INTERNATIONAL TOWER - HILL - MINES LTD

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April 10, 2017

VIA SEDAR

British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission

Dear Sirs:

Re: International Tower Hill Mines Ltd. ("Company") – Filing of Technical Report entitled "Prefeasibility Study of The Livengood Gold Project, Livengood, Alaska, USA" dated April 6, 2017 by Colin A. Hardie, P.Eng., Ryan T. Baker, P.E., Michael E. Levy, P.E., Timothy J. Carew, P.Geo., Scott E, Wilson, CPG, and Timothy J. George, P.E. ("Report")

On October 24, 2016, the Company filed a Technical Report entitled "Prefeasibility Study of The Livengood Gold Project, Livengood, Alaska, USA" dated October 24, 2016 by Colin A. Hardie, P.Eng., Ryan T. Baker, P.E., Michael E. Levy, P.E., Timothy J. Carew, P.Geo., Scott E, Wilson, CPG, and Timothy J. George, P.E. ("October Report"). Recently, it was determined that the calculation of All In Sustaining Costs for the Livengood Project ("AISC"), as contained in Table 22-2 on page 22-6 of the October Report, was incorrect as it included, contrary to World Gold Council guidance, both initial capital costs and mining and income taxes in the AISC calculation.

Accordingly, the Company is today filing the Report reflecting the following changes:

1. The AISC calculation has been corrected to be in accordance with World Gold Council guidance, and a corrected Table 22-2 has been included. The corrected AISC number has also been included in Table 1-11 on pages 1-24 and 1-25.

2. On January 12, 2017, the Company paid USD $14.7 million for the timely and full satisfaction of the final derivative payment due with respect to the acquisition of certain mining claims and related rights in the vicinity of the Livengood Project and the Company is now in full ownership and has no further liability with respect to this acquisition. The disclosure regarding the Livengood Property Description and Location in section 4.1.7, pages 4-5 and 4-6, has been updated accordingly.

Yours Truly,
INTERNATIONAL TOWER HILL MINES LTD.

Per: (signed) *Lawrence W. Talbot*
Lawrence W. Talbot,
General Counsel

NI 43-101 Technical Report

PRE-FEASIBILITY STUDY
OF THE LIVENGOOD GOLD PROJECT

Livengood, Alaska, USA

Prepared for:

International Tower Hill Mines Ltd.

TOWER HILL MINES
LIVENGOOD GOLD PROJECT

Effective Date: March 8, 2017
Signature Date: April 10, 2017

By qualified persons:

Colin A. Hardie, P. Eng. .... *BBA Inc.*
Ryan T. Baker, P.E. .... *NewFields Mining Design & Technical Services, LLC*
Michael E. Levy, P.E. .... *SRK Consulting (U.S.) Inc.*
Timothy J. Carew, P. Geo. .... *SRK Consulting (Canada) Inc.*
Scott E. Wilson, CPG. .... *Metal Mining Consultants Inc.*
Timothy J. George, P.E. .... *Wildcat and Badger, LLC*

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DATE AND SIGNATURE PAGE

This report is effective as of the 8th day of March 2017.

"Original signed and sealed"	*April 10, 2017*
Colin A. Hardie, P. Eng. BBA Inc.	Date
"Original signed and sealed"	*April 10, 2017*
Ryan T. Baker, P.E. NewFields Mining Design & Technical Services, LLC	Date
"Original signed and sealed"	*April 10, 2017*
Michael E. Levy, P.E. SRK Consulting (U.S.) Inc.	Date
"Original signed and sealed"	*April 10, 2017*
Timothy J. Carew, P. Geo. SRK Consulting (Canada) Inc.	Date
"Original signed and sealed"	*April 10, 2017*
Scott E. Wilson, CPG Metal Mining Consultants Inc.	Date
"Original signed and sealed"	*April 10, 2017*
Timothy George, P.E. Wildcat and Badger, LLC	Date

CERTIFICATE OF QUALIFIED PERSON

Colin A. Hardie, P. Eng.

I, Colin A. Hardie, P. Eng., do hereby certify that:

1. I am a Senior Process Metallurgist and Study Manager– Mining and Metals with the firm BBA Inc. located at 2020 Robert-Bourassa Blvd., Suite 300, Montréal, Quebec, H3A 2A5, Canada.
2. I am a graduate of the University of Toronto, Ontario, Canada with a BASc in Geological and Mineral Engineering (1996). In 1999, I graduated from McGill University of Montreal, Quebec, Canada, with a Master of Engineering in Metallurgy (Mineral Processing). In 2008, I received a Master of Business Administration (MBA) degree from the Université de Montréal (HEC), Montreal, Quebec, Canada.
3. I am a member in good standing of the Professional Engineers of Ontario (Member Number: 90512500) and of the Canadian Institute of Mining, Metallurgy, and Petroleum (Member Number: 140556). I have practiced my profession continuously since my graduation.
4. I have been employed in mining operations, consulting engineering and applied metallurgical research for over 20 years. My relevant project experience includes metallurgical testwork analysis, flowsheet development, cost estimation and financial modeling. Since joining BBA in 2008, I have worked as a senior process engineer and/or lead study integrator for numerous North American iron ore, precious metal, industrial mineral, and base metal projects.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of chapters 2, 3, 4, 5, 6, 13, 17, 18, 19, 20, 21, 22, 23, and 24, with the exception of section 18.17 and sections 21.2.1 to 21.2.3 and 21.4.3. I am also responsible for the relevant portions of chapters 1, 25, 26, and 27 of the technical report entitled "Pre-Feasibility Study of the Livengood Gold Project, Livengood, Alaska, USA" (the "Technical Report"), with an effective date of March 8, 2017.
8. I personally visited the property that is the subject to the Technical Report on August 15, 2016.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: April 10, 2017

"Original signed and sealed"

Colin Hardie, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Ryan T. Baker, P. E.

I, Ryan Baker, P. E., do hereby certify that:

1. I am currently employed as Principal Engineer by NewFields Mining Design & Technical Services, LLC, 9400 Station Street., Suite 300, Lone Tree, Colorado 80124.
2. I graduated with a Bachelor of Science degree in Civil Engineering from the Colorado State University, Fort Collins, Colorado in 1993.
3. I am a Registered Professional Engineer in Nevada (#13947), Alaska (#11172), Idaho (#10226), Colorado (#36988), Missouri (PE2008000049), and New Mexico (#22110). I am a Registered Member of the Society for Mining, Metallurgy and Exploration (#4204584).
4. I have been employed as an engineer continuously for a total of 23 years. My experience includes heap leach facility, tailings storage facility and mine surface infrastructure design and inspection.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of section 18.17, and the relevant portions of chapters 1, 25, 26, and 27 of the technical report entitled "Pre-Feasibility Study of the Livengood Gold Project, Livengood, Alaska, USA" (the "Technical Report"), with an effective date of March 8, 2017.
8. I personally visited the property that is the subject to the Technical Report on two occasions most recently on March 1-2, 2012.
9. I have had prior involvement with the property that is the subject of the Technical Report. I was part of the design team and a technical reviewer of the report titled "*Livengood Gold Project, Feasibility Study Geotechnical Design Report*" dated August 6, 2013.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: April 10, 2017

"Original signed and sealed"

Ryan T. Baker, P. E.

CERTIFICATE OF QUALIFIED PERSON

Michael Levy, P.E., P.G.

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

1. I am a Professional Engineer, employed as a Principal Geotechnical Engineer with SRK Consulting (U.S.), Inc. with an office at Suite 600, 1125 17th Street, Denver, CO, 80202.
2. I received a bachelor's degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004.
3. I am a registered Professional Engineer in the states of Colorado (#40268), California (#70578) and Arizona (#61372) and a registered Professional Geologist in the state of Wyoming (#3550).
4. I have practiced my profession continuously since March 1999 and have been involved in a variety of geotechnical projects specializing in advanced analyses and design of soil and rock slopes.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of section 16.2 and the relevant portions of chapter 1, 25, 26 and 27 of the technical report entitled "Pre-Feasibility Study of the Livengood Gold Project, Livengood, Alaska, USA" (the "Technical Report"), with an effective date of March 8, 2017.
8. I personally visited the property that is the subject to the Technical Report on June 20-22, 2012.
9. I have had prior involvement with the property that is the subject of the Technical Report. I was responsible for preparation of section 16.3 of report titled "*Livengood Gold Project, Feasibility Study*" dated September 4, 2013.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: April 10, 2017

"Original signed and sealed"

Michael E. Levy, P.E., P.G.

CERTIFICATE OF QUALIFIED PERSON

Timothy J. Carew, P. Geo.

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

1. I am a Principal Consultant (Geology) with the firm SRK Consulting (Canada) Inc. with an office at 5250 Neil Road, Suite 300, Reno, NV 89502.
2. I graduated from the following institutions:

University of Rhodesia, B.Sc. Geology	1973
University of Rhodesia, B.Sc. (Hons) Geology	1976
University of London (RSM), M.Sc. Mineral Production Management	1982

3. I am a member in good standing 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 over 35 years and, during that period, have been involved in geologic work in similar lithotectonic 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 the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of chapters 7, 8, 9, 10, 11, and 12. I am also responsible for sections 14.1 to 14.9, and the relevant portions of chapters 1, 25, 26, and 27 of the technical report entitled “Pre-Feasibility Study of the Livengood Gold Project, Livengood, Alaska, USA” (the “Technical Report”), with an effective date of March 8, 2017.
8. I personally visited the property that is the subject to the Technical Report on six occasions, most recently in May, 2012.
9. 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 that is the subject of the Technical Report, I had not had any prior involvement with the property. Since 2009, I have participated in the preparation of four NI 43-101 technical reports on the property, including the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: April 10, 2017

“Original signed and sealed”

Timothy J. Carew, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Scott E. Wilson, CPG

I, Scott E. Wilson, CPG, do hereby certify that:

1. I am currently employed as President by Metal Mining Consultants Inc., 9137 S. Ridgeline Blvd., Suite 140, Highlands Ranch, Colorado 80129.
2. I graduated with a Bachelor of Arts degree in Geology from the California State University, Sacramento in 1989.
3. I am a Certified Professional Geologist and member of the American Institute of Professional Geologists (CPG #10965) and a Registered Member (#4025107) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as a geologist continuously for a total of 27 years. My experience included resource estimation, mine planning, geological modeling, and geostatistical evaluations of numerous projects throughout North and South America.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of sections 14.10 to 14.14, and the relevant portions of chapter 1, 25, 26, and 27 of the technical report entitled "Pre-Feasibility Study of the Livengood Gold Project, Livengood, Alaska, USA" (the "Technical Report"), with an effective date of March 8, 2017.
8. I personally visited the property that is the subject to the Technical Report on August 2, 2011.
9. Prior to being retained by International Tower Hill Mines Ltd. for the technical report, entitled "Summary Report on the Livengood Project, Tolovana District, Alaska", dated August 29, 2011, I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: April 10, 2017

"Original signed and sealed"

Scott E. Wilson, CPG

CERTIFICATE OF QUALIFIED PERSON

Timothy J. George, PE

I, Timothy J. George, PE, do hereby certify that:

1. I am a consulting mining engineer and Principal of Wildcat and Badger, LLC, 3690 Bozeman Drive, Reno, NV 89511.
2. I am a graduate of the University of Arizona, with a BS in Mining Engineering.
3. I am a licensed Professional Engineer in the States of Colorado, USA (No. 47109) and I am a member of the Society for Mining Metallurgy & Exploration.
4. I have practiced my profession continuously for 9 years as a mining engineer, with experience in pit optimization, mine design, production scheduling and project evaluation in projects throughout North and South America, Africa and New Zealand.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of chapters 15 and 16, with the exception of section 16.2. I am also responsible for sections 21.2.1 to 21.2.3 and 21.4.3, as well as the relevant portions of chapter 1, 25, 26, and 27 of the technical report entitled "Pre-Feasibility Study of the Livengood Gold Project, Livengood, Alaska, USA" (the "Technical Report"), with an effective date of March 8, 2017.
8. I personally visited the property that is the subject to the Technical Report on October 15, 2016;
9. I have had prior involvement with the property that is the subject of the Technical Report. I conducted pit optimization and mine design iterations contributing to chapters 15 and 16 of the report titled "*Livengood Gold Project, Feasibility Study*" dated September 4, 2013;
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated: April 10, 2017

"Original signed and sealed"

Timothy J. George, PE

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TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
A	Ampere
a	Annum (year)
AARL	Anglo American Research Laboratories
ADEC	Alaska Department of Environmental Conservation
ADF&G	Alaska Department of Fish & Game
Ag	Silver
AGA	AngloGold Ashanti
Ai	Abrasion index
AISC	All-in sustaining costs (as defined by World Gold Council)
AMHT	Alaska Mental Health Trust
amsl	Above Mean Sea Level
ANFO	Ammonium Nitrate Fuel Oil
ANILCA	Alaska National Interest Lands Conservation Act
APR	Annual Percentage Rate
ARD	Acid Rock Drainage
Au	Gold
BBA	BBA Inc.
BLM	United States Bureau of Land Management
BWi	Bond Work index
CSS	Contact Support Services
CFR	Code of Federal Regulations
CIL	Carbon In Leach
CIM	Canadian Institute of Mining
CIP	Carbon In Pulp
CN	Cyanide
CND	Cyanide Detoxification
CN_T	Cyanide (Total)
CN_{WAD}	Cyanide (Weak Acid Dissociable)
Cu	Copper
CWi	Crusher Work index
DOT	Department of Transportation
DWT	JK Drop Weight Test
EIS	Environmental Impact Study
EPA	United States Environmental Protection Agency
EPCM	Engineering, Procurement, Construction Management
EPS	Electric Power Systems, Inc.

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
FNSB	Fairbanks North Star Borough
FOB	Freight On Board
FS	Feasibility Study
FWR	Fresh Water Reservoir
G&A	General and Administration
GRG	Gravity Recoverable Gold
GVEA	Golden Valley Electrical Association
ICP	Inductively Coupled Plasma
IRR	Internal Rate of Return
ISA	Inter-ramp Slope Angle
IT	Information Technology
ITH	International Tower Hill Mines, Ltd.
LLC	Limited Liability Company
LLDPE	Linear Low Density Polyethylene
LOM	Life of Mine
LPI	Livengood Placers, Inc.
MACRS	Modified Accelerated Cost Recovery System
MIK	Multiple Indicator Kriging
ML	Metal Leaching
MS	Mineral Survey
MSHA	Mine Safety and Health Administration
MWMP	Meteoric Water Mobility Potential
My	Million Years
NAD	North American Datum (Topographical Surveying)
NEPA	National Environmental Policy Act
NPI	Net Profits Interest
NPV	Net Present Value
NSR	Net Smelter Return
OCS	O'Connor Creek Substation
OK	Ordinary Kriging
PAG	Potentially Acid Generating
PFS	Pre-feasibility Study
POF	Probability of Failure
PLT	Point Lead Test
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
RC	Reverse Circulation
RMR	Rock Mass Rating
ROM	Run of Mine
ROW	Right of Way
RT	Rock Type
RQD	Rock Quality Designation
RWi	Rod Work index
SABC	Comminution circuit consisting of a SAG mill, ball mill and pebble crusher
SAG	Semi-Autogenous Grinding
Sb	Antimony
SMC	SAG Mill Comminution
SMU	Selective Mining Unit
SPI	SAG Power Index
SRIL	SR International Logistics
SRK	SRK Consulting (Canada and US) Inc.
SVC	Static VAR Compensator
SWPPP	Storm Water Pollution Protection Plan
TAPS	Trans-Alaska Pipeline
THM	Tower Hill Mines, Inc.
TMF	Tailings Management Facility
UCS	Uniaxial Compressive Strength
US	United States
USACE	United States Army Corps of Engineers
USD	United States Dollars
USGS	United States Geological Survey
UTM	Universal Transverse Mercator Coordinate System
W&B	Wildcat and Badger LLC
WBS	Work Breakdown Structure
WOL	Whole Ore Leach
WSR	Water Storage Reservoir
XRF	X-ray Fluorescence

TABLE OF ABBREVIATIONS – UNITS OF MEASURE			
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
ac	acre	ha	hectare
B	Billion	B	Billion
Btu	British thermal units	g-Cal	gram - calories
cfm	cubic feet per minute	m^3/m	cubic meters per minute
cfs	cubic feet per second	m^3/m	cubic meters per second
cP	centipoise (viscosity)	cP	centipoise (viscosity)
d	day (24 hours)	d	day (24 hours)
deg or ^{o}	angular degree	deg or ^{o}	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	gallon	L	Liter
gal/h	gallons per hour	L/h	Liters per hour
gpm	(US) gallons per minute	L/m	Liters per minute
lb/lb	pounds per pound	g/g	grams per gram
hr	hour (60 minutes)	hr	hour (60 minutes)
hp	horsepower	kW	kilowatt
Hz	Hertz	Hz	Hertz
in	inch	mm	millimeter
in	inch	µm	micron
in Hg	inches of mercury	mm Hg	millimeters of mercury
in WC	inches Water Column	mm WC	millimeters Water Column

TABLE OF ABBREVIATIONS – UNITS OF MEASURE			
Imperial		**Metric**	
Units	**Description**	**Units**	**Description**
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
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)
Mt	Million ton	Mmt	Million metric tonne
mph	miles per hour	km/h	kilometers per hour
MW	Megawatt	MW	Megawatt
oz	Troy ounce	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	second	s	second
scfm	standard cubic feet per minute	m^3/min	cubic meters per minute
sg	specific gravity	sg	specific gravity

TABLE OF ABBREVIATIONS – UNITS OF MEASURE			
Imperial		**Metric**	
Units	**Description**	**Units**	**Description**
t	ton (2,000 lbs)	mt	tonne (1,000 kg)
t/d	(short) tons per day	mt/d	tonnes per day
t/h	(short) 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	yard (36 inches)	m	meter
yd^3	cubic yard	m^3	cubic meter

1. SUMMARY

1.1 Introduction

The Livengood Gold Project (herein also referred to as "the Project") is a gold exploration project located 70 mi (113 km) northwest of Fairbanks, Alaska, USA. The Project is located in an active mining district that has been mined for gold since 1914.

This Technical Report (the "Report") was prepared and compiled by BBA Inc. under the supervision of the Qualified Persons named herein at the request of Tower Hill Mines, Inc. (THM), a wholly-owned subsidiary of International Tower Hill Mines Ltd. (ITH). The purpose of the Report is to summarize the results of the Pre-feasibility Study (PFS) for the Livengood gold deposit on the THM property. This Report has been prepared in accordance with the provisions of National Instrument 43-101 Standards of Disclosure for Mineral Projects, including Companion Policy 43 101CP and Form 13 101F1. The Report supports the ITH September 8, 2016 news release "International Tower Hill Mines Announces Study Results Showing Improved Livengood Gold Project" announcing the results of the study, and the March 8, 2017 news release "International Tower Hill Mines Announces Document Correction to Previously Reported All-In Sustaining Costs".

The PFS and this Report are based on an updated resource estimate, effective as of August 26, 2016, and has an optimized Project configuration of 52,600 t/d compared to the 100,000 t/d project evaluated in the September 2013 feasibility study (the "FS"). The Project configuration in the PFS remains a conventional, owner-operated surface mine that will utilize large-scale mining equipment in a blast/load/haul operation. Mill feed would be processed in a 52,600 t/d (47,000 mt/d) comminution circuit consisting of primary and secondary crushing, wet grinding in a single semi-autogenous (SAG) mill and single ball mill, followed by a gravity gold circuit and a conventional carbon in leach (CIL) circuit. As a result of the changes to the Project as summarized in this Report, including differences in the economic parameters applied to the geologic block model that resulted in a change in resources (gold price, recovery, CAPEX, and OPEX), the original project as evaluated in the FS is no longer considered current and the FS should therefore no longer be relied upon.

This Report assumes 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. Every effort has been made to clearly display the appropriate units being used throughout this Report.

However, it is important to note that both the Livengood Gold Project drillhole database and the block model were originally created in metric units and have been consistently maintained in metric units. Therefore some tables and figures in this Report may be presented in metric units only to minimize the risk of data unit conversion errors.

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

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

All monetary units are in United States dollars ($), unless otherwise specified. Costs are based on third quarter (Q3) 2016 dollars.

1.2 Contributors

The independent PFS was prepared through the collaboration of a number of industry-recognized consulting firms, including BBA Inc. ("BBA", Montreal, Quebec, Canada), NewFields Mining Design & Technical Services, LLC ("NewFields" Lone Tree, Colorado, USA), SRK Consulting (U.S.) Inc. ("SRK" Lakewood, Colorado, USA), SRK Consulting (Canada) Inc. ("SRK" Vancouver, British Columbia, Canada), Metal Mining Consultants Inc. ("MMC" Highlands Ranch, Colorado, USA), and Wildcat and Badger, LLC "W&B" Reno, Nevada, USA). Qualified persons as per NI 43-101 guidelines from these firms provided resource estimates, design parameters and cost estimates for mine operations, process facilities, major equipment selection, waste and tailings storage, reclamation, permitting, operating and capital expenditures. A summary of contributors to the PFS is included in Table 1-1.

Table 1-1: PFS Contributors

Qualified Person	Consulting Firm or Entity	Scope of Services
Colin A. Hardie, P. Eng. *(APEO No. 90512500)*	BBA Inc.	▪ Surface infrastructure design and capital costs; ▪ Metallurgical testwork analysis, processing plant design; ▪ Process plant capital and operating costs; ▪ Environmental Studies and Permitting; ▪ General and Administration operating costs; ▪ Financial analysis; ▪ Overall NI 43-101 integration.
Ryan T. Baker, P.E. *(Nevada No. 11172)*	New Fields Mining Design & Technical Services, LLC	▪ Geotechnical Engineering; ▪ Waste Rock and Water Management; ▪ Tailings Management Facility (TMF) design and capital costs; ▪ Closure Plan and Costs.
Michael E. Levy, P.E. *(Colorado No. 40268)*	SRK Consulting (U.S.), Inc.	▪ Rock mechanics and mine slope stability.

Qualified Person	Consulting Firm or Entity	Scope of Services
Timothy J. Carew, P. Geo. (APEGBC *Professional Geoscientist No.19706)*	SRK Consulting (Canada) Inc.	▪ Geology, drilling and MIK model.
Scott E. Wilson, CPG (No.10965)	Metal Mining Consultants Inc.	▪ Resource estimation.
Timothy J. George, P.E. *(Colorado No. 47109)*	Wildcat and Badger, LLC	▪ Mine engineering; ▪ Mine capital and operating costs; ▪ Reserve Estimation.

1.3 Key project outcomes

The reader is advised that the results of the PFS summarized in this Report are intended to provide an initial, high-level review of the proposed optimized project configuration and revised design options. The PFS mine plan, execution plan and economic model include numerous assumptions. There is no guarantee that the Project economics described herein will be achieved.

The key outcomes of this PFS are the following:

- The Livengood Gold Project mineral resource is estimated at 497.3 M measured tonnes at an average grade of 0.68 g/mt (10.84 Moz) and 28.0 M indicated tonnes at an average grade of 0.69 g/mt (0.62 Moz), for a total of 525.4 Mmt at an average grade of 0.68 g/mt (11.5 Moz).
- This PFS has converted a portion of these mineral resources into proven reserves of 377.7 Mmt at an average grade of 0.71 g/mt (8.62 Moz) and probable reserves of 14.0 Mmt at an average grade of 0.72 g/mt (0.353 Moz), for a total of 391.7 Mmt at an average grade of 0.71 g/mt (8.97 Moz).
- Annual mining rate of 55 Mmt and a life of mine waste rock to ore ratio of 1.3:1. Maximum size of the low grade stockpile is 131 Mmt.
- The PFS mine plan would provide sufficient ore (LOM gold head grade of 0.71 g/mt) to support an average annual production rate during Years 1-5 of 378,300 oz/y and an annual production rate of approximately 294,100 oz/y over an estimated 23 year mine life, producing a total of approximately 6.8 Moz.
- Metallurgical testwork has confirmed the preferred flowsheet consisting of primary crushing, secondary crushing and a comminution circuit (SABC configuration) producing a final grind size of 180 µm (P_{80}), with gravity recovery followed by whole ore leaching of the gravity tailings. LOM gold recovery is estimated to be 75.3% based on the rock types tested and mine plan.
- The initial capital cost (-20% / +25% accuracy) of the open pit mine, 52,600 t/d (47,700 mt/d) process plant and general site infrastructure is estimated at $1.84B, including a contingency of $231M.

- LOM project sustaining capital costs total $866M, including reclamation costs of $201M.
- The mining cost is estimated at $1.73/t mined, process plant operating cost is estimated at an average of $7.48/t ore processed, and general and administrative costs of $1.28/t ore processed.
- All-in sustaining cost (AISC) of production of 976 $/oz over LOM, including reclamation expenses, royalties and sustaining capital.
- Base case ($1,250/oz) negative project NPV of $-552M at a 5% discount rate and an IRR of 0.5% after mining and income taxes. Payback period is 22.1 years.

1.4 Property description, location and access

The Livengood property is located approximately 70 mi (113 km) by road (47 mi (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. These placer deposits have been actively mined since 1914 and have 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 ft (914 m) runway is located 3.73 mi (6 km) to the southwest of the Project and is suitable for light aircraft.

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°F to 72°F (7°C 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 about -40°F (-40°C). Annual precipitation is approximately 15.7 in (400 mm) water equivalent. Winter snow pack depth is approximately 26 in (66 cm).

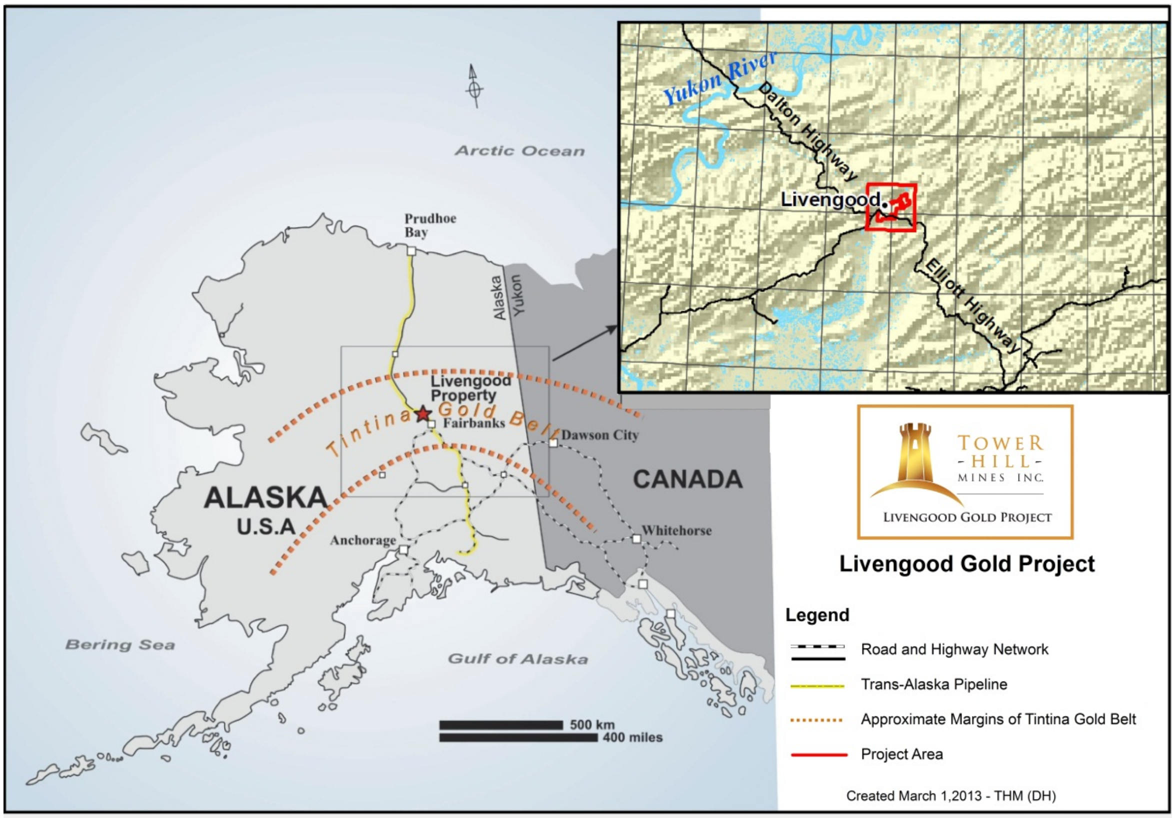

Figure 1-1: Project location map

1.5 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.

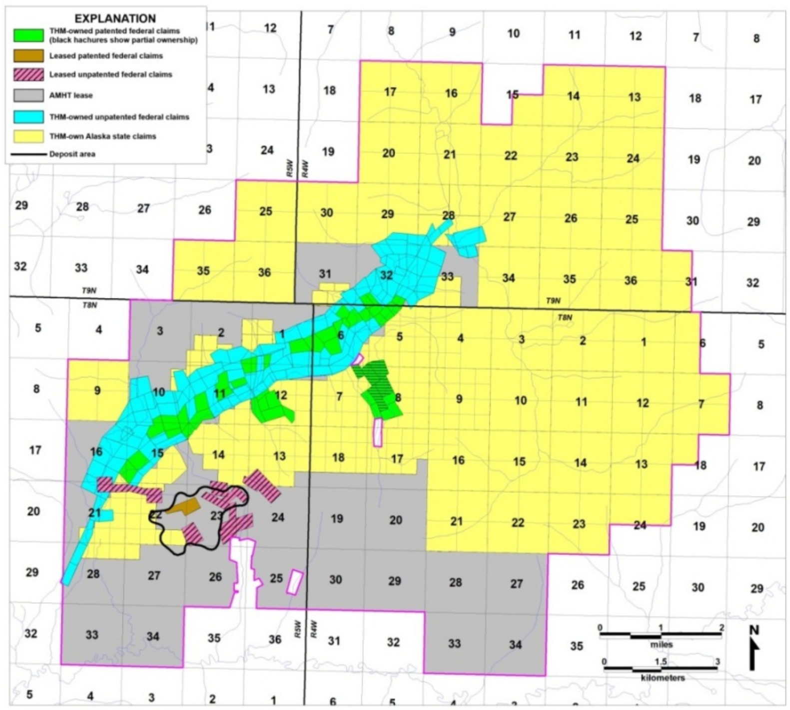

Figure 1-2: Livengood land holdings

1.6 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 has been produced. From 1914 through the 1970s, 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, primarily in the 1950s and in the form of dozer trenches, was carried out for lode type mineralization in the vicinity of Money Knob. However, no significant lode production has occurred to date.

Since the 1970s, 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 × 0.5 mi (1.6 × 0.8 km) square area with anomalous gold in soil. Scattered anomalous samples continue along strike for an additional 1.2 mi (2 km) to the northeast and 1 mi (1.6 km) to the southwest. Eight reverse circulation (RC) 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 ITH. In the same year, THM drilled a 4,026 ft (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. Concomitant programs have included geotechnical, engineering, and metallurgical work, along with the collection of environmental baseline data. As of the end of 2014, completed exploration and delineation drilling totals 574,599 ft (175,138 m) in 621 RC holes, and 140,854 ft (42,932 m) in 151 core drillholes.

Beginning in 2009, technical studies were performed to generate preliminary surface mine designs, to generate metallurgical data for process definition, 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, which was completed in August 2013.

From 2013 through 2016, additional metallurgical testwork was performed, along with various techno-economic trade-off studies, to form the basis for the optimized project configuration presented within this Report.

1.7 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.

1.7.1 Status of exploration

Cambior was chiefly responsible for outlining the sizeable area of anomalous gold in soil samples, which THM 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, 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.

1.8 Mineral processing and metallurgical testing

Several phases of testwork have been completed since the FS was issued.

A new round of comminution testing and simulation was completed, incorporating comminution data generated for the FS and new data generated from the PFS. SMC testwork was conducted to increase understanding of the ore variability by rock type in support of the grinding circuit development. The result of the work was the selection of a SAG mill / ball mill circuit in a SABC configuration with a pre-crushing (secondary) stage, treating 52,600 t/d (47,700 mt/d), operating to a target grind size P_{80} of 180 µm. The design relies upon an optimized drill & blast strategy, to achieve the rated throughput with a SAG mill (D × L) (36 ft × 20 ft) with 14 MW of installed power and a ball mill (26 ft × 40.5 ft) with 15 MW of installed power. The SAG mill is operated in closed circuit with a pebble crusher, and the ball mill is operated in closed circuit with two banks of hydrocyclones.

The back end of the plant, all that follows comminution, underwent several positive changes as a result of the PFS work, including a detailed analysis of previous work that was undertaken by BBA as well as the completion of five new rounds of testwork, completed since the issue of the FS. The various test programs (Continuous and Phases 7-10) were conducted to expand on knowledge developed through the course of the FS optimization and FS variability test programs. In the process of completing the five rounds of PFS testwork, several key conclusions were drawn:

- Increasing the target particle size from a P_{80} of 90 to 180 µm resulted in a decline in recovery of approximately 2% averaged across all rock types. The benefit of the coarser grind, which outweighs the recovery loss, is the higher throughput that facilitates a higher daily gold production.
- The effectiveness of gravity recovery was further confirmed as a result of the PFS testwork, using samples generated from both drill core and RC rig drill chips.

- Carbon in leach (CIL) of whole gravity tails (WOL) was retained over an option of conducting CIL on a flotation concentrate generated from the gravity tails. This decision was based on both testwork and an ensuing trade-off study, which showed that, under even optimistic recovery assumptions for the flotation option, the competing NPVs would at best be comparable. The WOL option was retained because of the lower associated risk.
- Analysis of PFS kinetic testwork (CIL leaching of gravity tails) with 3 hours of preconditioning using both oxygen (O_2) and lead nitrate has indicated that a reduction in leach residence time from 32 hours (FS) to 21 hours (PFS) is possible, without incurring a substantive penalty in recovery. The results are confirmed by earlier observation of CIL kinetic testwork from the FS optimization testwork.
- The same particle size, leaching and preconditioning conditions previously noted, also resulted in a significant reduction in cyanide consumption.

Gold recoveries (Gravity+CIL) were established for each of the Livengood ore rock types and are presented in Table 1-2. Note that this includes an approximately 2% recovery reduction applied to most rock types based on the change from P_{80} of 90 to 180 µm.

Table 1-2: Average gold recovery (Gravity+CIL) estimated for each rock type

Rock Type	Au Recovery (%)
RT4	81.8
RT5	84.7
RT6	75.6
RT7	62.4
RT9	69.6

In another trade-off study, BBA's review of the FS tailings detoxification option, based on metabisulphite, led to the identification of certain process reconfigurations and the adoption of sulphur burning technology at site to supply the SO_2 required by the proposed Inco SO_2/air tailings detoxification system. This option was deemed to have the highest payback of all options examined.

1.9 Mineral resource estimate

The current resource estimate for the Project (effective as of August 26, 2016) is based on the statistical analysis of data from 783 drillholes, totaling 717,435 ft (218,674 m), within a model area covering 3.1 mi^2 (7.9 km^2). The three dimensional geology was modeled and the structural/stratigraphic units have been used to constrain the resource model. The current mineral resource model is based on drilling, which still remains current as of this report.

Multiple indicator kriging (MIK) was used to calculate the gold grades for the blocks (15 m × 15 m × 10 m) in the model using the assay data composited to 10 m lengths. Statistical analysis indicated a significant relationship between the tenor of mineralization and the individual structural/stratigraphic units, consequently the resource interpolation for each individual geologic unit was restricted to: 1) the composite data within that unit; 2) contained within a 0.10 g/t gold grade shell, and 3) where data were available from a minimum of two octants and from two separate drillholes. Spatial statistics indicate that the mineralization shows very reasonable continuity within the range of anticipated operational cutoff grades. Bulk density for blocks within each of the structural stratigraphic units was assigned the mean value for density measurements of core and RC samples from that unit (total of 98 measurements for all the units). The resource model (15 × 15 × 10 blocks) was estimated using nine indicator thresholds, and then a change-of-support correction was imposed on the model, based on the assumption of 7.5 × 7.5 × 10 selectable mining units (SMUs). Resource classification into Measured, Indicated, and Inferred categories was based on estimation variance.

Mineral resources are reported at cut-off grades unique to the rock units of the Project. Rock type 7 (RT7) has a variable cut-off grade related to metallurgical ratios of the percentage of quartz - stibnite + jamesonite mineralization in each estimated model block. To determine the quantities of materials with "reasonable prospects for economic extraction" by open pit methods, the author determined pit constraining limits using the Lerchs-Grossman © economic algorithm, which constructs lists of related blocks that should or should not be mined. The final list defines a surface pit shell that has the highest possible total value, while honoring the required surface mine slopes and economic parameters. Input parameters were based on the three year trailing average gold price of $1,230/oz at August 26, 2016 and are described in Table 1-3.

Table 1-3: Constraining parameters used for the Livengood Gold Project

Parameter	Unit	Rock type 4	Rock type 5	Rock type 6	Rock type 7	Rock Type 8	Rock Type 9
Mining Cost	$/total mt	1.77	1.77	1.77	1.77	1.77	1.77
Gold Cut-Off	g/mt	0.33	0.32	0.35	0.40-0.85	0.38	0.38
Processing Cost	$/process mt	9.03	9.55	9.42	9.25	9.87	9.87
Gold Recovery	%	80.4	86.5	78.3	31-67	75.4	75.4
Administrative Cost	$/process mt	1.07	1.07	1.07	1.07	1.07	1.07
Royalty	%	3	3	3	3	3	3
Gold Selling Price	$/oz	1,230	1,230	1,230	1,230	1,230	1,230
Overall Slope Angle	Degrees	40	40	40	40	40	40

Mineral Resources for the Project are enumerated in Table 1-4.

Table 1-4: Livengood Gold Project pit constrained mineral resource estimate

Classification	Tonnes (Mmt)	Au Grade (g/mt)	Contained Au oz (000's)
Measured	497.34	0.68	10,840.84
Indicated	28.04	0.69	620.33
Total Measured and Indicated (M & I)	**525.38**	**0.68**	**11,461.17**
Inferred	52.80	0.66	1,127.21

Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution beyond that integral to the MIK model. Mineral resources are limited to mineralized material that occurs within the pit shells and which could be scheduled to be processed based on the defined cut-off grades.

1.10 Mineral reserve estimate

Mineral reserves have been recalculated for the Project at $1,250/oz of gold. The Project would be developed as a standalone open pit mining operation with a mine life of 23 years. Mineral reserves are confined to pit designs that meet geotechnical constraints. Proven reserves are identified as Measured Mineral Reserves contained within the pit shapes, above cut-off grades. Probable reserves are identified as Indicated Mineral Resources contained within pit designs, above cut-off grades. Mining methods and pit designs are detailed in Chapter 16 of this report. The Proven and Probable reserves for the Project are summarized in Table 1-5.

Table 1-5: Proven and Probable Reserves

Classification	Tonnes (Mmt)	Au Grade (g/mt)	Contained Au Oz (000's)
Proven	377.65	0.71	8,620.43
Probable	14.01	0.72	352.86
Total Proven and Probable (P & P)	**391.66**	**0.71**	**8,973.29**

1.11 Mining

The Project is a conventional surface mine that will utilize large-scale mining equipment for standard open pit mining of a blast/load/haul operation. The mineralized material will be crushed and processed by a gravity-whole ore CIL plant. Mining was scheduled to provide a mill feed of 52,600 t/d (47,700 mt/d). Preproduction stripping of 86.7 Mt (78.6 Mmt) of waste rock material is required for the construction of process facilities and site infrastructure. Mineralized material mined during preproduction will be stockpiled for mill feed later in the production period, after mill start-up.

The optimized production schedule provides an operating life of 23 years. A two-year preproduction period is required for removal of waste rock material and site construction. Mine production will produce mill feed mineralized material for 16 years. During the mine production period, low grade mineralized material will be stockpiled to be used for future mill feed, while waste rock material will be placed in a waste rock stockpile. Portions of the low grade, stockpiled, mineralized material will be sent to the mill during the mine production period, supplementing direct mine production tonnes to maintain constant mill throughput. After mining is complete, the mill will be fed from the remaining low grade stockpile for seven years. A summary production schedule, which shows the material movement of production material by period, is presented in Table 1-6 and Figure 1-3.

Table 1-6: Summary production schedule

Production Period	Pre-production	Mill and Stockpile	Stockpile Reclaim	Total
Years	**-2 to -1**	**1 to 16**	**17 to 23**	**23**
Mineralized Material to Mill (Kt)	-	254,377	-	**254,377**
Mineralized Material to Stockpile (Kt)	29,085	148,267	-	**177,352**
Stockpile to Mill (Kt)	-	48,931	128,422	**177,352**
Waste rock (Kt)	86,658	468,168	-	**554,825**
Strip Ratio (overall LOM)	-	1.84	-	**1.3**
Gold Grade (g/mt)	-	0.82	0.46	**0.71**
Contained Gold Koz	-	7,245	1,727	**8,972**

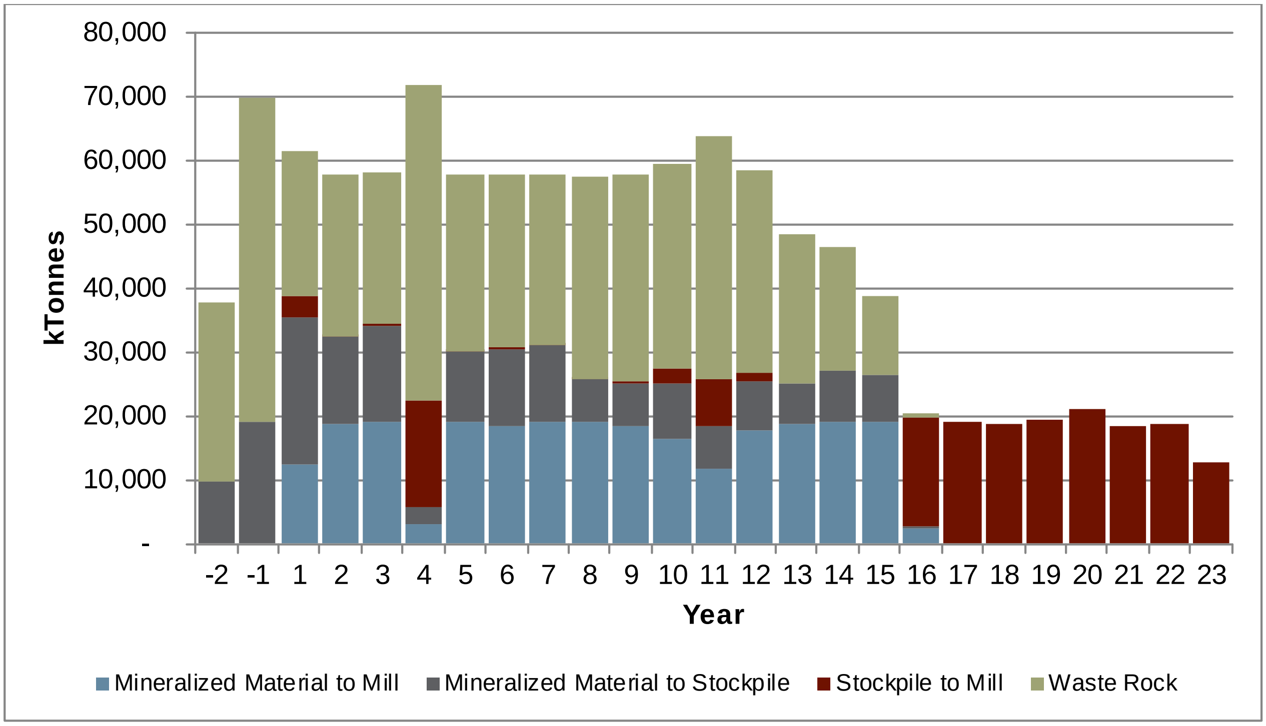

Figure 1-3: Production tonnage by material

1.12 Recovery methods

The recovery methods for the Project were established on the basis of previously noted laboratory-scale testwork programs, information from equipment suppliers and on BBA's experience on similar projects. Significant process plant configuration changes implemented within the PFS compared to the FS include the addition of secondary crushing ahead of the SAG mill for more efficient use of power, inclusion of a single line SAG/ball mill configuration, and simplification of the mill foundation and pebble re-grind circuit. Recent metallurgical testwork completed has also resulted in the grind size being coarsened from 90 to 180 µm (P_{80}), as well as a reduced leach circuit retention time from 32 to 24 hours.

The nominal Livengood process plant capacity at 92% is 52,600 t/d (47,700 mt/d) resulting in an annual capacity of 19.2 Mt/y (17.4 Mmt/y). Run of mine ore is transported to the primary gyratory (54/75) crusher, where it is crushed and stockpiled in a covered pile, then conveyed to the secondary crushing (1000 hp) building. Crushed product (1.65 in (42 mm)) will then be conveyed and processed in a comminution circuit (SABC) consisting of wet grinding in a single semi-autogenous (SAG) mill ((D×L) 36 ft × 20 ft / 18,774 hp) in closed circuit with a pebble crusher (1,000 hp) and a single ball mill (26 ft × 40.5 ft / 20,115 hp). The ball mill is in closed circuit with hydro-cyclones. A pulp stream will be bled from the cyclone feed and treated with a bank of eight centrifugal gravity gold separators. The gravity tails will be pre-treated with oxygen and lead nitrate and then leached in a conventional CIL circuit (2 rows of 7 tanks). The gravity gold will be intensively leached from the gravity concentrate with an intensive leach reactor (ILR) system.

Gold from the leach circuit will be recovered by an adsorption-desorption-recovery (ADR) circuit, where the final product will be doré. Two thickeners (131 ft / 40 m diameter) (Pre-leach and Pre-Detox) will be used to maximize water and cyanide recovery. The Inco SO_2/air cyanide detoxification method will be used to reduce the cyanide content of the process tailings to acceptable concentrations prior to being discharged to the tailings management facility (TMF). A preliminary water balance indicates that approximately 286 gpm (65 m^3/hr) of fresh water will be required during operations.

The gyratory crushing, secondary crushing and main process plant will operate 24 hours per day and 7 days per week. The operating teams will work on a schedule of two 12-hour shifts. The main process plant will be stopped periodically to perform preventive maintenance on equipment, for which there is no standby unit. The process plant is designed to operate with an availability of 92%.

Process plant reagents, including cyanide, lime, elemental sulphur, hydrochloric acid, lead nitrate, carbon and flocculants, will be delivered to site by transport truck and stored in the process facility as required.

Figure 1-4, a simplified process flow diagram, describes the conceptual process flow from the ore delivery to the crusher through to doré production and tailings management. The average gold head grade for plant feed will be 0.71 g/mt with an overall gold recovery of 75.3% based on the LOM plan.

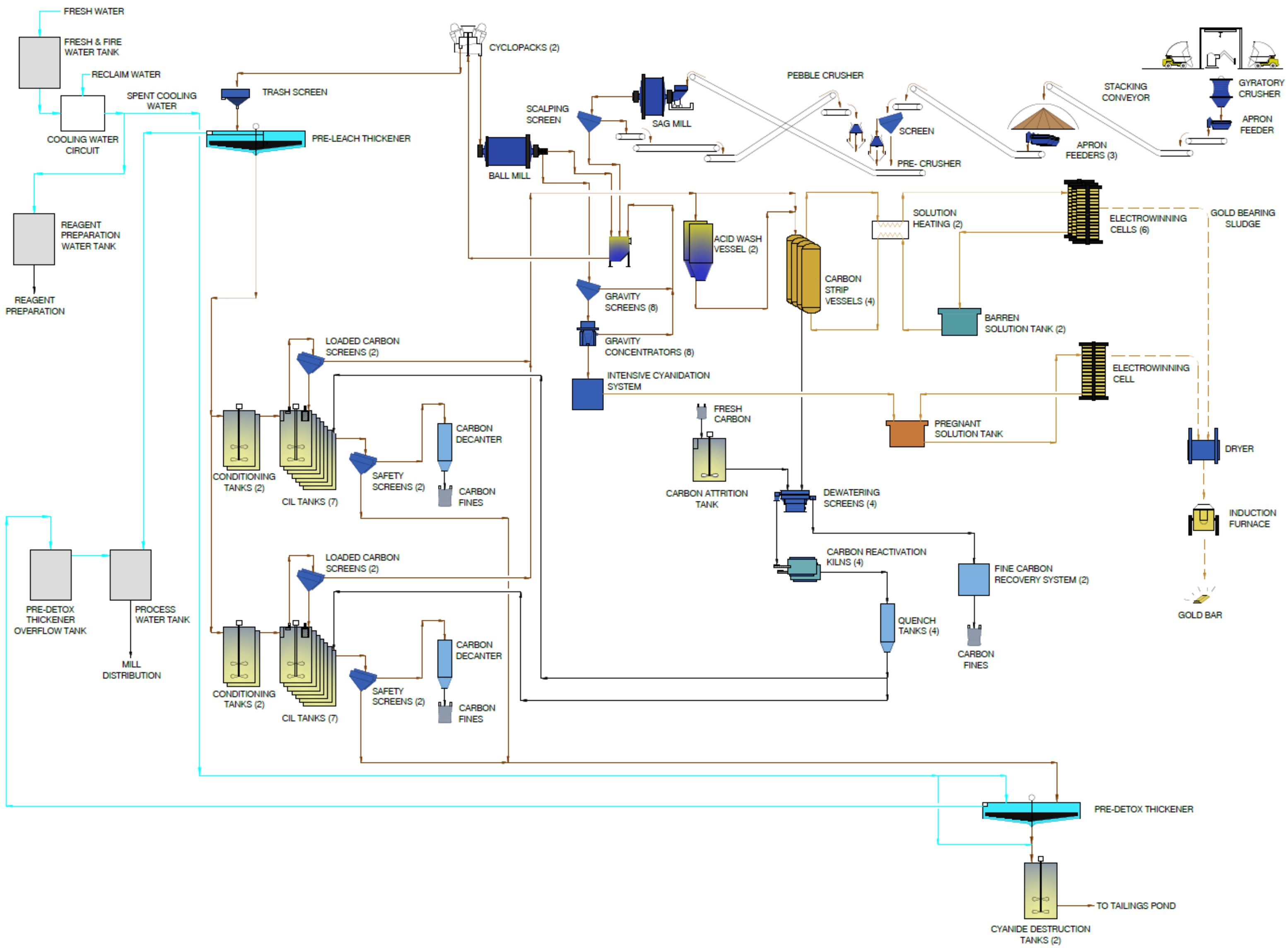

Figure 1-4: Simplified process flow diagram

The following Figure 1-5 shows the process plant feed grade and gold production per year based on the LOM mine plan. Annual gold production will be approximately 294,100 ounces per year.

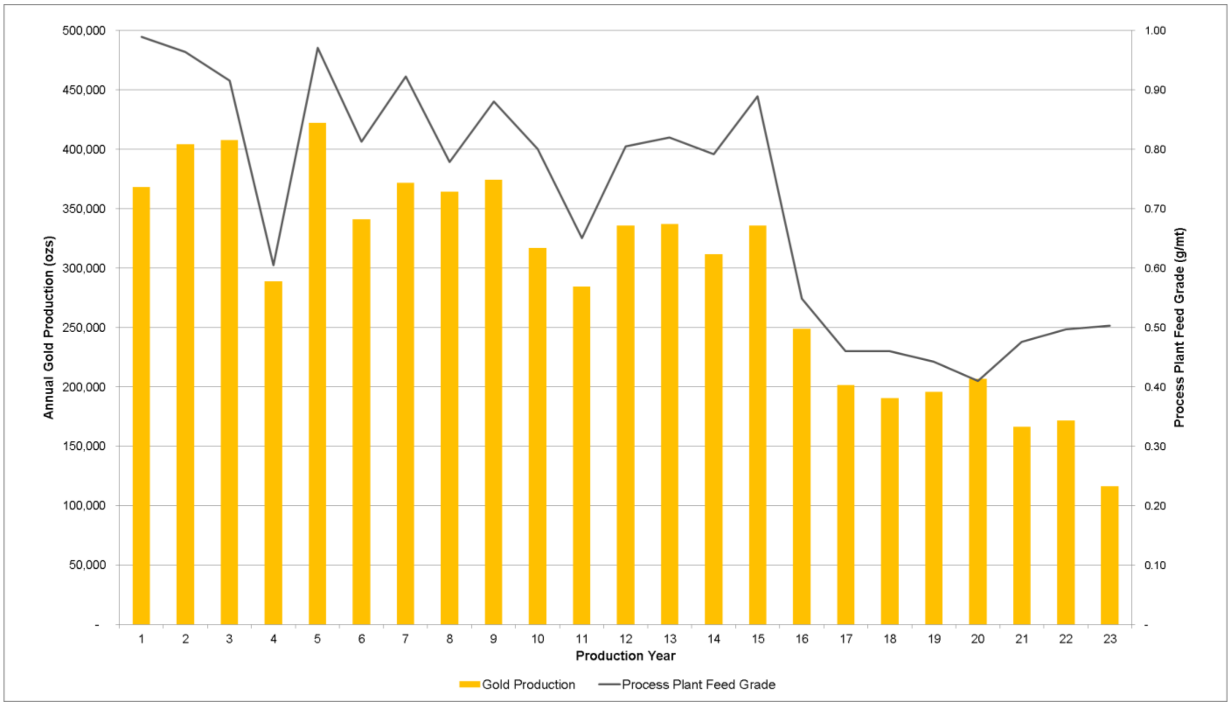

Figure 1-5: Gold production schedule (oz/year)

The process plant facilities include a wet laboratory, mill offices, a mill dry and maintenance shops. A total of 140 employees are required in the process plant, including 26 salaried staff and 114 hourly workers.

1.13 Local resources and Project infrastructure

1.13.1 Local resources

The Fairbanks North Star Borough (FNSB), which has a population of approximately 100,000 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 two paved highways, a railroad, an interlinked electrical grid and communications infrastructure. The city has an international airport serviced by up to three 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.

1.13.2 Project infrastructure

To the extent practicable, the infrastructure facilities for the Project have been designed for optimum construction access and operational efficiency as well as to take advantage of the existing roads and infrastructure.

Tailings, Mine Waste Rock and Water Management Facility

The tailings management facility (TMF) has been designed to provide safe and secure storage of approximately 450 Mt of mill tailings along with a supernatant pond. The TMF has sufficient area to expand up to 775 Mt capacity, dependent on future modifications that would be required at the Gertrude Creek overburden stockpile area and stormwater management infrastructure.

The TMF embankment is situated across Livengood Valley. Both the TMF embankment and impoundment area are designed as geomembrane-lined facilities that are constructed in phases. 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 impoundment area will be covered with a layer of rock to provide a stable foundation for the installation of the geomembrane.

Solution management systems at the TMF include a groundwater underdrain system and a tailings underdrain system. The groundwater underdrain system will be located below the impoundment geomembrane and positioned within the main drainages. This drain system will capture near surface groundwater flow and convey it to sumps located downstream of the TMF embankment. The collected water will be pumped into the TMF impoundment and used in the processing of ore at the mill. The tailings underdrain system is located above the impoundment geomembrane and will collect process solutions draining from the deposited tailings mass. This system will return the collected solutions to the supernatant pond for recycling back to the mill.

Non-economic mine waste rock produced by mining activities at the Livengood site will either be incorporated into the construction of site facilities, such as the TMF, or hauled and stockpiled in Gertrude Creek valley. The current design of the overburden stockpile is for 320 Mt of overburden storage, with expansion potential up to 700 Mt. An embankment constructed at the mouth of the Gertrude Creek valley will serve as a buttress for the overburden stockpile in addition to providing containment for tailings within the TMF.

Low-grade ore will also be stockpiled within the upper reaches of the Gertrude Creek valley. The current design of the stockpile is sufficient to store 140 Mt of material. The stockpile can be expanded, as needed, by modifying the design of the overburden stockpile.

The surface water management structures required to support the TMF are three surface water management pump stations and one surface water diversion channel. These structures will be used to manage and divert surface water generated from precipitation events and the spring freshet. The surface water management pump stations are positioned to capture streamflow and precipitation runoff within the Livengood Creek, Amy Creek and Lucky Creek watersheds. They consist of small embankments with pump systems. The surface water diversion channel is an existing channel located on the north hillside of Livengood Valley. This existing channel will require upgrading to handle the 100-year/24-hour storm event. The upgrades will consist of widening and deepening of the existing structure in addition to adjustments to the profile in some areas.

Surface Infrastructure

The Project envisions construction of the following key infrastructure facilities:

- Access light vehicle and mine haulage roads;
- O'Connor Creek substation and 50 mi of new 230 kV transmission line;
- Process plant and ancillary buildings;
- Administration, dry, maintenance, and warehouse complex;
- Mine truck wash and fueling facilities;
- Bulk fuel storage and delivery system;
- Water and sewage treatment;
- Fresh water pumping and distribution system;
- Waste rock, ore and growth media stockpiles;
- Surface water management pump stations and pipelines;
- Temporary construction camp;
- Fairbanks employee parking area.

Site Power

The total power demand of the Project is estimated to be approximately 55 MW (including network losses and a 5 MW contingency) based on the connected loads, load and efficiency factors and operating availability. A study completed by Electric Power Systems has determined that the local utility in Fairbanks (Golden Valley Electric Association) can provide the 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. A new 138/230 kV substation at O'Connor Creek (OCS) will be required to connect the transmission line to the GVEA system.

Emergency power systems (4.16 kV and 600 V) are planned for the purpose of supplying the critical installations when the main power is lost. Critical loads will be grouped into different categories, where some will be attended to automatically and others controlled manually.

Communications and IT

A site-wide telecommunication infrastructure will be installed to provide internet access, an IP phone system, a security access system, interconnection of the fire detection system, surveillance and process video cameras, as well as a mobile radio system for personnel and site vehicles.

1.14 Environmental and permitting

THM has been conducting environmental baseline studies at the 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, cultural resources and noise studies.

Table 1-7: Environmental baseline studies (2008-2016)

Baseline Study	2008	2009	2010	2011	2012	2013	2014	2015	2016
Surface Water									
Surface Water Quality		•	•	•	•	•	•	•	•
Sediment Quality						•	•	•	•
Hydrology									
Surface Water Flow and Snow			•	•	•	•	•	•	•
Hydrogeology			•	•	•	•	•	•	•
Groundwater Quality			•	•	•	•	•	•	•
Hydrogeological Modeling			•	•	•	•	•	•	•
Permafrost Studies			•	•	•	•	•	•	•
Wetlands and Vegetation									
Wetlands Delineations		•	•	•	•	•	•		
Meteorology & Air Quality									
Meteorological Data			•	•	•	•	•	•	•
Precipitation			•	•	•	•	•	•	•
Ambient Air				•					

Baseline Study	2008	2009	2010	2011	2012	2013	2014	2015	2016
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 (Chapter 11)				●	●	●			
Noise Studies									
Noise Surveys					●	●			

In early 2011, project engineers identified a 50 mi (80 km) 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 have been 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 agency 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.

1.15 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 2013, the census area held a total population of 5,650 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, which is approximately 80 mi (129 km) from the Project, have road access to Fairbanks.

The Fairbanks area is the service and supply hub for Interior and Northern Alaska. Construction of 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 operation of a local refinery, 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 economy included 38,150 non-agricultural wage and salary jobs in 2012. In 2011, using decennial census data, average employment of 39,018 wage and salary jobs, accounted for $1.81B in annual payroll.

Most of the small communities in rural interior Alaska are largely dependent on subsistence. Seventy-five percent (75%) of the Native families in Alaska's smaller villages acquire 50% of their food through subsistence activities (Federal Subsistence Board, 1992). For families that 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 PFS estimates a total of 6.8 million man-hours during project construction with a peak construction workforce of 1,050. The average wages of those workers is estimated at $40.00/h. During the two years of preproduction mine development, the owner's crew will be approximately 175 employees. During operation, the average employee count is estimated at 331 and an annual average wage of approximately $100,000. Total annual wages paid during operations is estimated to be approximately $32M.

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 355 residents in 2013. 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.

1.16 Capital cost and operating cost estimates

1.16.1 Capital costs

The total estimated preproduction capital cost (-20% / +25%) to design, procure, construct and commission the Project facilities including indirect costs, contingencies and funding of reclamation activities, is estimated to be $1.84B. The estimated sustaining capital cost required by the Project is $866M. Items such as salvage value, sales taxes, land acquisition, permitting, licensing, feasibility study and financing costs are not included in the cost estimate. The cumulative life-of-mine capital expenditure (preproduction and sustaining capital) is estimated to be $2.501B. Table 1-8 summarizes the initial capital and sustaining capital costs by major area.

Table 1-8: Initial capital and sustaining capital costs by major area
($ Millions)

Cost Item/Area	Initial ($M)	Sustaining ($M)
Mine Equipment	173	123
Mine Development	146	0
Process Facilities	446	24
Infrastructure Facilities	454	442
Power Supply	79	0
Owners Costs	307	0
Contingency	213	76
Subtotal before Reclamation	***1,818***	***665***
Funding of Reclamation Trust Fund (1)	18	201
Total	**$ 1,836**	**$ 866**

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

(1) Includes initial funding, total $342M estimated costs. The difference of $123M is projected trust fund earnings.

1.16.2 Operating costs

The operating cost estimate for the Project includes all expenses incurred to operate the mine and process plant, from the start of Year 1 through Year 23, at a daily average production rate of 52,600 t (47,700 mt). The expected accuracy for the estimate is +-/ 20% and does not contain any allowances for contingency or escalation beyond Q3 2016. The average operating cost including royalties and smelting/refining fees over the life of mine is estimated to be $12.95/t ($14.27/mt) milled. It is anticipated that the Project's workforce requirements will average 331 employees over the life of mine.

The total and unit operating cost estimate summaries are shown below in Table 1-9 for the three major operating cost areas: mining, processing, and general and administrative (G&A). The unit costs areas are shown in terms of total cost life of mine (LOM) per ore ton milled and total cost per troy ounce of gold produced.

Table 1-9: Total operating cost breakdown (LOM average)

Cost Item / Area	Life of Mine ($M)	Cost per ton ($/t mined)	Cost per ton ($/t milled)	Cost per oz ($/oz)	OPEX (%)
Mining (including stockpile reclaim)	1,505	1.73	3.49	223	27
Processing	3,228	-	7.48	477	58
General and Administration	552	-	1.28	82	10
On-site Mine Operating Costs	***5,286***	**-**	***12.24***	***781***	***95***
Royalties	252	-	0.58	37	4
Smelting, Refining and Transport	54	-	0.13	8	1
Total	**$ 5,592**	**-**	**$ 12.95**	**$ 827**	**100 %**

1.17 Project economics

A financial analysis for the Project was carried out using a discounted cash flow approach. The internal rate of return ("IRR") on total investment was calculated based on 100% equity financing even though THM may decide in the future to finance part of the Project using alternative sources of capital. The Net Present Value ("NPV") was calculated from the cash flow generated by the Project based on a discount rate of 5%. The payback period based on the undiscounted annual cash flow of the Project was also indicated as a financial measure.

No inflation or escalation exists in the economic model. THM compiled the taxation calculations for the Project with assistance from third-party taxation experts. The model calculates pre-tax and after-tax returns, and includes Alaska state taxes and Federal taxes based on the 2016 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. The major inputs and assumptions used for the development of the financial model are listed in Table 1-10.

Table 1-10: Financial model inputs

Execution Plan	
Construction Period	27 months
Mine Life (after preproduction)	23 years
LOM Ore Tons (millions)	432
LOM Gold Grade (g/mt Au)	0.71
Average Annual Process Gold Production Rate (oz)	294,100
Metal Pricing	
Gold Price ($/oz)	1,250
Cost and Tax Criteria	
Estimate Basis	Q3 2016
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Income Tax	AK State, Federal
Royalties	
Royalty on Net Smelter Return (NSR)	3%
Gold Transportation and Insurance, Refining, and Payable Charges	
Gold ($/oz)	8.05
Payable Terms	
Gold	99.50%

Table 1-11 below presents the results of the pre-feasibility study.

Table 1-11: Summary of pre-feasibility study results

	Value	Units
Production Metrics		
Mill Throughput	52,600	Dry tons/day
Head Grade – LOM	0.71	g/mt
Gold Recovery	75.3	%
Mine Life	23	Years
Total oz Produced	6,763,900	oz
Average Annual Production – LOM	294,100	oz
Total Ore Processed	432	Million tons
Total Waste Rock (not including pre-production)	468	Million tons
Annual Mining Rate	60	Million tons
Low grade stockpile size (maximum)	145	Million tons

	Value	Units
Capital and Operating Costs		
CAPEX – Initial	1.84	$Billion
CAPEX – Sustaining	665	$Million
Reclamation & Closure	342	$Million
OPEX – Mining - LOM	1.73	$/t mined
OPEX – Processing - LOM	7.48	$/t ore
OPEX – G&A - LOM	1.28	$/t ore
OPEX – Operating Cost – LOM	877	$/oz
All-in Sustaining Cost (AISC)	976	$/oz
All-in Cost Pre-Tax (CAPEX+OPEX) – LOM	1,247	$/oz
All-in Cost After-Tax (CAPEX+OPEX) – LOM	1,263	$/oz
Pre-Tax Financial Metrics		
Pre-Tax NPV (@ 5%)	- 507.1	$
Pre-Tax IRR	1.0	%
Pre-Tax Payback	17.2	Years
After-Tax Financial Metrics		
After-Tax NPV (@ 5%)	- 552.0	$
After-Tax IRR	0.5	%
After-Tax Payback	22.1	Years

The pre-tax internal rate of return (IRR) is 1.0% and the pre-tax net present value (NPV) using a 5% discount rate over the mine life is a loss of $ -507.1M.The after-tax IRR is 0.5% and the after-tax NPV at a discount rate of 5% over the mine life is a loss of $ -552.0M.

The results of the after–tax sensitivity analysis performed are summarized in Figure 1-6 and Figure 1-7. This sensitivity analysis shows that both gold price and recovery variations cause the greatest and almost equivalent impact on project value. A 30% increase in gold price to $1,625/oz would yield an IRR of 8.6% and a NPV of $511M. A 30% decrease in gold price to $825/oz would yield a reduced IRR of -39.6% and NPV of $ -1,887M. The impact of variations in operating and capital cost on both financial metrics is fairly similar with the operating cost changes resulting in marginally larger project returns than capital cost changes, meaning reducing operating expenses would benefit the Project more than reducing capital costs by the same percentage.

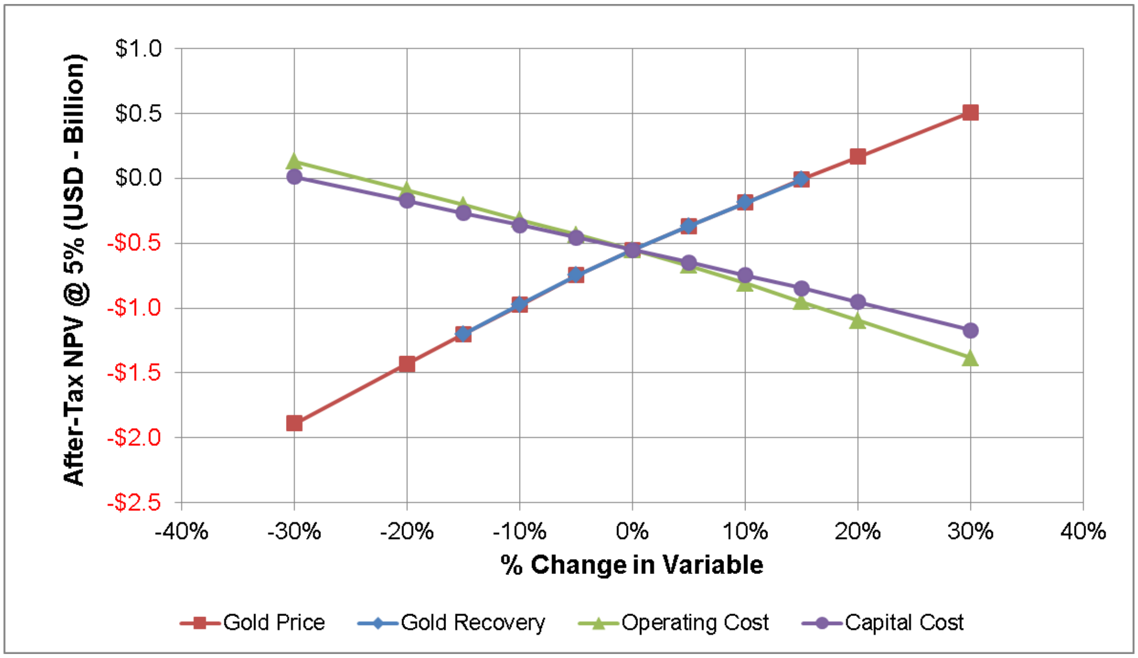

Figure 1-6 After-tax sensitivity analysis for project net present value (NPV @ 5% discount rate)

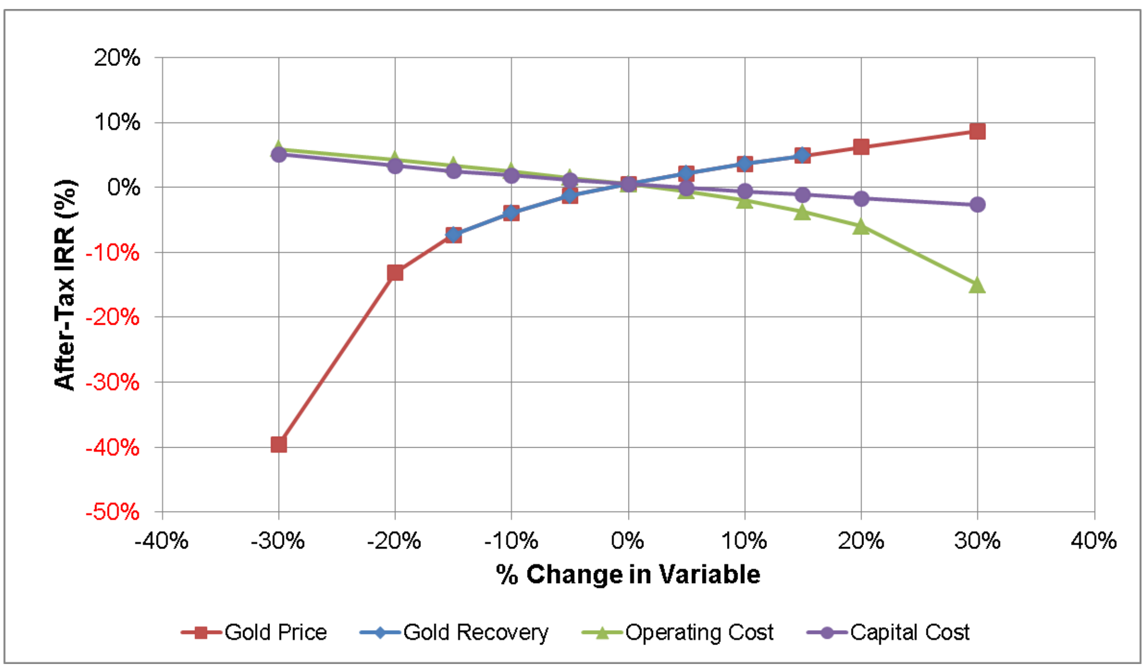

Figure 1-7 After-tax sensitivity analysis for project internal rate of return (IRR %)

1.18 Project schedule

A hypothetical execution schedule for permitting, engineering, pre-development and construction of the Project was developed as part of the pre-feasibility study. The plan is conceptual in nature and contingent on the eventual completion of the positive feasibility study, during which it will be adjusted and refined. The major project activity milestones are presented in Table 1-12.

