,488	6,788,676,164	7,165,824,840	$7,542,973,515	7,920,122,191	8,297,270,867	8,674,419,543	9,051,568,218
After Tax IRR	7.0%	5.8%	4.5%	3.1%	1.7%	0.1%	-1.8%	-4.2%	-7.2%
After Tax NPV @ 5%	$297,868,649	$112,477,162	($71,327,795)	($254,948,240)	($439,714,744)	($627,190,205)	($833,572,824)	($1,055,440,019)	($1,288,808,774)
Capital Cost	1,790,698,318	2,019,812,036	2,262,713,409	2,519,402,438	$2,789,879,122	3,074,143,462	3,372,195,458	3,684,035,109	4,009,662,416
After Tax IRR	10.0%	7.5%	5.3%	3.4%	1.7%	0.1%	-1.3%	-2.5%	-3.7%
After Tax NPV @ 5%	$460,265,969	$253,898,100	$35,112,025	($196,092,257)	($439,714,744)	($695,755,438)	($964,214,339)	($1,245,091,445)	($1,538,386,758)



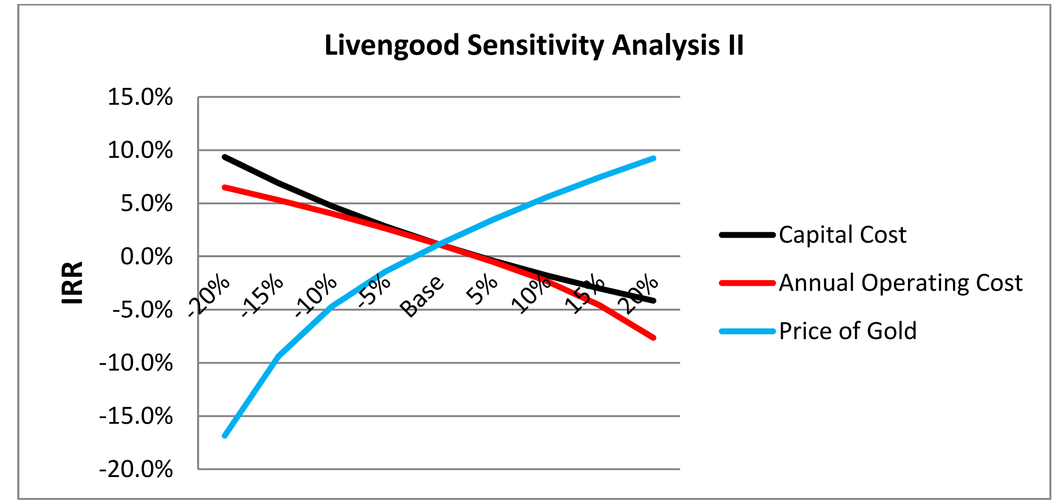
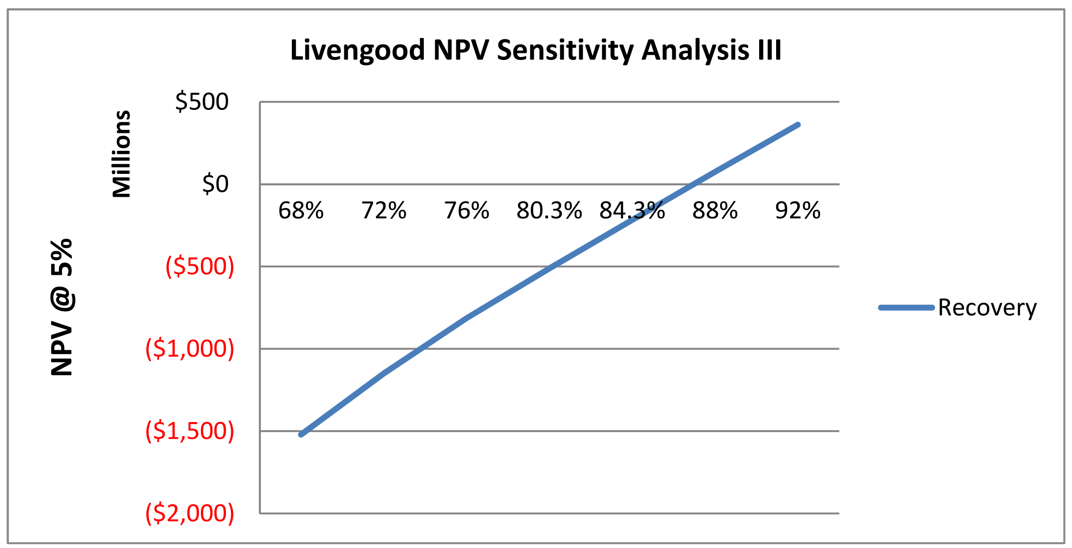


Figure 22.1 **After Tax Sensitivity Analysis for Capital Cost***
***Operating Cost and Price of Gold – NPV @ 5%**

Figure22.2 **After Tax Sensitivity Analysis for Capital Cost***
***Operating Cost and Price of Gold**



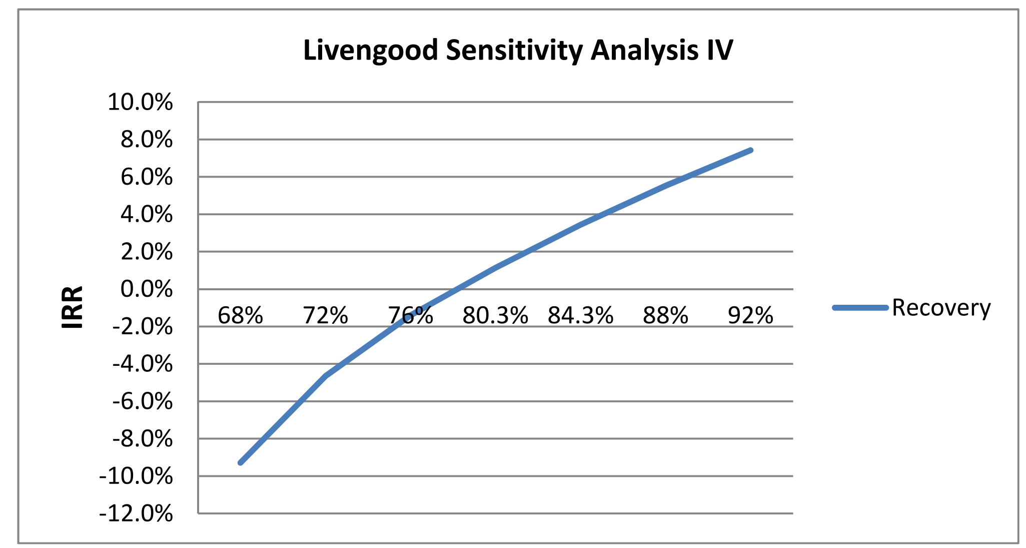


Figure 22.3 **After Tax Sensitivity Analysis for Recovery – NPV @ 5%**

Figure 22.4 **After Tax Sensitivity Analysis Recovery**

Gold price sensitivity – in $100/oz increments is shown in Table 22.9. This analysis indicates, given all other factors remain constant, the Project's value based on gold prices from $1,200/oz to $2,200/oz.

Table 22.9
Livengood Special Sensitivity Analysis – After Tax IRR and NPV(5)

Price of Gold	$1,200	$1,300	$1,400	$1,500	$1,600	$1,700	$1,800	$1,900	$2,000	$2,100	$2,200
After Tax IRR	-16.1%	-7.2%	-1.9%	1.7%	4.6%	7.3%	9.7%	12.0%	14.1%	16.0%	17.8%
After Tax NPV @ 5%	($1,835,098,612)	($1,336,289,755)	($853,779,863)	($439,714,744)	($49,710,670)	$336,384,540	$722,957,063	$1,109,126,533	$1,493,167,084	$1,869,459,328	$2,219,173,658
After Tax Payback Period Years	Beyond LOM	Beyond LOM	Beyond LOM	10.77	8.82	7.19	6.07	5.22	4.63	4.18	3.84

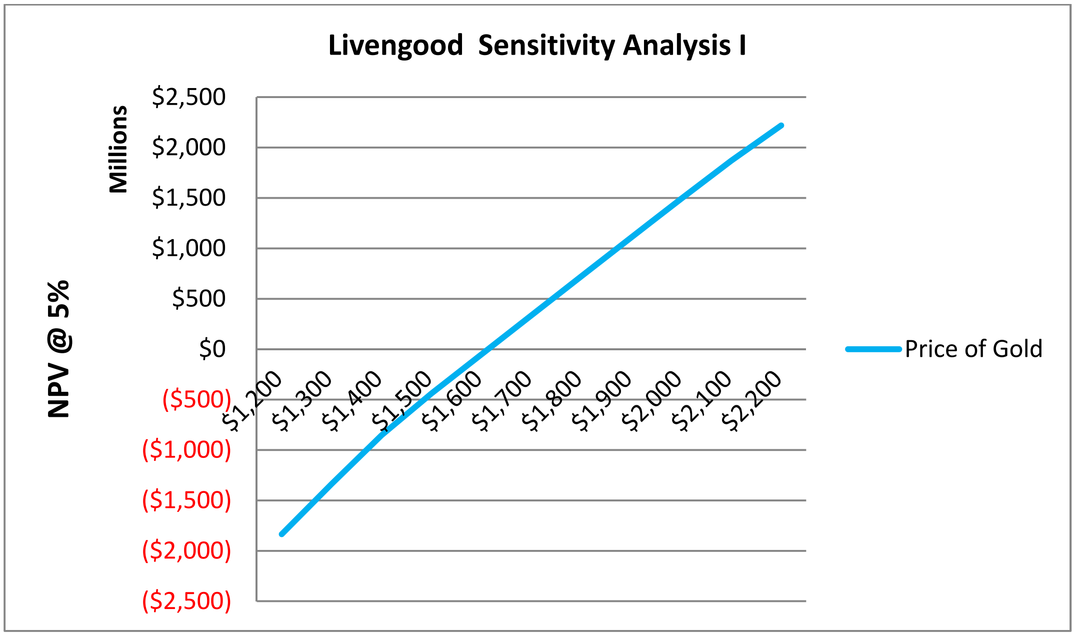


Figure 22.5 After Tax Sensitivity Analysis for Price of Gold

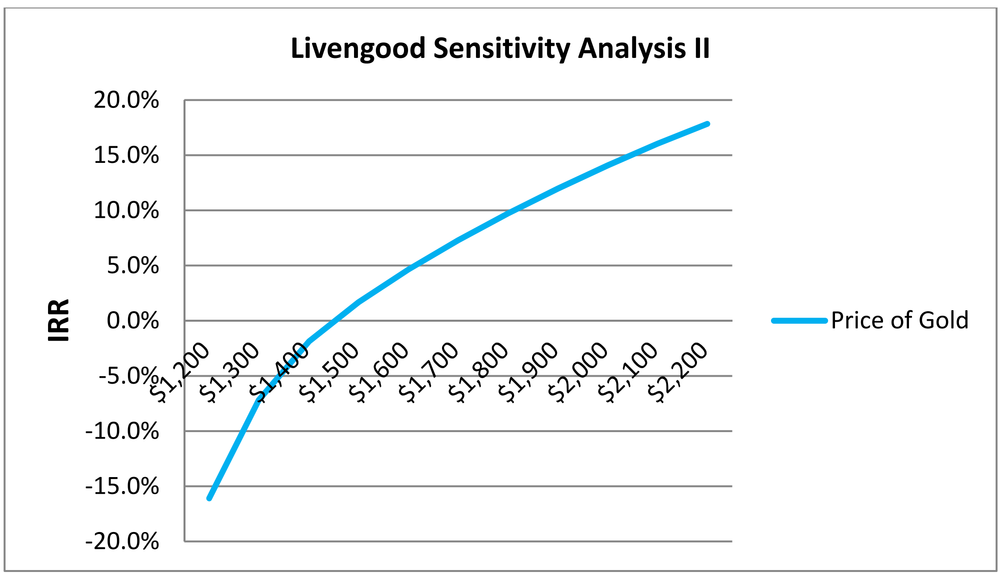


Figure 22.6 After Tax Sensitivity Analysis for Price of Gold

The two highest operating costs over the life of mine are power and cyanide. The Project's sensitivity to power cost is given in Figure 22.7 and 22.8 while cyanide sensitivity is given in Figure 22.9 and 22.10. These results show that the Project is slightly more sensitive to power costs than cyanide costs. A 20% reduction in power would yield a 1.2% improvement in IRR; a 20% reduction in cyanide cost would yield a 1.1% improvement in IRR.

