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Preliminary Economic Assessment
Fairbanks North Star Borough, Alaska USA

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LIST OF ACRONYMS

Acronym	Definition	Acronym	Definition
ADL	Alaska Division of Lands	LR2000	US Bureau of Land Management online Legacy Rehost System (BLM land status)
ADEC	Alaska Department of Environmental Conservation	MHT	Alaska Mental Health Trust Land Authority
ADR	Absorption, Desorption and Refining	MRSF	Mine Rock Storage Facility
AOI	Area of influence	MSGP	Multisector Stormwater General Permit
APDES	Alaska Pollution Discharge Elimination System	NEPA	National Environmental Policy Act
APMA	Application for Permits to Mine in Alaska	NAD	North American Datum
AQIA	Air Quality Impact Assessment	NOAA	National Oceanic and Atmospheric Administration
ADNR	Alaska Department of Natural Resources	NRCS	National Resource Conservation Service
CAPEX	Capital cost estimate	NSR	Net Smelter Royalties
CEQ	Council of Environmental Quality	OPEX	Operating expenses
CFS	Code of Federal Regulations (U.S. Federal Code)	POD	Point of diversion
CO2	Carbon dioxide	PSD	Prevention of significant deterioration
CWA	Clean Water Act	PTE	Potential to emit
DDH	Diamond drillhole	PZM	Precipitation Zone Method
DEM	Digital Elevation Model	QAPP	Quality Assurance Project Plan
DRI	Desert Research Institute	RAB	Reverse Air Blast (drillhole)
DST	Dry stack tailings	RC	Reverse circulation (drillhole)
EA	Environmental Assessment	RCH	MODFLOW recharge
EIS	Environmental Impact Statement	SAG	Semi-autogenous grinding
EPM	Equivalent porous media	SCS	Soil Conservation Service (or NRCS, National Resource Conservation Service)
ET	Evapotranspiration	SDR	Standard dimension ratio
FA/AA	Fire Assay with Atomic Absorption finish, analytical technique for gold analysis	SEDAR	System for Electronic Document Analysis and Retrieval
FEI	Fairbanks Exploration Inc.	SFR	MODFLOW Stream Flow Routing
FGMI	Fairbanks Gold Mining Inc.	SWWB	Site-wide water balance
F.M.	Fairbanks Meridian	TMT	Tentative Minimum Tax
FNSB	Fairbanks North Star Borough	TSF	Tailings Storage Facility
GHB	General head boundaries	TU	Tritium Unit
GIS	Geographic Information System	UIC	Underground injection control
GMWL	Global Metric Water Line	USACE	U.S. Army Corps of Engineers
GPS	Global Positioning System	USEPA	U.S. Environmental Protection Agency
HDPE	High density polyethylene	USGS	U.S. Geological Survey
HLP	Heap Leach Storage Facility	UTM	Universal Transverse Mercator
ICP	Inductively Coupled Plasma (geochemical analytical method)	WEL	MODFLOW well
IP	Induced polarization	WMB	Water management basin
LLDPE	Linear Low-Density Polyethylene	WMC	Water Management Consultants
LMPT	Large Mine Permitting Team	WRCC	Western Region Climate Center

LIST OF ABBREVIATIONS

Abbreviation	Definition	Abbreviation	Definition
$\mu g/m^3$	micrograms per cubic meter	Kz	vertical conductivity
µm	micrometers (microns)	lb	pound
ac-ft	acre-feet	lb/t	pounds per ton
amsl	above mean sea level	LF	linear foot
cfm	cubic feet per minute	LoM	life of mine
cfs	cubic feet per second	Ma	million years ago
cm/s	centimeters per second	m	meter
cy	cubic yards	m^2	square meter
d	day	mg/L	milligrams per liter
dmt	dry metric tonne	mg/m^3	milligrams per cubic meter
dst	dry short ton	mm	millimeter
fpm	feet per minute	MMBtu	million British thermal units
ft	feet	mph	miles per hour
ft/d	feet per day	MVA	megavolt-ampere
ft/hr	feet per hour	MW	megawatt
ft^2	square foot	opt	ounces per ton
ft^2/tpd	square feet per ton per day	Oz	ounce
ft^3	cubic foot	PAG	potentially acid generating
ft^3/d	cubic foot per day	Pcf	pounds per cubic foot
ft^3/hr	cubic foot per hour	PGM	plant growth medium
ft^3/t	cubic foot per ton	pH	hydrogen ion concentration
G	gram	PIW	pounds per inch of width
g/cc	grams per cubic centimeter	PoO	Plan of Operations
g/t	grams per tonne	ppm	parts per million
gpd	gallons per day	psf	pounds per square foot
gpm	gallons per minute	psi	pounds per square inch
h; hr	hour	Rb/Sr	Rubidium-Strontium
Hp	horsepower	Rpm	revolutions per minute
In	inch	SG	specific gravity
in/yr	inches per year	st/h	short tons per hour
Kg	kilogram	Tc	time of concentration
kg/m^2hr	kilograms per square meter per hour	Tlag	lag time
km	kilometer	TDS	total dissolved solids
kV	kilovolt	t/m^3	tonnes per cubic meter
kVA	kilovolt-ampere	toz	troy ounce
kW	kilowatt	tpd	tons per day
kWh	kilowatt hour	tph	tons per hour
kWh/t	kilowatt hour per ton	tpy	tons per year
Kxy	horizontal hydraulic conductivity	yd^2	square yard

ABBREVIATIONS OF THE PERIODIC TABLE

actinium = Ac	aluminum = Al	amercium = Am	antimony = Sb	argon = Ar
arsenic = As	astatine = At	barium = Ba	berkelium = Bk	beryllium = Be
bismuth = Bi	bohrium = Bh	boron = B	bromine = Br	cadmium = Cd
calcium = Ca	californium = Cf	carbon = C	cerium = Ce	cesium = Cs
chlorine = Cl	chromium = Cr	cobalt = Co	copper = Cu	curium = Cm
dubnium = Db	dysprosium = Dy	einsteinum = Es	erbium = Er	europium = Eu
fermium = Fm	fluorine = F	francium = Fr	gadolinium = Gd	gallium = Ga
germanium = Ge	gold = Au	hafnium = Hf	hahnium = Hn	helium = He
holmium = Ho	hydrogen = H	indium = In	iodine = I	iridium = Ir
iron = Fe	juliotium = Jl	krypton = Kr	lanthanum = La	lawrencium = Lr
lead = Pb	lithium = Li	lutetium = Lu	magnesium = Mg	manganese = Mn
meltnerium = Mt	mendelevium = Md	mercury = Hg	molybdenum = Mo	neodymium = Nd
neon = Ne	neptunium = Np	nickel = Ni	niobium = Nb	nitrogen = N
nobelium = No	osmium = Os	oxygen = O	palladium = Pd	phosphorus = P
platinum = Pt	plutonium = Pu	polonium = Po	potassium = K	prasodymium = Pr
promethium = Pm	protactinium = Pa	radium = Ra	radon = Rn	rhodium = Rh
rubidium = Rb	ruthenium = Ru	rutherfordium = Rf	rhenium = Re	samarium = Sm
scandium = Sc	selenium = Se	silicon = Si	silver = Ag	sodium = Na
strontium = Sr	sulfur = S	technetium = Tc	tantalum = Ta	tellurium = Te
terbium = Tb	thallium = Tl	thorium = Th	thulium = Tm	tin = Sn
titanium = Ti	tungsten = W	uranium = U	vanadium = V	xenon = Xe
ytterbium = Yb	yttrium = Y	zinc = Zn	zirconium = Zr	

UNITS OF MEASURE

All dollars are presented in U.S. dollars unless otherwise noted. Common units of measure and conversion factors used in this report include:

Weight:

1 oz (troy)	=31.1035 g

Analytical Values:

	percent	**grams per metric tonne**
1%	1%	10,000
1 g/t	0.0001%	1.0
10 ppb		
100 ppm		

Linear Measure:

1 inch (in)	=2.54 centimeters (cm)
1 foot (ft)	=0.3048 meters (m)
1 year (yd)	=0.9144 meters (m)
1 mile (mi)	=1.6093 kilometers (km)

Area Measure:

1 acre	=0.4047 hectare	
1 square mile	=640 acres	=259 hectares

PROJECT WORK BREAKDOWN STRUCTURE

Technical work executed by the Project team has been organized around the following work areas. In addition, the Technical Economic Model follows this hierarchical decomposition of the work. The Work Breakdown Structure can be found on the following page.

WBS #	Description	WBS #	Description	WBS #	Description
10	**Mining**	**320**	**ADR Plant**	**470**	**Plant Services**
100	**Capitalized Costs**	320.01	Carbon Columns	470.01	Control System Upgrade
100.01	Site Preparation	320.02	Stripping Circuit/Acid Wash	470.01	Expert System
100.02	Haul Roads	320.03	Electrowinning (Gold Room)	470.02	Plant Air Compressor, 434 cfm
110	**Mobile Equipment**	320.04	Dore Furnace (Gold Room)	470.02	Air Receivers, 250 gal
110.01	Rope Shovel P&H 2800	320.05	Kiln, 3'x20'	470.02	Air Dryer
110.02	Wheel Dozer Cat 854	320.05	Dewatering Screen, 4'x8', Single Deck	470.03	Fresh Water
110.03	Loader Caterpillar 992	320.05	Motor	470.04	Instrument Air
110.04	Atlas DM-45	320.06	Caustic Tank, 15,000 gal	470.05	Plant Water
110.05	Haul Truck Cat 793	320.06	Caustic Pump, 30 gph	470.06	Metallurgical/Assay Laboratory
110.06	Dozer Caterpillar D10T	320.06	Acid Tank, 15,000 gal	470.07	Sample Preparation Lab
110.07	Grader Caterpillar 16M	320.07	Acid Pump, 30 gph	470.08	Pickup Trucks
110.08	Water Wagon	320.08	Building	470.08	Skid Steer
110.09	Lube/Fuel	**330**	**Plant Services**	470.08	Forklift, Rugged Terrain
110.10	Service	330.01	Control Systems	470.08	Pickup Trucks, Flatbed, Work
110.11	Tire Truck	330.02	Plant Air	**480**	**Structures**
110.12	Caterpillar IT38H	330.03	Pit Dewatering to ADR (Pump)	480.01	Plant Building/Warehouse
110.13	Caterpillar 430E	330.03	Pit Dewatering to ADR (Pipe)	480.02	Laboratory Building
110.14	Caterpillar 256E	330.06	Pickup Trucks	**50**	**Tailings Storage Facility**
110.15	Pickups	330.06	Skid Steer	**500**	**Earthworks**
110.16	Rough Terrain Forklift	330.06	Forklift, Rugged Terrain	500.01	Site Preparation
110.17	Warehouse Forklift	330.06	Pickup Trucks, Flatbed, Work	500.02	Underdrain Installation
110.18	50 Ton Mobile Crane	**40**	**Process Plant (Sulfide)**	500.03	Liner Installation
110.19	ANFO Truck	**400**	**Site Preparation**	500.04	Overdrain Installation
110.20	Light Plant	400.01	Plant Earthworks	500.05	Embankment Construction
110.21	Mobile Crushing/Screening	**410**	**Grinding**	500.06	Reclaim Pond
110.22	Mobile Disp Sys. (90 unit)	410.01	Ball Mill, 16'x30'	**510**	**Tailings Pumping & Piping**
120	**Facilities**	410.01	Motor, Ball Mill	510.03	Tailings Slurry Piping
120.01	Mine Dry	410.01	Cyclone Feed Pumps, 10,000 gpm	510.04	Reclaim to Pond Pump
120.02	Mine Shop/Warehouse	410.01	CFP Motor	510.04	Reclaim Pond to TSF Pump
120.03	Fuel/Lube Storage Facilities	410.01	Cyclone Cluster, 26"x5	510.04	Reclaim to Pond Pipe
130	**Mine Services**	**420**	**Flotation**	510.04	Reclaim Pond to TSF Pipe
130.01	Explosives Handling	420.01	Conditioning Tank, 54,800 gal, 21x32	**60**	**Infrastructure**
130.02	Overland Conveyor	420.01	Flotation Feed Pump, 5,000 gpm	**610**	**Structures**
130.03	Gyratory Crusher	420.02	Rougher Flotation Cells, 3,500 ft^3	610.01	Mine Gate
130.04	Crusher Pocket Build	420.02	Floatation Cell Motors	610.02	Administrative Office
130.05	Crusher Install	420.02	Rougher Tailings Pump, 5,000 gpm	610.03	Security Gate
130.06	Pit Dewatering	420.02	Rougher Conc Pump	610.04	Access Roads
130.07	In-Pit Substations	420.02	Rougher Conc Pump Motor	610.05	Support Roads
20	**Crushing Circuit**	420.03	Cyanide Isotainers	610.06	Perimeter Fence
200	**Heap Leach**	420.03	Cyanide Mixing Skid	610.07	Process Fence
200.01	RoM Pad & Reclaim	420.03	Cyanide Pumps, 10 gpm	**620**	**Power Lines/Substations**
200.02	Crusher, Std Cone, 7' dia.	420.03	Cyanide Tanks, 17,000 gal	**630**	**Water Management**
200.02	Crusher, Short Head, 7' dia.	420.03	Lime Slaking Plant, 4,000 lb/hr	630.01	Fire Water Tank & Foundation
200.02	Screen, Inclined, 8x16, Double Deck	420.03	Flotation Chem System Tanks, 4,500 gal	630.01	Fire Water Pipe
200.02	Screen, Motor	420.03	Pumps, 30 gph	630.02	Potable Water
210	**Sulfide Plant**	420.04	Float Conc Thickener, 20'	630.03	Waste Water Treatment Plant
210.01	RoM Stockpile & Reclaim System	420.04	O/F Pump, 500 gpm	630.04	Pond Excavation
210.02	Motor, SAG Mill	420.04	O/F Pump Motor	630.04	Geosynthetic Liner
210.02	Screen, Inclined, 8x16, Double Deck	420.04	U/F Pump, 200 gpm	**640**	**Communications**
210.02	Motor Screen	420.04	U/F Pump Motor	**650**	**Mobile Equipment**
210.02	Crusher, Pebble	**430**	**Bioxidation**	**70**	**Construction**
30	**Heap Leach (Oxide)**	430.01	Bioxidation Tanks	700	Construction Labor
300	**Site Preparation**	430.01	Bioxidation Tank Agitators	710	Piping
300.01	Site Preparation	**440**	**CIL Plant**	720	Electrical & Instrumentation
300.01	Wells	440.01	Bioxidation Wash Thickener, 20'	730	Concrete
300.01	Underdrain Installation	440.01	O/F Pump, 1000 gpm	740	Structural Steel
300.01	Impoundment Liner Installation	440.01	O/F Pump Motor	750	Painting & Insulation
300.01	Overdrain Installation	440.01	U/F Pump, 200 gpm	**80**	**Indirects**
300.01	Embankment	440.01	U/F Pump Motor	800	Construction Indirects
300.01	Drip Lines	440.02	CIL Tanks, 35'x35'	810	Spares & Inventory
310	**HLP Equipment**	440.02	CIL Tank Agitators	820	First Fills
310.01	Tank, Barren Solution, 185 kgal	**450**	**Ancillary Equipment**	830	Freight & Logistics
310.02	Conveyors, Grasshoppers, 36"x100'	450.01	CIL Blowers, 3,200 cfm	840	Commissioning & Start-Up
310.03	Pumping & Piping	450.01	Bioxidation Blowers, 3,200 cfm	850	EPCM
310.03	PLS Soln Pump	**460**	**Cyanide Detoxification**	860	Vendor & Consultant Assistance
310.03	Barren Soln Pump	460.01	Cyanide Detoxi Tank, 43,000 gal	**90**	**Owner's Costs**
310.03	PLS Soln Pipe	460.01	Tailings Pumps, 5,000 gpm	900	Project Management
310.03	Barren Soln Pipe	460.01	Tailings Pump Motors	910	Environmental & Permitting
310.03	HLP Sub-Headers	460.01	Tailings Thickener, 100' dia	920	Mine Closure & Reclamation
310.04	Cyanide Isotainers	460.01	O/F Pump, 500 gpm	930	Exploration & Infill Drilling
310.04	Cyanide Mixing Skid	460.01	O/F Pump Motor	940	Engineering Studies
310.04	Cyanide Tank, 17,000 gal, 21'x7.25'	460.01	U/F Pump, 5,000 gpm	950	Legal
310.04	Cyanide Tank Pumps, 10 gpm	460.01	U/F Pump Motor	960	Insurance

1.0 SUMMARY

1.1 INTRODUCTION

Freegold Ventures Limited (Freegold) retained Tetra Tech, Inc. (Tetra Tech) with Mark J. Abrams, C.P.G. and Gary H. Giroux, P.Eng to prepare this preliminary economic assessment (PEA or "Report") for the Golden Summit Project (the Project) in the Fairbanks Mining District, Alaska. The purpose of this Report is to provide Freegold with an independent opinion of the technical aspects of the Project and make recommendations for future work. This Report is in compliance with National Instrument 43-101 (NI 43-101).