Table 1-12: Key project activities (preliminary)

Activity	Start date	Completion date	Duration (months)
Draft Environmental & Social Impact Study	Q1 YR -7	Q3 YR -3	48
Engineering Studies and Support for EIS approval	Q1 YR -7	Q3 YR -3	48
Process Plant Detailed Engineering	Q1 YR -3	Q3 YR -2	21
NEPA Project Authorization		Q3 YR -3	
Pit Pre-stripping / Waste Rock Supply for Construction	Q3 YR -3	Q4 YR -1	30
Tailings Management Embankment Construction	Q3 YR -3	Q4 YR -1	30
Process Plant Construction	Q4 YR -3	Q4 YR -1	27
Process Plant Dry Commissioning Completed		Q1 YR 1	
Start Process Plant Ramp-up to Full Production	Q1 YR 1		

1.19 Interpretations and conclusions

This Report was prepared by a group of independent consultants (QPs) to demonstrate the economic viability of an open pit mine and process plant complex based on the reserves estimated for the Livengood Gold Project. The process plant capacity is planned to be 52,600 t/d (47,700 mt/d).

This Report provides a summary of the results and findings from each major area of investigation to a level that is considered to be equivalent and normally expected for a PFS of a resource development project. Standard industry practices, equipment and processes were used in this study. The authors of this report, on the date of publication, are not aware of any unusual or significant risks or uncertainties that could materially affect the reliability or confidence in the Project based on the information available.

The results of the PFS indicate that the proposed Project is technically feasible, but is not economic at the base case gold price of $1,250/oz. However, development of the Project could have the potential to generate positive results. The Project QPs recommend that prior to advancing the Project to the feasibility study level, an optimization phase be completed to improve the Project's economics, study potential opportunities and reduce overall implementation risk.

An analysis of the results of the investigations has identified a series of risks and opportunities associated with each of the technical aspects considered for the development of the Project.

The key risks include:

- Large earthwork quantities required to construct the Project;
- Gold recovery less than expected due to complex orebody and metallurgy.

The key opportunities include:

- Alternative geological model could improve projected head grades;
- Improved geological model could better define zonation and lead to better projected overall gold recovery;
- Further variability metallurgical testing to improve gold recovery and lower operating costs (reagents).

1.20 Recommendations

It is recommended that a number of optimization studies be initiated prior to advancing the Project to the feasibility study stage. It is also recommended that environmental work and permitting continue as needed to support THM's development plans.

Based on the list of recommendations presented in Chapter 26, it is estimated that the optimization studies, laboratory testwork and supporting fieldwork will cost approximately $6.3M, including a 20% contingency.

2. INTRODUCTION

2.1 Overview

This Technical Report (Report) was prepared and compiled by BBA Inc. under the supervision of the Qualified Persons (QPs) named herein at the request of Tower Hill Mines, Inc. (THM), a wholly-owned subsidiary of International Tower Hill Mines Ltd. (ITH). BBA Inc. is an independent engineering consulting firm headquartered in Montreal, Quebec, Canada.

The purpose of the Report is to summarize the results of the Pre-feasibility Study (PFS) for the Livengood gold deposit on the THM property. This Report has been prepared in accordance with the provisions of National Instrument 43-101 Standards of Disclosure for Mineral Projects, including Companion Policy 43 101CP and Form 13 101F1. The Report supports the ITH September 8, 2016 news release “International Tower Hill Mines Announces Study Results Showing Improved Livengood Gold Project”, and the March 8, 2017 news release “International Tower Hill Mines Announces Document Correction to Previously Reported All-In Sustaining Costs”.

This Report was prepared under the supervision of the QPs named herein with contributions from BBA Inc., NewFields Mining Design & Technical Services LLC, SRK Consulting (Canada and U.S.) Inc., Metal Mining Consultants Inc., and Wildcat and Badger LLC.

The Livengood property (65˚30’16’’N, 148˚31’33’’W) 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 (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 System (TAPS) corridor, which transports crude oil from the North Slope south.

2.2 Important notes

2.2.1 Status of land acquisition and all-in sustaining costs (AISC)

On January 12, 2017, the Company paid $14.7 million for the timely and full satisfaction of the final derivative payment due with respect to the acquisition of certain mining claims and related rights in the vicinity of the Project. On January 17, 2017, the Full Deed of Reconveyance releasing the Deed of Trust on the acquired property was recorded and the Company is now in full ownership and has no further liability with respect to this acquisition.

On March 8, 2017, the Company announced that it had corrected previously reported all-in sustaining costs (AISC) downward to comply with the World Gold Council guidance. The results of the PFS set forth in a news release on September 8, 2016 and as filed with the NI 43-101 report on October 24, 2016 had included, contrary to World Gold Council guidance, both initial capital costs and mining and income taxes in the $1,263/oz previously reported as all-in sustaining costs. As a result of the restatement, the AISC for the Project is now projected to be $976/oz.

For these reasons, the Company has decided to re-issue this Report. With the exception of the changes outlined above, the report remains largely unaltered, save for some typographical errors that were corrected.

2.3 Important notes

2.3.1 2013 feasibility study (FS)

This Report is based on an updated resource estimate effective as of August 26, 2016 and has an optimized project configuration of 52,600 t/d (47,700 mt/d) compared to the 100,000 t/d (90,700 mt/d) project evaluated in the September 2013 feasibility study (the “FS”) summarized in the following NI 43-101 technical report:

- Kunter, R., Rehn C., Prenn, N., Carew, and T., Levy, 2013: NI 43-101 Technical Report On the Livengood Gold Project - Feasibility Study, Livengood, Alaska: Technical report prepared by Samuel Engineering for International Tower Hill Mines Ltd., effective date September 4, 2013.

As a result of the changes to the Project evaluated in this Report, including differences in the economic parameters applied to the geologic block model that resulted in a change in resources (gold price, recovery, CAPEX, and OPEX), the original project as evaluated in the September 2013 Feasibility Study is no longer considered current, and therefore should no longer be relied upon by the reader.

2.4 Basis of the technical report

This Report presents a summary of the results of the PFS for the development of the Livengood Gold Project. THM requested engineering consulting group BBA Inc. to lead and perform the PFS, including contributions from a number of independent consulting firms, including NewFields Mining Design & Technical Services LLC, SRK Consulting (Canada and U.S.) Inc., Metal Mining Consultants Inc., and Wildcat and Badger LLC.

This Report was prepared at the request of Mr. Karl Hanneman, Chief Operating Officer (COO) of ITH. As of the date of this Report, ITH is an exploration and development company trading on the Toronto Exchange (TSX) under the trading symbol (ITH) and the New York Stock Exchange (NYSE.MKT) under the trading symbol (THM).

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Fairbanks, AK, USA
99701

Telephone: (907) 328-2800
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2.5 Study contributors

A summary of the PFS contributors and their general areas of input are presented in Table 2-1.

Table 2-1: Primary PFS contributors

Consulting Firm or Entity	Scope of Services
BBA Inc.	▪ Surface infrastructure design and capital costs ▪ Metallurgical testwork analysis, processing plant design ▪ Process plant capital and operating costs ▪ Environmental studies and permitting ▪ General and administration operating costs ▪ Financial analysis ▪ Overall NI 43-101 integration
NewFields Mining Design & Technical Services, LLC (“Newfields”)	▪ Geotechnical engineering ▪ Waste rock and water management ▪ Tailings management facility (TMF) design and capital costs ▪ Closure plan and costs
SRK Consulting (U.S.), Inc.	▪ Rock mechanics and mine slope stability
SRK Consulting (Canada) Inc.	▪ Geology, drilling and MIK model
Metal Mining Consultants Inc. (“MMC”)	▪ Resource estimation
Wildcat and Badger, LLC (“W&B”)	▪ Mine engineering ▪ Mine capital and operating costs ▪ Reserve estimation

2.6 Report responsibility and qualified persons

The individuals listed in Table 2-2, by virtue of their education, experience and professional association, are considered QPs as defined by NI 43-101, and are members in good standing of appropriate professional institutions. All persons and their respective companies listed are independent of ITH and THM, as defined by NI 43-101.

The QPs have supervised the preparation of this Report and take responsibility for the contents of the Report as set out in Table 2-2. Each QP has also contributed relevant figures, tables and portions of Chapters 1 (Summary), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References).

Table 2-2: Qualified persons chapter/section responsibilities

Qualified Person	Consultant	Site Visit	Chapter/Section Responsibility
Colin A. Hardie	BBA Inc.	August 15, 2016	Chapters 2, 3, 4, 5, 6, 13, 17, 18, 19, 20, 21, 22, 23, 24, and relevant portions of Chapters 1, 25, 26, and 27 with the exception of Section 18.17 and Sections 21.2.1 to 21.2.3 and 21.4.3.
Ryan T. Baker	NewFields Mining Design & Technical Services, LLC	March 1-2, 2012	Section 18.17, and the relevant portions of Chapters 1, 25, 26, and 27.
Michael E. Levy	SRK Consulting (U.S.) Inc.	June 20-22, 2012	Section 16.2 and the relevant portions of Chapters 1, 25, 26 and 27.
Timothy J. Carew	SRK Consulting (Canada) Inc.	May, 2012	Chapters 7, 8, 9, 10, 11, 12, Sections 14.1 to 14.9, and the relevant portions of Chapters 1, 25, 26, and 27.
Scott E. Wilson	Metal Mining Consultants Inc.	August 2, 2011	Chapter 14, with the exception of Sections 14.1 to 14.9, and the relevant portions of Chapters 1, 25, 26, and 27.
Timothy J. George	Wildcat & Badger, LLC	October 15, 2016	Chapters 15, 16, Sections 21.2.1 to 21.2.3 and 21.4.3, as well as the relevant portions of Chapters 1, 25, 26 and 27 with the exception of Section 16.2.

2.7 Personal inspection of the Livengood property

The QPs inspected the Livengood Property on the dates shown in the above Table 2-2.

2.8 Effective dates and declaration

The Report includes a number of effective dates as follows:

- Date of the Mineral Resource and Mineral Reserve estimate: August 26, 2016;
- Date of Financial Analysis: September 8, 2016;
- Latest project information: March 8, 2017.

The overall effective date of the Report is taken to be the date of the latest information on the project and is March 8, 2017.

As of the effective date of this Report, the QPs are not aware of any known litigation potentially affecting the Livengood Gold Project. The QPs did not verify the legality or terms of any underlying agreement(s) that may exist concerning the permits, royalties or other agreement(s) between third parties.

The results of this Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between THM and the QPs. The QPs are being paid a fee for their work in accordance with the normal professional consulting practice.

The opinions contained herein are based on information collected throughout the course of the investigations by the QPs, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results can be significantly more or less favorable.

2.9 Sources of information

The reports and documentation listed in Chapter 3 (Reliance on Other Experts) and Chapter 27 (References) of this Report were used to support the preparation of this Report. Additional information was sought from THM personnel where required. Sections from reports authored by other consultants may have been directly quoted or summarized in this Report and are so indicated, where appropriate.

2.9.1 General

This Report has been completed using the aforementioned sources of information, as well as available information contained in, but not limited to, the following reports, documents and discussions:

- Technical discussions with THM personnel;
- QPs' personal inspections of the Livengood gold property;
- Reports detailing mineralogical, metallurgical and grindability characteristics of the Livengood deposit, conducted by industry recognized metallurgical testing laboratories on behalf of THM;
- Resource Block Model provided by THM (SMU_Block_Model_Mar_30_2016);
- A conceptual process flowsheet developed by BBA based on the specific project testwork and similar operations;
- Internal and commercially available databases and cost models;
- Various reports covering site hydrology, hydrogeology, geotechnical and geochemistry;
- Internal unpublished reports received from THM; and
- Additional information from public domain sources.

2.9.2 BBA

The following individuals provided specialist input to Mr. Colin Hardie, QP:

- Mr. Jorge Torrealba, PhD (BBA), Mr. André Allaire, PhD, P. Eng. (BBA), and Mr. Guy Deschênes, PhD, P. Eng. (BBA), provided input to the comminution and metallurgical data interpretations as summarized in the Report (Chapters 13 and 17).
- Mr. Langis Charron, P. Eng. (BBA) and Mr. Jocelyn Marcoux (BBA) provided input on the process plant and infrastructure capital costs as well as input on the project construction strategies as summarized in the Report (Chapter 21).
- Mr. Claude Catudal (BBA) provided input on the project execution strategy and schedule as summarized in the Report (Chapter 24).

2.9.3 NewFields

The following individuals provided specialist input to Mr. Ryan Baker, QP:

- Mr. Troy Thompson, P.E. (Ecological Resource Consultants Inc.) provided input for the water balance analysis and storm water diversion channel sizing as summarized in the Report (Chapter 18).
- Mr. Ron Arlian, (NewFields) provided input for the tailings management facility, mine stockpiles, and surface water management infrastructure capital costs as summarized in the technical report (Chapter 21).

2.10 Currency, units of measure, and calculations

This Report assumes 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. Every effort has been made to clearly display the appropriate units being used throughout this 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 some tables and figures in this Report may be presented in metric units only to minimize the risk of data unit conversion errors.

Unless otherwise specified or noted, this Report uses the following assumptions and units:

- Currency is in US dollars (USD or $);
- All ounce units are reported in troy ounces, unless otherwise stated: 1 oz (troy) = 31.1 g;
- All metal prices are expressed in US dollars (USD or $);
- For financial modeling, ore tonnage is reported in short tons (t), with all costs reported in $/t;
- All cost estimates have a base date of the third quarter (Q3) of 2016.

This Report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs consider them immaterial.

2.11 Important notice

This Report is intended to be used by International Tower Hill Mines Ltd. subject to the terms and conditions of its agreements with BBA 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. The user of this document should ensure that this is the most recent Report for the property as it is not valid if a new Report has been issued.

2.12 Acknowledgements

The authors would like to acknowledge the general support provided by the THM management and Alaska development team personnel during this assignment. The Report benefitted from the knowledge and specific input of the following individuals:

- Thomas Irwin, Senior Advisor
- Karl Hanneman, Chief Executive Officer and Chief Operating Officer
- Chris Puchner, Chief Geologist
- Denise Herzog, Environmental Affairs Manager
- Debbie Evans, Corporate Controller

Their commitment, contributions, and teamwork are gratefully acknowledged and appreciated.

3. RELIANCE ON OTHER EXPERTS

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

Colin Hardie, 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 August 15, 2016 as set forth in Section 4.1, Appendix A, and the relevant portions of Chapter 1. The various agreements under which THM holds title to the mineral claims for this Project have not been reviewed by the Qualified Person, and the Qualified Person offers no legal opinion as to the validity of the mineral title claimed.

Colin Hardie, 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 August 15, 2016 as set forth in Sections 20.1, 20.3, 20.4 and the relevant portions of Chapter 1.

Colin Hardie, 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, Government and Investor Relations for THM, as of August 15, 2016 as set forth in Section 20.6 and the relevant portions of Chapter 1.

Colin Hardie, QP, relied upon THM for the information on taxes, royalties, and other government levies or interests applicable to revenue or income from the Livengood Gold Project, relevant to and incorporated into the financial model developed as of September 8, 2016 as summarized in Chapter 22 and the relevant portions of Chapter 1.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property description

The Livengood Gold Project covers approximately 48,300 acres (19,546 hectares), all of which is controlled by the Company through its wholly-owned subsidiary, Tower Hill Mines, Inc. (THM) (Figure 4-1). The Livengood Gold Project is comprised of multiple land parcels: 100% owned patented mining claims, 100% owned State of Alaska mining claims, 100% owned federal unpatented placer mining claims; land leased from the Alaska Mental Health Trust (AMHT); land leased from holders of State of Alaska mining claims, patented claims, federal unpatented lode and placer mining claims, and undivided interests in patented mining claims. The property and claims controlled through ownership, leases or agreements are summarized below. All of the 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 mining claims and the federal unpatented mining claims of the Hudson/Geraghty lease, none of the properties have been surveyed.

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 Nelson Mining Company, to operate a placer mine on MS 2447 through May 11, 2018;
- 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 Samuel Eaves and Patricia Eaves to operate a placer mine on MS 1956 through June 1, 2017;
- With respect to portions of U.S. Mineral Survey 1626, located on lower Amy Creek:
  - 100% of No. 2 Above Discovery Amy 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 of Alaska mining claims acquired by purchase. (Appendix A, Table A1);
- 153 State of Alaska mining claims acquired by location. (Appendix A, Table A2).

4.1.3 100% Owned federal unpatented placer mining claims

- 29 federal unpatented placer mining claims, subject to the December 2011 land purchase agreement described below. (Appendix A, Table A3)

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 mining claims, subject to the December 2011 land purchase agreement described below. (Appendix A, Table A4);
- 108 federal unpatented placer mining claims, subject to the December 2011 land purchase agreement described below. (Appendix A, Table A5);
- 24 State of Alaska mining claims, subject to the December 2011 land purchase agreement described below. (Appendix A, Table A6).

4.1.5 Leased property

- Alaska Mental Health Trust Lease. A lease of the AMHT mineral rights having a term commencing 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. A net smelter return (NSR) production royalty of between 2.5% and 5.0% (depending upon the price of gold) is payable to the lessor with respect to the lands subject to this lease. In addition, an NSR production royalty of 1% is payable to the lessor with respect to the unpatented federal mining claims subject to the lease described in the Hudson/Geraghty Lease below and an NSR production royalty of between 0.5% and 1.0% (depending upon the price of gold) is payable to the lessor with respect to the lands acquired by THM as a result of the purchase of LPI in December 2011. As of December 31, 2015, there were 9,970 acres (4.035 hectares) included in the AMHT lease;

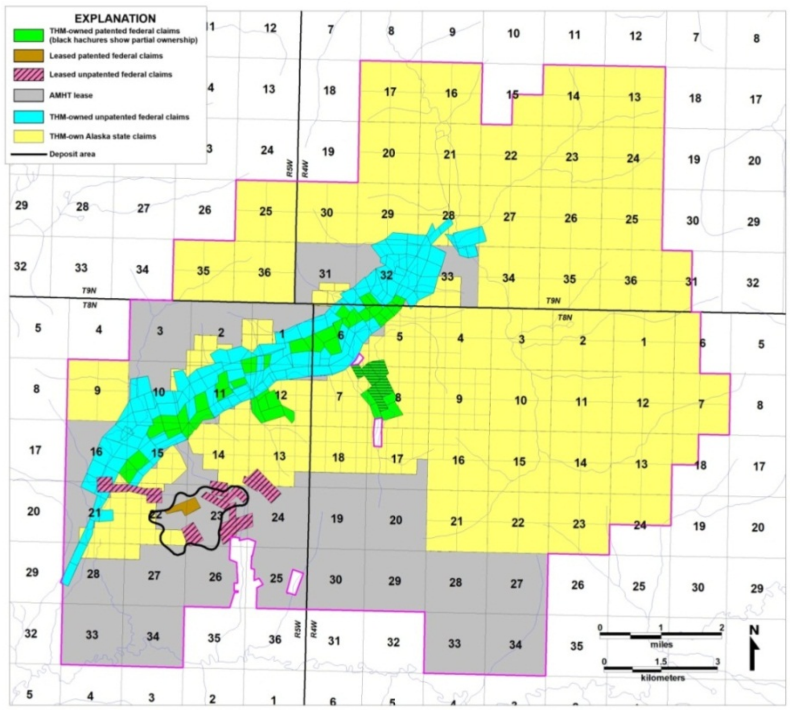


Figure 4-1: Map illustrating the company's Livengood Gold Project land holdings
(As of September 1, 2016 by tenure type, referenced to the Fairbanks Meridian township, range and section grid.)

- Hudson/Geraghty Lease. A lease of 20 federal unpatented 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 A, Table A7);
- 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 mining 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, $125,000 payable 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 A, Table A8).

4.1.6 Patented mining claims (undivided interests less than 100%)

- An undivided 5/6th 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 5/9th 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.

4.1.7 Other land obligations

State of Alaska mining claims

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 per 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 four for each 160-acre claim. As a consequence of the annual rentals due, all State of Alaska 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 1st each year) or cash-in-lieu of labor. An affidavit evidencing that such work has been performed is required to be filed on or before November 30th 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 mining claims can be held indefinitely. The work completed by THM during the 2015 field season was filed as assessment work, and the value of that work is sufficient to meet the assessment work requirements through September 1, 2019 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.

Federal unpatented mining claims

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.

Water and land use considerations

Holders of State of Alaska and federal unpatented mining claims have the right to use the land and 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, the Company 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 Livengood Placers, Inc. The aggregate consideration for the claims and rights was $13,500,000 in cash plus an additional payment based on the five-year average daily gold price ("Average Gold Price") from the date of the acquisition ("Additional Payment"). The Additional Payment equaled $23,148 for every dollar that the Average Gold Price exceeded $720 per troy ounce. If the Average Gold Price was less than $720, there was no additional consideration due. As at December 12, 2016, the five-year average daily gold price was $1,354.79 resulting in a derivative liability of $14,694,169. The obligation to make the contingent payment was secured by a Deed of Trust over the rights of the Company in the purchased claims in favor of the vendors. On December 28, 2016, the Company closed a non-brokered private placement financing of 45,833,334 common shares at a price of $0.48 per share for gross proceeds of $22.0 million. On January 12, 2017, the Company paid $14,694,169 for the timely and full satisfaction of the final derivative payment due with respect to the acquisition of these certain mining claims and related rights in the vicinity of the Livengood Gold Project.

The subject ground was previously vacant or was used for placer gold mining. No placer mineral reserves or mineral resources have been established on the ground subject to this agreement. However, records exist for 2,370 placer drillholes 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 drillholes. The 1,425 drillholes completed between 1984 and 2000 were 8-in RC rotary drillholes utilizing a center return tri-cone bit. All lands controlled by the Company, including the lands acquired pursuant to this agreement, were evaluated as appropriate for integration into the September 2016 PFS for the Livengood Gold Project.

4.1.8 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 (hard rock exploration, temporary water use), U.S. Bureau of Land Management (plan of operations), U.S. Corps of Engineers (Section 404 and nationwide wetlands), Alaska Department of Environmental Conservation (Section 401, storm water), and Alaska Department of Fish and Game (fish habitat) authorizations. Permits required to support project development are discussed in Chapter 20.

4.1.9 Environmental liabilities

With over 100 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. THM 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 mi (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).

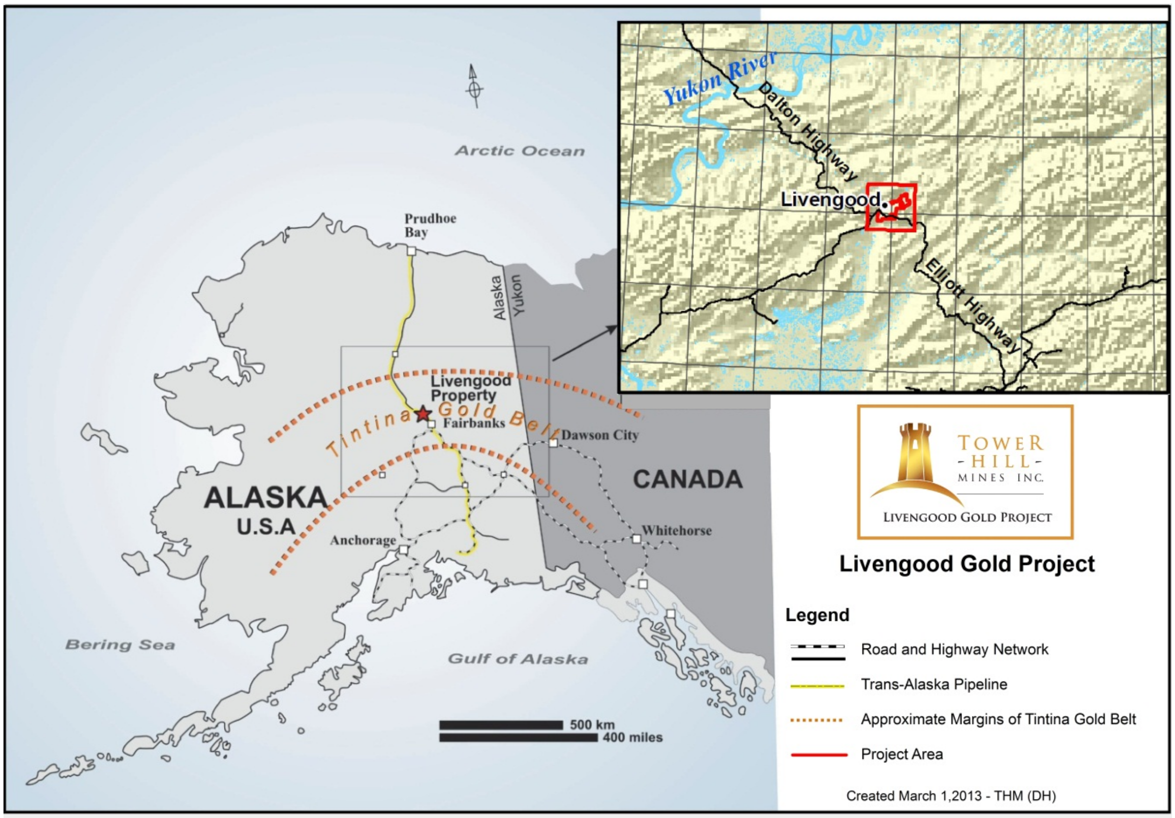


Figure 4-2: Project location map

5. 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-ft (914 m) runway is located 3.7 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 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°F to 72°F (7°C 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 in the order of 15.7 in (400 mm) water equivalent. Winter 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 population 223) is approximately 40 mi (64 km) southwest of the Project, and Manley Hot Springs (2012 population 116) is approximately 80 mi (129 km) southwest of the Project area at the western terminus of the Elliott Highway. The Fairbanks North Star Borough 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 two paved highways, a railroad, an interlinked electrical grid, and communications infrastructures. 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 55 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 Consulting 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). A water balance study was completed by Ecological Resource Consultants (ERC) based on available and collected data. The study indicates 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, waste 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 conspicuous, particularly in Livengood Creek and lower Goldstream Creek.

6. 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. From 1914 through the 1970s, 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, primarily in the 1950s and in the form of dozer trenches, was carried out for lode mineralization in the vicinity of Money Knob. However, no significant lode production has occurred to date.

Modern corporate exploration for lode gold mineralization in the vicinity of Money Knob and the Livengood placer deposits was initiated in 1976, continued intermittently though 1999, and included extensive soil sampling, trenching and 25 shallow drillholes. 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 these results, prior soil surveys, and geological concepts, four diamond core holes were drilled in late 2004. The two drill programs intersected broad and 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. Concomitant programs have included geotechnical, engineering and metallurgical work, along with the collection of environmental baseline data. As of the end of 2014, AGA and THM completed exploration and delineation drilling totaling 575,078 ft (175,284 m) in 604 RC holes and 138,726 ft (42,284 m) in 149 core drillholes.

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 the ITH press release dated February 27, 2012.

There are no known historical mineral resource estimates for hard rock minerals.

7. 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.

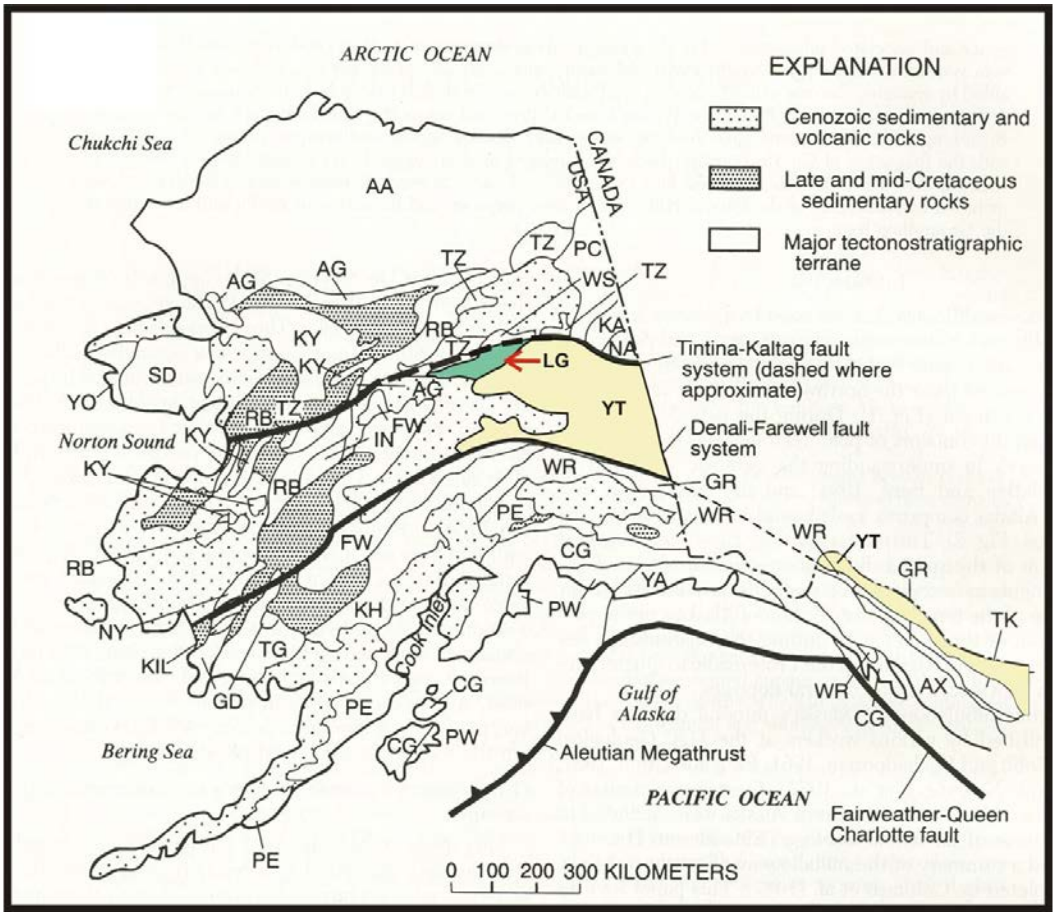


Figure 7-1: Terrane map of Alaska showing Livengood Terrane (LG: red arrow)

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 920 ft (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 (Figure 7-2 and Figure 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.

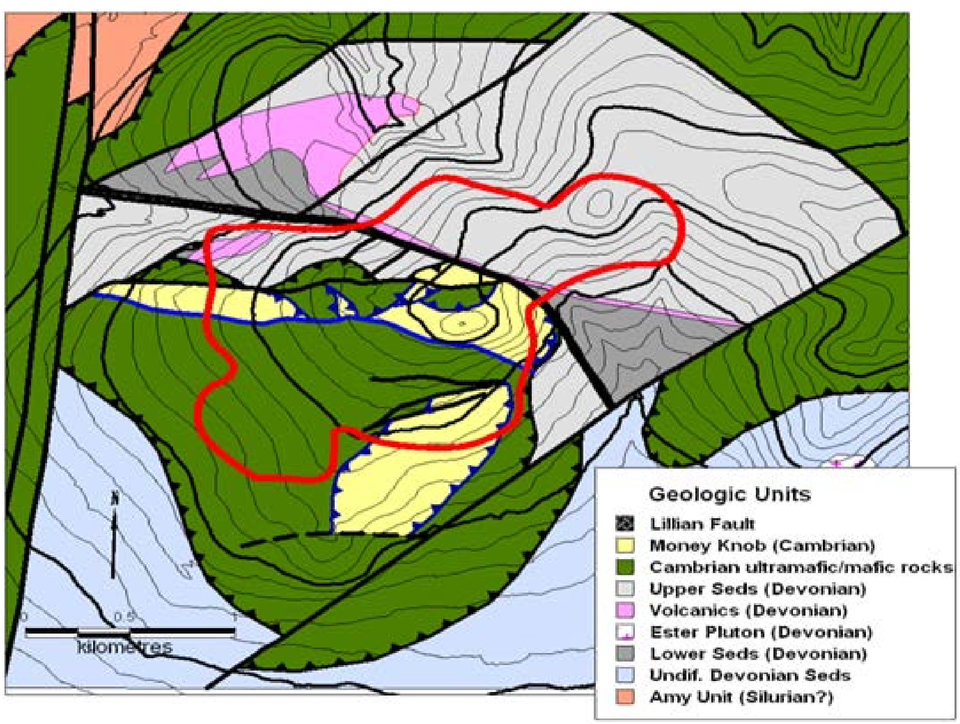


Figure 7-2: Generalized geologic map of the Money Knob area based on geologic work by THM
(Red outline is the surface projection of the gold deposit)

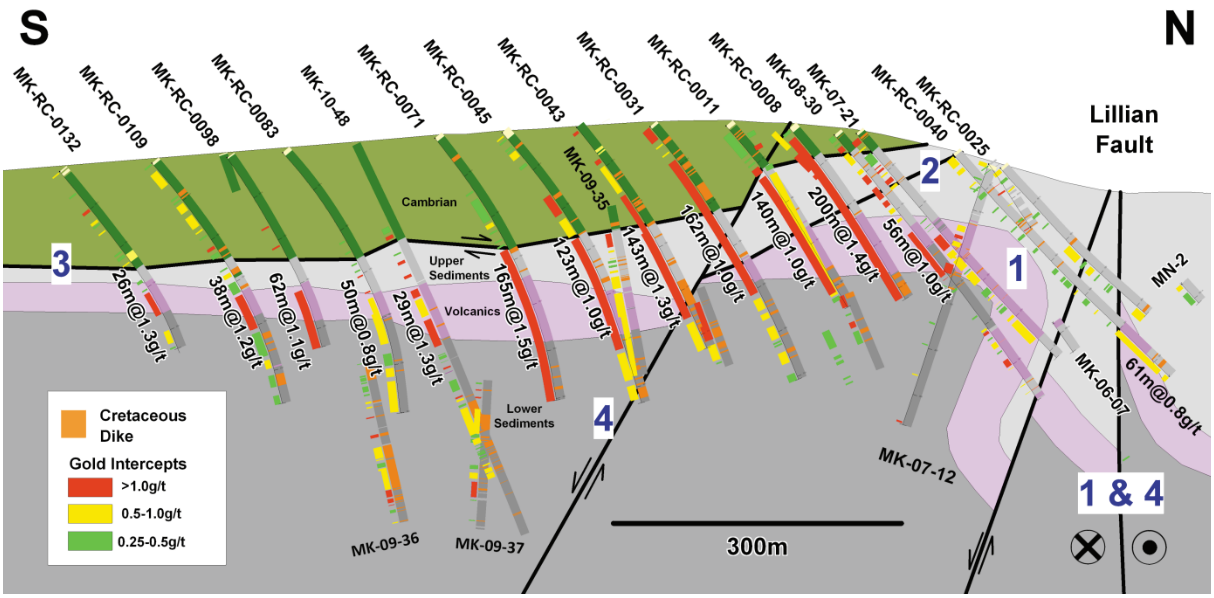


Figure 7-3: Cross section through the deposit

(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. 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. EXPLORATION

9.1 Exploration history

Multiple companies have explored the Livengood area as outlined in Chapter 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) by 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.

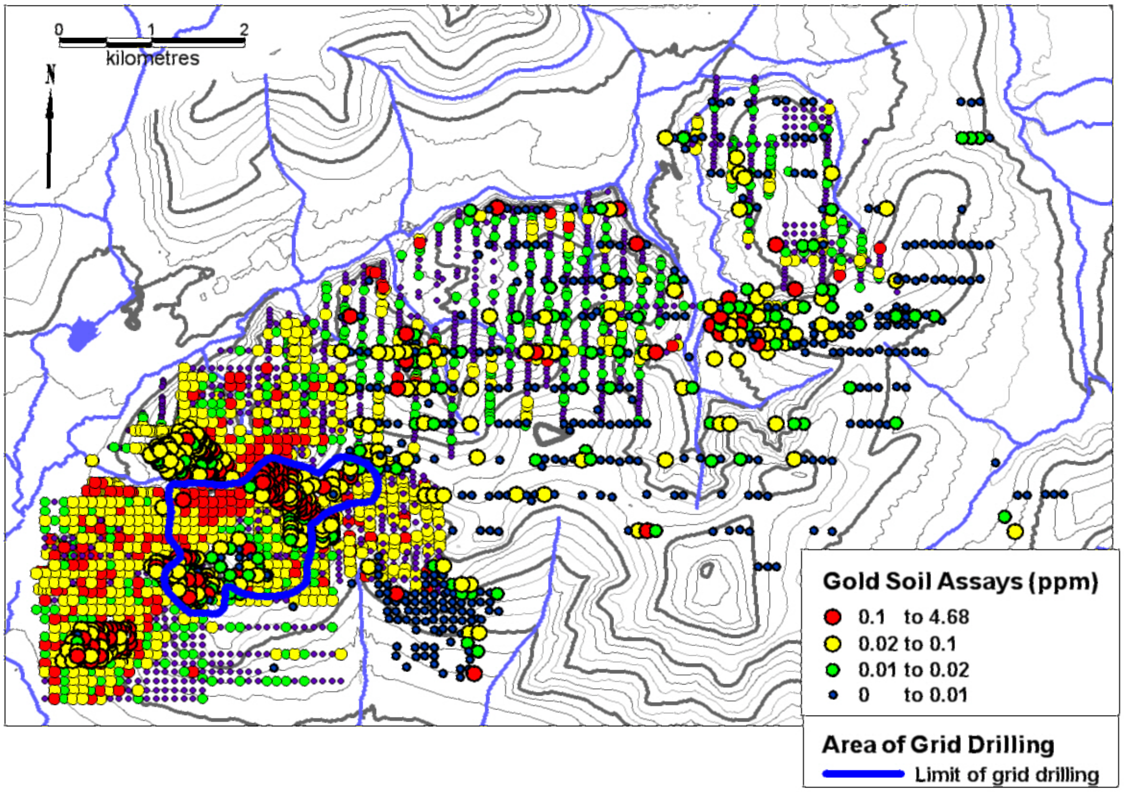


Figure 9-1: Plot of gold values in soil samples

(The surface projection of the known deposit is outlined in blue in the lower left corner of the figure.)

10. 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 2012 season, THM and others have completed a total of 717,435 ft (218,674 m) of exploration and delineation drilling, of which 574,599 ft (175,138 m) was RC drilling and 140,854 ft (42,932 m) was core drilling.

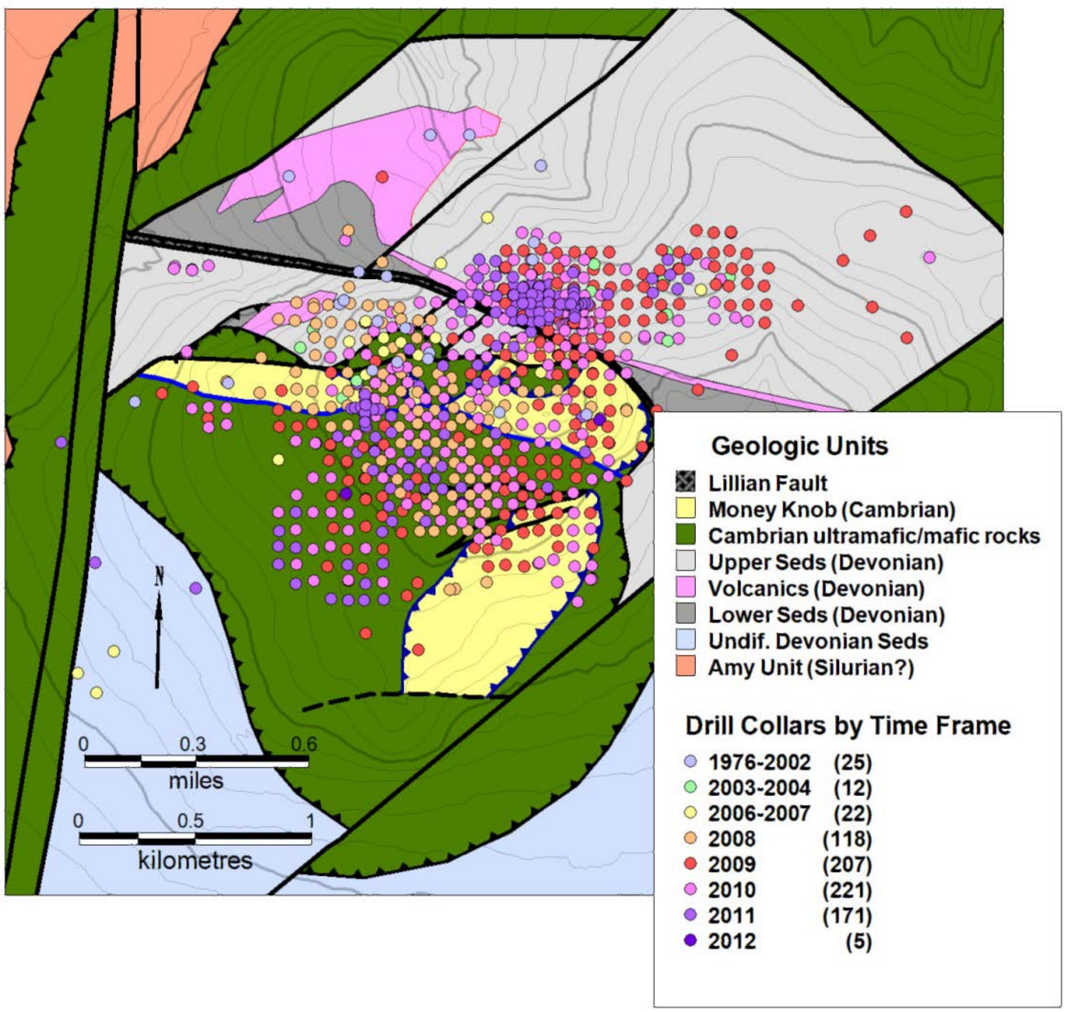


Figure 10-1: Distribution of resource / delineation drill holes in Money Knob area over time
(All holes completed after 2004 were drilled by THM. Drilling illustrated through 2011 were dedicated to exploration and delineation; 2012 holes shown are geotechnical.)

Nearly all resource drill holes at Livengood have been drilled in a northerly direction at an inclination of -50° (RC) and -60° (core), 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 and 246 ft (75 m) apart; subsequent infill drilling in the center of the 246 ft (75 m) square 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 downhole 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 a 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^{TM} or the EZMark^{TM} 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 of the collar. Downhole 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^{TM} 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 downhole contamination. These factors are addressed in the chapter on data verification (Chapter 12).

11. SAMPLE PREPARATION, ASSAYING 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 in Chapter 10, 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 drillhole 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 ALS 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 accreditations.

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 against 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 sample 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 1 kg 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 that 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 drillhole database containing all the data collected in the field, including drillhole locations, downhole 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 drillhole database utilizing custom queries. These data include RC drill logs and NITON™XRF geochemistry, collar locations and gyroscopic downhole 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 drillhole 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

Mr. Tim Carew, P.Geo., of SRK (Canada) has witnessed and reviewed sample and data collection in the field, inspected ALS' 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 drillhole data. Mr. Carew is satisfied that the THM data collection, management and verification procedures are adequate and diligently followed.

12. DATA VERIFICATION

12.1 Third party confirmation

In addition to the reviews described in Chapter 11, Mr. Tim Carew, P.Geo., of SRK (Canada), 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 and both RC and core drillholes 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). Mr. Carew 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). Mr. Carew has not verified all sample types or material reported, but to the best of his knowledge, THM has been diligent in their sampling procedures and efforts to maintain accurate and reliable results.

12.2 Reverse circulation vs core drilling

On other projects, the use of reverse circulation (RC) drilling beneath the water table has resulted in inaccurate assay data, due to cyclicity and/or downhole 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 drillhole prior to the addition of drill rods, which 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 six holes and one entire drillhole (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 downhole, 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 drillholes is statistically comparable; significant downhole 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 suggests that, on average for the deposit, core gold grades (split HQ) 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.

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%**

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 drillholes 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 samples. All Area 50 samples were composited to 16.4 ft (5 m) 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).

Because the gold at Livengood is relatively coarse, the relative sample volume (e.g. RC with a 5 in (127 mm) diameter, whole PQ core with a 3.3 in (83 mm) diameter, and HQ core with a 2.4 in (61 mm) diameter that has been halved) is likely the root cause of the grade discrepancies between core and RC, due to the nugget effect. Split 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.

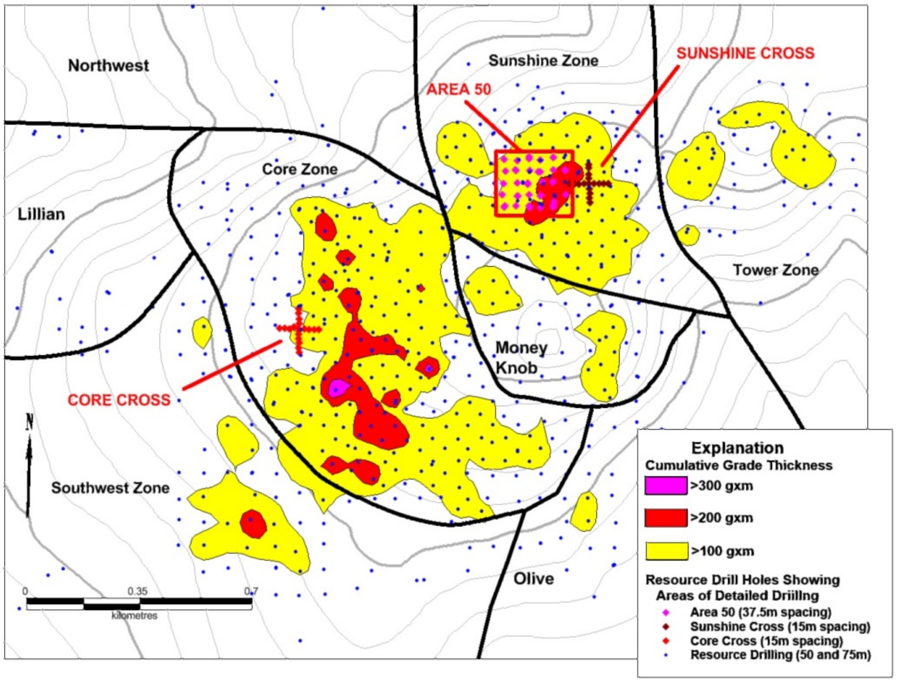


Figure 12-1: Map showing location of areas of detailed drilling
(Area 50, Sunshine Cross and Core Cross)

In addition, the 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 partially oxidized, the carbonate and sulfide is leached out, rendering the veinlets friable with the core often breaking along them. The most probable explanations for the greater discrepancies in grade in the Sunshine Zone above the water table 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%.

Table 12-2: Calculated resources for Area 50 by drill sample type
(Ordinary kriging of 32.8 ft (10 m) composites, 0.25 g/mt cut-off)

Drill Sample Type	Tonnes (Mmt)	Tonnage Ratio	Au Grade (g/mt)	Grade Ratio	Au (oz)	Au 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

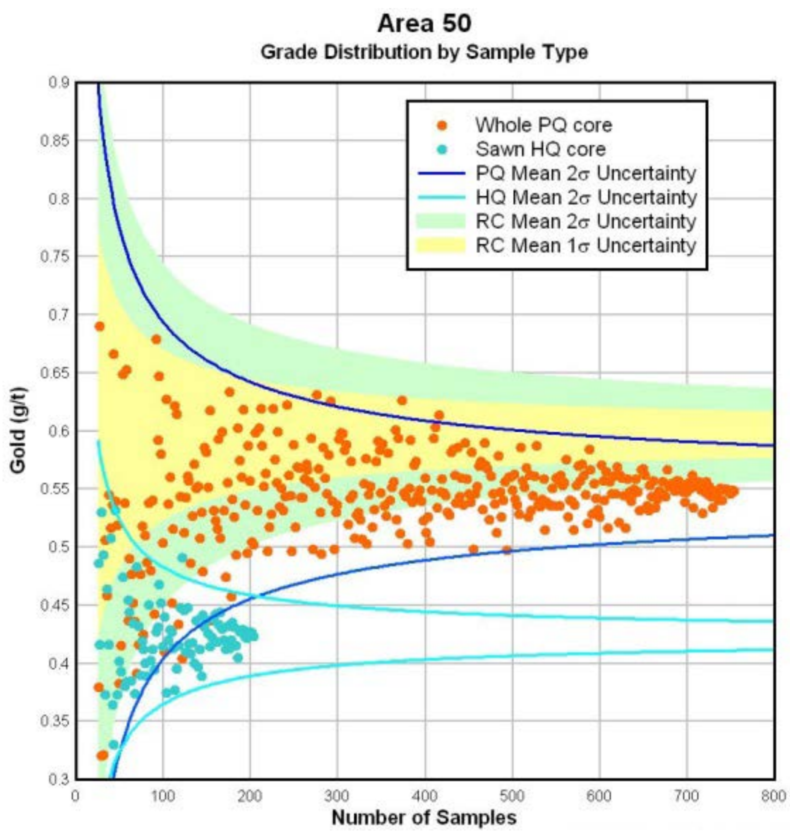


Figure 12-2: Models for RC, Whole PQ, and Sawn HQ from Area 50
(Based on 869 RC Composites, 753 PQ Core Composites, and 203 HQ Core Composites (all composited to 16.4 ft (5 m)). The modeled grade means for the RC, PQ and HQ composites in Area 50 are 0.597, 0.549 and 0.424 g/mt gold, respectively.)

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 640 ft (195 m) by 787 ft (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 ounces of gold is negligible (less than 1%; see Table 12-3), indicating that current grid spacing adequately defines the resource.

Table 12-3: Calculated resources for the Core Cross, Sunshine Cross and Area 50
(Ordinary kriging, 0.25 g/mt cut-off)

Area, Drillhole Spacing [1]	Tonnes (Mmt)	Tonnage Ratio (all/grid)	Au Grade (g/mt)	Grade Ratio	Au (oz)	Au Ratio (all/grid)
Core Cross, 50 m grid & 15 m infill	15.67		0.481		242,401	
Core Cross, 50 m grid drilling only	15.37	1.020	0.477	1.008	235,715	1.028
Sunshine Cross, 50 m grid & 15 m infill	9.82		0.553		174,647	
Sunshine Cross, 50 m grid drilling only	9.81	1.001	0.566	0.977	178,556	0.978
Area 50, all drilling (37.5 m)	16.04		0.562		289,685	
Area 50, 50 m grid drilling only	16.13	0.994	0.550	1.022	285,136	1.016
All areas (averages)		**1.005**		**1.002**		**1.007**

[1] 1 m = 3.28 ft

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This chapter presents both the pertinent results from the testwork leading up to the 2013 feasibility study (FS) as well as the post-FS test results that were obtained leading up to the 2016 pre-feasibility study (PFS). The chapter begins with an outline of sample selection and preparation for the FS test programs (Section 13.2). This is followed by a discussion on the mineralogy and gold deportment of the Livengood gold ore rock types (Section 13.3), work that had been completed for the FS.

Comminution testing and the results of grinding simulations as they relate to mill circuit design and throughput estimation are covered in Section 13.4. Comminution testing was conducted in the following test programs:

- FS – Design Comminution Test Program;
- FS – Variability Comminution Test Program;
- PFS – SMC Testwork (2015-2016).

Metallurgical testing results and how these relate to back-end (post-comminution) plant design are discussed in Section 13.5. Metallurgical testing was performed in the following test programs:

- FS – Optimization Test Program;
- FS – Variability Test Program;
- PFS – Continuous Test Program;
- PFS Phase 7 – Assay procedures and water source testing;
- PFS Phase 8 – Grind, leach recovery, gravity, flotation testing;
- PFS Phase 9 – SGS and FLS / Curtin University test program, grind, leach recovery, gravity testing;
- PFS Phase 10 – Stirred tank reactor (STR) testing of rock types RT7 and RT9.

The metallurgical testwork chapter includes discussion on gravity recovery, flotation, leach pre-conditioning, carbon in leach (CIL), intensive leach (IL) testing, settling, cyanide detoxification and other topics as they relate to plant design. Phase 9 represents the first test program that used RC rig duplicate rock chips in composite samples. All other test programs were based on drill core composite samples.

The chapter closes with a discussion on recovery equations (Section 13.6) and consolidates all testwork conclusions and a number of trade-offs as they relate to process flowsheet development (Section 13.7); potential opportunities for future testwork are then given (Section 13.8).

13.2 FS – Sample selection and preparation

As part of the work leading up to the FS, samples were selected by THM and RPA (Altman, K., 2013) and submitted to SGS for design and variability comminution composite preparation (Tajadod, J. and Lang, J., 2013).

Sample selection focused on the preparation of large bulk composite samples, which were used for flowsheet optimization testing and comminution testing. A number of variability samples were selected to test the variation in the orebody and to examine how the metallurgical response changes based on the feed grade for each of the rock types.

A mine production schedule that was developed prior to the 2013 FS was used to establish average gold grade targets to help guide the sample selection.

SGS Vancouver received two shipments in February and March 2012, originating from the Livengood property and submitted by THM. The material that was shipped was composed of approximately 3,000 individual samples, which were used for the optimization, design comminution and variability testing (Table 13-1).

Table 13-1: Livengood gold ore sample selection weights (kg) used in the FS test programs

FS Test Program	Sample weight (kg)
Optimization	4,800
Design comminution	2,700
Variability	3,000

The Livengood rock types were identified on the basis of their lithology. The six rock types identified in Table 13-2 below accounted for 100% of the reserve at that time.

Table 13-2: Definition of Livengood rock types (FS)

Rock Type	Description	% Ounces (of P&P) (1)	% Tons
RT4	Cambrian	13.1	13.9
RT5	Upper Sediments – Sunshine Zone	23.5	28.2
RT6	Upper Sediments	19.5	18.4
RT7 Bleached	Lower Sediments – South of Lillian Fault	13.5	12.1
RT8	Volcanics – North of Lillian Fault	1.9	2.0
RT9	Volcanics – South of Lillian Fault	28.5	25.4

(1) Proven & Probable

During the FS, rock type RT7 was further designated as “bleached” or “unbleached” material to account for the differences in the alteration and other factors of the samples. RT7 unbleached was not included in potential ore. The sample compositing instructions did contain some errors, so some of the RT7 samples were mixed up and in other cases bleached and unbleached material was combined.

For the design comminution test program, each sample interval was selected and added to the composite, blended, and homogenized. From each composite, 20 rocks (-3 /+2 in) were selected for the Bond low-energy impact (CWi) test. Each composite was then crushed to 100% minus 2½ in and 65 kg was split for the JK Drop Weight (DWT) test. The remaining sample was crushed to nominal 1¼ in and 5 kg was split for the Bond abrasion (Ai) test. The remaining sample was stage-crushed to ½ in and 15 kg was split for the Bond rod mill grindability (RWi) test. Finally, the remaining sample was stage-crushed to 6 mesh and 10 kg was split for the Bond ball mill grindability (BWi) test. The FS Design comminution sample preparation flowsheet is illustrated in Figure 13-1.

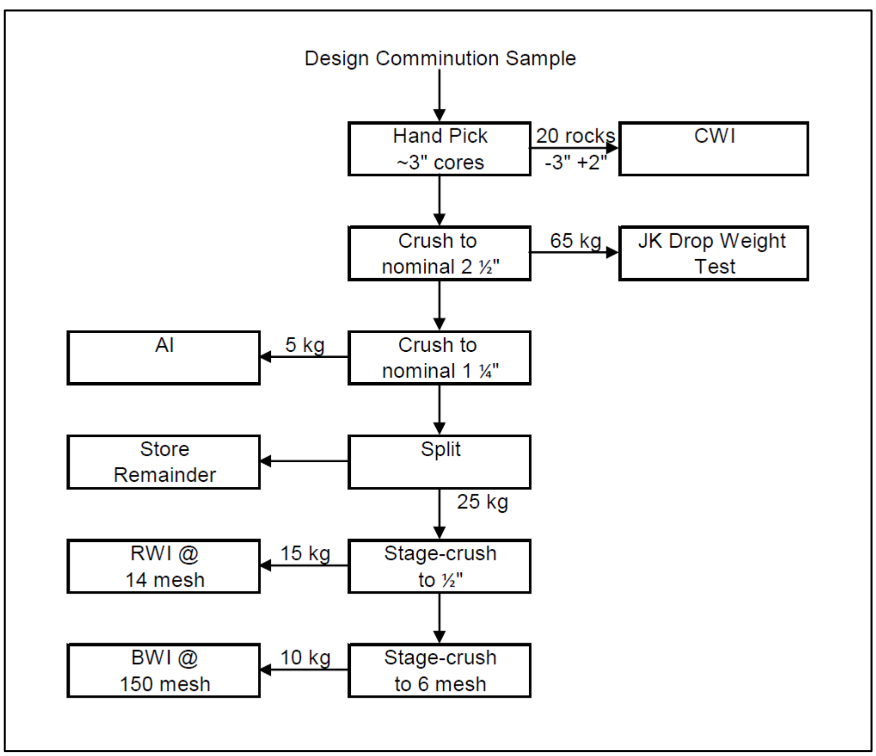


Figure 13-1: FS Design comminution sample preparation flowsheet (SGS report)

For the comminution portion of variability testing, each sample interval was selected and added to the composite, blended and homogenized. Every sample was crushed to nominal 2½ in and 10 kg was split for the SPI test. The remaining sample was stage-crushed to nominal 6 mesh, blended, and a 10 kg portion was split for the BWi test. The comminution variability sample preparation flowsheet is illustrated in Figure 13-2.

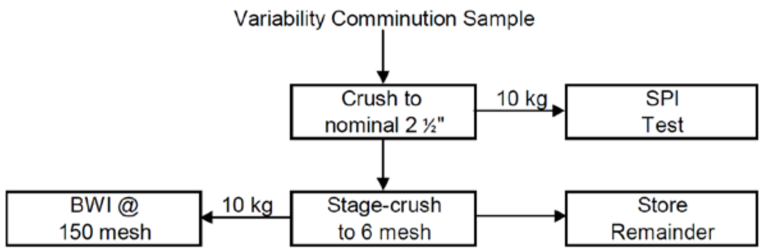


Figure 13-2: FS Variability comminution sample preparation flowsheet

Both the design comminution and variability samples were selected from the major rock types (RT4, RT5, RT6 and RT9). Rock types RT7-Bleached, RT7-Unbleached and Stibnite were also tested in the comminution variability test program. RT8 was not tested in any of the test programs. RT7-Bleached and RT7-Unbleached labels were later removed and sample results were combined and renamed RT7.

13.3 FS – Mineralogy and gold deportment study

SGS (Wang, Z. and Prout, S., 2013) undertook a high definition mineralogical examination of the Livengood samples that were used for the FS metallurgical testwork. Examination of four samples, which were identified as RT4, RT5, RT6, and RT9, was carried out using 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 iron sulfides and gold-bearing minerals, as well as to complete a mass balance of microscopic gold.

The RT4 sample consisted of carbonates (22.5%), talc (18.6%), quartz (16.0%), 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 accounted for 1.9% and pyrite for 0.9%. Gold minerals were tentatively quantified in the sample at less than 0.001%.

The RT5, RT6 and RT9 samples consisted 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 accounted for 2.9-10.5%, arsenopyrite (1.0-1.4%). Gold minerals were tentatively quantified in the samples at less than 0.001%.

In the four samples, gold occurred mainly in its native form (defined as Au 75-100%), and carried an average of 90.8-93.5 wt% Au, while all other elements were less than 1.0 wt%.

The results of the 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 had pyrite as the dominant sulfide mineral over arsenopyrite. Rock type RT9 maintained this trend, but with <10% arsenopyrite (relative to pyrite) 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 showed arsenopyrite to be the dominant sulfide mineral. However, the abundance of sulfide minerals was generally 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 Livengood Gold Project demonstrated a consistent trend for the majority 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 the gravity tailings.

13.4 Comminution testing

Comminution testwork programs were completed as part of the FS and the current PFS. In both cases, the objective of the programs were to generate the information needed to size the crushing and grinding circuits for the Livengood Gold Project.

13.4.1 FS – Comminution testing

Comminution testing was performed on samples that comprised part of the optimization samples, as well as the variability samples. Samples were selected based on the potential mill supplier's recommendations.

Design comminution samples were prepared in accordance with Figure 13-1. A total of 12 DWTs were performed on rock types RT4, RT5, RT6, RT7-Bleached, RT7-Unbleached and RT9. Priority was given to the DWT, due to limitations in the availability of PQ core.

A total of 36 samples were prepared for comminution testing, including: Bond Work index (BWi), Rod Work index (RWi), Crusher Work index (CWi) and Abrasion index (Ai). These indexes were applied in the crusher and mill sizing calculations as well as for determination of consumables, such as balls and liners.

Additional SAG power index (SPI) and BWi tests were completed using variability samples. The total number of BWi tests was 136.

The average BWi, RWi, CWi and Ai for each of the above rock types are presented in Table 13-3.

Table 13-3: Comminution data (FS)

Rock Type	Work Index Metric (kWh/mt)			
	BWi	**RWi**	**CWi**	**Ai**
RT4	12.3	13.1	13.3	0.14
RT5	11.9	15.7	14.1	0.15
RT6	14.4	17.3	14.4	0.12
RT7	14.1	14.5	7.7	0.17
RT9	14.3	16.3	7.4	0.35
Total Number of tests	**136**	**26**	**48**	**48**

JK Drop Weight (DWT) tests were performed on selected rock type samples. The data obtained was analyzed to determine the JKSimMet comminution parameters. These parameters were combined with equipment details and operating conditions to analyze and/or predict grinding circuit performance. While the A and b values of the DWT are not independent and cannot be used for direct comparison between ore types, their product (A×b) provides a good parameter for comparison. Lower A×b values indicate a higher resistance to abrasion breakage and also a greater resistance to impact breakage. Table 13-4 below, shows the average A and b values for each rock type. The results indicated that RT4 and RT7 would require less comminution energy than the other rock types. The numbers are indicative of a medium hard rock type.

Table 13-4: Average JK drop weight parameters by rock type (FS)

Rock Type	Number of Tests	A	b	A×b
RT4	2	62.1	0.83	51.5
RT5	2	67.6	0.50	33.8
RT6	2	50.7	0.64	32.5
RT7	4	55.4	0.89	49.3
RT9	2	60.5	0.58	35.4
Total tests	**12**			

13.4.1.1 FS – JKSimMet simulations

Analysis of the JK Drop Weight parameters was performed by Mark Richardson of Contract Support Services (CSS) using JKSimMet, a software package used to analyze the grinding circuit, which was comprised of a single (D×L) 40 ft × 25 ft SAG mill, followed by two (D×L) 28 ft × 45 ft ball mills, with a pebble crusher operated in closed circuit with the SAG mill. Following optimization, the JKSimMet results led to the conclusion that the selected circuit would process about 92,600 t/d (84,000 mt/d).

It should be noted that one vendor recommended the use of a (D) 42 ft SAG mill to achieve the target throughput. Consideration was given to this size of mill, but it was decided that a "first of its kind" (D) 42 ft SAG was not warranted, due to a lack of reference sites with proven track record in the industry at the time of the FS.

After further consultation, the JKSimMet model was rerun using the following new parameters:

- Circuit target grind of 90 µm (P_{80});
- Daily throughput of 100,000 t/d (90,718 mt/d);
- BWi (14.3 kWh/mt) corresponding to the 75th percentile of LOM hardness.

The simulation resulted in a circulating load of 15% through the pebble crusher and a circulating load of 350% running through the ball mill circuit. The proposed circuit used a single (D × L) 40 ft × 26 ft SAG mill with 27 MW of installed power and two 28 ft × 46 ft ball mills with 29.5 MW of installed power each.

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

13.4.2 PFS – Comminution testing

BBA completed a review of the FS comminution testwork ("Comminution testing of samples from the Livengood Property." SGS report 50223-001-Phase III, com Report 3. February 26, 2013). Based on the review, BBA made the recommendation to carry out additional comminution testwork (SMC testing) to increase the level of confidence in the parameters used to design the grinding circuit and gain further insights into the variability of the Livengood gold ore's comminution properties.

13.4.2.1 PFS – SMC testwork program (2015-2016)

SMC testwork was performed in January 2016 at SGS Vancouver to increase understanding of the ore variability by rock type in support of the grinding circuit development.

Ten composites were prepared for each rock type. Each composite was made up of several drill core intervals. The composite weights ranged from 12 to 26 kg. The samples making up a composite were all properly bagged and labeled according to the rock type (e.g. RT4) and composite number (1-10), i.e. RT4-1, RT4-2, up to RT4-10. The samples that made up the composites were bagged and labeled according to drillhole number and sample number. All samples within a composite came from a single drillhole.

BBA requested that Stephen Morrell (SMC Testing®) be engaged to assist in calibrating the SMC test results using the DWT data from the 2013 FS. This procedure is a required step in BBA's practices to ensure that the calibration of the SMC results is performed using data from DWT tests from the same deposit and ore types as opposed to using generic databases available through JKTech (owners of JKSimMet). The calibration of the SMC results for each of the rock types (RT4, RT5, RT6, RT7, and RT9) was completed using the DWT data that corresponded to each specific rock type.

Table 13-5 shows the average as well as the 50^{th} and 80^{th} percentile results of the SMC testwork. Based on BBA's experience and internal database, the RT5 and RT9 ore could be classified as hard, as the 50^{th} percentile (50^{th}) of the A×b data is in the low 30s. On this same basis, rock types RT6 and RT7 would be considered medium hard (50^{th} in the 40s), and RT4 would be considered the softest of the rock types present in the Livengood deposit (50^{th} = 73).

It is important to note that for most of the rock types there is only a small difference between the 50^{th} and 80^{th} A×b values. With the exception of RT4, the results suggest that there will not be significant grinding throughput variability from one rock type to another. In the case of RT4, ore blending with other rock types should be considered to moderate this issue.

When comparing the comminution results from the PFS to the 2013 FS, the 50th A×b results for the RT4 rock type was lower (the ore was softer) than the average of the DWT results from the FS. The 50th A×b results for rock types RT5, RT6, and RT9 were of the same order as the average of the DWT results from the FS. In the case of RT7, the 50th was slightly harder than the average of the DWT results from the FS.

Table 13-5: SMC testwork statistical analysis (PFS)

RT	Number of tests	Samples ID	A×b average	A×b 50th percentile	A×b 80th percentile
4	10	RT4-1 to 10	75.0	73.1	57.1
5	10	RT5-1 to 10	36.7	33.4	29.3
6	10	RT6-1 to 10	44.4	38.5	31.3
7	10	RT7-1 to 10	47.9	40.1	33.8
9	10	RT9-1 to 10	37.9	36.7	31.8
Total	**50**				

13.4.3 Testwork summary for crushing and grinding circuit design

A database was prepared with all available results from both the FS and current PFS comminution testwork. Table 13-6 and Table 13-7 present the results of a statistical analysis by rock type using the results from the FS and PFS programs.

Table 13-6: Comminution test statistical analysis by rock type

Percentile	Rock Type	SG (g/cm^3)	JK Drop Weight Parameters (DWT and SMC test)		CWi	RWi	BWi	Ai
			A × b	ta	kWh/mt	kWh/mt	kWh/mt	g
50th	RT4	2.73	65.2	0.72	14.5	13.4	12.0	0.13
	RT5	2.68	33.4	0.36	14.2	15.7	12.0	0.15
	RT6	2.73	36.7	0.41	15.1	17.6	13.0	0.10
	RT7	2.71	42.7	0.41	7.9	14.2	12.1	0.15
	RT9	2.74	36.0	0.38	6.9	16.3	13.7	0.29
80th	ALL	2.77	32.0	0.61	15.5	17.1	13.7	0.25

Table 13-7: Comminution test statistics using all FS and PFS testwork data

Statistic	SG (g/cm^3)	JK Drop Weight Parameters		CWi	RWi	BWi	Ai
		A × b	ta	kWh/mt	kWh/mt	kWh/mt	g
Max	2.87	23.8	1.26	19.7	19.1	14.9	0.59
90%	2.79	28.9	0.79	17.0	17.9	14.3	0.33
80%	2.77	32.0	0.61	15.5	17.1	13.7	0.25
75%	2.76	33.0	0.58	14.3	16.4	13.4	0.20
50%	2.71	41.0	0.41	9.4	14.9	12.6	0.16
25%	2.66	52.9	0.32	7.2	13.1	11.7	0.10
10%	2.57	78.6	0.28	5.8	11.6	11.1	0.08
Min	2.39	121.0	0.23	4.8	11.2	10.2	0.05
Average	**2.70**	**47.1**	**0.49**	**10.8**	**14.8**	**12.6**	**0.18**

Crushing circuit simulations used the 80^{th} percentile of the Crusher Work index (CWi) (Table 13-7).

Originally, the 80^{th} percentiles of the DWT and BWi of the hardest ores (RT5 and RT9) were used by BBA to estimate the initial grinding circuit design parameters. This was because SMC data was not available at the time. The final grinding circuit design parameters (Table 13-8) were taken from the data point (RT6 sample ID DC5) that was closest to the 80^{th} percentile of the A×b values of rock types RT5 and RT9. For design purposes, those results were considered the 80^{th} percentile. Note that this same test sample's BWi value was also used for design purposes (13.1 kWh/mt).

Figure 13-3 (A×b for DWT and SMC) and Figure 13-4 (BWi) show the cumulative distributions from the comminution testwork programs. Figure 13-3 indicates the preliminary A×b design point as (Design_RT5&RT9_DWT). Similarly, Figure 13-4 indicates the preliminary BWi design point as (Design_RT5&RT9_DWT).