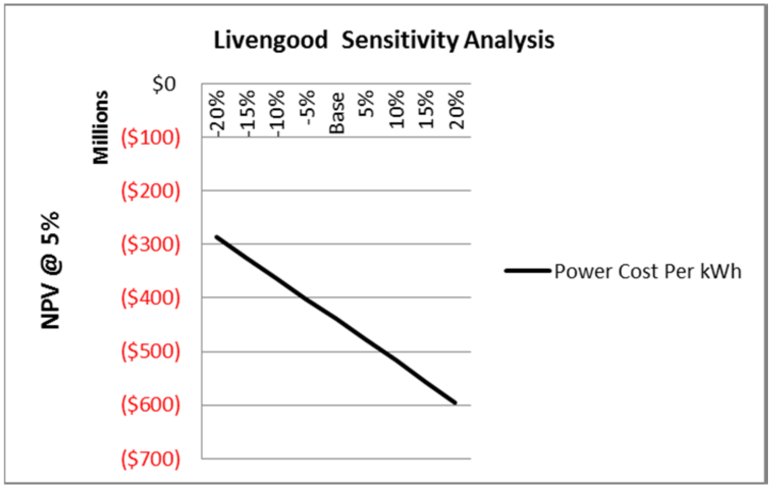


Figure 22.7 After Tax Sensitivity Analysis Power Cost

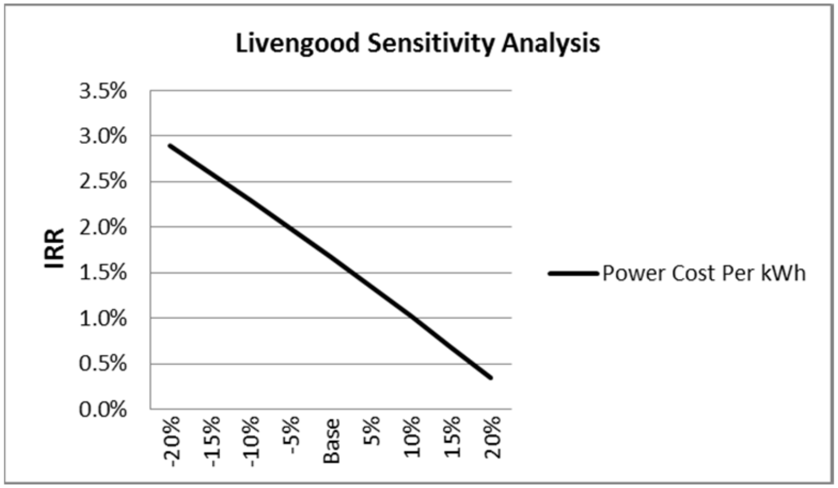


Figure 22.8 After Tax Sensitivity Analysis Power Cost

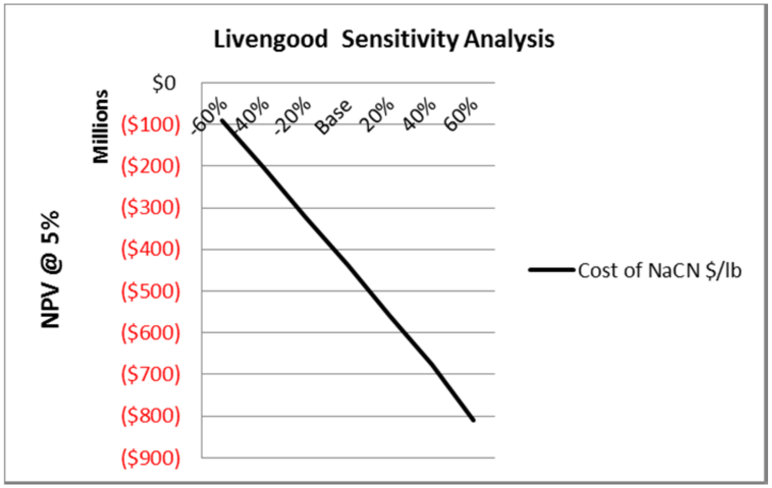


Figure 22.9 After Tax Sensitivity Sodium Cyanide Cost

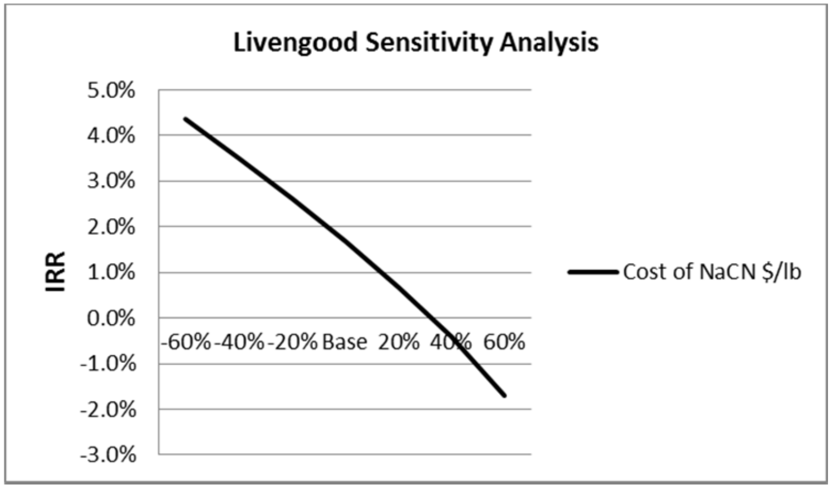


Figure 22.10 After Tax Sensitivity Sodium Cyanide Cost

22.8.2 Financial Model Cash Flow

Table 22.10
Livengood Feasibility Study Financial Model Cash Flow

Cash Flow Forecast
Tower Hill Mines, Inc.
Livengood Gold Project
Revision 2
July 18, 2013