1.2 KEY OUTCOMES

The PEA evaluates a two-phase, 24-year open pit mine generating two gold streams, each operating at 10,000 tonnes per day (tpd). Processing operations for the oxide and sulfide resource are heap leach and bioxidation respectively. All values are presented in US$.

Based on a gold price of $1,300/oz, highlights of the Project PEA include:

- A post-tax $NPV_{5\%}$ and IRR of $188 million and 19.6% respectively;
- A mine life of 24 years with peak annual gold production of 158 thousand ounces (koz) and average annual gold production of 96 koz;
- 2,358 koz of doré produced over the life of mine;
- Total cash cost estimated at $842/oz Au (including royalties, refining and transport);
- Ability to execute Phase 1 with low initial capital; initial and sustaining capital costs, including contingency, estimated at $88 million and $348 million respectively;
- A payback of 3.3 years post-tax; and
- Favorable geopolitical climate; completion risk is offset through strong legislative and financial support at state and federal levels.

Value-enhancing opportunities, such as leased mine equipment, improved metallurgical performance through additional testing, liquid natural gas, local labor surveys, power generation sets, and local power contracts will be further investigated as the Project moves towards the Preliminary Feasibility stage. Additionally, there is potential for immediate resource expansion with continued drilling efforts within the oxide zone. Work completed to date which includes geophysical, geochemical and geological studies, indicates that there is a strong possibility to expand upon the known resource. A similar geochemical and geophysical signature over the known resource appears to extend to the north, west and southwest over distances in excess of one kilometer.

1.3 PROJECT DESCRIPTION & OWNERSHIP

The Golden Summit Property (the Property) is located 18 miles (29 km) by road northeast of the City of Fairbanks, Alaska, United States of America. It is located in the north portion of the Fairbanks Mining District (**Figure 1-1**), a northeast trending belt of lode and placer gold deposits that compose one of the largest gold producing areas in the state of Alaska.

The Property consists of 50 patented claims, 94 unpatented federal claims, and 268 State of Alaska claims which cover a total area of 14,630 acres (5,921 hectares).The Property is situated in Township 3N, Range 1E, 2E and 3E of the Fairbanks Meridian, centered at approximately 479250 E, 7215464 N (UTM Zone 6 NAD 27 Alaska).

1.4 HISTORY

Placer or lode gold mining has occurred almost continuously in the Project area since gold was discovered in the district in 1902. Over 9.5 million ounces of placer gold have been recovered from the Fairbanks Mining District, of which 6.75 million ounces have been recovered from streams that drain the Project (Freeman, 1992e). In addition, over 506,000 ounces of lode gold were recovered from past producing mines on the Project (Freeman and others, 1996). More than 80 lode gold occurrences have been documented in the Project area. Recent exploration discoveries in the Tintina Gold Belt have underscored the potential for bulk tonnage and high-grade deposits, both of which are known to exist in the Project area (McCoy and others, 1997; Flanigan and others, 2000).

Table 1-1 provides a chronology of exploration activities conducted for the property and adjacent prospects.

Table 1-1: Summary of Exploration (1969-2015) Conducted for the Property and Adjacent Prospects

Company	Years	Exploration/Mining Activity	Principle Targets
International Minerals & Chemicals	1969	Trenching RC drilling	Saddle Zone Circle Trail Zone
Placid Oil Company	1978 – 1986	Trenching Core & RC drilling Adit excavation Christina feasibility study	Christina Vein Pioneer Vein American Eagle Vein Hi Yu Vein
SC	1980 – 1981	Diamond core drilling RC drilling Resource estimate	Tolovana Shear Zone
Fairbanks Exploration	1988	Bulk sampling	Christina Vein
Keystone Mines Partnership	1989	Bulk sampling of mine waste dumps	American Eagle, Hi Yu, Cleary Hill areas
British Petroleum/Fairbanks Exploration(FEI) JV	1987 – 1988	Trenching, RC drilling	Too Much Gold prospect Saddle Zone Circle Trail Zone Christina Vein
Freegold/FEI JV	1991	Property-wide data compilation	Property-wide
Freegold/Amax Gold JV	1992 – 1994	Trenching, soil sampling, RC drilling, aerial geophysical surveys (EM), bottle roll testing, baseline water quality surveys, aerial photos, EDM surveys	Too Much Gold prospect Cleary Hill area

Company	Years	Exploration/Mining Activity	Principle Targets
Freegold	1995 – 1996	RC drilling	Dolphin area Cleary Hill area
Freegold/Barrick JV	1997 – 1998	Property-wide grid-base soils, recon & prospect mapping, grab sampling, limited RC and core drilling	Property-wide Goose Creek prospect North Extension prospect Coffee Dome Dolphin area Newsboy area Wolf Creek area
Freegold	2000	Limited core drilling	Cleary Hill area
Freegold	2002	Trenching	Cleary Hill area (Currey Zone)
Freegold	2003	Limited core drilling	Cleary Hill area (Currey Zone)
Freegold/Meridian Minerals JV	2004	Trenching, core drilling	Tolovana area Cleary Hill area
Freegold	2005 – 2006	Trenching	Cleary Hill area Wackwitz Vein area Beistline Shaft area
Freegold	2007 – 2008	Trenching, RAB drilling, core drilling, bulk sampling	Cleary Hill area Tolovana Mine area
Freegold	2010	Induced Polarization Survey	Dolphin area
Freegold	2011	Induced Polarization Survey, Geochemical Surveys, Core Drilling,	Dolphin area Cleary Hill area, Christina Prospect
Freegold	2012	Induced Polarization Survey, Geochemical Surveys, Trenching, Metallurgical Work, Core Drilling	Dolphin/Tolovana area, Cleary Hill area, Chatham, Christina Prospect
Freegold	2013	Core Drilling, Geophysics,	Dolphin, Coffee Dome area
Freegold	2014	Water Quality Sampling, Cultural Resource Studies, Metallurgical tests, Geochemical Surveys	Dolphin and Cleary Hill areas
Freegold	2015	Water Quality Sampling, Cultural Resource Studies and Geochemical Surveys	Dolphin and Cleary Hill areas

Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Freegold Ventures Limited***	Fig1-1_Location.dwg	**FIGURE 1-1** **GOLDEN SUMMIT PROJECT** **GENERAL LOCATION MAP**
	Project: Golden Summit Project Preliminary Economic Assessment	Project Number: 114-910054	
	Project Location: Fairbanks Mining District, Alaska	Date of Issue: March/2016	

1.5 Geology & Mineralization

Gold mineralization on the Golden Summit property occurs in three main forms, including 1) intrusive-hosted sulfide-quartz stockwork veinlets (such as the Dolphin gold deposit), 2) auriferous sulfide-quartz veins (exploited by historic underground mines), and 3) shear-hosted gold-bearing veinlets. All three types are considered to be part of a large-scale intrusive-related gold system on the property. The Dolphin gold deposit is hosted in the Dolphin stock, which consists largely of granodiorite and tonalite, similar to the Pedro Dome pluton. It is the only large intrusive body known on the property at this time. The Dolphin stock is approximately the same age as the nearby Fort Knox pluton, which hosts Kinross Gold's Fort Knox gold mine. Freegold made the initial discovery of widespread low-grade gold mineralization in the Dolphin stock during the initial drilling campaign on the prospect in 1995; however resource definition drilling only commenced in 2011. A total of 87 holes have been drilled within the resource area since 2011 totaling 24,156 meters.

1.6 Mineral Resources

An update of the resource reported in December 2012 (Abrams and Giroux, 2012) was estimated incorporating an additional ten drillholes completed in 2013. The update also subdivides the resource into oxide and sulfide portions. The effective date for this resource is May 31, 2013, the date that the data was received. There were three drillholes completed since this date which do not have a material effect on this resource and as a result this resource remains current. The three new holes were compared to the estimated blocks they pass through and found to correlate well. Of the total 330 drillholes on the property, 185 penetrated the three dimensional geologic Dolphin Stock solid and were used for the estimate. The gold grade distribution identified multiple overlapping lognormal populations present. Erratic gold assays were capped at 88 g/t. Uniform down-hole composites three m in length were formed to honor the solid boundaries. The gold distribution of three meter composites also identified overlapping lognormal populations and an indicator approach was used for the estimate. Semivariograms for the high grade gold indicator and low grade background were produced and used to define and orient the various search ellipses. Grades for gold were interpolated into blocks 10 x 10 x 5 meters in dimension by a combination of Indicator and Ordinary Kriging. A total of 66 specific gravity measurements showed no correlation to gold grades and as a result an average value of 2.51 was used above the oxide surface and 2.67 below this surface to convert volume to tonnage. Estimated blocks were classified based on geologic and grade continuity into Indicated and Inferred. As part of this study, a conceptual open pit, based on $1300/oz Au, has been developed. As a result only blocks falling within this pit are now reported as a Resource within the following tables.

Table 1-2: Dolphin Zone Indicated Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes >Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	oz Au
0.20	82,650,000	0.58	47,610	1,531,000
0.25	71,140,000	0.63	45,030	1,448,000
0.30	**61,460,000**	**0.69**	**42,410**	**1,363,000**
0.35	53,460,000	0.74	39,770	1,279,000
0.40	46,690,000	0.80	37,260	1,198,000
0.50	35,590,000	0.91	32,320	1,039,000
0.60	26,720,000	1.03	27,440	882,000
0.70	20,030,000	1.15	23,110	743,000
0.80	15,030,000	1.29	19,390	623,000
0.90	11,450,000	1.43	16,350	526,000
1.00	8,870,000	1.57	13,910	447,000
1.10	6,990,000	1.71	11,940	384,000
1.20	5,560,000	1.85	10,300	331,000
1.30	4,490,000	2.00	8,960	288,000

Table 1-3: Dolphin Zone Inferred Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes >Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	oz Au
0.20	95,920,000	0.58	55,350	1,779,000
0.25	82,910,000	0.63	52,400	1,685,000
0.30	**71,500,000**	**0.69**	**49,260**	**1,584,000**
0.35	61,640,000	0.75	46,050	1,480,000
0.40	52,690,000	0.81	42,730	1,374,000
0.50	38,800,000	0.94	36,510	1,174,000
0.60	28,710,000	1.08	30,980	996,000
0.70	21,700,000	1.22	26,450	850,000
0.80	16,910,000	1.35	22,880	736,000
0.90	12,890,000	1.51	19,460	626,000
1.00	10,090,000	1.67	16,820	541,000
1.10	8,350,000	1.80	15,000	482,000
1.20	7,050,000	1.92	13,500	434,000
1.30	5,880,000	2.05	12,050	387,000

Table 1-4 and **Table 1-5** show the resource present above the oxide surface, within the Conceptual Pit while **Table 1-6** and **Table 1-7** show the resource present below the oxide surface again within the Conceptual Pit.

Table 1-4: Oxide Zone Indicated Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes >Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	oz Au
0.20	22,520,000	0.55	12,270	395,000
0.25	18,960,000	0.61	11,490	369,000
0.30	**16,180,000**	**0.66**	**10,730**	**345,000**
0.35	13,990,000	0.72	10,020	322,000
0.40	12,160,000	0.77	9,340	300,000
0.50	9,180,000	0.87	8,000	257,000
0.60	6,850,000	0.98	6,730	216,000
0.70	5,030,000	1.10	5,550	178,000
0.80	3,700,000	1.23	4,560	147,000
0.90	2,800,000	1.36	3,790	122,000
1.00	2,100,000	1.49	3,130	101,000
1.10	1,650,000	1.61	2,660	85,000
1.20	1,330,000	1.72	2,290	74,000
1.30	1,040,000	1.86	1,930	62,000

Table 1-5: Oxide Zone Inferred Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes >Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	oz Au
0.20	14,660,000	0.47	6,950	223,000
0.25	11,810,000	0.53	6,310	203,000
0.30	**9,620,000**	**0.59**	**5,700**	**183,000**
0.35	8,120,000	0.64	5,220	168,000
0.40	6,910,000	0.69	4,770	154,000
0.50	4,940,000	0.79	3,890	125,000
0.60	3,360,000	0.90	3,020	97,000
0.70	2,330,000	1.01	2,360	76,000
0.80	1,690,000	1.11	1,880	61,000
0.90	1,160,000	1.23	1,430	46,000
1.00	720,000	1.41	1,020	33,000
1.10	510,000	1.57	800	26,000
1.20	360,000	1.75	630	20,000
1.30	270,000	1.91	510	17,000

Table 1-6: Sulfide Zone Indicated Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes >Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	oz Au
0.20	60,130,000	0.59	35,360	1,137,000
0.25	52,180,000	0.64	33,550	1,079,000
0.30	**45,280,000**	**0.70**	**31,650**	**1,018,000**
0.35	39,470,000	0.76	29,800	958,000
0.40	34,530,000	0.81	27,930	898,000
0.50	26,410,000	0.92	24,300	781,000
0.60	19,870,000	1.04	20,720	666,000
0.70	14,990,000	1.17	17,550	564,000
0.80	11,330,000	1.31	14,820	476,000
0.90	8,650,000	1.45	12,550	404,000
1.00	6,770,000	1.59	10,780	347,000
1.10	5,340,000	1.74	9,280	298,000
1.20	4,230,000	1.89	8,010	257,000
1.30	3,450,000	2.04	7,030	226,000

Table 1-7: Sulfide Zone Inferred Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes >Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	oz Au
0.20	81,260,000	0.60	48,350	1,554,000
0.25	71,100,000	0.65	46,070	1,481,000
0.30	**61,880,000**	**0.70**	**43,560**	**1,401,000**
0.35	53,520,000	0.76	40,840	1,313,000
0.40	45,780,000	0.83	37,950	1,220,000
0.50	33,860,000	0.96	32,610	1,048,000
0.60	25,360,000	1.10	27,970	899,000
0.70	19,360,000	1.24	24,080	774,000
0.80	15,210,000	1.38	20,990	675,000
0.90	11,730,000	1.54	18,040	580,000
1.00	9,370,000	1.69	15,810	508,000
1.10	7,840,000	1.81	14,200	456,000
1.20	6,700,000	1.93	12,900	415,000
1.30	5,610,000	2.06	11,530	371,000

1.7 Mineral Processing & Metallurgical Testing

1.7.1 Mineral Processing

Gold recovery from the Golden Summit deposit will come from two separate processing methods. Oxide material will be crushed prior to loading onto a 10,000 tpd heap leach facility. The crushed oxide material will then be leached with a sodium cyanide solution. Gold from the pregnant leachate solution will then be recovered onto activated carbon and further refined in an elution/electrowinning circuit. The product from the electrowinning cells will be further refined into gold doré. Oxide gold recoveries of 80% are expected during operation.

Sulfide material containing gold will be processed in a 10,000 tpd bio-oxidation plant. The sulfide material will be processed by crushing and grinding the material prior to flotation and bio-oxidation of the sulfide concentrate. The oxidized slurry will be sent to a carbon-in-leach (CIL) circuit for cyanide leaching and recovery onto activated carbon. Gold loaded onto the activated carbon will then be recovered in the same elution circuit used for the oxide material, to produce gold doré. Sulfide gold recoveries of 90% are expected during operation.