Table 13-6 shows the 50^{th} percentile of the Abrasion index (Ai) for each rock type. The Ai values are classified as medium-low in abrasiveness and were used to calculate media consumption

Table 13-8: Grinding circuit design values

Percentile	Rock Type	A × b	ta	RWi	BWi
				kWh/mt	
80^{th}	RT5+RT9	29.6	-		11.9
Design Value DWT	RT6	29.3	0.58	17.1	13.1

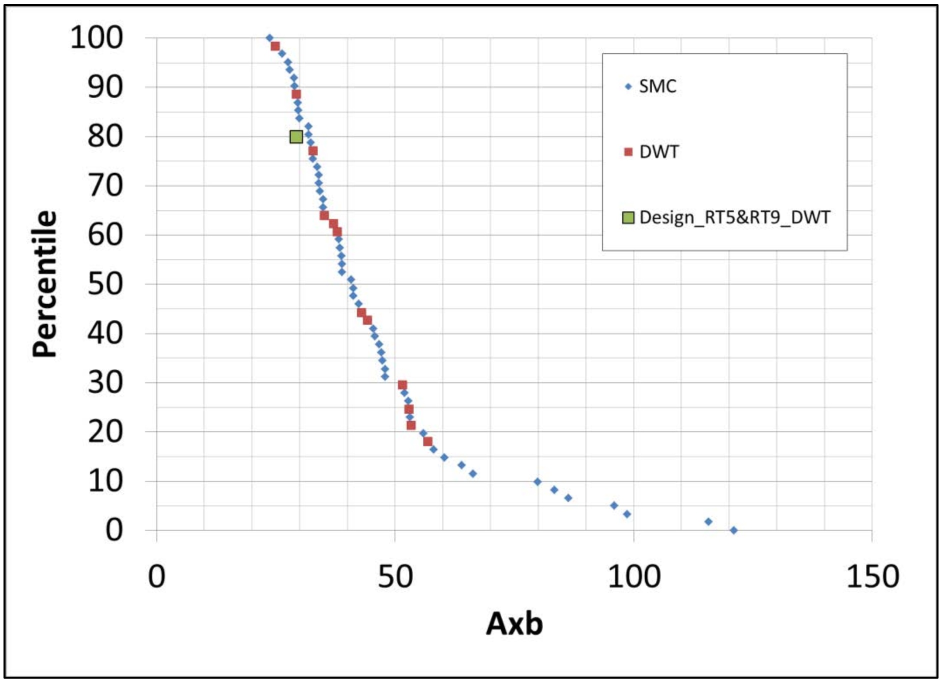


Figure 13-3: Cumulative A × b (DWT + SMC) results for the Livengood Gold Project

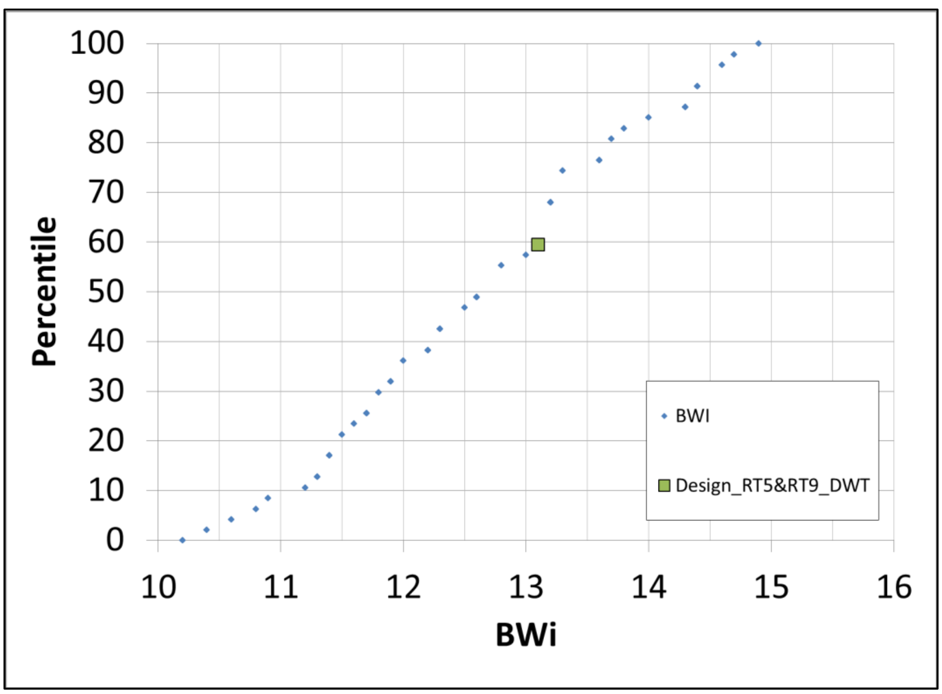


Figure 13-4: Cumulative BWi results for the Livengood Gold Project

13.4.4 Project throughput estimation

Three scenarios were simulated for the study:

a) Scenario A was a circuit based on two lines (SABC, Figure 13-5) with pre-crushing and a final product of 90 µm (P_{80}). SABC stands for a comminution circuit consisting of a semi-autogenous grinding mill (SAG), ball mill and pebble crusher.

b) Scenario B was a circuit based on one line of the same configuration as Scenario A, but with a final product of 180 µm (P_{80}).

c) Scenario C was based on the same circuit configuration as Scenario B, but with optimized blasting, resulting in a finer (F_{80}) feed.

The grind of 90 µm (P_{80}) that was used in Scenario A was based on the FS design criteria. The selection of 180 µm (P_{80}) in Scenarios B and C was the result of integrating the gold leaching results, which indicated at most a 2% difference in leaching recovery between 90 and 180 µm.

The grindability results from historical testwork contained in the BBA database were used to benchmark grinding circuit configurations. Crusher and mill specifications were extracted from recent projects from the BBA database.

Bruno (version 3.62) modeling software was used for the crushing simulations and JKSimMet (version 5.3) was used for the grinding simulations.

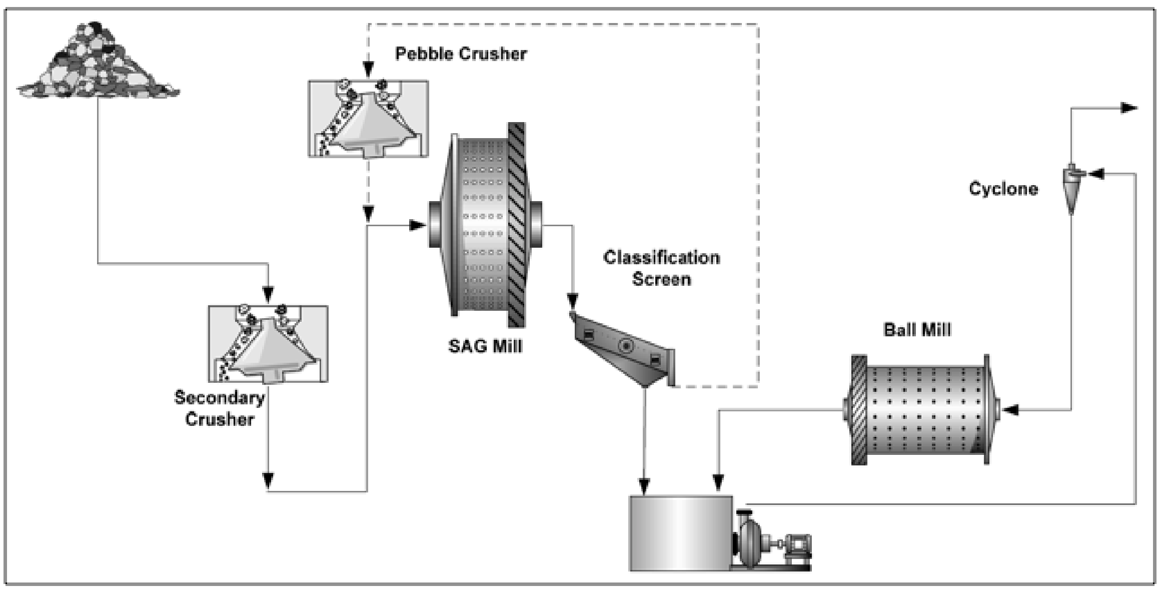


Figure 13-5: SABC with pre-crushing (secondary crusher) circuit configuration

13.4.4.1 Specific energy and throughput estimations

Preliminary power calculations were completed using Moly-Cop Tools (Moly-Cop v3) and JKMRC Estimator (Power Draw Estimation Spreadsheet tools for JKSimMet V5.3). The input parameters used by the two software packages are presented in Table 13-9.

Table 13-9: SAG and ball mill design criteria for simulations

	Units	SAG Mill	Ball Mill
Nominal Dimensions (D×L)	ft × ft	36 × 20	26 × 40.5
Effective Diameter	ft	35.3	25.5
Effective Length	ft	17.5	39.5
Mill Critical Speed	%	74.5	74.8
Charge Filling	%	28	30
Balls Filling	%	15	30
Percent Solids in Mill	%	75	76.4
Ore Density	mt/m^3	2.72	2.72
Losses	%	5	5
Ball Density	mt/m^3	7.75	7.75
Feed Cone Angle	(°)	15	24.3
Discharge Cone Angle	(°)	15	24.3
Trunnion diameter	ft	8.2	6.6

The design tonnage is estimated by an iterative process using Excel's "goal and seek" function, where the installed power is the target of the function and is based on known mill specifications. The mill tonnage is varied until the estimated power consumption matches the installed power. The result is the design tonnage of the grinding circuit.

Table 13-10 presents the results of the simulations, completed using the 80th percentile of the grindability results, which are used for calculating the grinding equipment design throughput. The table also presents the 50th percentile of the grindability results for each rock type, which is used for calculating the average throughput used to design the back end (post-comminution) portion of the plant. The average plant throughput was calculated as a weighted average of the throughput for each rock type multiplied by the percentage of each rock type in the deposit, based on the latest LOM summary by rock type, reference "160614 LVG 355k Prod 45M TPA Max Unlimited Stockpile RT9 67%.xlsx".

Table 13-10: Throughput estimations for each scenario in metric tonnes per day (mt/d)

	50th Percentile	80th Percentile
	Throughput, mt/d	
Scenario A – SABC × 2 + Pre-crusher 90 µm (P_{80})		
RT4	78,163	-
RT5	72,952	-
RT6	69,331	-
RT7	74,189	-
RT9	66,814	-
Weighted average of each rock type	**71,801**	-
All rock types combined	-	**66,284**
Scenario B – SABC × 1 + Pre-crusher 180 µm (P_{80})		
RT4	47,914	-
RT5	46,059	-
RT6	43,498	-
RT7	46,721	-
RT9	41,510	-
Weighted average of each rock type	**44,877**	-
All rock types combined	-	**41,577**
Scenario C – SABC × 1 + Pre-crusher (Optimized Blasting)180 µm (P_{80})		
RT4	51,181	-
RT5	49,128	-
RT6	46,081	-
RT7	49,570	-
RT9	44,160	-
Weighted average of each rock type	**47,745**	-
All rock types combined	-	**44,756**

The estimated throughputs highlighted in bold were the values used for trade-off analysis and for design purposes. The 80th percentile A×b parameter taken from the cumulative plot of all rock types (combined) was used to generate the 80th percentile throughput. This value represents the achievable throughput when the feed to the mill ranks in the 80th percentile (A×b) of all rock types. The 80th throughput value is used to design the comminution circuit.

The 50th percentile throughputs for each scenario are based on a weighted average throughput of the estimated throughputs for each rock type, which were generated through simulation using the 50th percentile A×b values that are associated with each rock type. The weighted average (50th) value is used to design the back end of the plant, which encompasses all elements of the process that follow comminution.

Table 13-11: Specific energy calculations for each scenario at design (80th percentile) A×b

Series	Parameter	Units	Scenario A (90 µm)		Scenario B (180 µm)		Scenario C (180 µm + Opt. D&B)	
			SAG Mill	Ball Mill	SAG Mill	Ball Mill	SAG Mill	Ball Mill
Number of grinding lines / Number of units per line			2/1	2/1	1/1	1/1	1/1	1/1
Mill Characteristics	Nominal dimension (D × L)	ft × ft	36.0 × 20.0	26.0 × 40.5	36.0 × 20.0	26.0 × 40.5	36.0 × 20.0	26.0 × 40.5
	Inside liner dimension (D × L)	m × m	10.77 × 5.33	7.77 × 12.04	10.77 × 5.33	7.77 × 12.04	10.77 × 5.33	7.77 × 12.04
	% of critical speed	%	74.5	74.7	75	74.6	75	74.6
	Cone angle	degree	15.0	24.3	15	24.3	15	24.3
Grinding Steel	Ball charge	% volume	15.0	30.0	14.7	30	14.7	30
Mill Power per Line	Required power	kW	13,846	14,960	13,846	14,949	13,846	14,950
		HP	18,568	20,061	18,568	20,046	18,568	20,048
	Installed power	kW	14,000	15,000	14,000	15,000	14,000	15,000
		HP	18,774	20,115	18,774	20,115	18,774	20,115
Total Circuit Power	Required power	kW	27,797	29,446	13,846	14,949	13,846	14,950
		HP	37,276	39,488	18,568	20,046	18,568	20,048
	Installed power	kW	28,000	30,000	14,000	15,000	14,000	15,000
		HP	37,549	40,231	18,774	20,115	18,774	20,115
Specific Energy	Motor output	kWh/t	8.4	9.5	6.7	7.6	6.2	7.0
		Total kWh/mt	17.9		14.3		13.2	

13.4.5 Comminution circuit simulations and design summary

13.4.5.1 Simulations

The simulations for Scenarios A, B and C were completed by BBA using the same SAG and ball mill design criteria described in Table 13-9. The SAG and ball mill specifications are based on an operation with a slightly higher ore hardness, where BBA has previously conducted design, commissioning, as well as technical support over the course of several years.

As part of BBA best practices, simulations were performed to balance the power draw in the SAG and ball mills to avoid mill throughput bottlenecks. The estimated power consumptions in Table 13-11 include adjustments for motor/drive efficiency (96%) and also ore variability factors, for which a value of 90% was assumed for the SAG mill and 95% for the ball mill.

Scenario A simulations concluded that the selected circuit (2 lines SABC + pre-crusher) would process approximately 79,145 t/d (71,800 mt/d), which is based on each line having a throughput of 39,573 t/d (35,900 mt/d) (P_{80} of 90 µm).

New leaching results became available at the time that the comminution work was being conducted. The new results indicated that, on average, leach recovery improved by approximately 2% with a grind size of (P_{80}) 90 µm versus 180 µm. A new scenario was modeled (Scenario B) to explore the throughput gain by relaxing the grind size. The Scenario B simulations led BBA to conclude that the selected circuit, based on a single line, would have a weighted average throughput of 49,470 t/d (44,877 mt/d) at the coarser target grind size of 180 µm.

The final optimization simulations were run by BBA using the following parameters:

- Circuit target grind of 180 µm (P_{80});
- Finer feed (F_{80}) assumed as a result of optimized blasting.

The simulation resulted in a 27% circulating load through the pebble crusher and the ball mill circuit running at 250% circulating load, generating a 180 µm (P_{80}) product.

13.4.5.2 Design recommendations

- The recommended configuration for the Livengood Gold Project is a single line SABC circuit with pre-crushing, and considers that the crushing and grinding plant will be fed by ore that has been treated with optimized blasting techniques. The conclusion is based on analysis of simulation results as well as CAPEX and OPEX calculations. The circuit that was selected was the configuration with the lowest specific energy consumption.
- The proposed circuit uses a single (D × L) 36 ft × 20 ft SAG mill with 14 MW of installed power and one 26 ft × 40.5 ft ball mill with 15 MW of installed power.
- Based on the information analyzed, the grinding circuit is designed to process 49,334 t/d (44,756 mt/d) when the ore is at the 80th percentile of all grindability results.

- Based on the information analyzed, the grinding circuit is designed to process 52,630 t/d (47,745 mt/d) when a weighted average of the 50th percentile grindability results for each rock type are assumed as the mill feed. This is the throughput used for sizing the back-end circuit.
- Coarsening of the grind from 90 to 180 µm (P_{80}), coupled with optimized blasting, which generates a finer (F_{80}) feed material, explains the increase in per line throughput between Scenarios A and C. For a single line, Scenario C is 33% higher (52,630 t/d (47,745 mt/d) vs 39,572 t/d (35,900 mt/d)), which has a direct impact on daily gold production.
- Similarly, the coarser grind and optimized blasting are also the basis for the reduction in specific energy between Scenarios A and C. Scenario C is 26% lower (13.2 vs 17.9 kWh/t), which translates into a lower per tonne operating cost for electricity.

13.5 Metallurgical testwork

13.5.1 FS – Metallurgical testwork

As part of the FS, metallurgical testwork was completed to evaluate the appropriate gold recovery process. Standard recovery trade-offs, such as whole ore leach vs flotation and CIL vs CIP were explored. The initial work was carried out to establish reagent consumption, leach residence time, and to determine the optimum leach feed particle size (P_{80}). The phases of testwork are outlined as follows:

- Optimization testing to establish preliminary ore design parameters;
- Variability testing to assess leaching response on selected gold grades and rock types.

The nature of the testwork and resulting conclusions are presented in the sections below.

13.5.2 FS – Optimization test program

Feasibility study optimization composites of the major rock types were prepared as indicated in Table 13-12. The assayed direct gold head grades for each of these samples are also summarized.

Table 13-12: Optimization composites used for testwork

Rock Type	Composite	Au (g/mt)
RT4	Optimization Composite 2 (RT4)	1.21
RT5	Optimization Composite 1 (RT5)	0.89
RT6	Optimization Composite 3 (RT6)	0.98
RT9	Optimization Composite 4 (RT9)	1.09
RT7	Mini optimization composite (RT7)	1.43

13.5.2.1 Gravity recovery

Various grinds, from 100 to 225 µm (F_{80}), were tested to optimize the grind for gravity recovery from each ore type (Figure 13-6). Analysis of the results indicated that a primary grind of 180 µm (P_{80}) was suitable for all of the ore types tested. Figure 13-6 also presents the results of GRG testwork conducted for each rock type. The GRG results are greater than 60% for RT4, RT5 and RT6 and greater than 55% for RT9. Typical gold operations recover 50% to 65% of the gold associated as GRG. It is observed in Figure 13-6 that the results of the batch gravity tests are in all cases greater than 50% of the GRG.

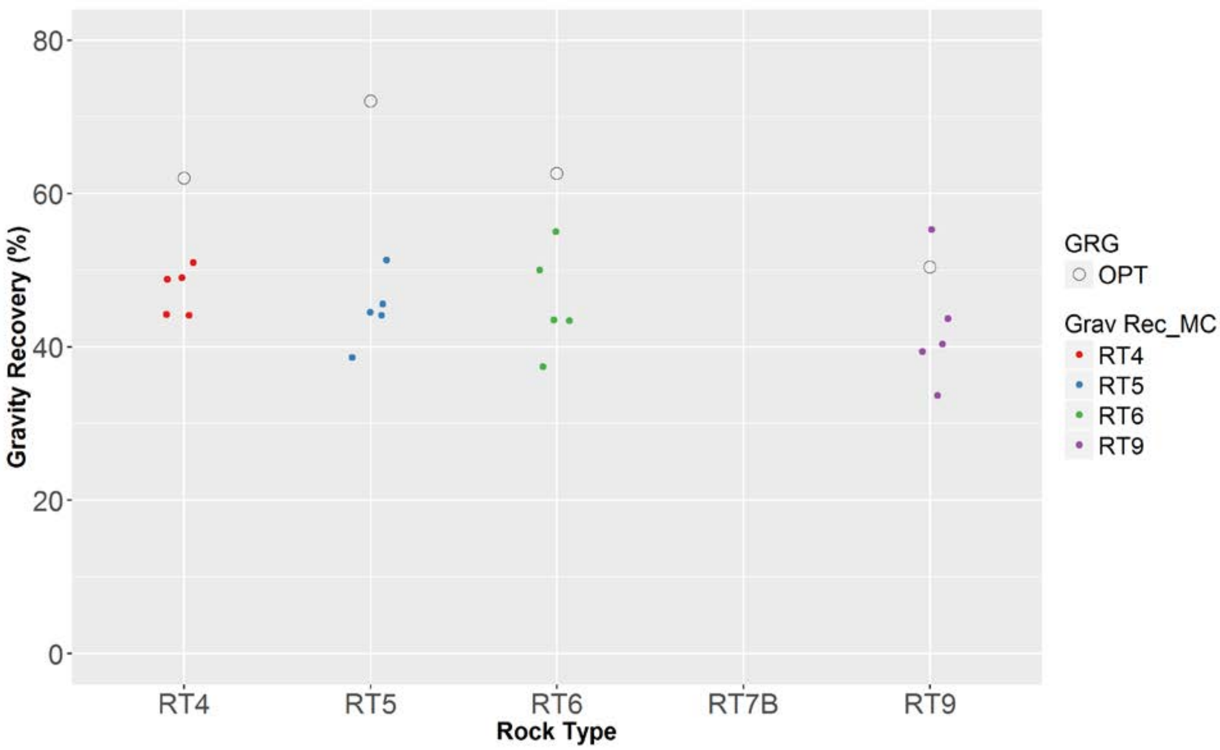


Figure 13-6: Gold gravity concentration grind-recovery relationships for RT4, RT5, RT6 and RT9 (FS)

Note from Figure 13-6, “Grav Rec_MC” is the gravity recovery to Mozley concentrate and GRG refers to the gravity recoverable gold of the optimization testwork.

Table 13-13: Comparison of gravity test results for different rock types (FS)

Test	Rock Type	Optimization Composite	Product	Mass	Grade, g/mt	Rec. %	Gravity Tail K_{80}
				%	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	-	

13.5.2.2 Flotation option

One of the options tested was to generate a flotation concentrate from the gravity tailings and leach only the concentrate. This would be compared to a second option of direct leaching of the gravity tailings.

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 that the effects of grind, cyanide concentration, and residence time be considered.

The RT4 rock type contained significant quantities of talc, which was difficult to separate and would increase the bulk of the potential flotation concentrate. Talc flotation cells were considered as a process option, but the decision to go to direct cyanidation leaching of the gravity tails, on the basis of the complete test results for the other three rock types, rendered this option moot.

Various grinds were tested to optimize the grind for rougher recovery from each ore type. The grind recovery data, represented below in Figure 13-7, indicated that a grind of 90 µm (P_{80}) was suitable for all of the ore types tested. The RT4 rock type did not respond well to flotation. At 12% rougher flotation mass pull, the projected rougher gold recoveries were 78%, 74%, 75% and 50% for RT5, RT9, RT6 and RT4, respectively.

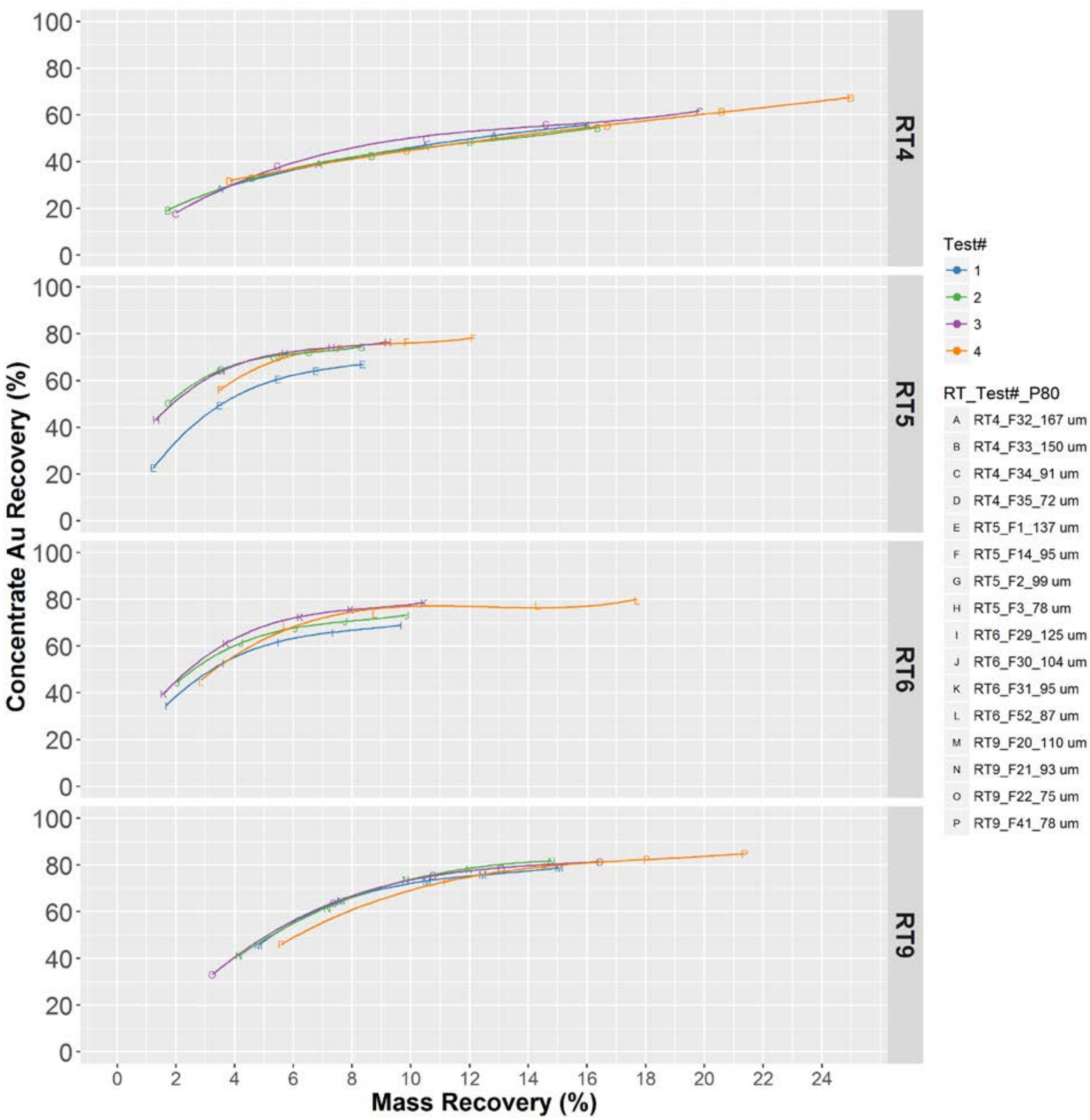


Figure 13-7: Effect of primary grind on gold rougher flotation test Kinetics for RT4, RT5, RT6 and RT9 (FS)

Flotation concentrates from RT5, RT6 and RT9 were subsequently leached (CIL) to determine recoveries. Figure 13-8 shows the gold recovery relative to time for these three rock types. Based on an analysis of the results, it became evident that the recovery of gold would be higher by applying CIL on the entirety of the gravity tails. Therefore, it was decided not to conduct any further flotation testing and CIL tests on flotation concentrate for RT4 were dropped.

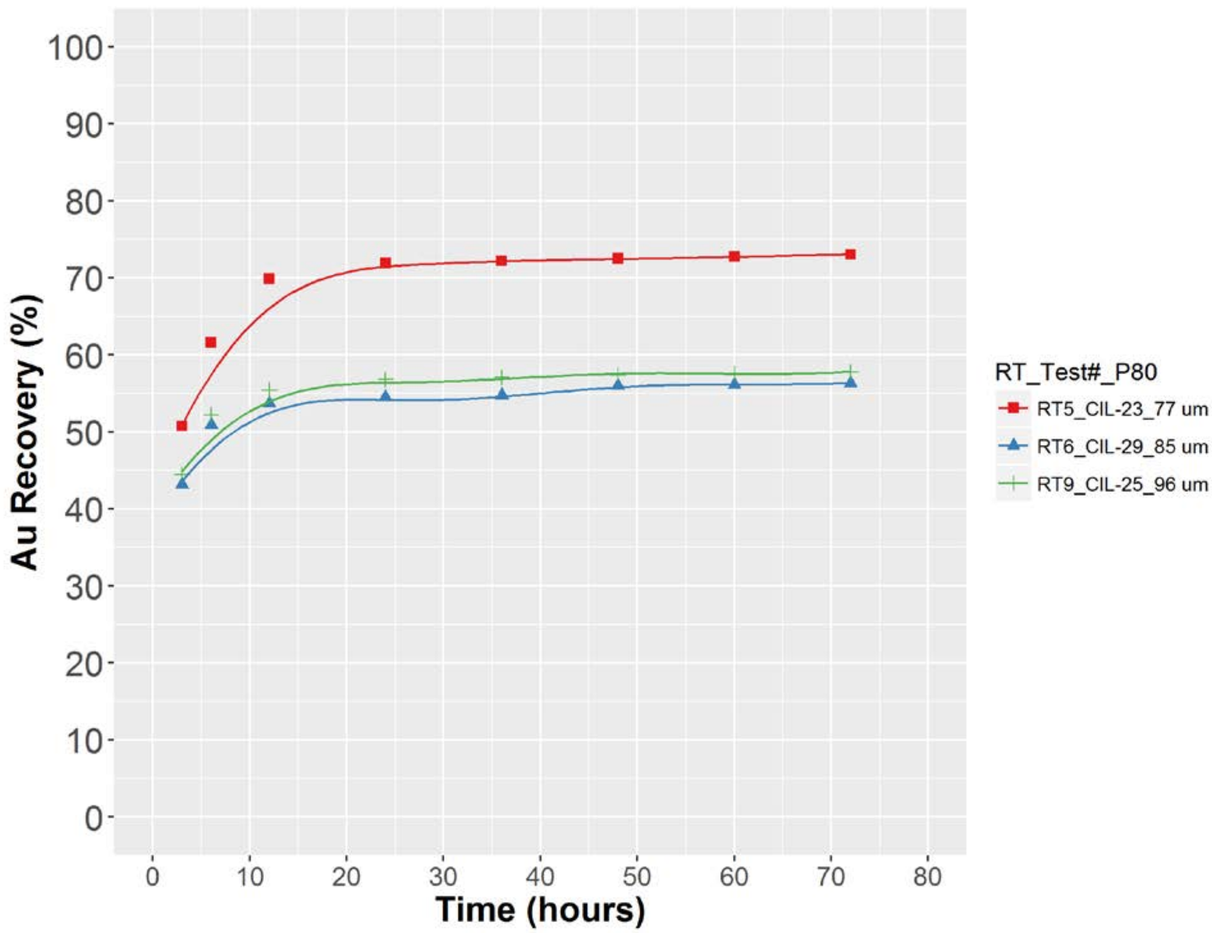


Figure 13-8: Flotation concentrates CIL test gold leach kinetics for different rock types (FS)

13.5.2.3 Flotation option recovery summary

The results derived for each rock type in this test series are summarized in Table 13-14.

Table 13-14: Gold recovery resulting from the combination of gravity, flotation and CIL (FS)

	Au Recovery (%)			
Rock Type	**Gravity**	**Flotation**	**CIL**	**Total**
RT4	49.0%	50%	-	-
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	**47.7%**	**69.8%**	**62.4%**	**70.5%**

13.5.2.4 Whole Ore Leach (WOL) option

The WOL 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. The grind recovery data, represented below in Figure 13-9, indicated that a grind of 90-100 µm (P_{80}) was suitable for CIL leaching of all of the rock types.

The observations in regards to Figure 13-9 are as follows:

- The incremental gold recovery at 72 hours (vs 24 hours) for RT5 and RT6 is less than 2.5%. For RT4 and RT9, it is less than 1%;
- There were no samples collected between 5 and 24 hours;
- The gold recovery variation (for each rock type) at the particle size range from 60 to 180 µm (P_{80}) was inconclusive given the single test at each grind size.

These observations were taken into consideration in the course of developing the optimized leaching conditions during the PFS.

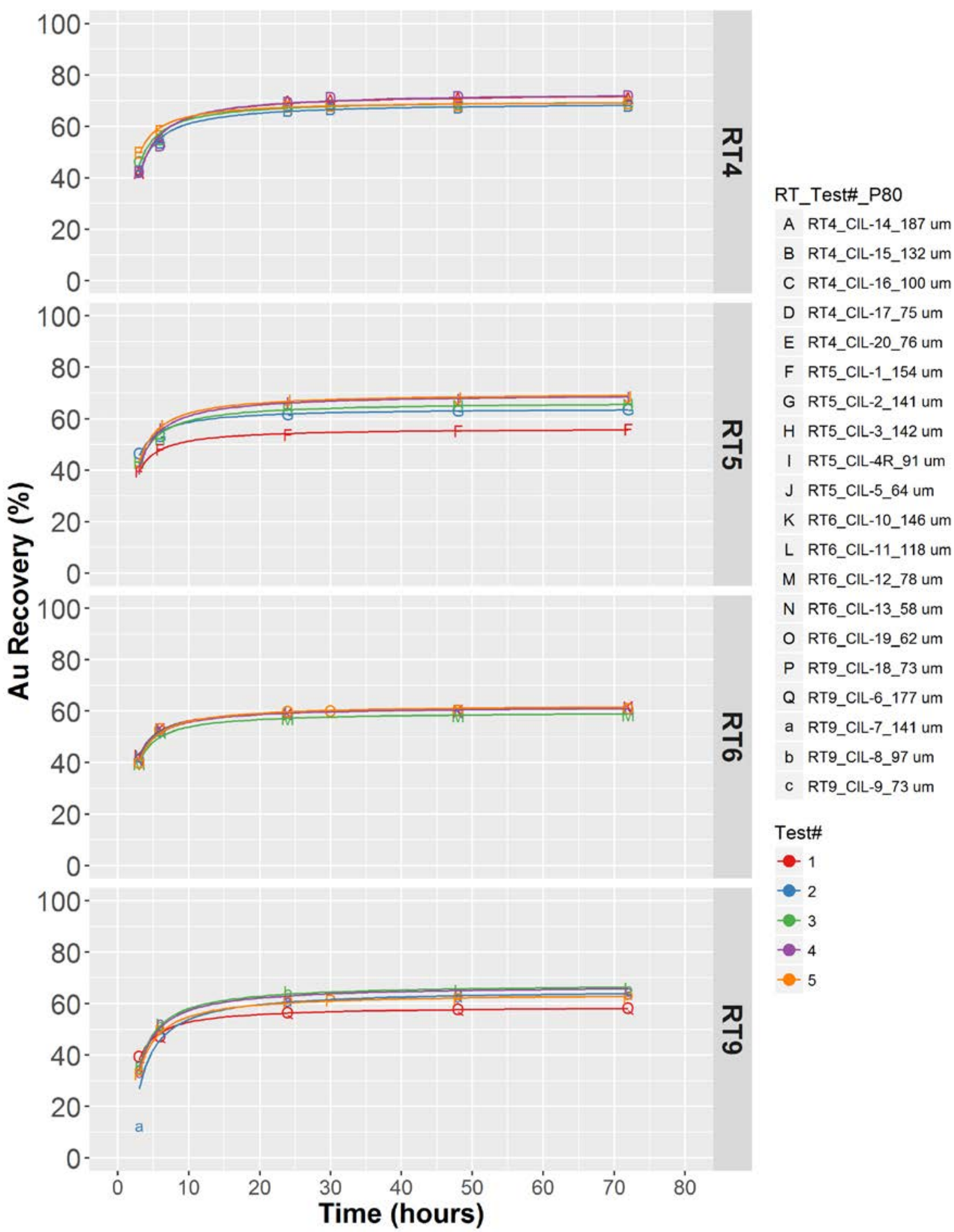


Figure 13-9: Effect of grind on gold extraction kinetics for RT4, RT5, RT6 and RT9 (FS)

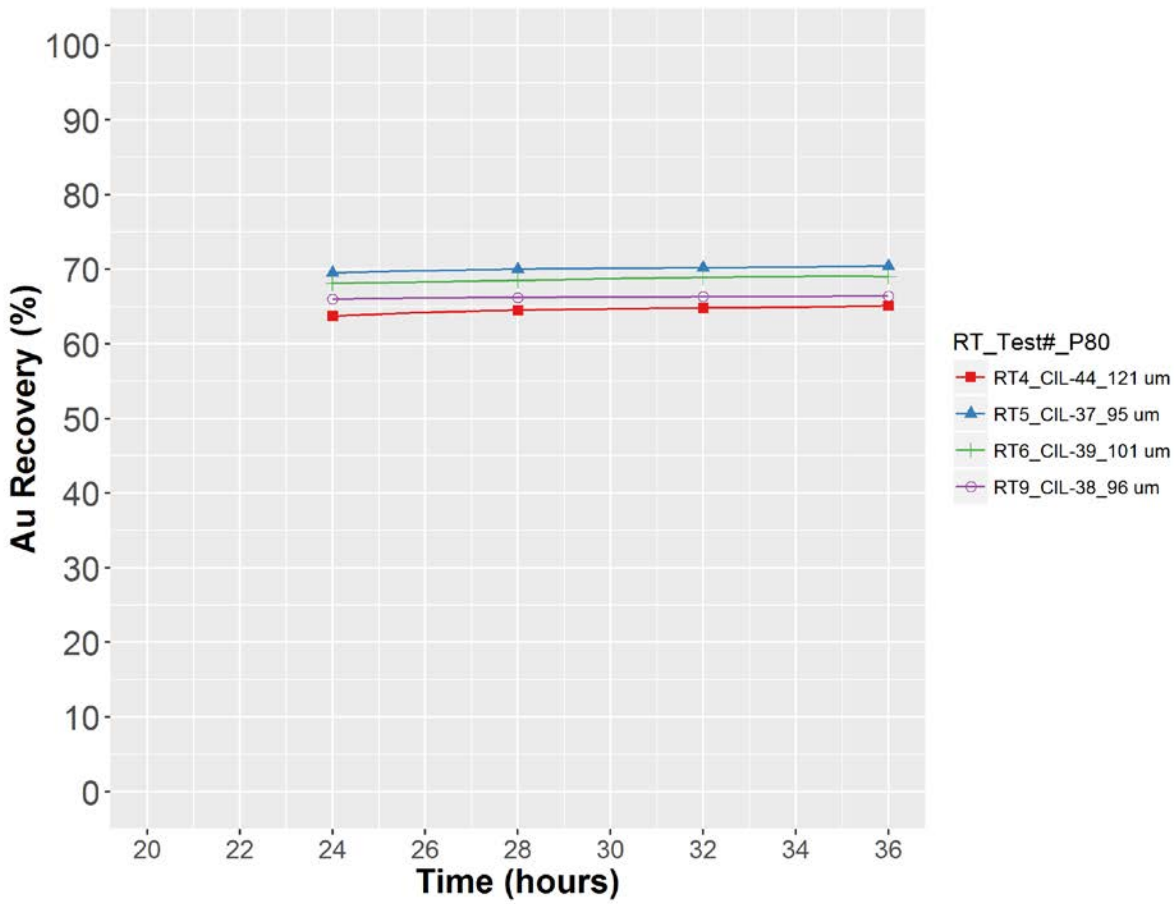


Figure 13-10: Mozley gravity tailings CIL test kinetics for different rock types (FS)

The above graph illustrates the very flat leach recovery curves for the gravity tail leach, indicating little (if any) increased extraction beyond 24 hours of leach time. Similar to the observation made for Figure 13-9, the leaching rate after 24 hours was very slow and it was decided to explore shorter leaching retention times in future testwork.

13.5.2.5 WOL option recovery summary

The analysis that was completed with the optimization samples led to the conclusion that the preferred flowsheet was gravity followed by CIL of the gravity tails. The gravity plus CIL leaching of the gravity tails produced a 9-12% improved gold recovery for all rock types compared to gravity plus flotation with CIL of flotation concentrate. The overall results of whole ore leaching can be seen below in Table 13-15.

Table 13-15: Gold recovery resulting from whole ore leaching (FS)

	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) (1)	24.3%	44.8%	58.2%

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

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

13.5.2.6 WOL vs Flotation

The overall gold recoveries achieved by both process options are summarized in Table 13-16 below.

Table 13-16: Overall gold recovery of optimization samples for both process options (FS)

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%	-

The important conclusions that are drawn from the FS optimization testing include:

- All rock types responded well to gravity separation, with 44% to 55% of the gold recoverable in the gravity circuit. At a grind of approximately (P_{80}) 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 noted significant 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, and contains more total carbon, total organic carbon (TOC), and carbonate;
- Overall gold extraction was increased 9% to 12% by leaching the gravity tails as compared to leaching the flotation concentrate.

A detailed analysis of the testwork results post FS by BBA indicated that there were additional opportunities to explore, such as reducing the leach retention time and targeting a coarser grind.

13.5.3 FS – Variability test program

Following on the optimization testing, the FS test program moved into a variability testing phase. The goal was to determine the variation that existed in the ore and to test the geological extremes of each rock type. In addition to the samples tested in the optimization phase, rock type RT7 and a rock type known as Stibnite was included in the variability testing program. The RT7 rock type, which contains varying levels of antimony (Sb) in the form of stibnite and jamesonite, was not evaluated in the initial optimization testing as it did not have a large presence in the early period of the mine life and only represented 12.1% by weight of the LOM reserve. The RT7 rock type was originally split into two sub-types, RT7-Bleached and RT7-Unbleached, as these sub-types exhibited different metallurgical responses. The Stibnite rock type represented a very small fraction of the mine’s ore, but had head grades in the multiple grams per tonne.

The most favorable process conditions that were established in the optimization phase were used for variability testing. The variability test results showed an overall lower average gold recovery than what was achieved in the optimization phase, which is reflective of 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 in Table 13-17.

Table 13-17: Variability sample gold recovery (FS)

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

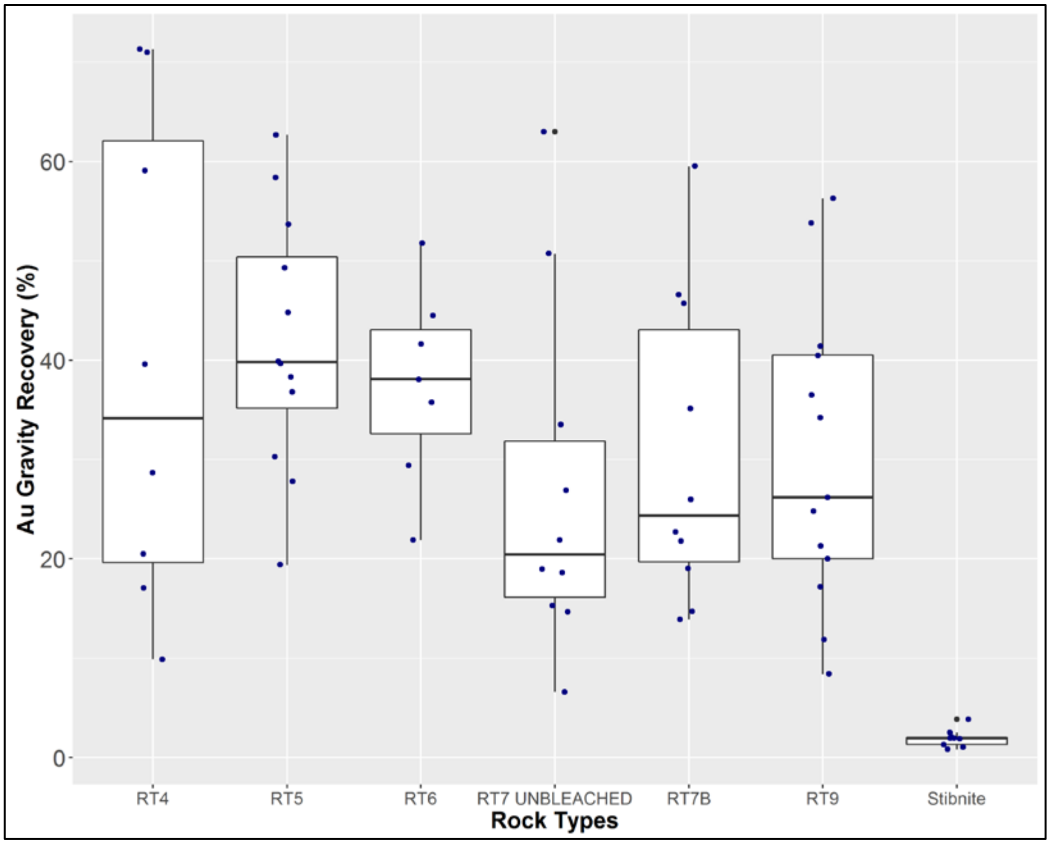


Figure 13-11: Gold gravity recovery box plots (FS)

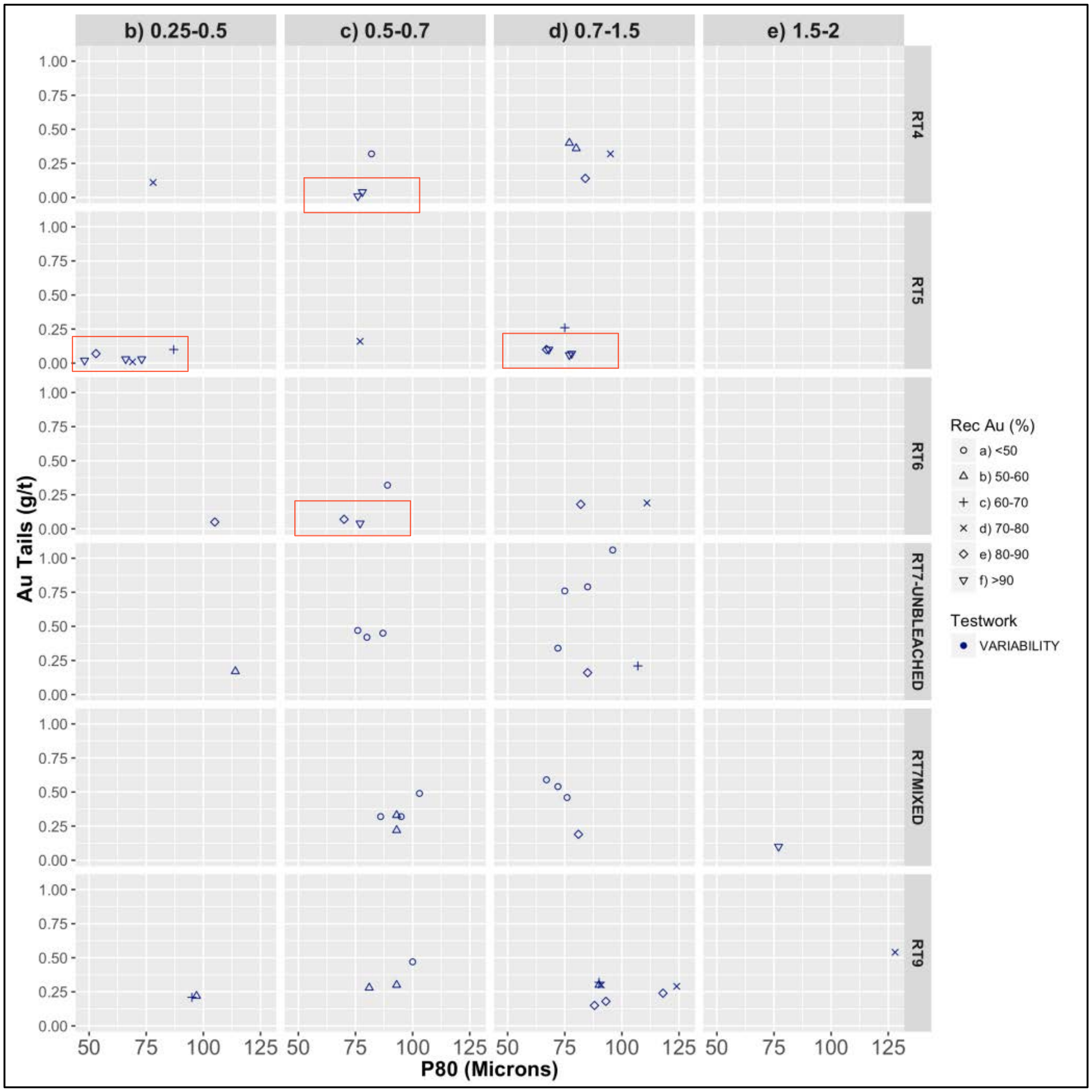


Figure 13-12: Gold in Residues from CIL testwork vs P_{80} for each rock type (FS)

The columns in Figure 13-12 represent the gold grade ranges in g/mt and the rows correspond to the different rock types.

Analysis of the results suggested that the RT4, RT5 and RT6 rock types did not show a correlation with grade, RT9 showed a weak correlation with grade, while RT7 presented a strong correlation to stibnite content and grade.

A post FS analysis led to the conclusion that the RT5 results suggested opportunities to increase gold recovery with finer grind. The red boxes associated with RT5 in Figure 13-12 highlight the lower gold in residues at the finer P_{80}(s). Similar observations have been made for RT4 and RT6. This was not observed for RT7 and RT9. BBA analyzed the information on testwork methodology that was available and a lack of consistency was observed in the residence time as well as the level of monitoring of dissolved oxygen (DO) levels during testwork preconditioning and leaching. This result was critical, because low dissolved oxygen levels during the initial hours of a leaching test will have an important and detrimental effect on the gold leaching performance. This observation was used during Phase 9 PFS testwork to standardize the O_2 preconditioning (4 hours) as well as monitoring and maintaining the DO levels (8 ppm).

Important follow-up observations from the analysis of the variability testwork include:

- The opportunity to lower the gold in residues by using finer grind (P_{80}) (RT5, Figure 13-11);
- The need to more closely monitor and control preconditioning and DO levels as these will have an impact on cyanide consumption.

13.5.4 FS – Solid / liquid separation testwork

As part of the FS, Livengood gold ore samples were submitted to Pocock Industrial, Inc. for solid liquid separation (SLS) testing. 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 current flowsheet has a pre-leach thickener and a tailings (pre-detox) thickener.

The Livengood Gold Project design criteria use a high rate thickening rise rate of 1.64 gpm/ft² (4.0 m^3/m^2h) for both the pre-leach thickener and the tailings thickener at a design P_{80} of 90 µm. Additional savings on reagents (flocculant) are expected for the present scenario, where the P_{80} is 180 µm, but further settling testwork will be required for confirmation.

13.5.5 FS – Cyanide detoxification tests

13.5.5.1 FS – Cyanide detoxification testwork

The CIL tailings generated from the leaching testwork was used for cyanide detoxification testing. The INCO SO_2/air process was used to remove cyanide and base metal complexes from the CIL tailings generated from each rock type. The objective of this phase of testing was to optimize cyanide detoxification (CND) of the CIL tailings. The “Interim Test Program” used a 10 kg sample of each rock type (RT4, RT5, RT6, RT7 and RT9).

The feed pulp density to cyanide detoxification was between 31-39%. The results showed that it was possible to treat the CIL tailings using the INCO process to bring 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, coupled with a retention time of 94-147 minutes, is ideal. The reagent consumptions from the Phase 1 testing are 8.2-14.7 g/g CN_{WAD} of equivalent SO_2, 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 were assumed to be:

- Lime = 0.82 lb/t
- Copper sulfate = 0.08 lb/t
- Sodium metabisulfite = 1.65 lb/t

13.5.5.2 Observations made in regards to cyanide detoxification

In the lead-up to the PFS, BBA reviewed the design of the cyanide detoxification system that was presented in the FS. The objective was to look for gaps and opportunities. The following are the major conclusions:

- The use of a sulfur burner to generate SO_2 instead of sodium metabisulfite was identified as an opportunity to lower the OPEX. Details are presented in the flowsheet development in Section 13.7;
- A model was developed to estimate the amount of cyanide that is recirculated to the leaching process via the pre-detox thickener. The result is less cyanide reporting to cyanide detoxification.

13.5.6 PFS – Metallurgical testwork

Five additional phases of testwork (Continuous, 7, 8, 9 and 10) have been completed since the FS was completed in 2013. Testwork was conducted to explore possible opportunities established through BBA's analysis of the FS testwork and/or to clarify certain questions regarding gold leach performance and reagent consumptions. The phases of testwork are outlined as follows:

- Continuous: Processing FS Optimization composites using recycled process solutions;
- Phase 7: Assay procedures and water source testing;
- Phase 8: Exploratory testing on selected rock types;
- Phase 9: SGS / FLS Curtin University testwork: Exploratory testing on selected rock types using reverse circulation (RC) drill chip composites;
- Phase 10: Stirred tank reactor (STR) controlled leach testwork, with focus on two problematic rock types (RT7 and RT9).

The nature of the testwork and resulting conclusions are discussed below.

13.5.7 PFS – Continuous testwork

Continuous testwork was conducted using 60 kg composites taken from the optimization master composites prepared for the FS. The objective of the continuous testwork was to evaluate the impact of recirculating streams as well as generating leach residues (CIL tailings) for the cyanide detoxification testwork. The continuous testwork conditions were developed from the optimization and variability testwork.

One of the important conclusions to be drawn from the continuous testwork is that the results indicated that using lower cyanide additions had a minimal impact on gold leaching performance, except on RT9, where cyanide starvation conditions were observed. The continuous results were used to estimate the addition of lead nitrate and cyanide for the Phase 9 test program (see Section 13.5.11).

13.5.8 PFS – Phase 7 - Assay procedures and water source testing

The Phase 7 testwork was conducted on 20 kg composites of RT4, RT5, and RT9. The objective of the Phase 7 testwork was to remove uncertainty related to the water source used for testing: SGS Vancouver water versus water that was sourced from the mine and to confirm the procedures necessary for improving assay repeatability. To improve assay repeatability, all samples had gravity recoverable gold removed prior to leaching by a combination of a centrifugal concentrator followed by gravity table, leaching was performed in triplicate, and all pulps were fire screen assayed in triplicate.

CIL testing with air sparging using both Vancouver and mine-sourced water was performed on Mozley gravity tails to compare the extractable gold using similar reagent conditions.

Gravity tail leach recoveries for duplicate samples were within 2% of each other for all three rock types using Vancouver and mine-sourced water and likely within the precision of the testwork. Cyanide consumption increased 0.3% with mine water and lime consumption decreased 8%.

Important conclusions from the Phase 7 testwork include:

- The results indicated that the gold recovery was not particularly sensitive to water source;
- Phase 7 results confirmed that performing triplicate screen metallic assays on gravity tail leach residue was the protocol required for precise work.

13.5.9 PFS – Phase 8 - Grind, recovery, gravity, flotation testing

The Phase 8 testwork program was comprised of several sub-phases and all work was conducted on 75 kg core samples. The objectives of the program were to explore CIL gold recovery sensitivity to particle size.

The sub-phases are described as follows:

- Gravity testing on 180 and 250 µm (P_{80}) samples;
- CIL sensitivity testwork on 90, 180 and 250 µm (P_{80}) samples;
- At 90 µm (P_{80}), CIL of gravity tails was compared to CIL of only a flotation concentrate generated from the gravity tails.

Phase 8a

- CIL testing was extended down to 60 and 75 µm (P_{80}):
  - Knelson and Mozley tails 250 µm (P_{80}) samples from the Phase 8 testwork program were combined and reground to 60 and 75 µm (P_{80}).
- Carbon handling protocols were compared:
  - The carbon handling protocol was explored. No significant difference was found between adding new carbon and retaining the original carbon for the duration of the testwork.
- Dissolved oxygen (DO) and cyanide (CN) consumption were evaluated:
  - Phase 8a was the first attempt to normalize the DO levels and CN additions between different tests. There had been indications of inconsistent preconditioning in previous testwork.

Phase 8b

- Evaluated gravity and leach sensitivity at 90, 180 and 250 µm (P_{80});
- At 90 µm (P_{80}), CIL of gravity tails was compared to CIL leaching of only a flotation concentrate generated from the gravity tails.

Phase 8c

- Completed intensive leach (IL) of flotation concentrate.

Phase 8d

- Tested flotation with sulfidization at 180 µm (P_{80}).

Phase 8d was designed to study the response of flotation to sulfidization at a grind of 180 µm (P_{80}). Only two rock types were tested.

The important conclusions to be drawn from the Phase 8 test program include the following:

- Flotation gold recoveries did not improve with slurry sulfidization;
- Gold recovery did not improve with a grind of 60 µm (P_{80});
- CN consumption was reduced by the pre-oxidation;
- The carbon handling protocol did not affect the gold recovery performance.

13.5.10 PFS – Phase 9 - SGS and FLS-Curtin University test program

The Phase 9 SGS / FLS-Curtin University test program was conducted on 500 kg composites made according to rock type. It was the first work conducted using reverse circulation (RC) drill chips.

The objectives of the Phase 9 test program were:

- To compare the performance of gravity recovery at 180 and 250 µm (P_{80});
- To study the impact of lead nitrate addition on intensive leach and CIL;
- To confirm and/or revise the cyanide addition to CIL;
- To study the impact of particle size on gold leaching at 75, 90, 135, 180 and 250 µm (P_{80}).

The Phase 9 program processed a large quantity of mass for each sample to confirm the process flowsheet developed in Phase 8, and to avoid having nugget effects influence the metallurgical recoveries.

The objectives of the FLS/Curtin University testwork were:

- To conduct gravity recoverable gold (GRG) testing and to perform an Integrated Liberation and Leaching Model (ILLM) characterization on the Livengood gold ore types.

Rock type splits of 100 kg each were sent to FLSmidth/AMIRA (Curtin University, Australia).

13.5.10.1 Phase 9 Metallurgical composite sample selection methodology

The Livengood Gold Project resource has been defined by approximately 800 drillholes, about 80% of which are reverse circulation (RC) and 20% are core. Prior to the Phase 9 test program for the PFS, all of the metallurgical testwork had been completed using individual core samples or core sample composites. The Phase 9 samples were the first to be composited from RC rig duplicate rock chips. The Phase 9 samples are bulk composites prepared for each of the five major rock types to represent the average grade and approximate grade distribution of the 2013 FS reserve estimate.

The RC rig duplicates (rock chips), which originated from an earlier phase reported under SGS project CAVM-50223-006, were received at SGS in June 2015. The gold head assays, determined by screened metallics, are presented in Table 13-18.

Table 13-18: Gold head assay

Rock type composite	Au, g/mt
RT4	0.64
RT5	0.72
RT6	0.81
RT7B	0.89
RT9	0.68

Composites were prepared from samples received in super sacks that had been sorted according to their rock type. The composites were labeled as follows with the following weights (Table 13-19):

Table 13-19: PFS composite naming and weights for Phase 9 (PFS) test program.

Composite Name	Mass (kg)
RT4 June 2015 Composite	453.6
RT5 June 2015 Composite	468.0
RT6 June 2015 Composite	472.8
RT7B June 2015 Composite	475.0
RT9 June 2015 Composite	477.5

Each composite was stage crushed to 100% minus 10 mesh, blended, and split to obtain two 100 kg splits and one bulk split as identified below:

- 100 kg forwarded to FLSmidth for GRG tests and leach tests performed at Curtin University;
- 100 kg stored in a freezer at SGS;
- Bulk split retained at SGS for use in the current testwork program.

Figure 13-13 illustrates the testwork that was conducted.

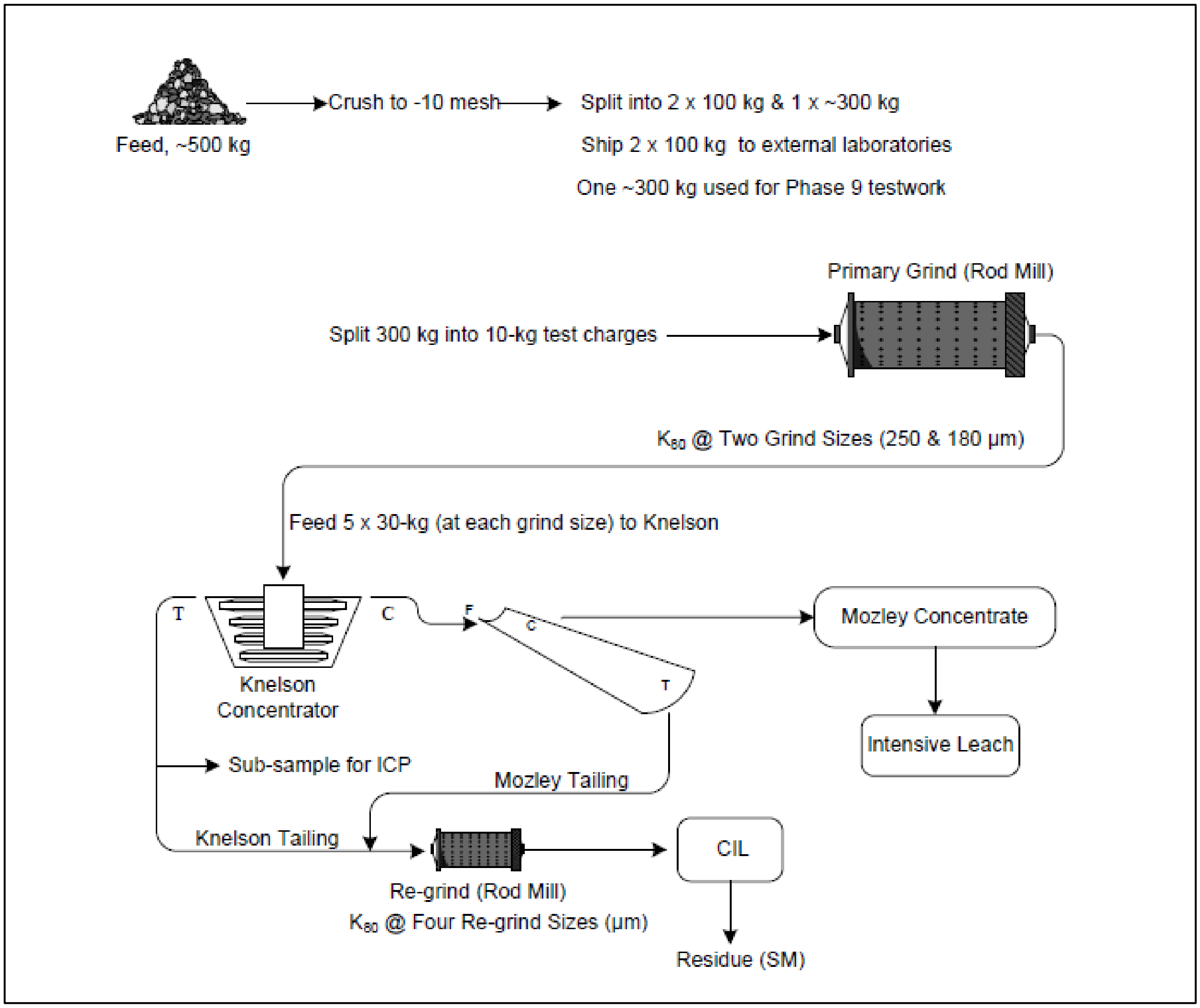


Figure 13-13: PFS (Phase 9) testwork outline

In an effort to minimize the differences found between calculated and direct head grades, possibly due to nugget effect, Phase 9 testwork was conducted following a different approach using larger samples (30 kg) versus the normal 2 kg samples used in the previous testwork.

13.5.10.2 Phase 9 – Cyanide and lead nitrate addition

The ICP analysis of the leach feed composites had not been available at the time that the leaching testwork was being completed. Instead of waiting for this information, BBA recommended that the lead nitrate addition be based on the antimony (Sb) content from the ICP analysis of each rock type from the continuous testwork program (Table 13-20).

Table 13-20: ICP Analysis of the CIL feed from the continuous testwork for each rock type (PFS)

Rock Type	Au	Ag	Cu	Fe	Ni	S	As	Pb	Sb	Te	Hg	Proposed Lead Nitrate addition, g/mt
RT4	0.5		45.1	5.29	1008	0.60	4748	21.6	64.0	0.18	4.3	150
RT5	0.4	0.4	51.5	4.2	121	0.97	2800	14.4	19.9	0.1	0.4	100
RT6	0.5	0.4	75.3	3.87	146	1.25	3360	29.4	62.3	0.17	1.4	150
RT7	0.6	0.5	62.8	4.12	206	1.98	3973	15.4	176.5	0.20	0.7	250
RT9	0.7	0.5	36.6	4.60	69	2.54	3995	24.1	101.8	0.13	3.1	250

Table 13-21: Comparison of cyanide addition in Phase 9 versus Continuous (PFS)

Rock Type	CN addition used during Continuous testwork kg/t	Proposed CN addition for Phase 9, kg/mt
RT4	0.52	0.71
RT5	0.71	0.71
RT6	0.51	0.8
RT7	0.67	1
RT9	0.57	1

13.5.10.3 Phase 9 – Gravity / intensive leach testwork

Phase 9 gravity testwork was performed on two particle sizes, 180 and 250 µm (P_{80}), at SGS Vancouver. It was observed that similar average results were realized for RT7 and RT9. On the other hand RT4 and RT5 show higher gravity results at 180 µm (P_{80}), and RT6 shows higher gravity recovery at the coarser grind size. In general, however, the gravity results were lower than those obtained in previous work.

Table 13-14 summarizes the gravity testwork results of Phase 9 (180 and 250 µm); the GRG results by FLS/Curtin University, along with the results obtained from the FS optimization test program.

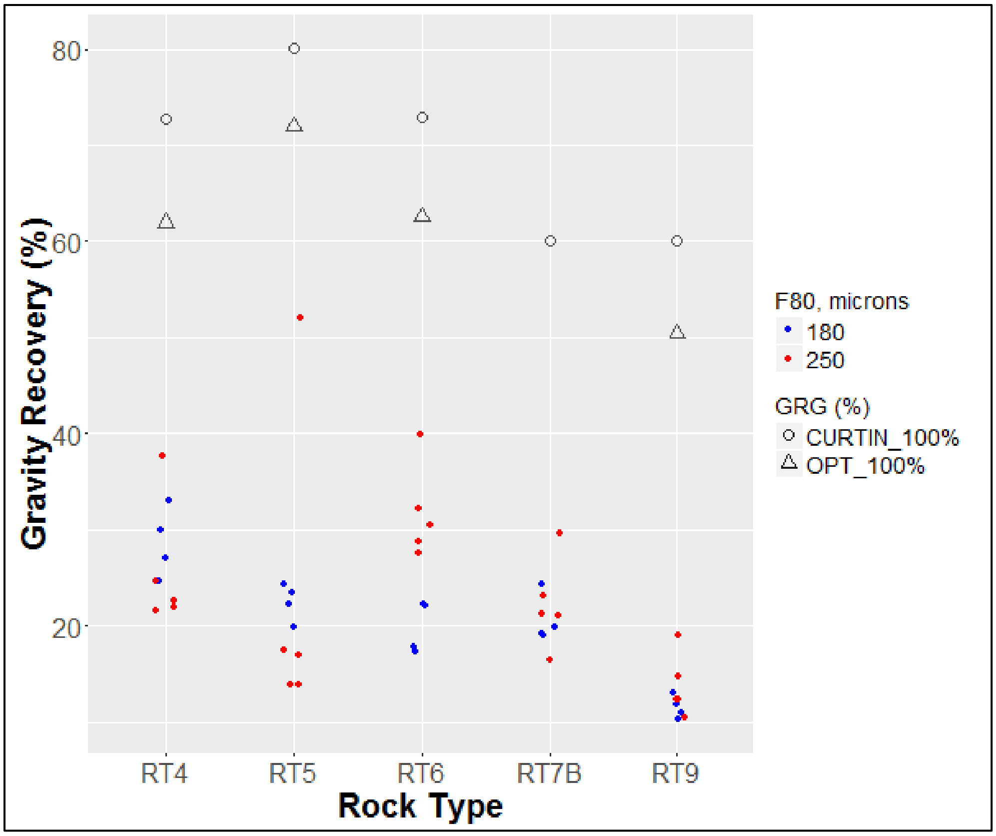


Figure 13-14: PFS (Phase 9) gravity recovery for all tock types
(GRG presented as the 3rd stage of GRG)

The GRG results from the FLS/Curtin University testwork are comparable to the results of the optimization GRG testwork, presented earlier in Figure 13-6, which is an important conclusion for a few reasons:

- The rock types tested represent 98.2% of the ore body;
- Two independent labs have produced these complementary results;
- The FS optimization testwork made use of drill core composites, whereas the FLS/Curtin testwork used RC drill chip composites, implying that two independent sample batches have been tested.

The gravity gold recovery in Phase 9, which followed the Knelson+Mozley table methodology, was lower compared to previous testwork; compare these results, for example, to those of Figure 13-11. BBA is placing less emphasis on these particular results. In future testwork, this method of testing will be applied. However, BBA recommends that a systematic review of the testing protocols be performed.

A series of benchmark graphs (Figure 13-15 to Figure 13-19) were prepared by FLS/Curtin University that compare the results of Livengood gold ores to the Curtin gravity testwork database. Figure 13-15 shows the high percentage of GRG of Livengood gold ore (60% to 80%) compared to the FLS/Curtin University database (15% to 55% for the same gold head grade).

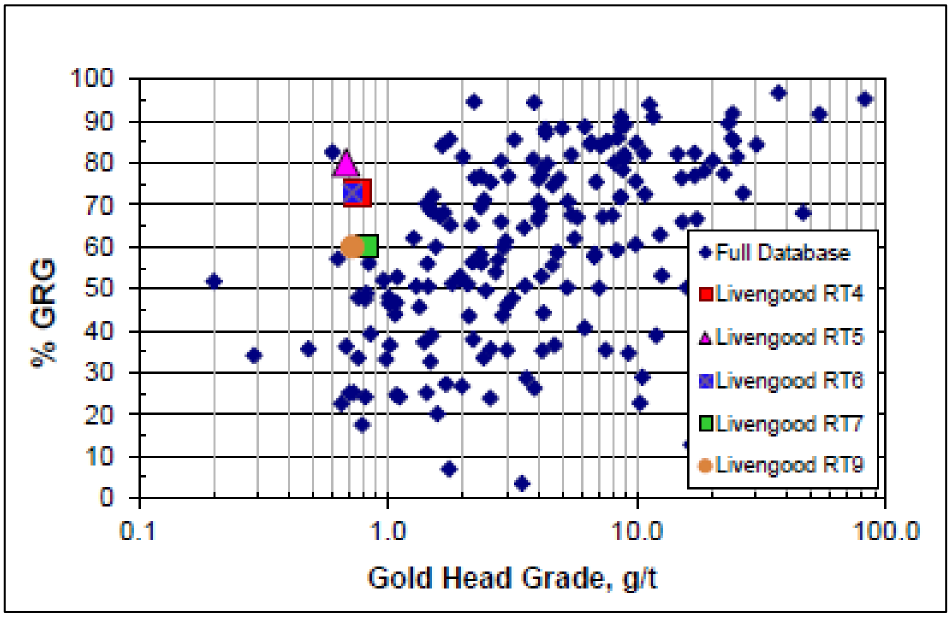


Figure 13-15: Livengood GRG results vs FLS/Curtin database (source Curtin report) (PFS)

Figure 13-16 plots the %GRG vs the gold recovered in stage 1 (of 3 stages) (P_{80} = 850 µm) of gravity testwork, showing that the results fall within the range of data contained in the full FLS/Curtin University database. This observation supports the good gravity recoverable potential of the Livengood gold ore.