	Units		Total or Average	Preproduction Year -3	Preproduction Year -2	Preproduction Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Production Summary	**Units**																			
Production Data																				
Ore Processed	Tons		501,136,789				29,325,742	36,713,533	36,613,293	36,613,284	36,613,300	36,713,548	36,613,317	36,613,315	36,613,303	36,713,632	36,613,212	36,613,273	36,613,270	32,150,768
Gross Income																				
Payable Metals																				
Recovered Au	Troy oz		8,086,373				763,177	844,156	594,038	671,348	619,743	558,273	590,252	582,288	554,171	562,911	720,733	421,611	321,423	282,247
Market Price																				
Au	$/oz	$ 1,500					1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Gross Revenue			**$ 12,129,558,858**				**$ 1,144,765,429**	**$ 1,266,234,500**	**$ 891,057,059**	**$ 1,007,022,725**	**$ 929,614,265**	**$ 837,409,973**	**$ 885,377,346**	**$ 873,432,308**	**$ 831,257,077**	**$ 844,367,249**	**$ 1,081,099,735**	**$ 632,417,007**	**$ 482,133,831**	**$ 423,370,354**
Refining Cost																				
Refining, Transportation and Insurance (Includes Payable Discount)	/oz	$ 9.30	$ (75,203,265)				(7,097,546)	(7,850,654)	(5,524,554)	(6,243,541)	(5,763,608)	(5,191,942)	(5,489,340)	(5,415,280)	(5,153,794)	(5,235,077)	(6,702,818)	(3,920,985)	(2,989,230)	(2,624,896)
NSR			**$ 12,054,355,593**				**$ 1,137,667,883**	**$ 1,258,383,846**	**$ 885,532,505**	**$ 1,000,779,184**	**$ 923,850,657**	**$ 832,218,031**	**$ 879,888,007**	**$ 868,017,028**	**$ 826,103,283**	**$ 839,132,172**	**$ 1,074,396,917**	**$ 628,496,021**	**$ 479,144,601**	**$ 420,745,458**
Production Royalty																				
Royalty on NSR	%	3.00	$ (361,630,668)				(34,130,036)	(37,751,515)	(26,565,975)	(30,023,376)	(27,715,520)	(24,966,541)	(26,396,640)	(26,040,511)	(24,783,098)	(25,173,965)	(32,231,908)	(18,854,881)	(14,374,338)	(12,622,364)
Gross Income			**$ 11,692,724,925**				**$ 1,103,537,847**	**$ 1,220,632,331**	**$ 858,966,530**	**$ 970,755,809**	**$ 896,135,137**	**$ 807,251,490**	**$ 853,491,367**	**$ 841,976,517**	**$ 801,320,184**	**$ 813,958,207**	**$ 1,042,165,009**	**$ 609,641,141**	**$ 464,770,263**	**$ 408,123,094**
Operating Cost																				
Unit Operating Costs																				
Mining	$/t ore Processed		$ 3.71				4.58	4.12	4.59	4.25	4.08	4.20	4.43	4.35	4.38	4.53	4.48	1.84	1.00	1.01
Processing	$/t ore Processed		$ 10.45				10.69	10.29	10.21	10.42	10.52	10.38	10.56	10.59	10.49	10.44	10.63	10.45	10.34	10.34
General & Administration	$/t ore Processed		$ 0.89				1.09	0.87	0.88	0.88	0.88	0.87	0.88	0.88	0.88	0.87	0.88	0.88	0.88	0.88
Project Unit Operating Cost	**$/t ore Processed**		**$ 15.05**				**16.36**	**15.28**	**15.68**	**15.54**	**15.47**	**15.46**	**15.86**	**15.81**	**15.74**	**15.84**	**15.98**	**13.17**	**12.21**	**12.22**
Project Unit Operating Cost	**$/Troy oz Au Produced**		**$ 932.80**				**628.57**	**664.68**	**966.22**	**847.72**	**914.17**	**1,016.57**	**983.87**	**994.19**	**1,040.23**	**1,033.13**	**811.91**	**1,143.30**	**1,391.18**	**1,392.36**
Mine Reclamation & Closure Cost	$/t ore Processed		$ 0.70																	
Annual Operating Costs																				
Mining	$		$ (1,861,590,070)				(134,308,140)	(151,201,814)	(168,177,449)	(155,541,177)	(149,456,781)	(154,370,279)	(162,163,475)	(159,291,354)	(160,253,600)	(166,398,935)	(163,898,634)	(67,443,421)	(36,606,770)	(32,478,240)
Processing	$		$ (5,236,646,116)				(313,356,883)	(377,847,430)	(373,747,556)	(381,531,042)	(385,045,094)	(381,106,711)	(386,519,856)	(387,567,940)	(384,163,652)	(383,116,962)	(389,226,383)	(382,539,414)	(378,505,091)	(332,372,101)
General & Administration	$		$ (444,737,330)				(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(32,045,940)	(28,140,114)
Reclamation & Closure Cost	$		$ (352,952,018)				-	-	-	-	-	-	-	-	-	-	-	-	-	(352,952,018)
Allocation of Closure Trust Funds			$ 352,952,018				-	-	-	-	-	-	-	-	-	-	-	-	-	352,952,018
Total Annual Operating Costs			**$ (7,542,973,515)**				**$ (479,710,962)**	**$ (561,095,184)**	**$ (573,970,944)**	**$ (569,118,158)**	**$ (566,547,815)**	**$ (567,522,929)**	**$ (580,729,271)**	**$ (578,905,234)**	**$ (576,463,192)**	**$ (581,561,837)**	**$ (585,170,957)**	**$ (482,028,775)**	**$ (447,157,801)**	**$ (392,990,456)**
Operating Income (Gross Income less Operating costs)			**$ 4,149,751,410**				**623,826,885**	**659,537,147**	**284,995,585**	**401,637,650**	**329,587,323**	**239,728,561**	**272,762,096**	**263,071,283**	**224,856,992**	**232,396,370**	**456,994,052**	**127,612,366**	**17,612,461**	**15,132,638**
Depreciation	$		$ (3,424,381,464)				(318,662,101)	(555,502,041)	(423,163,828)	(326,939,047)	(266,275,131)	(237,109,513)	(219,220,047)	(169,469,355)	(113,725,736)	(111,238,477)	(96,442,925)	(79,853,517)	(78,588,407)	(428,191,338)
Taxable Income Prior to Depletion			**$ 725,369,946**				**305,164,783**	**104,035,106**	**(138,168,243)**	**74,698,603**	**63,312,191**	**2,619,048**	**53,542,049**	**93,601,928**	**111,131,256**	**121,157,893**	**360,551,127**	**47,758,849**	**(60,975,946)**	**(413,058,700)**
Depletion	$		$ (620,215,148)				(149,653,352)	(50,659,501)	0	(36,963,009)	(31,053,029)	(1,284,577)	(25,997,713)	(45,090,028)	(52,988,738)	(55,251,148)	(148,266,775)	(23,007,278)	0	0
Taxable Income			**$ 105,154,798**				**155,511,431**	**53,375,605**	**(138,168,243)**	**37,735,594**	**32,259,162**	**1,334,471**	**27,544,336**	**48,511,900**	**58,142,518**	**65,906,744**	**212,284,353**	**24,751,571**	**(60,975,946)**	**(413,058,700)**
Taxes																				
Alaska State Income Tax	%	9.4	$ (49,220,699)				(11,716,158)	(5,432,208)	0	(242,182)	(203,459)	(8,417)	(1,263,497)	(3,697,075)	(5,899,374)	(6,647,741)	(14,035,216)	(75,372)	0	0
Alaska Mining License Tax	%	7	$ (31,182,479)				-	-	-	(1,302,987)	(2,208,806)	(91,372)	(1,829,749)	(3,146,670)	(3,683,116)	(4,007,855)	(13,243,002)	(1,668,922)	-	-
Alaska State Income and Mining Tax Credits			$ 20,000,000				5,858,079	2,716,104	-	772,585	1,206,133	49,894	1,546,623	3,421,872	4,428,710	-	-	-	-	-
Federal Income Tax	%	35	$ (159,743,708)				(37,070,432)	(18,440,058)	0	(1,345,454)	(1,130,330)	(46,759)	(7,019,429)	(16,412,770)	(19,287,901)	(20,111,418)	(38,460,425)	(418,732)	0	0
Net Income			**$ (114,992,088)**				**112,582,920**	**32,219,443**	**(138,168,243)**	**35,617,556**	**29,922,699**	**1,237,818**	**18,978,284**	**28,677,258**	**33,700,837**	**35,139,730**	**146,545,710**	**22,588,545**	**(60,975,946)**	**(413,058,700)**
Depreciation Add Back			$ 3,424,381,464				318,662,101	555,502,041	423,163,828	326,939,047	266,275,131	237,109,513	219,220,047	169,469,355	113,725,736	111,238,477	96,442,925	79,853,517	78,588,407	428,191,338
Depletion Add Back			$ 620,215,148				149,653,352	50,659,501	0	36,963,009	31,053,029	1,284,577	25,997,713	45,090,028	52,988,738	55,251,148	148,266,775	23,007,278	0	0
Net Profit After Tax			**$ 3,929,604,524**				**580,898,374**	**638,380,984**	**284,995,585**	**399,519,612**	**327,250,860**	**239,631,908**	**264,196,043**	**243,236,641**	**200,415,311**	**201,629,356**	**391,255,410**	**125,449,340**	**17,612,461**	**15,132,638**
Capital Cost																				
Initial Capital Costs (less Spare Parts/Consumables/Initial Fills)	$		$ (2,700,390,516)	(81,011,715)	(1,377,199,163)	(1,242,179,637)														
Spare Parts/Consumables/Initial Fills with Contingency	$		$ (57,140,606)			(57,140,606)														
Funding of Closure Trust Fund (pre-production)			$ (32,348,000)		(16,174,000)	(16,174,000)														
Total Initial Capital Costs			**$ (2,789,879,122)**	**$ (81,011,715)**	**$ (1,393,373,163)**	**$ (1,315,494,244)**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**
Funding of Closure Trust Fund	$		$ (226,436,000)				(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)	(16,174,000)
Recapture of Spare Parts/Consumables/Initial Fills	$		$ 57,140,606																	57,140,606
Sustaining Capital	$		$ (666,850,342)				(120,776,422)	(74,832,067)	(92,229,794)	(47,941,248)	(51,218,280)	(38,919,465)	(66,412,143)	(52,833,474)	(6,678,662)	(57,192,911)	(41,882,565)	(1,523,161)	(14,410,150)	-
Working Capital	$		$ (77,030,478)				(77,030,478)													
Working Capital Recapture	$		$ 77,030,478																	77,030,478
ANNUAL CASH FLOW (after tax)			**$ 303,579,667**	**$ (81,011,715)**	**$ (1,393,373,163)**	**$ (1,315,494,244)**	**$ 366,917,474**	**$ 547,374,917**	**$ 176,591,791**	**$ 335,404,364**	**$ 259,858,580**	**$ 184,538,443**	**$ 181,609,900**	**$ 174,229,166**	**$ 177,562,649**	**$ 128,262,445**	**$ 333,198,845**	**$ 107,752,179**	**$ (12,971,689)**	**$ 133,129,723**
CUMULATIVE CASH FLOW (after tax)				**$ (81,011,715)**	**$ (1,474,384,879)**	**$ (2,789,879,122)**	**$ (2,422,961,648)**	**$ (1,875,586,731)**	**$ (1,698,994,939)**	**$ (1,363,590,575)**	**$ (1,103,731,995)**	**$ (919,193,552)**	**$ (737,583,652)**	**$ (563,354,485)**	**$ (385,791,836)**	**$ (257,529,391)**	**$ 75,669,453**	**$ 183,421,632**	**$ 170,449,944**	**$ 303,579,667**
ANNUAL CASH FLOW (pre-tax)			**$ 523,726,552**	**$ (81,011,715)**	**$ (1,393,373,163)**	**$ (1,315,494,244)**	**$ 409,845,985**	**$ 568,531,080**	**$ 176,591,791**	**$ 337,522,402**	**$ 262,195,043**	**$ 184,635,096**	**$ 190,175,953**	**$ 194,063,809**	**$ 202,004,330**	**$ 159,029,459**	**$ 398,937,487**	**$ 109,915,205**	**$ (12,971,689)**	**$ 133,129,723**
CUMULATIVE CASH FLOW (pre-tax)				**$ (81,011,715)**	**$ (1,474,384,879)**	**$ (2,789,879,122)**	**$ (2,380,033,137)**	**$ (1,811,502,058)**	**$ (1,634,910,266)**	**$ (1,297,387,864)**	**$ (1,035,192,822)**	**$ (850,557,726)**	**$ (660,381,773)**	**$ (466,317,964)**	**$ (264,313,634)**	**$ (105,284,175)**	**$ 293,653,313**	**$ 403,568,518**	**$ 390,596,829**	**$ 523,726,552**

After tax	IRR	1.7% Middle of Period
	Disc	NPV
	0.0%	$ 303,579,667
	5.0%	$ (439,714,744)
	7.5%	$ (665,341,341)
	10.0%	$ (828,460,873)

Pre - Tax	IRR	2.8% Middle of Period
	Disc	NPV
	0.0%	$ 523,726,552
	5.0%	$ (300,286,677)
	7.5%	$ (551,788,498)
	10.0%	$ (734,636,264)

Payback Period After Tax	10.77 Years

23.0 Adjacent Properties

Bluestone controls the adjacent ground to the south of the Livengood project, and has sporadically been exploring it for gold mineralization in the past.

The Alaska Pipeline, the main means of transporting crude oil from Alaska's North Slope to the south coast of Alaska, runs northwest-southwest about 6 km to the west. This feature is not expected to have any impact on the project.

24.0 Other Relevant Data and Information

24.1 Plan of Execution

The purpose of the Plan of Execution (POE) is to provide a comprehensive plan for the development and implementation of the Livengood Gold Project. The POE addresses the overall Project including objectives, scope and strategies. The POE provides a tactical plan for engineering, procurement, construction, start-up and commissioning of the plant facilities and infrastructure and addresses roles and responsibilities, and management plans required to execute and manage the work. This feasibility execution plan will require refinement during the permit support engineering, the basic engineering and/or detailed engineering stages.

24.2 Project Schedule

Due to anticipated site and climatic conditions, it was determined that October was the most opportune starting time for construction. Therefore the project schedule was developed to support site earthwork beginning on October 1 with completion of construction 27 months later.

24.2.1 Summary

The project schedule shows summary level detail of project development including permit support engineering, basic engineering, detail engineering, major procurement, construction, start-up and commissioning. Schedule development involved identification of key activities, their durations, and proper Gantt logic ties required to determine the critical path, finish date and assess float. Activities considered low-risk or insignificant have been purposefully excluded from the schedule.

The schedule assumes a 29-month overall construction schedule. The critical path on the process plant runs through the grinding area and the tailings impoundment facilities.

The summary project schedule is shown below.