1.7.2 Metallurgical Testing

Sample composites from five different rock types were taken from various drill core for metallurgical testing. The five composites were subjected to over 60 cyanidation tests to investigate gold recoveries using various methods of sulfide oxidation and cyanidation. A total of 36 coarse bottle roll tests were also completed to define parameters for a single column leach test to simulate heap leaching conditions for the Oxide material. In addition to the leach tests, the five composites had Bond Ball Mill Work Indices conducted to determine comminution requirements. Head analyses for gold, silver, and sulfur were also conducted. Major conclusions from the test program include:

- Golden Summit oxide material leaches rapidly and achieves good recoveries under standard heap leaching parameters;
- Sulfide material responds favorably to multiple methods of oxidation and cyanidation;
- Gold recoveries greater than 80% were observed from the column tests; and
- Gold recoveries greater than 90% were observed from sulfide oxidation testwork.

1.8 Mining Methods

Due to the pit containing both sulfide and oxide material, there will be two methods of processing. Two sets of cut-off values were calculated; breakeven cut-off and the internal cut-off were calculated using $1,300/oz Au price for both the oxide material and the sulfide material. The oxide mine plan used a breakeven cut-off grade of 0.182 g/t Au, and an internal cut-off grade of 0.132 g/t Au. The sulfide mine plan used a breakeven cut-off grade of 0.611 g/t Au, and an internal cut-off grade of 0.566 g/t Au. The oxide will be processed via heap leach, while the sulfide will be processed through a plant. The mine has been scheduled to provide up to 3.5 million tonnes per year (Mtpy) of each material type. Oxide is mined in the early years, as it forms a cap over the sulfide material. Years in the middle of the production schedule have an overlap of oxide and sulfide production prior to completion of oxide mining. A detailed pit design was created using the pit optimizer cones as guidelines. The phases within the ultimate pit were developed to enhance the Project by scheduling higher-value material earlier in the mine life.

Oxide material will be mined and processed exclusively for the first eight years of the mine production. A small amount of sulfide material will be mined before year eight; this sulfide material (approximately 800,000 tonnes) will be stockpiled until the end of mine life. In year nine, the sulfide material comes online for production. Mining of the oxide material will continue through year 14 of the 24-year mine life. Mining of sulfide material will continue from year nine through the end of the 24 -year mine life.

During production, both oxide and sulfide material will be transported from the pit to the primary crusher located near the pit exit. After primary crushing, oxide and sulfide material will be transported by conveyor to its respective process area. The oxide will be leach processed in an area to the southeast of the pit, while the sulfide will be processed northwest of the pit.

Waste will be hauled by truck to the Mine Rock Storage Facility (MRSF). The MRSF has been designed to permanently contain the overburden and waste material associated with the pit. The current MRSF design, located to the northeast of the pits, is built around the hill. The MRSF was designed with a buffer around the nearby creeks. The total MRSF design will contain 100% of the expected waste material planned to be generated - approximately 239 million tonnes of swelled material.

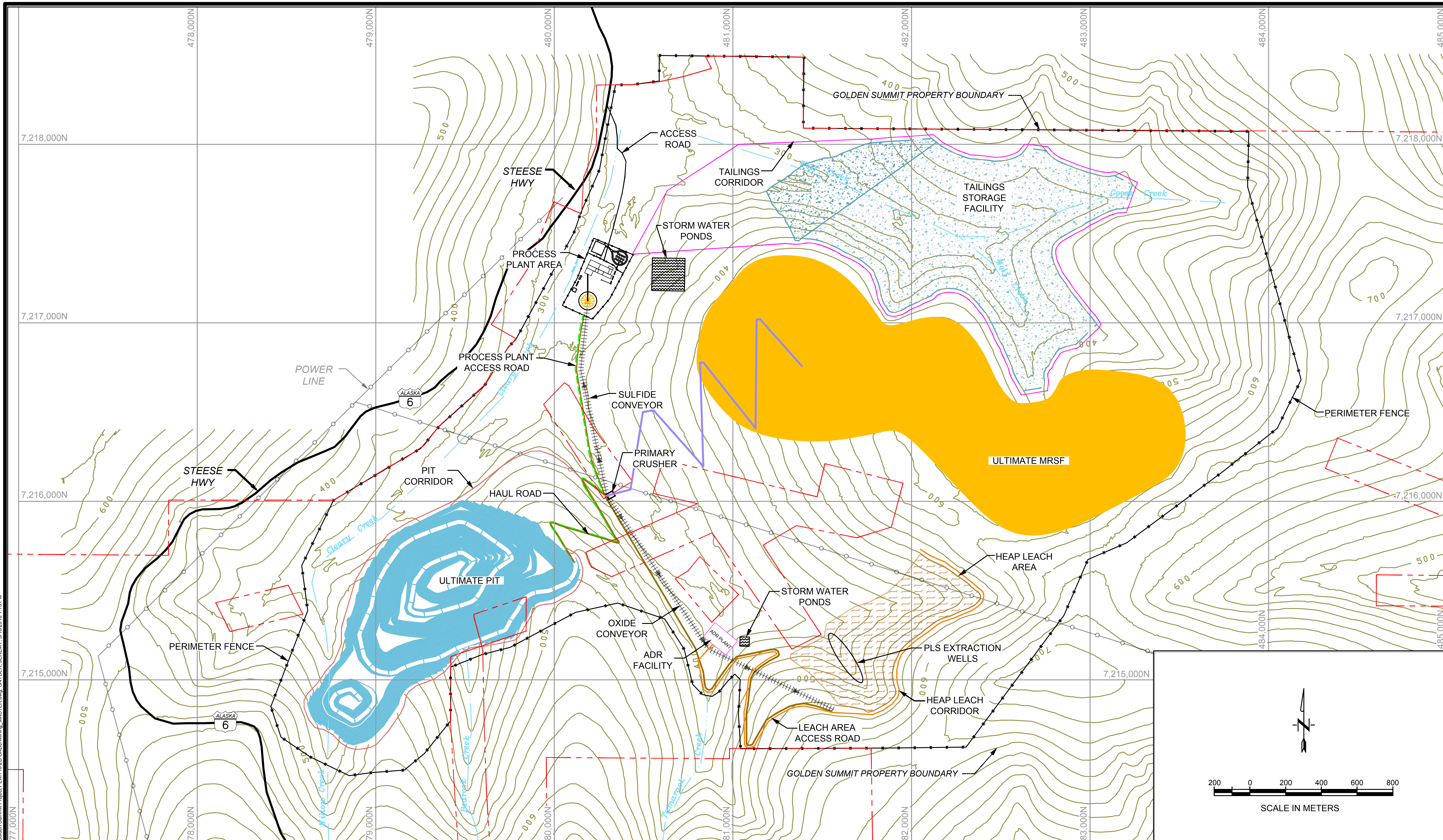
The mine has been planned using diesel blasthole drills, large haul trucks and rope shovels. Primary mine production is achieved using 64 Mt payload rope shovels along with 227 Mt payload haul trucks. The drills, shovels and haul trucks selected for the Project are scheduled to operate around the clock and require four crews on 12-hour shifts for complete shift coverage.

1.9 INFRASTRUCTURE

The following key infrastructure will support the mine and process facilities:

- From Fairbanks, Alaska the Project lies approximately 29 km (18 miles) northeast via State Highway 2 and State Highway 6 (the Steese Highway). The site holds a series of gravel roads which allow access to most areas of the property on a year-round basis. Fairbanks is served by the Alaska Railroad, and is connected to Anchorage and Whitehorse, Canada by well-maintained paved highways.
- Heap leach pad and solution storage;
- Conventional slurry tailings storage facility to serve the sulfide processing facility;
- Processing, truck shop, warehouse, and administration buildings;
- Substation and power distribution; and
- Potable water, fire water and sewage treatment systems.

Fairbanks and its surrounding area serves as the regional service and supply center for interior Alaska and comprises a total population of approximately 100,000. Labor will come from the Fairbanks area where there is ready access to trained personnel. In addition the State of Alaska allows $20M of exploration expenditures to be carried forward and recovered against State taxes due. The general site layout is provided in **Figure 1-2**.



B	EDIT TO MRSF	JS	03/2016	
A	EDIT TO MRSF	LA	12/2015	
Rev	Description	BY	Date	

REFERENCE

REVISIONS

ENGINEER'S SEAL

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

Scale: As Noted

Designed by: TETRA TECH
Drawn by: L. AGA
Checked by: E. LIPS
Approved by: V. SCHARNHORST

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Issued for: Freegold Ventures Limited

Issued by: TETRA TECH

GOLDEN SUMMIT PROJECT
GENERAL ARRANGEMENT

B REVISION

Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT

Project no.: 114-910054

Location: FAIRBANKS MINING DISTRICT, ALASKA

Date: March/2016

FIGURE 1-2

1.10 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

The Project area lies within the Cleary Creek watershed and in addition to Cleary Creek, includes the drainages of Willow Creek, Bedrock Creek, Chatham Creek, Fairbanks Creek, Too Much Gold Creek, and Wolf Creek. The Cleary Creek basin is tributary to the Chatanika River. To date, a limited amount of baseline environmental data have been collected in the Project area to characterize water resources, water quality, wetlands, aquatic resources, on-site meteorology, subsistence use and cultural resources. An evaluation under Section 106 of the National Historic Preservation Act (NHPA) concerning the historic status of a former ski area within the Project area has been conducted by the Alaska State Historic Preservation Office (SHPO). Additionally, an initial evaluation of waste rock geochemistry has also been conducted.

Baseline environmental data will be required including on-the-ground studies to delineate jurisdictional wetlands. These data will be required to meet a number of needs including permitting and mine design and location of facilities, mine construction and operations. Freegold has initiated consultation with the State's Large Mine Permitting Team (LMPT) to begin the process of project planning, development and environmental permitting. Through this process, the LMPT will assist in developing a broader environmental baseline program.

1.11 CAPITAL & OPERATING COSTS

1.11.1 Capital Costs

Life of mine (LoM) capital cost requirements are estimated at $437 million as summarized in **Table 1-8**. Initial capital of $88 million is required to commence operations and a sustaining capital of $348 million.

Table 1-8: LoM Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
Direct Costs				
10	Mining	$39,744	$110,784	$150,528
20	Crushing & SAG Mill Circuits	$3,921	$9,884	$13,805
30	Heap Leach (Oxide)	$11,410	$23,723	$35,133
40	Process Plant (Sulfide)	$0	$27,894	$27,894
50	Tailings Storage Facility	$0	$67,774	$67,774
60	Infrastructure	$10,131	$11,000	$21,131
70	Construction	$12,095	$56,903	$68,998
	Direct Costs	**$77,301**	**$307,962**	**$385,263**
Indirect Costs				
800	Construction Indirects	$456	$2,232	$2,688
810	Spares & Inventory	$342	$1,674	$2,016
820	First Fills	$342	$1,674	$2,016
830	Freight & Logistics	$799	$2,789	$3,588
840	Commissioning & Start-Up	$342	$1,674	$2,016
850	EPCM	$1,369	$4,184	$5,553
860	Vendor & Consulting Assistance	$228	$1,116	$1,344
	Indirect Costs	**$3,879**	**$15,342**	**$19,221**
90	Owner's Costs	$7,240	$24,984	$32,224
	Total Capital	**$88,420**	**$348,288**	**$436,708**

The open pit mine utilizes some leased mobile equipment. Leases are capitalized during the preproduction period, then reported in the operating costs during the production.

1.11.2 Operating Costs

LoM operating costs are summarized in **Table 1-9**. Open pit mining costs, as reported in this table, do not include the lease costs. Lease unit costs are shown separately.

Table 1-9: LoM Operating Costs

Description	$/t-moved	$/t-Mined	$/oz-gold
Mining	$3.04	$10.56	$441.68
Mining Lease	-	$1.06	$44.53
Crushing Circuit	-	$0.91	$38.10
Heap Leach (Oxide)	-	$1.20	$50.18
Process Plant (Sulfide)	-	$4.44	$185.59
Tailings Storage Facility	-	$0.12	$4.96
Infrastructure	-	$0.31	$13.09
Direct Operating Cost	**-**	**$18.60**	**$778.13**
Property Tax	-	$0.15	$6.10
Mining License Tax	-	$0.57	$23.74
Operating Cost	**-**	**$19.31**	**$807.97**

Refining charges, transportation, and royalties are not included in the operating cost estimate.

1.12 ECONOMIC ANALYSIS

The following preliminary economic assessment analysis includes inferred mineral resources which are considered too speculative geologically to have economic considerations applied to them, and are therefore not categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

Project cost estimates and economics are prepared on an annual basis. Based upon design criteria presented in this report, the level of accuracy of the estimate is considered ±35%.

Project economics are based primarily on inputs developed in the preliminary economic assessment. Economic results suggest the following conclusions:

- Mine Life: 24 years;
- Pre-Tax $NPV_{5\%}$: $213 million; IRR: 20.0%;
- Post-Tax $NPV_{5\%}$: $188 million; IRR: 19.6%;
- Payback (Post-Tax): 3.3 years;
- Federal Income Taxes Paid: $58 million;
- State Income Tax Paid: $21 million;
- Mining License Tax Paid: $55 million;
- Cash costs of $842/oz; and
- Initial project capital of $88 million, sustaining project capital of $348 million, and total project capital of $437 million.

1.13 INTERPRETATIONS & CONCLUSIONS

1.13.1 Geology

Three main rock units underlie the Property, including rocks of the Fairbanks Schist, rocks of the Chatanika Terrane, and intrusive rocks (**Figure 7-2**). The Fairbanks Schist and Chatanika Terrane have both been subjected to one or more periods of regional metamorphism. The intrusive bodies are post-metamorphism. Chatanika Terrane rocks are found structurally above the Fairbanks Schist and north of the Chatanika Thrust fault and comprise the northernmost portion of the property. Intrusive rocks are relatively minor on the Property, and are primarily represented by the Dolphin stock, although small granitic dikes are known in several locations.

The Dolphin stock is located on the ridge between Bedrock and Willow Creek. Initial diamond core logging identified five intrusive phases within the Dolphin stock, including: 1) fine- to medium-grained, equigranular to weakly porphyritic biotite granodiorite; 2) fine- to medium-grained, equigranular to weakly porphyritic hornblende-biotite tonalite; 3) fine-grained biotite granite porphyry; 4) fine-grained biotite rhyolite to rhyodacite porphyry; and 5) rare fine-grained, chlorite-altered mafic dikes (Adams and Giroux, 2012).

Limited drill data suggests the north and west contacts of the Dolphin stock are fault contacts (Adams and Giroux, 2012). The south and east contacts are largely intrusive contacts with minor faulting

1.13.2 Mining

Mine production constraints were imposed to ensure that mining wasn't overly aggressive with respect to the equipment anticipated for use at the Project. The schedule has been produced using mill targets and stockpiling strategies to enhance the project economics. The constraints and limits used are reasonable to support the project economics.

Pit designs were created using 10 m benches for mining with a catch bench every level. This corresponds to the resource model block heights, and Tetra Tech believes this to be reasonable with respect to mining loss and the equipment anticipated to be used in mining.

1.13.3 Groundwater Hydrogeology

Estimates of groundwater conditions at the project site are based on records from existing groundwater wells at and near the Project site and on conditions observed at the Fort Knox mine, which is approximately 5 km (3 mi) to the south of the project site and is considered to provide a good representation of the conditions at the project site.

Groundwater is expected to be present in two units: unconsolidated deposits consisting of alluvium and dredge tailings along the valley floors, and fractured bedrock throughout the property. The degree of bedrock fracturing, and therefore the hydraulic conductivity, are expected to be highly variable. Reported depths to groundwater in nearby water wells ranged from 2.1 m (6.9 ft) below the land surface in the valley bottoms to 68.6 m (225 ft) below the land surface in upland areas. Reported yields of water supply wells ranged from 16 to 491 m^3/day (3 to 90 gpm), and dewatering wells at the Fort Knox mine were reported to have capacities up to approximately 1,000 m^3/day (183 gpm). Groundwater flow on a local scale is anticipated to be from bedrock in the upland areas toward the valleys and thence down-valley in the alluvial deposits or dredge tailings. Regional-scale groundwater flow cannot be determined from available data.