A good agreement was also found between the % GRG and the gold recovery of stage 1 vs feed size (F_{80}) of the gold particles and the FLS/Curtin University database (Figure 13-17 and Figure 13-18).

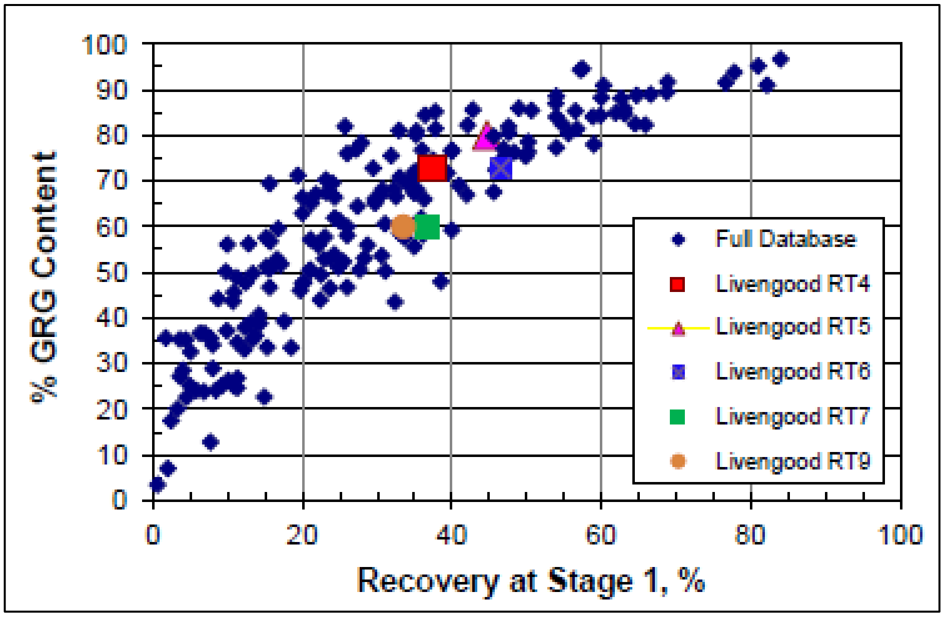


Figure 13-16: Livengood GRG results vs FLS/Curtin database (source Curtin report) (PFS)

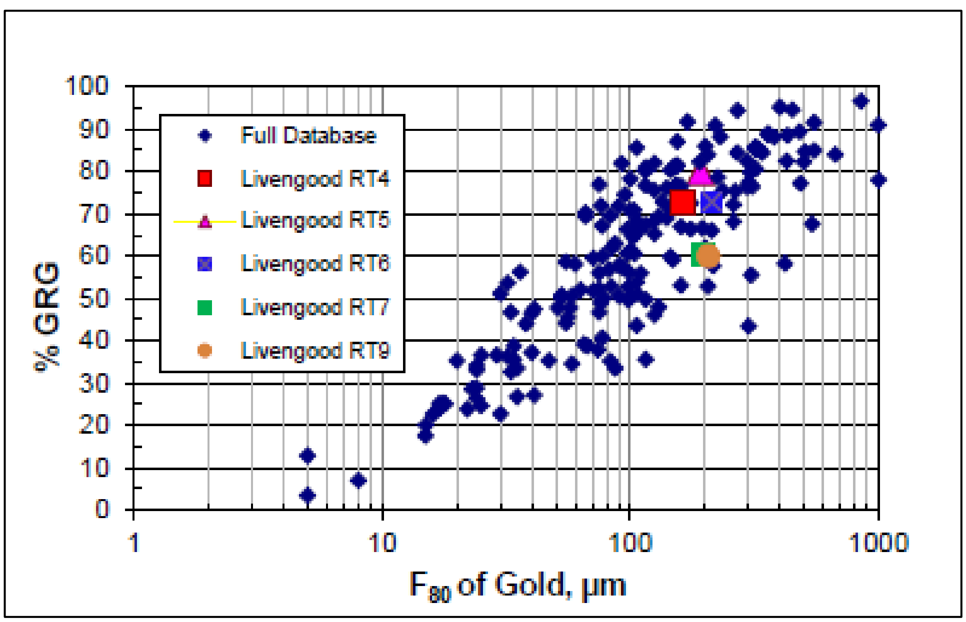


Figure 13-17: Livengood GRG results vs FLS/Curtin database (source Curtin report) (PFS)

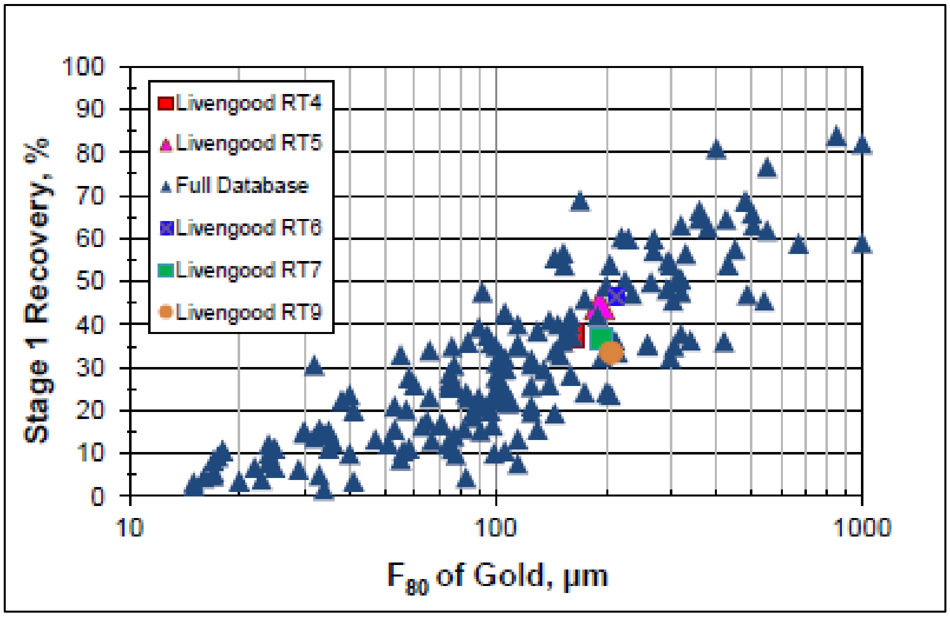


Figure 13-18: Livengood GRG results vs FLS/Curtin database (source Curtin report) (PFS)

Figure 13-19 shows the benchmarking of the results of Livengood versus two operations with similar %GRG. The operations processing ores "sample B" and "sample D" typically recover gravity gold in the order of 45% to 55%. FLS/Curtin University indicates that operations typically should achieve recoveries in the order of 50% to 66% of the plant feed GRG.

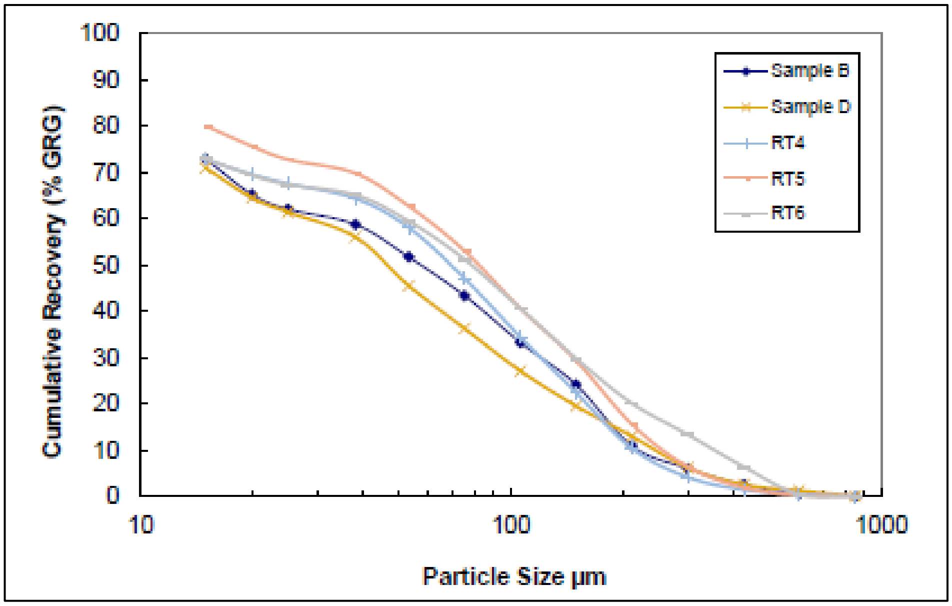


Figure 13-19: Livengood GRG results vs FLS/Curtin database (source Curtin report) (PFS)

13.5.10.4 Phase 9 – Intensive leach kinetic testwork

Intensive leach testwork was conducted on Mozley concentrates. Table 13-22 presents the results of the testwork.

Table 13-22: Intensive leach results (PFS)

Test ID	Gravity K_{80}	Intensive Leach K_{80}	NaCN		CaO		Au in Residue	Calc Head	Au Recovery
	µm	µm	Add kg/mt	Cons kg/mt	kg/mt	kg/mt	g/mt	g/mt	%
IL-1	180	180	72.5	31.5	2.29	0.81	8.2	476.6	98.3
IL-2	250	250	61.7	25.2	1.84	<1	10.1	368.7	97.3
IL-3	180	180	74	30.5	1.84	0.65	9.64	368.6	97.4
IL-4	250	250	66.8	35	1.51	0.59	10.9	295	96.3
IL-5	180	180	73.6	30.7	0.74	<1	9.16	392.7	97.7
IL-6	250	250	80.2	29.9	0.88	<1	13.2	535.6	97.5
IL-7	180	180	72.5	30.3	1.36	0.28	8.68	389	97.8
IL-8	250	250	78.2	27.8	1.66	0.24	9.01	331.1	97.3
IL-9	180	180	74.7	31.6	1.18	<1	7.53	189.9	96
IL-10	250	250	74.4	35.4	0.8	0.8	7.32	214.6	96.6

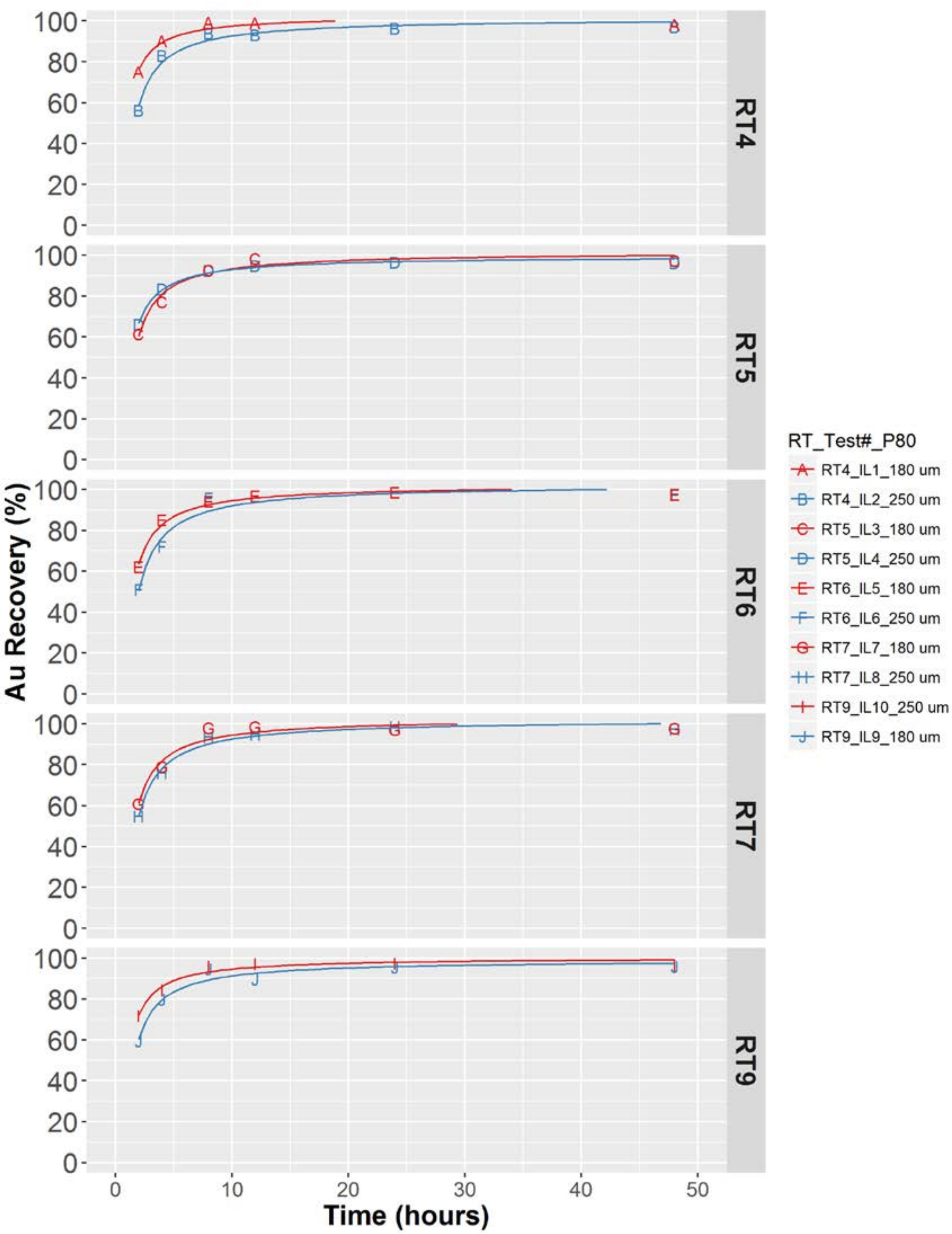


Figure 13-20: Intensive leach of Mozley concentrate

13.5.10.5 Phase 9 – Leach kinetic testwork

Following observations from the FS optimization test program, the decision was taken to add extra sampling times (12 and 18 hours) to the leach testwork to better characterize the gold leaching kinetics for each rock type. Figure 13-21 shows the results of kinetics tests from gravity tails at 180 µm (P_{80}). Results using 250 µm (P_{80}) gravity tails presented similar trends.

The leaching kinetics results were analyzed and it was found that for each rock type, after 18 hours of leach time, there was no extra recovery or the increment was not sufficient to justify the addition of an extra leach tank.

The latter observation was used to reduce the leaching retention time from 32 to 21 hours. The reduction in the leaching retention time translates into lower cyanide and lime consumption. An example of the analysis is presented on Table 13-23.

The important conclusions to be drawn from the Phase 9 and the FLS/Curtin testwork include:

- A high gravity recoverable gold content was confirmed;
- Improved leach results were obtained from Curtin on all samples (pH 10);
- The first intensive leach of gravity concentrates achieved excellent gold recoveries on gravity concentrates from all rock types ranging from 96% to 98%;
- Pre-conditioning and lead nitrate led to a reduction in leaching time;
- A reduction in cyanide consumption and required leach time were realized.

Table 13-23: Kinetic results from Phase 9 (PFS)

Rock type	Item	Phase 9					
		90 µm			180 µm		
		Leaching time, hours			Leaching time, hours		
		18	24	32	18	24	32
RT4	Au head grade, g/mt	0.44			0.52		
	Au residue, g/mt	0.07	0.07	0.07	0.08	0.08	0.08
	Au Recovery, %	83.0	83.2	83.2	83.9	84.5	84.6
	CN consumption, kg/mt	0.26	0.28	0.34	0.19	0.23	0.32
	CaO consumption, kg/mt	2.29	2.56	2.51	2.14	2.35	2.30
RT5	Au head grade, g/mt	0.50			0.65		
	Au residue, g/mt	0.09	0.09	0.09	0.12	0.11	0.11
	Au Recovery, %	81.1	81.8	82.0	81.6	82.7	83.0
	CN consumption, kg/mt	0.20	0.24	0.31	0.19	0.24	0.31
	CaO consumption, kg/mt	1.75	1.77	1.75	1.77	1.78	1.74
RT6	Au head grade, g/mt	0.55			0.57		
	Au residue, g/mt	0.18	0.18	0.18	0.20	0.20	0.20
	Au Recovery, %	67.5	67.5	67.5	64.9	65.1	65.2
	CN consumption, kg/mt	0.29	0.32	0.38	0.27	0.31	0.35
	CaO consumption, kg/mt	1.59	1.68	1.62	1.46	1.55	1.52
RT7B	Au head grade, g/mt	0.53			0.75		
	Au residue, g/mt	0.28	0.28	0.28	0.35	0.34	0.33
	Au Recovery, %	47.0	47.2	47.2	53.8	55.2	55.7
	CN consumption, kg/mt	0.48	0.51	0.53	0.36	0.41	0.51
	CaO consumption, kg/mt	1.86	2.17	2.14	1.73	2.05	2.02
RT9	Au head grade, g/mt	0.62			0.55		
	Au residue, g/mt	0.29	0.29	0.29	0.31	0.30	0.30
	Au Recovery, %	53.0	53.1	53.1	43.5	44.9	45.5
	CN consumption, kg/mt	0.39	0.40	0.47	0.27	0.36	0.43
	CaO consumption, kg/mt	1.58	1.70	1.67	1.32	1.34	1.30

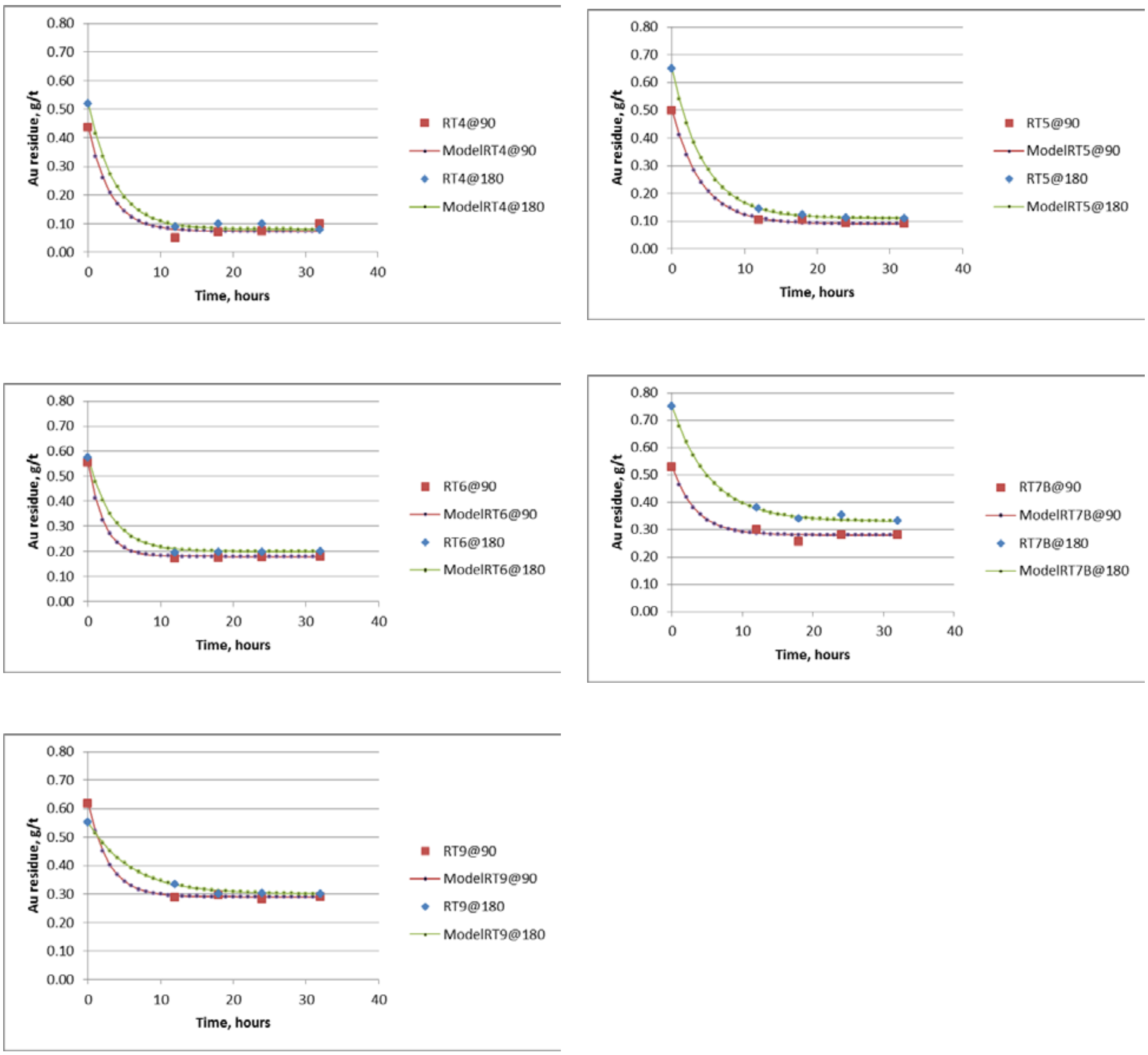


Figure 13-21: PFS (Phase 9) - Leach kinetics analyses according to rock type

13.5.11 PFS – Phase 10 - Stirred tank reactor (STR) leach tests

Phase 10 was conducted under the direction of Guy Deschênes (BBA), owing to his expertise on leaching gold ores containing antimony (Sb). The test program included the use of lead nitrate and control of dissolved oxygen levels under controlled leach conditions. This approach had previously been demonstrated effective at the Fort Knox Mine to control the adverse effects of antimony.

The objective of the Phase 10 test program was to determine whether antimony minerals were responsible for some low gold extractions experienced by rock types (RT7 and RT9). The antimony content of the RT7 sample tested (590 ppm) is an order of magnitude higher than that of RT9 (60 ppm or less).

Phase 10 included testing using 20 kg composites of RT7 and RT9.

The gold content in samples RT7-GR11, RT9-GR14 and RT9-V86, V89 were 0.46 g/t, 0.72 g/t and 0.50 g/t respectively. The RT7 sample contained 0.016% Cu, 4.7% Fe, 0.026% Zn and 2.0% S (S_{Tot}), whereas RT9 contained 0.014% Cu, 5.3% Fe, 0.013% Zn and 2.0% S (S_{Tot}).

Tests were conducted in stirred tank reactors (STR) under controlled conditions of the following variables; agitation, temperature, pH, free cyanide and dissolved oxygen (DO). The leaching conditions that were applied were those used for processing orebodies containing antimony minerals.

The results were compared to baseline conditions (0.5 h pre-treatment, pH 10.7, DO 4 ppm; 32 h leaching, 200 ppm NaCN, pH 10.7, DO 8 ppm). Test results for the RT7 and RT9 composite samples indicated only a modest improvement in gold extraction of 2-5%, when calculated on the basis of a gold balance developed around the leach solution and residues, i.e. gold recovery based on leach testwork results. This improvement resulted from a pre-treatment of four hours, with the addition of 100 g/mt lead nitrate and oxygen. However, if the interpretation is based upon the gold content of the leach residues only, which may be valid because the assayed (direct) head grades are the same for each rock type, it suggests no improvement.

The leaching profiles of the baseline conditions and new conditions using lead nitrate are comparable, which would indicate no sign of passivation of antimony minerals. The lack of interference by antimony minerals might be explained by no or insufficient liberation during grinding, or that surface passivation had already taken place in the prior treatment of the samples. Other explanations for a lack of improvement may be that that the gold is not liberated at the particle size selected, or that the gold is refractory, i.e. gold is in solid solution with the mineral. Seven tests with repeats indicated good reproducibility of the results.

Table 13-24: Phase 10 results (PFS)

Sample	Feed Size P_{80}, µm	CN Test No.	Time (h)	Pre-aeration pH	Pre-aeration DO (ppm)	$Pb(NO_3)_2$ (g/mt)	Leach pH	Leach D. O. mg/L	Reagent Cons. NaCN (kg/mt)	Reagent Cons. CaO (kg/mt)	Residual NaCN g	Au Extraction (%) 2h	5h	24h	32h	Residue grade Au, g/mt	Head grade (g/mt Au) Calc	Head grade (g/mt Au) Direct
GR11:1-3,7-9:RT7 RT7B:repeats	90	1	0.5	10.7	4	-	10.7	8	0.67	0.9	0.11	14	20	35	21.5	0.40	0.51	0.46
		2	4	9.8	8	-	9.9	8	0.68	0.6	0.22	8	8	14	26.5	0.36	0.49	
		3	4	9.8	6	100	9.9	8	0.71	0.5	0.20	7	28	28	27.1	0.39	0.54	
		7	2	10.0	9	100	10.2	8	0.74	0.3	0.13	14	20	30	23.5	0.38	0.50	
		7R	2	10.0	6	100	10.3	8	0.65	0.4	0.15	11	20	47	28.3	0.39	0.55	
		8	4	10.1	8	100	10.2	16	0.65	0.2	0.13	15	23	25	23.9	0.39	0.52	
		8R	4	10.1	16	100	10.2	19	0.61	0.3	0.17	19	21	30	25.7	0.38	0.52	
		9	4	10.1	6	200	10.2	8	0.63	0.1	0.18	15	21	27	25.4	0.40	0.53	
		9R	4	10.0	7	200	10.2	7	0.53	0.4	0.19	18	22	28	21.5	0.40	0.50	
GR14 RT9	86	4	0.5	10.2	4	-	10.6	8	0.47	0.7	0.16	37	55	58	44.8	0.34	0.62	0.72
		5	4	10.2	8	-	9.9	8	0.62	0.4	0.20	38	40	52	46.3	0.32	0.60	
		6	4	10.2	7	100	9.9	8	0.60	0.3	0.21	41	49	50	50.2	0.32	0.64	
GR14 RT9 (V86 & V89)	62	10	2	10.2	9	100	10.2	8	0.76	0.1	0.12	53	55	59	54.7	0.21	0.46	0.50
		10R	2	10.0	8	100	10.0	7	0.63	0.4	0.16	52	49	37	42.0	0.29	0.49	
		11	4	10.1	12	100	10.2	16	0.59	0.1	0.16	55	57	57	53.9	0.22	0.48	
		11R	4	10.2	16	100	10.2	15	0.47	0.3	0.16	54	56	52	55.5	0.20	0.46	
		12	4	9.9	8	200	10.1	8	0.52	0.2	0.15	50	52	54	54.0	0.22	0.48	
		12R	4	10.0	9	200	10.1	8	0.52	0.4	0.20	49	51	53	55.4	0.21	0.47	

Table 13-25: Reproducibility of cyanidation tests on the Livengood Gold Project (PFS)

Sample number	CN	Reagent Cons.		Gravity Tail Leach Extraction (% Au)	Residue Grade Au, g/mt	Head grade (g/mt Au)
	Test No.	NaCN (kg/mt)	CaO (kg/mt)			Calc
RT 7 and RT7B	7	0.74	0.3	23.5	0.38	0.50
	7R	0.65	0.4	28.3	0.39	0.55
	8	0.65	0.2	23.9	0.39	0.52
	8R	0.61	0.3	25.7	0.38	0.52
	9	0.63	0.1	25.4	0.40	0.53
	9R	0.53	0.4	21.5	0.40	0.50
RT9	10	0.76	0.1	54.7	0.21	0.46
	10R	0.63	0.4	42.0	0.29	0.49
	11	0.59	0.1	53.9	0.22	0.48
	11r	0.47	0.3	55.5	0.20	0.46
	12	0.52	0.2	54.0	0.22	0.48
	12R	0.52	0.4	55.4	0.21	0.47

Important conclusions that can be drawn from the Phase 10 test program include:

- For the samples tested, there was no clear evidence of passivation in the leaching profiles using conditions that are efficient for ores containing antimony minerals;
- Given the level of antimony and arsenic minerals that are present, this is a very unusual response. Either these minerals did not interfere, perhaps because they were not liberated, or the samples tested were altered by the previous grinding/gravity tests that were performed on them. Ageing may have also been a contributing factor;
- The response runs counter to the good recovery results of RT9 in the FS optimization, where the average gravity tail leach extraction was 62.9%. It also runs counter to the RT7 mini-optimization testwork that resulted in an average gold recovery of 53.6%. Both of these sets of tests were run on fresh core, where the impact of antimony should have been quite pronounced, but yet gold recoveries were higher than the Phase 10 testwork. Particle size cannot be used to explain the differences in the Phase 10 tests, because the P_{80} of 60-90 µm was similar to that used in the FS optimization tests;
- The results reinforce the need to consider detailed gold deportment analysis of gravity and leaching products in the next phase of testwork;
- Seven tests with repeats indicated a good reproducibility of the results;
- Lead nitrate addition may have increased the gold leaching kinetics.

13.6 PFS Recovery equations

Recovery equations were developed using the results of the optimization, variability, continuous, and Phase 9 and Phase 10 test programs.

The following table presents the average recovery (gravity + CIL) estimated for each of the five rock types within the Livengood gold deposit. With the exception of RT9, the gold recovery results include a 2% reduction for coarser grind at 180 µm (P_{80})

Table 13-26: Average gold recovery (gravity + CIL) estimated for each rock type

Rock Type	Au Recovery (%)
RT4	81.8
RT5	84.7
RT6	75.6
RT7	62.4(1)
RT9	69.6(2)

(1) Weighted average based on recovery correlation to quartz – stibnite + jamesonite

(2) Weighted average based on grade/frequency distribution of the 15 × 15 ×10 meter block model.

The data from all of the testwork programs was analyzed using several criteria to discard possible testwork with less-than-ideal or erroneous conditions, i.e. tests with low DO or low CN level, wrong particle size, etc. The filtered data (qualified data) of tests including all grind sizes was used to develop recovery estimates for all rock types based on calculated head grade. For example, the optimization and Phase 9 results were averaged and each testwork program contributed one point to the data set. It was understood that this was the most appropriate treatment, especially considering that both optimization and Phase 9 testwork was conducted on a master composite of each rock type.

Rock Types: RT4, RT5 and RT6

The results from the entire body of testwork were analyzed with the objective of developing relationships to characterize the gold leaching performance of rock types RT4, RT5 and RT6. It was not possible to develop a gold grade vs gold recovery model(s), based on the available data for these rock types.

An average gold recovery for each rock type was estimated from the results of the different testwork programs. A 2% recovery reduction was applied when converting leach test results from 90 to 180 µm (F_{80}).

The cyanide and lime consumptions were estimated as an average of the reagent consumptions observed from both the continuous and Phase 9 testwork programs. Variability or optimization results were not used, because when comparing testwork results, it was found that the higher cyanide additions did not improve the gold recovery results.

Rock Types: RT7 and RT9

Testwork results were analyzed to characterize the most important gold recovery drivers for the RT7 and RT9 rock types. A strong relationship between quartz - stibnite + jamesonite and grade was found for RT7, which is depicted in Figure 13-22. Stibnite and jamesonite are antimony-bearing minerals.

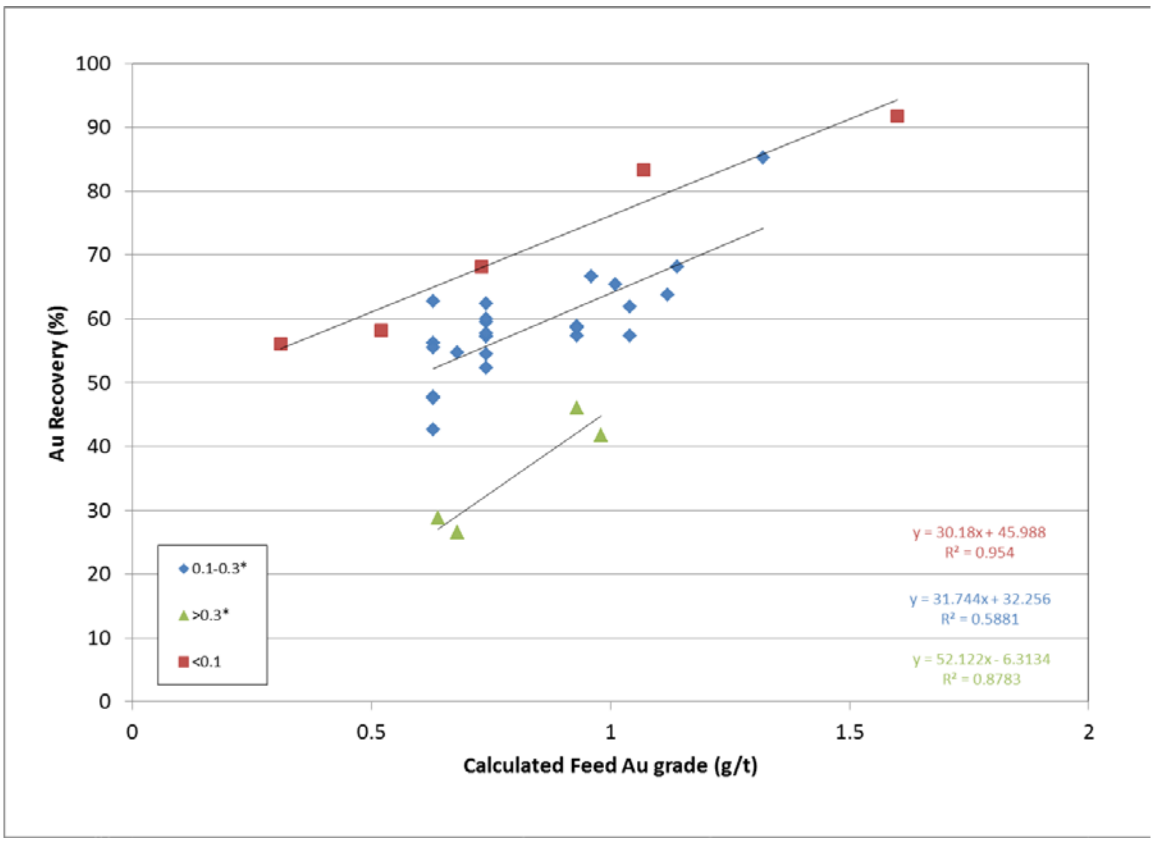


Figure 13-22: PFS (Phase 9) - RT7 gold recovery vs head grade at different Quartz-Stibnite+Jamesonite levels

The RT9 testwork results were examined using advanced statistical techniques (R/ggplot2 software) in a number of ways in an attempt to establish the most defensible relationship to estimate gold recovery.

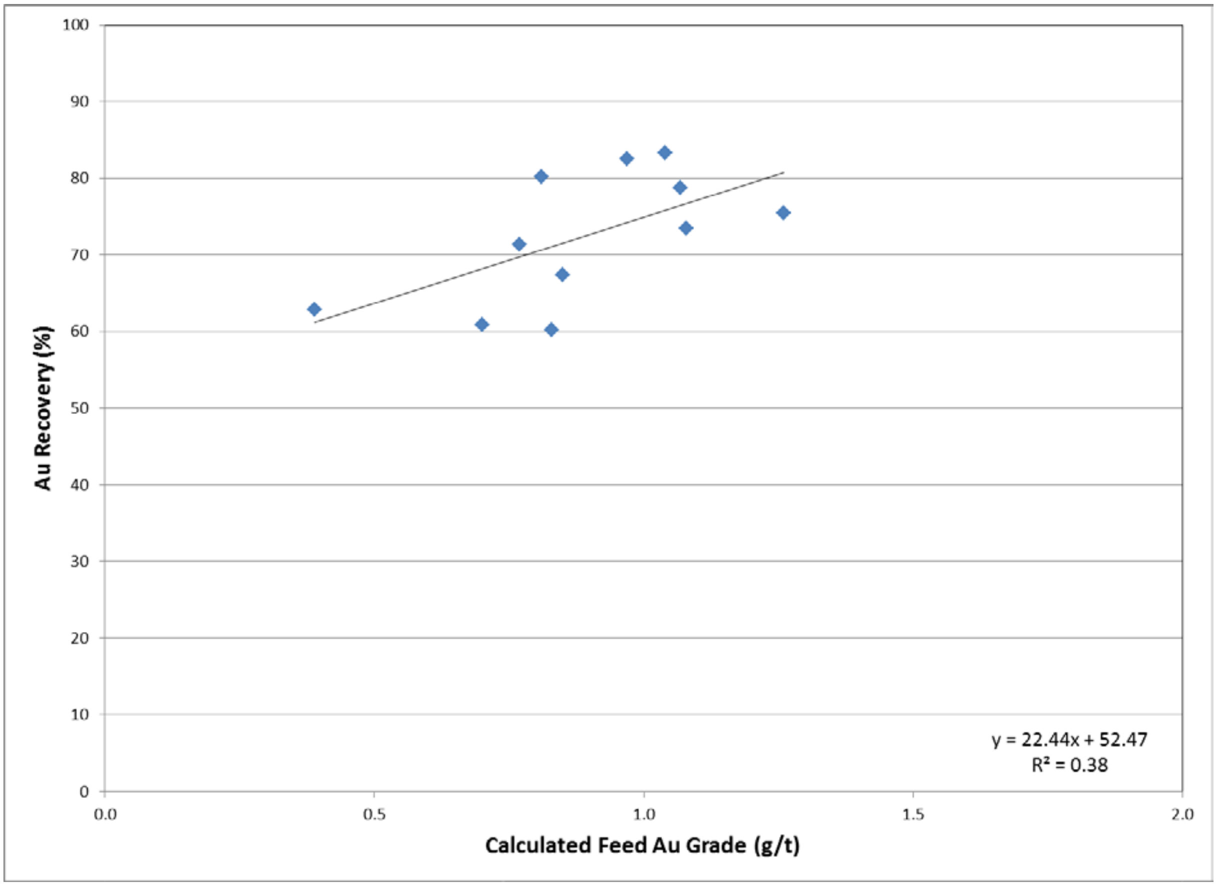


Figure 13-23: PFS (Phase 9) - RT9 gold recovery vs head grade

In the case of rock type RT9, a head grade/recovery relationship was found (Figure 13-23), but it is probable that there is a quartz - stibnite + jamesonite or antimony relationship as well. However, the current available data suggests that the quartz - stibnite + jamesonite index is under 0.1 and it is not possible to establish a strong relation showing any detrimental effect on gold leaching.

Given that the curve for rock type RT9 was developed using all qualified data, including grinds of between (P_{80}) 80 and 250 µm, it was decided not to apply a 2% deduction in gold recovery to compensate for a coarser grind of product (P_{80} of 180 µm).

13.7 PFS Flowsheet development

13.7.1 Comparative studies

13.7.1.1 Comminution optimization with drilling and blasting (D&B)

Establishing Run-of-Mine (ROM) particle size distribution (PSD) estimates represents an important step for developing a baseline for mineral processing costs. Given that the drilling and blasting process is typically regarded as the first stage of comminution, its efficiency will directly impact the subsequent activities, namely crushing and grinding. To assess and quantify these impacts for the Livengood Gold Project, various blast design scenarios were compiled and simulated for each ore-bearing geological domain, namely RT4 (Cambrian), RT5 and RT6 (Upper and Lower Seds), and RT9 (Volcanics).

The first step towards generating ROM PSD curve estimates consisted of compiling all available geological and geo-mechanical parameters. These parameters were then imported into a break radius modeling software (AEGIS), which estimated the degree of breakage and area of influence of a typical blast hole charge. Based on the resultant break radii, preliminary burden and spacing values were then determined for each rock type and/or explosive charge.

After determining burden and spacing values, the remaining blast design and geo-mechanical parameters were compiled and integrated in a JKMRC Fragmentation (software) model. The software will use these inputs to generate PSD curves for the ROM material produced by various blast designs in the different geological domains. This is referred to as a drill & blast (D&B) analysis.

The results of the D&B exercise were used in conjunction with comminution design software (Bruno and JKSimMet) to study the impact of the PSD on throughput and specific energy.

The impact of the D&B in the current study was an increase of 6.4% in the average throughput of the project from 49,468 to 52,630 t/d (or 44,877 to 47,745 mt/d).

Future work

With regards to the Volcanics domain, it must be noted that geo-mechanical test results were not available and were therefore assumed. To confirm the resulting ROM PSD values obtained for the Volcanics (RT9) domain, a re-iteration of the simulation work is recommended, once geo-mechanical testing is completed.

13.7.1.2 Comminution optimization with pre-crushing

Simulations were conducted to study the opportunities to increase throughput by adding a pre-crusher. The simulations indicate that a 25% to 30% increase in tonnage can be achieved by including a pre-crushing step.

13.7.1.3 Throughput studies

The higher tonnage comminution circuit from the FS was challenged during the development of the PFS via an extensive throughput rationalization study. This study investigated the impact that grinding circuit configuration, ROM particle size, pre-crushing and target particle size, would have on equipment size, power efficiency, overall throughput, OPEX and CAPEX. The scenarios that were investigated include the following:

- Pre-crushing + single line SABC circuit;
- Dual line pre-crushing + SABC circuit;
- SAG mill motor type (twin pinion vswrap around);
- Grinding circuit product size target of (P_{80}) 90 µm vs180 µm;
- Impact of drill and blast (finer ROM) on throughput.

Analysis of the leaching testwork conducted in parallel to the throughput studies indicated that the gold recovery was relatively insensitive to grind in the range of (P_{80}) 90 µm to 180 µm. Based on this observation, it was decided to coarsen the grind to (P_{80}) 180 µm, which resulted in a significant throughput increase of 25%, which more than compensated for the estimated gold recovery losses of 2%.

Due to the significantly reduced capital cost and lower project execution risk, a single line (SABC + pre-crushing) circuit was adopted for further development and use as the base case for the PFS, despite having a lower throughput capability than the circuit proposed by the FS study. The final configuration also assumes additional throughput by applying optimized drill and blast techniques to produce a finer ROM product for the primary crusher.

13.7.1.4 Gravity concentration

The GRG results indicate significant potential for gravity gold recovery. Based on the results of the FLS simulations completed for the FS, a gravity circuit with two parallel lines, each with four gravity concentrators, was incorporated into the conceptual design.

13.7.1.5 Leach time, lead nitrate, and pre-oxidation

Testwork evidence also indicates that the gold recovery kinetics slow down significantly after 21 hours of leaching time. The gain/loss in gold recovery, estimated at ±1%, does not justify the addition of extra leaching tanks; six additional leach tanks would be required to extend the residence time from 21 to 32 hours. Lead nitrate can help to improve kinetics, but will require more testwork to quantify the magnitude of reduction of leaching time.

Pre-oxidation was normalized to four hours in the course of completing the Phases 8 and 9 testwork, which showed that by combining lead nitrate and O_2 during the pre-conditioning stage, it was possible to reduce the leaching time and also reduce the cyanide consumption as a result of reducing leach time and by oxidizing any sulfides that could consume cyanide.

13.7.1.6 WOL vs Flotation

A trade-off study was conducted between a whole gravity tails CIL configuration (WOL) and a flotation configuration (FLOT), where gravity tails float concentrates undergo CIL.

The result of the trade-off study supported the decision to select gravity, followed by CIL of the gravity tailings as the design process.

Recovery – WOL vs Flotation

The summarized results of WOL and FLOT testing from both the FS and Phase 8 (PFS) are presented in Table 13-27. At the bottom of the table the differences in recovery between the WOL and FLOT options are also presented. Due to different composites being used in the FS as compared to the Phase 8 testwork, the differences calculated for the Phase 8b and Phase 8d results were calculated, not against the WOL recovery results from the FS, but against corresponding WOL results from the same samples of the Phase 8 test program. Some results suggested slightly higher recoveries for the FLOT option, but generally, the WOL option resulted in a significantly higher recovery.

Table 13-27: Summary of recovery results from different testwork programs

Testwork Program	FS		Phase 8a	Phase 8b	Phase 8b	Phase 8d
Configuration	WOL	FLOT	WOL	FLOT	FLOT	FLOT
P_{80} (µm)	90	90	60/75	90	180	180
Rock types	**Au Recovery (%)**					
RT4	84.2	-	-	-	-	-
RT5	87.7	76.1	82	81	68	72.6
RT6	76.7	67.4	-	-	-	-
RT7	58.2	-	-	-	-	-
RT9	78.1	66.8	62	68	65	67.4
Rock types	**Au Recovery difference compared to WOL (%)**					
RT4	-	-	-	-	-	-
RT5	-	-11.6	-	-1	-14	-9.4
RT6	-	-9.3	-	-	-	-
RT7	-	-	-	-	-	-
RT9	-	-11.3	-	5	2	4.4

In the absence of consistent comparables between WOL and FLOT between the different composites, the decision was taken to assume a recovery difference (FLOT – WOL) for each rock type that was likely to be favorable to the FLOT option. In the case of rock types RT4, RT5, and RT6, a relative difference of -5% was assumed, which was generally less than what had been observed, at least for RT5 and RT6. In the case of RT7 and RT9, the recovery difference was assumed to be +5%, implying a higher recovery for the FLOT option as compared to the WOL option. These assumptions were developed as a means of evaluating the FLOT option in the best light for the purpose of conducting the trade-off (Table 13-28). If the WOL option delivered higher NPV than the FLOT option, even under these assumptions, then it would validate the selection of the WOL flowsheet. Using these assumed recovery differences, weighted average recoveries were calculated for both options, with the result of 76 wt% avg. gold recovery for WOL and 74 wt% avg. gold recovery for FLOT.

Table 13-28: Simulated gold recoveries for the WOL vs FLOT trade-off

Rock types	Relative difference	WOL	FLOT
RT4	-5%	78%	73%
RT5	-5%	85%	80%
RT6	-5%	76%	71%
RT7	5%	62%	67%
RT9	5%	69%	74%
Wt. Avg.		**76%**	**74%**

Capital cost estimate

The current crushing and grinding configuration produces a particle size (P_{80}) of 180 µm that is fed to each of the configurations (WOL or FLOT). Both process configurations are equipped with the same gravity circuit.

In the FLOT configuration a flotation concentrate (12% mass pull) is produced and is fed to a CIL circuit that is substantially smaller than the CIL circuit in the WOL configuration. Additionally, the equipment required for cyanide detoxification is smaller.

In the WOL configuration a greater volume of slurry would need to go through thickening and detoxification, prior to going to the tailings management facility (TMF). The cyanide detoxification tanks are smaller in the FLOT than WOL configurations due to the smaller volumetric flow of CIL tails in the FLOT configuration. On the other hand the CN concentration in the FLOT configuration is higher than WOL, meaning that the unit requirements of SO_2 are higher in the Detox system of the FLOT configuration.

All equipment costs for the WOL and FLOT configurations were estimated using equipment cost information from BBA's projects database. Total CAPEX indicates an increase of $11.7M by adopting the WOL option.

Operating cost estimate

Operating cost estimates were prepared for both alternatives. The WOL option indicated a slight increase in operating cost over the FLOT option $7.44/t ($8.21/mt) vs $7.13/t ($7.86/mt).

Cash flow analysis

Discounted cash flow models (5% discount rate and $1,250/oz gold) where developed to determine the Net Present Value (NPV) for each alternative based on the revenues, capital costs and operating costs. The weighted average distribution of rock types from the PFS LOM plan was used to determine the overall gold recovery for each alternative.

The NPV values were very similar for both configurations: $5,322M for WOL and $5,200M for FLOT, with the WOL alternative being slightly more profitable (+$120M). Since the WOL and FLOT alternatives have similar capital costs, this result could be explained by WOL having a better gold recovery, while the FLOT alternative had lower operating costs.

Conclusion

The results indicate that the selection of WOL was adequate, because the FLOT configuration was slightly less economic on an NPV basis than WOL, despite giving every advantage to the FLOT option with respect to recovery potential.

13.7.1.7 CIL vs CIP

The review of the underlying geology has allowed for a better understanding of the preg-robbing nature and distribution of the deposit. Using the preg-robbing index, the main observation is that the volcanics typically present very low preg-robbing values, while both sediment rock types (upper and lower) present a higher level of preg-robbing. This can be classified as a very systematic behavior. On that basis, the Livengood resources are probably best processed using carbon in leach (CIL), instead of carbon in pulp (CIP). Furthermore, the sediment rock types are important contributors to the gold resource and will likely have to be mined concurrently to the main volcanics. As well, in some cases there is the inclusion of sediments cutting through the volcanics that will induce preg-robbing.

13.7.1.8 Sulphur burner

In the FS, sodium metabisulfite (SMBS) was used to supply SO_2 to the cyanide detoxification process at a rate of 1.63 lb/t (0.82 kg/mt). At 100,000 t/d (90,718 mt/d) and based on a price of $0.37/lb, the total annual operating cost for producing SO_2 with SMBS was approximately $22.0M. Upon review of the SMBS consumption estimate, BBA concluded that an opportunity to reduce the cost of SO_2 was highly probable.

A trade-off was conducted by BBA comparing the available options on the basis of their operating and capital costs. This comparative study evaluated three (3) possible options for the production or supply of SO_2: (1) mixing of sodium metabisulfite (base case), (2) burning of elemental sulfur using a sulfur burner, and (3) direct injection of liquid SO_2.

Key assumptions

The following list contains the assumptions used in conducting this study:

- The throughput of the process plant for all options was 100,000 t/d (90,718 mt/d) or the plant throughput of the FS;
- SO_2 to CN ratio was determined by testwork (SGS post feasibility testwork program - Project report 50223-002 – December 2013);
- This trade-off study covers only the cost, capital and operating, for the supply of SO_2 for the cyanide detoxification process. Any other costs outside of this scope are not covered, including mining, front-end process, infrastructure, tailings pond and tailings management. These other costs are neglected in the analysis, since they would not impact the selection of the SO_2 supply;
- Equipment pricing was determined through updated budget quotes for the major equipment or historical prices. The other equipment costs were determined using BBA's equipment cost database and were based on the required equipment size;

- A quotation has been recently obtained from the supplier for elemental sulfur and sodium metabisulfite. The cost of liquid SO_2 was estimated based on BBA's pricing database;
- When using a sulfur burner with less than 100% availability, SMBS is used as a back-up during operation downtime.

Feasibility study versus trade-off cost comparison

Table 13-29 presents the annual operating cost and the cost of the reagent.

Table 13-29: Annual operating cost comparison

	Annual Cost (M$/y)	Reagent Cost ($/t)	Reagent Reference
Sodium metabisulfite	22.0	820	2013 Feasibility Study
Elemental sulfur	5.4	552	2016 Supplier Quote
Liquid SO_2	34.0	1,830	BBA Estimate

Even using the feasibility study consumption numbers, burning of elemental sulfur to produce SO_2 would have been advantageous. The yearly operating cost for the project would have been approximately $17.0M less.

Conclusions and recommendations

Based on the cash flow analysis, the liquid SO_2 option is the most costly followed by the SMBS option. Although the highest initial capital cost expenditure is required, burning of elemental sulfur in a sulfur burner presents the lowest cost method to produce SO_2 over the LOM. Even at 50% sulfur burner availability and SMBS compensating for the difference, the sulfur option is the most economical with over $100M in savings over the LOM.

The sensitivity analysis on the price of sulfur also demonstrates that the sulfur burner option is the most attractive. Payback of the equipment is within one (1) year even at double the sulfur price.

BBA recommends that THM pursue the sulfur burner option for the Livengood Gold Project.

13.7.2 Flowsheet development summary

Livengood gold ore has demonstrated that it is very amenable to gravity concentration as a substantial proportion of the gold is free and liberated at a reasonably coarse grind. GRG results confirmed the excellent potential for gravity recoverable gold.

The fact that Livengood gold 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 whether test condition changes were making improvements to the process. 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.

Gold deportment studies indicated that a substantial amount of the finer gold had at least a 25% or greater exposure, allowing it to be recovered by cyanidation.

However, some of the exposed gold 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.

On the basis of the substantial testwork conducted on the major rock types and trade-off study (WOL vs FLOT), 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 gold 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 a 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.

Analysis of leaching (CIL) kinetic tests with preconditioning with O_2 (3h) and lead nitrate has shown that the gold is leached within 21 hours of retention time. The reduction of leaching time from 32 in the FS to 21 hours impacts CAPEX (fewer leach tanks) and OPEX (lower CN consumption).

The addition of pre-crushing was recommended by BBA to enhance the operation of the SAG mill, by providing a narrower feed particle size, thereby reducing variability, which will translate into increased efficiency. The estimated increase in throughput from the addition of pre-crushing is 25 to 30%.

Grinding simulations of a single line SABC + pre-crushing circuit has shown that there is a 33% increase in throughput if the grind size is relaxed from 90 to 180 µm (P_{80}) and by using optimized drill and blast techniques.

Sulphur dioxide (SO_2) produced by a sulfur burner will significantly reduce the OPEX costs for cyanide detoxification.

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

13.8 Opportunities for further investigation

Sample characterization

The preparation of ore blends based on the LOM, i.e. years 1-3, 4-6, etc., is recommended to test the influence of lead nitrate on gold recovery. It is understood that the different rock types require different amounts of lead nitrate, so it is important to study the gold leaching response of the ore blend to a lead nitrate requirement estimated from the antimony content of each ore blend.

Gravity

Based on gravity modeling, it is recommended that new gravity circuit simulations based on the results of the latest GRG work be conducted. Two scenarios should be compared: gravity circuit with cyclone feed vs gravity circuit with cyclone underflow feed.

Based on the high GRG content of the Livengood ore, it is recommended to conduct a gold deportment study based on the gravity products for each rock type. The criteria for the gravity testwork should be based on the results of the gravity simulations, i.e. the future circuit configuration. The analysis of the Knelson gravity tails should be analyzed through microscopy to investigate gold liberation characteristics and also by diagnostic leach techniques to investigate potential gold minerals that are refractory to cyanide.

BBA does not recommend the blending of the Knelson gravity concentrator and the Mozley tails. Mozley tails should be cyanide leached using intensive testwork conditions.

Variability testwork

Based on the current understanding of the ore response to leaching, it is recommended to perform gold grade leaching variability testwork using samples with bins having higher Quartz + Stibnite + Jamesonite indices.

Optimization testwork

Optimization testwork should be conducted to validate the conclusions and observations of Phase 9 and repeatability should be studied in this phase of work.

Solid / Liquid separation testwork

The new product size of 180 µm (P_{80}) for the Livengood Gold Project requires new settling testwork; static as well as dynamic settling testwork are recommended. The static work can be used to screen potential flocculant suppliers for the Livengood gold ore. Dynamic settling testwork by vendors is recommended as part as the equipment sizing and bidding process. Additional savings on reagents (flocculant) are expected for the coarser grind size.

Cyanide detoxification testwork

For cyanide detoxification, the following reagent consumptions were assumed for the PFS:

- Lime = 0.62 lb/t (0.28 kg/mt);
- Copper sulfate = 0.09 lb/t (0.045 kg/mt);
- S (elemental) = 0.52 lb/t (0.26 kg/mt);
- Sodium metabisulfite = 0.38 lb/t (0.19 kg/mt).

Confirmatory detoxification testwork with a particle size distribution target of 180 µm (P_{80}) is recommended. Potential reagent savings are expected as a result of lower liberation of detrimental metals that could otherwise consume cyanide reagents.

Stirred tank reactor optimization

After the optimization testwork is completed, further stirred tank reactor (STR) testwork should be conducted to validate the optimum results. In particular, the gold leaching performance of RT7 and RT9 should be studied.

WOL-CIP vs CIL testwork

Early leaching testwork (McClelland 3526, 2011) without carbon (CN tests) indicated a significant recovery reduction as compared to leaching testwork with carbon. Analysis of the testwork indicates that the leaching testwork was conducted without a gravity recovery step and cyanide consumption in the CIL testwork was five times higher than the CN tests. However, in the recent FLS/Curtin University leaching testwork, which leached gravity tails from GRG testwork, the results indicate similar and generally better gold recoveries as compared to the CIL testwork at SGS Vancouver, i.e. leaching in the presence of carbon. This observation should be validated in future testwork due to the potential advantages of using a leach + CIP circuit, which is a leach followed by dedicated carbon-in-pulp tanks. An additional observation was that similar lower cyanide consumptions were achieved in the Curtin testwork program compared to the CIL testwork in Phase 9.

One disadvantage of CIL is the high volume of gold locked in the carbon due to the relatively high volume of carbon utilized. An alternative process, the CIP carrousel by Kemix, has the advantage of being able to increase the carbon concentration per tank to 50-60 g/l (vs 20 g/l for CIL) and reduce the carbon inventory in the carbon adsorption circuit. The impact of the higher concentration of carbon is that the carbon handling circuit in CIP has a smaller footprint than the CIL circuit, permitting the installation of the CIP circuit inside the process plant. Having the carbon handling system inside the building is more operator-friendly and permits better CIP system maintenance. This is as compared to the lower density of carbon of the CIL circuit that requires higher volumes of carbon to be treated by a larger absorption, desorption and reactivation (ADR) plant.

An opportunity that BBA recommends is further testing of the comparison of CIL vs leach and CIP of the gravity tails.

Grinding simulations

Simulations based on yearly composites are recommended. It is known that there is a higher percentage of RT4 and that it is softer than the rest of the rock types. Blending by year can be used to further optimize the financial models.

Carbon loading testwork and simulation (CIL)

Both a qualified laboratory and equipment vendor(s) should be approached to undertake carbon loading testwork and simulation work of the proposed CIL carbon handling system to confirm the assumptions made in this study. This work will lead to the selection of the most appropriate carbon elution system (high pressure ZADRA vs AARL).

Oxygen uptake tests

BBA recommends that oxygen uptake tests be performed by more than one service supplier to confirm the oxygen consumption for the Livengood Gold Project.

14. MINERAL RESOURCE ESTIMATE

14.1 Mineral resource estimation methodology

The global mineral resource estimate was prepared based on a resource model 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.

Note that the resource modeling work described and the analytical measures reported in this section were done using metric units. Where it is deemed pertinent (i.e. to support summary production statistics), the equivalent measure in imperial units have been provided.

14.2 Data used

The THM data available for the PFS model comprised 717,435 ft (218,674 m) of core and RC drilling, plus trench data. Historical drilling and sampling is shown in Table 14-1. Drilling performed by THM is shown in Table 14-2. The historical data represent about 2% of the total information used. The use of historical 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, 2008b, and 2009; Klipfel et al., 2009a and 2009b). For data validation purposes, in 2011, Mr. Tim Carew, P.Geo., of SRK (Canada), checked the assay data for a sample of drillholes (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 14-3.

Table 14-1: Historical drilling and sampling

Year	Company	Method	Number of Sites	Feet	Meters
1976	Homestake	Percussion	5	994	303
1981	Occidental	Percussion	6	988	301
1989	AMAX	Trench	2	525	160
1990	AMAX	RC	3	1,050	320
1997	Placer Dome	Core	8	3,467	1,057
2003	AngloGold	RC	8	4,968	1,514
2004	AngloGold	Trench	8	892	272
2004	AngloGold	Core	4	2,500	762
		Total	**44**	**15,384**	**4,689**

Table 14-2: THM resource drilling and sampling

Year	Company	Method	Number of Sites	Feet	Meters
2006	THM	Core	7	4,027	1,227
2007	THM	Core	15	14,471	4,411
2008	THM	Core	9	7,185	2,190
2008	THM	Trench	4	261	80
2008	THM	RC	109	93,402	28,469
2009	THM	Core	12	15,003	4,573
2009	THM	RC	195	196,243	59,815
2010	THM	Core	38	43,472	13,250
2010	THM	RC	195	184,717	56,302
2011	THM	RC	111	94,219	28,718
2011	THM	Core	53	44,260	13,490
2012	THM	Core	5	6,469	1,972
		Total	**753**	**703,730**	**214,497**

Table 14-3: 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**			

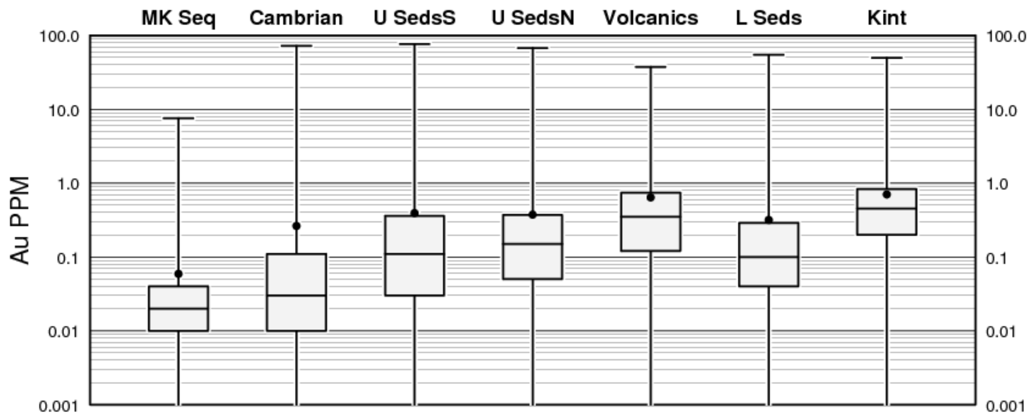
14.3 Data analysis

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 14-4. 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 14-4: Statistical summary of assay data

Element	Units	N	Mean	Maximum	Std. Dev.	C.V.
Au	ppm	34,786	0.40	56.2	1.22	3.0
Ag	ppm	12,969	0.41	440	4.07	10.0
Cu	ppm	12,969	42	1,120	34	0.8
Pb	ppm	12,969	19	9,240	128	6.7
As	ppm	12,971	2,169	137,000	4,181	1.9
Sb	ppm	12,969	221	138,000	2,394	10.8
Zn	ppm	12,969	186	3,440	221	1.2
Fe	%	12,708	4.3	21.3	1.4	0.3
Mo	ppm	12,969	5.5	74.0	6.9	1.3
S	%	12,081	1.4	18.4	1.4	1.0
Te	ppm	12,063	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

14.4 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 modeled material is undifferentiated Upper Sediments, with a small amount of Volcanics and Lower Sediments. These represent the major stratigraphic units that host the mineralization. No other geologic features with possible controls were modeled.

14.5 Composite statistics

All of the available drilling was composited into fixed length 10 m composites. Composite residuals less than 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 de-clustered by estimating a nearest-neighbor value into each block. The de-clustered composite statistics are tabulated below (Table 14-5).

Table 14-5: Gold composite statistics

Statistics	Value
Mean:	0.36
Variance:	0.32
C.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 14-6). With MIK, top cutting of the assays is not necessary. In this case all composite values higher than 2.0 g/mt Au (the highest threshold) are treated the same as "high grade".

Table 14-6: Gold indicator statistics

Rock Type	Threshold Au g/mt	Data		Metal		Median
		%	Cum%	%	Cum%	
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 and 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 drillhole, the grade profile was examined on either side of the contact.

Contacts were evaluated from the Cambrian into 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 the PFS did not appear to alter these relationships and the contact analysis was not repeated.

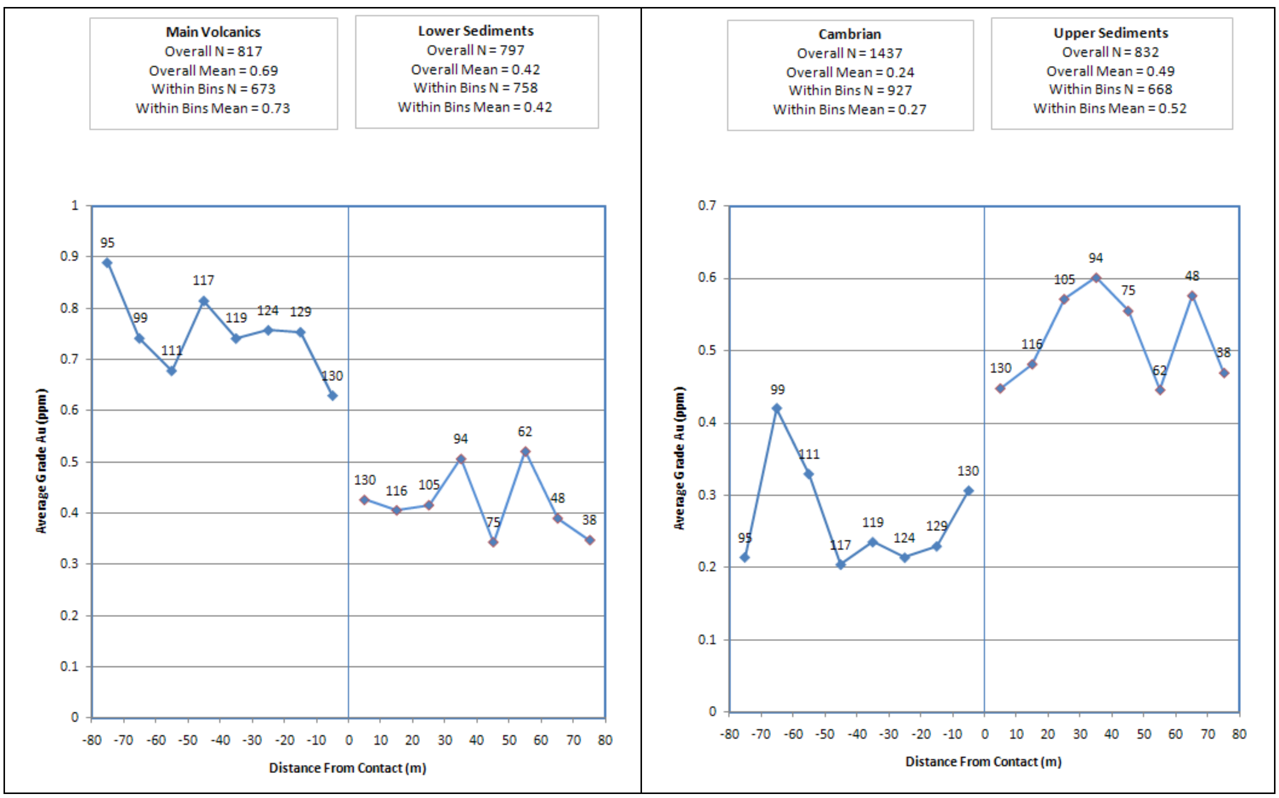


Figure 14-2: Contact plots

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.

14.6 Spatial statistics

There was no additional data available for this PFS as compared to the FS. Therefore, the variography for gold from the FS was retained.

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 14-7.

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 14-7: 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

Indicator	Sill	Range X	Range Y	Range Z
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
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

14.7 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 and used metric units of measure. The model extents are shown in Table 14-8.

The block size was selected based on the drillhole spacing of 50 m to 75 m.

Table 14-8: 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 14-9 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 drillholes for an estimate to be made. Each of the gold indicators was estimated independently.

Table 14-9: 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 × 4 × 1
Search Distances (m)	100 (Maj.), 100 (Semi-Maj.), 60 (Min.)
Search Rotation	Maj. -5º →190º, Semi-Maj. 100º

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 × 4 × 1
Search Distances (m)	200 (Maj.), 160 (Semi-Maj.), 120 (Min.)
Search Rotation	Maj. -5º →190º, Semi-Maj. 100º

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 × 4 × 1
Search Distances (m)	300 (Maj.), 250 (Semi-Maj.), 200 (Min.)
Search Rotation	Maj. -5º →190º, Semi-Maj. 100º

14.8 Model validation

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

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 de-clustered 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 favorably to the composite values, with areas of some divergence corresponding to swaths with a low number of samples.

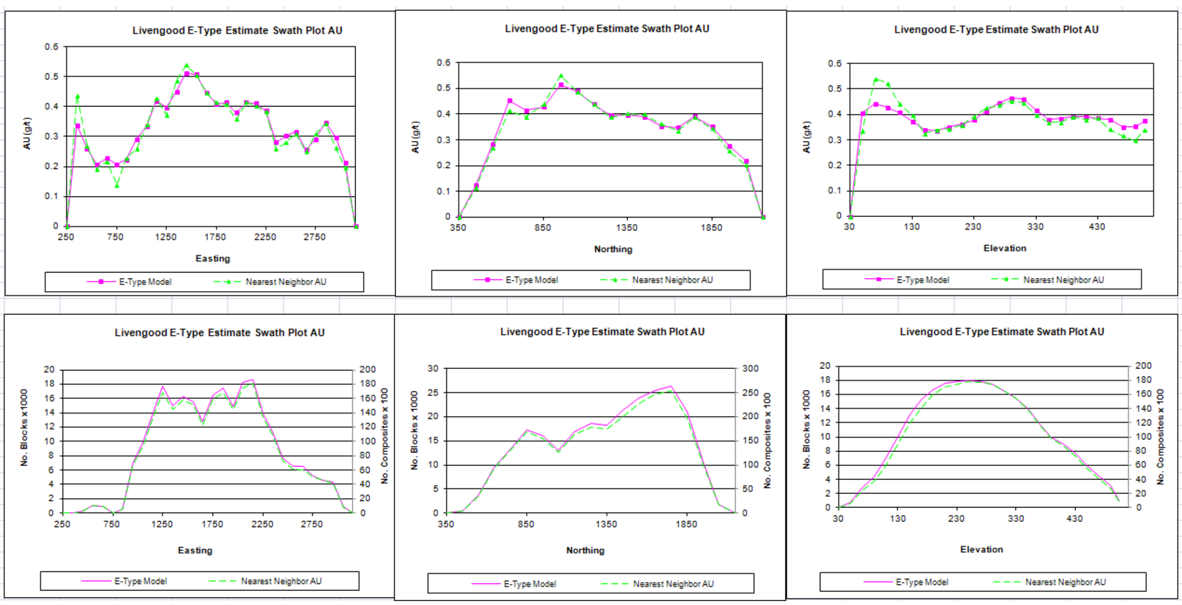


Figure 14-3: Swath plots of E-Type estimate vs nearest neighbor

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 drillholes, 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).

14.9 Post-processing of the MIK model

The post-processing of the MIK model 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:

$$Zmax = Zcn + 3(Zn - Zcn)$$

In this formula, Zcn is the uppermost indicator threshold and Zn is the mean of values 'greater than' Zcn. Considering a mean of 3.45 g/mt (raw composites), the maximum grade used in the post-processing was calculated to be 6.35 g/mt.

The MIK 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 40 m × 40 m, or approximately half the drill spacing.

The method used for the change of support was an affine 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 initially assumed to be 5 m × 5 m × 10 m. 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 7.5 m × 7.5 m × 10 m.

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.

14.10 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 drillholes 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 34 m of the nearest composite, and are informed by 27 composites from at least eight drillholes. On average, Inferred blocks are within 84 m of the nearest composite, and are informed by 20 composites from at least six drillholes.

14.11 Mineral resource estimation

The CIM Definition Standards for Mineral Resources and Mineral Reserves defines a mineral resource as:

"A concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for eventual economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

14.12 Pit constraining optimization criteria

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. The deposit gold mineralization is amenable for open pit extraction. To determine the quantities of material offering “reasonable prospects for eventual economic extraction” by an open pit, the author used the Lerchs-Grossman © economic algorithm, which constructs lists of related blocks that should or should not be mined. The final list defines a surface pit shell that has the highest possible total value, while honoring the required surface mine slope and economic parameters.