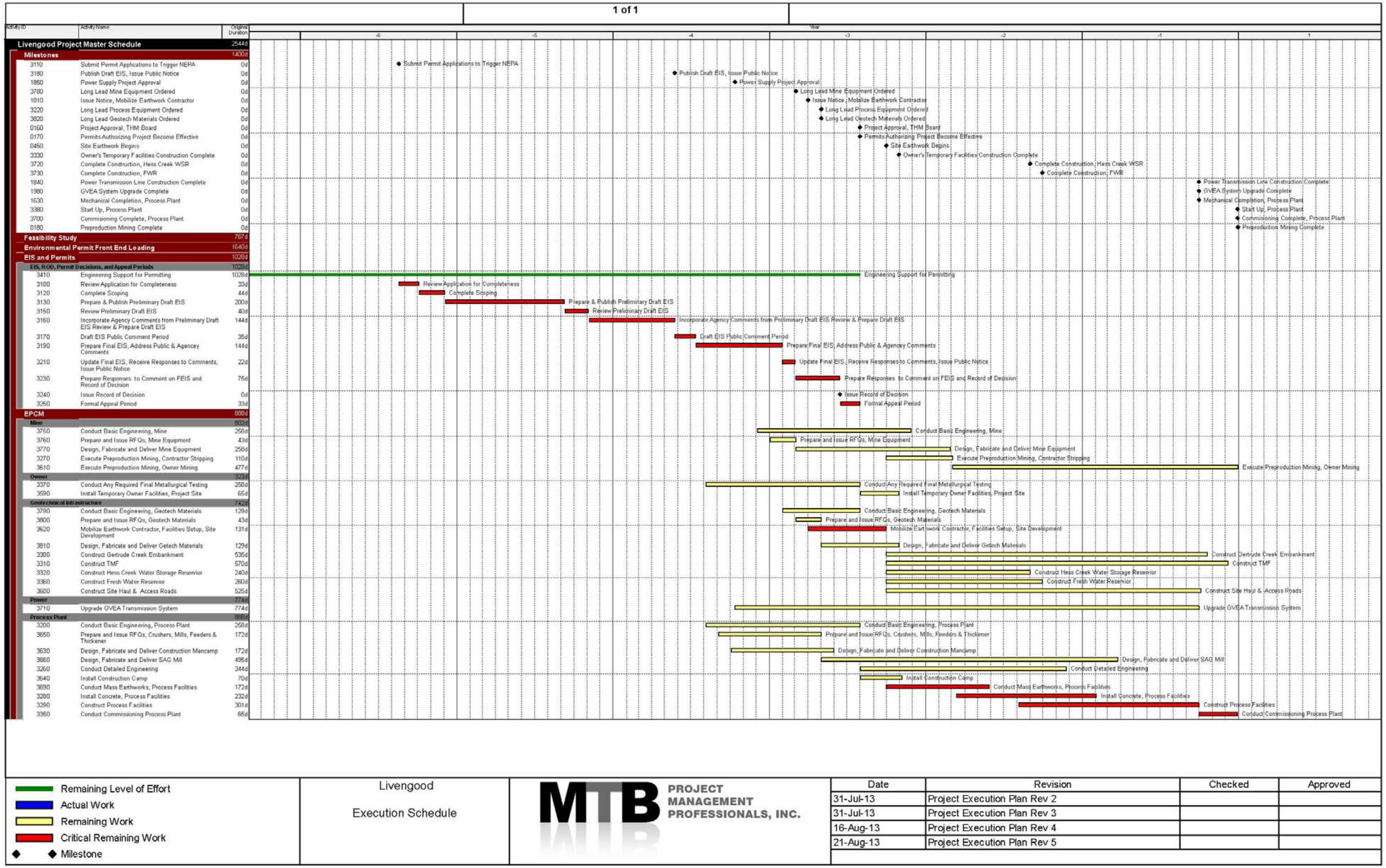


Figure 24.1 Project Schedule

25.0 Interpretations and Conclusions

25.1 Conclusions

- The Livengood Gold Project mineral resource is estimated at 731 million measured tonnes at an average grade of 0.61 g/mt (14.4 million oz at 0.3 g/mt cut-off) and 71 million indicated tonnes at an average grade of 0.56 g/mt (1.3 million oz at 0.3 g/mt cut-off), for a total of 802 million tonnes at an average grade of 0.61 g/mt (15.7 million ounces at 0.3 g/mt cut-off).
- The FS has converted a portion of these mineral resources into proven reserves of 434 million tonnes at an average grade of 0.689 g/mt (9,621,500 oz) and probable reserves of 20.1 million tonnes at an average grade of 0.702 g/mt (454,000 oz), for a total of 454 million tonnes at an average grade of 0.689 g/mt (10,075,600 oz).
- The FS mine plan would provide sufficient ore to support an annual production rate of approximately 577,600 ounces per year over an estimated 14 year mine life, producing approximately 8 million ounces.
- Metallurgical testwork has identified the preferred flowsheet of gravity recovery followed by whole ore leaching of the gravity tailing for an overall LOM recovery of 80.3%.
- The initial capital cost of a 100,000 t/d mill and associated 234,000 t/d mine is estimated at $2.79 billion.
- The mining cost is estimated at $1.67/t mined, process operating cost is estimated at an average of $10.45/t ore processed, and general and administrative costs of $0.89/ton ore processed.
- Using the trailing three year gold price of $1,500 per ounce, the project generates a minimal positive return.

25.2 Risks

Although it is the judgment of the authors of this study that the project can be completed as designed, the following risks have been identified and need to be managed appropriately:

- Large earthworks quantity

The Livengood Gold Project requires excavation, processing, movement, placement, and preparation of a large quantity of soil, colluvium, alluvial material, and rock, including construction of multiple engineered structures. There is a risk that the contractors' and owner's crews and equipment may not be able to move this material as efficiently as estimated. The result could have significant negative implications to both the execution schedule and project cost.

- Unknown subsurface conditions

The Livengood Gold Project has a large surface footprint. While subsurface ground conditions have been investigated by drilling in support of this Feasibility Study the actual subsurface ground conditions encountered during construction may be different than currently understood. The result could have significant negative implications to both the execution schedule and cost.

- Large area of liner installation

The Livengood Gold Project will require the surface preparation and placement of approximately 38 Mft^2 of LLDPE liner at the Tailings Management Facility during the two planned summer construction seasons prior to production. There is risk that the contractor may not be able to place the quantity of liner required in the time available. The result could have significant negative implications to both the execution schedule and cost.

- Seasonal sensitivity of project release date

The Feasibility Study execution plan assumes an August 1 mobilization date, with construction to begin on October 1. The actual project release date is uncertain, given the combination of market variables and the multi-year permitting process that must be completed prior to a construction decision. There is a risk that a project release date that is substantially different than August 1 could have negative implications to both the execution schedule and project cost.

26.0 Recommendations

- The optimized final pit contains over 44 Mmt of inferred material that is above cutoff grade. Additional drilling may improve the classification of this material. The cost of this work is estimated to be $5 million.
- The optimized final pit extends to the bottom of the current grade model. It is apparent that deeper drilling is warranted to develop material below the current grade model bottom. It is recommended that this work be completed mid-way through the life of the mine.
- Metallurgical testing has consistently shown higher calculated head grades compared to the average assay obtained from composited drill core assays that make up the metallurgical test samples. This result is consistent with the bi-modal size distribution of the gold in the Livengood deposit. An extensive check assay program by metallic screen assays also showed small gains after adjusting for sample distribution; however the average grade of the metallic screen was about the same as the average fire assay. In this series of tests the metallic assay sample may have been too small, and contained more samples of higher grade materials than the average distribution in the database. More follow up is suggested as there is a significant amount of information that suggests the drill hole assays may be 10-15% lower than the actual grade. The cost of this work is estimated to be $1 million.
- After the flow sheet was fixed for the purposes of the feasibility study, additional analysis by Knelson suggested that a 1-3% improvement in overall gold recovery may be achievable if an intensive cyanide leach reactor is used in place of the shaking tables contemplated in the study. There are potential space savings and operational improvements associated with use of a reactor, which together with the potential recovery improvement, warrant further study. It is recommended that this work would cost approximately $200,000 and should be deferred until detailed engineering.
- Pursue mill throughput and capital cost studies to evaluate the optimum scale for the project. This would require feasibility level work at an estimated cost of $2 million dollars.
- There is an opportunity to enhance mill head grades in early years by a more aggressive stockpile management strategy than is assumed in the feasibility study. The additional production schedule optimization would cost approximately $50,000.

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28.0 Date and Signature Page

The effective date of the technical report, "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood, Alaska" is September 4, 2013.

Original signature and seal on file

Richard S. Kunter, FAusIMM (CP), Q.P. MMSA

Original signature and seal on file

Charles C. Rehn, P.E.

Original signature and seal on file

Neil Prenn, P.E.

Original signature and seal on file

Tim Carew, P. Geo.

Original signature and seal on file

Michael Levy, P.E.

29.0 Certificates of Qualified Persons

The effective date of the technical report, "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood, Alaska" is September 4, 2013.

AUTHOR'S CERTIFICATE

I, Charles C. Rehn, P.E., do hereby certify that:

1. I am a Principal of AMEC Environment and Infrastructure with an office at:

   AMEC Environment and Infrastructure
   9856 South 500 West
   Salt Lake City, Utah 84070 USA

2. I graduated from Iowa State University with a B.S. in Civil Engineering in 1975.

3. I am a registered Professional Engineer with the state of Utah (#7261612).

4. I have practiced my profession continuously for 38 years and, during that period, have been involved in geotechnical and civil engineering on similar projects in Minnesota, Colorado, Missouri, Alberta, Idaho, Montana, Alaska, Oregon, Washington, California, Nevada, New Mexico, Arizona and the Philippines. These projects were all mining related projects. These projects included tailing dam design ranging in height from 50 to over 500 feet in height of which several were built over very soft soils and rugged climate conditions similar in nature to this project. In the late 1970's I worked on one of the first tailing dam projects in the United States utilizing geo membranes instead of conventional compacted clay liners. There were multiple projects where I performed foundation design for mill sites and associated structures for both precious metal and coal mines. I have designed waste rock facilities for mines where the height of the piles has exceeded 700 feet. I have worked on multiple projects involving the new design of dams for water storage to be used as process water for mine facilities. I have worked on engineering projects in the state of Alaska.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of sections, 18, 20 and 21, and the relevant portions of Sections 1, 2, 25 and 26, of the technical report titled "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood Alaska", dated September 4, 2013 (the "Technical Report").

7. I personally visited the property that is the subject of the Technical Report on October 8-11, 2012.

8. Prior to being retained by International Tower Hill Mines Ltd. in connection with the preparation of the Technical Report I have not had prior involvement with the property that is the subject of the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read NI 43-101, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated this 4th of September, 2013

/Original signature and seal on file/

"*Charles C. Rehn*"

Charles C. Rehn, P.E.

AUTHOR'S CERTIFICATE

I, Timothy J. Carew, P. Geo., do hereby certify that:

1. I am an independent consultant with an office at:

Reserva International LLC
P.O. Box 19848
Reno, NV 89511 USA

2. I graduated from the following institutions:

1.	University of Rhodesia, B.Sc. Geology	1973
2.	University of Rhodesia, B.Sc. (Hons) Geology	1976
3.	University of London (RSM), M.Sc. Mineral Prod. Management	1982

3. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (Professional Geoscientist 19706), and the Institute of Mining, Metallurgy and Materials (Professional Member 46233).

4. I have practiced my profession continuously for 35 years and, during that period, have been involved in geologic work in similar lithotechtonic terranes (Cassiar, northern British Columbia) and resource estimation of vein and disseminated type gold deposits in the U.S. (Florida Canyon, Nevada), South America (Nassau, Suriname) and Asia (Boroo, Mongolia).

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 19 and 23, and the relevant portions of Sections 1, 2, 25 and 26, of the technical report titled "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood Alaska", dated September 4, 2013 (the "Technical Report").

7. I personally visited the property that is the subject to the Technical Report on six occasions, most recently in May, 2012.

8. Prior to being retained by International Tower Hill Mines Ltd. in 2009 in connection with the preparation of an NI 43-101 report on the property which is the subject of the Technical Report, I had not had any prior involvement with the property that is the subject of the Technical Report. Since 2009, I have participated in the preparation of three NI 43-101 technical reports on the property, including the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and

technical information that is required be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read NI 43-101, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated this 4th of September, 2013

/Original signature and seal on file/

"*Tim Carew*"

Timothy J. Carew, P. Geo.

AUTHOR'S CERTIFICATE

I, Neil Prenn, P.E., do hereby certify that:

1. I am a Mining Engineer practicing at Mine Development Associates with an office at:

Mine Development Associates

210 South Rock Boulevard

Reno, Nevada 89502 USA

2. I graduated from the Colorado School of Mines with an Engineer of Mines degree in 1967.

3. I am a registered Professional Engineer with the state of Nevada (#7844).

4. I have practiced my profession continuously for 46 years and, during that period, have been involved in completing numerous resource and reserve calculations for 16 years with Cyprus Mines Corporation, two years with California Silver, and 24 years with Mine Development Associates, Inc.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of sections 15, 16.1, 16.2 and 16.4 -16.10, and the relevant portions of Sections 1, 2, 25 and 26, of the technical report titled "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood Alaska", dated September 4, 2013 (the "Technical Report").