Planned open pit mining at the property would extend below the water table, and dewatering would be required for maintaining pit wall stability and dry conditions within the pit. Because of weather conditions, a well system would likely be the most feasible dewatering method. The mine pit would intersect the water approximately six months to one year after the start of mining, but dewatering would need to start earlier in order for the pumping effects to extend throughout the required area. The estimated annual average pumping rate was approximately 410 m^3/day (75 gpm) initially, increased to approximately 4,460 m^3/day (818 gpm) by the third year of mining, declined slightly through the eighth year of mining, and then increased gradually to approximately 6,600 m^3/day (1,210 gpm) near the end of the mine life. The number of wells required for dewatering is estimated to range from two initially to 16 later in the mine life.

Data would need to be collected to characterize the site-specific hydrogeologic conditions and develop site-specific designs for dewatering.

1.13.4 Metallurgy & Process

Sufficient metallurgical testwork has been completed on samples from the Project deposit to determine the preferred processing methods to recover gold from oxide and sulfide materials at a PEA level study. The oxide material was shown to be highly amenable to heap leaching. The testwork showed that oxidation of the sulfide material was needed to achieve acceptable gold recoveries. The oxidation methods tested were able to achieve acceptable recoveries, but high capital cost requirements made those methods un-feasible for this PEA study. The processing method chosen for the sulfide material was bio-oxidation followed by cyanide leaching. While bio-oxidation testwork has not been performed on the deposit material, the high recoveries achieved throughout the testwork indicate that the sulfide material would be amenable to bio-oxidation.

1.13.5 Environmental

Development of the project will require extensive environmental baseline analyses, assessment of environmental impacts and evaluation, and associated permitting requirements reflective of the direct, indirect and cumulative impacts associated with full project build-out, and the sensitive environment in which it is to be constructed. The complexity of the environmental impact review and permitting of the various facilities will be dependent on siting of facilities in relationship to the various creeks and valleys surrounding the project development target areas. This PEA provides preliminary siting information of facilities such as tailings disposal, waste rock, and leach pads. Baseline and environmental studies that will be required to move the project toward permitting can now be planned, implemented, and modified as necessary as the project progresses through the prefeasibility and feasibility planning process.

Required environmental data for this Project will include on-the-ground studies to delineate jurisdictional wetlands. These data will be required to meet a number of needs including permitting, mine design, location of facilities, mine construction and operations. Freegold has initiated consultation with the State’s Large Mine Permitting Team (LMPT) to begin the process of project planning, development and environmental permitting. Through this process the LMPT will assist in developing a broader environmental baseline program.

1.14 RECOMMENDATIONS

Based on the results of the PEA, and the resultant economic evaluation, it is recommended that this study be followed by a preliminary feasibility study in order to further assess the economic viability of the Project. Additional drilling, metallurgical testing, environmental analyses, other permitting and property confirmation activities will need to be undertaken as part of this next level of study. The approximate cost of this study is estimated at $700,000.

Detailed recommendations are provided in **Section 26.0** of this report.

2.0 INTRODUCTION

2.1 BACKGROUND INFORMATION

Freegold Ventures Limited (Freegold) retained Tetra Tech, Inc. (Tetra Tech) to prepare this Technical Report for the Golden Summit Project (the Project) in the Fairbanks Mining District, Alaska. The purpose of this Report is to provide Freegold with an independent opinion of the technical aspects of the Project and make recommendations for future work.

The data from Abrams and Giroux (2012) was reviewed and validated by the authors and subsequent new information generated by Freegold was evaluated and incorporated in this report.

The authors have been provided documents, maps, reports and analytical results by Freegold. Additionally, Freegold personnel — Kristina Walcott, President and CEO and Alvin Jackson, Vice President, Exploration and Development — accompanied the authors to the property May 25 and 26, 2012, and on May 6, 2014 and discussed the geology and explained the past and proposed exploration activities. During these visits the authors reviewed the geology, areas of historical activities, claim corners/locations monument locations, drillholes, open cuts and other pertinent features of the property. The authors also reviewed core in Freegold's Fairbanks core storage facility.

The work completed by Freegold, along with historical data available to the authors, forms the basis of this report. These data include reports from previous operators, including but not limited to, annual, monthly, operations, geological, engineering, metallurgy and production reports as well as new metallurgical testwork.

2.2 TERMS OF REFERENCE

This Report is prepared for Freegold by Tetra Tech, Inc. (Tetra Tech), Mark Abrams (Abrams) and Giroux Consultants Ltd. (Giroux Consultants).

This Technical Report has been prepared in accordance with Section 4.2(1)(j)(ii) of Canadian National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101); and in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions, adopted by the CIM Council on May 10, 2014; and in accordance with the Canadian Securities Administrators Staff Notice 43-307, dated August 16, 2012. CIM defines a "preliminary economic assessment" (PEA) as a study, other than a pre-feasibility or feasibility study, that includes an economic analysis of the potential viability of mineral resources. By definition, the PEA is preliminary in nature, and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves; and as such, there is no certainty that the PEA would be realized. The reason there are no Mineral Reserves is because reserves require a positive prefeasibility study of the indicated resource estimates, and the Project has not reached that stage of advancement.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report.

2.3 Scope of Work

The scope of work conducted by Tetra Tech per the request of Freegold was the development of a PEA Study that evaluates a three-phase open pit and a two-phase process operation. This Report is based on the May 31, 2013 Resource estimate.

2.4 Sources of Information & Data

Principal technical documents and files relating to the Project, used in the preparation of this Report, are listed in Section 27.0.

2.5 Units of Measure

Unless otherwise noted, all costs contained in this report are denominated in United States (U.S.) dollars (US$1.00 = CDN$1.00), which would likely be $1.30 or better.

All units of measurement used in this report are metric unless otherwise stated. Historical grade and tonnage are reported as originally published. Gold grades are reported as referenced and conversion factors are listed below. The Project site is on the Universal Transverse Mercator (UTM) coordinate system, NAD 27 Alaska, Fairbanks Meridian (F.M.).

2.6 Detailed Personal Inspections

1. Mark J. Abrams visited the property on May 25 and 26, 2012.
2. Jackie Blumberg has not visited or inspected the property.
3. Gary Giroux has not visited or inspected the property.
4. Chris Johns has not visited or inspected the property.
5. Ed Lips has visited and inspected the property on May 6, 2014.
6. Nick Michael has not visited or inspected the property.
7. Dave Richers has visited and inspected the property on May 6, 2014.
8. Vicki Scharnhorst has visited and inspected the property on May 6, 2014.
9. Erik Spiller has not visited or inspected the property.
10. Keith Thompson has not visited or inspected the property.

3.0 RELIANCE ON OTHER EXPERTS

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information as required. This Report includes technical information, which 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 Consultants do not consider them to be material.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Golden Summit Property (the Property) is located 18 miles (32 km) by road northeast of the City of Fairbanks, Alaska, United States of America. It is located in the north portion of the Fairbanks Mining District (**Figure 4-1**), a northeast trending belt of lode and placer gold deposits that compose one of the largest gold producing areas in the state of Alaska.

The Property comprises 50 patented claims, 94 unpatented federal claims (managed by the U.S. Department of the Interior, Bureau of Land Management (BLM)), and 268 State of Alaska claims (managed by the State of Alaska Department of Natural Resources (DNR)) and covers a total area of 14,630 acres (5,921 hectares). The Property is situated in Township 3N, Ranges 1E, 2E, and 3E of the Fairbanks Meridian, centered at approximately 479250 E, 7215464 N (UTM Zone 6 NAD 27 Alaska).

4.2 CLAIMS & AGREEMENTS

No annual payments or work are required by law in connection with patented federal mining claims. Annual claim maintenance fees or rents for unpatented federal claims or state claims vary according to the type of claims, claim size, and age, are adjusted every five to ten years, and are due and payable by August 31 of each year (for unpatented federal claims) and November 30 of each year (for state claims). Annual maintenance fees and rents that currently must be paid to maintain the claims in good standing are $14,570 (BLM) and $42,450 (DNR). No minimum amount of work is required by law to be performed on or for the benefit of the unpatented federal claims to maintain them in good standing. To maintain state claims in good standing, however, at least $2.50 per acre per year of work must be performed on or for the benefit of state claims, though work performed in excess of the minimum may be carried forward and used to satisfy future work requirements for up to four years. All unpatented federal claims and state claims included in the Property currently are in good standing with the BLM or DNR (as the case may be), with excess work banked the maximum four years into the future.

Other than the 50 patented mining claims (fee simple lands), claims included in the Project have not been surveyed by a registered land or mineral surveyor and there is no State or federal law or regulation requiring such surveying. Survey plats for the townships in which the Project is situated and for all patented mining claims are open to public inspection at the BLM.

Freegold currently holds a valid Five Year Hardrock Exploration Permit from the State of Alaska (2012-2016) as well as a Department of Army Permit POA-2007-510; which authorizes APMA 9726, a Hard Rock Exploration permit to conduct exploration at the Project. The land on which the Project is situated is zoned as Mineral Land by the Fairbanks North Star Borough, giving mineral development activities first priority use. But as the Project moves forward, additional permits and approvals will need to be acquired from federal, state, and local regulatory agencies. Freegold also expects that it will need or desire to acquire certain additional property rights. For example, depending on how the Project moves forward, Freegold may need or wish (a) to extend or amend one or more of the agreements described in **Sections 4.2.1-4.2.7**, (b) to purchase or lease the undivided 50% interest that it does not currently own or control in two claims (unless Freegold were to acquire this outstanding 50% interest, Freegold will need to account to the co-owner of this claim for its “fair share of the profits” from such claims), (c) to include additional lands in its MHT lease described in Section 4.2.6 below, or (d) to acquire certain surface rights from DNR or other third parties.

Figure 4-2 shows the current land status and extent of the Property. A summary of the claims held by Freegold is shown in **Table 4-1**.

Table 4-1: Summary of Claims Comprising the Golden Summit Property

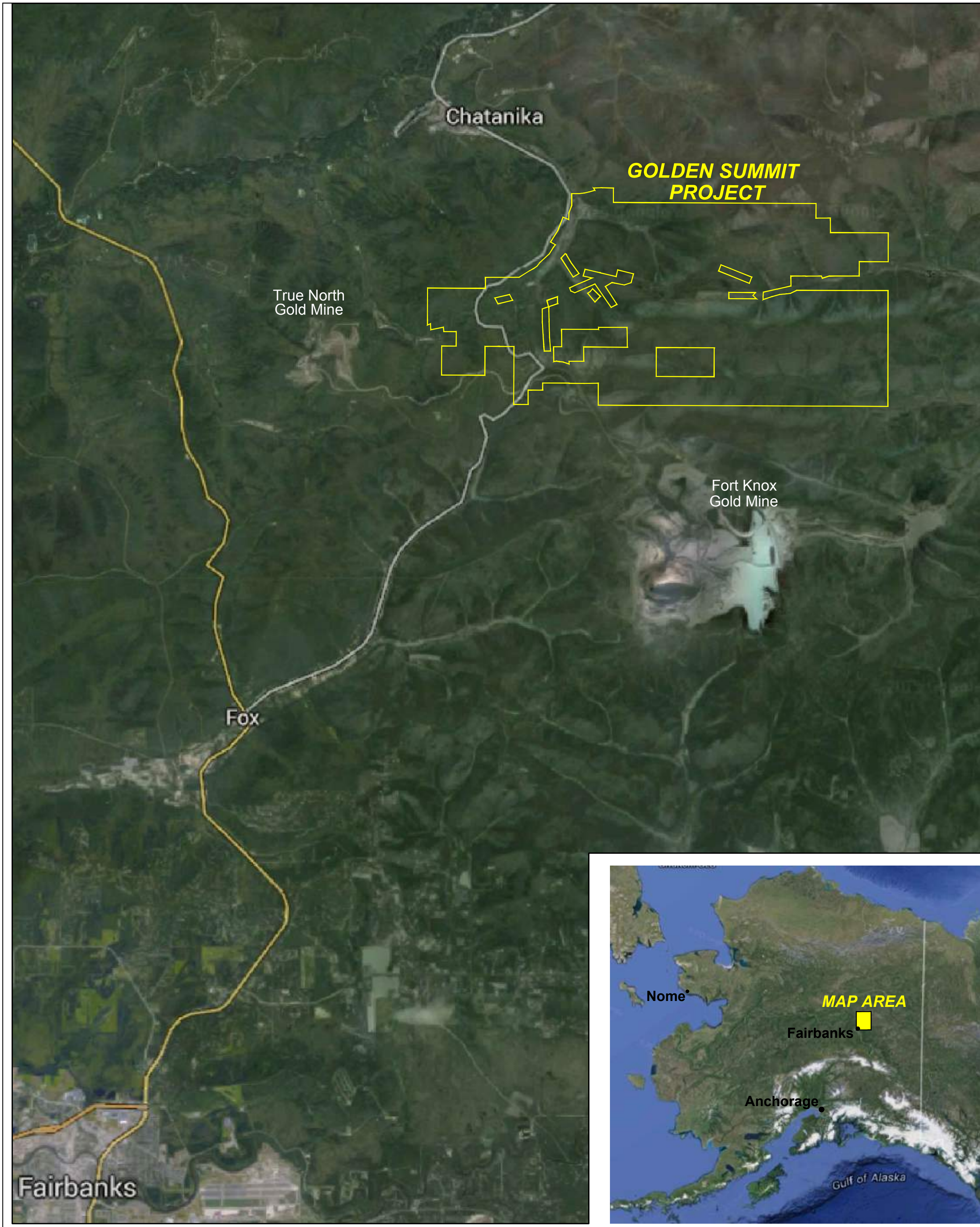
Claim Type	Total Claims	Total Area (sq. mi)	Total Area (acres)	Total Area (hectares)
Federal Patented	50	1.08	693.6	280.6
Federal Unpatented	94	2.93	1,880	760.8
State of Alaska	268	24.44	15,640	6,329.28
Total	**412**	**28.45**	**18,213.62**	**7,370.68**

The agreements under which Freegold holds non-owned claims are summarized below. Total acreage under claim is greater than total area as there are overlapping state and federal claims.

Some of the claims included in the Project are owned outright by Freegold; others are held by Freegold under long-term leases. Claims included in the Project are subject to various NSR royalties ranging from 2% to 5%, and all state claims are subject to a royalty payable to the State of Alaska equal to 3% of net income.

For the claims included in the Project that are subject to long-term leases, Freegold is required to make lease and/or payments as per the following schedules.