Economic parameters used in the analysis are based on the three year trailing average gold price ($1,230/oz) at August 26, 2016, and the following general assumptions (Table 14-10):

Table 14-10: Pit constraining parameters used for the Livengood Gold Project

Parameter	Unit	Rock Type 4	Rock Type 5	Rock Type 6	Rock Type 7	Rock Type 8	Rock Type 9
Mining Cost	$/total mt	1.77	1.77	1.77	1.77	1.77	1.77
Au Cut-Off	g/mt	0.33	0.32	0.35	0.40-0.85	0.38	0.38
Processing Cost	$/process mt	9.03	9.55	9.42	9.25	9.87	9.87
Au Recovery	%	80.4	86.5	78.3	31-67	75.4	75.4
Administrative Cost	$/process mt	1.07	1.07	1.07	1.07	1.07	1.07
Royalty	%	3	3	3	3	3	3
Au Selling Price	$/oz	1,230	1,230	1,230	1,230	1,230	1,230
Overall Slope Angle	Degrees	40	40	40	40	40	40

The parameters listed in Table 14-10 define a realistic basis to estimate the Mineral Resource for the Livengood Gold Project and are representative of similar mining operations throughout North America. The Mineral Resource has been limited to mineralized material that occurs within the pit shells and that could be scheduled to be processed based on the defined cut-off grade by rock type. Rock Type 7 was assumed to have a variable cut-off grade related to metallurgical ratios of the percentage of quartz-stibnite + jamesonite mineralization in each estimated model block. All other material within the defined pit shells was characterized as non-mineralized material.

The reader is cautioned that the results from the pit optimization are used solely for testing the “reasonable prospects for eventual economic extraction” by an open pit and do not represent an economic study as is required to evaluate mineral reserves. The author considers that blocks located within a conceptual pit shell are amenable for open pit extraction and can be reported as the Mineral Resource for the Project.

The Mineral Resource Estimate for the Project is summarized in Table 14-11. The estimate is reported in both metric and imperial measures to maintain report consistency. Mineral resources are reported at various cut-off grades to reflect the throughput factors and varying costs by rock type, for processing at Livengood. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability

Table 14-11: Livengood Gold Project mineral resource estimate

Classification	Tonnes (Mmt)	Au (g/mt)	Contained Au (Koz)
Measured	497.34	0.68	10,840.84
Indicated	28.04	0.69	620.33
Total M & I	**525.38**	**0.68**	**11,461.17**
Inferred	52.80	0.66	1,127.21

Notes:

- The Independent and Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Scott Wilson, CPG.
- The effective date of the Estimate is August 26, 2016.
- Mineral Resources are inclusive of Mineral Reserves.
- These mineral resources are not mineral reserves as they do not have demonstrated economic viability.
- The reported mineral resources are considered to have reasonable prospects for economic extraction.
- Ounce (troy) = metric tonnes x grade / 31.10348. Calculations used metric units (meters, tonnes and g/t). Metal contents are presented in thousands of ounces (Koz).
- The number of metric tonnes (Mmt) was rounded. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.

14.13 Grade sensitivity analysis

The mineral resource is highly sensitive to cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates are presented at various cut-off grades ranging from 0.3 to 1.0 g/mt Au (Table 14-12). The reader is cautioned that the table should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. This cut-off grade sensitivity is also illustrated as grade tonnage curves (Figure 14-4).

Table 14-12: Sensitivity of block model to cut-off grade

Cut-off Au g/mt	Measured			Indicated			Measured & Indicated			Inferred		
	Tonnes (000)	Grade Au g/mt	Au oz (000)	Tonnes (000)	Grade Au g/mt	Au oz (000)	Tonnes (000)	Grade Au g/mt	Au oz (000)	Tonnes (000)	Grade Au g/mt	Au oz (000)
0.3	1,049,884	0.61	20,590	108,539	0.59	2,059	1,158,423	0.61	22,649	451,452	0.54	24,707
0.4	938,674	0.64	19,314	95,293	0.62	1,899	1,033,967	0.64	21,214	376,255	0.58	23,113
0.5	681,137	0.71	15,548	65,909	0.69	1,462	747,046	0.71	17,010	211,587	0.68	18,472
0.6	431,071	0.81	11,226	39,267	0.79	997	470,338	0.81	12,223	117,551	0.79	13,220
0.7	266,753	0.91	7,804	23,714	0.89	679	290,467	0.91	8,483	70,142	0.90	9,161
0.8	170,732	1.00	5,489	14,340	0.98	452	185,072	1.00	5,941	43,062	0.99	6,393
0.9	108,375	1.10	3,833	8,582	1.08	298	116,957	1.10	4,131	26,150	1.09	4,429
1.0	68,609	1.18	2,603	5,171	1.17	195	73,780	1.18	2,797	15,828	1.18	2,992

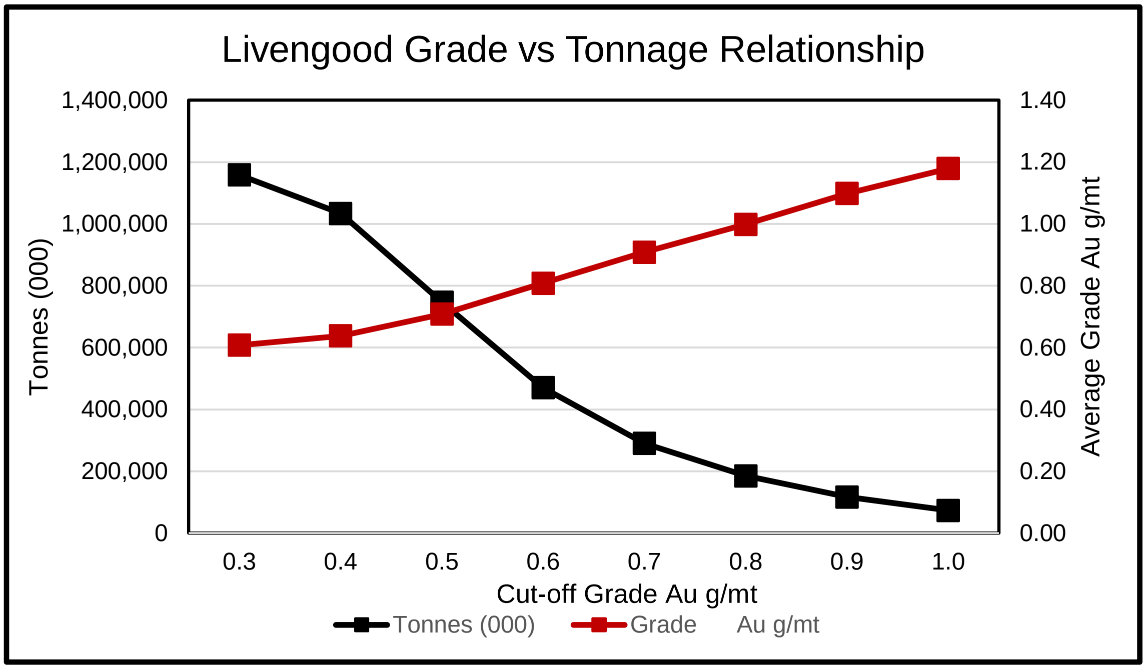


Figure 14-4: Livengood grade vs tonnage relationship

14.14 Sensitivity of mineralization to gold price

The sensitivity of mineralization defined by the evaluation of the mineralization inventory at different gold prices was performed for gold prices of $984/oz (-20%), $1,230/oz (resource base case) and $1,476/oz (+20%). The input parameters defined in Table 14-10 above were used in the analysis. Table 14-13 lists the amount of the mineralization contained within the pit shells that could be scheduled to process.

Table 14-13: Sensitivity of mineralization inventory contained in pit shells defined by Whittle™ Analyses at different gold prices within pit shells

Whittle™ Pit Gold Price	Classification	Tonnes (Mmt)	Au (g/t)	Contained Au (Koz)
$984	Measured	322.75	0.79	8,166.23
	Indicated	15.06	0.83	399.36
	Total M & I	337.81	0.79	8,565.59
	Inferred	19.77	0.79	504.68
$1,230	Measured	497.34	0.68	10,840.84
	Indicated	28.04	0.69	620.33
	Total M & I	525.38	0.68	11,461.17
	Inferred	52.80	0.66	1,127.21
$1,476	Measured	663.11	0.61	13,004.65
	Indicated	46.76	0.60	899.03
	Total M & I	709.88	0.61	13,903.68
	Inferred	115.01	0.56	2,070.62

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 beyond those parameters that are integral to MIK block modelling techniques. 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.

15. MINERAL RESERVE ESTIMATE

15.1 Introduction

The mine design and mineral reserve estimates have been completed to a level appropriate for pre-feasibility studies. The mineral reserve estimate stated herein is consistent with the CIM Standards on Mineral Resources and Mineral Reserves and is suitable for public reporting. As such, the mineral reserves are based on measured and indicated resources, and do not include any inferred resources.

15.2 Pit optimization

The process of pit optimization utilized Gemcom Whittle™ software to determine pit extents and phasing. The Project consists of a block model encompassing the mining area. The model used for pit optimization and to produce reserves based on the mine plan is the same 15 × 15 × 10 meter MIK estimated model used to report the resource as detailed in Chapter 14 of this report. The Livengood block model is an SMU model, which accounts for the smallest unit that can be mined and accounts for selectivity and dilution in the model. It was assumed that the resources converted to reserves were based on a recoverable resource model. Therefore, additional operational dilution beyond what is integral to the MIK model was not included in the estimation of proven and probable reserves.

A nested pit analysis was completed to find the optimal pit based on gold price. Three daily throughput rate scenarios for the mill were analyzed with the optimized pit to determine updated mining operating costs, which in turn where used to generate a new ultimate pit with phased internal pits to be used for scheduling. The resulting pit shells were used to produce mine designs and production schedules for economic analysis.

15.2.1 Nested pit analysis

The process of pit optimization utilized Whittle™ to determine pit extents. The block model was manipulated in Maptek Vulcan™ software and then loaded into Whittle™. Analysis of nested pit shells was performed to determine the optimal pit shell for scheduling. The values of the nested pits were calculated by comparing the cost per tonne to mine, the cost per tonne to process and recoverable ounces of gold at a price of $1,250/oz of gold. The nested pit based on a gold price per ounce of $1,000 indicated a high net present value (NPV) and high recoverable ounces, before production costs at higher tonnages started to show diminishing NPV values. The higher gold ounces in the economic pit will help to create a favorable operating cost per ounce for the Project. Table 15-1 shows the results of pit optimization for gold prices between $500 and $1,500/oz of gold; the $1,000 pit is highlighted. Figure 15-1 shows the nested pit results in graphical form.

Table 15-1: Nested pit optimization results

Nested Pit ($ per Au oz)	Mineralized Material (Kmt)	Au Grade (gpt)	Au Contained (kg)	Waste Material (Kmt)	Au Recoverable (kg)	Cash Flow ($ x 1000)	Cash Flow NPC at 5% ($ x 1000)
$500	1,906	0.952	1,814	893	1,440	$32,643	$32,421
$550	3,843	0.882	3,390	1,745	2,671	$56,782	$55,994
$600	18,759	0.747	14,012	5,460	11,367	$216,420	$204,613
$650	55,132	0.744	41,025	23,288	32,138	$572,995	$499,846
$700	83,330	0.740	61,673	47,566	48,250	$831,767	$681,818
$750	136,674	0.756	103,377	117,257	79,374	$1,301,213	$947,257
$800	183,396	0.730	133,797	155,591	102,981	$1,609,592	$1,060,806
$850	254,722	0.720	183,497	241,988	139,468	$2,047,426	$1,166,069
$900	337,204	0.712	240,150	350,012	180,545	$2,497,644	$1,203,891
$950	397,651	0.708	281,682	448,551	210,494	$2,789,994	$1,218,792
$1,000	419,749	0.704	295,537	477,514	220,424	$2,868,823	$1,210,275
$1,050	448,206	0.696	311,725	517,654	232,658	$2,943,596	$1,169,522
$1,100	463,749	0.692	320,851	540,022	239,186	$2,977,312	$1,155,354
$1,150	486,170	0.685	332,990	576,431	248,363	$3,009,618	$1,119,390
$1,200	508,819	0.679	345,261	623,972	257,856	$3,031,699	$1,065,838
$1,250	525,445	0.674	354,222	657,129	264,464	$3,037,455	$1,033,825
$1,300	538,727	0.671	361,290	687,768	269,701	$3,032,926	$1,006,482
$1,350	558,283	0.665	371,451	732,993	277,129	$3,015,237	$970,092
$1,400	576,667	0.663	382,119	791,690	284,427	$2,983,604	$938,077
$1,450	591,423	0.661	390,726	841,441	290,188	$2,949,255	$912,974
$1,500	600,987	0.659	395,980	876,073	293,861	$2,920,422	$894,119

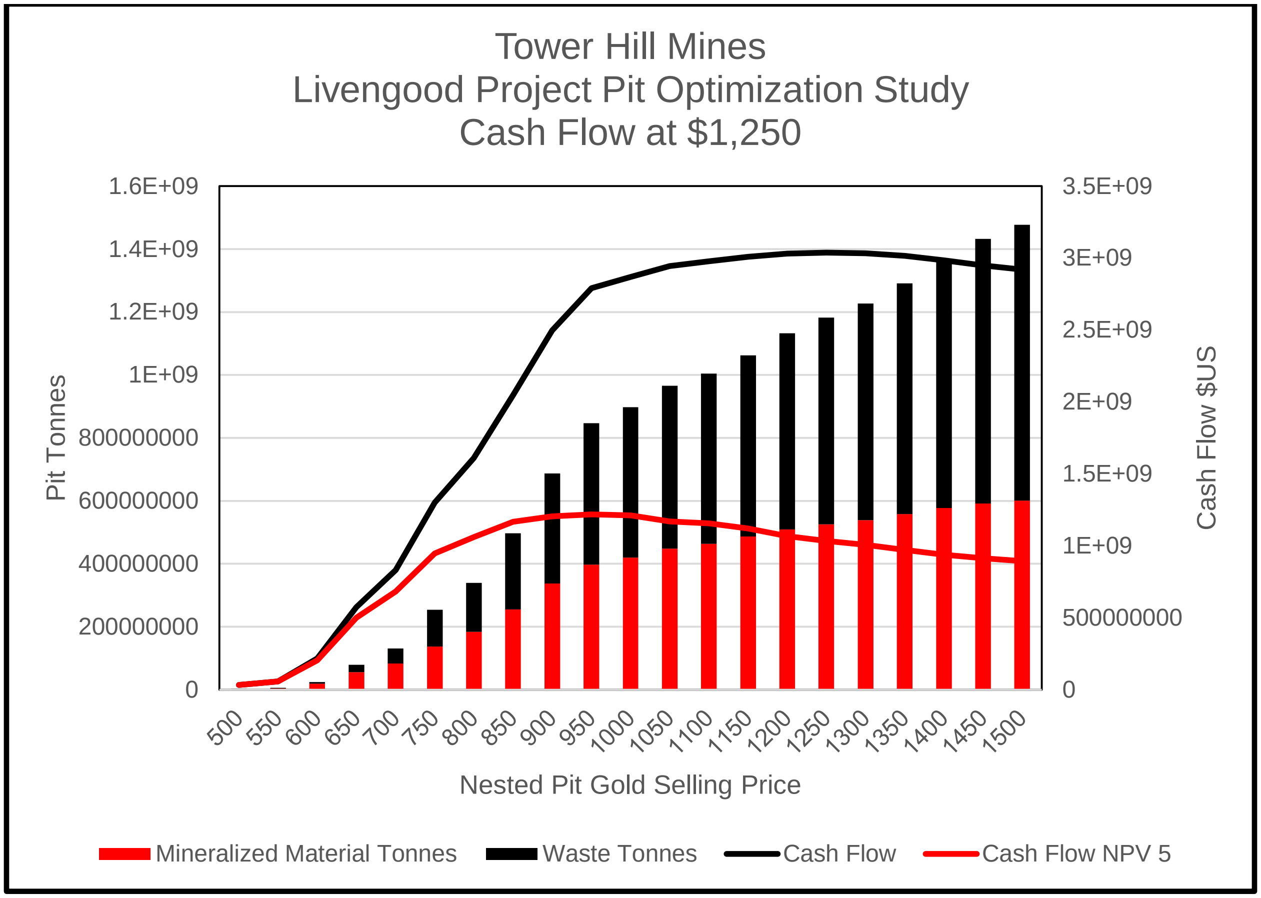


Figure 15-1: Nested pit optimization results

15.2.2 Throughput analysis

Three production schedules were prepared for economic analysis by varying daily mill throughput production rates. The three throughput scenarios were 35,500 mt/d, 47,700 mt/d, and 71,000 mt/d. Production schedules based on the $1,000/oz ultimate pit for the three cases were run in Whittle™ to optimize the mining schedules. The schedules were optimized by taking into account preproduction waste requirements for infrastructure earthwork for mine development, in addition to the production from pit mineralized material and a low grade mineralized material stockpile, to meet the mill throughput requirements. Tonnes mined per annum reflect mining ramp-up in preproduction and a ramp-down to a favorable rate for production years. For each scenario, NPV was calculated at 5% based on the Whittle™ schedules, calculated revenue from $1,250 per gold ounce and the mining and processing operating costs. Table 15-2 shows the preproduction waste requirements. Table 15-3 through 15-5 present the summary schedules and the NPV for each throughput scenario.

Table 15-2: Preproduction waste requirements

Throughput Scenario	Waste Tonnes
35,500 mt/d	70 million
47,700 mt/d	79 million
71,000 mt/d	95 million

Table 15-3: Summary production schedule for 35,500 mt/d scenario

Production	Preproduction	Mill and Stockpile	Stockpile Reclaim	Total
Years	**-2 to -1**	**1 to 17**	**18 to 28**	**28**
Mineralized Material to Mill (Kmt)	-	201,399	-	**201,399**
Mineralized Material to Stockpile (Kmt)	23,171	130,214	-	**153,385**
Stockpile to Mill (Kmt)	-	15,574	137,811	**153,385**
Waste (Kmt)	69,829	380,169	-	**449,998**
Strip Ratio	-	1.89	-	**2.23**
Au Grade (gpt)	-	0.90	0.48	**0.73**
Contained Au (Koz)	-	6,246	2,105	**8,351**
Au Recovery (%)	-	78%	80%	**78%**
Total Recovered Au (Koz)	-	4,858	1,672	**6,530**
NPV at 5% ($ x 1000)				**$1,181,110**

Table 15-4: Summary production schedule for 47,700 mt/d scenario

Production	Preproduction	Mill and Stockpile	Stockpile Reclaim	Total
Years	**-2 to -1**	**1 to 17**	**18 to 24**	**24**
Mineralized Material to Mill (Kmt)	-	250,611	-	**250,611**
Mineralized Material to Stockpile (Kmt)	26,122	121,923	-	**148,045**
Stockpile to Mill (Kmt)	-	41,786	106,259	**148,045**
Waste (Kmt)	78,878	430,117	-	**508,994**
Strip Ratio	-	1.72	-	**2.03**
Au Grade (gpt)	-	0.82	0.47	**0.72**
Contained Au (Koz)	-	7,689	1,597	**9,285**
Au Recovery (%)	-	76%	75%	**76%**
Total Recovered Au (Koz)	-	5,769	1,196	**6,965**
NPV at 5% ($ x 1000)				**$1,540,852**

Table 15-5: Summary production schedule for 71,000 mt/d scenario

Production	Preproduction	Mill and Stockpile	Stockpile Reclaim	Total
Years	**-2 to -1**	**1 to 12**	**13 to 15**	**15**
Mineralized Material to Mill (Kmt)	-	239,339	-	**239,339**
Mineralized Material to Stockpile (Kmt)	36,509	97,364	-	**133,872**
Stockpile to Mill (Kmt)	-	66,602	67,270	**133,872**
Waste (Kmt)	93,486	375,611	-	**469,097**
Strip Ratio	-	1.57	-	**1.96**
Au Grade (gpt)	-	0.78	0.47	**0.73**
Contained Au (Koz)	-	7,714	1,010	**8,724**
Au Recovery	-	79%	78%	**78%**
Total Recovered Au (Koz)	-	6,026	781	**6,808**
NPV at 5% ($ x 1000)				**$1,781,706**

The 52,600 t/d (47,700 mt/d) throughput was selected for economic assessment and mine planning, moving forward with the Livengood Gold Project, based on the recoverable ounces produced, the NPV value and lower capital costs. There are also operational and capital processing benefits identified that were considered, but were out of the scope of the pit optimization analysis.

15.2.3 Pit optimization and phasing

The Livengood Gold Project is planned to be mined by surface mining methods with the mineralized material crushed and processed by a plant designed for 52,600 t/d (47,700 mt/d), 365 d/y. The mine must supply 19.2 Mt (17.4 Mmt) of mineralized material to the plant annually. The production is planned to be lower during the first year of operation before full capacity is achieved. Preproduction waste rock requirements of 87.1 Mt (79 Mmt) are required to be mined the two years before mill start-up. Total material movement is capped at 60.6 Mt/y (55 Mmt/y). Mining is planned on 32.8 ft (10 m) bench intervals with pit optimization slopes set at 42° to accommodate ramps and other design criteria. The only restriction on phasing was the removal of the waste and low grade stockpiling was forced to take place the first two years. This was done to account for the preproduction period and the material removal from that period so that it would not be held against the production phases tonnages. Utilizing the costs for the selected mill throughput of 52,600 t/d (47,700 mt/d) and associated production requirements, a new optimization was generated to determine the new pit extents and phasing. This optimization was run with the parameters listed in Table 15-6 and Table 15-7.

Table 15-6: Input parameters for the Whittle™ analysis

Parameter	Value
Mining Cost ($/mt)	$1.77
Stockpile Reclaim Cost ($/mt)	$0.82
RT4 Processing Cost ($/mt)	$11.31
RT5 Processing Cost ($/mt)	$11.87
RT6 Processing Cost ($/mt)	$11.76
RT7 Processing Cost ($/mt)	$11.56
RT8 Processing Cost ($/mt)	$12.24
RT9 Processing Cost ($/mt)	$12.24
Gold Price ($/oz)	$1,250
Pit Slopes (degrees)	42

Table 15-7: Process recoveries and cut-off grades by rock type

Rock Type	Au Recovery	Cut-off Grade (gpt)
4	84.4%	0.347
5	86.5%	0.335
6	78.3%	0.366
7 (1)	64.0%	0.441
8	75.4%	0.443
9	75.4%	0.443

(1) Recovery for rock type 7 should be calculated Rec=80.78*QSJ+66.86,
QSJ=% quartz stibnite + % jamesonite mineralization per block,
64% recovery hard coded for cut-off calculations.

The new ultimate pit was produced with phased internal pits shells. The resulting pit shells were used as a basis for the designed pits used to generate production schedules for economic analysis. The final optimized pit is shown in Figure 15-2. The designed ultimate pit based on the optimized pit is shown in Figure 15-3.

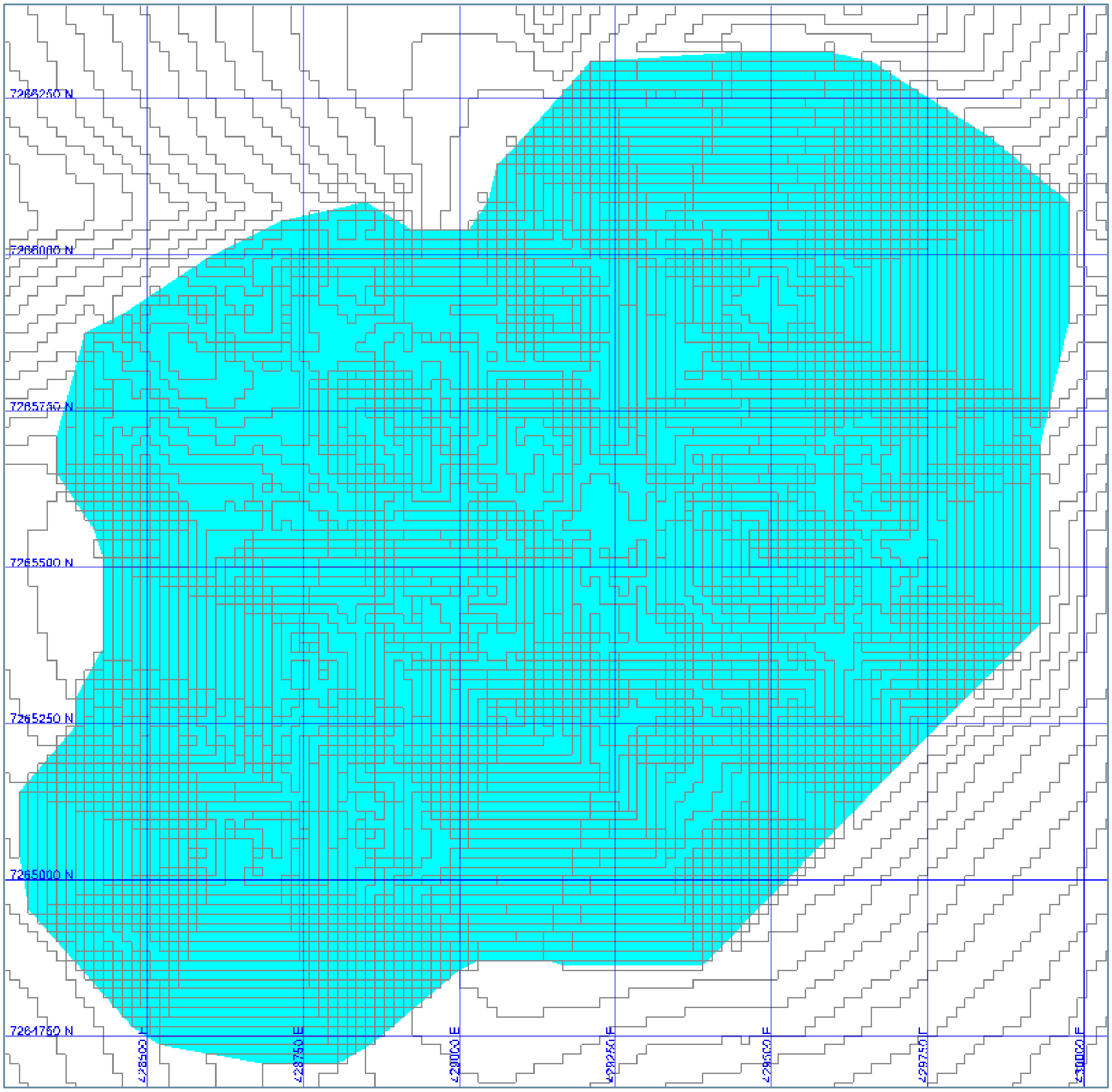


Figure 15-2: Optimized final pit

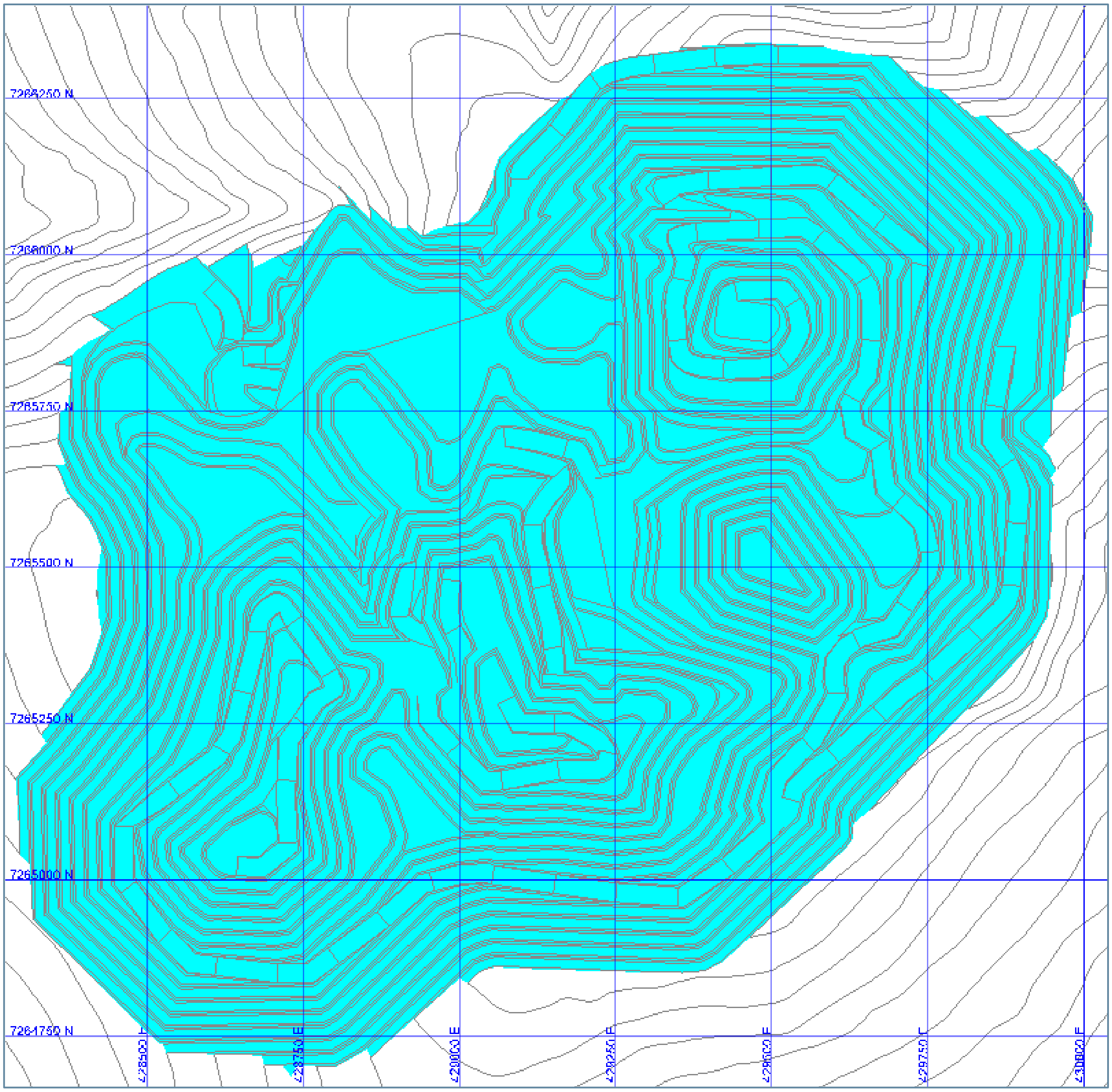


Figure 15-3: Designed ultimate pit

15.3 Summary of reserves from detailed mine design

Mineral reserves are confined to pit designs meeting geotechnical constraints. Additionally mineral reserves are reported at the above mentioned cut-off grades. Proven reserves are identified as Measured Mineral Reserves contained within the pit shapes, above cut-off grades. Probable reserves are identified as Indicated Mineral Resources contained within pit designs, above cut-off grades. Mining methods and pit designs are detailed in Chapter 16 of this report. The Proven and Probable reserves, which are contained in the ultimate pit, are summarized in Table 15-8 and match the production schedule.

Table 15-8: Livengood reserves from designed ultimate pit

Class	Mineralized Material Kmt	Au Grade gpt	Contained Au Koz
Proven			
Rock Type 4	56,373	0.65	1,178
Rock Type 5	88,282	0.61	1,733
Rock Type 6	88,712	0.69	1,974
Rock Type 7	48,102	0.79	1,214
Rock Type 8	4,807	0.73	112
Rock Type 9	90,211	0.83	2,405
Total Proven	**377,650**	**0.71**	**8,620**
Probable			
Rock Type 4	5,133	0.73	120
Rock Type 5	728	0.58	14
Rock Type 6	2,321	0.65	49
Rock Type 7	2,426	0.73	57
Rock Type 8	1,426	0.70	32
Rock Type 9	3,142	0.84	85
Total Probable	**14,010**	**0.72**	**353**
Proven and Probable Totals [1]	**391,660**	**0.71**	**8,973**

[1] Numbers may not add up due to rounding.

16. MINING METHODS

16.1 Introduction

The Livengood Gold Project is a conventional surface mine that will utilize large-scale mining equipment for standard open pit mining of a blast/load/haul operation. The mineralized material will be crushed and processed by a gravity-whole ore carbon in leach (CIL) plant. Mining was scheduled to provide a mill feed of 52,600 t/d (47,700 mt/d). Preproduction stripping of 86.7 Mt (78.6 Mmt) of waste rock material is required for the construction of process facilities and site infrastructure. Mineralized material mined during preproduction will be stockpiled for mill feed later in the production period, after mill start-up.

Multiple pit optimizations and production schedules were produced for different mining and processing scenarios to develop the best operating scenario with the greatest value. Pits were optimized using varying gold prices to determine the optimal pit for scheduling based on net present value (NPV). After the metal price was chosen, pits and schedules were produced based on varying mill throughput scenarios. The selected mill throughput scenario was used to generate mining operating costs to generate a new ultimate pit with phased internal pits to be used for scheduling. The phased pits were used for subsequent mine designs, production scheduling, and economic analysis. Schedule iterations to maximize NPV and overall project economics included limiting of stockpiling, minimizing mining capital, increasing recoverable ounces and maximizing mill head grades. The final production schedule included a mixture of elements from the previous schedules with focus on mill head grade.

The optimized production schedule provides an operating life of 23 years. A two-year preproduction period is required for removal of waste rock material and site construction. Mine production will produce mill feed mineralized material for 16 years. During the mine production period, low grade mineralized material will be stockpiled, to be used for future mill feed, while waste rock material will be placed in a waste rock stockpile. Portions of the low grade, stockpiled, mineralized material will be sent to the mill during the mine production period, supplementing direct mine production tonnes to maintain constant mill throughput. After mining is complete, the mill will be fed from the remaining low grade stockpile for seven years. A summary production schedule is presented in Table 16-1.

Table 16-1: Summary production schedule

Production Period	Pre-production	Mill and Stockpile	Stockpile Reclaim	Total
Years	**-2 to -1**	**1 to 16**	**17 to 23**	**23**
Mineralized Material to Mill (Kt)	-	254,377	-	**254,377**
Mineralized Material to Stockpile (Kt)	29,085	148,267	-	**177,352**
Stockpile to Mill (Kt)	-	48,931	128,422	**177,352**
Waste rock (Kt)	86,658	468,168	-	**554,825**
Strip Ratio (overall LOM)	-	1.84	-	**1.3**
Au Grade (g/mt)	-	0.82	0.46	**0.71**
Contained Au Koz	-	7,245	1,727	**8,972**
Au Recovery	-	76%	74%	**75%**
Total Recovered Au Koz	-	5,515	1,249	**6,764**

16.2 Pit slope geotechnical evaluation

The following information summarizes the findings of the SRK report “Feasibility Pit Slope Evaluation, Livengood Project”, dated July 2013, as well as supplemental work completed later and described in a later SRK report “Geotechnical Optimization of Years 1 and 3 Pit Designs, Livengood Gold Project” dated April 19, 2016.

16.2.1 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, it was not possible to carry out geotechnical mapping 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 (6,775 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 in 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 to orient the core and observe the core in its most undisturbed state. The distribution of the 24 combined geotechnical drillholes used in the analysis is shown on Figure 16-1.

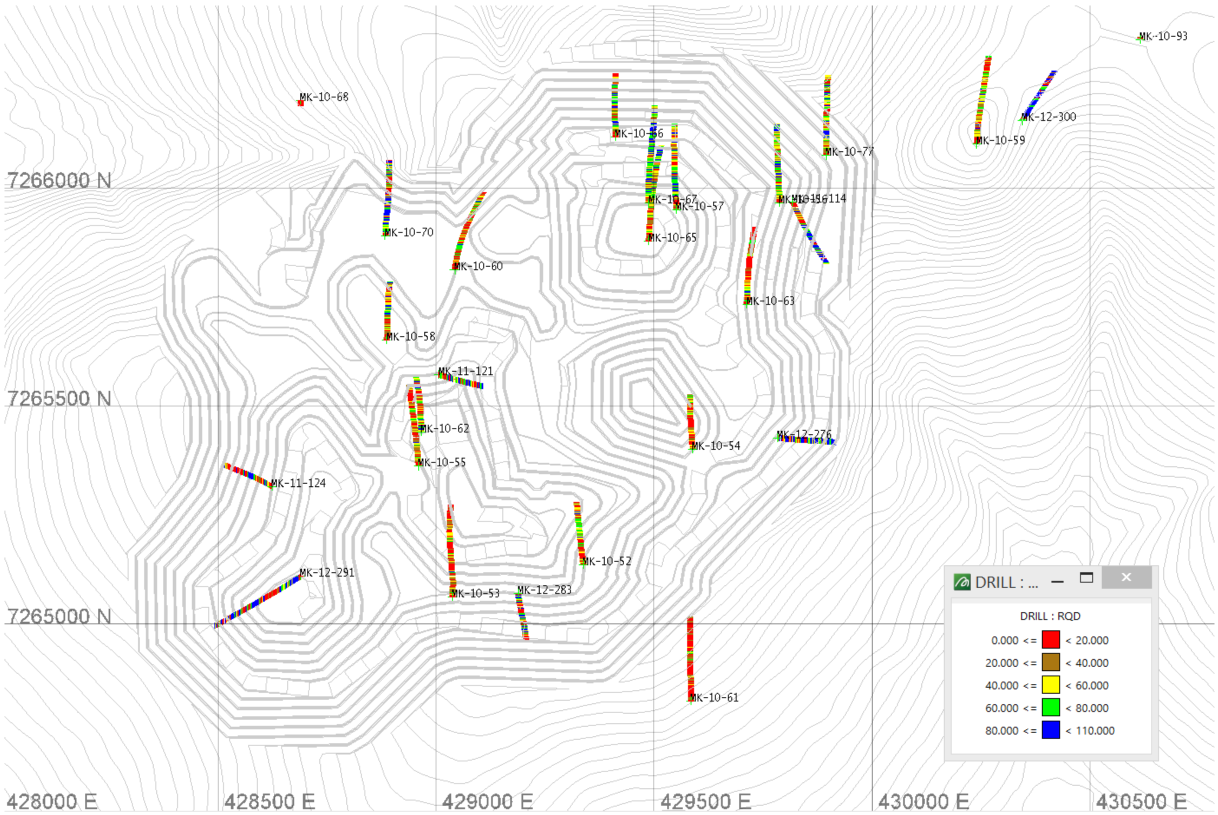


Figure 16-1: Location of drillholes used for geotechnical analysis

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 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.

16.2.2 Geotechnical model

The 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 16-2.

Table 16-2: Distributions of RMR (Bieniawski, 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 Sediments (Core Zone)	211	56	14
Lower Sediments	190	53	13
Upper Sediments (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. 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 a UCS value for each PLT. A correlation factor was developed for each individual 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 16-3 contains a statistical summary of the overall UCS data per engineering unit.

Table 16-3: 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 Sediments (Core Zone)	249	32	34
Lower Sediments	290	36	26
Upper Sediments (Sunshine)	808	59	42

16.2.3 Slope stability analyses

SRK evaluated both global and bench scale stability for the proposed 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 inter-ramp and/or overall slopes. Bench scale failures typically involve only one or two bench levels and can be described as block type displacements 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 the overall and high inter-ramp slopes of the ultimate pit. The critical sections were selected to represent the anticipated, most adverse stability conditions, such as where the slope height is at its maximum, pit wall materials are low strength and/or pore water pressures may be the highest. 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.

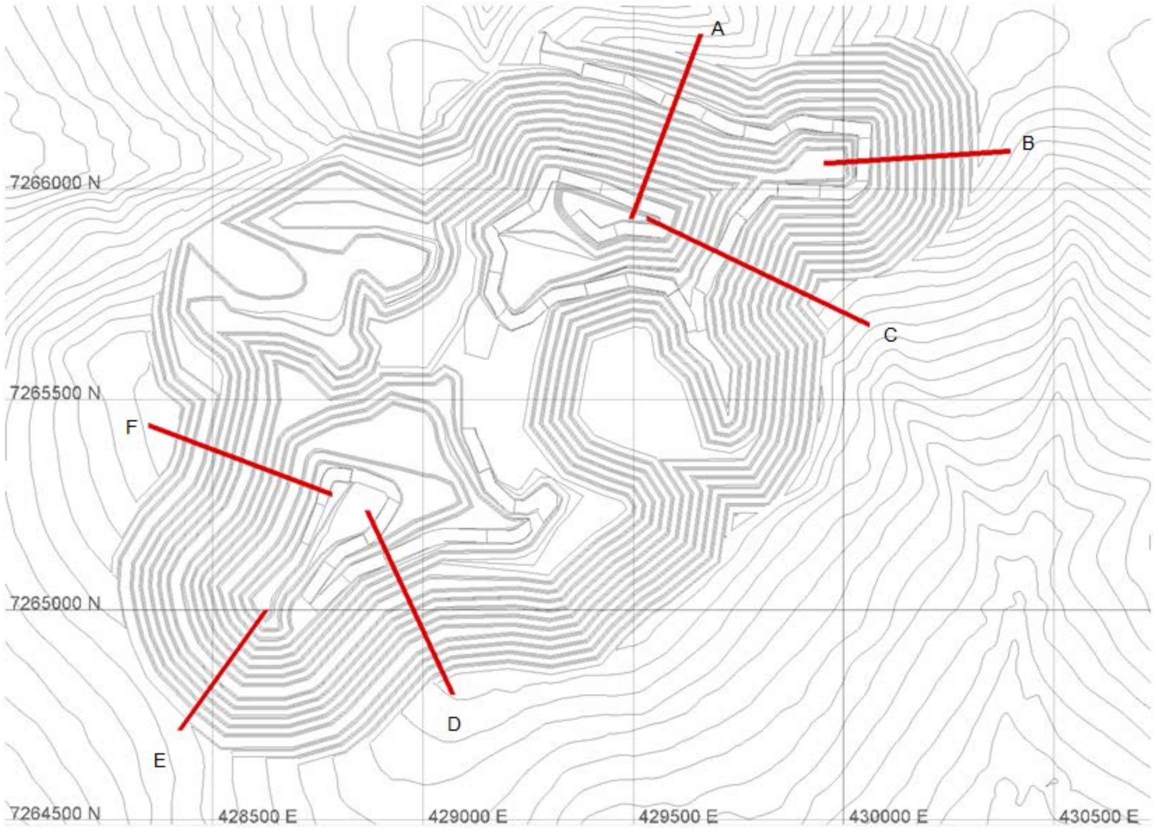


Figure 16-2: Critical slope stability sections for the (2013) ultimate pit

Based on accepted engineering experience, inter-ramp/overall slope designs, subject to probabilities of failure (POF) 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 acceptable for the non-critical slopes. Results of the analyses are summarized in Table 16-4. While Section C demonstrates 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 instability occur.

Table 16-4: Overall slope stability analysis results for the ultimate pit

Section	Probability of Failure	Mean Factor of Safety	Recommended Geotechnical Berm Elevations
A	3%	1.7	NA
B	3%	1.9	NA
C	22%	1.2	NA
D	8%	1.4	Elev. 220, 320
E	14%	1.3	Elev. 220
F	18%	1.3	Elev. 220, 320

Geotechnical berms were added as necessary to reduce the overall slope, and where necessary to achieve the acceptable design criteria. Geotechnical berms are defined as extra wide catch benches designed to break up high inter-ramp bench stacks and to provide a wide catchment, should an unexpected instability occur above. The geotechnical berms are designed at a total width of 82 ft (25 m).

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

16.2.4 Pit slope design recommendations

The final pit slope design recommendations for the ultimate pit are summarized in Table 16-5, with corresponding sectors shown on Figure 16-3.

Table 16-5: Pit slope design recommendations for the ultimate pit

Pit Sector	Max. Overall Slope Angle	Max. Inter-ramp Slope Angle	82 ft (25 m) Geotechnical Berms (Elev.)	Bench Height (ft)	[(1)] Bench Width (ft)	[(1)] Bench Face Angle
A	40	42	120, 220, 320	65.6	39.4/48.9	63/70
B	41	42	220, 320	65.6	39.4/48.9	63/70
Remaining Areas	42	42	N/A	65.6	39.4/48.9	63/70

[(1)] The 42° inter-ramp may be achieved by either 48.9 ft (14.9 m) width with 70° bench face angles or 39.4 ft (12 m) width with 63° bench face angles.

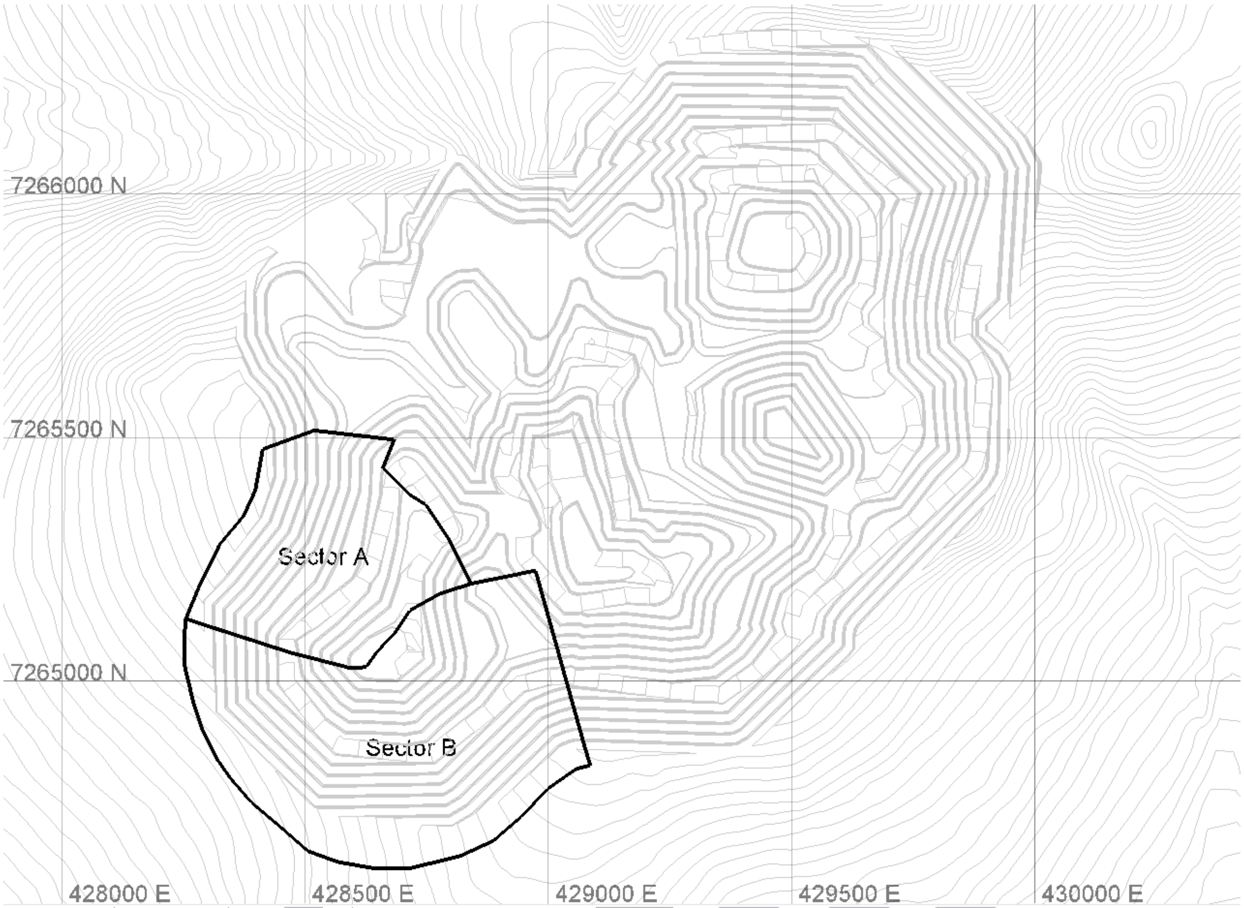


Figure 16-3: Pit slope design sectors for the ultimate pit

Recommendations are given for both 63º and 70º bench face angle configurations. The 63º 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 70º bench face angle may be more practical. Considering the relatively wide catch benches of 48.9 ft (14.9 m) that would be required to achieve the 42º inter-ramp angle, localized bench sloughing that may occur is expected to be retained by the catch bench beneath. Regardless of which bench configuration is selected, inter-ramp slope angles should not be increased over 42°.

16.2.5 Pit slope design recommendations for early mine phase interim pit walls

Subsequent to the FS, additional analyses were completed to optimize interim pit wall angles to minimize waste handling during the critical payback period (SRK, 2016). A total of six critical slope stability sections were analyzed for the interim slopes with the maximum inter-ramp slope heights ranging between 394 ft (120 m) and 525 ft (160 m).

Both deterministic and probabilistic limit equilibrium methods of analysis were used. For the deterministic analyses, the average properties were used to represent the rock mass strength inputs for each of the primary rock types that are summarized in Table 16-2 and Table 16-3. Groundwater surfaces were estimated for each model based on the SRK (2013) hydrogeologic field programs and numerical modeling. A minimum safety factor of 1.3 and a maximum probability of failure (POF) of 5% to 10% were selected as the acceptability criteria for the particular inter-ramp slopes analyzed, corresponding to high failure consequence slopes due to the proximity of the haul roads to the slopes. The minimum resulting safety factors range between 1.5 and 2.4, while the maximum POF for the sections analyzed range between 1% and 10% for the interim slopes analyzed.

The results indicate that the stability of lower, interim slope heights are anticipated to be controlled primarily by achievable bench face angles and to a lesser extent, the stability of high inter-ramp and overall slopes. Calculated safety factors could be considered relatively high for typical open pit slope designs. However, steepening of the inter-ramp slope angles beyond 47° would require either steeper bench face angles or a reduction in the design catch bench, which SRK does not recommend at the feasibility (PFS or FS level) level, due to the lack of rock exposure and actual geologic structural information. With detailed geotechnical bench face mapping and good quality wall control blasting practices, opportunity may exist to steepen the inter-ramp angles, based on more accurate information acquired during pit development.

Based on the (SRK, 2013) feasibility study geotechnical characterization and subsequent slope stability analyses (SRK, 2016) described above, SRK recommends that a maximum inter-ramp slope angle of 47° be used for interim inter-ramp slope heights of less than 525 ft (160 m).

16.3 Mine design

Using the phased pits generated by Whittle™ as guidance for pit shapes, designed pit phases were completed using Vulcan™, with design parameters provided by the geotechnical slope recommendations from SRK referred to in Section 16.2. All pits were designed with bench heights of 65.6 ft (20 m) and bench face angles of 70°. As the mining height is 32.8 ft (10 m) a bench, the mining of the pit walls will be double benched; catch benches were designed on alternating mining benches. Catch bench width varies depending upon the pit wall's relation to the ultimate pit. The inter-ramp slope angle (ISA) for the ultimate pit is 42°, requiring a catch bench width of 48.9 ft (14.9 m). Pit phases that are internal to the ultimate pit were designed with a steeper ISA of 47°, giving a 37.4 ft (11.4 m) catch bench width. These internal pits are mined earlier in the production sequence and the internal pit walls are mined out by subsequent phase pushbacks, until the final pit slope of the ultimate pit is reached. The internal pits are scheduled to be mined in a period of around three years to minimize the time exposed to the steepened slopes. Slope heights for the internal pits were also limited to 395 to 525 ft (120 to 160 m) maximum between haul roads, step outs, and areas of shallower slopes. The pit designs also accounted for haul road access to all mining areas and minimum practical mining widths, based on the specified mining equipment. Haul roads were designed at a grade of 10% with a width of 100 ft (30.5 m), which is 3.5 times the width of the selected 320 t (290 mt) haul trucks (28.5 ft or 8.7 m). The haul road network internal to the ultimate pit limits was designed to one pit exit location. Haulage destinations external to the pit limits follow the same route from the pit exit for the life of mine. This network was designed to minimize external pit haul road construction and associated costs. Table 16-6 presents the mine design parameters. Figure 16-4 through 16-12 show the designed pit phases.

Table 16-6: Mine design parameters

Parameter	Internal Pits	Ultimate Pit
Inter-ramp Slope Angle (degrees)	47	42
Bench Face Angle (degrees)	70	70
Bench Height (ft)	65.6	65.6
Catch Bench Width (ft)	37.4	48.9
Haul Road Width (ft)	100	100
Haul Road Grade (percent)	10%	10%

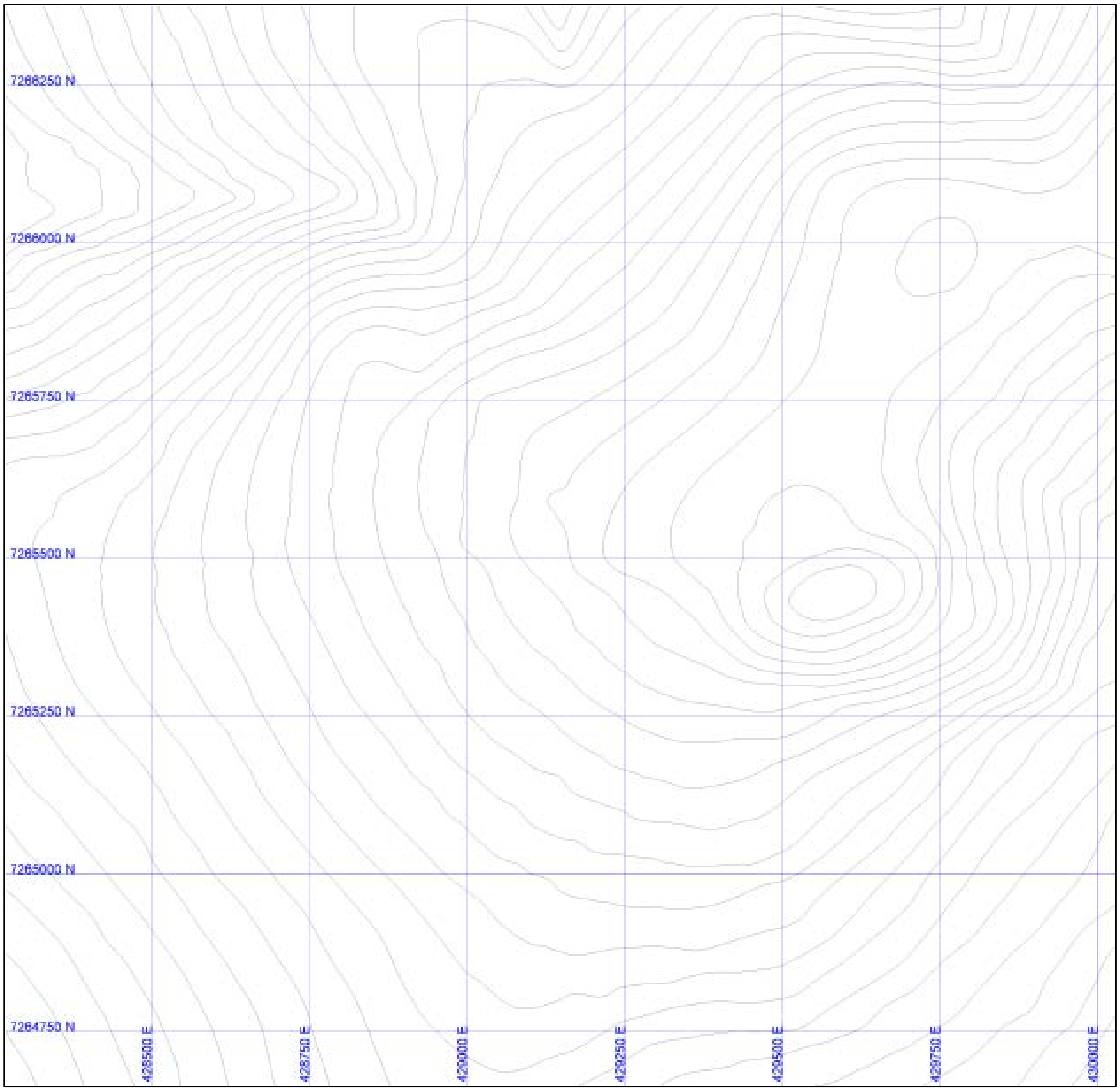


Figure 16-4: Pre mining surface

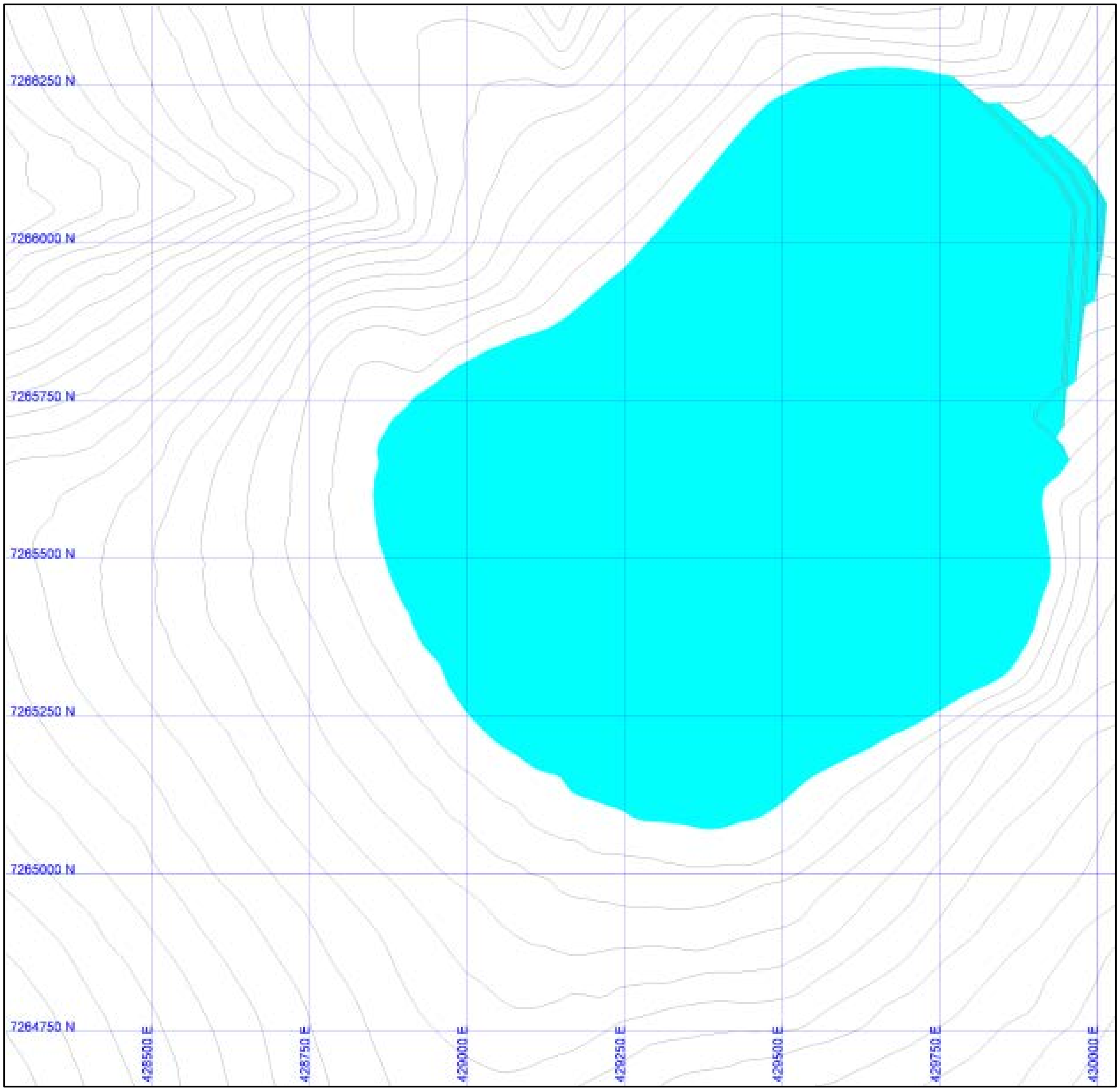


Figure 16-5: Phase 1 (preproduction)

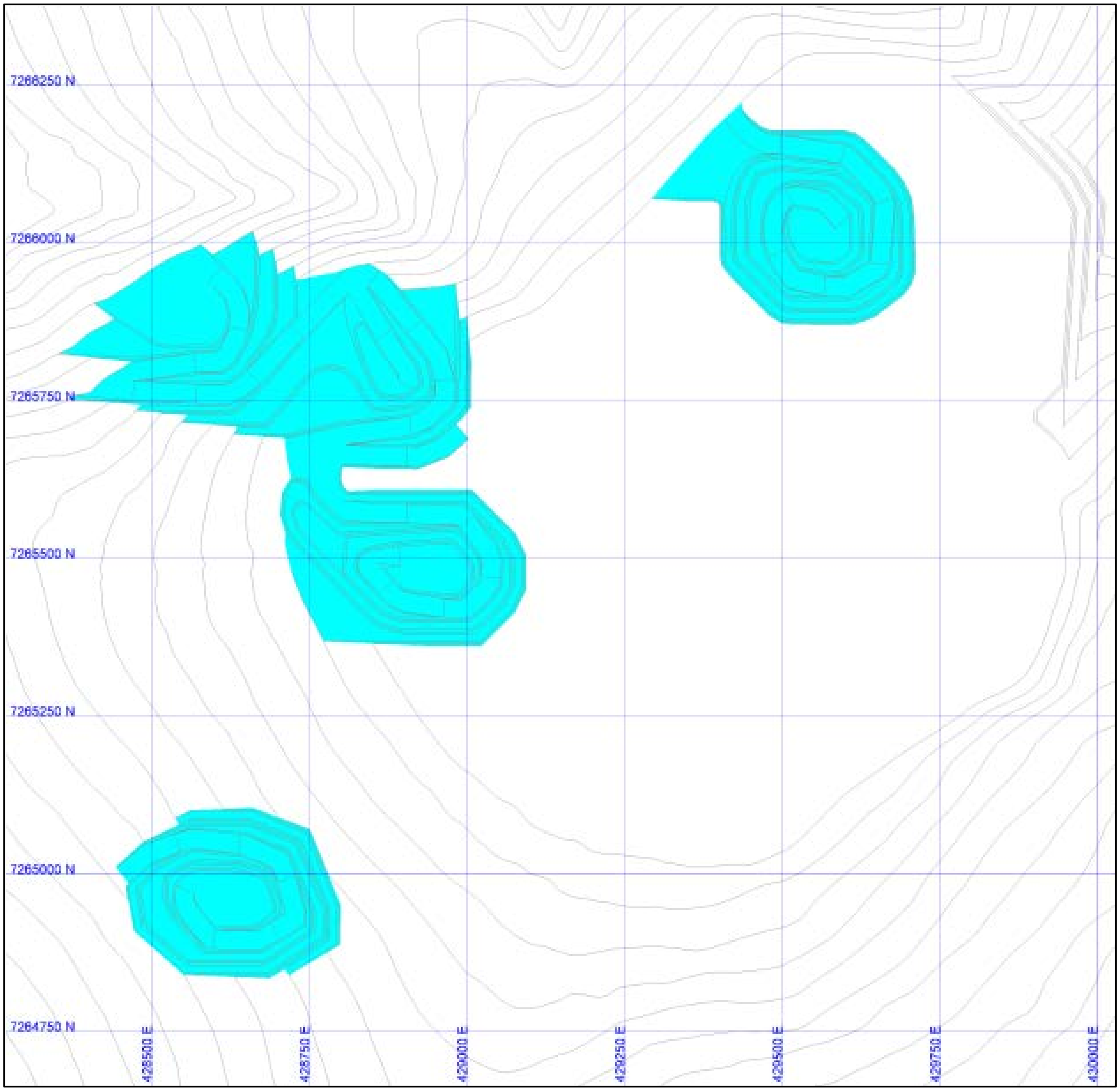


Figure 16-6: Phase 2 (start of production mining)