7. I personally visited the property that is the subject to the Technical Report on October 9-10, 2012.

8. Prior to being retained by International Tower Hill Mines Ltd. in connection with the preparation of the Technical Report I have not had prior involvement with the property that is the subject of the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

11. I have read NI 43-101, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated this 4th of September, 2013

/Original signature and seal on file/

"*Neil Prenn***"**

Neil Prenn, P.E.

AUTHOR'S CERTIFICATE

I, Richard S. Kunter, FAusIMM (CP), QP MMSA, do hereby certify that:

1. I am a Principal of AMEC Environment and Infrastructure with an office at:

   Samuel Engineering, Inc.
   8450 E Crescent Pkwy., Ste. 200
   Greenwood Village, CO 80111

2. I graduated from the University of Idaho with a BS in Metallurgical Engineering in 1967 and an MS in Metallurgical Engineering in 1969.

3. I am a Chartered Professional Fellow of the Australasian Institute of Mining and Metallurgy (No. 100346) and a Qualified Professional of the Mining and Metallurgical Society of America (No. 01217QP).

4. I have practiced my profession continuously for 42 years and, during that period, have been involved in the preparation and review of technical and/or competent person's reports, metallurgical recovery from mineral resources and other similar reports and studies on various properties domestically and internationally during the past 22 years. Prior to that I have held operating and technical positions in the mining and process industries for Western Mining Corporation in Australia as Research and Process Metallurgist, Newmont Mining as Mill Superintendent at Telfer Gold, Homestake Mining Company as Senior Corporate Metallurgist, Artech Recovery Systems as VP Technical, and Advanced Science as Project Manager, in aggregate covering 20 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of sections 3, 13, 17, 22, and 24, and the relevant portions of Sections 1, 2, 25 and 26, of the technical report titled "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood Alaska", dated September 4, 2013 (the "Technical Report").

7. I personally visited the property that is the subject of the Technical Report on October 9-10, 2012.

8. Prior to being retained by International Tower Hill Mines Ltd. in connection with the preparation of the Technical Report I have not had prior involvement with the property that is the subject of the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contains all scientific and

technical information that is required be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read NI 43-101, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated this 4th of September, 2013

/Original signature and seal on file/

"Richard S. Kunter"

Richard S. Kunter, FAusIMM (CP), QP MMSA

AUTHOR'S CERTIFICATE

I, Michael Levy, P.E., do hereby certify that:

1. I am Senior Geotechnical Engineer with SRK Consulting, Inc. with an office at:

   SRK Consulting, Inc.
   7175 W Jefferson Ave.
   Lakewood, CO 80235

2. I graduated from the University of Iowa with a B. Sc. In Geology in 1998 and a M.Sc. in Civil-Geotechnical Engineering in 2004.

3. I am a registered Professional Engineer with the states of Colorado (#40268) and California (#70578) and a registered Professional Geologist with the state of Wyoming (#3550).

4. I have practiced my profession continuously for 15 years and, during that period, have been involved in a variety of geotechnical projects specializing in advanced analysis and design of soil and rock slopes.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of section 16.3 of the technical report titled "Canadian National Instrument 43-101 Technical Report on the Livengood Gold Project Feasibility Study, Livengood Alaska", dated September 4, 2013 (the "Technical Report").

7. I personally visited the property that is the subject of the Technical Report on June 20 – 22, 2012.

8. Prior to being retained by International Tower Hill Mines Ltd. in connection with the preparation of the Technical Report I have not had prior involvement with the property that is the subject of the Technical Report.

9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read NI 43-101, and the portions of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated this 4th of September, 2013

/Original signature and seal on file/

"Michael *Levy*"

Michael Levy, P.E.

30.0 APPENDICES

30.1 State of Alaska Claims – 100% Owned

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	330936	LUCKY 55	F009N004W33	40	1
Tower Hill Mines, Inc.	330937	LUCKY 56	F009N004W33	40	2
Tower Hill Mines, Inc.	330938	LUCKY 64	F009N004W32 F009N004W33	40	3
Tower Hill Mines, Inc.	330939	LUCKY 65	F009N004W33	40	4
Tower Hill Mines, Inc.	330940	LUCKY 66	F009N004W33	40	5
Tower Hill Mines, Inc.	330941	LUCKY 72	F008N004W05	40	6
Tower Hill Mines, Inc.	330942	LUCKY 73	F008N004W05	40	7
Tower Hill Mines, Inc.	330943	LUCKY 74	F008N004W05	40	8
Tower Hill Mines, Inc.	330944	LUCKY 75	F008N004W04	40	9
Tower Hill Mines, Inc.	330945	LUCKY 76	F008N004W04	40	10
Tower Hill Mines, Inc.	330946	LUCKY 82	F008N004W05	40	11
Tower Hill Mines, Inc.	330947	LUCKY 83	F008N004W05	40	12
Tower Hill Mines, Inc.	330948	LUCKY 84	F008N004W05	40	13
Tower Hill Mines, Inc.	330949	LUCKY 85	F008N004W04	40	14
Tower Hill Mines, Inc.	330950	LUCKY 86	F008N004W04	40	15
Tower Hill Mines, Inc.	330951	LUCKY 91	F008N004W05	40	16
Tower Hill Mines, Inc.	330952	LUCKY 92	F008N004W05	40	17
Tower Hill Mines, Inc.	330953	LUCKY 93	F008N004W05	40	18
Tower Hill Mines, Inc.	330954	LUCKY 94	F008N004W05	40	19
Tower Hill Mines, Inc.	330955	LUCKY 95	F008N004W04	40	20
Tower Hill Mines, Inc.	330956	LUCKY 96	F008N004W04	40	21
Tower Hill Mines, Inc.	330957	LUCKY 101	F008N004W05	40	22
Tower Hill Mines, Inc.	330958	LUCKY 102	F008N004W05	40	23
Tower Hill Mines, Inc.	330959	LUCKY 103	F008N004W05	40	24
Tower Hill Mines, Inc.	330960	LUCKY 104	F008N004W05	40	25
Tower Hill Mines, Inc.	330961	LUCKY 105	F008N004W04	40	26
Tower Hill Mines, Inc.	330962	LUCKY 106	F008N004W04	40	27
Tower Hill Mines, Inc.	330963	LUCKY 202	F008N004W08	40	28
Tower Hill Mines, Inc.	330964	LUCKY 203	F008N004W08	40	29
Tower Hill Mines, Inc.	330965	LUCKY 204	F008N004W08	40	30
Tower Hill Mines, Inc.	330966	LUCKY 205	F008N004W09	40	31
Tower Hill Mines, Inc.	330967	LUCKY 206	F008N004W09	40	32
Tower Hill Mines, Inc.	330968	LUCKY 207	F008N004W09	40	33
Tower Hill Mines, Inc.	330969	LUCKY 208	F008N004W09	40	34
Tower Hill Mines, Inc.	330970	LUCKY 302	F008N004W08	40	35
Tower Hill Mines, Inc.	330971	LUCKY 303	F008N004W08	40	36
Tower Hill Mines, Inc.	330972	LUCKY 304	F008N004W08	40	37
Tower Hill Mines, Inc.	330973	LUCKY 305	F008N004W09	40	38
Tower Hill Mines, Inc.	330974	LUCKY 306	F008N004W09	40	39
Tower Hill Mines, Inc.	330975	LUCKY 307	F008N004W09	40	40
Tower Hill Mines, Inc.	330976	LUCKY 308	F008N004W09	40	41
Tower Hill Mines, Inc.	330977	LUCKY 404	F008N004W08	40	42
Tower Hill Mines, Inc.	330978	LUCKY 405	F008N004W09	40	43
Tower Hill Mines, Inc.	330979	LUCKY 406	F008N004W09	40	44
Tower Hill Mines, Inc.	338477	LUCKY 198	F008N004W07	40	45

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	338478	LUCKY 199	F008N004W07	40	46
Tower Hill Mines, Inc.	338479	LUCKY 295	F008N005W12	40	47
Tower Hill Mines, Inc.	338480	LUCKY 296	F008N005W12	40	48
Tower Hill Mines, Inc.	338481	LUCKY 297	F008N004W07	40	49
Tower Hill Mines, Inc.	338482	LUCKY 298	F008N004W07	40	50
Tower Hill Mines, Inc.	338483	LUCKY 299	F008N004W07	40	51
Tower Hill Mines, Inc.	338484	LUCKY 392	F008N005W11	40	52
Tower Hill Mines, Inc.	338485	LUCKY 395	F008N005W12	40	53
Tower Hill Mines, Inc.	338486	LUCKY 396	F008N005W12	40	54
Tower Hill Mines, Inc.	338487	LUCKY 397	F008N004W07	40	55
Tower Hill Mines, Inc.	338488	LUCKY 398	F008N004W07	40	56
Tower Hill Mines, Inc.	338489	LUCKY 399	F008N004W07	40	57
Tower Hill Mines, Inc.	338490	LUCKY 400	F008N004W07 F008N004W08	40	58
Tower Hill Mines, Inc.	338491	LUCKY 491	F008N005W11	40	59
Tower Hill Mines, Inc.	338492	LUCKY 492	F008N005W11	40	60
Tower Hill Mines, Inc.	338493	LUCKY 493	F008N005W12	40	61
Tower Hill Mines, Inc.	338494	LUCKY 494	F008N005W12	40	62
Tower Hill Mines, Inc.	338495	LUCKY 495	F008N005W12	40	63
Tower Hill Mines, Inc.	338496	LUCKY 496	F008N005W12	40	64
Tower Hill Mines, Inc.	338497	LUCKY 497	F008N004W07	40	65
Tower Hill Mines, Inc.	338498	LUCKY 498	F008N004W07	40	66
Tower Hill Mines, Inc.	338499	LUCKY 499	F008N004W07	40	67
Tower Hill Mines, Inc.	338500	LUCKY 500	F008N004W07 F008N004W08	40	68
Tower Hill Mines, Inc.	338501	LUCKY 504	F008N004W08	40	69
Tower Hill Mines, Inc.	338502	LUCKY 505	F008N004W09	40	70
Tower Hill Mines, Inc.	338503	LUCKY 589	F008N005W14	40	71
Tower Hill Mines, Inc.	338504	LUCKY 590	F008N005W14	40	72
Tower Hill Mines, Inc.	338505	LUCKY 591	F008N005W14	40	73
Tower Hill Mines, Inc.	338506	LUCKY 592	F008N005W14	40	74
Tower Hill Mines, Inc.	338507	LUCKY 593	F008N005W13	40	75
Tower Hill Mines, Inc.	338508	LUCKY 594	F008N005W13	40	76
Tower Hill Mines, Inc.	338509	LUCKY 595	F008N005W13	40	77
Tower Hill Mines, Inc.	338510	LUCKY 596	F008N005W13	40	78
Tower Hill Mines, Inc.	338511	LUCKY 597	F008N004W18	40	79
Tower Hill Mines, Inc.	338512	LUCKY 598	F008N004W18	40	80
Tower Hill Mines, Inc.	338513	LUCKY 599	F008N004W18	40	81
Tower Hill Mines, Inc.	338514	LUCKY 689	F008N005W14	40	82
Tower Hill Mines, Inc.	338515	LUCKY 690	F008N005W14	40	83
Tower Hill Mines, Inc.	338516	LUCKY 691	F008N005W14	40	84
Tower Hill Mines, Inc.	338517	LUCKY 692	F008N005W14	40	85
Tower Hill Mines, Inc.	338518	LUCKY 693	F008N005W13	40	86
Tower Hill Mines, Inc.	338519	LUCKY 694	F008N005W13	40	87
Tower Hill Mines, Inc.	338520	LUCKY 697	F008N004W18	40	88
Tower Hill Mines, Inc.	338521	LUCKY 698	F008N004W18	40	89