A complete list of claims is available at the end of the section.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Freegold Ventures Limited***	Fig4-1_Location.dwg	**FIGURE 4-1** **GOLDEN SUMMIT PROJECT** **GENERAL LOCATION MAP**
	Project: Golden Summit Project Preliminary Economic Assessment	Project Number: 114-910054	
	Project Location: Fairbanks Mining District, Alaska	Date of Issue: March/2016	

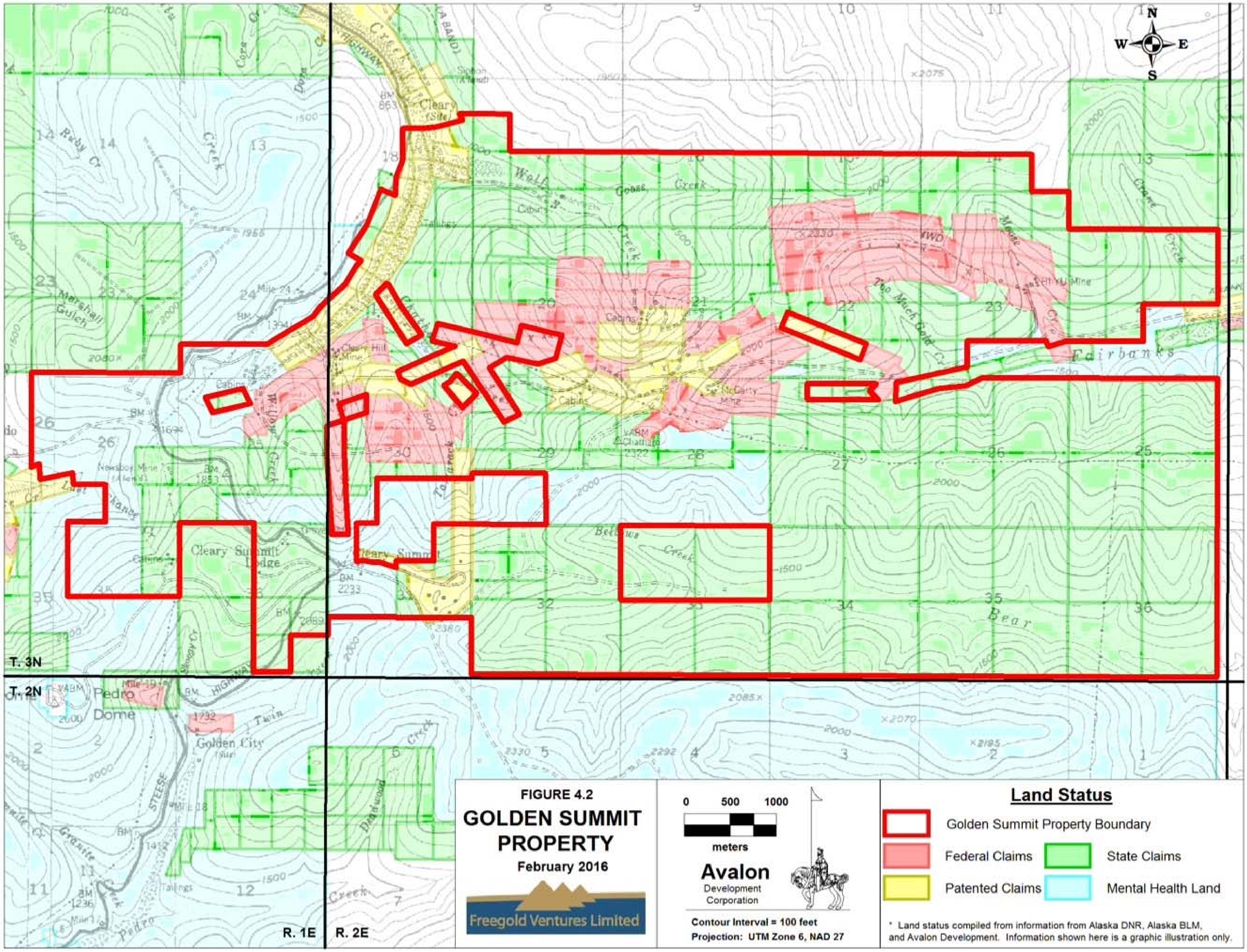


FIGURE 4.2
GOLDEN SUMMIT PROPERTY
February 2016
Freegold Ventures Limited
0 500 1000
meters
Avalon
Development
Corporation
Contour Interval = 100 feet
Projection: UTM Zone 6, NAD 27
Land Status
Golden Summit Property Boundary
Federal Claims
State Claims
Patented Claims
Mental Health Land
* Land status compiled from information from Alaska DNR, Alaska BLM, and Avalon Development. Information shown here is a graphic illustration only.
T. 3N
T. 2N
R. 1E
R. 2E
Cleary Summit
Cleary Summit Lodge
Cleary Hill Mine
McCarty Mine
Pedro Dome
Golden City
Fairbanks
Bear
Too Much Gold Cr
Wolf
Goose Creek
Bellows Creek
Marshall Gulch
Newsboy Mine
Hi Yu Mine
Cleary
Chatham
Twin
Deadwood
Steese Highway

4.2.1 Keystone Claims

On May 17, 1992, Freegold entered into an agreement with Keystone whereby Freegold agreed to make payments of $15,000 per year. In May 2000, the agreement was renegotiated and on October 15, 2000, a $50,000 signing bonus was paid. On November 30, 2001, Freegold restructured the payments.

Table 4-2: Keystone Claims Royalty Payments

Time Period	Amount ($)	Status
1992 – 1998 (US$15,000 per year)	$105,000	(paid)
2000	$50,000	($25,000 paid in cash and $25,000 with 9,816 treasury shares issued)
2001 – 2006 (US$50,000 per year)	$300,000	(paid)
2007	$150,000	(paid)
2008	$150,000	(paid $75,000 in 2008 with the remaining $75,000 paid in 2009, subject to a payment extension)
2009	$150,000	(paid)
2010	$150,000	(paid)
2011	$150,000	(paid)
2012	$150,000	(paid)
2013	$150,000	(paid)
2014	$150,000	(paid)
2015		$75,000 paid to date ($75,000 due August 1st, and November 1st – deferred

This property is subject to a 3% net smelter returns (NSR) royalty. Fifty percent (50%) of the payments shall be credited against future production. In 2011 Freegold negotiated an extension of the Lease for so long as there is either active exploration or production on the Project. In December 2015, Freegold re-negotiated the lease to reduce the annual payments to $75,000 payable in two equal installments on August 1 and November 1, until such time as the price of gold reaches $1,400 for a sustained period. In addition Freegold will undertake to conduct $75,000 in exploration expenditures on the property as consideration for the reduced payments.

4.2.2 Tolovana Claims

In May 2004, Freegold entered into an agreement with a third party (the “Seller”) whereby the Seller transferred to Freegold 100% of the rights under a 20-year lease on this property.

Under the terms of the agreement, Freegold assumed all of the Seller’s obligations under the lease, which include making annual payments of $1,000 per month for the first 23 months increasing to $1,250 per month for the 24th to the 48th months and increasing to $1,500 after the 49th month and for the duration of the lease. These payments are current.

This property is subject to a sliding scale NSR royalty as follows: 1.5% NSR if gold is below $300 per ounce, 2.0% NSR in the event the price of gold is between $300 to $400 per ounce, and 3.0% NSR in the event that the price of gold is above $400 per ounce. Freegold has the right to purchase 100% of the rights to the property including the NSR for US $1 million, less any payments made to date. In addition, Freegold made a cash payment of $7,500 on signing and issued 66,667 shares on regulatory approval. An additional 33,333 shares were to be issued within 30 days, if a minimum 200,000 ounce mineral resource being calculated on the property if the resource was established in five years or less from the date of the agreement. No resource was calculated during the prescribed time frame so these shares were not issued.

4.2.3 Newsboy Claims

By lease agreement dated February 28, 1986 and amended March 26, 1996, Freegold assumed the obligation to make payments of $2,500 per year until 1996 (paid) and $5,000 per year until 2006 (paid). During 2006, the Company renewed the existing lease term for an additional five years on the same terms and conditions. In 2011 Freegold extended the lease for another five years through 2016 and the payments increased to $12,000 per year. These payments are current. In addition Freegold has the opportunity to further extend the lease for another 5 years by making a one-time payment of $50,000. The claims are subject to a 4% NSR royalty. Freegold has the option to purchase the royalty for the greater of the current value or $1,000,000, less all payments made.

4.2.4 Green Claims

By lease agreement dated December 16, 2010, Freegold acquired from Christina Mining Company, LLC (CMC) certain mineral claims known as the Green Property. The property is controlled by Freegold through a long-term lease agreement. The claims are subject to a 3% NSR royalty. Commencing in December 2014 all annual payments shall be credited against future production. Freegold must make annual cash payments and exploration expenditures as shown in **Table 4-3**.

Table 4-3: Green Claims Royalty Payments

Time Period	Payments	Exploration Expenditures
1 December 2010	$100,000 (paid)	-
1 December 2011	$100,000 (paid)	$250,000 (completed)
1 December 2012	$100,000 (paid)	$500,000 (completed)
1 December 2013	$100,000 (paid)	$750,000 (completed)
1 December 2014	$100,000 (paid)	$1,000,000 (completed)
1 December 2015 to 2019	$100,000 per year	-
1 December 2020 to 2029	$200,000 per year	-
Total	**$3,000,000**	**$2,500,000 (completed)**

In December 2015, an amendment was signed to reduce the annual advance royalty for 2015 to US $50,000 and payment was deferred until March 31st, 2016.

4.2.5 *Chatham Claims*

Freegold holds certain mineral claims known as the Chatham Property. The property is controlled by Freegold through a four-year lease agreement. The claims are subject to a 2% NSR royalty. Freegold must make annual cash payments and exploration expenditures as follows.

Table 4-4: Chatham Claims Royalty Payments

Time Period	Payments	Exploration Expenditures
11 July 2011	$20,000 (paid)	-
11 July 2012	$30,000 (paid)	$50,000
11 July 2013	$40,000 (paid)	$50,000
11 July 2014	$50,000 (waived)	$50,000
11 July 2015	$50,000*	$50,000
Total	**$140,000**	**$200,000**

*By mutual agreement the July 2015 payment was deferred.

Freegold has the option to purchase one-half of the NSR representing 1% for $750,000. Freegold also has the option to purchase the property for US$750,000, less the amount already paid.

4.2.6 *Alaska Mental Health Trust Authority Land*

Freegold entered into a long term lease agreement with the Alaska Mental Health Trust Authority, Trust Land Office (MHT) for land and minerals with an effective date of June 1, 2012 (and subsequently amended twice increasing the acreage to 1,576 acres). With respect to the annual rental payments and work commitments to be made for the amendments the date of execution of the amendment shall govern the work and payment requirements. The property is controlled by Freegold through a three-year lease agreement, which may be extended for two extensions of three years each. In 2015 a first extension of the lease was granted. The land is subject to the following sliding scale royalty.

Table 4-5: MHT Sliding Scale Royalty

Price of Gold ($/oz)	Net Royalty
$500 – or below	1.0%
$500.01 - $700	2.0%
$700.01 - $900	3.0%
$900.01 - $1,200	3.5%
Above $1,200	4.5%

Freegold must make annual cash payments and exploration expenditures as follows.

Time Period	Annual Payments	Exploration Expenditures
Execution of agreement	$20,000 (paid)	-
Years 1 -3	$10.00 per acre (paid)	$125.00 per acre per year (completed)
Year 4-6	$15.00 per acre	$235.00 per acre per year
Years 7-9	$20.00 per acre	$355.00 per acre per year

4.2.7 Former Fairbanks Exploration Claims

In 1997, Freegold acquired certain claims from Fairbanks Exploration Inc (FEI), subject to a 7% carried working interest held in trust by Freegold for FEI. After production is achieved, FEI must contribute 7% of any future approved budget. The same claims are also subject to a 2% NSR payable to FEI. Freegold has a 30-day right of first refusal in the event that the 7% carried working interest of FEI or the NSR is to be sold. Freegold can also purchase the NSR at any time following the commencement of commercial production, for a price equal to its then net present value (NPV) as determined in accordance with an agreed upon formula.

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
1	Blueberry	21	T3N	R2E	Fairbanks	308497
2	Robin 1	28	T3N	R2E	Fairbanks	308498
3	Robin 2	29	T3N	R2E	Fairbanks	308499
4	Robin 3	29	T3N	R2E	Fairbanks	308500
5	Robin 4	29	T3N	R2E	Fairbanks	308501
6	Robin 5	29	T3N	R2E	Fairbanks	308502
7	Robin 6	30	T3N	R2E	Fairbanks	308503
8	Ing Fraction	22	T3N	R2E	Fairbanks	315014
9	Gene Fraction	22	T3N	R2E	Fairbanks	315015
10	Beta Fraction	22	T3N	R2E	Fairbanks	315016
11	Alpha Fraction	21,22	T3N	R2E	Fairbanks	315017
12	Arnold Fraction	22	T3N	R2E	Fairbanks	315018

Federal

No.	Claim Name	Section	Township	Range	Meridian	BLM F#
1	Alabama	30	T3N	R2E	Fairbanks	F45603
2	Disc. on Bedrock Cr.	24,25	T3N	R1E	Fairbanks	F45604
3	July #1	30	T3N	R2E	Fairbanks	F45605
4	July #2	30	T3N	R2E	Fairbanks	F45606
5	July #3	30	T3N	R2E	Fairbanks	F45607
6	July Frac. #4	30	T3N	R2E	Fairbanks	F45608
7	Liberty Lode #1	30	T3N	R2E	Fairbanks	F45609
8	Liberty Lode #2	30	T3N	R2E	Fairbanks	F45610
9	Liberty Lode #3	30	T3N	R2E	Fairbanks	F45611
10	Millsite Fraction	30	T3N	R2E	Fairbanks	F45612
11	New York Mineral	24,25	T3N	R1E	Fairbanks	F45613
12	No Name	30	T3N	R2E	Fairbanks	F45614
13	#1 Above Disc. on Bedrock Cr	30	T3N	R2E	Fairbanks	F45615
14	Snow Drift	19	T3N	R2E	Fairbanks	F45616
No.	Claim Name	Section	Township	Range	Meridian	BLM F#
15	Texas	19	T3N	R2E	Fairbanks	F45617
16	Wyoming Quartz	30	T3N	R2E	Fairbanks	F45618
17	Wyoming Frac.	25	T3N	R1E	Fairbanks	F45619
18	Button Weezer	27,28	T3N	R2E	Fairbanks	F45620
19	Caribou Frac.	21,28	T3N	R2E	Fairbanks	F45621
20	Caribou #1	21,22	T3N	R2E	Fairbanks	F45622
21	Caribou #2	21,22	T3N	R2E	Fairbanks	F45623
22	Fern	28	T3N	R2E	Fairbanks	F45624
23	Free Gold	21	T3N	R2E	Fairbanks	F45625
24	Henry Ford #1	28	T3N	R2E	Fairbanks	F45626
25	Henry Ford #2	21	T3N	R2E	Fairbanks	F45627
26	Henry Ford #3	28	T3N	R2E	Fairbanks	F45628
27	Henry Ford #4	28	T3N	R2E	Fairbanks	F45629
28	Laughing Water	21	T3N	R2E	Fairbanks	F45630
29	Little Jim	28	T3N	R2E	Fairbanks	F45631
30	Minnie Ha Ha	21	T3N	R2E	Fairbanks	F45632
31	Pennsylvania	21	T3N	R2E	Fairbanks	F45633
32	Ruth Frac.	21	T3N	R2E	Fairbanks	F45634
33	Speculator	28	T3N	R2E	Fairbanks	F45635
34	Wolf Lode	20,21	T3N	R2E	Fairbanks	F45636
35	Bonus	22	T3N	R2E	Fairbanks	F45637
36	Don	15,22	T3N	R2E	Fairbanks	F45638
37	Durando	22	T3N	R2E	Fairbanks	F45639
38	Edythe	15,22	T3N	R2E	Fairbanks	F45640
39	Flying Joe	22	T3N	R2E	Fairbanks	F45641

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
40	Gold Point	22	T3N	R2E	Fairbanks	F45642
41	Helen S.	23	T3N	R2E	Fairbanks	F45643
42	Hi Yu	23	T3N	R2E	Fairbanks	F45644
43	Hi Yu Millsite	23	T3N	R2E	Fairbanks	F45645
44	Homestake	23	T3N	R2E	Fairbanks	F45646
No.	Claim Name	Section	Township	Range	Meridian	BLM F#
45	Inez	22	T3N	R2E	Fairbanks	F45647
46	Insurgent #1	23	T3N	R2E	Fairbanks	F45648
47	Insurgent #2	23	T3N	R2E	Fairbanks	F45649
48	Julia	15, 22	T3N	R2E	Fairbanks	F45650
49	Jumbo	22	T3N	R2E	Fairbanks	F45651
50	Laura	22	T3N	R2E	Fairbanks	F45652
51	Lillian	23	T3N	R2E	Fairbanks	F45653
52	Long Shin	23	T3N	R2E	Fairbanks	F45654
53	Mame	14,15	T3N	R2E	Fairbanks	F45655
54	Mayflower	22,27	T3N	R2E	Fairbanks	F45656
55	Mohawk	22	T3N	R2E	Fairbanks	F45657
56	#1 Moose Gulch	23	T3N	R2E	Fairbanks	F45658
57	#2 Moose Gulch	23	T3N	R2E	Fairbanks	F45659
58	N.R.A.	15	T3N	R2E	Fairbanks	F45660
59	Nars	22,23	T3N	R2E	Fairbanks	F45661
60	O'Farrel Frac.	23	T3N	R2E	Fairbanks	F45662
61	Ohio	22	T3N	R2E	Fairbanks	F45663
62	Rand	23	T3N	R2E	Fairbanks	F45664
63	Red Top	22	T3N	R2E	Fairbanks	F45665
64	Rob	23	T3N	R2E	Fairbanks	F45666
65	Royalty	15	T3N	R2E	Fairbanks	F45667
66	Santa Clara Frac.	23	T3N	R2E	Fairbanks	F45668
67	Summit	22,23	T3N	R2E	Fairbanks	F45669
68	Sunnyside	22	T3N	R2E	Fairbanks	F45670
69	Teddy R.	23	T3N	R2E	Fairbanks	F45671
70	Yankee Doodle	23	T3N	R2E	Fairbanks	F45672
71	Insurgent #3	14,23	T3N	R2E	Fairbanks	F45673
72	Roy	23	T3N	R2E	Fairbanks	F45674