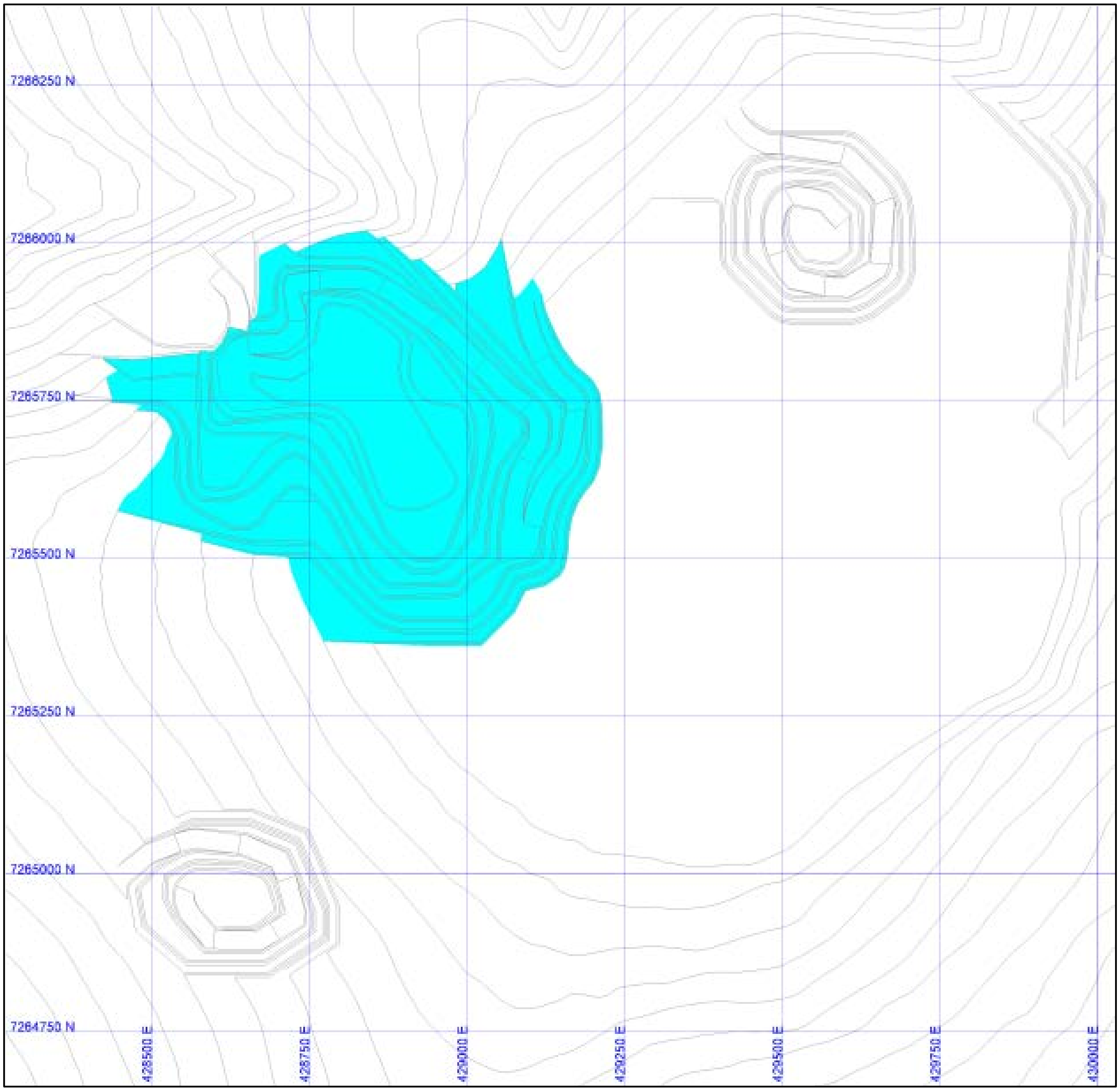


Figure 16-7: Phase 3

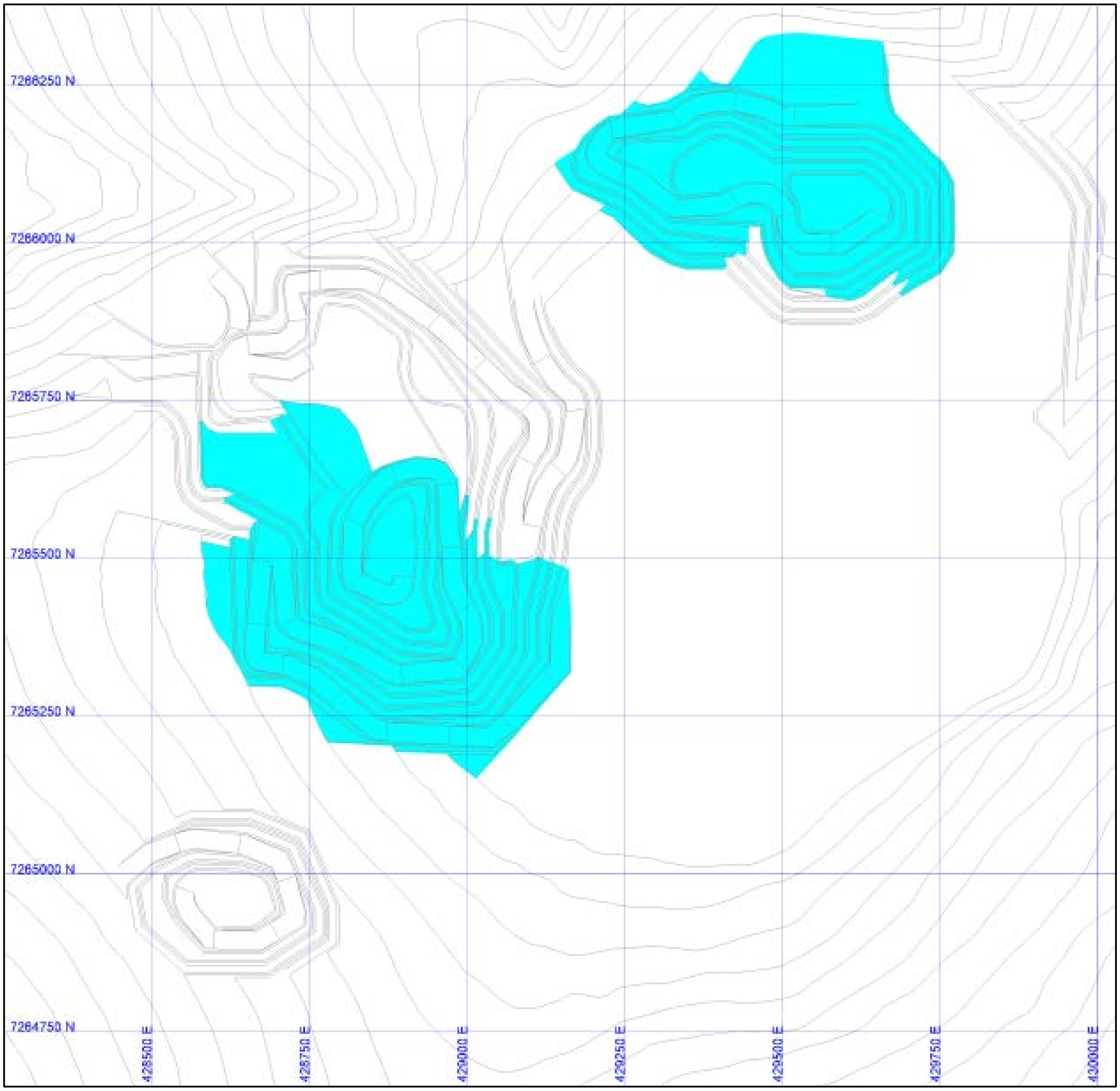


Figure 16-8: Phase 4

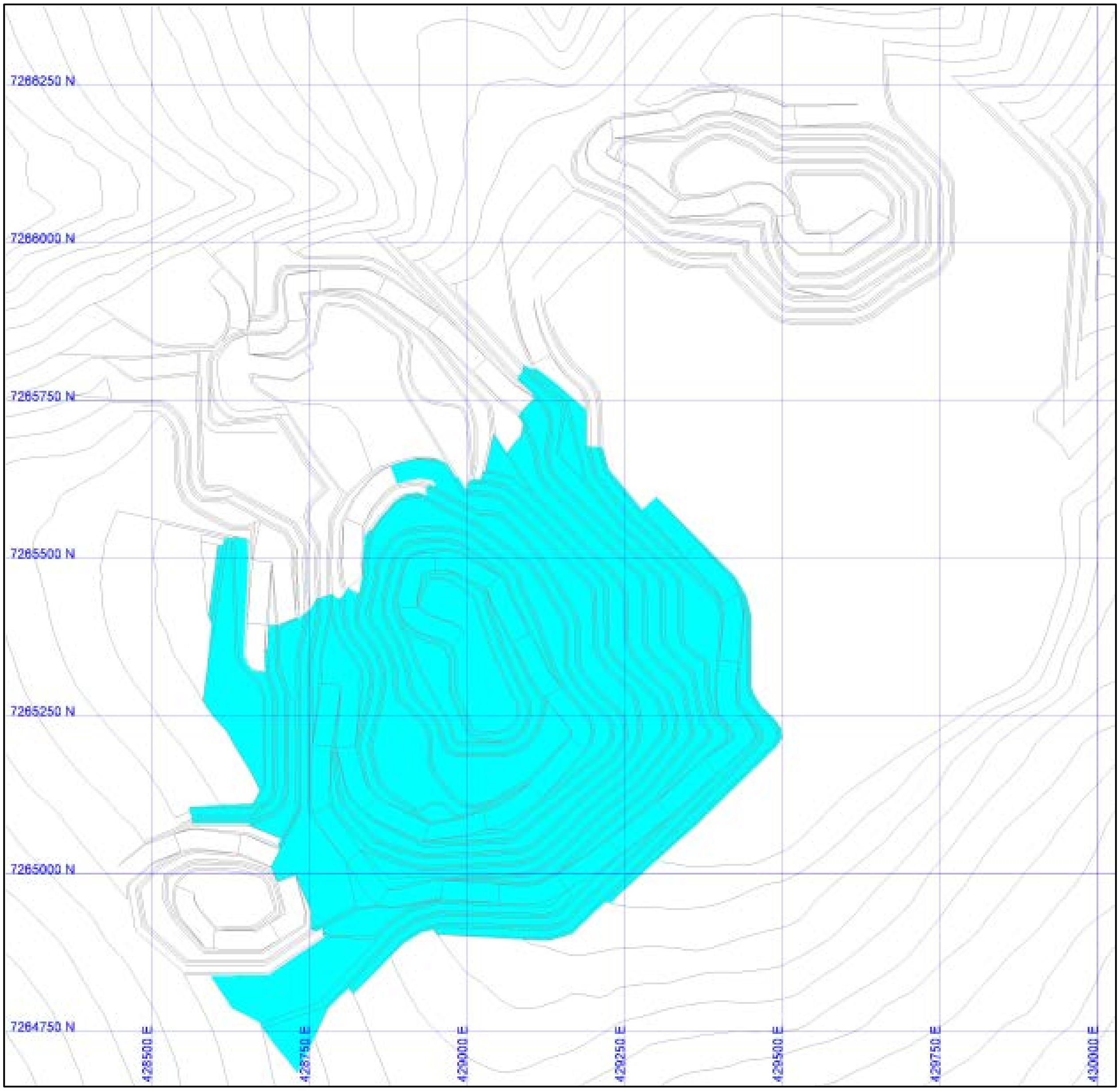


Figure 16-9: Phase 5

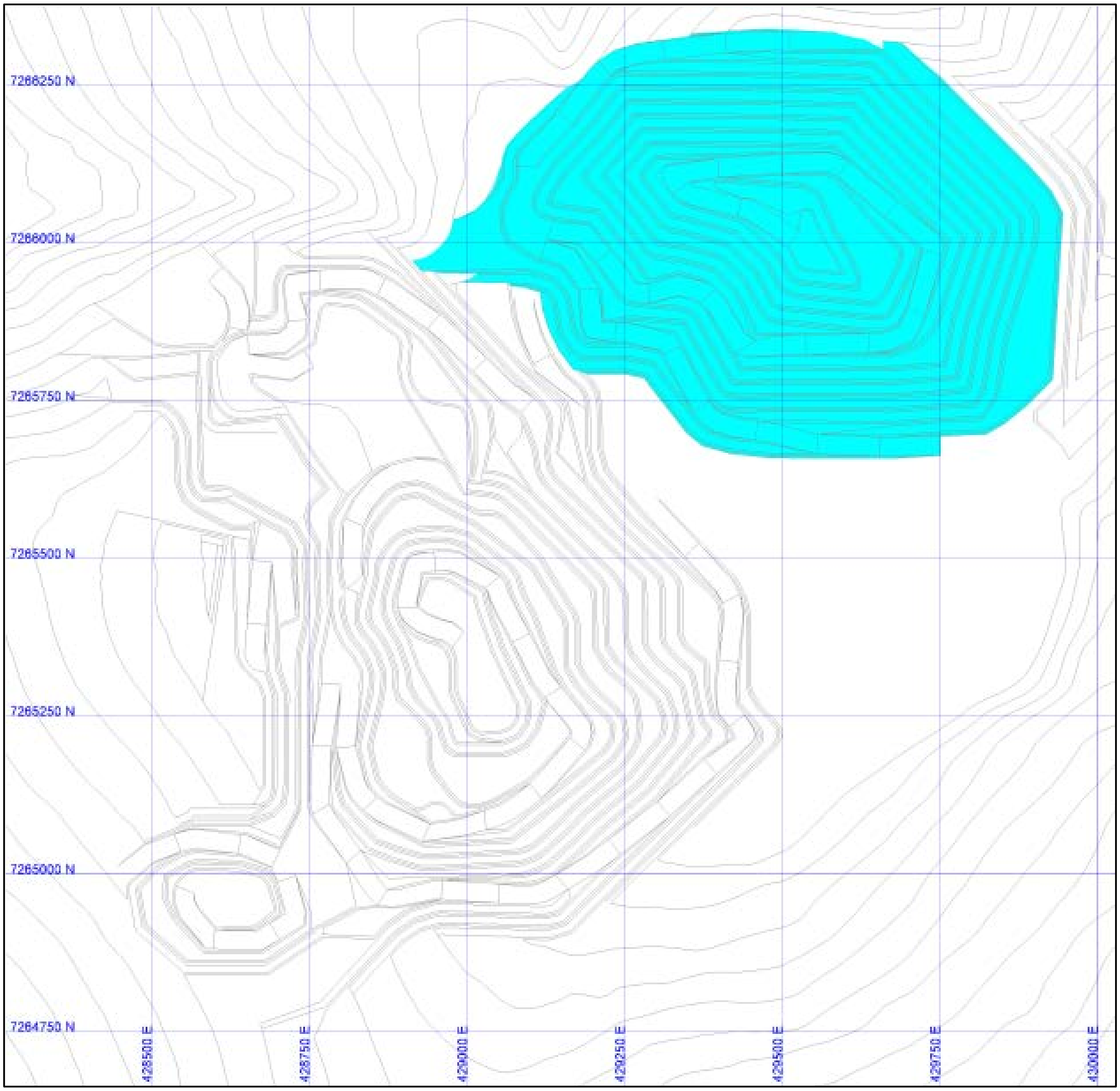


Figure 16-10: Phase 6

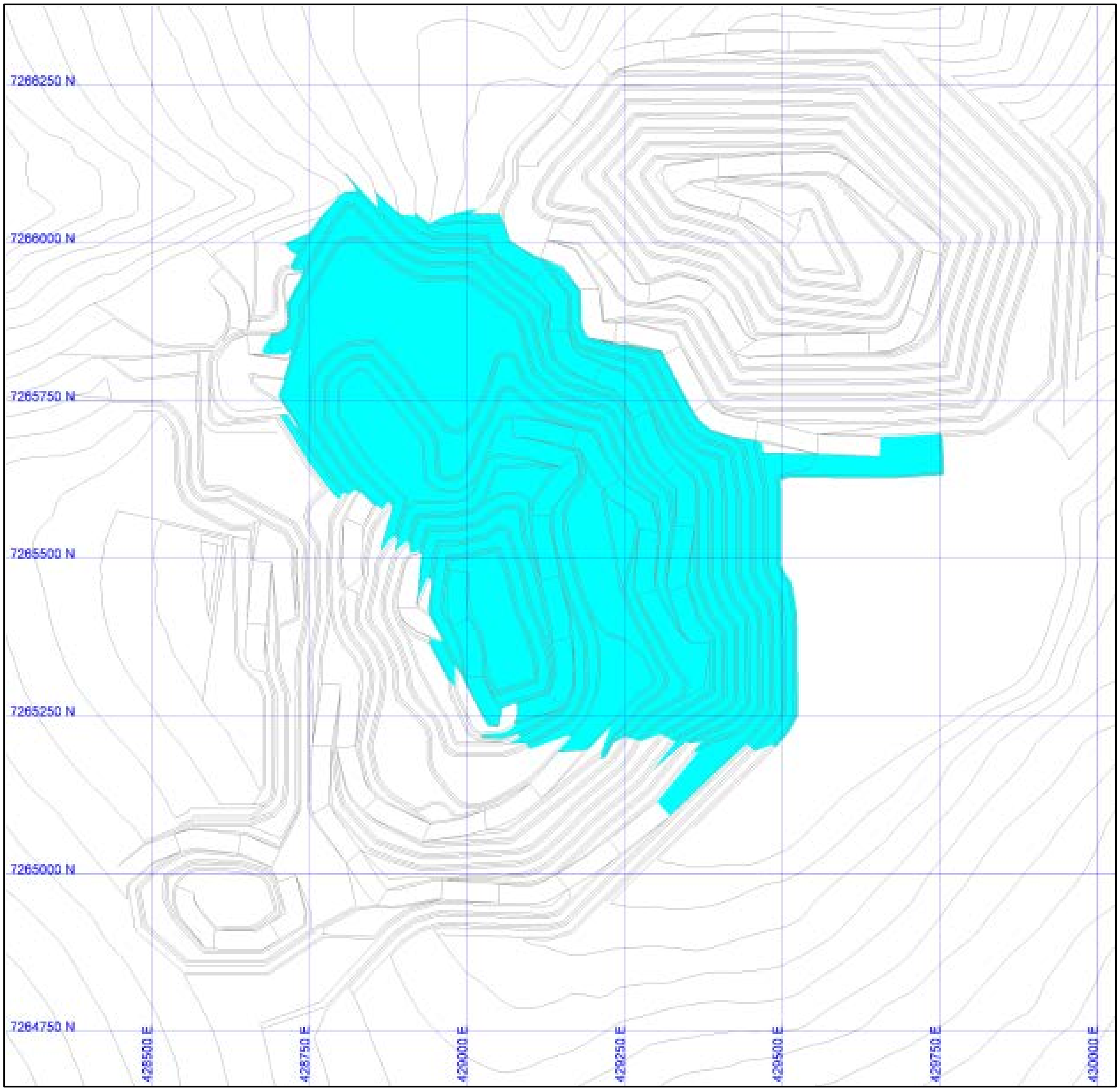


Figure 16-11: Phase 7

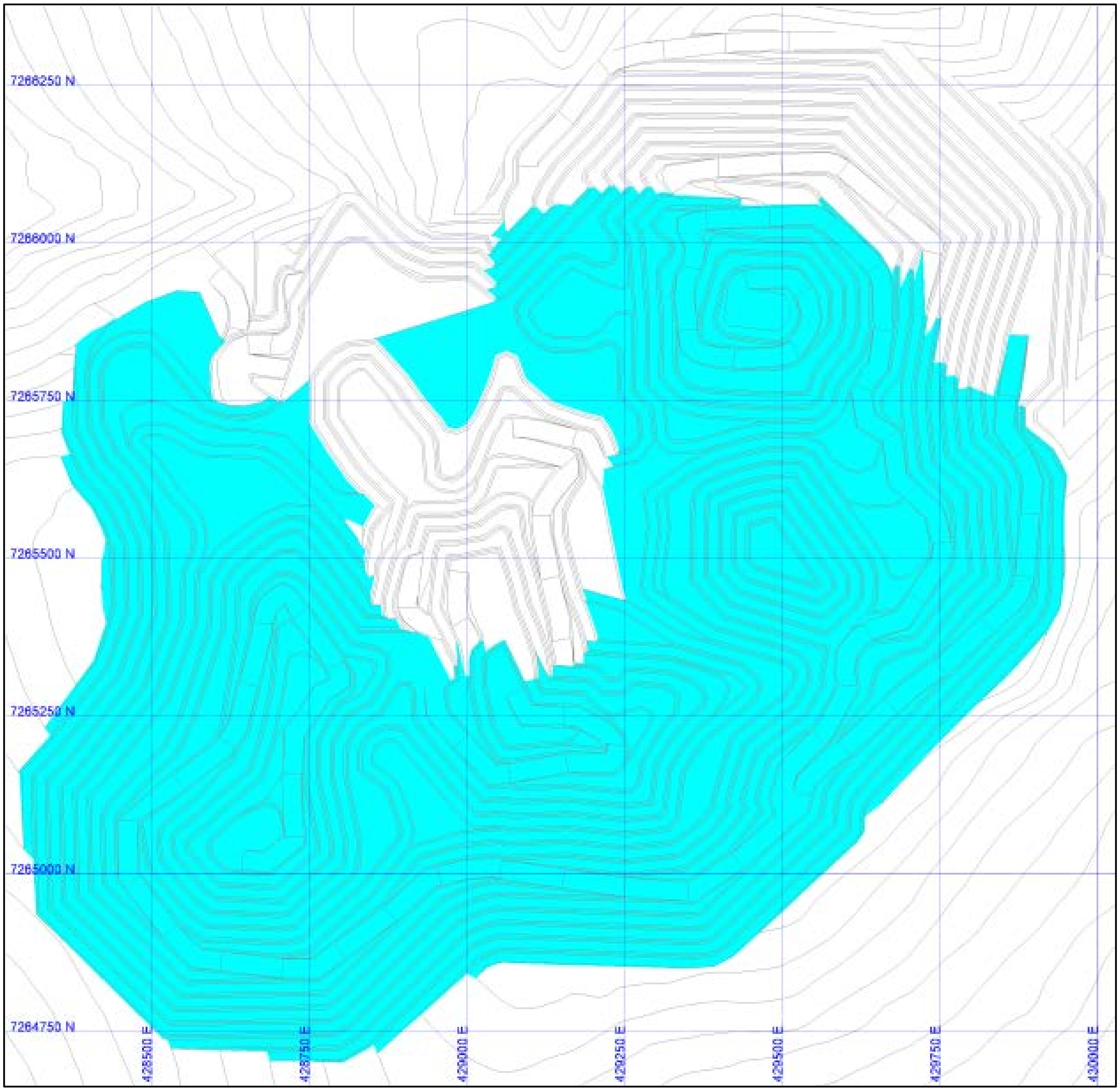


Figure 16-12: Phase 8 (ultimate pit)

16.4 Schedule

The mine production schedule was based on the designed pits scheduled on an annual basis in Whittle™. The schedule was optimized for stockpiling to ensure higher mill head grade and to account for preproduction waste rock requirements. Further constraint was applied to balance mining rates and minimize equipment requirements. Tonnages mined per annum reflect mining ramp-up in preproduction and a ramping down to an optimized rate for production years.

The mine will commence waste rock removal during the preproduction period of Years -2 and -1, waste rock stripping was scheduled to meet the 86.7 Mt (78.6 Mmt) required for infrastructure earthwork. Total tonnages mined were limited in Year -2 to reflect the ramp-up of tonnage as major mining equipment is being assembled on site. As loading equipment and haul trucks come on line, waste rock removal and low grade mineralized material stockpiling rates will increase until the earthwork requirements are met and the mill is ready for production. In the preproduction period, 29.1 Mt (26.4 Mmt) of low grade mineralized material will be stockpiled. The low grade stockpile will continue to be used for the life of the Project.

Production mining will begin in Year 1 and last until Year 16, with a total annual movement cap of 60.6 Mt (55 Mmt). During these years, mineralized material will be routed to the mill or the stockpile, based on grades, while waste rock will be sent to the waste rock stockpile, the tailings management facility, when needed, or used for project work as required. In Year 1, high grade mineralized material being sent to the mill was limited, based on tonnage and rock type blending requirement for mill production start-up and general mill production ramp-up. Based on these requirements 15.4 Mt (14 Mmt) of mineralized material is scheduled in Year 1. In the mine production period from Year 2 until Year 16, mill throughput will be maintained at 18.7 Mt (17 Mmt) per year of mineralized material combined from both the open pit and the stockpile, based on grade, rock type blending and scheduling requirements such as waste rock stripping. Once mine resources are exhausted in Year 16, the mill will be fed from the remaining low grade stockpile from Year 17 through Year 23.

Waste rock stripping varies throughout the mine life, as does mill feed and stockpiling of mineralized material. Over the life of the mine, the schedule has an overall average waste rock to mineralized material strip ratio of 1.3. Specific information about the Project infrastructure and Project earthwork, including the low grade and waste rock stockpiles, is detailed in Chapter 18 of this report. Table 16-7 contains the complete production schedule. Table 16-8 shows the mill tonnages by rock type and by year.

Table 16-7: Livengood production schedule

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	Total
Mineralized Material to Mill (Kt)	-	-	12,378	18,748	19,242	3,113	19,108	18,662	19,266	19,256	18,459	16,598	11,946	17,787	18,987	19,059	19,266	2,503	-	-	-	-	-	-	-	254,377
Mineralized Material to Stockpile (Kt)	9,963	19,122	23,031	13,686	15,102	2,868	11,219	11,894	11,954	6,464	6,677	8,587	6,508	7,867	6,214	8,299	7,409	488	-	-	-	-	-	-	-	177,352
Stockpile to Mill (Kt)	-	-	3,358	24	95	16,455	22	191	136	248	454	2,379	7,323	1,217	-	-	-	17,029	19,040	18,778	19,539	21,130	18,395	18,823	12,717	177,352
Waste Rock (Kt)	30,823	55,835	25,217	28,194	26,283	54,644	30,298	30,071	29,406	34,907	35,492	35,441	42,173	34,972	25,624	21,164	13,623	659	-	-	-	-	-	-	-	554,825
Strip Ratio	-	-	2.04	1.50	1.37	17.55	1.59	1.61	1.53	1.81	1.92	2.14	3.53	1.97	1.35	1.11	0.71	0.26	-	-	-	-	-	-	-	2.18
Au Grade (g/mt)	-	-	0.99	0.96	0.92	0.60	0.97	0.81	0.92	0.78	0.88	0.80	0.65	0.81	0.82	0.79	0.89	0.55	0.46	0.46	0.44	0.41	0.48	0.50	0.50	0.71
Contained Au (Koz)	-	-	454	528	516	345	542	447	522	443	486	443	366	446	454	440	500	313	256	252	252	253	255	273	187	8,972
Au Recovery	0%	0%	81%	76%	79%	84%	78%	76%	71%	82%	77%	72%	78%	75%	74%	71%	67%	80%	79%	76%	78%	82%	65%	63%	62%	75%
Total Recovered Au (Koz)	-	-	368	404	408	289	422	341	372	364	375	317	285	336	337	312	336	249	201	190	196	207	167	172	116	6,764

Note: Tonnages reported are expressed in kilo short tons (Kt), whereas gold grades are reported in grams per metric ton (g/mt).

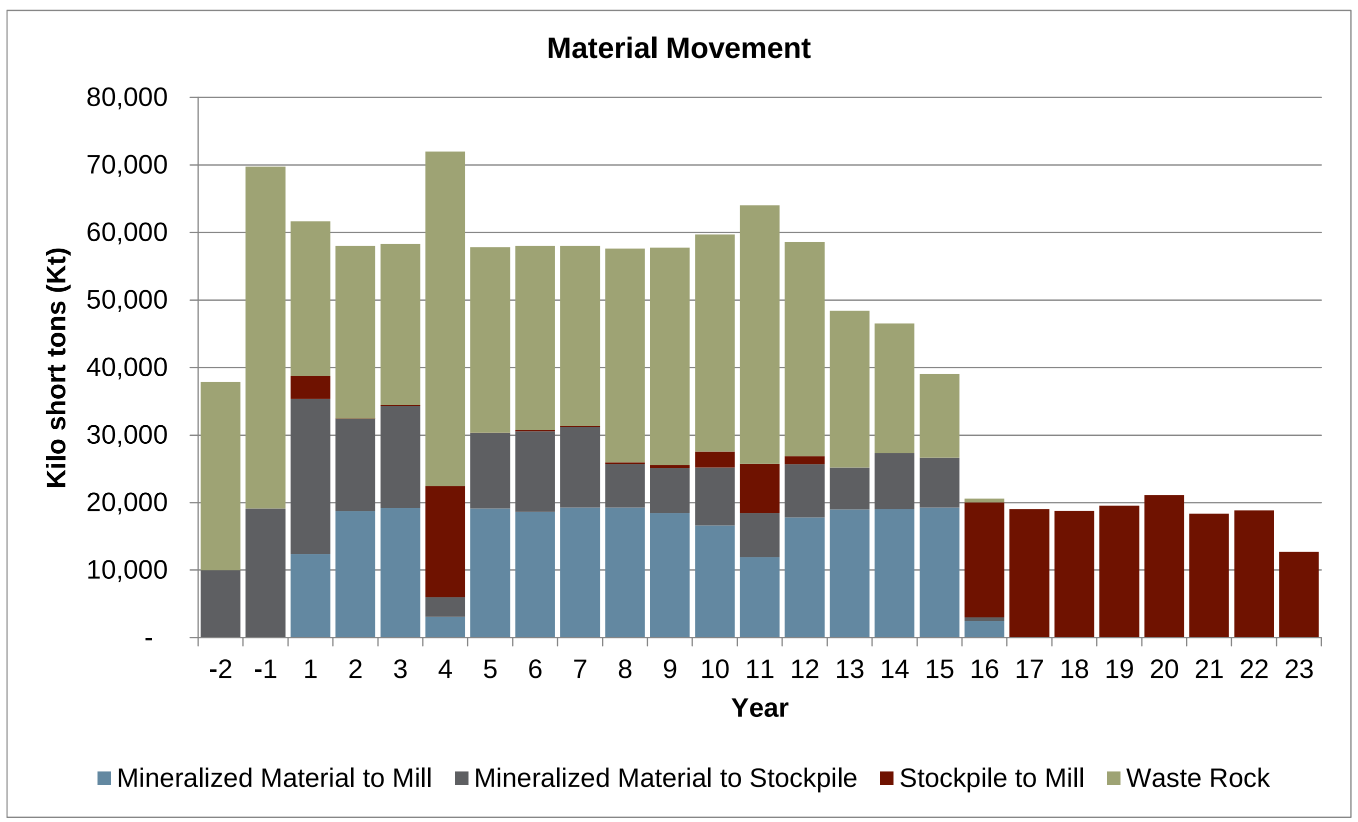


Figure 16-13: Production tonnage by material

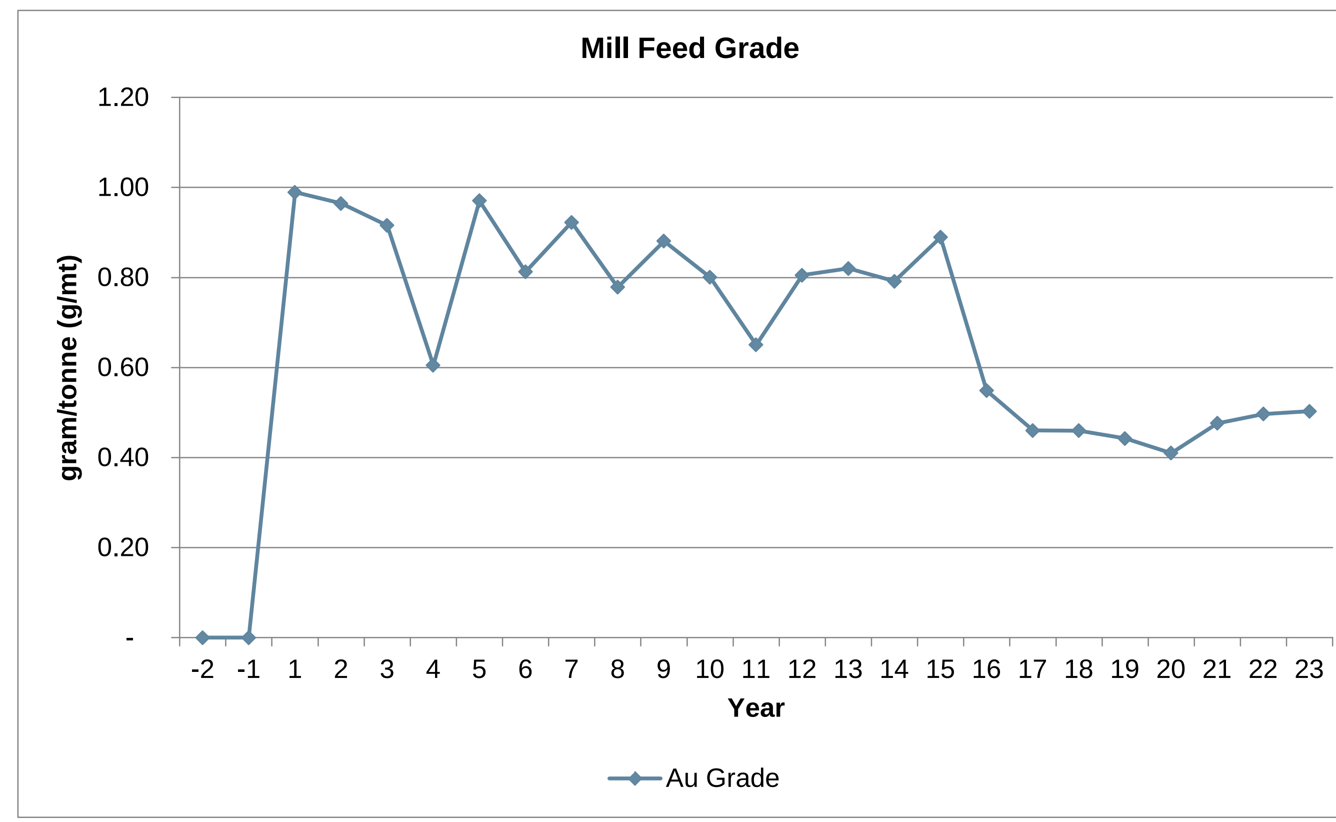


Figure 16-14: Mineralized material head grade

Table 16-8: Livengood mineralized material production schedule by rock type (RT)

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	Total
RT 4 to Mill (Kt)	-	-	7,091	2,147	3,147	2,007	5,702	217	1,339	1,092	1,487	1,189	2,653	4,291	3,968	1,356	1	-	-	-	6,840	21,130	2,137	-	-	67,798
Au Grade (g/mt)	-	-	0.96	0.84	0.78	1.09	0.90	0.93	0.75	0.96	1.01	0.89	0.77	0.75	0.71	0.65	0.39	-	-	-	0.41	0.41	0.41	-	-	0.66
Contained Au Koz	-	-	199	52	71	64	149	6	29	31	44	31	60	94	82	26	0	-	-	-	82	253	26	-	-	1,298
Au Recovery	-	-	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	-	-	-	82%	82%	82%	-	-	82%
RT 5 to Mill (Kt)	-	-	4,906	1,270	6,947	16,855	22	6,259	4,316	14,964	5,537	2,382	7,428	1,547	604	984	440	17,029	6,626	-	-	-	-	-	-	98,116
Au Grade (g/mt)	-	-	0.97	0.80	0.79	0.54	0.53	0.58	0.62	0.74	0.79	0.46	0.47	0.51	0.64	0.73	0.83	0.46	0.46	-	-	-	-	-	-	0.61
Contained Au Koz	-	-	139	30	159	265	0	106	78	324	127	32	101	23	11	21	11	229	89	-	-	-	-	-	-	1,745
Au Recovery	-	-	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	-	-	-	-	-	-	85%
RT 6 to Mill (Kt)	-	-	2,837	8,110	5,360	563	6,713	2,592	1,704	933	3,086	4,103	5,583	3,478	4,045	5,069	2,273	3	12,414	18,778	12,700	-	-	-	-	100,346
Au Grade (g/mt)	-	-	1.06	0.98	1.09	0.78	0.90	0.83	0.73	0.78	0.87	0.78	0.74	0.79	0.84	0.74	0.80	0.92	0.46	0.46	0.46	-	-	-	-	0.69
Contained Au Koz	-	-	88	232	170	13	177	63	36	21	79	93	120	80	99	110	53	0	167	252	170	-	-	-	-	2,022
Au Recovery	-	-	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	-	-	-	-	76%
RT 7 to Mill (Kt)	-	-	-	41	472	94	8	1,485	7,302	13	670	4,189	557	768	1,558	5,410	10,928	1,746	-	-	-	-	-	7,024	11,537	55,697
Au Grade (g/mt)	-	-	-	1.01	0.91	0.97	0.86	0.78	1.11	0.82	0.81	0.94	0.69	0.75	0.78	0.85	0.91	1.21	-	-	-	-	-	0.51	0.50	0.78
Contained Au Koz	-	-	-	1	12	3	0	34	237	0	16	114	11	17	35	134	291	62	-	-	-	-	-	116	187	1,271
Au Recovery	-	-	-	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	62%	-	-	-	-	-	62%	62%	62%
RT 8 to Mill (Kt)	-	-	-	30	52	49	-	909	259	809	271	28	106	284	256	532	366	1	-	-	-	-	2,896	-	-	6,871
Au Grade (g/mt)	-	-	0.86	0.95	1.10	1.14	-	0.88	0.78	0.87	1.02	0.74	0.74	0.79	0.84	0.92	0.94	1.00	-	-	-	-	0.49	-	-	0.72
Contained Au Koz	-	-	1	1	2	2	-	23	6	21	8	1	2	7	6	14	10	0	-	-	-	-	41	-	-	144
Au Recovery	-	-	72%	74%	77%	78%	0%	72%	70%	72%	75%	69%	69%	70%	71%	73%	74%	75%	-	-	-	-	63%	-	-	69%
RT 9 to Mill (Kt)	-	-	877	7,174	3,359	1	6,687	7,391	4,482	1,692	7,862	7,088	2,940	8,637	8,556	5,708	5,256	753	-	-	-	-	13,362	11,079	-	102,903
Au Grade (g/mt)	-	-	1.06	1.01	1.04	0.73	1.10	0.99	1.03	0.93	0.93	0.83	0.83	0.89	0.88	0.81	0.88	1.00	-	-	-	-	0.48	0.48	-	0.83
Contained Au Koz	-	-	27	212	101	0	215	214	135	46	212	172	72	225	221	136	135	22	-	-	-	-	189	157	-	2,490
Au Recovery	-	-	76%	75%	76%	69%	77%	75%	76%	73%	73%	71%	71%	73%	72%	71%	72%	75%	-	-	-	-	63%	63%	-	71%
Total Mill (Kt)	-	-	15,734	18,772	19,335	19,569	19,130	18,853	19,402	19,504	18,913	18,977	19,267	19,005	18,987	19,059	19,266	19,533	19,040	18,778	19,539	21,130	18,395	18,823	12,717	431,729
Total Recovered Au Koz	-	-	368	404	408	289	422	341	372	364	375	317	285	336	337	312	336	249	201	190	196	207	167	172	116	6,764

Note: Tonnages reported are expressed in kilo short tons (Kt), whereas gold grades are reported in grams per metric ton (g/mt).

16.5 Mine equipment

Mine equipment requirements were developed using detailed equipment calculations for drills, shovels, loaders, and haul trucks. Support equipment requirements were developed using comparisons to other similarly scaled operations along with specific project needs.

16.5.1 Drilling and blasting

The primary production drill is a 59,400 lb (27,000 kg) pull-down rotary blast hole drill capable of drilling 10 in (251 mm) blast holes on 32.8 ft (10 m) benches with 4.9 ft (1.5 m) of sub-grade. Drilling estimates were completed using a drillhole spacing of 22.3 ft (6.8 m) × 26.2 ft (8 m), except for the 22.3 ft (6.8 m) × 22.3 ft (6.8 m) pattern needed for the increased fragmentation required for rock types 4 and 9. The closer pattern spacing for rock types 4 and 9 (Cambrian and Volcanics, respectively) allows for increased fragmentation, which has a downstream benefit on mill processing cost and mill throughput. Each blast hole will produce 1,386 t (1,258 mt) of shot material at a specific gravity of 2.72. These drills are projected to complete 750 ft (228.7 m) of drilling or 19.9 holes per shift. A smaller 6.7 in (171 mm) rotary drill is included for back-up drilling and trim blasting. A summary of drilling requirements is shown in Table 16-9.

Table 16-9: Drill requirements

Year	10 in (251 mm) Drill			6.7 in (171 mm) Drill		
	Annual Drill Hours	Drills Required	Rounded Drills	Annual Drill Hours	Drills Required	Rounded Drills
-2	12,630	2.4	4	1,972	0.4	1
-1	23,284	4.4	4	5,953	1.1	1
1	20,619	3.9	4	6,029	1.1	1
2	20,353	3.8	4	5,913	1.1	1
3	20,524	3.9	4	5,988	1.1	1
4	17,983	3.4	4	4,886	0.9	1
5	20,164	3.8	4	5,832	1.1	1
6	20,185	3.8	4	5,840	1.1	1
7	20,244	3.8	4	5,866	1.1	1
8	19,751	3.7	4	5,653	1.1	1
9	19,699	3.7	4	5,630	1.1	1
10	19,704	3.7	4	5,632	1.1	1
11	19,101	3.6	4	5,370	1.0	1
12	19,746	3.7	4	5,650	1.1	1
13	16,884	3.2	4	5,252	1.0	1
14	16,415	3.1	4	5,246	1.0	1

Year	10 in (251 mm) Drill			6.7 in (171 mm) Drill		
	Annual Drill Hours	Drills Required	Rounded Drills	Annual Drill Hours	Drills Required	Rounded Drills
15	13,986	2.6	3	2,602	0.5	1
16	1,319	0.2	3	258	0.0	1
17	-	-	-	-	-	-
18	-	-	-	-	-	-
19	-	-	-	-	-	-
20	-	-	-	-	-	-
21	-	-	-	-	-	-
22	-	-	-	-	-	-
23	-	-	-	-	-	-
Total Hours	**322,591**			**89,572**		

A powder factor of 0.50 lb/t (0.25 kg/mt) was used to calculate explosives requirements, except for rock types 4 and 5, which use a powder factor of 0.74 lb/t (0.37 kg/mt) to increase fragmentation. Exclusive use of emulsion with a 70/30 blend (70% Emulsion / 30% ANFO blend) for all rock types was used for calculations and costing. The use of emulsion only for loading holes, as opposed to an ANFO or emulsion combination, eliminates the need of a mixed fleet of equipment for loading and transporting, removes the requirements for hole dewatering and associated costs, and the higher density gives higher drill pattern expansion that can reduce drilling costs or help increase fragmentation, depending on drill pattern spacing. Table 16-10 shows the required explosives by year.

Table 16-10: Explosives requirements

Year	70/30 Blend Emulsion Tons (t)	Fuel Oil Gallons (gal)
-2	11,057	222,150
-1	20,404	409,682
1	18,217	365,988
2	17,960	360,821
3	18,124	364,136
4	15,673	314,895
5	17,778	357,166
6	17,799	357,561
7	17,855	358,715

Year	70/30 Blend Emulsion Tons (t)	Fuel Oil Gallons (gal)
8	17,379	349,165
9	17,330	348,150
10	17,334	348,237
11	16,752	336,549
12	17,374	349,052
13	14,884	299,035
14	14,495	291,212
15	12,380	248,713
16	1,171	23,532
17	-	-
18	-	-
19	-	-
20	-	-
21	-	-
22	-	-
23	-	-
Totals	**283,871**	**5,704,762**

16.5.2 Loading and hauling

Waste rock will be mined using 47 yd^3 (36 m^3) face shovels, allowing five pass loading for the 320 t (290 mt) capacity haul trucks. A 40 yd^3 (30.6 m^3) front end loader will be used to augment production when high mobility is needed. The front end loader will load the haul trucks in six passes. Mineralized material mining would be done with the same machines. Table 16-11 shows the loading hours and calculated shovels and loaders required by year. Haul distances and cycle times were estimated for all material destinations from each mining phase. Haul trucks with 320 t (290 mt) capacity were chosen as a size that is sufficient to maintain efficiencies and to keep loading equipment utilized. Table 16-12 shows the truck hours and calculated haul trucks required by year.

Table 16-11: Shovel and loader requirements

Year	47 yd³ (36 m³)Shovel			40 yd³ (30.6 m³) Loader		
	Annual Shovel Hours	Shovels Required	Rounded Shovels	Annual Loader Hours	Loaders Required	Rounded Loaders
-2	6,747	1.2	2	923	0.2	1
-1	12,382	2.2	3	1,721	0.3	1
1	21,990	1.7	2	4,627	0.5	1
2	21,990	1.7	2	6,533	0.3	1
3	21,990	1.7	2	8,515	0.3	1
4	21,990	1.8	2	14,126	1.0	1
5	21,990	1.7	2	15,968	0.3	1
6	21,990	1.7	2	17,864	0.3	1
7	21,990	1.7	2	19,764	0.3	1
8	21,990	1.7	2	21,530	0.3	1
9	21,990	1.7	2	23,335	0.3	1
10	21,990	1.7	2	25,668	0.4	1
11	21,990	1.8	2	29,153	0.3	1
12	21,990	1.7	2	31,182	0.3	1
13	21,990	1.4	2	32,715	0.3	1
14	21,990	1.4	2	34,277	0.3	1
15	21,990	1.1	2	35,692	0.2	1
16	21,990	0.1	1	40,497	-	-
17	21,990	0.6	1	40,497	-	-
18	21,990	0.6	1	40,497	-	-
19	21,990	0.6	1	40,497	-	-
20	21,990	0.7	1	40,497	-	-
21	21,990	0.6	1	40,497	-	-
22	21,990	0.6	1	40,497	-	-
23	21,990	0.4	1	40,497	-	-
Total Hours	**524,902**			**647,567**		

Table 16-12: Truck requirements

Year	Annual Truck Hours	Trucks Required	Rounded Trucks	Cycle (Min) Mineralized Material Shovel	Cycle (Min) Mineralized Material Loader	Cycle (Min) Stockpile Shovel	Cycle (Min) Stockpile Loader	Cycle (Min) Waste Rock Shovel	Cycle (Min) Waste Rock Loader
-2	45,888	7.1	8	17.8	19.9	15.3	16.9	23.4	25.5
-1	84,115	13.0	13	17.8	19.9	15.3	16.9	23.4	25.5
1	60,124	9.3	10	16.8	18.9	15.3	16.9	20.4	22.5
2	57,614	8.9	9	16.8	18.9	15.3	16.9	20.4	22.5
3	49,819	7.7	8	17.6	19.7	15.3	16.9	13.7	15.8
4	112,058	17.3	18	28.3	30.4	15.3	16.9	31.9	34.0
5	93,345	14.4	15	28.3	30.4	15.3	16.9	31.9	34.0
6	93,459	14.4	15	28.3	30.4	15.3	16.9	31.9	34.0
7	61,928	9.6	10	17.9	20.0	15.3	16.9	21.5	32.4
8	63,080	9.7	10	17.9	20.0	15.3	16.9	21.5	32.4
9	68,151	10.5	11	19.7	21.8	15.3	16.9	23.3	25.4
10	69,863	10.8	11	19.7	21.8	15.3	16.9	23.3	25.4
11	95,099	14.7	15	26.1	28.2	15.3	16.9	29.7	32.4
12	88,437	13.6	14	26.1	28.2	15.3	16.9	29.7	32.4
13	72,610	11.2	12	26.1	28.2	15.3	16.9	29.7	32.4
14	68,767	10.6	11	26.1	28.2	15.3	16.9	29.7	32.4
15	56,449	8.7	9	26.1	28.2	15.3	16.9	29.7	32.4
16	5,018	0.8	1	26.1	28.2	15.3	16.9	29.7	32.4
17	31,709	4.9	5	-	-	15.3	16.9	-	-
18	31,273	4.8	5	-	-	15.3	16.9	-	-
19	15,075	2.3	3	-	-	15.3	16.9	-	-
20	16,303	2.5	3	-	-	15.3	16.9	-	-
21	14,193	2.2	3	-	-	15.3	16.9	-	-
22	14,522	2.2	3	-	-	15.3	16.9	-	-
23	9,812	1.5	2	-	-	15.3	16.9	-	-
Total Hours	**1,378,710**								

16.5.3 Production and support equipment fleet

Using the calculated required production equipment, the schedule for purchases of new equipment and rebuilds of primary production equipment was produced. Replacement and rebuild schedules are based on equipment hours as well as general time periods used by similarly scaled operations with equipment rebuild programs. Excess capacity in drilling, loading and hauling is taken into account to size the production fleets properly for each period. In addition to the primary production equipment, support equipment requirements were also assessed. Support equipment needed to keep all haul roads watered and graded, shovel and loading sites leveled and cleaned, and drilling sites leveled and cleaned, will also aid in projects in the preproduction period and throughout the life of mine. The type and size of the support equipment is based on information from similarly scaled operations. The number of required units takes into account the number of potential loading sites, potential material destinations, as well as haul road lengths in a given production period. Table 16-13 shows a summary of the annual equipment requirements for the mine.

Table 16-13: Livengood mine equipment schedule

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Production Equipment																									
Rotary Drill (10 in / 251 mm)	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-
Rotary Drill (6.7 in / 171 mm)	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3	-	-	-	-	-	-	-
Shovel (47 yd^3 / 36 m^3)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1
Loader (40 yd^3 / 30.6 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Truck (320 t / 290 mt)	8	13	13	13	13	18	18	18	18	18	18	18	18	14	14	14	14	14	5	5	5	3	3	3	3
Truck Bed (320 t / 290 mt)	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Spare Bucket (47 yd^3 / 36 m^3)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Spare Bucket (40 yd^3 / 30.6 m^3)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Support Equipment																									
Dispatch System	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader (15.7 yd^3 / 12 m^3)	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Spare Bucket (15.7 yd^3 / 12 m^3)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front End Loader (3.3 yd^3 / 2.5 m^3)	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Spare Bucket (3.3 yd^3 / 2.5 m^3)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
13,000 gal (49,210 L) water Truck	1	2	2	2	2	2	2	2	2	2	2	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	2	2	2	2	2	2	2	2	2
Dozer 800-900 HP	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Rubber Tire Dozer 1050 HP	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Grader Blade (16 ft / 4.9 m)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader Blade (24 ft / 7.3 m)	-	1	1	1	1	1	1	1	1	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	1	1	1	1	1	1	1	1	1

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Mass Excavator (5.25 yd^3/ 4 m^3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
300 HP Roller/Compactor	1	1	1	1	1	1	1	1	1	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	8	8	8	8	8	8	8	8	8
Stemming Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-	-	-	-	-
Low Boy for Drill, Dozer Transport	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Explosives Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-	-	-	-	-
Fuel and Lube - 40 T	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Truck	1	1	1	1	1	1	1	1	1	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	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	2	2	2	2	2	2	2	2	2
Rough Terrain Crane (22 t / 20 mt)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Crane (55 t / 50 mt)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Generators and Portable Crusher	1	1	1	1	1	1	1	1	1	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	1	1	1	1	1	1	1	1	1	1
Radios and Base Stations	1	1	1	1	1	1	1	1	1	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	1	1	1	1	1	1	1	1	1
GPS & Technology	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Flatbed Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Crew Vans	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Forklift (13.2 t / 12 mt)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
ATV	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2

16.6 Manpower

The mine operations manpower requirements were estimated based on the production and support equipment requirements. Mine maintenance manpower requirements were estimated based on ratios of the mine operations to mine maintenance personnel requirements of similarly sized mining operations. Mine support staff, including mine engineering, geology and ore control personnel, were estimated based on mine operations of similar size. Table 16-14 shows the estimated annual mine personnel.

Table 16-14: Mine personnel

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
MINE OPERATIONS																									
Mine Manager	.	1	1	1	1	1	1	1	1	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	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	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	1	1	1	1	1	1	1	1	1
Mine Trainer	2	2	2	2	1	1	1	1	1	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	3	3	3	3	3	3	3	3	3
Load and Haul Operator	29	48	36	34	31	58	50	50	36	36	39	39	50	47	39	36	31	17	17	17	12	12	12	12	9
Drill and Blasting Superintendent	1	1	1	1	1	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	6	6	6	6	6	6	6	6	6	6	-	-
Blasting Foreman	1	1	1	1	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	2	2	2	2	-	-	-	-	-	-	-
Blasting Helper	3	3	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	-	-	-	-	-	-	-
Driller	9	15	18	18	18	18	18	18	18	12	12	12	12	12	12	12	9	3	-	-	-	-	-	-	-
Support Equipment Operators	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19
Trainee	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	-	-	-	-	-	-	-
Subtotal Mine Operations	82	107	100	98	94	121	113	113	99	93	96	96	107	104	96	93	85	63	49	49	44	44	44	38	35
MINE MAINTENANCE																									
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	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	6	6	6	6	6	3	3	3	3	3	3	3
Planning Engineer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
Mechanic	19	28	26	25	24	32	30	30	26	24	24	24	28	27	23	22	20	14	12	12	10	10	10	10	9
Electrician	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Welder	3	3	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	3	3	3	3	3	3	3
Servicemen	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
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	2	2	2	2	2	2	2	2	2	2	2	2
Mechanic Trainee	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal Mine Maintenance	45	54	54	53	52	59	57	57	53	51	51	51	55	52	47	46	42	36	28	28	26	26	26	26	25
Total Mine Operations	127	161	154	151	146	180	170	170	152	144	147	147	162	156	143	139	127	99	77	77	70	70	70	64	60
MINE ENGINEERING																									
Chief Mining Engineer	1	1	1	1	1	1	1	1	1	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	1	1	1	1	1	1	1	1	1
Sr Mining Engineer	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1
Surveyor	1	1	1	1	1	1	1	1	1	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	2	2	1	1	1	1	1	1	1
Subtotal Engineering	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	5	5	5	5	5	5	5
GEOLOGY AND GRADE CONTROL																									
Chief Geologist	1	1	1	1	1	1	1	1	1	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	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	2	2	2	2	2	2	2	2	2	2	2	2
Sampler	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	1	1	1	1	1
Subtotal Geology and Grade Control	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	5	5	5	5	5	5	5
TOTAL MINE STAFF	141	141	175	168	165	160	194	184	184	166	158	161	161	176	170	157	153	141	113	87	87	80	80	80	74

16.7 Production

The mine is planned to produce mineralized material and waste rock year round with two 12-hour shifts per day, 7 days a week. After the preproduction mill ramp-up periods, daily mill throughput will be 52,600 t (47,700 mt) of mineralized material. Low grade mineralized material will be stockpiled and reclaimed as mill feed, as required. Table 16-15 shows daily production rates for each year.

Table 16-15: Daily mine production tons per day (t/d) averages by year

Year	Mineralized Material	To Stockpile	From Stockpile	Waste Rock	Total Material
-2	-	27,288	-	84,422	111,710
-1	-	52,374	-	152,929	205,303
1	33,901	63,084	9,196	69,070	175,251
2	51,349	37,485	67	77,220	166,122
3	52,702	41,362	259	71,990	166,314
4	8,527	7,856	45,072	149,671	211,127
5	52,338	30,730	61	82,986	166,115
6	51,115	32,576	523	82,363	166,578
7	52,769	32,743	371	80,543	166,426
8	52,742	17,705	679	95,608	166,734
9	50,559	18,287	1,243	97,210	167,298
10	45,464	23,519	6,515	97,072	172,570
11	32,718	17,826	20,055	115,511	186,110
12	48,718	21,550	3,335	95,787	169,390
13	52,003	17,020	-	70,185	139,209
14	52,202	22,731	-	57,967	132,901
15	52,768	20,292	-	37,313	110,373
16	6,856	1,338	46,643	1,806	56,643
17	-	-	52,150	-	52,150
18	-	-	51,433	-	51,433
19	-	-	53,518	-	53,518
20	-	-	57,875	-	57,875
21	-	-	50,385	-	50,385
22	-	-	51,554	-	51,554
23	-	-	34,832	-	34,832

17. RECOVERY METHODS

17.1 Introduction

The recovery methods for the Livengood Gold Project were established on the basis of laboratory-scale testwork as described in Chapter 13, equipment information from suppliers, and BBA's experience on similar projects. The resulting flowsheet reflects the results of this initial testwork and forms the basis for the plant design, capital costs and operating costs developed in this study.

17.2 Process plant production schedule

The mine is scheduled to deliver an average tonnage of 52,600 t/d (47,700 mt/d) of ore to the primary crusher and process plant on a 365 day per year basis. The process plant is designed to operate with an availability of 92%. The primary crushing and main process plant will operate 24 hours per day and 7 days per week. The operating teams will work on a schedule of two 12 hour shifts. The main process plant will be stopped periodically to perform preventive maintenance on equipment, for which there is no standby unit.

The overall gold recovery of the proposed circuit is estimated at 75.3% based on the rock types to be processed according to the LOM plan. Average annual gold production is estimated to be 378,000 oz/year for the first 5-years and approximately 294,000 oz/year life of mine.

17.3 Conceptual process flow diagram

The processing plant consists of primary crushing, ore reclaiming, pre-crushing, grinding, gravity recovery, carbon in leach (CIL) with ADR (adsorption, desorption and reactivation) circuits, cyanide detoxification, water and tailings management, and reagent preparation circuits geared to produce gold doré for delivery to the refinery. Figure 17-1 describes the conceptual process flow from the ore delivery to the crusher to doré production and tailings and water management.

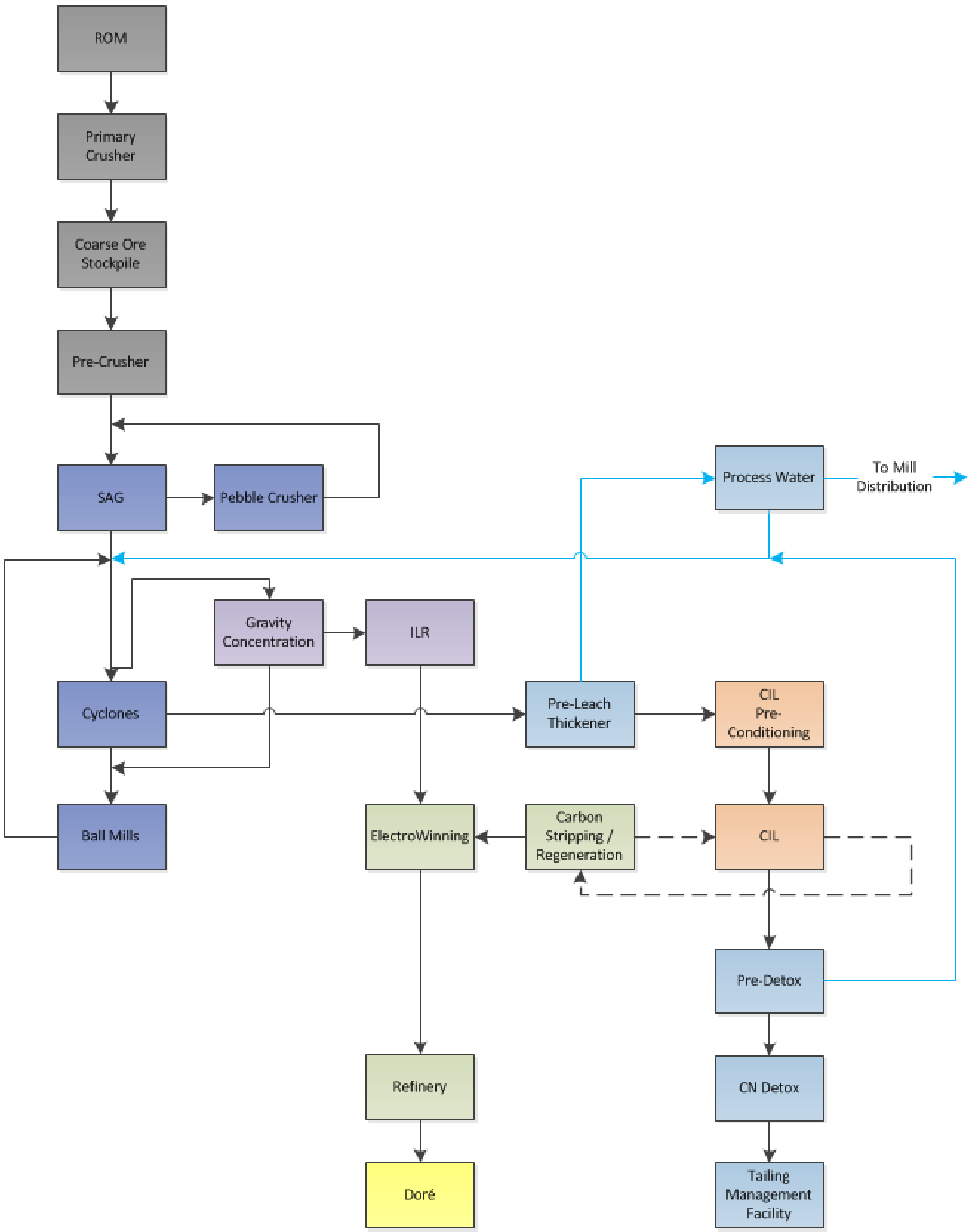


Figure 17-1: Conceptual process block flow diagram

17.4 Plant operating design parameters

The design criteria to determine sizing of the equipment is based on a nominal daily processing plant throughput capability of 52,600 t (47,700 mt) with a 92% availability factor. A design factor of 1.1 was typically considered for areas where flow rates are not affected by the feed grades of the ore processed. Table 17-1 presents an overview of the main design criteria factors employed and sizes of the most significant process equipment.

Table 17-1: General process design criteria

Criterion	Unit	Value
General Design Data		
Process Plant Operating Life	y	23
Overall Process Plant Availability	%	92
Operating Hours Per Year	hr	8,059
Design Factor		1.1
Production Rates		
Life of Mine	Mt (Mmt)	403 (365.6)
Annual	Mt/y (Mmt/y)	19.2 (17.4)
Daily	t/d (mt/d)	52,600 (47,700)
Process Plant Feed		
Gold Grade (LOM Average)	g/mt	0.71
Feed Size (ROM, F_{80})	in (mm)	39.4 (1,000)
Primary Crushing		
Crusher Type / Size - (Gyratory (54' × 75'))	hp (kW)	600 (450)
Utilization	%	65
Product Size (P_{80})	in (mm)	4.4 (138)
Hourly Throughput	t/h (mt/h)	3,371 (3,058)
Stockpile Retention Time (Live)	hr	12
Secondary Crushing (Pre-crushing)		
Crusher Type / Size - Cone	hp (kW)	1000 (750)
Product Size (P_{80})	in (mm)	1.65 (42)

Criterion	Unit	Value
Grinding and Pebble Crushing		
Hourly Throughput	t/h (mt/h)	2,381 (2,160)
Number of SAG Mills	No.	1
SAG Mill Size	D × L, ft	36 x 20
Scalping Screen Transfer Size (T_{80})	µm	2,800
SAG Mill Specific Energy (motor output)	kWh/mt	5.4
SAG Mill (Installed Power)	hp (kW)	18,774 (14,000)
Pebble Crusher No./Type/Size		1 x Cone (1000 hp)
Pebble Crusher Product Size (P_{80})	in (mm)	0.5 (14)
Number of Ball Mills	No.	1
Ball Mill Size	D × L, ft	26 × 40.5
Ball Mill (Installed Power)	hp (kW)	20,115 (15,000)
Ball Mill Specific Energy (motor output)	kWh/mt	6.6
Ball Mill Product Size (P_{80})	µm	180
Ball Mill Circulating Load	%	250
Gravity Circuit		
Screens	No.	8 (1 / gravity concentrator)
Gravity Concentrator Size (Diameter)	in	48
Number of Gravity Concentrators	No.	8 (2 lines x 4 units)
Intensive Leach Reactor (ILR)	No.	1
Cyanide Leaching and ADR		
Pre-Leach Thickener diameter	ft (m)	131 (40)
Pre-Conditioning Tank Dimension	D x H, ft	47 x 56
Number of Pre-Conditioning Tanks	No.	4 (2 lines x 2 tanks)
Pre-Conditioning Retention Time	hr	3
CIL Circuit Volume	m^3	68,250
CIL Tank Dimension	D x H, ft	59 x 63
Number of CIL Tanks	No.	14 (2 lines × 7 tanks)
CIL Retention Time	hr	21
pH		10.5
Carbon Concentration	g/L	20
Carbon Tonnage per tank	t (mt)	107 (98)
Carbon Transfers per Day	No.	1
Average Carbon Loading	g/mt	4,000 [(1)]
Carbon Stripping, Regeneration Capacity	t/batch (mt/batch)	44 (40)

Criterion	Unit	Value
Cyanide Detoxification		
Pre-Detox Thickener Diameter	ft (m)	131 (40)
CIL Discharge Cyanide Concentration	ppm (mg/L)	120
Detoxification Tank Dimension	D x H, ft	47 x 56
Detoxification Circuit Retention Time	hr	1.5
Sulphur Burner Capacity	t SO_2/d	23.7
Detox Circuit Discharge Target (WAD Cyanide)	ppm (mg/L)	22
Tailings Slurry Density	%	50

(1) This assumption was retained from the FS and is to be confirmed through additional CIL testwork and simulations.

17.5 Process plant facilities description

The Livengood process facilities will consist of a comminution circuit (one SAG and one ball mill) followed by a gravity concentration circuit. The tailings from the 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 intensively leached from the gravity concentrate. Figure 17-2 presents a schematic process flow diagram while the following subsections describe the selected flowsheet in more detail.

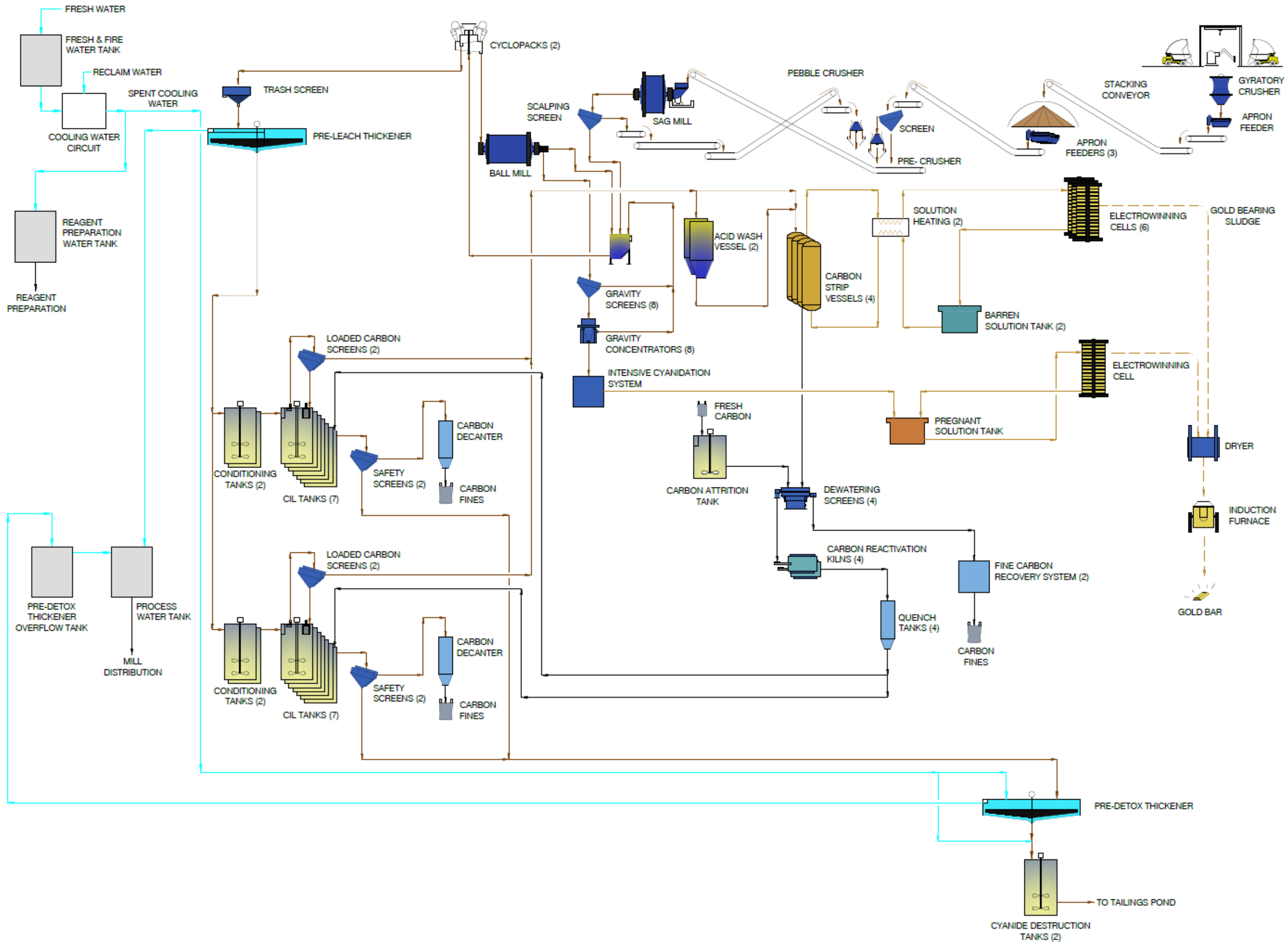


Figure 17-2: Conceptual process flowsheet

17.5.1 Primary crushing

The primary crushing system is a single stage open circuit (54 × 75) gyratory crusher (600 hp, 450 kW). The crusher selection is based upon a feed top size of 39.4 in (1,000 mm) and a product (P_{80}) of 5.4 in (138 mm), with an expected utilization of 65% at 52,600 t/d (47,700 mt/d). The live capacity of the feed and discharge hoppers to the gyratory crusher are designed for slightly over two truckloads, assuming a nominal payload of 320 t (290 mt). The gyratory crusher's instantaneous capacity is 3,370 t/h (3,058 mt/h) and the system is equipped with a sacrificial conveyor.

17.5.2 Crushed ore stockpile

The crushed ore storage pile is designed for a live capacity corresponding to approximately 12 hours of crushing or 26,290 t (23,850 mt). The total capacity of the storage pile (live + dead) is 87,756 t (79,612 mt). The coarse ore stockpile is covered by a dome.

17.5.3 Secondary crushing (pre-crushing)

Ore reclaim from the stockpile is fed from a reclaim tunnel. The reclaim tunnel is equipped with three apron feeders that feed a secondary cone crusher installed in an open circuit. A screen 118 in × 287 in (3 m × 7.3 m) receives the gyratory crusher product, which directs oversize material to a cone crusher (1,000 hp, 746 kW) that crushes the oversize to a P_{80} of 1.65 in (42 mm). The screen undersize and secondary crusher product is subsequently fed to the SAG mill. The secondary crusher is equipped with a by-pass chute to maintain high plant availability.

17.5.4 Grinding and pebble crushing

A SAG mill / ball mill, in a SABC configuration has been selected (Figure 13-5) for the Livengood Gold Project; this configuration provides increased efficiency for competent to medium hard ores. In a SABC circuit, the SAG mill operates in closed circuit with a pebble crusher. The SAG mill is equipped with pebble ports, which evacuate the hard, critical size pebbles that are then conveyed to the pebble crusher, before being returned to the SAG mill. The ball mill operates in closed circuit with hydrocyclones. The required SAG mill power is estimated at 6.0 kWh/t (5.4 kWh/mt), while the required ball mill power is estimated at 7.3 kWh/t (6.6 kWh/mt), for a combined total of 13.3 kWh/t (12.1 kWh/mt) at the pinion, excluding the pebble crusher and secondary crusher power. The grinding circuit product used to design the mill power is 180 µm (P_{80}). The total power required to grind from primary crusher to final ball mill product is 13.9 kWh/t (12.6 kWh/mt). Note that all estimated power values cited are based on the motor output.

The grinding circuit is based on one grinding line, which is comprised of a SAG mill (D×L: 36 ft × 20 ft,) with installed power of 18,774 hp (14,000 kW) and a ball mill (26 ft × 40.5 ft) with installed power of 20,115 hp, (15,000 kW).

The product from the SAG mill will fall onto a classification screen. The oversize from the scalping screen will be conveyed to a single cone (pebble) crusher (1000 hp (746 kW)), the product of which is returned to the SAG mill. A scalping screen undersize product of 2,880 µm (P_{80}) is discharged into a SAG/ball mill (combination) pumpbox, from which the slurry is pumped to a pair of hydrocyclone clusters. The cyclone underflow is fed to the ball mill, with discharge from this mill returning to the combination pumpbox. A portion of cyclone feed of each hydrocyclone cluster (25%) is directed to a distributor that feeds four (4) gravity screens (4 per hydrocyclone cluster), each of which feeds its own gravity concentrator (48 in). The gravity screen oversize and gravity concentrator tails are returned to the combination pumpbox. The gravity concentrate, amounting to approximately 1 wt% mass pull (to be validated with additional studies), is sent to the intensive cyanidation (ILR) system.

Pebble lime will be added continuously at the ball mill to maintain ball mill discharge pH above 9.0 to promote sodium cyanide leaching downstream and limit the amount of conditioning required prior to CIL.

17.5.5 Gravity and intensive leaching

The Livengood gold ore contains significant amounts of free gold, which responds well to gravity concentration. The gravity circuit is fed by a portion of cyclone feed that is redirected to the gravity circuit. Based on testwork and simulations conducted by Knelson Gravity Solutions, the design gold recovery of the gravity circuit is estimated to be 40% for an average feed blend.

A batch intensive cyanidation system will be used to process the gravity concentrate. The extraction performance of gold from the gravity concentrate by the intensive cyanidation system is designed at 99%. The pregnant solution will be pumped to a tank in the gold room, followed by electrowinning in a dedicated cell.

17.5.6 Carbon in leach

The hydrocyclone overflow product will be pumped to a trash screen, before discharging into the pre-leach high rate thickener with a diameter of 131.2 ft (40 m). This thickener will thicken the slurry to 50 wt% in the thickener underflow stream. The thickener overflow will report to the process water tanks. The underflow from the thickener will feed CIL lines 1 and 2 at the pre-conditioning stage.

Pre-conditioning with oxygen and lead nitrate will be conducted in four large tanks. The designed retention time is four hours when the plant operates at 52,600 t/d (47,700 mt/d).

The CIL circuit is comprised of two lines of 7 large CIL tanks each and all within their own concrete containment areas. The designed retention time is 21 hours when the plant operates at 52,600 t/d (47,700 mt/d). The average design carbon loading remains 4000 g/mt from the FS, but is to be confirmed by additional CIL testwork and simulation.

The slurry will flow counter-currently to the carbon from tank 1 through to tank 7. Fresh carbon will be added to tank 7 and flow to tank 1, by way of the carbon advance pumps located in each CIL tank. Slurry will exit tank 7 over the carbon safety screens, before heading to the pre-detox thickeners. Loaded carbon exiting tank 1 will report to the carbon stripping system (ADR) for recovery of the adsorbed metals.

17.5.7 Adsorption desorption and recovery (ADR)

Loaded carbon from the CIL tanks reports to the loaded carbon stripping circuit, where gold will be stripped and the carbon reactivated for recycle to the CIL circuit. Based on the information available, it is assumed that one strip per day will be sufficient to recover the gold loaded onto the carbon.

The ADR circuit includes an acid wash stage (two vessels) and High Pressure "modified" Zadra process for gold stripping from the loaded carbon. The Zadra stripping circuit (4 vessels) is considered "modified", as the electrowinning is done in-line, with no pregnant tank between stripping and electrowinning. The barren solution is collected in two 31,700 gal (120 m^3) barren tanks. 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 kilns (4) that each possess the same carbon capacities and conservatively include provision for 100% regeneration of the carbon. 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 flow of pregnant solution from the Zadra circuit and the gravity ILR is split to feed a total of seven electrowinning cells. The refining equipment is designed to handle both the gold from the stripping circuit and from the gravity recovery system. The electrowinning sludge is filtered, dried, and mixed with fluxes, before being smelted in an induction furnace.

17.5.8 Pre-detox thickening and cyanide detoxification

The underflow of the pre-detox thickener (131.2 ft (40 m) diameter) is diluted from 64% to 55% using reclaimed water to dilute the slurry prior to cyanide detoxification.

The selected cyanide detoxification process is the Inco SO_2/air process. A sulphur burner will be used to produce SO_2. Cyanide detoxification was designed with 1.5 hours of retention time in two tanks. The tailings slurry will be pumped at 50 wt% solids coming out of the cyanide detoxification unit. It will then be pumped into the tailings management facility via two parallel pipelines, each having a design capacity of 11,446 gpm (2,600 m^3/h).

Thickener overflow reports to the process water tank and will be used for water needs upstream. Water recovered by the reclaim barge pumps from the settled tailings will be returned upstream to meet process water requirements.

All cyanide process tanks are provided with appropriately sized secondary containments and all process solution pipelines are contained within the mill complex (mill building, CIL/leach tank farm, and detoxification plant) and are provided with secondary containment in association with the major tanks that they serve.

17.6 Consumables

The main consumables for the processing plant are represented by the grinding media and liners for the SAG and ball mills, as well as the reagents used in the leaching, gold recovery and cyanide detoxification circuits.