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	338522	LUCKY 699	F008N004W18	40	90
Tower Hill Mines, Inc.	347943	LC 407	F008N004W09	40	91
Tower Hill Mines, Inc.	347944	LC 408	F008N004W09	40	92
Tower Hill Mines, Inc.	347945	LC 502	F008N004W08	40	93
Tower Hill Mines, Inc.	347946	LC 503	F008N004W08	40	94
Tower Hill Mines, Inc.	347947	LC 506	F008N004W09	40	95
Tower Hill Mines, Inc.	347948	LC 507	F008N004W09	40	96
Tower Hill Mines, Inc.	347949	LC 600	F008N004W17 F008N004W18	40	97
Tower Hill Mines, Inc.	347950	LC 601	F008N004W17	40	98
Tower Hill Mines, Inc.	347951	LC 602	F008N004W17	40	99
Tower Hill Mines, Inc.	347952	LC 603	F008N004W17	40	100
Tower Hill Mines, Inc.	347953	LC 604	F008N004W17	40	101
Tower Hill Mines, Inc.	347954	LC 605	F008N004W16	40	102
Tower Hill Mines, Inc.	347955	LC 695	F008N005W13	40	103
Tower Hill Mines, Inc.	347956	LC 696	F008N005W13	40	104
Tower Hill Mines, Inc.	347957	LC 700	F008N004W17 F008N004W18	40	105
Tower Hill Mines, Inc.	347958	LC 701	F008N004W17	40	106
Tower Hill Mines, Inc.	347959	LC 702	F008N004W17	40	107
Tower Hill Mines, Inc.	347960	LC 703	F008N004W17	40	108
Tower Hill Mines, Inc.	347961	LC 704	F008N004W17	40	109
Tower Hill Mines, Inc.	347962	LC 790	F008N005W14	40	110
Tower Hill Mines, Inc.	347963	LC 791	F008N005W14	40	111
Tower Hill Mines, Inc.	347964	LC 792	F008N005W14	40	112
Tower Hill Mines, Inc.	347965	LC 793	F008N005W13	40	113
Tower Hill Mines, Inc.	347966	LC 794	F008N005W13	40	114
Tower Hill Mines, Inc.	347967	LC 795	F008N005W13	40	115
Tower Hill Mines, Inc.	347968	LC 796	F008N005W13	40	116
Tower Hill Mines, Inc.	347969	LC 797	F008N004W18	40	117
Tower Hill Mines, Inc.	347970	LC 798	F008N004W18	40	118
Tower Hill Mines, Inc.	347971	LC 799	F008N004W18	40	119
Tower Hill Mines, Inc.	347972	LC 800	F008N004W17 F008N004W18	40	120
Tower Hill Mines, Inc.	347973	LC 801	F008N004W17	40	121
Tower Hill Mines, Inc.	347974	LC 802	F008N004W17	40	122
Tower Hill Mines, Inc.	347975	LC 803	F008N004W17	40	123
Tower Hill Mines, Inc.	347976	LC 891	F008N005W14	40	124
Tower Hill Mines, Inc.	347977	LC 892	F008N005W14	40	125
Tower Hill Mines, Inc.	347978	LC 893	F008N005W13	40	126
Tower Hill Mines, Inc.	347979	LC 894	F008N005W13	40	127
Tower Hill Mines, Inc.	347980	LC 895	F008N005W13	40	128
Tower Hill Mines, Inc.	348802	LC 688	F008N005W15	40	129
Tower Hill Mines, Inc.	348803	LC 787	F008N005W15	40	130
Tower Hill Mines, Inc.	348804	LC 788	F008N005W15	40	131
Tower Hill Mines, Inc.	348805	LC 884	F008N005W16	40	132

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	348806	LC 885	F008N005W15	40	133
Tower Hill Mines, Inc.	348807	LC 886	F008N005W15	40	134
Tower Hill Mines, Inc.	348808	LC 887	F008N005W15	40	135
Tower Hill Mines, Inc.	348809	LC 888	F008N005W15	40	136
Tower Hill Mines, Inc.	348810	LC 984	F008N005W21	40	137
Tower Hill Mines, Inc.	348811	LC 985	F008N005W22	40	138
Tower Hill Mines, Inc.	348812	LC 986	F008N005W22	40	139
Tower Hill Mines, Inc.	348813	LC 987	F008N005W22	40	140
Tower Hill Mines, Inc.	348814	LC 1083	F008N005W21	40	141
Tower Hill Mines, Inc.	348815	LC 1084	F008N005W21	40	142
Tower Hill Mines, Inc.	348816	LC 1085	F008N005W22	40	143
Tower Hill Mines, Inc.	348817	LC 1086	F008N005W22	40	144
Tower Hill Mines, Inc.	348818	LC 1183	F008N005W21	40	145
Tower Hill Mines, Inc.	348819	LC 1184	F008N005W21	40	146
Tower Hill Mines, Inc.	348820	LC 1185	F008N005W22	40	147
Tower Hill Mines, Inc.	348821	LC 1186	F008N005W22	40	148
Tower Hill Mines, Inc.	348822	LC 1282	F008N005W21	40	149
Tower Hill Mines, Inc.	348823	LC 1283	F008N005W21	40	150
Tower Hill Mines, Inc.	348824	LC 1284	F008N005W21	40	151
Tower Hill Mines, Inc.	348825	LC 1285	F008N005W22	40	152
Tower Hill Mines, Inc.	348826	LC 1286	F008N005W22	40	153
Tower Hill Mines, Inc.	348827	LC 1287	F008N005W22	40	154
Tower Hill Mines, Inc.	348828	LC 1288	F008N005W22	40	155
Tower Hill Mines, Inc.	348829	LC 1382	F008N005W28	40	156
Tower Hill Mines, Inc.	348830	LC 1383	F008N005W28	40	157
Tower Hill Mines, Inc.	348831	LC 1384	F008N005W28	40	158
Tower Hill Mines, Inc.	348832	LC 1385	F008N005W27	40	159
Tower Hill Mines, Inc.	361326	LUCKY 90	F008N004W06	40	160
Tower Hill Mines, Inc.	361327	LUCKY 100	F008N004W06	40	161
Tower Hill Mines, Inc.	361328	LUCKY 200	F008N004W07	40	162
Tower Hill Mines, Inc.	361329	LUCKY 294	F008N005W12	40	163
Tower Hill Mines, Inc.	361330	LUCKY 300	F008N004W07	40	164
Tower Hill Mines, Inc.	361331	LUCKY 394	F008N005W12	40	165
Tower Hill Mines, Inc.	361332	LUCKY 401	F008N004W08	40	166
Tower Hill Mines, Inc.	361333	LUCKY 402	F008N004W08	40	167
Tower Hill Mines, Inc.	361334	LUCKY 403	F008N004W08	40	168
Tower Hill Mines, Inc.	361335	LUCKY 501	F008N004W08	40	169

30.2 State of Alaska Claims – 100% Owned

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	669377	LVG 1	F008N004W09	40	170
Tower Hill Mines, Inc.	669378	LVG 2	F008N004W16	40	171
Tower Hill Mines, Inc.	669379	LVG 3	F008N004W16	40	172
Tower Hill Mines, Inc.	669380	LVG 4	F008N004W16	40	173
Tower Hill Mines, Inc.	669381	LVG 5	F009N004W20	160	174
Tower Hill Mines, Inc.	669382	LVG 6	F009N004W20	160	175
Tower Hill Mines, Inc.	669383	LVG 7	F009N004W21	160	176
Tower Hill Mines, Inc.	669384	LVG 8	F009N004W21	160	177
Tower Hill Mines, Inc.	669385	LVG 9	F009N004W22	160	178
Tower Hill Mines, Inc.	669386	LVG 10	F009N004W22	160	179
Tower Hill Mines, Inc.	669387	LVG 11	F009N004W20	160	180
Tower Hill Mines, Inc.	669388	LVG 12	F009N004W20	160	181
Tower Hill Mines, Inc.	669389	LVG 13	F009N004W21	160	182
Tower Hill Mines, Inc.	669390	LVG 14	F009N004W21	160	183
Tower Hill Mines, Inc.	669391	LVG 15	F009N004W22	160	184
Tower Hill Mines, Inc.	669392	LVG 16	F009N004W22	160	185
Tower Hill Mines, Inc.	669393	LVG 17	F009N005W25	160	186
Tower Hill Mines, Inc.	669394	LVG 18	F009N005W25	160	187
Tower Hill Mines, Inc.	669395	LVG 19	F009N004W30	160	188
Tower Hill Mines, Inc.	669396	LVG 20	F009N004W30	160	189
Tower Hill Mines, Inc.	669397	LVG 21	F009N004W29	160	190
Tower Hill Mines, Inc.	669398	LVG 22	F009N004W29	160	191
Tower Hill Mines, Inc.	669399	LVG 23	F009N005W25	160	192
Tower Hill Mines, Inc.	669400	LVG 24	F009N005W25	160	193
Tower Hill Mines, Inc.	669401	LVG 25	F009N004W30	160	194
Tower Hill Mines, Inc.	669402	LVG 26	F009N004W30	160	195
Tower Hill Mines, Inc.	669403	LVG 27	F009N004W29	160	196
Tower Hill Mines, Inc.	669404	LVG 28	F009N004W29	160	197
Tower Hill Mines, Inc.	669405	LVG 29	F009N005W35	160	198
Tower Hill Mines, Inc.	669406	LVG 30	F009N005W35	160	199
Tower Hill Mines, Inc.	669407	LVG 31	F009N005W36	160	200
Tower Hill Mines, Inc.	669408	LVG 32	F009N005W36	160	201
Tower Hill Mines, Inc.	669409	LVG 33	F009N005W35	160	202
Tower Hill Mines, Inc.	669410	LVG 34	F009N005W35	160	203
Tower Hill Mines, Inc.	669411	LVG 35	F009N005W36	160	204
Tower Hill Mines, Inc.	669412	LVG 36	F009N005W36	160	205
Tower Hill Mines, Inc.	669413	LVG 37	F008N005W03	160	206
Tower Hill Mines, Inc.	669414	LVG 38	F008N005W03	160	207
Tower Hill Mines, Inc.	669415	LVG 39	F008N005W03	160	208
Tower Hill Mines, Inc.	669416	LVG 40	F008N005W03	160	209
Tower Hill Mines, Inc.	669417	LVG 41	F009N004W27	160	210
Tower Hill Mines, Inc.	669418	LVG 42	F009N004W27	160	211
Tower Hill Mines, Inc.	669419	LVG 43	F009N004W27	160	212
Tower Hill Mines, Inc.	669420	LVG 44	F009N004W27	160	213
Tower Hill Mines, Inc.	669421	LVG 45	F009N004W34	160	214
Tower Hill Mines, Inc.	669422	LVG 46	F009N004W34	160	215