Patented

No.	Claim Name	Section	Township	Range	Meridian	Pat. #
1	Freegold	19	T3N	R2E	Fairbanks	MS821
2	Colorado	19,30	T3N	R2E	Fairbanks	MS1639
3	California	19,30	T3N	R2E	Fairbanks	MS1639
4	Pauper's Dream	30	T3N	R2E	Fairbanks	MS1639
5	Idaho	30	T3N	R2E	Fairbanks	MS1639
6	Keystone	20,21	T3N	R2E	Fairbanks	MS1607
7	Kawalita	20,21	T3N	R2E	Fairbanks	MS1607
8	Fairbanks	21	T3N	R2E	Fairbanks	MS1607
9	Hope	21	T3N	R2E	Fairbanks	MS1607
10	Willie	21	T3N	R2E	Fairbanks	MS2198
11	Marigold	21,28	T3N	R2E	Fairbanks	MS2198
12	Pioneer	21	T3N	R2E	Fairbanks	MS2198
13	Henry Ford	21,28	T3N	R2E	Fairbanks	MS2198
14	Henry Clay	21	T3N	R2E	Fairbanks	MS2198

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #

FREEGOLD

State

NO.	Claim Name	Section	Township	Range	Meridian	ADL #
1	FRG # 1	31	T3N	R2E	Fairbanks	558129
2	FRG # 2	31	T3N	R2E	Fairbanks	558130
3	FRG # 3	31	T3N	R2E	Fairbanks	558131
4	FRG # 4	31	T3N	R2E	Fairbanks	558132
5	FRG # 5	32	T3N	R2E	Fairbanks	575592
6	FRG # 6	32	T3N	R2E	Fairbanks	575593
7	Erik 1	18	T3N	R2E	Fairbanks	574226
8	Erik 2	18	T3N	R2E	Fairbanks	574227
9	Erik 3	18	T3N	R2E	Fairbanks	574228
10	Kelly 1	27	T3N	R2E	Fairbanks	574122
11	Kelly 2	27	T3N	R2E	Fairbanks	574123
12	Kelly 3	27	T3N	R2E	Fairbanks	574124
NO.	Claim Name	Section	Township	Range	Meridian	ADL #
13	Kelly 4	27	T3N	R2E	Fairbanks	574125
14	Kelly 5	27	T3N	R2E	Fairbanks	574126
15	Kelly 6	27	T3N	R2E	Fairbanks	574127
16	Starbuck 1	16	T3N	R3E	Fairbanks	574128
17	Starbuck 2	16	T3N	R3E	Fairbanks	574129
18	Starbuck 3	16	T3N	R3E	Fairbanks	574130
19	Starbuck 4	16	T3N	R3E	Fairbanks	574131
20	Butterfly 1	33	T3N	R3E	Fairbanks	575583
21	Butterfly 2	33	T3N	R3E	Fairbanks	575584
22	Butterfly 3	33, 34	T3N	R3E	Fairbanks	575585
23	Butterfly 4	3, 4	T2N	R3E	Fairbanks	575586
24	Butterfly 5	3	T2N	R3E	Fairbanks	575587
25	Butterfly 6	34	T3N	R3E	Fairbanks	575588
26	Butterfly 7	34	T3N	R3E	Fairbanks	575589
27	Butterfly 8	33	T3N	R3E	Fairbanks	575590
28	Eldorado #1	27	T3N	R1E	Fairbanks	575591
29	Lauren #9	18	T3N	R2E	Fairbanks	604794
30	3 Above 2 T LL	18, 19	T3N	R2E	Fairbanks	519698
31	4 Above 2 T LL	18, 19	T3N	R2E	Fairbanks	519699
32	FRG 7	26	T3N	R2E	Fairbanks	714368
33	FRG 8	26	T3N	R2E	Fairbanks	714369
34	FRG 9	26	T3N	R2E	Fairbanks	714370
35	FRG 10	26	T3N	R2E	Fairbanks	714371
36	FRG 11	26	T3N	R2E	Fairbanks	714372
37	FRG 12	25	T3N	R2E	Fairbanks	714373
38	FRG 13	25	T3N	R2E	Fairbanks	714374
39	FRG 14	27	T3N	R2E	Fairbanks	714375
40	FRG 15	27	T3N	R2E	Fairbanks	714376
41	FRG 16	26	T3N	R2E	Fairbanks	714377
NO.	Claim Name	Section	Township	Range	Meridian	ADL #
42	FRG 17	26	T3N	R2E	Fairbanks	714378
43	FRG 18	25	T3N	R2E	Fairbanks	714379
44	FRG 19	25	T3N	R2E	Fairbanks	714380
45	FRG 20	32	T3N	R2E	Fairbanks	714381
46	FRG 21	32	T3N	R2E	Fairbanks	714382
47	FRG 22	31	T3N	R2E	Fairbanks	714383
48	FRG 23	32	T3N	R2E	Fairbanks	714384
49	FRG 24	32	T3N	R2E	Fairbanks	714385

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
50	FRG 25	32	T3N	R2E	Fairbanks	714386
51	FRG 26	34	T3N	R2E	Fairbanks	714387
52	FRG 27	34	T3N	R2E	Fairbanks	714388
53	FRG 28	35	T3N	R2E	Fairbanks	714389
54	FRG 29	35	T3N	R2E	Fairbanks	714390
55	FRG 30	36	T3N	R2E	Fairbanks	714391
56	FRG 31	36	T3N	R2E	Fairbanks	714392
57	FRG 32	31	T3N	R2E	Fairbanks	714393
58	FRG 33	32	T3N	R2E	Fairbanks	714394
59	FRG 34	32	T3N	R2E	Fairbanks	714395
60	FRG 35	33	T3N	R2E	Fairbanks	714396
61	FRG 36	33	T3N	R2E	Fairbanks	714397
62	FRG 37	34	T3N	R2E	Fairbanks	714398
63	FRG 38	34	T3N	R2E	Fairbanks	714399
64	FRG 39	35	T3N	R2E	Fairbanks	714400
65	FRG 40	35	T3N	R2E	Fairbanks	714401
66	FRG 41	36	T3N	R2E	Fairbanks	714402
67	FRG 42	36	T3N	R2E	Fairbanks	714403
68	FRG 43	36	T3N	R1E	Fairbanks	714966
69	FRG 44	36	T3N	R1E	Fairbanks	717880
70	FRG 45	36	T3N	R1E	Fairbanks	717881
71	FRG 46	36	T3N	R1E	Fairbanks	717882
72	FRG 47	24	T3N	R1E	Fairbanks	619290
	FRG 47	19	T3N	R2E	Fairbanks	
73	FRG 48	24,25	T3N	R1E	Fairbanks	619291
	FRG 48	19,30	T3N	R2E	Fairbanks	
NO.	Claim Name	Section	Township	Range	Meridian	ADL #
72	STARBUCKS 5	16	T3N	R3E	Fairbanks	717870
73	STARBUCKS 6	16	T3N	R3E	Fairbanks	717871
74	STARBUCKS 7	15	T3N	R3E	Fairbanks	717872
75	STARBUCKS 8	9	T3N	R3E	Fairbanks	717873
76	STARBUCKS 9	9	T3N	R3E	Fairbanks	717874
77	STARBUCKS 10	10	T3N	R3E	Fairbanks	717875
78	STARBUCKS 11	10	T3N	R3E	Fairbanks	717876
79	STARBUCKS 12	10	T3N	R3E	Fairbanks	717877
80	STARBUCKS 13	10	T3N	R3E	Fairbanks	717878
81	STARBUCKS 14	10	T3N	R3E	Fairbanks	717879
82	FRG 47	19,24,25	T3N	1E,2E	Fairbanks	619290
83	FRG 48	19,24,25,30	T3N	1E,2E	Fairbanks	619291

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #

Patented **Freegold**

No.	Claim Name	Patent #
1	No. 9 Number Nine Above Discovery On Cleary Creek	1687
2	Bench Claim No. 9 Above Discovery, Left Limit Cleary Creek	1671
3	No. 8 Above Discovery On Cleary Creek	1670
4	No. 7 Above Discovery On Cleary Creek	1670
5	No. 6 Above Discovery Cleary Creek	1670
6	Side Claim No. 8, Above Left Limit On Cleary Creek, Placer	807
7	Side Claim No. 8, Above Left Limit, Cleary Creek, Placer	524
8	Side Claim No. 8, Above Left Limit, Cleary Creek	1968
9	No. 7 Above Discovery, 1st Tier, Left Limit Placer	1968
10	Placer Mining Claim No. 6, 1st T.LL. Above Discovery on Cleary Creek Placer	1972
11	Bench No. 5, Above Discovery On Left Limit Cleary Creek	367
12	No. 5 Above Discovery On Cleary Creek	365
13	No. 4 Above Discovery On Cleary Creek	365
No.	Claim Name	Patent #
14	No. 5 Above Discovery L.L. First Tier, Placer	836
15	The Lower Divided One Half of the Upper One Half of Number 4 Above Left Limit Bench Placer	1793
16	The Lower Half of Number 4 Above Discovery Creek Claim Placer	1793
17	Claim No. Three (3) Above Discovery On Cleary Creek Placer	1793
18	Fraction No. Three Above Discovery First Tier Left Limit Placer	1793
19	No. 3 Above Discovery, First Tier, Left Limit on Cleary Creek, Placer	1919
20	Discovery Placer	805
21	No. 1 Above Discovery	805
22	No. 2 Above Discovery	805
23	No. 2 Side Claim, Left Limit, Cleary Creek, Placer	1798
24	No. Two Above Fraction Placer	1798
25	No. 1 One Above Discovery on the Left Limit of Cleary Creek, Placer	1605
26	Discovery Bench Left Limit Cleary Creek, Placer	1926
No.	Claim Name	Patent #
27	Side Claim on Right Limit of Discovery Cleary Creek, Placer	1794
28	Discovery Claim on Wolf Creek Placer	1901
29	Bench Claim Right Limit Opposite Discovery on Wolf Placer	1920

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
TOLOVANA						
State						
NO.	Claim Name	Section	Township	Range	Meridian	ADL #
1	VDH-AMS #1	25	T3N	R1E	Fairbanks	344681
2	VDH-AMS #2	25	T3N	R1E	Fairbanks	344682
3	VDH-AMS #3	25	T3N	R1E	Fairbanks	344683
Federal						
No.	Claim Name	Section	Township	Range	Meridian	BLM F#
1	Willow Creek #1	25, 26	T3N	R1E	Fairbanks	24963
2	Willow Creek #2	25	T3N	R1E	Fairbanks	24964
3	Willow Creek #3	25	T3N	R1E	Fairbanks	24965
4	Willow Ck. #1 Placer	25	T3N	R1E	Fairbanks	24966

FAIRBANKS EXPLORATION

State

NO.	Claim Name	Section	Township	Range	Meridian	ADL #
1	What's Next #1	24	T3N	R2E	Fairbanks	501821
2	What's Next #2	24	T3N	R2E	Fairbanks	501822
3	What's Next #3	24	T3N	R2E	Fairbanks	501823
4	What's Next #4	24	T3N	R2E	Fairbanks	501824
5	What's Next #5	22	T3N	R2E	Fairbanks	502196
6	What's Next #6	22	T3N	R2E	Fairbanks	502197
7	What's Next #7	22	T3N	R2E	Fairbanks	502198
NO.	Claim Name	Section	Township	Range	Meridian	ADL #
8	What's Next #8	22	T3N	R2E	Fairbanks	502199
9	Crane #1	24	T3N	R2E	Fairbanks	502551
10	Crane #2	24	T3N	R2E	Fairbanks	502552
11	Crane #3	24	T3N	R2E	Fairbanks	502553
12	Crane #4	24	T3N	R2E	Fairbanks	501930
13	Anticline #1	24	T3N	R2E	Fairbanks	501825
14	Anticline #2	24	T3N	R2E	Fairbanks	501836
15	Ruby 3A Fraction	25	T3N	R1E	Fairbanks	515911
16	Ruby 4A Fraction	25	T3N	R1E	Fairbanks	515912
17	Ruby 5 Fraction	25	T3N	R1E	Fairbanks	515913
18	Ruby 6 Fraction	25	T3N	R1E	Fairbanks	515914
19	Ruby 7 Fraction	25	T3N	R1E	Fairbanks	515915
20	Ruby 8 Fraction	30	T3N	R2E	Fairbanks	515916
21	Ruby 9 Fraction	30	T3N	R2E	Fairbanks	515917
22	Ruby 10 Fraction	30	T3N	R2E	Fairbanks	515918
23	Ruby 11 Fraction	30	T3N	R2E	Fairbanks	515919
24	Ruby 12 Fraction	29	T3N	R2E	Fairbanks	515920
25	Ruby 13 Fraction	29	T3N	R2E	Fairbanks	515921
26	Ruby 14 Fraction	29	T3N	R2E	Fairbanks	515922
27	Ruby 15 Fraction	29	T3N	R2E	Fairbanks	515923
28	Ruby 16 Fraction	28	T3N	R2E	Fairbanks	515924
29	Ruby 17 Fraction	28	T3N	R2E	Fairbanks	515925
30	Ruby 18 Fraction	28	T3N	R2E	Fairbanks	515926
31	Ruby 19 Fraction	28	T3N	R2E	Fairbanks	515927