All process reagents are contained in a separate area within the process plant building to prevent any contamination of any surrounding areas in case of a spill. Safety showers are provided in the different reagent mixing and utilization areas for safety, in case of contact with the reagents. HCN monitors will be installed in appropriate locations to ensure the safety of the employees. Grinding media will be located in pits located indoors, close to usage points.

The primary reagents used in the process include sodium cyanide (NaCN), lime (CaO), oxygen (O_2), elemental sulphur (S), sodium hydroxide (NaOH), hydrochloric acid (HCl), carbon, copper sulphate ($CuSO_4$), and flocculant. Consumption rates are mostly based on results from bench-scale testwork, with reductions as deemed applicable to recycle streams and implementation of control strategies at industrial scale.

Table 17-2 and Table 17-3 list all reagents, media, areas of usage and their purpose.

Table 17-2: Reagents and area of use

Reagent	Area	Use	Consumption (mt/y)
Lead nitrate ($PbNO_3$)	Pre-treatment ahead of carbon in leach and intensive leach	Surface passivation	2,300
Sodium cyanide (NaCN)	Carbon in leach and carbon elution	Dissolution of gold and elution	7,100
Oxygen (O_2)	Carbon -In-Leach	Dissolution of gold	36,500
Carbon (C)	Carbon in leach	Gold adsorption	600
Quick-lime (CaO)	Ball mills Carbon in leach Cyanide detoxification	pH control	29,500
Sodium hydroxide (NaOH)	Acid wash and Carbon elution	Neutralization of acid and elution	1,500
Hydrochloric acid (HCl)	Carbon elution	Acid wash	1,000
Elemental sulphur (S)	Cyanide detoxification	Cyanide detoxification	5,000
SMBS ($Na_2S_2O_5$)	Cyanide detoxification	Cyanide detoxification (backup)	3,600
Copper sulphate ($CuSO_4$)	Cyanide detoxification	Catalyst cyanide detoxification reaction	900
Flocculant	Pre-leach thickener Pre-detox thickener	Flocculate solids to assist in solid/liquid separation	1,700

Table 17-3: Grinding media and area of use

Media	Area	Consumption (mt/y)
5-in forged steel ball	SAG mill	4,504
3-in forged steel ball	Ball Mill	7,334

Carbon is consumed regularly through abrasion in the CIL tanks and in transfer pumps, and by thermal disintegration from regeneration, etc. Carbon will be delivered by truck in 1,100 lb (500 kg) bulk super sacks. Before it can be used in the process, fresh carbon must be wetted and abraded in an attrition tank.

Caustic soda (50%) and hydrochloric acid (35%) will be used in the carbon stripping process. Caustic soda will be supplied by bulk tanker and the acid will be delivered in rubber-lined ISO containers. The individual feed lines will be equipped with flow meters and control valves to ensure that the appropriate dosages are achieved.

Sodium cyanide (NaCN), used in the cyanide leaching (CIL and ILR) and carbon stripping processes, will be delivered in briquette form in bulk super sacks or boxes. The briquettes will be dissolved in water and the solution will be stored in a tank, from which it will be distributed by pumps to the appropriate process areas.

Oxygen is fundamental for gold leaching. At the moment of the present report it is not clear whether oxygen has better leaching performance than air (source of 20% of oxygen v/v), but based on the recent BBA trade-off studies it was found that the operating costs of an oxygen plant are typically lower than an air blower system because of lower maintenance costs. Based on this finding, BBA recommends the implementation of a vacuum pressure swing adsorption (VPSA) oxygen production plant to meet the oxygen requirements of the Project. Oxygen will then be bottom-sparged to the pre-treatment tank and CIL tanks. A liquid oxygen back-up system will be available when the VPSA plant is not in operation.

Flocculant will be required for the pre-leach and pre-detox thickeners. It will be delivered in solid form and dissolved in batches in a mixing tank using process water. The batch will then be pumped to a storage tank, from where the reagent will be continuously metered into the thickener feed slurry.

Copper sulphate ($CuSO_4$) will be delivered in solid form in 2,200 lb (1,000 kg) super sacks and dissolved in batches in a mixing tank using fresh water. The batch will then be pumped to a storage tank, from which the reagent will be continuously pumped to the cyanide detoxification tank.

Quick-lime (CaO) is used in the cyanide detoxification and cyanide leaching processes (pre-aeration and CIL tanks). It will be delivered by truck in bulk containers and transferred pneumatically to a lime storage silo. A screw conveyor will transfer the quick-lime from the silo to the lime slaker, where it will be wetted with water. The slaked lime will pass through grit separators and into the quick-lime mixing tank, to which more water will be added to create slurry of the appropriate density. Grit will be removed from the separator and disposed of using a screw conveyor. The hydrated lime slurry will be pumped continuously from the mixing tank to the appropriate process areas.

Lead nitrate ($PbNO_3$), used in cyanide leaching, will be delivered in 2,200 lb (1,000 kg) bulk super sacks. Lead nitrate is dissolved in batches in a mixing tank. The solution will then be pumped to a holding tank, from which the reagent will be continuously pumped to the pre-conditioning tank.

The Inco SO_2/air process will be used for cyanide detoxification. The sulphur dioxide (SO_2) will be generated using a sulphur burner that will burn elemental sulphur delivered to site in 2,200 lb (1,000 kg) bulk super sacks, which will be transferred to a storage silo. The SO_2 will be produced on demand and will be delivered to the cyanide detoxification tanks.

Sodium Metabisulfite ($Na_2S_2O_5$) will be the back-up source of SO_2, when the sulphur burner plant is under maintenance. Sodium metabisulfite will be received in 2,200 lb (1,000 kg) super sacks. Each sack will be emptied into the sodium metabisulfite mix tank and mixed with fresh water to 34 wt% and then transferred to the sodium metabisulfite solution holding tank. Sodium metabisulfite solution will be pumped by the sodium metabisulfite distribution pump to the cyanide detox reactor tanks.

Copper sulphate ($CuSO_4$) and quick-lime (CaO) will be added to control the chemical reaction at the cyanide detoxification reactors. The detoxified slurry will then be pumped to the tailings management facility.

Refining flux will be delivered to site in bags or buckets. Pre-mixed fluxes are mixed with the dried electrowinning sludge to adjust the chemistry of the material for refining. The proper flux mix and quantity, based on the electrowinning sludge chemistry, will be established by the smelting flux supplier during the first months of operation.

17.7 Ancillary facilities

The process plant building will house various maintenance facilities including shops for mechanical, electrical and instrumentation repairs. Equipment requiring specialized maintenance or major rebuilds will either be dispatched to shops in the Fairbanks area or back to their suppliers.

Other facilities within the process plant building include a centralized control room located near the grinding area, metallurgical and sample preparation laboratory, change-rooms (dry), lunch room, as well as offices, conference and training rooms.

17.8 Process plant controls

A plant control system with open architecture and a unique platform will be used. The main communication backbone will be provided by redundant Ethernet fiber optic cables. Where equipment is supplied as a packaged unit, the vendor packages will have standardized controllers that will communicate with and be controlled by the plant network.

The control system will include operator workstations with historian software to enable reporting of plant data, calculations, statistical analysis of process data, and to allow for metallurgical optimization of the plant operations.

An information system and an information management system will allow certain staff to monitor the process and the variables from their PCs, connected to the management information platform.

Monitors will be installed for a closed-circuit television (CCTV) system, a calling and searching system, fire protection system, centralized panel, and other dedicated systems that require monitoring or controlling by the operator.

17.9 Process water

Process water is distributed throughout the processing facility to dilute streams to the necessary solids pulp density that is required in each unit operation. The majority of the process water is reclaimed from the CIL thickener overflow stream and from the tailings pond via the reclaim pumps. The fresh water required for the process is taken from the wells located on the north side of Livengood valley. Based on the preliminary mass and water balance, fresh make-up requirements will be approximately 286 gpm (65 m^3/h)

17.10 Energy requirements

The total operating power demand for the process plant will be approximately 43 MW. The crushing and grinding circuit represents approximately 70% of the total operating power used by the plant. The processing power demand is shown in Table 17-4.

Table 17-4: Process plant power demand by area

Area	Power Demand (MW)
Crushing and Grinding	30
Balance of Plant (Gravity, CIL, and Tailings)	13
Total Power Demand	**43**

Liquefied natural gas will be used for heating within the process plant building.

17.11 Process plant arrangement

Figure 17-3 presents a conceptual layout of the proposed Livengood process plant and ancillary facilities. It will be located approximately midway between the open pit mine to the south east and the tailings management facility (TMF) to the north. The process plant comprises four main buildings: the primary crushing building, the covered stockpile area, the secondary/pebble crushing building and the main process plant building. Secondary process facilities such as the thickeners, reagent silos and the sulphur burner are in proximity to the main process building.

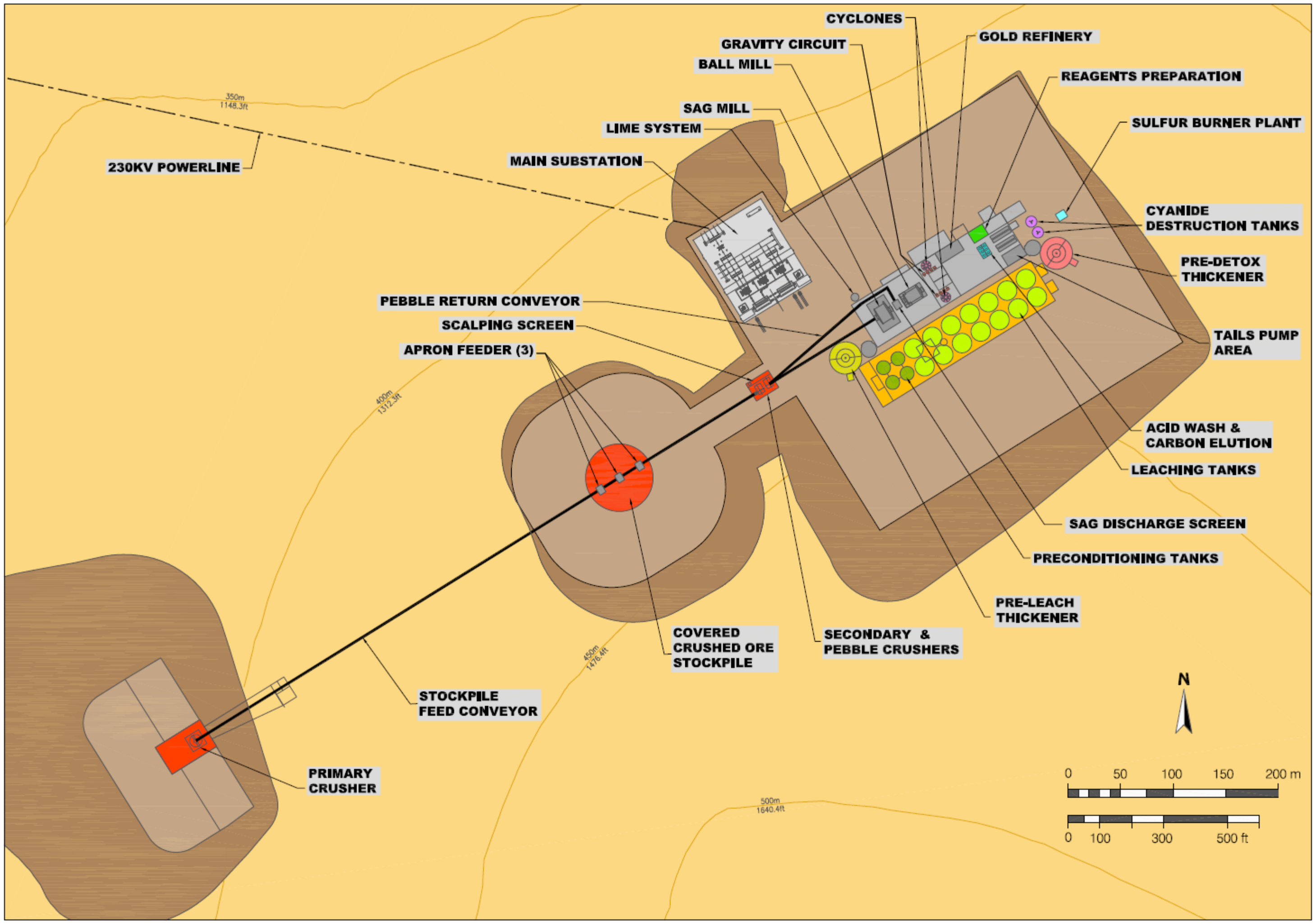


Figure 17-3: Process plant general arrangement

17.12 Process plant personnel

A total of 140 employees are required in the process plant, including 26 salaried staff and 114 hourly workers divided into management and technical services, operations and maintenance departments.

Table 17-5 and Table 17-6 present the salaried and the hourly manpower requirements, respectively, for the processing plant.

Table 17-5: Process plant salaried manpower

Position	Number of Employees
Mill manager	1
Mill secretary / clerk	1
Mill operations supervisor	4
Safety trainer & coordinator	1
Chief metallurgist	1
Metallurgist	2
Chief assayer	1
DCS engineer	1
Process data analyst	1
Mill maintenance superintendent	1
Electrical superintendent	1
E&I supervisor	2
Electrical engineer	1
Electrical maintenance planner	1
Mechanical engineer	1
Mill maintenance supervisor	2
Mill maintenance planner	2
Crusher supervisor	1
Tailings supervisor	1
Total – Salaried	**26**

Table 17-6: Process plant hourly manpower

Position	Number of Employees
Mill Operations	
Primary crushing operator	4
Crusher / conveyor helper	4
Mill control room operator	4
Grinding operator	4
Grinding helper	4
Gravity operator	4
Gravity helper	4
Leach / CIL operator	8
Stripping operator	4
Refiner	4
Detox operator	4
Tailing operator	4
Reagents operator	4
Metallurgical technician	2
Assayer	4
Sampler	4
Sub-total	**66**
Mill Maintenance	
Millwright	20
Mechanic / welder	16
Elect. / Inst.	4
Electrician	4
Instrument technician	4
Sub-total	**48**
Total – Hourly	**114**

18. PROJECT INFRASTRUCTURE

18.1 Introduction

The Livengood Gold Project area is located approximately 70 mi (113 km) by road (47 mi (75 km) by air) northwest of Fairbanks and is accessed by state Highway 2 (Elliott Highway), which provides paved, year round access from Fairbanks. The property is adjacent to the Trans-Alaska Pipeline System (TAPS) corridor, which transports crude oil from the North Slope south and also contains a fiber optic communications cable. A second fiber optic cable runs parallel to the Elliott Highway. 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 General site arrangement

To the extent practicable, the infrastructure facilities for the Project have been designed to avoid or minimize impacts to wetlands by avoiding direct use of the Tolovana River watershed and by establishing a footprint as compact as possible within the historically mined Livengood Creek basin. The Project site has been configured for optimum construction access and operational efficiency as well as to take advantage of the existing roads and infrastructure.

The Project envisions construction of the following key infrastructure items:

- Access light vehicle and mine haulage roads;
- O'Connor Creek substation and 50 miles of new 230 kV transmission line;
- Process plant and ancillary buildings;
- Administration, dry, maintenance, and warehouse complex;
- Mine truck wash and fueling facilities;
- Bulk fuel storage and delivery system;
- Water and sewage treatment;
- Emergency generators;
- Fresh water pumping and distribution system;
- Waste rock, ore and growth media stockpiles;
- Communications and information technology networks;
- Mine tailings and water management facilities;
- Temporary construction camp;
- Fairbanks employee parking area.

18.3 Power demand

The total power demand of the Project is estimated to be approximately 55 MW, including network losses and a 5 MW contingency, based on the connected loads, load and efficiency factors and operating availability. Table 18-1 shows the estimated power demand breakdown by sector. The power demand was calculated using a factored approach based on the largest installed loads (primary and secondary crushers, SAG mill, ball mill and pebble crusher motors) as well as past project experience for the remaining areas. The projected annual energy use is estimated to be approximately 365,000 MWh.

Table 18-1: Estimated total project power demand

Area		Power Demand (MW)	% Site
Open Pit Mine		0.8	1
Process Plant	*Crushing and Grinding*	*30.0*	*55*
	Balance of Plant	*13.0*	*24*
	Total	43.0	79
Infrastructure Facilities		2.0	4
Tailings and Water Management		3.0	5
Network Losses		1.0	2
Contingency		5.0	9
	Site Total	**54.7**	**100**

18.4 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.

In 2012, to support the FS, 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 up to 100 MW of power, if required. To supply the 55 MW required for the PFS configuration, the additions and modifications to the electrical system that will be required include:

- A new substation at O’Connor Creek;
- GVEA transmission system upgrades;
- 50 mi (80 km) 230 kV transmission line.

18.4.1 O'Connor Creek substation

A new 138/230 kV substation at O'Connor Creek (OCS) will be required to connect the Livengood transmission line to the GVEA system. The OCS will contain two 100/150 MVAR transformers, each of which will be capable of transmitting the Livengood load. GVEA has obtained a lease from the Fairbanks North Star Borough for the land parcel required for the substation. The substation will be a 3-ring bus configuration and will step up the voltage from 138 kV to 230 kV for transmission to Livengood.

18.4.2 GVEA transmission system upgrades

GVEA currently provides 138 kV service to the Fort Knox mine through the Fort Knox transmission line that connects to the grid at the Gold Hill substation. The OCS will be built adjacent to and connect to the Fort Knox transmission line. Since a large part of the GVEA generation will be coming from their facilities near North Pole, Alaska, located approximately 15 mi (24 km) southeast of Fairbanks, upgrades to the GVEA transmission system will be required. Upgrades to the GVEA transmission system include double circuiting approximately 15 mi (24 km) of existing line and replacing 18 mi (29 km) of various sections of 69 kV and 138 kV lines with new 138 kV transmission line.

18.4.3 230 kV transmission line

Dryden & LaRue completed the design for the 50 mi (80 km) 230 kV transmission line. The route generally consists of flat to gently rolling terrain. It follows the Trans-Alaska Pipeline (TAPS) for the first 42.5 mi (68.4 km) from O'Conner Creek substation, crossing it three times. The route then traverses north and east away from the TAPS corridor for 7.25 mi (11.7 km) to the mine site substation.

The preliminary design is based on constructing the 230 kV transmission line with wood H-frame structures with guyed angle and dead-end structures. Wood poles will be directly embedded with native backfill where favorable soils exist. Where ice-rich permafrost or swampy conditions exist, driven pile foundations will be used to support the wood poles. The transmission line would be permitted in conjunction with the Project, would be constructed by THM, and operated by GVEA. EPS determined that a 25 MVAR Static Var Compensator (SVC) is required at the Livengood mine site substation to modulate the transient effects of the Project to GVEA specifications.

18.5 Site electrical distribution

The main substation at the plant site will consist of one 230 kV primary circuit-breaker and three main 230 – 13.8 kV, 42/56/70 MVA, outdoor transformers. Each transformer is connected to the main breaker via a 230 kV disconnect switch to allow its isolation from the network. The substation will distribute power to the plant at 13.8 kV, 60 Hz from switchgears installed in two separate pre-fabricated buildings located in the main substation. The main loads, each at 13.8 kV, are dedicated to the SAG and ball mills. This equipment will be driven by low-speed synchronous motors that will be run by active front end drives complete with their own transformers. One remote substation related to the process will be built: a 13.8 kV-600 V, 500 kVA substation for the fresh water pumping station.

The electrical distribution to the site infrastructure (security gate, mine garage/administration complex and other facilities) will consist of a dedicated 13.8 kV overhead line distribution network. Approximately 10 mi (16 km) of aerial lines will supply all of the infrastructure loads.

18.5.1 Emergency power

A group of two 2,500 kW diesel engine driven generators will serve as the emergency electrical power source for the whole plant, providing power at 4.16 kV. The generator sets will provide backup power to the plant for selected process loads that need emergency power to allow an orderly shutdown of the process in case of a main power failure or to simply maintain them in operation if they are critical.

Five smaller generators at 600 V with automatic transfer switches (2 x 1,200 and 3 x 250 kW) will provide backup power to the plant control system, critical remote 600 V loads and the security system.

Generators purchased for the construction camp will be used as backup generators for the infrastructure area once the operations phase commences. No emergency power capacity is planned for the mine area.

18.6 Site access

The main road and security gate to access the site will be located near the existing Alaska Department of Transportation facilities. Site access will be controlled with a guard/security house located at the entrance to the site on the main access road. The guard house will be a modular, pre-fabricated wood-frame building, with separate entrance and exit doors, potable water cooler (bottled), and a small toilet and sink connected to a pumpable holding tank. Visitor car, and bus and truck parking bays will be provided next to the guard house. The security gate will be manned full time and is equipped with a weigh scale to monitor delivery of all bulk items required by the operation.

18.7 Site roads

The Project site is very well situated and will make use of existing roads when possible. Project site roads will consist of both light vehicle access roads and mine truck haul roads. These roads will be constructed during the initial construction of the Project with adjustments to the alignments and profiles during the operational years as facilities change in size and shape. The on-site roads will be constructed of crushed waste rock available from site and other available materials. A dedicated mobile aggregate crushing plant will be utilized for the entire life of project, including the period post ex-pit operation, when stockpiles are being reclaimed, to provide aggregate for continually resurfacing haul roads.

18.7.1 Light vehicle roads

Site roads are light vehicle access roads located throughout the Project site. Approximately 12 mi (20 km) of new site roads are planned to be constructed. These roads are designed to provide access to the tailings facility, tailings pipeline, fresh water wells, primary crusher and process plant facilities, administration/garage complex and the storm water pumping stations. These site roads have been designed with a two-way travel width of 26 ft (8 m) and 3 ft (1 m) high safety berms along each road shoulder. If necessary, transit of these roads by large vehicles will be by controlled one-way traffic.

18.7.2 Mine haul roads

Mine haul roads will be built to connect the open pit to the primary crusher, the mine garage, the tailings management facility and the various low grade blending and waste rock stockpiles. The haul roads have been designed for a two-way travel width of 100 ft (30.5 m) and 6.5 ft (2.0 m) high safety berms along each road shoulder, which is suitable for the 320 t (290 mt) class trucks planned for Project use. Over the life of the mine, approximately 4 mi (6.4 km) of haul roads (ex-pit) will be built.

18.8 Process plant

The process plant area consists of the primary crushing facility, covered stockpile, secondary/pebble crushing and main process plant building. The main process plant enclosed structure is approximately 165 ft (50 m) wide by 560 ft (170 m) long, and will house the grinding area (SAG and ball mills), carbon stripping, electrowinning, refining and reagent preparation areas as well as tailings pumps, mechanical services, maintenance areas, offices and the metallurgical laboratory. The pre-leach thickeners, CIL leach tanks, pre-detox, detox cyanide destruction tanks, and lime and sulphur burner facility are to be located outside, around the process building. The process building will be heated with liquefied natural gas (LNG).

18.9 Administration and mine services facility

Due to the sub-arctic conditions experienced at the Project site, the administration offices, mine dry, truck-shop and warehouse will be housed in a single continuous two-story structure with a dimension of approximately 230 ft (70 m) wide by 525 ft (160 m) long located on a large pad area.

This combined mine facility is designed as a permanent building with an expandable maintenance bay structure that will accommodate the addition of mining vehicles over time. A symmetrical design allows for repair bays to be added in pairs. The fleet bay dimensions, bay door sizing and overhead crane lifting capacities (70 t / 63 mt) are all based on a fleet of 320 t (290 mt) class mining trucks. All vehicle bays will have the same dimensions to allow for operations flexibility. As with the process facilities, this building will be heated with liquefied natural gas (LNG).

18.9.1 Lube storage and distribution

The mine fleet shop will be equipped with an enclosed lube storage and distribution system for mine fleet maintenance. The storage area will consist of multiple vertical steel tanks sized according to their consumption rate and located within a containment dike. The storage area will be placed alongside the mine garage facility, and the large used oil and coolant tanks will be located outdoors for ease of access and servicing. A long-range overhead dispensing and evacuation system for transfer of oil, grease, transmission fluids, cooling fluids, windshield washer, service water and compressed air is planned for the mine fleet shop.

18.9.2 Warehouse and storage

The warehouse storage facility is located adjacent to the mine fleet shop, with direct access between the sections, to facilitate heavy component transfer and increased productivity. Warehouse storage requirements will be defined according to the type of storage required. The assumption is made that only one set of major components will be housed at the site and sufficient “rolling” storage will be provided. Moreover, an exterior cold storage area has been allocated adjacent to the building.

18.9.3 Mechanical workshop

The mechanical workshop will also serve as a light vehicle maintenance bay, and be equipped with a 10 t (9 mt) overhead crane and small bay doors. An allowance for equipment has been included in the capital cost estimate.

18.9.4 Administration offices

The mine offices are an integral part of the mine garage facility and are located on two floors. The mine offices will include sufficient closed offices and workstations to accommodate the administration group, mining operations, mine maintenance, environmental, technical services and warehouse personnel.

18.9.5 Employee dry

The mine dry facilities will consist of locker rooms and shower facilities for both men and women. Each mine employee will be assigned two distinct lockers. The shower facilities will be sufficient to handle the shift crossover.

18.10 Other structures

Additional surface infrastructure facilities are located on the same pad area near the administration and mine services facility. The facilities will be positioned on the pad to ensure free and safe movement of the heavy vehicles and will include a "ready line" parking area for the mine haul trucks. These additional facilities include the following:

- Truck Wash Facility:
  - The wash facility will accommodate the mine trucks and auxiliary vehicles. This facility will have a specialized truck wash system, which will include a mud settling basin, oil separator, and water filtration and recirculation system to reduce overall water consumption.
- Diesel Fuel Storage:
  - Diesel storage will consist of ten 13,000 gal (50,000 L) tanks, providing up to an average of seven-day storage capacity, based upon 24 hrs/d operation for the LOM. The tanks are double walled and self-contained with leak detectors.
- Fuel Island:
  - The fuel dispensing system for mine fleet vehicles will consist of an open-ended pre-engineered building with high speed dispensers and hose reels. A concrete pad will be installed under the enclosure and will be equipped with a spill catchment. The fuel dispensing area will also serve as a top-off area for engine coolant, oil, grease and windshield washer fluid. For safety and practical reasons, a separate fuel dispensing station for light vehicles will be located nearby. Gasoline usage is minor and will be satisfied by purchase from local retail suppliers. The mine's light vehicle fleet is expected to consist primarily of diesel pickup trucks.

- Sewage Treatment Plant:
  - A skid mounted sewage treatment plant will treat sewage from the administration and mine services facility and process plant. Sludge from the sewage treatment plant will be collected by a vacuum truck and may be transported off site for disposal.
- Potable Water Treatment Plant:
  - A skid mounted potable water treatment plant is planned to supply water to the administration and mine services facility and process plant. The treatment process consists of filtration, chlorination and UV sterilization units to produce potable quality water.
- Emergency services building:
  - The emergency services building is a modular and pre-fabricated wood-frame building. It contains two offices, an examination room, a treatment room, and a waiting room. Within the same complex, a covered garage houses the ambulance and fire truck.

18.11 Communications / information technology (IT)

The internet and phones services will be provided for the Project by a regional internet service provider, utilizing one of the existing networks currently installed near the Trans-Alaska Pipeline (TAPS) corridor. A redundant fiber optic network will interconnect critical site areas including the gate house, administration and mine services facility and the process plant. Telecommunication services for non-critical remote locations will be provided by a wireless network. A hand-held radio system will be used for voice communication between personnel in the field. The site-wide fiber optic network will be utilized by the following systems:

- Process plant control system (process control network and electrical systems);
- Corporate IT (phone and data);
- Operations, maintenance and warehouse management systems;
- Fire detection;
- Video surveillance and access control systems.

The mining operation plans to use mobile mine radios with base stations to communicate between equipment operators and the mine staff. Some equipment will also be equipped with GPS technology to provide accurate location information through Wi-Fi communication.

18.12 Fire protection

An underground fire water distribution network of 14-in pipe, feeding sprinkler systems, with 6-in hydrants will be installed around the process plant and the administration and mine services facility. The network will be supplied by a combined fresh/fire water tank and a dedicated fire water pump with sufficient water to meet demand for two hours.

Each facility will also be protected by manual fire alarm systems and will have portable fire extinguishers located at strategic points throughout.

18.13 Fresh water

Fresh water for potable and process plant use will be sourced from an aquifer system located on the north side of the Livengood valley. Pumping tests and hydrological studies conducted in 2015 (L. Cope, SRK 2016) of the Amy Carbonate unit indicate that the aquifer could support 5 to 10 water supply wells each, producing 500 to 1,000 gpm (1,900 to 3,800 L/m). It has been assumed that 8 wells will be able to support the process plant start-up and operations requirements. Water from the wells will be collected in a storage tank and pumped via a heat traced pipeline across the valley to the process plant pad for further distribution to other areas as required.

18.14 Construction camp

A temporary 1,050 person construction camp will be mobilized for the construction phase of the Project. The camp would have single occupancy rooms in a common bathroom arrangement. The construction camp is planned to include a kitchen, dining complex, offices, recreation room, and laundry and gym facilities. Once construction activities are completed, the construction camp will be removed and sold.

18.15 Personnel transportation

As a permanent camp is not planned for the operations phase of the Project, all personnel will be transported between Fairbanks and the mine site by charter buses. Multiple buses will be required to operate on different schedules to accommodate varying work schedules. All costs related to personnel transportation are covered by the General & Administration operating cost estimate (Section 21.4.5).

18.16 Fairbanks infrastructure

A pre-fabricated guard house, bus waiting building, small receiving area and parking lot for 300 vehicles are planned to be located in the city of Fairbanks.

18.17 Mine ore waste and water management

18.17.1 Waste rock storage area

Non-economic waste rock produced by mining activities at the Project will either be used to construct site infrastructure such as haul roads and the tailings management facility (TMF) or hauled and stockpiled in the Gertrude Creek valley. The current design of the mine waste rock stockpile has a capacity of 320 Mt (290 Mmt). The remainder of the waste rock produced will be utilized in the construction of the infrastructure. Dependent upon the storage requirements for the low grade ore stockpile and subject to further refinement, the facility has the potential to store up to 700 Mt (635 Mmt) of waste rock.

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 waste rock 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 mine waste rock facility. The shear key starter embankment will be lined on the upslope side and pumps will be installed to collect runoff within the storage facility for discharge into the TMF.

18.17.2 Tailings management facility

The TMF has been designed as a fully lined facility to provide safe and secure storage of approximately 450 Mt (408 Mmt) of mill tailings along with a supernatant pond for ore processing solutions. The TMF has expansion potential up to 775 Mt (703 Mmt). Expansions would require evaluations and design modifications to the Gertrude Creek embankment and water management/storm water diversion infrastructure.

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 six 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, with the first expansion being constructed during the first two years of operation. The remaining five expansions will take approximately three years each to construct and will be completed every four years. After completion of the seven TMF phases, the embankment will have an ultimate height of approximately 296 ft (90 m) and a storage capacity of 450 Mt (408 Mmt).

The TMF embankment will be the primary structure for the TMF impoundment and 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.

18.17.3 Low grade ore stockpile

Low grade ore (133 Mt or 120 Mmt) will be stockpiled in upper Gertrude Creek during the mine life at a facility with a design capacity of 140 Mt (127 Mmt). Runoff from the low grade ore stockpile will be collected and discharged into the TMF.

18.17.4 Water management

Surface water management structures consist of three surface water management pump stations and one surface water diversion channel. These structures will be used to manage and divert surface water generated from precipitation events and the spring freshet. The pump stations are identified as the Phase 1, Amy Creek and Lucky Creek. The surface water diversion channel consists of upgrading an existing channel on the north hillside of the Livengood Valley. This upgrade consists of increasing the channel width and depth and in some areas, the channel profile will require slight steepening.

The surface water management pump stations will capture and divert some of the stream and precipitation runoff within the Livengood, Amy and Lucky Creek watersheds to reduce run-on into the TMF. They consist of small embankments and basins and are capable of transferring 40% to 50% of the annual surface water from these watersheds to the surface water diversion channel. During Phase 1 of the TMF, only the Livengood pump station will be in operation. From the beginning of Phase 2 through the end of operations, only the Amy Creek and Lucky Creek stations will be in service.

The surface water diversion channel is located immediately uphill from the TMF impoundment with an alignment generally oriented northeast-southwest parallel to the northern boundary of the TMF impoundment. The channel will convey flows generated by storm events up to the 100-year/24-hour storm. The capacity of the channel varies from 90 ft^3/s (cfs) at the upper reaches of the channel, to approximately 500 cfs at the channel termination. The erosion protection varies along the channel alignment, with riprap being utilized for the majority of the channel, grass/vegetation lining near the upper limits of the channel, and the Myrtle Creek drop structure, armored with grouted riprap, which will convey the water to the bottom of the valley west of the TMF embankment.

19. MARKET STUDIES AND CONTRACTS

19.1 Introduction

The Livengood Gold Project will produce gold in the form of doré bars. Neither BBA, nor Tower Hill Mines (THM) has contacted any precious metal refineries for competitive treatment bids for the doré expected to be produced by the Project.

19.2 Market studies

Gold is a freely traded precious metal commodity on the world market, for which there is a steady demand from numerous buyers. The gold market is global in nature and is unlikely to be affected by production from the Project. Neither BBA, nor THM have conducted a market study in relation to the doré that will be produced by the Project.

Due to its widely traded nature, it is not difficult to determine the market value of gold at any particular time. Gold doré bullion is typically sold through commercial banks and metals traders with sales price obtained from the World Spot or London fixes. These contracts are easily transacted and standard terms apply. BBA expects that the terms of any sales contracts would be typical of, and consistent with, standard industry practices and would be similar to contracts for the supply of doré elsewhere in the world. Limited additional effort is expected to be required to develop the doré marketing strategy.

19.3 Gold price projections

The gold price of $1,250/oz (base case) used within the financial model (Chapter 22) to estimate revenue from the Project is a consensus price derived from bank analysts long term forecasts, historical metal price averages and prices used in publically disclosed comparable studies that were deemed to be credible. The forecasted gold price is kept constant and is meant to reflect the average metal price expectation over the life of the Project. It should be noted that metal prices can be volatile and that there is the potential for deviation from the LOM forecasts.

19.4 Contracts

There are no refining agreements or sales contracts currently in place for the Project that are relevant to this Technical Report. BBA expects that terms contained within any sales contract that could be entered into would be typical of and consistent with standard industry practices, and be similar to contracts for the supply of gold elsewhere in the world. In the opinion of Colin Hardie, QP, THM will be able to market gold produced from the Project.

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

None of the aforementioned companies have been contacted by THM to provide a competitive treatment bid.

This pre-feasibility study assumes a refining, transportation and insurance charge of $8.05/oz of doré and payable terms of 99.5% for gold content.

20. 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 100 years. Portions of the resource area on Money Knob have also hosted intermittent hard rock 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 Chapter 4). 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 (2008-2016)

Baseline Study	2008	2009	2010	2011	2012	2013	2014	2015	2016
Surface Water									
Surface Water Quality		●	●	●	●	●	●	●	●
Sediment Quality						●	●	●	●
Hydrology									
Surface Water Flow and Snow			●	●	●	●	●	●	●
Hydrogeology			●	●	●	●	●	●	●
Groundwater Quality			●	●	●	●	●	●	●
Hydrogeological Modeling			●	●	●	●	●	●	●
Permafrost Studies			●	●	●	●	●	●	●
Wetlands & Vegetation									
Wetlands Delineations		●	●	●	●	●	●		
Meteorology & Air Quality									
Meteorological Data			●	●	●	●	●	●	●

Baseline Study	2008	2009	2010	2011	2012	2013	2014	2015	2016
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 20.6)				●	●	●			
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 and around 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. Monitoring has continued since 2013 at three stations located on the Tolovana River and the West Fork of the Tolovana River. 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 and regional mineralization.
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 has maintained stream gauges in the Project area since 2010. Snow surveys have been completed in a variety of aspects, elevations, and vegetation types in late spring 2010-2016. Three years of surface flow data have been collected from Lower Amy Creek and the first year of data collection from Livengood Creek in the vicinity of the ADOT maintenance facility is underway. Regional data sources were 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, which forms the basis of the water balance model.

Baseline Study	Program Summary
Hydrogeology	
Groundwater Quality	THM has sampled 54 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 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 northeast of the resource area. The hydraulic conductivities observed down-gradient from the proposed pit and in the rocks of the Livengood Valley are relatively high. The lowest hydraulic conductivity values were observed to the north and east of the resource area. 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 (>-2ºC) and discontinuous. Permafrost depths at the Project have been measured to reach nearly 600 ft (183 m) below ground surface.
Wetlands and Vegetation	A 62,000-acre (25,090 ha) wetlands map of the Project area and power line corridor was completed in December 2013. This mapping will form the basis for wetlands minimization, avoidance, and mitigation during mine design and permit application preparation. Approximately 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 13 invasive species found within the study area, most of the populations are relatively small. The control and containment of these species will be considered during the 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, northeast of the resource area, and has collected 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 has collected the same meteorological parameters as well as seasonal evaporation data. Two fine particulate matter ($PM_{2.5}$) meters were co-located with this station to monitor ambient air quality in 2011. In 2013, an all-season precipitation gauge was installed at the ADOT maintenance facility in the Livengood Creek Valley.
Aquatic Resources	
Resident Fish Surveys	As the most populous fish in the Project area, young of the year Arctic Grayling were targeted for full-body tissue analysis. Fish tissue sampling was conducted from 2009-2012. Tissues of the resident fish in the area contain detectable metals concentrations, as do many regional streams in naturally mineralized areas. 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

Baseline Study	Program Summary
	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.
Bio-monitoring	Macro-invertebrate sampling was conducted in 2009-2012; periphyton sampling was conducted in 2009-2016. 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.
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. Seventy-five humidity cell tests have undergone multi-year testing. Samples from the datasets have also been tested for meteoric water mobility potential (MWMP) and sequential MWMP. Twenty-eight 550 lb (250 kg) barrels of resource and regional materials are also undergoing on-site multi-year testing to establish scalability factors. The data indicates that certain stratigraphic units are potentially acid generating (PAG), while other rock types are non-PAG. Several rock types have metal leaching (ML) potential, with arsenic, antimony, and selenium being of primary interest. Mineral content and ARD potential tend to decrease outside the resource area. Management of these materials is discussed in Section 20.1.3.
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 forests.
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 to be nesting.
Cultural Resources	Cultural resource surveys have been completed on nearly 16,000 acres (6,475 ha) of the Project area and 5,000 acres (2,023 ha) of the power line corridor. To date, 124 historic features and 21 prehistoric sites have been 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. 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 and summer noise monitoring was completed in March 2013 and July 2013, respectively. Seven locations were monitored employing two different techniques (short term and 24 hour).

20.1.3 Environmental management strategies

Tailings Management Facility – The TMF has been designed to safely contain process plant 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 and restricts communication with the deep groundwater.

Mine Waste Rock Facility – 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 water that infiltrates 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 Project is to incorporate as many environmental considerations into the initial design process as possible. These considerations are reflected in the PFS design and include the characterization studies of the mine waste rock and overburden, process plant tailings and water that have been underway since 2009.

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 34-year period, starting in production Year 21 with the construction of a water treatment plant, and ending in Year 54 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 relevant government 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 Project will need to prepare a U.S. Army Corps of Engineers Compensatory Mitigation Plan for mitigating unavoidable wetlands impacts that will include input from many reclamation and mitigation banking 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 activities and will take approximately five years to complete.

Water Treatment Plant

A 5,500 gpm (1,249 m^3/h) water treatment plant will be constructed during Mine Year 21 and 22 to treat water removed from the TMF supernatant pond and seepage from the TMF underdrain system and the mine waste rock stockpile 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 (114 m^3/h) units, so that over time, as the treatment requirements reduce, modules can be taken out of service.

Tailings Management Facility

A dry closure of the TMF has been incorporated into its design. The supernatant pond will be removed and treated. Four years will be required to place a 3 ft (0.92 m) thick layer of mine waste rock over the entire tailings surface. A 1.5 ft (0.46 m) 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.

Mine Waste Rock and Ore Stockpiles

The mine waste rock stockpile has been designed to minimize the impacts from potentially acid-generating waste rock. During closure, the waste rock will be contoured, covered with 1.5 ft (0.46 m) 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. Any runoff 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 growth media, seeded and fertilized. Pre-construction drainage patterns will be restored or enhanced to minimize storm water impacts. Safety berms will be dozed over the road slope or into road ditches to further enhance drainage.

20.2.2 Post closure activities

The post closure period includes six 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. This list is based on government agency guidance and past Alaskan mining project development experience.

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 permitting 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	CWA Section 404 Permit (wetlands dredge and fill)
	Section 106 Historical and Cultural Resources Protection
U.S. Environmental Protection Agency	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	Notice of Controlled Firing Area (Blasting)
	Structure Warning Lights
Federal Communication Commission	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
	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)

Agency	Authorization
	Water Appropriation Permits
	Road Right of Way/Access
	Power Line Right of Way
	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)
	Storm Water Pollution Prevention Plan (SWPPP) Review Approval
	Oil Discharge Prevention and Contingency Plan Review Approval
	Plan Review and Approvals to Construct and Operate a Public Water Supply System
	Plan Review and Construction Approval for Domestic Wastewater System
	Solid Waste Management Permit
	Food Establishment Permit
	Air Quality Construction Permit (first 12 months)
	Air Quality PSD Permit
	Air Quality Title V Operating Permit
Alaska Department of Fish & Game	Fish Collection, Habitat, and 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	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	Health Impact Assessment
Other Entities	
Alyeska Pipeline	Trans- Alaskan Pipeline System(TAPS) Right of Way (ROW) access/crossing approvals

The proposed preliminary project execution plan for the development and construction of the Livengood Gold Project summarized in Chapter 24 incorporates the permits previously noted in Table 20-3.

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

A mine closure plan featuring dry closure of the tailings management facility has been developed. Closure costs track reclamation and closure expenses from Year 21 through 55. The reclamation and stabilization effort occurs from Year 22 through 32 and includes deconstruction of the facilities and closure of the tailings management facility, mine waste rock facility, roads and water storage reservoirs as described in Section 20.2. These costs total $281.9M, including contractor indirect costs. Subsequent post-closure costs incurred during Years 33 through 55 include pumping, water treatment, maintenance and post-closure monitoring. These costs total $59.8M. Year 55 is the last year with planned closure expenses.

The total closure cost is $341.7M, which is applied to the cash flow in Year 24. This cost, which includes indirect costs, includes closure of the mine waste rock stockpile, tailings management facility, solid waste landfill and ancillary facilities.

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 $9.1M during Years -2 through 24. The model includes trust fund earnings at 3.0% annual percentage rate (APR), applied to the fund balance until closure is complete in Year 55.

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. The district has produced over 500,000 oz of placer gold, with two-thirds of that production coming prior to World War II. In 2016, there were three 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 2013, the Census Area held a total population of 5,650 widely dispersed residents in 20 communities, of which approximately 70% were Alaska Natives. Both Minto, which is approximately 40 mi (64 km) from Livengood, and Manley Hot Springs, approximately 80 mi (129 km) away from the Project, 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 System (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, operation of a local refinery and the operation and maintenance of TAPS. Today, the University of Alaska, the Fairbanks Memorial 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 38,150 non-agricultural wage and salary jobs in 2012. In 2011, using decennial census data, 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 as the primary land use for the 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 Department of Labor, will be used to attract, train and retain an Alaskan workforce for the various construction and operating jobs available.

The PFS estimates a total of 6.8 M man-hours during Project construction, with a peak construction workforce of 1,050. The average wages of those workers is estimated at $40.00/hr. During the two years of preproduction mine development, the Owner's crew will be approximately 175 employees. During operation, the peak employee count is estimated at 387 and an annual average wage of approximately $100,000/y. Total annual wages paid during operations is estimated to be $32M.

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 FNSB. 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 $175M on consumables, supplies and purchases.

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 355 residents in 2013. 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 Project will be designed to promote safety, environmental stewardship, efficient production, and local hire.

21. CAPITAL AND OPERATING COSTS

The capital and operating cost estimates presented in this study are based on the development, construction and start-up of an open pit mine, process plant and tailings management facility capable of processing on average 52,600 t/d (47,700 mt/d) of gold bearing material. All capital and operating cost estimates cited in this report are referenced in nominal third quarter 2016 United States dollars. No provisions have been included to offset future escalation.

21.1 Capital cost summary and basis

THM 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). BBA consolidated the cost information from all sources to determine the overall project capital cost.

Table 21-1: Capital cost estimate contributors

Scope / Responsibility	Contributor(s)
Mine Equipment and Development	Tim George (W&B)
Process Plant & Ancillary Facilities	Colin Hardie (BBA)
Surface Infrastructure and Buildings	Colin Hardie (BBA)
Waste Rock and Tailings Management Facility	Colin Hardie and Ryan Baker (BBA and NewFields)
Electrical Line and Substations	Colin Hardie (BBA)
Indirect Cost	Colin Hardie (BBA)
Owner's Cost	Colin Hardie (BBA)
Reclamation and Remediation	Ryan Baker (NewFields)
Contingency	All

The total estimated preproduction capital cost (-20% / +25%) to design, procure, construct and commission the Livengood Gold Project facilities, including funding of reclamation activities, is estimated to be $1.84B. The estimated sustaining capital cost required by the Project is $866M. This estimate includes the addition of certain contingencies and indirect costs. The cumulative life of mine capital expenditure (preproduction and sustaining capital) is estimated to be $2.501B. Table 21-2 summarizes the initial capital and sustaining capital costs by major area.

Table 21-2: Initial capital and sustaining capital costs by major area
($ Millions)

Cost Item/Area	Initial ($M)	Sustaining ($M)
Mine Equipment	173	123
Mine Development	146	0
Process Facilities	446	24
Infrastructure Facilities	454	442
Power Supply	79	0
Owners Costs	307	0
Contingency	213	76
Subtotal before Reclamation	**1,818**	**665**
Funding of Reclamation Trust Fund (1)	18	201
Total	**$1,836**	**$866**

Note: Rounding of some figures may lead to minor discrepancies in totals.
(1) Includes initial funding, total $342M estimated costs. The difference of $123M is projected trust fund earnings.

21.1.1 Accuracy

The overall capital cost estimate developed in this study generally meets the AACE class 4 requirements and has an accuracy range of -20% and +25%. Estimate accuracy ranges are projections based upon cost estimating methods and are not a guarantee of actual project costs. The capital cost estimate of this PFS forms the basis for the approval of further development of the Project by means of a FS.

21.1.2 Assumptions

The capital cost estimate is based on the following assumptions:

- Reflects general accepted practices in the cost engineering profession;
- Assumes contracts will be awarded to reputable contractors on a lump sum basis;
- Craft all-in rates are trade union rates calculated based on an assumed 70-hour work week with two 10-hour shifts worked per day. Rotation for craft and supervision personnel is 20 days on and 10 days off;
- Waste rock generated during the mine pre-stripping will be of suitable quality and quantity to be used as backfill material to construct the tailings management facility and other geotechnical facilities;
- Construction will consist of a mixture of contracted work and work performed by mine personnel;

- The following activities will be performed by the THM owners team (mine personnel) to support the construction of the tailings management facility and other geotechnical facilities:
  - Crushing and screening of waste rock for construction aggregate;
  - Load, haul and placement (spreading and compaction) of rock fill from mine;
- Soil conditions will not require special foundation designs such as piling;
- All excavated material will be disposed of on site;
- Project will adhere to the schedule in construction execution plan as detailed in Chapter 24;
- The estimate assumes that the contingency will be spent.

21.1.3 Exclusions

General exclusions from the capital estimate are as follows:

- Sunk costs (costs prior to a production decision);
- Land acquisition, permitting, licensing costs;
- Allowance for special incentives (schedule, safety, etc.);
- Interest and financing costs;
- Escalation beyond Q3 2016;
- Taxes and import duties;
- Salvage value, except for sale of construction camp;
- Risk due to labor disputes, permitting delays, weather delays or any other force majeure occurrences;
- Issues beyond the control of the Owner.

21.2 Initial capital costs

21.2.1 Open pit mine

The initial capital cost for mine development activities over a 24-month period and acquisition of mining equipment is $319M and summarized in Table 21-3.

Table 21-3: Open pit mine initial capital costs
($ Millions)

Cost Item/Area	Initial ($M)
Mine Development	146
Mine Equipment	173
Total	**$319**

21.2.2 Mine Development

The open pit mining development cost for mine waste rock removal, waste removal, man hours and ore stockpile activities, based on the initial mine plan as described in Chapter 16, is $146M, or $1.27/t based on using the same equipment as required by the operations phase. In total, 86.7 Mt (78.6 Mmt) of waste removal and the stockpiling of 29.1 Mt (26.4 Mmt) of low grade mineralized material will be required during the development phase.

21.2.3 Mining equipment

Open pit mining mobile equipment and ancillary equipment costs were estimated based on recent supplier quotations and W&B's in-house database. The initial mine equipment requirements are based on operating hours and production needs as described in Chapter 16. The mobile support equipment consists of dozers, graders, compactors, fuel trucks and cranes required to support the operation of the mine and waste/tailings management facilities. The initial mine equipment requirements along with the capital costs are detailed in Table 21-4.

Table 21-4: Mining equipment initial capital costs
($ Millions)

Cost Item / Area	No.	Initial ($M)
Rotary Drill (10 in / 251 mm)	4	9
Rotary Drill (7 in / 171 mm)	2	1
Hydraulic Shovel (47 yd^3 / 36 m^3)	2	27
Loader (40 yd^3 / 31 m^3)	1	10
Haul Truck (320 t / 290 mt)	13	71
Mobile Support Equipment	-	39
Portable Crushing Plant	1	2
Communication System	1	1
GPS and Dispatch System (35 units)	1	4
Spare Parts (including buckets and truck beds)	-	9
Total		**$173**

21.2.4 Process plant

The design and capital costs of the crusher area, the crushed ore stockpile area and the process plant has largely been based on BBA's experience on recent projects. To estimate the capital cost of the process plant, BBA used its project cost database, which includes as-built capital costs for a number of similar large gold processing facilities. Based on the proposed plant capacity, preliminary general arrangement layouts and project location, the capital costs were adjusted to match the requirements of the Project. For the major process and mechanical equipment packages, equipment datasheets and summary specifications were prepared and budget pricing obtained from qualified suppliers. Regional data from Northern Canada and Alaska was compared to assess and adjust the labor and crew rates and productivity factors for Alaska based on BBA's standard estimating spreadsheet. The process plant preproduction capital costs are detailed by area in Table 21-5:

Table 21-5: Process plant capital costs by major area
($ Millions)

Cost Item / Area	Initial ($M)
Process Building	106
Primary Crushing	50
Stockpile, Pre-Crushing and Pebble Crushing	58
Primary and Secondary Grinding	72
Gravity Separation	9
Leaching	59
Carbon Stripping and Gold Room	11
Cyanide Destruction and Tailings	20
Reagents	7
Common Services	26
Spare Parts	12
Initial Fills	16
Total	**$446**

21.2.5 Power supply

The capital costs related to the electrical transmission line, O'Connor Creek substation and the Golden Valley Electrical Association (GVEA) system upgrade were estimated by specialized local firms (Dryden & LaRue and Electric Power Systems) and integrated into the estimate by BBA. The main on-site substation was estimated by BBA based on other recent projects of similar size, power rating and layout. Table 21-6 summarizes the initial capital cost estimate for the off-site and on-site electrical facilities.

Table 21-6: Power supply capital costs by major area
($ Millions)

Cost Item / Area	Initial ($M)
230 KV Transmission Line	31
O'Connor Creek Substation	10
GVEA Transmission System Upgrades	19
Primary Substation and Site Distribution	19
Total	**$79**

21.2.6 Infrastructure facilities

The capital cost of infrastructure facilities required by the Project was estimated by BBA and NewFields. BBA estimated the initial capital costs based on the site/building layout drawings, specific project requirements and its in-house database for the following site infrastructure facilities:

- Site preparation and common underground services;
- Site security and main access gate;
- Mine haul and site access roads;
- Mine garage, dry, warehouse and administration complex;
- Mine truck wash and fuel/lubrication facility;
- Office, garage and warehouse equipment;
- Site communications and emergency power;
- Water and sewage treatment;
- Fresh water wells, pumping station and piping;
- Storm water management pumping stations (Amy Creek and Lucky Creek);
- Process plant tailings and water reclaim systems;
- Fairbanks guardhouse, storage and employee parking area (off-site).

NewFields developed the preliminary designs and estimated material quantities for the tailings management facility and related infrastructure such as:

- Livengood Valley and Gertrude Creek TMF starter embankments (lined facility with 29.8 Mt (27 Mmt) storage capacity equivalent to two years of production);
- TMF North access road and pipe corridor;
- Surface water diversion ditches and drop structures;
- Ground water collection systems;
- Growth media, waste rock and ore stockpiles.

The general approach of utilizing mine waste rock from the surface mine delivered by the mine operations to satisfy the major fill requirements for the TMF was employed to maximize savings in construction costs. Based on recent project experience in Northern Canada, BBA assisted NewFields in developing earthwork unit costs and overhead costs using a mixture of contracted work and work performed by mine personnel. To support the TMF cost estimate, budgetary quotes for the supply of the principal purchased materials, such as geosynthetics and piping, was obtained from potential vendors. Table 21-7 summarizes the initial infrastructure capital costs by area.

Table 21-7: Infrastructure capital costs by area
($ Millions)

Cost Item / Area	Initial ($M)
Site Preparation and Common Services	54
Main Control Gate and Access Roads	13
Truck Shop and Administration Building	48
Mine Truck Wash	3
General Infrastructure Buildings and Temporary Facilities	14
Fuel Storage and Dispensing Facility	2
Tailings, Waste Rock and Water Management Infrastructure	305
Site Communications	4
Emergency Power	2
Offsite Infrastructure (Fairbanks Storage and Parking Area)	1
Total	**$446**

21.2.7 Indirect and Owner's costs

For the Project, indirect costs included within the preproduction capital cost estimate, an itemized list of elements has been used to generate factored estimates. The Owner's costs were calculated using BBA's database, data from the 2013 Feasibility Study, THM requirements and adjusted to the specifics of the Livengood Gold Project. The following costs have been covered within the estimate:

- Indirect costs:
  - Construction camp (1,050 rooms) procurement (including resale) and operations;
  - Engineering, procurement and construction management (EPCM);
  - Engineering, construction quality assurance, third party testing and surveying;
  - Construction of temporary facilities, erection and operation;
  - Land and ocean freight for process and major electrical equipment;
  - Pre-operational verifications, commissioning and start-up support;
  - Relocation costs to move the Alaska Department of Transport (DOT) Garage Facilities;
  - Site and process plant mobile and light vehicles;
  - Vendor representatives during construction.
- Owner's costs:
  - Construction insurance;
  - Preproduction employment and training;

- Corporate services and site support operations;
- Environmental monitoring and community development;
- Right of Way (ROW) and land acquisition;
- Legal permits.

It should be noted that costs related to mine and mill initial fills, commissioning spares, start-up and capital spares, normally shown as indirects, are included in their respective facility capital cost areas. Table 21-8 provides a breakdown of the indirect and Owner's costs by area:

Table 21-8: Indirect and Owner's initial capital costs by area
($ Millions)

Cost Item / Area	Initial ($M)
Construction Camp (including resale)	63
Construction Operations Costs	48
Alaska DOT Garage Relocation Costs	18
EPCM Services	71
Sub-Consultants and Third Party Services	6
Land and Ocean Freight	33
Vendor Representatives	2
Site Mobile and Construction Equipment	7
Owner's Costs	59
Total	**$307**

21.2.8 Contingency

Contingency provides an allowance to the capital cost estimate for undeveloped details within the scope of work covered by the estimate. Contingency is not intended to take into account items such as labor disruptions, weather-related impediments, changes to the scope of the Project from what is defined in the study, nor does contingency take into account price escalation or currency fluctuations.

To establish an adequate contingency estimate, BBA along with the other contributors, reviewed the overall capital cost estimate and categorized the major project work items in terms of level of definition and the nature of how the costs were established for labor, materials and equipment. Depending on the level of confidence, contingencies were allocated to each of the work items. Table 21-9 provides a summary of the contingency by major work area. The total contingency cost for the Livengood Gold Project is estimated to be $213M or approximately 13% of the Project's overall direct and indirect costs.

Table 21-9: Contingency by major area
($ Millions)

Cost Item / Area	Initial ($M)
Mine Equipment and Preproduction Work	8
Process Plant, Surface Facilities and Project Indirects	162
Tailings, Waste Rock and Water Management Facilities	42
Total	**$213**

21.3 Sustaining capital costs

The total estimated sustaining capital cost for the Livengood Gold Project is $866M and was developed by BBA, W&B and NewFields. This is the estimated expense required to maintain operations over the proposed 23-year mine life. Sustaining capital costs included are as follows:

- Open pit mining equipment (new and replacements) and spare parts:
  - 2 x Rotary Drills (10 in / 251 mm);
  - 2 x Rotary Drills (7 in / 171 mm);
  - 1 x Loader (40 yd^3 / 31 m^3);
  - 9 x Haul Trucks (320 t / 290 mt);
  - Additional mobile support equipment (loaders, dozers, compactors, fuel trucks, etc.);
  - Spare parts, including buckets and truck beds.
- Phased tailings management facility and water management system upgrades to achieve their ultimate capacity based on the design provided by NewFields;
- Lengthening and relocation of the process plant tailings pumping and pipeline systems;
- Relocation and upgrades to the Amy Creek and Lucky Creek storm water management pumping and pipeline systems;
- Contingency related to the previously listed activities;
- Annual funding of the reclamation trust fund for eventual site closure in Year 23.

Table 21-10 summarizes the sustaining capital requirements over life of mine.

Table 21-10: Sustaining capital costs by major area
($ Millions)

Cost Item / Area	Sustaining ($M)
Process Facilities	24
Infrastructure Facilities	442
Mine Equipment	123
Contingency	76
Subtotal before Reclamation	**665**
Funding of Reclamation Trust Fund	201
Total	**$866**

21.4 Operating cost summary and basis

The operating cost estimate for the Livengood Gold Project includes all expenses incurred to operate the mine and process plant from the start of Year 1 through Year 23 at a daily average production rate of 52,600 t (47,700 mt). The expected accuracy for the operating cost estimate is that of a pre-feasibility study level (+-/ 20%) and does not contain any allowances for contingency or escalation beyond Q3 2016. Any ore excavated during the preproduction period is considered as a capital expense.

Table 21-11: Operating cost estimate contributors

Scope / Responsibility	Contributor(s)
Mine Operations	Tim George (W&B)
Process Plant Operations	Colin Hardie (BBA)
General and Administration (G&A)	Colin Hardie (BBA) and THM

THM engaged various consultants to provide estimation support for various cost portions of the Project that fall within their specialized scope of work (see Table 21-11). Operating costs were estimated using cost models, laboratory testwork, budgetary quotations from suppliers, general knowledge, and recent experience on similar projects. THM, in consultation with BBA and W&B, provided a list of personnel, based on mining and process plant requirements, along with the salaries benefits and bonuses associated with each position.

The three major operating cost (on-site) areas are mining, processing, and general and administration (G&A). Table 21-12 provides the breakdown of the projected operating costs for the Project. The unit costs areas including royalties and smelting, refining and transport costs are shown in terms of total cost life of mine (LOM) per ton mined, per ore ton processed and total cost per ounce of gold produced. The average operating cost, including royalties and smelting/refining fees over the life of mine, is estimated to be $12.95/t ($14.27/mt) milled.

Table 21-12: Total operating cost breakdown (LOM average)

Cost Item / Area	Total ($M)	Average ($/t mined)	Average ($/t milled)	Average ($/oz)	OPEX (%)
Mining (including stockpile reclaim)	1,505	1.73	3.49	223	27
Processing	3,228	-	7.48	477	58
General and Administration	552	-	1.28	82	10
Onsite Mine Operating Costs	***5,286***	**-**	***12.24***	***781***	***95***
Royalties	252	-	0.58	37	4
Smelting, Refining and Transport	54	-	0.13	8	1
Total	**$5,592**	**-**	**$12.95**	**$827**	**100%**

The operating cash costs per ounce of gold vary significantly depending on the mill feed grade, rock type composition, mine strip ratio and stockpiling activities. The annual variation in operating costs per ounce of gold produced can be seen in Figure 21-1. It should be noted that due to the processing of lower grade stockpile material (0.4 to 0.5 g/t Au), the overall operating costs per ounce increase significantly during the later years.

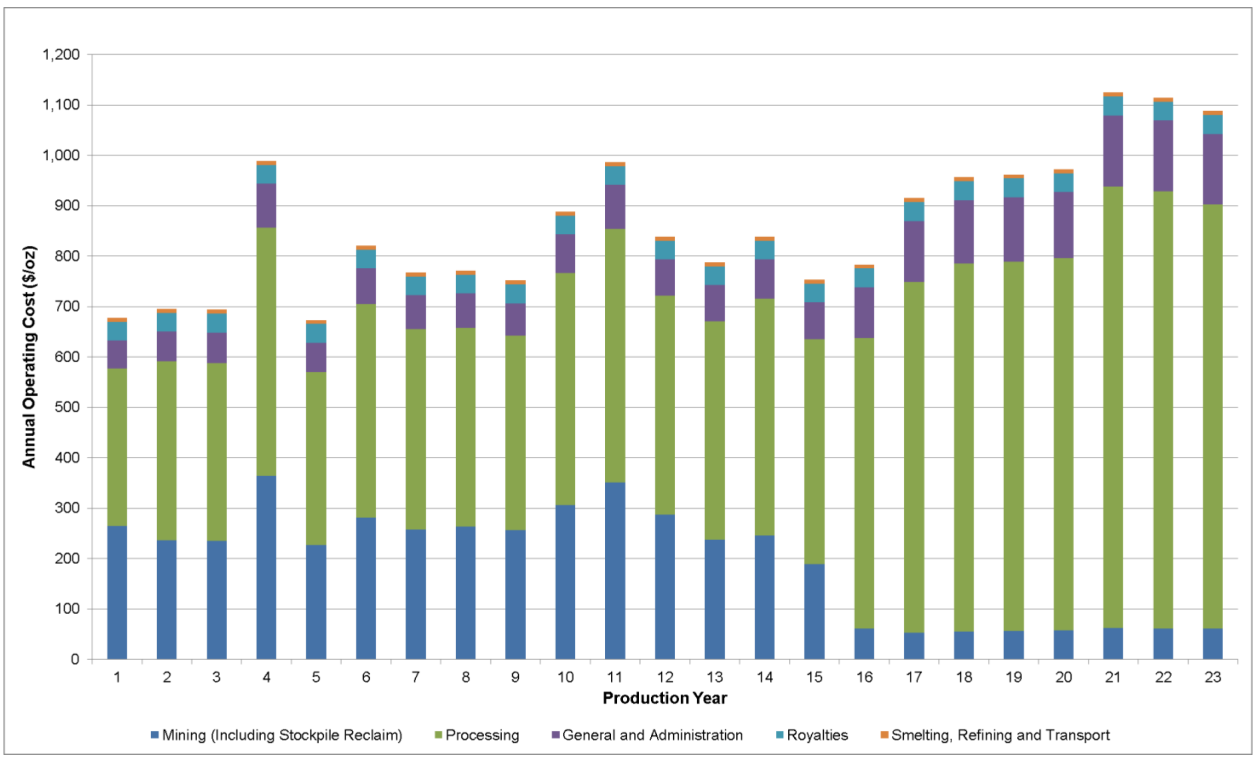


Figure 21-1: Annual operating cash costs ($/oz)

21.4.1 Electricity, diesel and LNG

The cost of electrical power for the Project was estimated based on the GVEA projected 2016 industrial rate of $0.13 / kWh provided by THM. A diesel fuel unit cost of $1.75 / gal was used for estimating the operating costs of the mine and infrastructure mobile equipment. Liquefied natural gas (LNG) is planned to be used as the heat source for the process and ancillary facilities. At present, no LNG supplier has been identified and it is assumed that LNG will be available at the time mine operations commence. A supply unit rate of $9.14 / MMBTU has been used for LNG in this estimate.

21.4.2 Project personnel

The mine and mill are planned to operate 365 days per year, primarily with two 12-hour shifts per day. Various crew schedules will be employed, including crews with 4 days on, 4 days off rotation, crews with 7 days on, 7 days off rotation, and staff with 4 days on, 3 days off. Most General and Administration personnel will work 12-hour day shifts with 4 days on, 3 days off rotation. Personnel will be transported to site from Fairbanks on a daily basis by third party contract highway coach.

The number of employees required by the Project during the production phase (Years 1 to 23) consists of personnel from the open pit mine, process plant and site administration (G&A). On average, over the life of mine, the total number of personnel will be approximately 331. As shown in Figure 21-2, the process plant and general and administrative employees remain fairly constant throughout the mine life, while the mine employees vary on an annual basis due to changes in operations and maintenance personnel requirements. The mine personnel requirements drop significantly in Year 17, due to the end of open pit mining and all process plant feed requirements being met with 100% stockpile material.

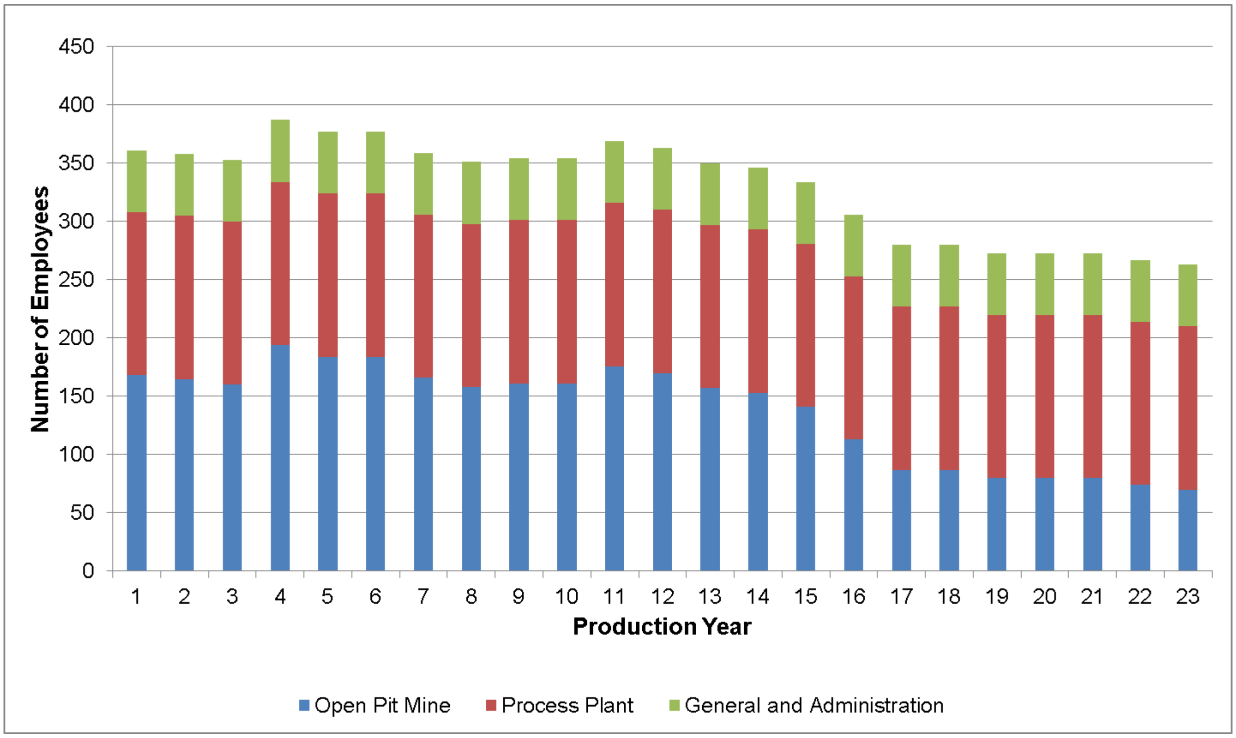


Figure 21-2: Average number of personnel

The total personnel for the Livengood Gold Project peaks in Year 4 at 387 employees as shown in Table 21-13.

Table 21-13: Project peak personnel (Year 4)

Area	No. of Employees
Open Pit Mine	194
Process Plant	140
General and Administration	53
Total	**387**

21.4.3 Open pit mine

21.4.3.1 Production schedule summary

The schedule accounts for two years of mine development during the preproduction stripping by the Owner's mine crews to prepare the mine for full scale production. The mining schedule calls for the mine to operate in production through Year 16. The production schedule specifies that the plant runs through Year 23. The low grade mineralized material stockpile would feed the plant between Years 16 and 23. Section 16.5 details the mine and processing production schedules for life of mine. Key LOM schedule parameters for the three different production periods are summarized in Table 21-14.

Table 21-14: Mine production summary schedule

Production Period	Preproduction	Mill and Stockpile	Stockpile Reclaim	Total
Years	**-2 to -1**	**1 to 16**	**17 to 23**	**23**
Direct Mine Ore to Mill Tons ('000)	-	254,377	-	**254,377**
Mine Ore to Stockpile Tons ('000)	29,085	148,267	-	**177,352**
Stockpile to Mill Tons ('000)	-	48,931	128,422	**177,352**
Mine Waste Rock Tons ('000)	86,658	468,167	-	**554,825**
Strip Ratio (1)	-	1.84	-	**1.3**
Gold Grade (g/mt)	-	0.82	0.46	**0.71**
Contained Gold (Koz)	-	7,245	1,727	**8,972**

(1) Strip ratio calculated as total mine waste rock tons / direct mine to mill tons. Overall project strip ratio is 1.28 (mine waste rock tons / (direct mine ore to mill tons + stockpile to mill tons)).

21.4.3.2 Mine operating costs

The LOM operating costs include all expenses incurred to operate the mine from the start of Year 1 through Year 23. Mine preproduction 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 the production drills, including labor and materials over the life of mine. Blasting costs include explosive materials and labor required to break the ore and mine waste rock loose from the surface mine, including increased costs for rock types RT4 and RT9 to achieve target fragmentation, which has a downstream benefit on mill throughput and processing costs. Loading costs include labor and operating costs to operate the front shovels and production front end loaders, and place the blasted rock into 320 t (290 mt) haul trucks. Hauling costs cover the labor, fuel and maintenance required to haul the waste 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 levelled and cleaned, and drilling sites levelled and cleaned. Mining operating costs are based on production hours required to perform required tasks.