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	669423	LVG 47	F009N004W34	160	216
Tower Hill Mines, Inc.	669424	LVG 48	F009N004W34	160	217
Tower Hill Mines, Inc.	669425	LVG 49	F008N004W04	160	218
Tower Hill Mines, Inc.	669426	LVG 50	F008N004W03	160	219
Tower Hill Mines, Inc.	669427	LVG 51	F008N004W03	160	220
Tower Hill Mines, Inc.	669428	LVG 52	F008N004W02	160	221
Tower Hill Mines, Inc.	669429	LVG 53	F008N004W02	160	222
Tower Hill Mines, Inc.	669430	LVG 54	F008N004W04	160	223
Tower Hill Mines, Inc.	669431	LVG 55	F008N004W03	160	224
Tower Hill Mines, Inc.	669432	LVG 56	F008N004W03	160	225
Tower Hill Mines, Inc.	669433	LVG 57	F008N004W02	160	226
Tower Hill Mines, Inc.	669434	LVG 58	F008N004W02	160	227
Tower Hill Mines, Inc.	669435	LVG 59	F008N004W10	160	228
Tower Hill Mines, Inc.	669436	LVG 60	F008N004W10	160	229
Tower Hill Mines, Inc.	669437	LVG 61	F008N004W11	160	230
Tower Hill Mines, Inc.	669438	LVG 62	F008N004W11	160	231
Tower Hill Mines, Inc.	669439	LVG 63	F008N004W10	160	232
Tower Hill Mines, Inc.	669440	LVG 64	F008N004W10	160	233
Tower Hill Mines, Inc.	669441	LVG 65	F008N004W11	160	234
Tower Hill Mines, Inc.	669442	LVG 66	F008N004W11	160	235
Tower Hill Mines, Inc.	669443	LVG 67	F008N004W16	160	236
Tower Hill Mines, Inc.	669444	LVG 68	F008N004W15	160	237
Tower Hill Mines, Inc.	669445	LVG 69	F008N004W15	160	238
Tower Hill Mines, Inc.	669446	LVG 70	F008N004W14	160	239
Tower Hill Mines, Inc.	669447	LVG 71	F008N004W14	160	240
Tower Hill Mines, Inc.	669448	LVG 72	F008N004W16	160	241
Tower Hill Mines, Inc.	669449	LVG 73	F008N004W16	160	242
Tower Hill Mines, Inc.	669450	LVG 74	F008N004W15	160	243
Tower Hill Mines, Inc.	669451	LVG 75	F008N004W15	160	244
Tower Hill Mines, Inc.	669452	LVG 76	F008N004W14	160	245
Tower Hill Mines, Inc.	669453	LVG 77	F008N004W14	160	246
Tower Hill Mines, Inc.	669454	LVG 78	F008N004W21	160	247
Tower Hill Mines, Inc.	669455	LVG 79	F008N004W21	160	248
Tower Hill Mines, Inc.	669456	LVG 80	F008N004W22	160	249
Tower Hill Mines, Inc.	669457	LVG 81	F008N004W22	160	250
Tower Hill Mines, Inc.	669458	LVG 82	F008N004W23	160	251
Tower Hill Mines, Inc.	669459	LVG 83	F008N004W23	160	252
Tower Hill Mines, Inc.	669460	LVG 84	F008N004W21	160	253
Tower Hill Mines, Inc.	669461	LVG 85	F008N004W21	160	254
Tower Hill Mines, Inc.	669462	LVG 86	F008N004W22	160	255
Tower Hill Mines, Inc.	669463	LVG 87	F008N004W22	160	256
Tower Hill Mines, Inc.	669464	LVG 88	F008N004W23	160	257
Tower Hill Mines, Inc.	669465	LVG 89	F008N004W23	160	258
Tower Hill Mines, Inc.	700008	LVG 90	F009N004W17	160	259
Tower Hill Mines, Inc.	700009	LVG 91	F009N004W17	160	260
Tower Hill Mines, Inc.	700010	LVG 92	F009N004W16	160	261

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	700011	LVG 93	F009N004W16	160	262
Tower Hill Mines, Inc.	700012	LVG 94	F009N004W17	160	263
Tower Hill Mines, Inc.	700013	LVG 95	F009N004W17	160	264
Tower Hill Mines, Inc.	700014	LVG 96	F009N004W16	160	265
Tower Hill Mines, Inc.	700015	LVG 97	F009N004W16	160	266
Tower Hill Mines, Inc.	700016	LVG 98	F008N005W09	160	267
Tower Hill Mines, Inc.	700017	LVG 99	F008N005W09	160	268
Tower Hill Mines, Inc.	700018	LVG 100	F008N005W09	160	269
Tower Hill Mines, Inc.	700019	LVG 101	F008N005W09	160	270
Tower Hill Mines, Inc.	703377	LVG 116	F009N004W14	160	271
Tower Hill Mines, Inc.	703378	LVG 117	F009N004W14	160	272
Tower Hill Mines, Inc.	703379	LVG 118	F009N004W13	160	273
Tower Hill Mines, Inc.	703380	LVG 119	F009N004W13	160	274
Tower Hill Mines, Inc.	703381	LVG 120	F009N004W15	160	275
Tower Hill Mines, Inc.	703382	LVG 121	F009N004W14	160	276
Tower Hill Mines, Inc.	703383	LVG 122	F009N004W14	160	277
Tower Hill Mines, Inc.	703384	LVG 123	F009N004W13	160	278
Tower Hill Mines, Inc.	703385	LVG 124	F009N004W13	160	279
Tower Hill Mines, Inc.	703386	LVG 125	F009N004W23	160	280
Tower Hill Mines, Inc.	703387	LVG 126	F009N004W23	160	281
Tower Hill Mines, Inc.	703388	LVG 127	F009N004W24	160	282
Tower Hill Mines, Inc.	703389	LVG 128	F009N004W24	160	283
Tower Hill Mines, Inc.	703390	LVG 129	F009N004W23	160	284
Tower Hill Mines, Inc.	703391	LVG 130	F009N004W23	160	285
Tower Hill Mines, Inc.	703392	LVG 131	F009N004W24	160	286
Tower Hill Mines, Inc.	703393	LVG 132	F009N004W24	160	287
Tower Hill Mines, Inc.	703394	LVG 133	F009N004W26	160	288
Tower Hill Mines, Inc.	703395	LVG 134	F009N004W26	160	289
Tower Hill Mines, Inc.	703396	LVG 135	F009N004W25	160	290
Tower Hill Mines, Inc.	703397	LVG 136	F009N004W25	160	291
Tower Hill Mines, Inc.	703398	LVG 137	F009N004W26	160	292
Tower Hill Mines, Inc.	703399	LVG 138	F009N004W26	160	293
Tower Hill Mines, Inc.	703400	LVG 139	F009N004W25	160	294
Tower Hill Mines, Inc.	703401	LVG 140	F009N004W25	160	295
Tower Hill Mines, Inc.	703402	LVG 141	F009N004W35	160	296
Tower Hill Mines, Inc.	703403	LVG 142	F009N004W35	160	297
Tower Hill Mines, Inc.	703404	LVG 143	F009N004W36	160	298
Tower Hill Mines, Inc.	703405	LVG 144	F009N004W36	160	299
Tower Hill Mines, Inc.	703406	LVG 145	F009N003W31	160	300
Tower Hill Mines, Inc.	703407	LVG 146	F009N004W35	160	301
Tower Hill Mines, Inc.	703408	LVG 147	F009N004W35	160	302
Tower Hill Mines, Inc.	703409	LVG 148	F009N004W36	160	303
Tower Hill Mines, Inc.	703410	LVG 149	F009N004W36	160	304
Tower Hill Mines, Inc.	703411	LVG 150	F009N003W31	160	305
Tower Hill Mines, Inc.	703412	LVG 151	F008N004W01	160	306
Tower Hill Mines, Inc.	703413	LVG 152	F008N004W01	160	307

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Tower Hill Mines, Inc.	703414	LVG 153	F008N003W06	160	308
Tower Hill Mines, Inc.	703415	LVG 154	F008N004W01	160	309
Tower Hill Mines, Inc.	703416	LVG 155	F008N004W01	160	310
Tower Hill Mines, Inc.	703417	LVG 156	F008N003W06	160	311
Tower Hill Mines, Inc.	703418	LVG 157	F008N004W12	160	312
Tower Hill Mines, Inc.	703419	LVG 158	F008N004W12	160	313
Tower Hill Mines, Inc.	703420	LVG 159	F008N003W07	160	314
Tower Hill Mines, Inc.	703421	LVG 160	F008N003W07	160	315
Tower Hill Mines, Inc.	703422	LVG 161	F008N004W12	160	316
Tower Hill Mines, Inc.	703423	LVG 162	F008N004W12	160	317
Tower Hill Mines, Inc.	703424	LVG 163	F008N003W07	160	318
Tower Hill Mines, Inc.	703425	LVG 164	F008N003W07	160	319
Tower Hill Mines, Inc.	703426	LVG 165	F008N004W13	160	320
Tower Hill Mines, Inc.	703427	LVG 166	F008N004W13	160	321
Tower Hill Mines, Inc.	703428	LVG 167	F008N003W18	160	322
Tower Hill Mines, Inc.	703429	LVG 168	F008N004W13	160	323
Tower Hill Mines, Inc.	703430	LVG 169	F008N004W13	160	324
Tower Hill Mines, Inc.	703431	LVG 170	F008N004W24	160	325
Tower Hill Mines, Inc.	703432	LVG 171	F008N004W24	160	326

30.3 Federal Unpatented Placer Claims – 100% Owned

Township	Range	Section	BLM Claim #	Claim Name
8 North	5 West	15NW	61477	Patsy Bench
9 North	4 West	31SE	61478	Black Bench
9 North	4 West	32SW	61479	Little Ben Bench
9 North	4 West	32SW	61480	Oregon
9 North	4 West	32SW	61481	Moonshine
9 North	4 West	32SW	61482	Blue Bird
9 North	4 West	32SW	61483	Nerma Fisko
9 North	4 West	32NE	61484	Prosper
9 North	4 West	32NE	61485	#2 Below Heine Creek
9 North	4 West	32NE	61486	Windy Association
9 North	4 West	32NE	61487	Triangle
9 North	4 West	32NE	61488	Black Dimond
9 North	4 West	29SE	61489	Robin
9 North	4 West	28SW	61490	Dimond Ski Association
9 North	4 West	28SW	61491	Hoover Devide
9 North	4 West	29SE	61492	Mellon
8 North	5 West	6SW	61498	#9 Above Discovery Association
8 North	4 West	6NE	61499	#10 Above Bench
8 North	4 West	5NW	61500	Gem Association
9 North	4 West	32SW	61501	#18 Above Discovery Association
9 North	4 West	32SE	61502	Sunshine
9 North	4 West	32SE	61503	Last Chance Fraction
9 North	4 West	32SE	61504	#23 above Discovery Association
9 North	4 West	32SE	61505	Star Association
9 North	4 West	32SE	61506	May Association
9 North	4 West	32SE	61507	Hot Air Association
9 North	4 West	32SE	61508	Option Association
9 North	4 West	32NE	61493	Tomtit Association
9 North	4 West	1SE	61494	LaFrance Association