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
EHB LLC						
State						
No.	Claim Name	Section	Township	Range	Meridian	ADL #
1	Greenback 1	35	T3N	R1E	Fairbanks	359771
2	Greenback 2	35	T3N	R1E	Fairbanks	359772
3	Greenback 3	26	T3N	R1E	Fairbanks	361184
4	Greenback 4	25	T3N	R1E	Fairbanks	505192
5	Newsboy	26	T3N	R1E	Fairbanks	333135
6	Newsboy Extension	25	T3N	R1E	Fairbanks	333136
CHATHAM (BURGGRAF)						
Patented						
No.	Claim Name	Section	Township	Range	Meridian	Pat #
1	Chatham #2 Lode	20, 29	T3N	R2E	Fairbanks	1713
2	Fey Lode	20, 29	T3N	R2E	Fairbanks	1713
3	Colby #2 Lode	29	T3N	R2E	Fairbanks	1713
4	Colby Lode	28, 29	T3N	R2E	Fairbanks	1713
5	Fay Claim #2 Lode	20, 28, 29	T3N	R2E	Fairbanks	1713
6	I.B. Claim	28	T3N	R2E	Fairbanks	1676
7	Margery Daw Claim	28, 29	T3N	R2E	Fairbanks	1676
CHRISTINA MINING LLC						
Federal						
No.	Claim Name	Section	Township	Range	Meridian	BLM F#
1	Christina	20,	T3N	R2E	Fairbanks	F58503
2	Fraction #1	20, 21	T3N	R2E	Fairbanks	F58504
3	Fraction #2	20, 21	T3N	R2E	Fairbanks	F58505
4	Fraction #3	20	T3N	R2E	Fairbanks	F58506
5	Carrie A	20	T3N	R2E	Fairbanks	F58507
6	Carrie A #1	20	T3N	R2E	Fairbanks	F58508
7	Carrie A #2	20	T3N	R2E	Fairbanks	F58509
8	Grace E	20	T3N	R2E	Fairbanks	F58510
9	Grace E #1	20	T3N	R2E	Fairbanks	F58511
No.	Claim Name	Section	Township	Range	Meridian	BLM F#
10	Grace E #2	20	T3N	R2E	Fairbanks	F58512
11	Grace Eva #1	20	T3N	R2E	Fairbanks	F58513
12	Grace Eva #2	20	T3N	R2E	Fairbanks	F58514
13	Grace Eva #3	30	T3N	R2E	Fairbanks	F58515
14	Wolf Lode #1	20, 21	T3N	R2E	Fairbanks	F58516
15	Wolf Lode #2	20, 21	T3N	R2E	Fairbanks	F58517
16	Fairbanks #1	21	T3N	R2E	Fairbanks	F58518
17	Fairbanks #2	21	T3N	R2E	Fairbanks	F58519
18	Fairbanks #3	21	T3N	R2E	Fairbanks	F58520
State						
No.	Claim Name	Section	Township	Range	Meridian	ADL #
1	RAM 1	17	T3N	R2E	Fairbanks	303366
2	RAM 2	17	T3N	R2E	Fairbanks	303367
3	RAM 3	17	T3N	R2E	Fairbanks	303368
4	RAM 4	17	T3N	R2E	Fairbanks	303369
5	RAM 5	16	T3N	R2E	Fairbanks	303370
6	RAM 6	16	T3N	R2E	Fairbanks	303371
7	RAM 7	16	T3N	R2E	Fairbanks	303372
8	RAM 8	16	T3N	R2E	Fairbanks	303373
9	RAM 9	15	T3N	R2E	Fairbanks	303374
10	RAM 10	15	T3N	R2E	Fairbanks	303375
11	RAM 11	15	T3N	R2E	Fairbanks	303376

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
12	RAM 12	15	T3N	R2E	Fairbanks	303377
13	RAM 13	17	T3N	R2E	Fairbanks	303378
14	RAM 14	17	T3N	R2E	Fairbanks	303379
15	RAM 15	17	T3N	R2E	Fairbanks	303380
16	RAM 16	17	T3N	R2E	Fairbanks	303381
17	RAM 17	16	T3N	R2E	Fairbanks	303382
18	RAM 18	16	T3N	R2E	Fairbanks	303383
19	RAM 19	16	T3N	R2E	Fairbanks	303384
20	RAM 20	16	T3N	R2E	Fairbanks	303385
21	RAM 21	15	T3N	R2E	Fairbanks	303386
22	RAM 22	15	T3N	R2E	Fairbanks	303387
23	RAM 23	15	T3N	R2E	Fairbanks	303388
24	RAM 24	15	T3N	R2E	Fairbanks	303389
25	RAM 25	17	T3N	R2E	Fairbanks	303390
26	RAM 57	14	T3N	R2E	Fairbanks	303422
27	RAM 59	14	T3N	R2E	Fairbanks	303423
28	RAM 60	14	T3N	R2E	Fairbanks	303424
29	RAM 62	14	T3N	R2E	Fairbanks	303426
30	RAM 63	14	T3N	R2E	Fairbanks	303427
31	RAM 64	14	T3N	R2E	Fairbanks	303428
32	RAM 65	14	T3N	R2E	Fairbanks	303429
33	RAM 66	20	T3N	R2E	Fairbanks	306460
34	RAM 67	20	T3N	R2E	Fairbanks	306461
35	RAM 68	20	T3N	R2E	Fairbanks	306462
36	RAM 69	20	T3N	R2E	Fairbanks	306463
37	RAM 70	21	T3N	R2E	Fairbanks	306464
38	RAM 71	21	T3N	R2E	Fairbanks	306465
39	RAM 72	20	T3N	R2E	Fairbanks	306466
40	RAM 73	20	T3N	R2E	Fairbanks	306467
41	RAM 74	20	T3N	R2E	Fairbanks	306468
42	RAM 75	20	T3N	R2E	Fairbanks	306469
43	RAM 76	21	T3N	R2E	Fairbanks	306470
44	RAM 2A	20	T3N	R2E	Fairbanks	302892
45	RAM 3A	20	T3N	R2E	Fairbanks	302893
46	RAM 58	19	T3N	R2E	Fairbanks	302894
47	RAM 58A	19	T3N	R2E	Fairbanks	302895
48	RAM 58B	19	T3N	R2E	Fairbanks	302896
49	RAM 58C	19	T3N	R2E	Fairbanks	302897
50	RAM 58D	19	T3N	R2E	Fairbanks	302898
51	RAM 58E	19	T3N	R2E	Fairbanks	302899
52	RAM 58F	20	T3N	R2E	Fairbanks	302900
53	RAM 58G	20	T3N	R2E	Fairbanks	302901
54	RAM 58H	20	T3N	R2E	Fairbanks	302902
55	RAM 58I	18	T3N	R2E	Fairbanks	302903
56	RAM 58J	20	T3N	R2E	Fairbanks	302904
57	RAM 58K	20	T3N	R2E	Fairbanks	302905
58	RAM 58L	20	T3N	R2E	Fairbanks	302906
59	VD 1	20	T3N	R2E	Fairbanks	302907
60	VD2	20	T3N	R2E	Fairbanks	302908
61	GOOSE 1	20	T3N	R2E	Fairbanks	342763
62	GOOSE 2	20	T3N	R2E	Fairbanks	342764
63	GOOSE 3	20	T3N	R2E	Fairbanks	342765
64	GOOSE 4	20	T3N	R2E	Fairbanks	342766
65	GOOSE 5	21	T3N	R2E	Fairbanks	342767

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
66	GOOSE 6	21	T3N	R2E	Fairbanks	342768
67	MOOSE FRACTION 1	23	T3N	R2E	Fairbanks	344966
68	MOOSE FRACTION 2	23	T3N	R2E	Fairbanks	344967
69	MOOSE FRACTION 3	23	T3N	R2E	Fairbanks	344968
70	MOOSE FRACTION 4	23	T3N	R2E	Fairbanks	344969
71	OAKIE FRACTION 1	30	T3N	R2E	Fairbanks	342791
72	OAKIE FRACTION 2	30	T3N	R2E	Fairbanks	342792
73	OAKIE FRACTION 3	30	T3N	R2E	Fairbanks	342793
74	OAKIE FRACTION 4	25	T3N	R1E	Fairbanks	342794
75	OAKIE FRACTION 5	19	T3N	R2E	Fairbanks	348966
76	OAKIE FRACTION 6	19	T3N	R2E	Fairbanks	348967
77	OAKIE FRACTION 7	19	T3N	R2E	Fairbanks	348968
78	OAKIE FRACTION 8	19	T3N	R2E	Fairbanks	348969
79	OAKIE FRACTION 9	19	T3N	R2E	Fairbanks	348970
80	OLD GOLD 1	21	T3N	R2E	Fairbanks	322801
81	OLD GOLD FRACTION 2	21	T3N	R2E	Fairbanks	322802
82	OLD GOLD FRACTION 3	21	T3N	R2E	Fairbanks	322803
83	OLD GOLD 4	21	T3N	R2E	Fairbanks	322804
84	OLD GOLD FRACTION 5	21	T3N	R2E	Fairbanks	322805
85	OLD GOLD FRACTION 6	21	T3N	R2E	Fairbanks	322806
86	OLD GOLD FRACTION 7	21	T3N	R2E	Fairbanks	322807
87	OLD GOLD FRACTION 8	21	T3N	R2E	Fairbanks	322808
88	OLD GOLD FRACTION 9	23	T3N	R2E	Fairbanks	322809
89	OLD GOLD FRACTION 11A	22	T3N	R2E	Fairbanks	336671
90	OLD GOLD FRACTION 13	22	T3N	R2E	Fairbanks	336672
91	OLD GOLD FRACTION 14	22	T3N	R2E	Fairbanks	336673
92	OLD GOLD FRACTION 15	23	T3N	R2E	Fairbanks	336674
93	OLD GOLD FRACTION 16	22	T3N	R2E	Fairbanks	336675
94	OLD GOLD FRACTION 17	22	T3N	R2E	Fairbanks	336676
95	OLD GOLD FRACTION 18	22	T3N	R2E	Fairbanks	336677
96	OLD GOLD 19	23	T3N	R2E	Fairbanks	336666
97	OLD GOLD FRACTION 20	23	T3N	R1E	Fairbanks	336678
98	OLD GOLD FRACTION 21	23	T3N	R1E	Fairbanks	336679
99	OLD GOLD FRACTION 22	23	T3N	R1E	Fairbanks	336680
100	OLD GOLD FRACTION 23	22	T3N	R1E	Fairbanks	336681
101	OLD GOLD FRACTION 24	22	T3N	R1E	Fairbanks	336682
102	OLD GOLD FRACTION 25	22	T3N	R1E	Fairbanks	336683
103	OLD GOLD FRACTION 26	23	T3N	R1E	Fairbanks	336667
104	OLD GOLD FRACTION 34	22	T3N	R1E	Fairbanks	336684
105	OLD GOLD FRACTION 35	22	T3N	R1E	Fairbanks	336685
106	OLD GOLD FRACTION 36	28	T3N	R1E	Fairbanks	336686
107	OLD GOLD FRACTION 37	27	T3N	R1E	Fairbanks	336687
108	OLD GOLD FRACTION 38	27	T3N	R1E	Fairbanks	336688
109	OLD GOLD FRACTION 39	27	T3N	R1E	Fairbanks	336689
110	OLD GOLD FRACTION 40	27	T3N	R1E	Fairbanks	336690
111	OLD GOLD FRACTION 41	27	T3N	R1E	Fairbanks	336691
112	OLD GOLD FRACTION 42	28	T3N	R1E	Fairbanks	336692
113	OLD GOLD FRACTION 43	27	T3N	R1E	Fairbanks	336668
114	OLD GOLD FRACTION 44	27	T3N	R1E	Fairbanks	336669
115	OLD GOLD FRACTION 45	27	T3N	R1E	Fairbanks	336670
116	RUBY 1	25	T3N	R1E	Fairbanks	354215
117	RUBY 2 FRACTION	25	T3N	R1E	Fairbanks	354216
118	RUBY 3 FRACTION	25	T3N	R1E	Fairbanks	354217
119	RUBY 4 FRACTION	25	T3N	R1E	Fairbanks	354218

Table 4-6: Claim List

NO.	CLAIM NAME	SECTION	Township	Range	Meridian	ADL #
120	WW FRACTION 1	20	T3N	R2E	Fairbanks	342778
121	WW FRACTION 2	20	T3N	R2E	Fairbanks	342779
122	WW FRACTION 3	20	T3N	R2E	Fairbanks	342780
123	WW FRACTION 4	20	T3N	R2E	Fairbanks	342781
124	WW FRACTION 5	20	T3N	R2E	Fairbanks	342782
125	WW FRACTION 6	20	T3N	R2E	Fairbanks	342783
126	WW 7	29	T3N	R2E	Fairbanks	342784
127	WW FRACTION 8	29	T3N	R2E	Fairbanks	342785
128	WW FRACTION 9	29	T3N	R2E	Fairbanks	342786
129	WW FRACTION 10	29	T3N	R2E	Fairbanks	342787
130	WW FRACTION 11	19	T3N	R2E	Fairbanks	342788
131	WW FRACTION 12	30	T3N	R2E	Fairbanks	342789
132	WW FRACTION 13	30	T3N	R2E	Fairbanks	342790
133	WW FRACTION 14	30	T3N	R2E	Fairbanks	506514
Mental Health Trust						
No.	Claim Name	Section	Township	Range	Meridian	ADL #
TOTAL	**1,576 Acres**					
	NW1/4(Excluding portion of MS2376, MS2448 and ADL344682)	25	T3N	R1E		
	E1/2NE1/4	26	T3N	R1E		
	87.5 Acres					
	(S1/2S1/2)	24	T3N	R1E		
	(NW1/4NE1/4)	25	T3N	R1E		
	92.12 Acres	25	T3N	R1E		
	S1/2S1/2					
	11.3 Acres	19	T3N	R2E		
	S1/2S1/2					
	1,173 Acres - contained within					
	5 irregularly shaped parcels	26	T3N	R1E		
		35	T3N	R1E		
	portions of	28-31	T3N	R2E		

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Golden Summit Property (Property) is situated close to the city of Fairbanks, the second largest city in Alaska (population of the greater Fairbanks area is approximately 100,000). Fairbanks serves as a major population and supply center for the interior region of Alaska.

Access to the Property from Fairbanks is by 29 km of paved highway (Steese Highway). The Steese Highway transects the Property and is connected to state and privately-maintained gravel roads which allows easy access to most areas of the property on a year-round basis. A high voltage electrical power line, land telephone lines, and a cellular phone net service the property.

5.2 CLIMATE

Sub-freezing temperatures are the norm in this region of Alaska during the six to eight months of winter. Following winter, four to six months of warm summer weather prevails. Precipitation in this part of Alaska averages 13 inches, occurring mostly as snowfall between October and March. Permafrost is discontinuous throughout the area. Drilling is possible on a year-round basis on the Property.

5.3 LOCAL RESOURCES

Fairbanks serves as the seat for the Fairbanks Northstar Borough, a region which supports a population of approximately 100,000 and has excellent labor and services infrastructure, including rail and international airport access. The Fairbanks International Airport is served by several major airlines with numerous scheduled daily flights. Fairbanks is also served by the Alaska Railroad, and is connected to Anchorage and Whitehorse, Canada by well-maintained paved highways.

The main campus of the University of Alaska is located in Fairbanks in addition to state and federal Offices. Major employers within the Fairbanks Area include Fort Knox (Kinross), Fort Wainwright (U.S. Army), the University of Alaska, as well as numerous state and federal Agencies. Exploration and development costs in the Fairbanks area are similar to those common in the western United States.

5.4 PHYSIOGRAPHY

The terrain in the Project areas is composed of low, rounded hills cut by steep sided valleys and a number of streams. Elevations on the property range from 1,000 feet (305 meters) to over 2,200 feet (670 meters). Outcrops are rare except in man-made exposures. Vegetation consists of a tundra mat that supports subarctic vegetation (alder, willow, black spruce, aspen and birch). A variably thick layer of aeolian silt covers most of the Property. Permafrost is limited to small discontinuous lenses on steep, poorly drained north-facing slopes, and does not pose an obstacle to mining activities.

6.0 HISTORY

Placer or lode gold mining has occurred almost continuously in the Project area since gold was discovered in the district in 1902. Over 9.5 million ounces of placer gold have been recovered from the Fairbanks Mining District, of which 6.75 million ounces have been recovered from streams that drain the Project (Freeman, 1992e). In addition, over 506,000 ounces of lode gold were recovered from past producing mines on the Project (Freeman and others, 1996). More than 80 lode gold occurrences have been documented in the Project area. Recent exploration discoveries in the Tintina Gold Belt have underscored the potential for bulk tonnage and high-grade deposits, both of which are known to exist in the Project area (McCoy and others, 1997; Flanigan and others, 2000).

Freegold acquired an interest in the Project in mid-1991 and since then has conducted extensive geologic mapping, soil sampling, trenching, rock sampling, geophysical surveys, core, reverse circulation, and rotary air blast drilling on the project (Freeman, 1991; Galey and others, 1993; Freeman and others, 1996; Freeman and others, 1998; Freeman, 2004; Freeman, 2005; Freeman, 2006 and Freeman, 2007, Adams and Giroux, 2012). Drilling completed by Freegold on the Project between 1991 and 2009 totaled 88,241 feet of core and reverse circulation in 214 holes and 80,822 feet of rotary air blast drilling in 2,028 holes before commencing a comprehensive property compilation in 2010.