The combined cost per ton of material mined for general mine expenses and engineering, drilling, blasting, loading, hauling, and support comes to $1.73/t ($1.91/mt) of total material mined or $3.41/t ($3.76/mt) of material milled, including the cost of stockpile reclamation over the LOM. Table 21-15 shows the unit operating costs for the mine based on the scheduled production, equipment, and support requirements per ton mined.

Table 21-15: Average annual and LOM operating costs – mining

Cost Item / Activity	Life of Mine Cost ($M)	Average Annual Cost ($M/y)	Cost per Ton ($/t mined)	Cost per Ton ($/t milled)	OPEX (%)
General Mine Expenses and Technical Services	150	6.7	0.17	0.35	10
Drilling	207	9.2	0.24	0.48	14
Blasting	302	13.4	0.35	0.70	20
Loading	139	6.2	0.16	0.32	9
Hauling	405	18.0	0.47	0.94	27
Support	301	13.4	0.35	0.70	20
Total	**$1,505**	**$66.9**	**$1.73**	**$3.49**	**100%**

21.4.3.3 Mine personnel

Mine labor requirements for operations, maintenance, engineering and geology have been estimated on an annual basis to support the mine plan developed in this study. Mine salaried and hourly personnel positions with corresponding headcounts were presented in Chapter 16. During the operations period, an average of 138 personnel will be employed by the open pit mine. The mine personnel headcount will peak at 194 in Year 4.

21.4.4 Process plant

Process plant operating costs over the 23-year mine life were calculated based on the metallurgical testwork program, the mine schedule, salary cost tables (THM), comparable projects, literature reviews and recent supplier quotations. Operating costs for each rock type were developed and then combined, based on the mine schedule, to calculate the overall operating cost on a per ton weighted average basis. The process plant operating costs are estimated to be $7.48/t ($8.25/mt) over the life of mine.

The average operating cost includes reagents, consumables, grinding media, personnel (Salaried and Labor), electrical power, liquefied natural gas and maintenance/operations parts. The consumables include spare parts, grinding media, liners and screen components. A breakdown of the process plant operating costs is shown in Table 21-16. The main cost areas for the process plant are electrical power, crushing and grinding steel, and reagents and chemicals. The majority of the reagent costs are associated with sodium cyanide and lime required for leaching.

Table 21-16: Average annual and LOM operating costs – process plant

Cost Item / Activity	Life of Mine Cost ($M)	Average Annual Cost ($M/y)	Cost per Ton ($/t milled)	OPEX (%)
Crushing and Grinding Steel	592	26.3	1.37	18
Reagents and Chemicals	1,054	46.9	2.44	33
Facility Power (Process Plant)	1,038	46.1	2.41	32
Ancillary Power (Admin and Garage)	16	0.7	0.04	0
Liquefied Natural Gas (LNG)	42	1.9	0.10	1
Maintenance supplies and material	165	7.3	0.38	5
Operations Supplies and materials	26	1.2	0.06	1
Hourly labor	205	9.1	0.47	6
Salaried labor	90	4.0	0.21	3
Total	**$3,228**	**$143.6**	**$7.48**	**100%**

21.4.4.1 Crushing and grinding steel

The replacement costs of major equipment consumables, such as the primary crusher liners, pre-crusher/pebble crusher mantles and bowls, SAG and ball mill liners, and screen decks, were calculated based on recommended change-out schedules, recent budgetary quotations and BBA's internal database.

The Livengood process flowsheet includes two types of grinding media for the SAG and ball mills. The consumption rates for the 5-in SAG mill and 3-in ball mill media were calculated using MolyCop (V 3.0) tools and the abrasion index (Ai) distribution measured at the 50th percentile for the five rock types to be processed over the life of mine. The input data considered the average operating conditions for the SAG and ball mills, in terms of power draw, rotational speed, pulp density and media loading. The wear and annual media consumption rates for each type are presented in Table 21-17. Crushing and grinding steel represents approximately 18% of the total process operating cost at $1.37/t milled.

Table 21-17: Average LOM media wear and consumption rates

Media Type	Wear Rate (lb/kWh)	Annual Consumption (t)
SAG mill – 5-in steel media	0.090	4,504
Ball mill – 3-in steel media	0.122	7,334

21.4.4.2 Reagents and chemicals

The reagent and chemical consumptions were estimated based on testwork, industrial references, literature and assumed operational practice. Sodium cyanide and lime have a higher consumption variability depending on rock type and, therefore, have been estimated based on an analysis of the various testwork campaigns performed to date, as well as adjusted using scale-up factors and assumed process water recirculation rates within the process plant.

The reagent unit costs ($/t reagent) were established through recent vendor quotations and comparison to prices at reference sites and include delivery to site. The Reagents and chemicals category represents approximately 33% of the total process operating cost at $2.44/t milled.

21.4.4.3 Electrical power

The largest power consumers within the process plant are the SAG and ball mills. The respective power required for the SAG mill and ball mill were calculated based on the comminution testwork program, which provided the material hardness indices (A x b value) for the SAG mill and the BWi of the ball mill for the five rock types expected to be processed during the life of mine.

The SAG mill specific energy (kWh/t) was estimated from the analyzed relationships derived from testwork between the A x b value and the SAG motor input specific energy as determined by JKSimMet. The ball mill specific energy (kWh/t) was calculated from the BWi and the Bond formula, assuming the ball mill will grind the rock from 2,900 µm (F_{80}) to 180 µm (P_{80}).

The overall process plant energy consumption was estimated based on the SAG and ball mill grinding energy requirements and factored balance of plant equipment running loads. Various factors (efficiency, load, diversity, and annual factors) were applied to adjust for equipment motor efficiency, the power used versus installed, the synchronous operation of equipment and average plant operating availability. The electrical power of the process plant represents approximately 32% of the total process operating costs at $2.41/t milled.

Electricity requirements (Ancillary Power) for the Project's surface infrastructure, such as the mine garage, administration building and the fresh water pumping system, were estimated based on the specific project requirements and similar sized installations. The ancillary power represents less than 1% of the total process operating costs at $0.04/t milled.

21.4.4.4 Liquefied natural gas

Liquefied natural gas (LNG) is planned to be used for heating of the primary and secondary crusher buildings, the main process plant building and surface ancillary facilities. LNG requirements have been estimated based on the building requirements and similar sized installations. LNG represents approximately 1% of the total process operating costs at $0.10/t milled.

21.4.4.5 Maintenance and operations supplies

Maintenance supplies and materials are intended to cover the costs of maintaining the process facilities. Operations supplies are intended to cover the cost of personnel protection wear, minor tools, oil and other consumables. The costs of maintenance and operations supplies were derived using a percentage (5.75%) of the capital cost of plant mechanical equipment. Combined maintenance and operations supplies represent approximately 6% of the total process operating costs at $0.44/t milled.

21.4.4.6 Personnel

A total of 140 employees (26 salaried and 114 hourly) divided into management and technical services, operations and maintenance departments are required in the process plant. No allowance for contractors has been allocated. The list of personnel (Chapter 17), along with the salaries, was provided by THM. The estimated personnel cost (salaried and hourly combined) represents approximately 9% of the total process operating cost at $0.68/t milled.

21.4.5 General and administration (G&A)

G&A costs are expenses not directly related to the production of goods and encompass items not included in the mining and processing sectors of the Project. These costs were developed based on THM's past project experience, similar sized operations and BBA's in-house database.

The General and Administration area includes the following items:

- Site administration, accounting and payroll labor;
- Human Resources, Information Technology (IT) and Health Services labor;
- Computer hardware and software costs/license fees;
- Health and Safety supplies;
- Insurance (Earthquake, Physical Plant, and Rolling Stock including loss of production);
- Security, maintenance, laundry, snow removal and janitorial service contracts;
- Warehouse administration and supplies;
- Waste collection and recycling services;

- Environmental testwork and permitting fees;
- Mobile equipment and building maintenance;
- Telecommunications and data service fees;
- Staff and labor training;
- Employee transportation fees.

The total G&A operating cost equals $1.28/t ($1.41/mt) milled. Table 21-18 shows life of mine and average annual operating costs for G&A expenses. The largest costs within the G&A category is employee transport, representing approximately 17%, while insurance is the second largest cost, accounting for approximately 15%. Corporate expenses followed by costs related to the Environmental, and Health and Safety departments are also significant contributors.

Table 21-18: Average annual and LOM operating costs – general and administration

Cost Item / Activity	Life of Mine Cost ($M)	Average Annual Cost ($M/y)	Cost per Ton ($/t milled)	OPEX (%)
General Management and Administration	21	1.0	0.05	4
Environmental	53	2.3	0.12	10
Community Relations	18	0.8	0.04	3
Human Resources	24	1.1	0.06	4
Health, Safety & Security	55	2.5	0.13	10
Accounting	31	1.4	0.07	6
Information Technology	23	1.0	0.05	4
Warehouse	26	1.1	0.06	5
Purchasing	14	0.6	0.03	3
Transportation (Bussing)	96	4.3	0.22	17
Land	11	0.5	0.03	2
Corporate	62	2.8	0.14	11
Mobile Equipment Operations and Maintenance	37	1.7	0.09	7
Insurance	81	3.6	0.19	15
Total	**$552**	**$24.6**	**$1.28**	**100%**

21.4.5.1 Personnel

A total of 53 employees are required by the general and administration group. The number of employees allocated to each administration department is shown in Table 21-19.

Table 21-19: G&A employee list

Department	No. of Employees
General Management and Administration	2
Environmental	10
Community Relations	1
Human Resources	5
Health, Safety & Security	13
Accounting	8
Information Technology	3
Warehouse and Purchasing	11
Total	**53**

21.5 Royalties

The annual royalty costs are based on the PFS mine design and production profile, along with the terms of the individual royalty agreements. Over the life of the Project, based on an assumed 3.0% average royalty fee, approximately $252M in royalties is expected to be paid.

21.6 Transportation and refining

A weekly shipment of doré bars will be transported to a refinery. A flat rate transportation cost will be incurred by the refinery in addition to a cost by weight and a variable liability fee. A treatment cost per troy ounce of material shipped to the refinery will also be charged. THM will be paid for a set recovery of the assayed gold content. Over the LOM, a transport and refining cost of $54M is estimated based on typical terms and pricing for a North American gold refinery.

22. ECONOMIC ANALYSIS

22.1 Introduction

The economic/financial assessment of the Livengood Gold Project is based on a financial model developed by Tower Hill Mines (THM) and BBA Inc. (BBA). The model calculates revenues based on contained ounces, head grade, recovery and a gold price of $1,250/oz (base case). 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. The economic evaluation was carried out using a discounted cash flow approach on a pre-tax and after-tax basis, based on Q3 2016 metal price projections. No provision was made for the effects of inflation. Current tax regulations were applied to assess the federal income tax liabilities, while the most recent state regulations were applied to assess the Alaska income and mining tax liabilities.

The internal rate of return (IRR) on total investment was calculated based on 100% equity financing, even though THM may decide in the future to finance part of the Project with debt financing. The Net Present Value (NPV) was calculated from the cash flow generated by the project, based on a discount rate of 5%. The payback period based on the undiscounted annual cash flow of the Project is also indicated as a financial measure. Furthermore, a sensitivity analysis has been performed for the after-tax base case to assess the impact of the following variations on the project economics: capital costs, operating costs, and price of gold.

The economic analysis presented in this section contains forward-looking information with regard to the mineral reserve estimates, commodity prices, proposed mine production plan, projected recovery rates, operating costs, construction costs and project schedule. The results of the economic analysis are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. The reader is cautioned that this PFS is preliminary in nature and there is no certainty that the PFS economics will be realized.

22.2 Assumptions and basis

The economic analysis was performed using the following assumptions and basis:

- The conceptual mine plan developed in Chapter 16 provided the following inputs to the financial model: mine life, annual ore and waste tons mined, and annual mill tons and head grade;
- The preproduction period and construction period financial inputs flow from the Project execution schedule developed in Chapter 24, taking into consideration key project milestones;

- The financial model applies metal pricing of $1,250/oz, which was estimated on the basis of discussions with experts, consensus analyst estimates and recently-published economic studies that were deemed to be credible. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. It is understood that metal prices can be volatile and that there is the potential for deviation from the LOM forecasts;
- All cost and sales estimates are in constant Q3 2016 United States dollars with no inflation or escalation factors taken into account;
- All metal products are assumed sold in the same year that they are produced;
- Class specific capital cost depreciation rates for tangible property under the Modified Accelerated Cost Recovery System (MACRS) are used for the purpose of determining the allowable taxable income;
- All project related payment and disbursements incurred prior to the effective date of this report are considered as sunk costs. Disbursements that may occur after the effective date of this report, but before the start of construction, are considered as sunk costs;
- Net present value (NPV) was calculated using the middle of period approach;
- The after tax model includes Alaska state taxes and Federal taxes according to 2016 guidelines;
- The model applies 3% royalties on net smelter returns across the life of mine based on an average royalty calculation;
- Project revenue is derived from the sale of gold doré into the international marketplace. No contractual arrangements for doré smelting or refining exist at this time. Provisions for gold transportation, insurance, refining and payable charges have been included in the financial model;
- Final rehabilitation and closure costs will be incurred after production Year 23.

This financial analysis was performed on both a pre-tax basis and after-tax basis with the assistance of an external tax consultant hired by THM. The general assumptions used for this financial model and the LOM plan tonnage and grade estimates are summarized in Table 22-1, and outlined in Table 22-3.

Table 22-1: Financial model criteria

Description	Value	Unit
Construction/Preproduction Period	36	Months
Mine Life (after preproduction)	23	Years
Total Ore Processed	432	Mt
Total Waste Mined (including 87Mt during preproduction)	555	Mt

Description	Value	Unit
Gold Grade (LOM)	0.71	g/mt
Gold Recovery (LOM)	75.3	%
Gold Production (LOM)	6,763,900	Troy oz
Average Annual Process Gold Production Rate	294,100	Troy oz
Daily Milling Rate	52,600	t/d
Open Pit Mining Operating Cost (LOM Avg.)	1.73	$/t mined
Processing Operating Cost (LOM Avg.)	7.48	$/t milled
General and Administration Operating Cost (LOM Avg.)	1.28	$/t milled
Gold Transportation and Insurance, Refining, and Payable Charges	8.05	$/oz
Doré Gold Payable Terms	99.5	%
Royalty on Net Smelter Return (NSR)	3.0	%
Base Case Gold Price	1,250	$/oz
Discount Rate	5.0	%
Initial Capital Cost	1.84	$B
Sustaining Capital Cost	665	$M
Reclamation and Closure Cost	342	$M

22.3 Royalties

The annual royalty costs are based on the conceptual open pit mine design and production profiles described in Chapter 16. Due to the fact that there are numerous individual royalty agreements, for the purposes of this financial evaluation, a fixed 3.0% NSR has been assumed. Over the life of the Project, approximately $252M in royalties is expected to be paid based on the base case metal prices and project assumptions.

22.4 Third party smelting, refining and transportation

A weekly shipment of doré bars will be transported to a refinery. A flat rate transportation cost will be incurred by the refinery, in addition to a cost by weight and a variable liability fee. A treatment cost per troy ounce of material shipped to the refinery will also be charged. THM will be paid for a set recovery (99.5%) of the assayed gold content. Over the life of the mine, a transport and refining cost of $8.05/oz is estimated and this is based on a budgetary quotation obtained from a North American gold refinery.

22.5 Taxes

The Livengood Gold Project is subject to three levels of taxation, including federal income tax, Alaska State income tax, and an Alaska State mining license tax. THM compiled the taxation calculations for the Project with assistance from third party taxation experts. This information was not verified by BBA.

The current US tax system applicable to mineral resource income was used to assess the annual tax liabilities for the Project. The US Federal corporate income tax, Alaska State corporate income tax and Alaska State license mining tax rates, currently applicable over the operating life of the Project, are 35.0%, 9.40% and 7.0% of taxable income, respectively.

The tax calculations are underpinned by the following key assumptions:

- The Project is held 100% by a corporate entity and the after-tax analysis does not attempt to reflect any future changes in corporate structure or property ownership;
- Assumes 100% equity financing and therefore does not consider interest and financing expenses;
- Projected payments relating to Net Smelter Return (NSR) or Net Profits Interest (NPI) royalties, as applicable, are allowed as a deduction for federal and state income tax purposes, but are added back for state mining tax purposes; and
- Actual taxes payable will be affected by corporate activities, and current and future tax benefits have not been considered.

The combined effect on the Project of the three (3) levels of taxation, including the elements described above, is a cumulative effective tax rate of 27%, based on the Project's LOM Operating Income (gross income less operating costs and depreciation). It is anticipated, based on the Project assumptions, that THM will make tax payments of approximately $104M ($16/oz) over the life of the Project.

22.6 Closure costs

NewFields developed a dry closure plan for the tailings management facility. Closure costs track reclamation and closure expenses over a period of 33 years (Year 21 through 54), including costs to build a water treatment plant in Year 21 and 22, prior to the termination of operations. The main closure construction effort occurs from Year 21 through 31, accounting for 79% of the overall closure costs. Costs for pumping and management operations are included in Years 23 through 54. Year 54 is the last year with planned closure expenses.

The total closure cost is $341.7M. This total closure cost is applied to the cash flow in Year 23. 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 $9.1M during Years -2 through 22. The model includes trust fund earnings at a 3.0% annual percentage rate (APR), applied to the fund balance until closure is complete.

22.7 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 and price of the saleable product produced, allowing for the following ramp-up, which corresponds to the mine production schedule:

Quarter 1:	11.7% 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% of 1st year production (75% of design capacity)

Revenue receipt was projected based on shipping and receipt of 85% of funds four (4) weeks after the shipping date, with the balance of 15% of funds received eight (8) 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 17, totaling $44.7M. The model contains this working capital cost in Year 1 and recovers the equivalent amount in Year 23.

22.8 Gold production

Figure 22-1 highlights the anticipated gold production schedule for the Livengood Gold Project. Total life of mine production is anticipated to be 6,763,900 oz or approximately 294,000 oz/y based on the PFS mine plan, estimated feed grade and recovery estimates. The average feed grade is expected to be 0.71 g/mt and process plant recovery is estimated to be 75.3% over the life of mine. Over the first five years, the operation is expected to produce approximately 378,000 oz/y due to higher grade material being preferentially sent to the process plant. Low grade material will be stockpiled in these early years to be used for future process plant feed.

During Years 17 through 23, the process plant feed will consist entirely of reclaimed ore from the low grade stockpile.

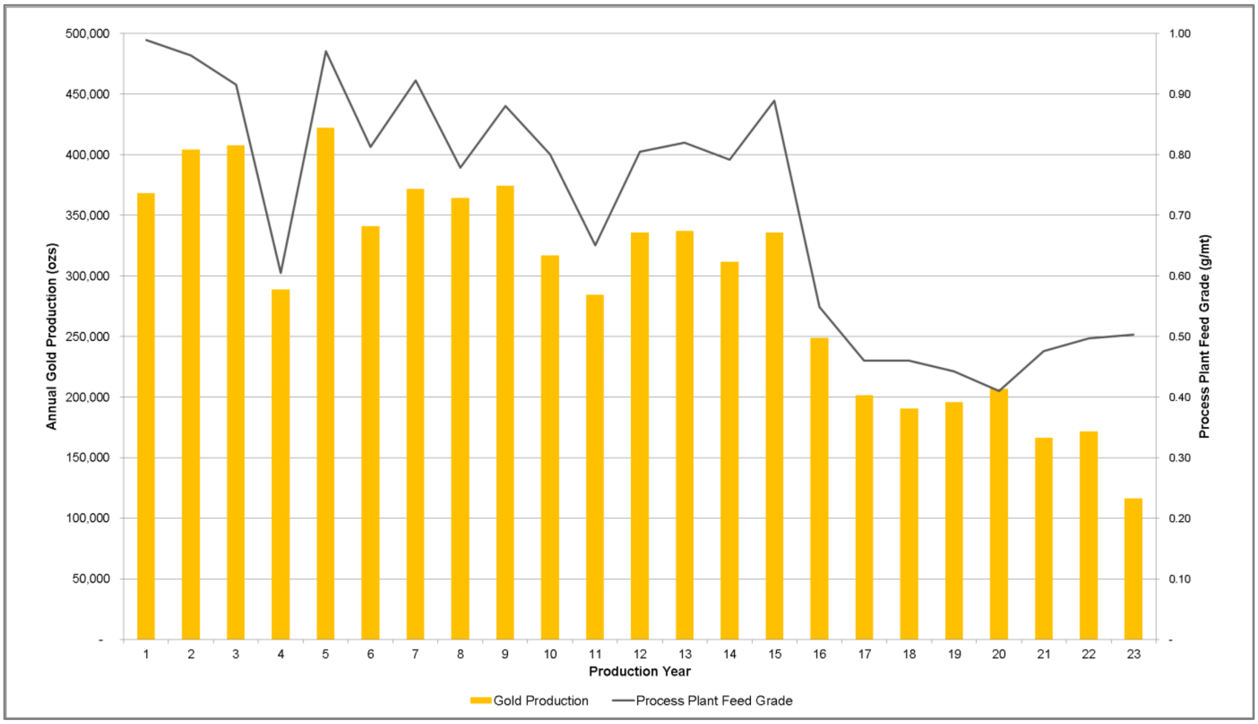


Figure 22-1: Annual gold production schedule

22.9 Production costs

The costs of production at Livengood total $877/oz before capital, the all-in sustaining costs (AISC) are $976/oz and the all-in costs are $1,247/oz including capital. Taxes add an additional $16/oz for a total production cost of $1,263/oz. Production costs before capital represent 70% of the all-in cost, while capital expenses represent 30%.

Table 22-2 highlights the costs of production over the life of the Project.

Table 22-2: Costs of production

Costs of Production	$/oz	LOM ($Million)
Operating Costs	**$877**	**$5,934**
Capital Expenditures (Sustaining) (1)	99	665
All-In Sustaining Costs (AISC)	**$976**	**$6,599**
Capital Expenditures (Non-sustaining) (1)	271	1,836
All-In Costs	**$1,247**	**$8,435**

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

(1) Excludes $18M upfront funding included in reclamation and remediation above and $37M of recoverable initial stores inventory.

22.10 Financial analysis

A 5% discount rate was applied to the cash flow to derive the NPV for the Project on a pre-tax and after-tax basis. The summary of the financial evaluation results for the Project base case, at a gold price of $1,250/oz, is presented in Table 22-3.

Table 22-3: Financial analysis summary (pre-tax and after-tax)

Description		Base Case	Units
Pre-Tax	Net Present Value (0% disc)	197.7	$M
	Net Present Value (5% disc)	- 507.1	$M
	Internal Rate of Return	1.0	%
	Simple Payback Period	17.2	Years
After-Tax	Net Present Value (0% disc)	93.7	$M
	Net Present Value (5% disc)	- 552.0	$M
	Internal Rate of Return	0.5	%
	Simple Payback Period	22.1	Years

The pre-tax base case financial model resulted in an IRR of 1% and a negative NPV of $ -507M using a discount rate of 5%. The simple pre-tax payback period is 17.2 years. On an after-tax basis, the base case financial model resulted in an IRR of 0.5% and a negative NPV of $ -552M with a discount rate of 5%. The simple after-tax payback period is 22.1 years.

The summary of the Livengood Gold Project discounted cash flow financial model (pre-tax and after-tax) is presented in Table 22-4.

Table 22-4: Simplified cash flow table

Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	Total/Average
Period	**Preproduction**			**Production**																							
Production Summary																											
Total Ore Mined (Mt)	0	10	19	35	32	34	6	31	31	31	25	25	25	19	25	25	28	26	3	0	0	0	0	0	0	0	**432**
Total Waste Mined (Mt)	0	31	56	25	29	26	55	30	30	30	35	35	35	42	35	25	21	13	1	0	0	0	0	0	0	0	**555**
Total Milled (Mt)	0	0	0	15	19	20	20	19	19	20	20	19	19	19	19	19	19	19	20	19	19	20	21	19	19	13	**432**
Mill Head Grade Au (g/mt)	0.00	0.00	0.00	0.99	0.96	0.92	0.60	0.97	0.81	0.92	0.78	0.88	0.80	0.65	0.81	0.82	0.79	0.89	0.55	0.46	0.46	0.44	0.41	0.48	0.50	0.50	**0.71**
Gold Recovery (%)	0.0	0.0	0.0	81.2	76.5	79.0	83.6	78.0	76.3	71.3	82.3	77.1	71.6	77.8	75.3	74.3	70.8	67.2	79.6	78.8	75.6	77.6	81.8	65.2	62.9	62.4	**75.3**
Revenue																											
Gross Revenue ($M)	0	0	0	461	505	510	361	528	426	465	456	468	396	356	420	422	390	420	311	252	238	245	258	208	215	146	**8,455**
Operating Expenditures																											
Mining ($M)	0	0	0	-98	-96	-96	-105	-96	-96	-96	-96	-96	-97	-100	-96	-80	-77	-64	-15	-11	-11	-11	-12	-10	-11	-7	**-1,474**
Processing ($M)	0	0	0	-115	-143	-144	-142	-145	-144	-148	-144	-145	-146	-143	-146	-146	-146	-149	-143	-140	-139	-143	-153	-146	-149	-98	**-3,259**
General and Administration ($M)	0	0	0	-20	-24	-25	-25	-24	-24	-25	-25	-24	-24	-25	-24	-24	-24	-25	-25	-24	-24	-25	-27	-24	-24	-16	**-552**
Smelting, Refining and Transport Costs ($M)	0	0	0	-3	-3	-3	-2	-3	-3	-3	-3	-3	-3	-2	-3	-3	-3	-3	-2	-2	-2	-2	-2	-1	-1	-1	**-54**
Royalty Payments ($M)	0	0	0	-14	-15	-15	-11	-16	-13	-14	-14	-14	-12	-11	-13	-13	-12	-13	-9	-8	-7	-7	-8	-6	-6	-4	**-252**
Capital Expenditures																											
Preproduction ($M) [(1)]	-53	-908	-856	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	37	**-1,781**
Sustaining ($M)	0	0	0	-91	-88	-25	-55	-25	-25	-21	-48	-56	-34	-16	-19	-18	-17	-18	-18	-21	-35	-8	-8	-8	-8	0	**-665**
Reclamation and Closure ($M)	0	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	-9	0	**-219**
Working Capital ($M)	0	0	0	-45	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	45	**0**
Pre-Tax Cash Flow																											
Annual Pre-Tax Cash Flow ($M)	-53	-917	-865	67	127	192	11	209	112	149	117	121	72	49	110	129	102	140	89	37	12	39	40	3	6	100	**198**
Cumulative Pre-Tax Cash Flow ($M)	-53	-971	-1,836	-1,770	-1,642	-1,450	-1,439	-1,230	-1,117	-968	-851	-730	-658	-608	-499	-369	-268	-128	-39	-2	10	48	88	92	97	198	**198**
Taxes																											
Alaska State Income and Mining Taxes ($M)	0	0	0	-0	0	0	0	-1	0	-1	-1	-2	-1	-0	-1	-2	-2	-4	-2	-1	-1	-3	-4	0	-0	0	**-27**
Federal Income Tax ($M)	0	0	0	-1	0	0	0	-1	0	-1	-1	-2	-1	-0	-1	-3	-12	-19	-13	-7	-5	-5	-5	0	-0	0	**-77**
After-Tax Cash Flow																											
Annual After-Tax Cash Flow ($M)	-53	-917	-865	66	127	192	11	207	112	148	114	117	71	49	107	124	88	117	74	29	6	31	31	3	5	100	**94**
Cumulative After-Tax Cash Flow ($M)	-53	-971	-1,836	-1,770	-1,643	-1,450	-1,439	-1,233	-1,120	-972	-858	-740	-670	-621	-514	-390	-302	-185	-111	-83	-77	-46	-15	-12	-7	94	**94**
Summary																											
Pre-Tax NPV @ 5% ($M)	**-507**																										
Pre-Tax IRR (%)	**1.0%**																										
After-Tax NPV @ 5% ($M)	**-552**																										
After-Tax IRR (%)	**0.5%**																										

[(1)] $37M of recoverable initial stores inventory in Year 23

Figure 22-2 shows the cumulative cash flows for the Project projected for the life of the mine on a pre-tax and after-tax basis.

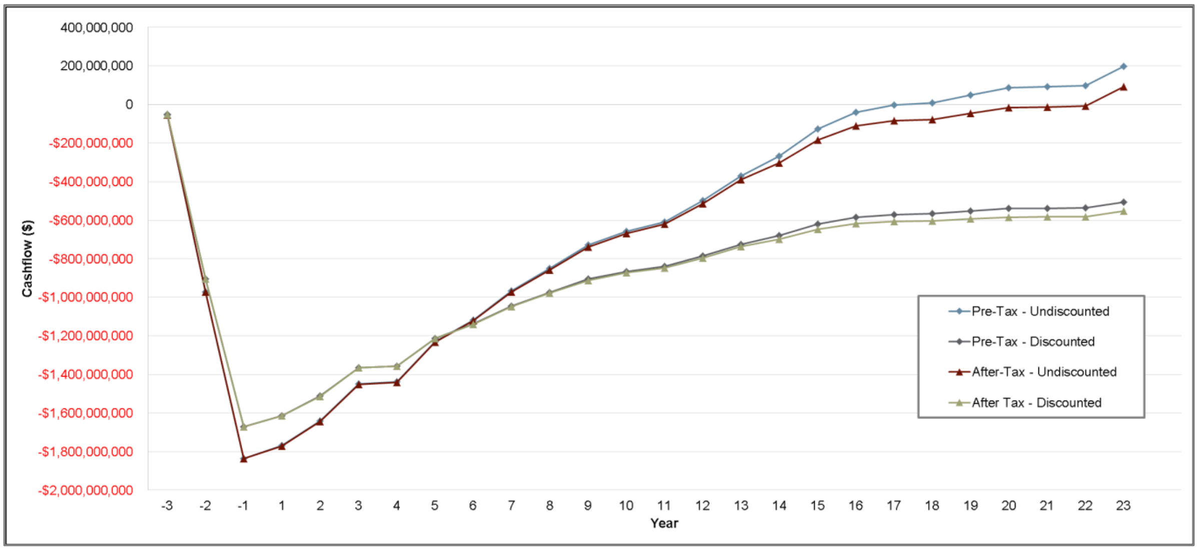


Figure 22-2: Life-of-mine cash flow projection (pre-tax and after-tax, discount rate: 5%)

22.11 Sensitivity analysis

The economic evaluation includes an analysis of the Project sensitivity to key financial parameters compared to the base case. Sensitivity measures how much impact a change in a given parameter has on the base project value, all other factors remaining constant. Table 22-5 presents the after-tax IRR and NPV (@ 5% discount rate) sensitivity results for varying gold recovery, gold price, total operating cost and total capital cost. Figure 22-3 and Figure 22-4 present each sensitivity analysis graphically, steeper curves represent greater sensitivity.

This sensitivity analysis shows that both gold price and recovery variations cause the greatest and almost equivalent impact on project value. A 30% increase in gold price to $1,625/oz would yield an IRR of 8.6% and a NPV of $511M. A 30% decrease in gold price to $825/oz would yield a reduced IRR of -39.6% and NPV of -$1,887M. The impact of variations in operating and capital cost on both financial metrics is fairly similar, with the operating cost changes resulting in marginally larger project returns than capital cost changes, meaning that reducing operating expenses would benefit the Project more than reducing capital costs by the same percentage.

Table 22-5: Project sensitivity analysis – after-tax IRR and NPV

Base Case Variance	-30%	-20%	-15%	-10%	-5%	Base	+5%	+10%	+15%	+20%	+30%
Gold Recovery (%)			64%	68%	72%	**75.3%**	79%	83%	87%		
After Tax IRR			-7.30%	-3.90%	-1.30%	**0.50%**	2.10%	3.60%	4.90%		
After Tax NPV @ 5% ($M)			-$1,200	-$973	-$748	**-$552**	-$369	-$188	-$11		
Gold Price ($/oz)	$875	$1,000	$1,063	$1,125	$1,188	**$1,250**	$1,313	$1,375	$1,438	$1,500	$1,625
After Tax IRR	-39.64%	-13.10%	-7.30%	-3.90%	-1.30%	**0.50%**	2.10%	3.60%	4.90%	6.20%	8.60%
After Tax NPV @ 5% ($M)	-$1,888	-$1,429	-$1,199	-$974	-$747	**-$552**	-$367	-$187	-$9	$165	$511
Operating Cost ($M)	$3,700	$4,229	$4,493	$4,757	$5,022	**$5,286**	$5,550	$5,815	$6,079	$6,343	$6,872
After Tax IRR	5.90%	4.30%	3.40%	2.50%	1.50%	**0.50%**	-0.60%	-2.00%	-3.80%	-6.00%	-15.00%
After Tax NPV @ 5% ($M)	$128	-$93	-$205	-$319	-$435	**-$552**	-$675	-$809	-$952	-$1,096	-$1,386
Capital Cost ($M)	$1,751	$2,001	$2,126	$2,251	$2,376	**$2,501**	$2,626	$2,751	$2,876	$3,001	$3,251
After Tax IRR	5.10%	3.30%	2.50%	1.80%	1.10%	**0.50%**	-0.10%	-0.60%	-1.10%	-1.70%	-2.70%
After Tax NPV @ 5% ($M)	$11	-$174	-$268	-$362	-$457	**-$552**	-$649	-$748	-$848	-$955	-$1,171

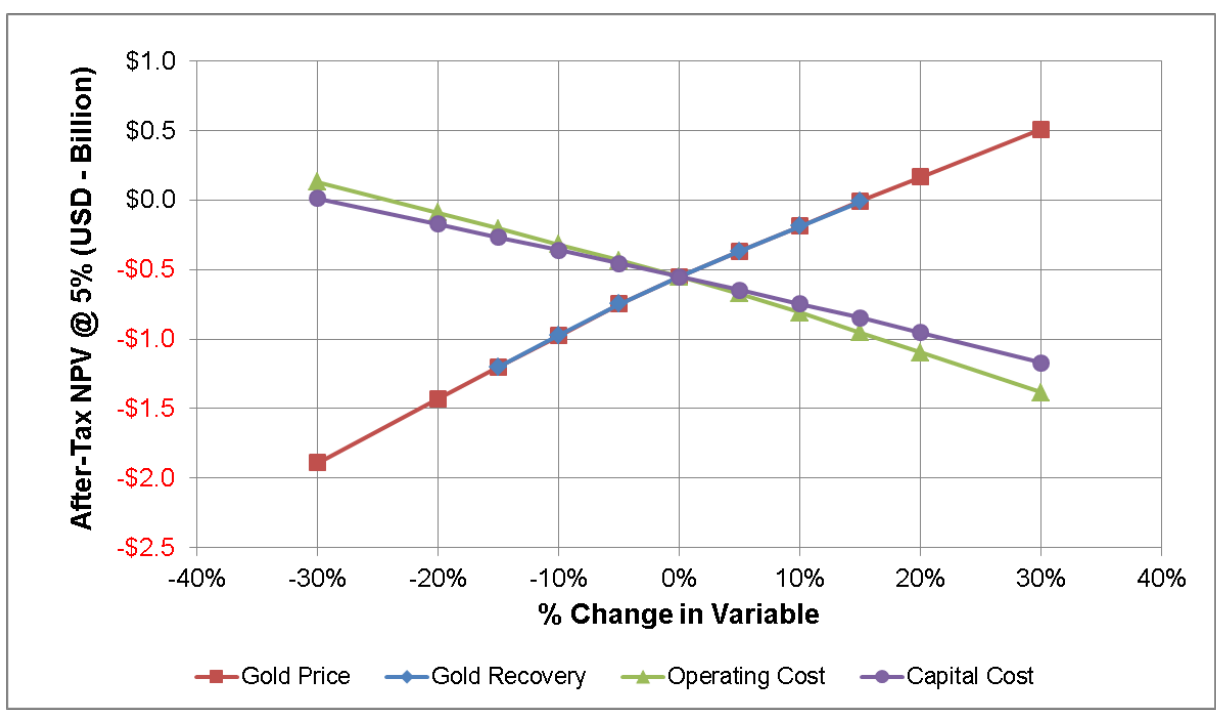


Figure 22-3: After-tax sensitivity analysis for project net present value (NPV @ 5% discount rate)

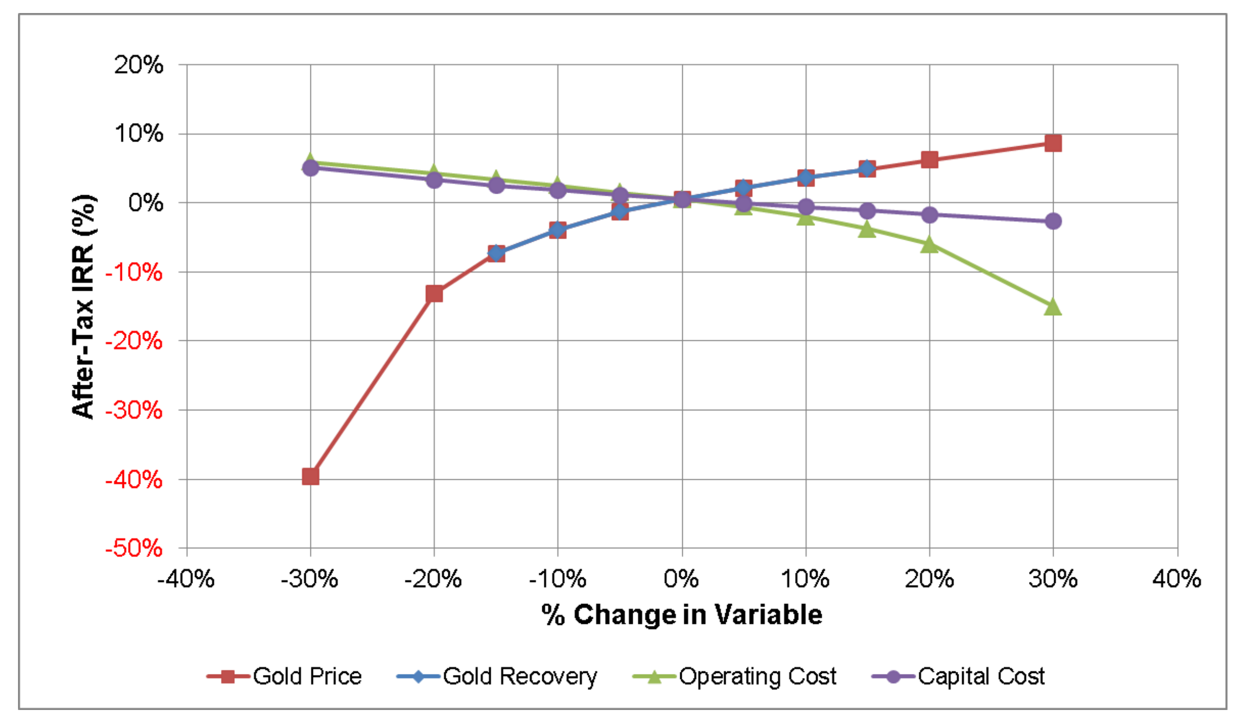


Figure 22-4: After-tax sensitivity analysis for project internal rate of return (IRR %)

23. ADJACENT PROPERTIES

This chapter provides public source information on producing and exploration properties adjacent to the Livengood Gold Project. The information related to adjacent properties is not necessarily indicative of the mineralization on the Livengood Gold property.

23.1 Introduction

The Project is located in the Tolovana mining district within the Tintina Gold Belt. The Project area is centered on a local topographic high point named Money Knob. This feature and the adjoining ridge lines have been considered by many to be the lode gold source for the placer gold deposits, which lie in the adjacent valleys and have been actively mined since 1914 with production of more than 500,000 oz of gold.

Running northwest-southwest, approximately 3.7 mi (6 km) to the west of the Project, is the Alaska Pipeline, which transports crude oil from Alaska's North Slope to the south coast of Alaska.

The community of Fairbanks, Alaska, located approximately 70 mi (113 km) southeast of the Project site, has developed significant logistical infrastructure in support of the mining industry. It has experienced mining contractors and suppliers, and a trained mining workforce, all of which supports regional exploration activities and the two major hard rock gold mines in the region.

23.2 Producing properties

The Fort Knox Gold Mine is an open pit mine owned and operated by Toronto-based Kinross Gold (TSX: K). A conventional gravity/carbon-in-pulp (CIP) mill processes up to 50,000 t/d (45,000 mt/d) of higher grade ore (0.6 g/mt), with a heap leach for lower grade ore (0.3 g/mt). The mine is located 26 mi (42 km) northeast from the city of Fairbanks via a combination of paved and unpaved roads. In production since 1996 and surpassing production of 7 Moz, Fort Knox is the single largest producer of gold in the history of the State of Alaska and is the largest single property taxpayer in the Fairbanks North Star Borough.

The Pogo Gold Mine is an underground mine owned and operated by Sumitomo Metal Mining Pogo LLC and affiliates. A conventional gravity/flotation/flotation concentrate CIP leach processes up to 3,000 t/d (2,722 mt/d) of ore generally greater than 10 g/mt. The mine is located 85 mi (137 km) southeast from the city of Fairbanks via a combination of paved and unpaved roads. In production since 2006 and surpassing production of 3 Moz, Pogo is the largest underground gold mine in Alaska.

23.3 Exploration projects

In 2014, Freegold Ventures Limited (FVL:TSX) acquired control of the Shorty Creek property comprising 27,000 acres (10,800 hectares) of State of Alaska mining claims directly adjacent to and south of the Livengood Gold Project. During 2015, the company released a technical report on the property (Abrams, Mark J, "Technical Report for the Shorty Creek Project, Livengood-Tolovana Mining District, Alaska", March 31, 2015), completed a geophysical program and conducted limited drilling.

In 2016, Freegold released an updated technical report on the property (Abrams, Mark J, "Updated Technical Report for the Shorty Creek Project, Livengood-Tolovana Mining District, Alaska", March 25, 2016), and conducted additional drilling. Hole SC 16-01 intersected 434.5 m grading 0.57% copper equivalent from the base of oxidation at 86.1 m to EOH at 520.6 m. Within this broad intercept, a higher-grade interval of 207 m grading 0.73% copper equivalent from 138.6 m to 345 m was also intersected. Mineralization remains open to depth with the last 12 m grading 0.82% copper equivalent. (Cu 0.55%, Au 0.145 g/t and Ag 9.67 g/t). (Freegold Ventures Limited press release September 8, 2016).

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Execution plan and schedule

The execution plan is conceptual in nature and will be adjusted and refined during a future phase of the Project. The plan covers the period from the initiation of the environmental impact study (EIS) process to commercial production in Q2 Year 1. It is based on a recommended Project configuration that includes an open pit mine, a processing plant with a capacity of 52,600 t/d (47,700 mt/d), and surface infrastructure facilities. The durations and milestones for the major Project activities are shown in Table 24-1 and Figure 24-1.

Table 24-1: Key project activities (preliminary)

Activity	Start date	Completion date	Duration (months)
Environmental Impact Statement and Permitting	Q1 YR -7	Q3 YR -3	48
Engineering Studies in Support of Permitting	Q1 YR -7	Q3 YR -3	48
Process Plant Detailed Engineering	Q1 YR -3	Q3 YR -2	21
Project Authorization	--	Q3 YR -3	--
Pit Pre-Stripping / Waste Rock Supply for Construction	Q3 YR -3	Q4 YR -1	30
Tailings Management Embankment Construction	Q3 YR -3	Q4 YR -1	30
Process Plant Construction	Q4 YR -3	Q4 YR -1	27
Process Plant Dry Commissioning Completed	--	Q1 YR 1	--
Start Process Plant Ramp-up to Commercial Production	Q1 YR 1	--	--

After the PFS, the Project plans to proceed with an optimization phase prior to initiating a full feasibility study. In parallel, environmental studies will be continued.

The Project schedule includes consideration of early work requirements, the permitting process, stakeholder engagement, engineering studies, the procurement of long lead items and critical equipment, construction, and facility commissioning, including the power line and main substation, processing plant, tailings management facility, and site infrastructure.

Off-site construction of a sub-station and a transmission line by others will need to be permitted, constructed, and operational by Q2 Year -1 to allow for commissioning of the processing facilities.

On-site construction at the Livengood site is planned to start with a major civil contractor preparing the site access roads, while the THM mining operations team begins preproduction stripping in Q3 Year -3. The overall construction period from start of the access road construction to completion of the process plant is expected to last 32 months. The civil contractor will begin the tailings management facility (TMF) embankment foundations in Q4 Year -3, while the ground is frozen. Waste rock excavated from the pit by the mining team will be used for the construction of the embankment, haul roads, and other facilities. Waste rock fill will be delivered, placed and compacted by the THM mining operations team. The civil contractor will be responsible for the installation of liners and smaller volume excavations and backfills. Once road foundations and embankment foundations are completed, work will be maximized in warmer weather and scaled back during the coldest winter months. The preproduction TMF embankment will be raised by the end of Q3 Year -1 to a height sufficient to accumulate process water required for start-up and operations.

The construction of other surface facilities, including the main substation, process plant and surface fleet maintenance shop will begin in Q4 Year -3 with the aim of completing construction and commissioning in Q1 Year 1. This schedule is in line with recent projects of similar scope and size.

An analysis of the construction schedule developed during the PFS facilitated the development of a preliminary site workforce plan, which is expected to peak at approximately 1,050 workers during construction. The total estimated workforce takes into account the development of the open pit, direct and indirect construction labor for the tailings and water management facilities, process plant construction, and the construction of other site facilities. The estimate also incorporates commissioning crews and an allowance for THM operating and supervision personnel. A construction camp will be built to lodge the labor force.

Figure 24-1 shows the summary schedule for the Project.

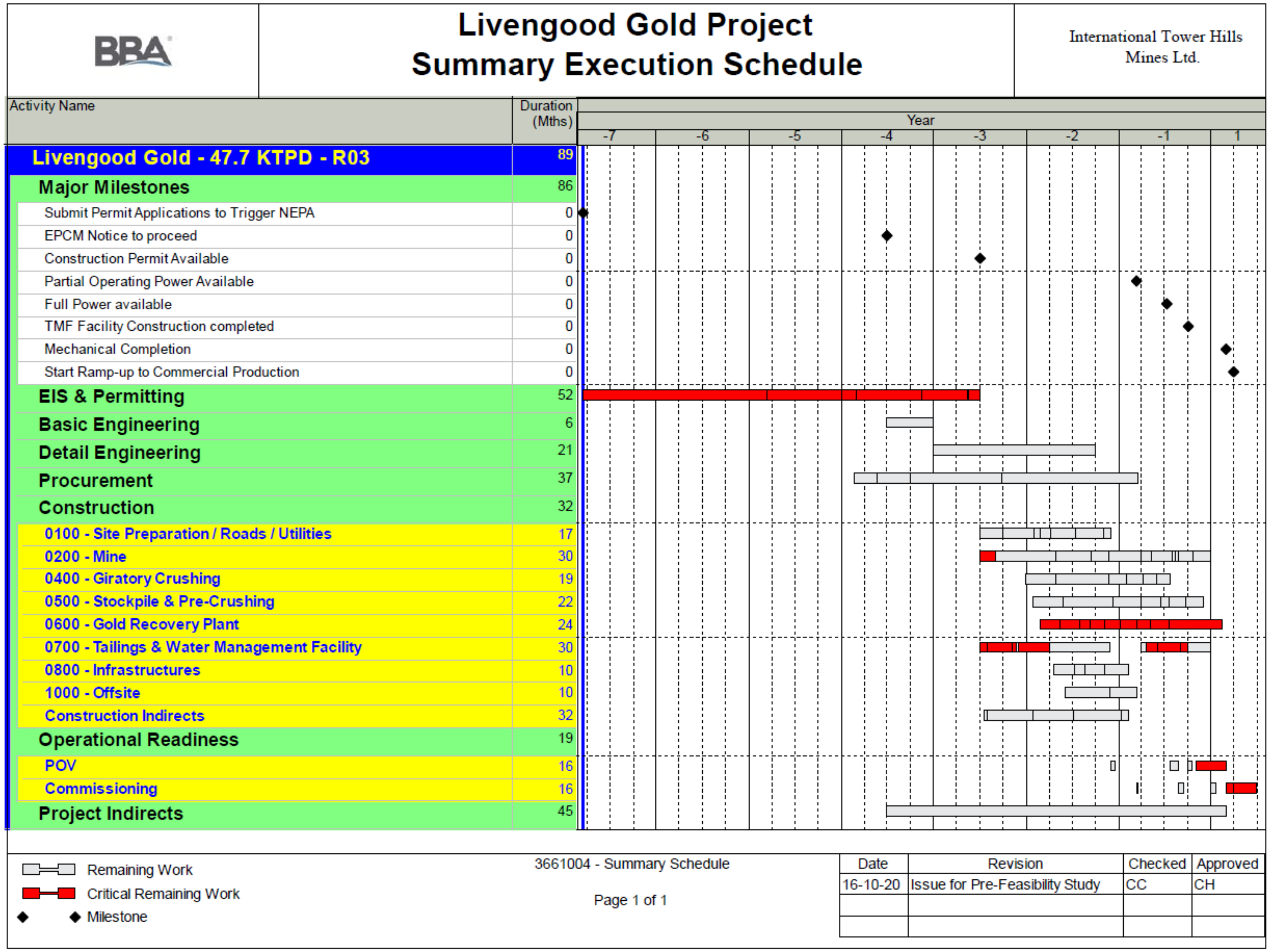


Figure 24-1: Summary project execution schedule

24.2 Logistics and transportation

24.2.1 Introduction

In 2012, SR International Logistics (SRIL) completed a logistics and transportation study to support the FS. This study is still relevant to the current PFS. SRIL reviewed and compiled extensive data to plan a seamless and uninterrupted 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 included pricing details for ocean freight, inland freight, air freight, heavy haul requirements, rail freight, consolidation and marshaling points, and warehousing.

24.2.2 Freight options considered

The construction and commissioning of the Project will require effective frontend planning and a complete, schedule-driven transportation and logistics plan. All freight for warding 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 that ocean freight and inland freight schedules are met and that freight inspections/inventories and import customs documentation are compliant with US 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 and taxes will also be based on this number.

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

THM will set up a primary receiving yard to hold and consolidate freight near the Project site. It is assumed that the primary receiving yard would be located on the northern outskirts of Fairbanks, near Highway 2. Alternatively, ITH 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 (Seattle, WA) 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 (792 km) (6 hrs) via State Highways 1 and 3. The road has year round state maintenance and regulations.
- Valdez Port to Fairbanks yard: 365 mi (803 km) (7 hrs) via State Highway 4 with year round state maintenance and regulations.
- Seward Port to Fairbanks yard: 485 mi (1,067 km) (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 (917 km) (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 and 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 rail service connects with US and Canadian lines via hydro-train barges.

Caterpillar, 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.

24.2.3 Recommended base routes

The preferred base route for most project equipment and materials contains four legs and is shown in Figure 24-2. The legs are listed below with the approximate distances:

Table 24-2: Preferred base route legs and distances

Number	Leg	Distance (miles)
1	EX-works to Puget Sound	-
2	Puget Sound to Anchorage	1,726
3	Anchorage to Fairbanks	352
4	Fairbanks to Livengood	71
Total	**Puget Sound to Livengood**	**2,109**

Figure 24-2: Primary route, Livengood logistics plan (Google Earth)

25. INTERPRETATIONS AND CONCLUSIONS

25.1 Overview

This Report was prepared by a group of independent consultants (QPs) to demonstrate the economic viability of an open pit mine and process plant complex based on the reserves estimated for the Livengood Gold Project. This Report provides a summary of the results and findings from each major area of investigation to a level that is considered to be equivalent and normally expected for a PFS of a resource development Project. Standard industry practices, equipment and processes were used in this study.

This Report is based on an updated resource estimate effective as of August 26, 2016 and has an optimized Project configuration and throughput of 52,600 t/d (47,700 mt/d), compared to the 100,000 t/d (90,700 mt/d) Project evaluated in the September 2013 FS.

25.2 PFS improvements

The Project configuration evaluated in the PFS remains a conventional, owner operated surface mine that will utilize large-scale mining equipment in a blast/load/haul operation. Mill feed would be processed in a 52,600 t/d (47,700 mt/d) comminution circuit consisting of primary and secondary crushing (pre-crushing), wet grinding in a single semi-autogenous (SAG) mill and single ball mill, followed by a gravity gold circuit and a conventional carbon in leach (CIL) circuit.

This improved configuration has reduced the capital costs by 34% or $950 million to $1.84B, the process operating cost by 28% or $2.97 per ton to $7.48 per ton, the all-in sustaining cost (AISC) to $976/oz and the all-in cost to $1,247/oz, all as compared to the Project evaluated in the FS.

The lower capital costs were achieved by a reduction in tonnage from 100,000 to 52,600 t/d (90,700 to 47,700 mt/d) resulting in a smaller process plant facility, elimination of two previously planned fresh water supply reservoirs due to the inclusion of a fresh water supply from a local aquifer, elimination of a permanent camp as a result of the planned daily transport of workers to the mine site during operation, changes in Project execution strategy for the placement of large development earthworks using mine waste by owner instead of contractor, and design changes to focus on bulk fills instead of cut/fills during construction.

The lower PFS OPEX was achieved by adjustments and optimization of the mine, process, and G&A. The following summarizes the major improvements realized:

- Mine design changes that lowered costs include a more direct haul route to the primary crusher and steeper pit slopes in the early phases. Changes that increased mine costs were a higher drill and blast cost for enhanced blast fragmentation to optimize mill throughput;

- Process plant configuration changes that contributed to an overall reduction in CAPEX and OPEX include the addition of secondary crushing ahead of the SAG mill for more efficient use of power, inclusion of a single line SAG / ball mill configuration with pebble crushing, and simplification of the mill foundation and pebble crushing circuit;
- Metallurgical studies completed since 2013 supported a significant process OPEX reduction through a combination of increasing grind size from a P_{80} of 90 µm to 180 µm, reducing leach circuit retention time from 32 to 21 hours, and reducing per ton reagent consumption. Lower power and reagent costs, based on updated data, also contributed to a reduction in OPEX;
- Total G&A costs went down due to reduced corporate overhead estimates based on current data, but went up on a unit basis due to lower throughput of the PFS.

25.3 Key outcomes

The key outcomes of this PFS study are:

- The Livengood Gold Project mineral resource is estimated at 497.3 M measured tonnes at an average grade of 0.68 g/mt (10.84 Moz) and 28.0 M indicated tonnes at an average grade of 0.69 g/mt (0.62 Moz), for a total of 525.4 Mmt at an average grade of 0.68 g/mt (11.5 Moz);
- This PFS has converted a portion of these mineral resources into proven reserves of 377.7 Mmt at an average grade of 0.71 g/mt (8.62 Moz) and probable reserves of 14.0 Mmt at an average grade of 0.72 g/mt (0.353 Moz), for a total of 391.7 Mmt at an average grade of 0.71 g/mt (8.97 Moz);
- Annual mining rate of 55 Mmt and a life of mine waste rock to ore ratio of 1.3:1. Maximum size of the low grade ore stockpile is 131 Mmt;
- The mine plan would provide sufficient ore (LOM Au head grade of 0.71 g/mt) to support an annual production rate of approximately 294,100 oz/y over an estimated 23 year mine life, producing a total of approximately 6.8 Moz;
- Metallurgical testwork has confirmed the preferred flowsheet consisting of primary crushing, secondary crushing and a comminution circuit (SABC configuration) producing a final grind size of 180 µm (P_{80}), with gravity recovery followed by whole ore leaching of the gravity tailings. LOM gold recovery is estimated to be 75.3% based on the rock types tested;
- Important Project surface infrastructure include:
  - O'Connor Creek Substation and 50 mi (81 km) of new 230 kV transmission line;
  - Administration, dry, maintenance, and warehouse complex;
  - Fresh water wells, pumping and distribution system;
  - Waste rock, low grade ore and growth media stockpiles;

  - Tailings management facility with capacity for approximately 450 Mt (408 Mmt) of mill tailings along with a supernatant pond. Design incorporates best practices including a lined rock fill structure with a lined tailings basin.
- The initial capital cost (-20% / +25% accuracy) of the open pit mine, 52,600 t/d (47,700 mt/d) process plant and general site infrastructure is estimated at $1.84B including a contingency of $213M;
- LOM Project sustaining capital costs total $866M, including reclamation costs of $219M;
- The mining cost is estimated at $1.73/t mined, process plant operating cost is estimated at an average of $7.48/t ore processed, and general and administrative costs of $1.28/t ore processed;
- All-in sustaining cost of production of $976/oz over the LOM including reclamation expenses, royalties and sustaining capital;
- The total power demand is estimated to be approximately 55 MW (including network losses and a 5 MW contingency);
- Over the life of mine, the total number of personnel averages 331, including mining, processing and G&A. The total personnel numbers peak in Year 4 at 387 employees;
- 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.

25.4 Indicative economics

The financial analysis performed as part of this PFS using the base case assumptions results in a negative after-tax net present value (NPV) of $ -552M at a 5% discount rate and an internal rate of return (IRR) of 0.5% after mining and income taxes. The payback period is 22.1 years. The all-in after tax cost of production is $1,263/oz over the LOM including capital, reclamation expenses, royalties, mining and income taxes.

The results of the PFS indicate that the proposed Livengood Gold Project is technically feasible, but is not economic at the base case gold price of ($1,250/oz). However, development of the Project could have the potential to generate positive results with additional efforts. The Project QPs recommend that, prior to advancing the Project to the feasibility study level, an optimization phase be completed to improve the Project economics, study potential opportunities, and reduce overall implementation risk.

25.5 Project risks and opportunities

As with most mining projects, there are risks that could affect the economic viability of the Project. Many of these risks are based on a lack of detailed knowledge and can be managed as more sampling, testing, design, and engineering are conducted at the next study stages. Table 25-1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches that could affect the technical feasibility and economic outcome of the Project.

External risks are, to a certain extent, beyond the control of the Project proponents and are much more difficult to anticipate and mitigate. Although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the Project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates.

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the Project. The major opportunities that have been identified at this time are summarized in Table 25-2 excluding those typical to all mining projects, such as changes in metal prices, exchange rates, etc. Further information and assessments are needed before these opportunities should be included in the Project economics.

Table 25-1: Project risks (preliminary risk assessment)

Area	Risk and Potential Impact	Possible Mitigation Approach
Geology and Resource Estimation	1. *Use of Reverse Circulation drilling.* ITH has used both core and reverse circulation (RC) drilling above and below the water table. The use of RC drilling beneath the water data can result in inaccurate assay data, due to cyclicity and/or downhole contamination.	1. Detailed analysis of drilling data indicated the potential for cyclicity contamination in portions of six holes and one entire drillhole. The data for the affected intervals was removed from the database used for resource calculation. Similar analyses for downhole migration of mineralized material indicated that significant downhole contamination is not an issue.
	2. *Resource Modeling.* The Multiple Indicator Kriging (MIK) method used for resource estimation block modeling provides an estimate of the proportion of a parent block/panel that is above a given cut-off grade, and the average grade of that material, based on an assumed size of Selective Mining Unit (SMU). An inappropriate SMU size could produce an inadequate assessment of the internal dilution encountered in mining.	2. The SMU size (one quarter of the parent block size of 15 m) assumed for MIK post-processing is consistent with the mining method/equipment assumed. Grade and tonnage detail was calculated at the SMU level and should provide adequate assessment of potential dilution at the mining stage.

Area	Risk and Potential Impact	Possible Mitigation Approach
Open Pit Mining	3. *Unable to meet schedule.* Owner's mining fleet may be unable to meet the construction schedule, resulting in delay and cost overruns.	3. Develop detailed Project plan to ensure adequate fleet size.
Metallurgy and Process Plant	4. *Reduction in gold recovery.* Preg-robbing or deleterious minerals/elements may cause a reduction in overall gold recovery. 5. *Unable to achieve throughput.* A limited number of samples with large top size were available for testing to size grinding mills. 6. *Gold recovery uncertainty.* Variability of the deposit could result in variances with respect to predicted overall gold recoveries. 7. *Undersized gravity circuit.* A large gravity recovery potential of the Livengood ore has been observed in different testwork phases. If the gravity circuit is undersized, a large proportion of coarse gold will report to the CIL circuit where, due to conventional leaching conditions, it will not be leached efficiently. 8. *Gold recovery*. Reduced gold recovery due to antimony. Evidence of detrimental effect of antimony on gold recovery has been reported in RT7 and RT9.	4. Conduct additional metallurgical testing to determine the difficult mineral zones or rock types and identify zones in the geological model so that selective mining, process, or stockpile blending could be implemented. 5. Obtain additional samples during feasibility study. 6. Perform additional testwork. 7. Conduct modeling of the different scenarios and design the gravity circuit to be able to handle more gravity gold (i.e. higher mass pull) than conventional operations. 8. CIL testwork using current optimized reagent conditions (O_2, lead nitrate, pH, etc.) should be conducted using well characterized samples (gold head grade, quartz/stibnite/Jamesonite, etc.) to evaluate and, where possible, reduce the current magnitude of the detrimental effect of stibnite on gold recovery.

Area	Risk and Potential Impact	Possible Mitigation Approach
Infrastructure	9. *Higher CAPEX due to unknown subsurface conditions:* The Project has a large surface footprint. While subsurface ground conditions have been investigated by drilling, not all areas have been completely investigated. 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.	9. Additional geotechnical investigations should be performed as the Project advances to further reduce the risk associated with unknown subsurface conditions.
	10. *Large earthworks quantity:* The Project requires excavation, processing, movement, placement, and preparation of a large quantity of soil, colluvium, alluvial material, and rock. 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.	10. A detailed Project execution plan should be developed that identifies contractor and owner work activities and schedules with critical path items identified along with key milestones.
Water Management	11. *Fresh water requirements may be higher than planned.* Permitting of a fresh water source could be a challenge for location and acceptance of the fresh water intake.	11. Plan for alternative freshwater pumping from the Tolovana flood plain.
	12. *Operation of storm water pump stations:* The water balance model for the tailings management facility indicates that the facility is appropriately sized for the operations and will contain all required tailings, process solutions and storm water. It also indicates that storm water diversion and pumping is critical to maintaining this capacity.	12. Further refinement of the site geotechnical and hydrological conditions at the storm water management pump stations is required to verify the capture and diversion efficiency of these structures. This will reduce the risk of excess water at the TMF due to storm water run-on.

Area	Risk and Potential Impact	Possible Mitigation Approach
Tailings Management	13. *Deleterious waste rock.* Some waste rock, proposed to be used as construction material, may have acid generating or arsenic leaching potential. 14. *Large area of liner installation:* The Project will require the surface preparation and placement of approximately 27 Mft^2 (2.5 Mm^2) of LLDPE liner at the TMF during the construction of the starter facility and prior to production. There is a 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.	13. Identify additional testing, incorporate the results into the geological block model. 14. A detailed Project execution plan should be developed, which will identify required milestones for the earthworks and production rates for the liner installation.
Execution Plan	15. *Less than optimum Project start date.* The PFS execution plan assumes a July 1 mobilization date for construction activities. 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 July 1 could have negative implications to both the execution schedule and Project cost.	15. A detailed Project execution plan should be developed, which will identify alternative approaches to minimize the construction timeline and Project cost.

Table 25-2: Project opportunities (preliminary opportunity assessment)

Area	Opportunity Explanation	Benefit
Geology and Resource Model	1. *Improve resource model to better support metallurgical modeling.* Analyze existing pulps to improve litho-structural and zonation models.	1. Potentially improve overall Project recovery by isolation of zones with varying recoveries.
	2. *Alternative Resource Model.* Develop an alternative resource model based on grade shells.	2. An alternative to the existing MIK model could support a mine production schedule with improved head grades. Improved Project economics.
Metallurgy and Process Plant	3. *Increase recovery.* Complete additional testwork to better optimize critical variables.	3. Improved Project economics.
	4. *Model feed source for gravity circuit based on the latest testwork results.* The modeling of the gravity circuit with focus on the gravity feed source (cyclone feed or cyclone underflow) should be evaluated together with gravity circuit layout.	4. The cyclone feed option presents potential savings in civil structure and slurry pumping if modeling confirms similar gravity recovery. Two stages of gravity recovery could also improve recovery and reduce downstream OPEX.
	5. *Conduct settling tests and continuous leaching and detoxification testwork at (P_{80})180* µm. Coarser grinding product has the advantage of having a lower level of reagent consumers, particularly cyanide consumption. Coarse material also has a higher settling rate.	5. Potential reduction on reagents and equipment sizing (i.e. smaller thickeners). Potential OPEX and CAPEX reduction.
	6. *Conduct grinding simulations based on yearly composite hardness data.* The simulations should include the annual mix of rock types to be sent to the process plant. There is an opportunity to increase throughput in the early years based on the high proportion of soft material (RT4) available.	6. Higher tonnages will be expected during the early years. Improved Project economics.

Area	Opportunity Explanation	Benefit
	7. *Optimize location of secondary crushing (pre-crushing).* It is recommended to perform a techno-economic tradeoff study of potential secondary crushing locations with the flowsheet to determine the impact on overall plant availability, OPEX and CAPEX. 8. *Model carbon loading/elution.* Modeling and simulations to aid in the selection of the most appropriate carbon elution circuit. 9. *Investigate the lease of an oxygen plant versus air compressors*. The oxygen produced by an oxygen plant is more concentrated than the oxygen delivered by air compressors. Even if it is not clear if gold recovery improves with oxygen, the maintenance costs are cheaper with an oxygen plant.	7. Locating secondary crushing before the stockpile has the benefit of producing material with better handling characteristics. The finer crushed ore could potentially improve SAG mill run time due to fewer maintenance interruptions. 8. An optimized carbon elution circuit will allow more batches per day based on the volume of carbon to treat per day from the CIL circuit and gold loading on carbon. 9. Based on previous project experience leasing an oxygen plant is more economic than installing air compressors for CIL circuit. Potential OPEX reduction.
Water and tailings management	10. *Reduce earthworks quantities.* Continue to refine the TMF design through a detailed grading of the basin and the south slope of the Livengood Valley, which constitutes approximately one-half of the required mine waste rock fill.	10. This opportunity could reduce the volume of waste rock required during construction. Potential CAPEX reduction
Execution Plan	11. *Alternative construction techniques.* Investigate the application of the following concepts to the Project: - Pre-assembly of leach tank bridges, structural steel, and pipe racks; - Pre-welding of tanks and piping offsite; - Use of pre-cast foundations; - Use of pre-fabricated buildings for offices and non-industrial use facilities.	11. These concepts could compress the construction schedule and reduce preproduction CAPEX.

Note: Opportunities 1-5 should be pursued during the proposed optimization phase; opportunities 6-11 should be deferred until the feasibility study is initiated.

26. RECOMMENDATIONS

26.1 Summary

The revised Livengood Gold Project configuration and flowsheet developed as a result of the 2016 PFS reduced the all-in after tax cost of production for the Project to $1,263/oz, as compared to the 2013 FS. The PFS identified additional optimization opportunities with the potential to improve recovery or further reduce costs, either of which could result in further improvement to the Project. It is recommended that these opportunities be pursued to better define overall project economics prior to initiation of a full feasibility study. It is also recommended that environmental work continue to support project development and maintain continuity of baseline information.

It is estimated that the optimization studies and supporting field work would cost approximately $6.30M. A breakdown of the key components of these studies is summarized in Table 26-1

Table 26-1: Cost estimate for optimization studies

Activities	Estimated Cost ($M)
Geology and Resource Modeling	$0.90
Mine	$0.25
Metallurgical Studies and Testwork	$3.09
Environmental	$1.00
Sub Total	**$5.24**
Contingency (20%)	$1.05
Total	**$6.30**

Sections 26.2 to 26.6 summarize the key recommendations arising from this study.

26.2 Geology and resource modeling

Optimization activities and recommendations include:

- Improve Project geological models to better support metallurgical modeling;
  - Metallurgical testwork has indicated that gold recovery of rock type 7 (RT7), and perhaps rock type 9 (RT9), is negatively impacted by the presence of antimony in the form of qtz-stibnite and jamesonite mineralization. The Project is currently limited in its ability to model the spatial distribution of this antimony mineralization in three dimensions (3D), because only 50% of the Project drill intercepts were assayed for antimony. To date, conservative estimates have been used to apply test data to the resource. Analyzing the pulps of all drill intercepts in the PFS pit would allow modeling and variance analysis to be completed to potentially improve the project litho-structural and zonation models and thereby potentially improve overall project recovery by isolation of zones with varying recoveries. – Cost $650,000;
- Develop grade-shell resource model;
  - A preliminary grade-shell resource model has indicated potential for developing a block model that could support a mine production schedule with improved economics compared to the MIK model used for the PFS. Further work is warranted to completely develop and validate this model. – Cost $250,000.

26.3 Mine

Optimization activities and recommendations include:

- Review and optimize the production schedule and stockpiling strategy based on updated resource model. – Cost $250,000.

26.4 Metallurgical testwork

Optimization activities and recommendations include:

- Advance metallurgical testwork to continue to optimize flowsheet and evaluate whether recovery can be improved;
  - The PFS work has indicated that the project economics are sensitive to recovery, grind size, reagent consumption and test conditions (oxygen, pH, lead nitrate). Completion of additional testwork on the remaining 100 kg of Phase 9 composites is recommended to allow better optimization of these critical variables. – Cost $850,000;
  - Complete gravity circuit simulations of cyclone feed vs. cyclone underflow to optimize gravity recovery. – Cost $135,000;

- After completion of the updated geologic and resource model, conduct additional testwork on new samples composited from available core, followed by testwork on several hundred kilogram composites prepared from existing inventory of reverse circulation rig duplicates. Samples should be composited with guidance from the new resource model, as appropriate, to include blends of early year production.
  – Cost $2,100,000 and includes the following sub-tasks:
  - Complete rheological testwork, including static and dynamic settling, on the composites prepared and processed above to evaluate potential reagent savings;
  - Complete confirmatory cyanide detoxification testwork at 180 µm (P_{80}) to evaluate potential reagent savings;
  - Complete confirmatory CIL vs. CIP testwork.

26.5 Environment

Continue to advance needed environmental baseline studies in support of future permitting. – Cost $1,000,000

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Appendix A: Properties and claims

Table A1: 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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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
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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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
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	40	105

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
			F008N004W18		
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
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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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

Table A2: 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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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
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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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
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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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
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

Claim Owner	ADL Number	Parcel Name	Meridian Township Range and Section	Acres	Count
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

Table A3: 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

Table A4: 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%

Table A5: 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

Township	Range	Section	BLM Claim #	Claim Name
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
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

Township	Range	Section	BLM Claim #	Claim Name
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
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

Table A6: 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

Table A7: 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

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

Table A8: Tucker Lease – Federal Unpatented 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

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