30.4 Patented Claims – 100% Owned

Mineral Survey	Patent Number	Claim Names	LPI Ownership
832	743623	Wagner Association Bench	**100%**
1604	1041577	Snow Bird Bench	**100%**
1604	1041577	Mint Bench	**100%**
1604	1041577	Black Jack	**100%**
1609	1043895	Navada Bench Placer	**100%**
1609	1043895	Gold Brick Fraction Placer	**100%**
1623	1073686	Italy	**100%**
1624	1073687	Trustworthy Association	**100%**
1624	1073687	Imperial Association	**100%**
1625	1075872	Etna-Sunnyside Association	***15/16***
1625	1075872	Sunny Bench Association	**100%**
1640	1069069	Duncan	**100%**
1641	1069097	Eureka or No. 22 Creek Above on Livengood	**100%**
1641	1069097	Placer Mining Claim No. 21 Above Discovery on Livengood Creek	**100%**
1641	1069097	Placer Mining Claim No. 20 Above Discovery on Livengood Creek	***3/4***
1641	1069097	Placer Mining Claim No. 19 Above Discovery on Livengood Creek	**100%**
1641	1069097	Last Chance	**100%**
1641	1069097	Tolovana Bench	**100%**
1960	1036259	No.1 Above Discovery on Livengood Creek	**100%**
1960	1036259	The Tolovana Placer Mining Bench Claim on Right Limit of Livengood Creek	**100%**
1960	1036259	No.1 Above Discovery Bench	**100%**
1960	1036259	No. One Bench Fraction Above Discovery Right Limit Livengood Creek	**100%**
1960	1036259	Ready Bullion Placer Mining Bench Claim on Right Limit of Livengood Creek	**100%**
1963	1045457	Deep Channel Association	**100%**
1966	1031406	Golden Rod Association	**100%**
2060	1117204	Eldorado Bench	**100%**
2071	1117929	Marietta Association	**100%**
2152	1127946	Hidden Treasure	**100%**
2152	1127946	Hot Day	**100%**

30.5 Federal Unpatented Placer Claims – 100% Owned

Township	Range	Section	BLM Claim #	Claim Name
8 North	5 West	11SE	61249	#5 above Discovery
8 North	5 West	11SE	61250	Star fraction
8 North	5 West	11SW	61256	#3 above discovery
8 North	5 West	11SE	61257	#4 above discovery
8 North	5 West	11NE	61258	Dickey-fraction
8 North	5 West	11SE	61259	#4-a above discovery
8 North	5 West	11NE	61260	#5 above discovery bench
8 North	5 West	11NE	61261	#5 bench fraction, 1st tier
8 North	5 West	11NE	61262	Leitrim a/k/a letruim, letrium, letram association
8 North	5 West	12NW	61263	#7 bench right limit 1st tier above discovery
8 North	5 West	12NW	61264	#7 above discovery
8 North	5 West	12NW	61265	Rosalind fraction
8 North	5 West	12NW	61266	#8 above discovery
8 North	5 West	1SW	61267	Chatham bench association
8 North	5 West	1SW	61268	Gold dollar association claim
8 North	4 West	7NW	61269	Basin association claim
8 North	4 West	6SW	61270	Dorothy association bench claim
8 North	4 West	6SW	61271	Riffle association claim
8 North	4 West	6SE	61272	Montana association
8 North	5 West	11NE	61273	High grade fraction
8 North	5 West	11NE	61274	Triangle fraction
8 North	5 West	12NW	61275	#6 above discovery
8 North	5 West	12NW	61276	o.k. fraction
8 North	5 West	12NW	61277	#1 frank (franklin) gulch
8 North	5 West	1SW	61278	#2 franklin gulch
9 North	4 West	33SW	61292	Cloud association
9 North	4 West	33SW	61293	Ruby bench
8 North	5 West	28SW	61322	Pete
8 North	5 West	28NW	61323	Mike
8 North	5 West	21SE	61324	Ike
8 North	5 West	21NE	61325	Carolyn
8 North	5 West	21SE	61326	Sunshine Fraction
8 North	5 West	16SE	61327	Frio
8 North	5 West	16SW	61328	Ring
8 North	5 West	16SW	61329	Pilot
8 North	5 West	16SE	61330	Dan
8 North	5 West	16SE	61331	Nyuk
8 North	5 West	16SE	61332	Sweede Association
8 North	5 West	15SW	61333	Eureka Banch claim
8 North	5 West	15SW	61334	Bessie Bench
8 North	5 West	15NW	61335	Jeanne
8 North	5 West	16NE	61336	Hawk
8 North	5 West	16NE	61337	Gypsy
8 North	5 West	15NW	61338	Reef Association
8 North	5 West	15NW	61339	California Fraction
8 North	5 West	15NW	61340	No. 1 Below Discovery
8 North	5 West	9SE	61341	Horse

Township	Range	Section	BLM Claim #	Claim Name
8 North	5 West	9SE	61342	Close
8 North	5 West	10SW	61343	No. 2 Below Myrtle Creek
8 North	5 West	15NW	61344	No. 1 Bench Right Limit
8 North	5 West	15NW	61345	No. 1 Bench Fraction
8 North	5 West	15NE	61346	Discovery Livengood Cr. Association
8 North	5 West	10SW	61347	Placer Mining Claim No. 1 Below Discovery
8 North	5 West	9SE	61348	Destiny
8 North	5 West	9NE	61349	Jackpot
8 North	5 West	10NW	61350	Nancy
8 North	5 West	10NW	61351	Paystreak Bench Claim
8 North	5 West	10NW	61352	Eureka Bench Claim on Left Limit
8 North	5 West	10SW	61353	Deep Channel Fraction
8 North	5 West	10NW	61354	Colorado Association
8 North	5 West	10SE	61355	George Association, 2nd Tier
8 North	5 West	10SE	61356	Gan Fraction, 2nd Tier right limit
8 North	5 West	10NE	61357	Colorado Fraction, 3rd tier right limit
8 North	5 West	10NE	61358	Sacramento Bench
8 North	5 West	10NE	61359	Three Star Association
8 North	5 West	10SE	61360	Toni Placer Mining Claim
8 North	5 West	10NE	61361	Little Butch
8 North	5 West	10NE	61362	Horseshoe claim
8 North	5 West	10NE	61363	Carryall
8 North	5 West	10NE	61364	Fish Association
8 North	5 West	11NW	61365	Homesite Bench
8 North	5 West	11NW	61366	Virgina Association
8 North	5 West	10NW	61367	Eagle Bench Association
8 North	5 West	11NE	61368	Birch Fraction
8 North	5 West	2SE	61369	Brendan or Brandon Bench
8 North	5 West	2SW	61370	Xmas
8 North	5 West	2SE	61371	Blanche
8 North	5 West	1SW	61372	Audrey Fraction
8 North	5 West	1SW	61373	Gold Dollar Fraction
8 North	5 West	1SW	61374	Livengood Bench Right Limit
8 North	5 West	1NW	61375	Snow
8 North	5 West	1NE	61376	Ice
8 North	5 West	1SE	61377	Harding (Pearson)
8 North	5 West	1SE	61378	Mayflower Claim
8 North	5 West	1SE	61379	Golden Gusher Bench Claim
8 North	4 West	6SW	61380	Bonznza Bench
8 North	4 West	6NW	61381	North Star Association
8 North	4 West	6NW	61382	Black Bear Association
8 North	4 West	6NW	61383	Tom Cat Bench
8 North	4 West	6NW	61385	Flat Association
8 North	4 West	6SW	61386	Magnus Opus
8 North	4 West	6NE	61387	Banner Bench claim
8 North	4 West	6NE	61388	Jewel Bench
8 North	4 West	6NW	61389	Wild Cat bench

Township	Range	Section	BLM Claim #	Claim Name
9 North	4 West	31SE	61391	Hum Dinger
8 North	4 West	6NE	61392	Red Claim
9 North	4 West	31SE	61393	Jerry Association
9 North	4 West	32SW	61394	Alaska
9 North	4 West	32NW	61395	California Association claim
9 North	4 West	32NW	61396	Gol Run Bench, 2nd Tier
9 North	4 West	29SE	61399	Spring Association
9 North	4 West	28SE	61406	Wedge Claim
9 North	4 West	28SE	61407	Bulldozer
9 North	4 West	28SE	61408	Eve
9 North	4 West	27SW	61409	Resavoir Association
9 North	4 West	28SW	61420	Alabam on the divide
9 North	4 West	29SW	63462	Dome a/k/a Dome Association
9 North	4 West	1SW	63466	Marjorie Bench

30.6 State of Alaska Claims – 100% Owned

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
Livengood Placers, Inc.	361349	Galaxy 1	F008N005W10	40	327
Livengood Placers, Inc.	361350	Galaxy 2	F008N005W10	40	328
Livengood Placers, Inc.	361351	Galaxy 3	F008N005W02	40	329
Livengood Placers, Inc.	361352	Galaxy 4	F008N005W02 F008N005W03 F008N005W10 F008N005W11	40	330
Livengood Placers, Inc.	361353	Galaxy 5	F008N005W10 F008N005W11	40	331
Livengood Placers, Inc.	361354	Galaxy 6	F008N005W02	40	332
Livengood Placers, Inc.	361355	Galaxy 7	F008N005W02 F008N005W11	40	333
Livengood Placers, Inc.	361356	Galaxy 8	F008N005W11	40	334
Livengood Placers, Inc.	361357	Galaxy 9	F008N005W02	40	335
Livengood Placers, Inc.	361358	Galaxy 10	F008N005W02 F008N005W11	40	336
Livengood Placers, Inc.	361359	Galaxy 11	F008N005W01 F008N005W02	40	337
Livengood Placers, Inc.	361360	Galaxy 12	F008N005W01 F008N005W02	40	338
Livengood Placers, Inc.	361361	Galaxy 13	F008N005W01 F008N005W02	40	339
Livengood Placers, Inc.	361362	Galaxy 14	F008N005W01	40	340
Livengood Placers, Inc.	361363	Galaxy 15	F008N005W01	40	341
Livengood Placers, Inc.	361364	Galaxy 16	F008N005W01	40	342
Livengood Placers, Inc.	361365	Galaxy 17	F008N004W06 F008N004W07 F009N004W31	40	343
Livengood Placers, Inc.	361366	Galaxy 18	F008N004W06 F009N004W31	40	344
Livengood Placers, Inc.	361367	Galaxy 19	F009N004W31	40	345
Livengood Placers, Inc.	361368	Galaxy 20	F009N004W31	40	346
Livengood Placers, Inc.	603474	FM9N4W28SW	F009N004W28	160	347
Livengood Placers, Inc.	603475	FM9N4W28SE	F009N004W28	160	348
Livengood Placers, Inc.	603476	FM9N4W28NE	F009N004W28	160	349
Livengood Placers, Inc.	603477	FM9N4W28NW	F009N004W28	160	350

30.7 Hudson/Geraghty Lease - Federal Unpatented Lode Claims

BLM File Number	Parcel Name	Owner
55452	SHARON	HUDSON
55453	DOROTHEA	HUDSON
55454	LENORA	HUDSON
55455	FOSTER	HUDSON
55456	VANCE	HUDSON
55457	TWERPIT	HUDSON
55458	SAUNDERS	HUDSON
55459	NICKIE	HUDSON
55460	PATRICK	HUDSON
55461	WHITE ROCK	HUDSON
55462	SUNSHINE #1	GERAGHTY
55463	SUNSHINE #2	GERAGHTY
55464	OLD SMOKY	HUDSON
55465	WITTROCK	HUDSON
55466	BLACK ROCK	HUDSON
55467	TRAPLINE	HUDSON
55468	PATRICIA	HUDSON
55469	ANNE	HUDSON
55470	EILEEN	HUDSON
55471	BRIDGET	HUDSON

30.8 Tucker Lease – Federal Unpatented Lode and Placer Claims

The Property consists of the following six (6) unpatented Federal Lode and Placer claims.

File Number	Parcel Name	Date Acquired	Acres	Type
37580	Lillian No. 1	30-Sep-1968	21	Lode Claim
37581	Satellite	30-Sep-1968	20	Lode Claim
37582	Nickel Bench R.L.	30-Jun-1972	20	Placer Claim
37583	The Nickel	12-Aug-1965	19	Placer Claim
37584	Overlooked	6-Sep-1975	18	Placer Claim
37585	The Lad	12-Aug-1965	20	Placer Claim