In the summer of 2010, a ground-based geophysical survey was undertaken on the Dolphin area in addition to the extensive compilation work on the Project. The results of the geophysical survey indicated that the alteration in the Dolphin Area is well defined with a low resistivity feature. Total exploration expenditures at Golden Summit in 2010 amounted to $293,378. In addition to the exploration and compilation work, Freegold also entered into a long term lease on 133 State of Alaska mining claims and 18 unpatented Federal mining claims in order to better strengthen its land position within the Project area. In March of 2011, Freegold completed its first NI 43-101 compliant Mineral Resource calculation using previous drilling completed in the Dolphin area. The Mineral Resource was completed by Giroux Consultants of Vancouver, British Columbia and, using a 0.3 g/t cut-off grade, included Indicated Resources totaling 7,790,000 tonnes grading 0.695 g/t (174,000 ounces) and Inferred Resources totaling 27,010,000 tonnes grading 0.606 g/t (526,000 ounces). Drilling aimed at increasing this Mineral Resource began in February 2011. During 2011 a total of 29 holes (20,766.5 feet/6,329.5 meters) were completed in the Dolphin area. The results of the Dolphin drilling were incorporated into the updated NI 43-101 which was released in December 2011 and using a 0.3 g/t cut-off resulted in an increase in the Indicated category to 17,270,000 tonnes at 0.62 g/t (341,000 contained ounces) and 64,440,000 tonnes at 0.55 g/t (1,135,000 contained ounces) in the Inferred category. 2011 also saw the further expansion of the Property with the addition of seven patented mining claims of the Chatham mine block. Ground based induced polarization (IP) geophysics and shovel soil sampling was also carried out during the summer and fall of 2011.

A total of 18 holes (11,515 feet/3,509.9 meters) were also drilled in the Cleary Hill area during 2011. This initial drilling was aimed at infilling historical drilling in the Cleary Hill mine area with the aim of linking the Dolphin/Cleary Hill areas in a future resource model. Total exploration expenditures in 2011 on the Project were $3,927,969.

In late 2011, Freegold also undertook its first drilling in the Christina prospect area, a high grade vein and bulk tonnage style target which lies three km to the east of the Dolphin – Cleary Hill area. A total of 12 holes were drilled (15,058 feet) (4,580 meters) in the Christina prospect during late 2011 and early 2012.

A total of 55 holes (54,470.5 feet/16,602.6 meters) were completed at the Project in 2012. In January 2012 drilling resumed with one drill rig at the Christina area and a second rig at the Dolphin/Cleary Hill area. From mid-May on, a single drill rig remained active on the Dolphin/Cleary Hill area through late September. In addition ground based geophysics and shovel soil sampling were also undertaken on the project. A mineral lease with the MHT was finalized in 2012 which expanded the project area by 212 acres to the west. The company also staked an additional 37 State of Alaska claims covering 4,720 acres along it southern boundary.

In October 2012, an updated NI 43-101 resource was again calculated this time expanding the Dolphin Resource to encompass the eastern portion of the Cleary Hill area as well (reference **Section 14**). Exploration expenditures to September 30, 2012 were $4,763,783.

Freegold drilled thirteen holes (16,860 feet/5,138 meters) in 2013. In addition, an updated NI 43-101 compliant gold resource was calculated for the Dolphin/Cleary area based on the ten holes completed during the winter drill program, of which eight were incorporated into the Resource. The additional three holes were drilled after the updated resource was completed, and as such, were not included in the Resource. An additional three State of Alaska claims which covered 120 acres were staked as well as an additional 191 acres were added to the MHT Lease.

No additional drilling was undertaken in 2014 and 2015. Activities were concentrated on metallurgical testing, cultural resource work, water quality sampling and geochemical surveys.

Table 6-1 provides a summary of exploration activities conducted for the property and adjacent prospects.

Table 6-1: Summary of Exploration (1969-2015) Conducted for the Property and Adjacent Prospects

Company	Years	Exploration/Mining Activity	Principle Targets
International Minerals & Chemicals	1969	Trenching RC drilling	Saddle Zone Circle Trail Zone
Placid Oil Company	1978 – 1986	Trenching Core & RC drilling Adit excavation Christina feasibility study	Christina Vein Pioneer Vein American Eagle Vein Hi Yu Vein
SC	1980 – 1981	Diamond core drilling RC drilling Resource estimate	Tolovana Shear Zone
Fairbanks Exploration	1988	Bulk sampling	Christina Vein
Keystone Mines Partnership	1989	Bulk sampling of mine waste dumps	American Eagle, Hi Yu, Cleary Hill Mines
British Petroleum/Fairbanks Exploration(FEI) JV	1987 – 1988	Trenching, RC drilling	Too Much Gold prospect Saddle Zone Circle Trail Zone Christina Vein
Freegold/FEI JV	1991	Property-wide data compilation	Property-wide
Freegold/Amax Gold JV	1992 – 1994	Trenching, soil sampling, RC drilling, aerial geophysical surveys (EM), bottle roll testing, baseline water quality surveys, aerial photos, EDM surveys	Too Much Gold prospect Cleary Hill Mine area
Freegold	1995 – 1996	RC drilling	Dolphin Deposit Cleary Hill Mine area

Company	Years	Exploration/Mining Activity	Principle Targets
Freegold/Barrick JV	1997 – 1998	Property-wide grid-base soils, recon & prospect mapping, grab sampling, limited RC and core drilling	Property-wide Goose Creek prospect North Extension prospect Coffee Dome Dolphin Deposit Newsboy Mine area Wolf Creek area
Freegold	2000	Limited core drilling	Cleary Hill Mine area
Freegold	2002	Trenching	Cleary Hill Mine area (Currey Zone)
Freegold	2003	Limited core drilling	Cleary Hill Mine area (Currey Zone)
Freegold/Meridian Minerals JV	2004	Trenching, core drilling	Tolovana Mine area Cleary Hill Mine area
Freegold	2005 – 2006	Trenching	Cleary Hill Mine area Wackwitz Vein area Beistline Shaft area
Freegold	2007 – 2008	Trenching, RAB drilling, core drilling, bulk sampling	Cleary Hill Mine area Tolovana Mine area
Freegold	2010	Induced Polarization Survey	Dolphin/Tolovana Area
Freegold	2011	Induced Polarization Survey, Geochemical Surveys, Core Drilling,	Dolphin Deposit Cleary Hill, Christina Prospect
Freegold	2012	Induced Polarization Survey, Geochemical Surveys, Trenching, Core Drilling	Dolphin/Tolovana Area, Cleary Hill, Christina Prospect
Freegold	2013	Core Drilling, Geophysics,	Dolphin, Coffee Dome Area
Freegold	2014	Water Quality Sampling, Cultural Resource Studies, Metallurgical tests, Geochemical Surveys	Dolphin/Tolovana Area, Cleary Hill,
Freegold	2015	Water Quality Sampling, Cultural Resource Studies,, and Geochemical Surveys	Dolphin/Tolovana Area, Cleary Hill,

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional, District & Property Geology

7.1.1 Regional Geology

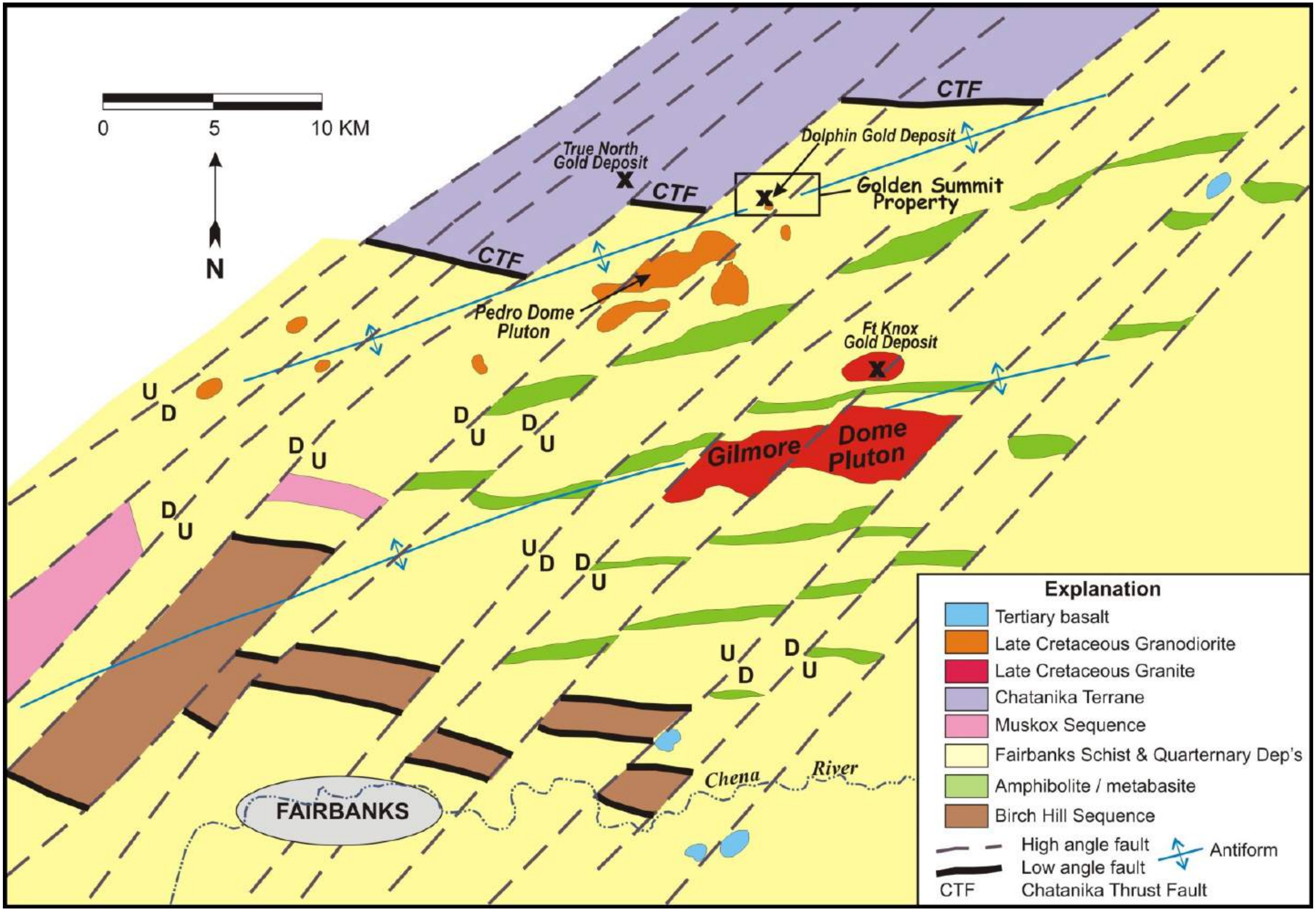
The following summary of the regional geology of eastern Interior Alaska is excerpted from Adams and Giroux (2012).

The Fairbanks Mining District is located in the north-central portion of the Yukon-Tanana Terrane (YTT). The YTT is a diverse lithotectonic terrane of largely continental affinity consisting primarily of quartzitic, pelitic, and calcic metasedimentary rocks; and local mafic and felsic meta-igneous rocks. These protoliths are intruded to a large extent by Mesozoic and Cenozoic granitic rocks (Foster and others, 1994; Newberry, 2000). The YTT is bound on the north by the Tintina-Kaltag fault system, and on the south by the Tanana-Denali-Farewell fault system. These fault systems form zones of major right lateral strike-slip movement, but are largely obscured by alluvial and other Quaternary deposits. Small subterranes of possible island-arc affinity occur along the south margin and in the northeast portion of the YTT (Nokleberg, et al, 1994).

Igneous rocks are widespread throughout the YTT, but are most abundant in the eastern portion of the province. Age dates of plutonic rocks in the YTT generally cluster into three distinctive groups: 1) 215-188 million years ago (Ma) (Late Triassic–Early Jurassic); 2) 110-85 Ma (mid- to Late Cretaceous); and 3) 70-50 Ma (Latest Cretaceous-Eocene). Within the 110-85 Ma group, most age dates cluster within a sub-group ranging in age from 95-90 Ma, and typically referred to as the “Tombstone” suite (Mortinson et al, 2000); plutonic compositions of the Tombstone suite ranges are dominantly granite, granodiorite, quartz monzonite and diorite. The Tombstone suite plutonic rocks are thought to be derived from crustal melts, but could also be mantle-derived melts with significant crustal material contamination. Volcanic rocks in the YTT are far less voluminous than plutonic rocks. Volcanic rocks ranging from Cretaceous to Cenozoic in age, and from rhyolite to basalt in composition, are found in scattered locations throughout the YTT.

7.1.2 Fairbanks District Geology

Bedrock geology of the Fairbanks Mining District is dominated by a N60-80E trending lithologic and structural trend covering a 30-mile by 15-mile area (Robinson and others, 1990; Newberry and others, 1996). The Project is situated in lower to middle Paleozoic metavolcanic and metasedimentary rocks of the Cleary sequence and Fairbanks Schist adjacent to an east-west trending thrust fault known as the Chatanika thrust (**Figure 7-1**). Rocks of the Fairbanks Schist and Cleary Sequences are exposed at Golden Summit in the Cleary antiform, the northern of two northeast trending antiformal belts which form distinctive marker horizons in the mineralized portions of the district. Lithologies within the Cleary Sequence include quartzite, massive to finely laminated mafic to intermediate flows and tuffs, calc-schist, black chloritic quartzite, quartz-sericite schist of hydrothermal origin and impure marble. Lithologies in the Fairbanks Schist include quartz muscovite schist, micaceous quartzite and biotite quartz mica schist. These lithologies have been metamorphosed to the lower amphibolite facies.



From: Newberry and others, 1996; modified by Avalon Department, 2008

FIGURE 7-1
GENERAL GEOLOGY OF THE FAIRBANKS MINING DISTRICT

Current maps for the Fairbanks District indicate rocks of the Fairbanks Schist and Cleary Sequence have been over thrust from the northeast by eclogite to amphibolite facies rocks of the Chatanika terrane (Newberry and others, 1996;). The Chatanika terrane consists of quartz muscovite schist, carbonaceous quartzite, impure marble, garnet feldspar muscovite schist, and garnet-pyroxene eclogite that have yielded Ordovician Ar40/Ar39 age dates ranging from 470 to 500 Ma (Douglas, 1997). Motion on the Chatanika thrust fault has been dated at approximately 130 million years and resulted in structural preparation of favorable host units in the Chatanika Terrane and adjacent lower plate rocks. Diamond drilling and trenching completed on the Project by Freegold have encountered Chatanika Terrane rocks over a zone extending up to one mile south of the mapped contact of the Chatanika Terrane. The location of these exposures suggests that the contact between the upper and lower plate is in fact a series of en-echelon low angle structures. This mixed terrane can be distinguished on airborne magnetics maps as a zone of intermediate magnetic intensity that is less than the highly magnetic rocks of the Chatanika Terrane but more magnetic than the Fairbanks Schist (Freeman, 2009). The ramifications of this hypothesis are discussed in **Section 7.2**.

Intrusives in the Fairbanks District have yielded Ar40/Ar39 and K-Ar dates of 85 to 95 million years (Freeman and others, 1996). These intrusives range in composition from diorite to granite and possess elevated Rb/Sr ratios indicative of significant crustal contribution to subduction generated magmas. Several granodiorite to aplite intrusive bodies are present in the Project area. The presence of hypabyssal intrusives and sporadic Au-W skarn mineralization in the Project area suggests the area may be underlain by more extensive intrusive bodies similar to those on Pedro Dome and Gilmore Dome (Freeman and others, 1998). This conclusion is supported by airborne geophysical surveys (DGGS, 1995) and by depth modeling conducted on these airborne data (PRJ, 1998). Mineralization within the Pedro Dome, Gilmore Dome and Dolphin intrusive complexes suggests plutonic rocks pre-date mineralization.

Rocks on the Project are folded about earlier northwest and northeast trending isoclinal recumbent fold axes followed by an open folded N60-80E trending event (Hall, 1985). Upper plate rocks of the Chatanika Terrane have been affected by more intense northwest and northeast trending isoclinal and recumbent folding followed by folding along the same N60-80E trending axis which affected lower plate rocks. Lithologic packages in both the upper and lower plates are cut by steeply dipping, high angle northwest and northeast trending shear zones, some of which are mineralized (**Figure 7-1**). Recent large-scale trenching in the Cleary Hill mine area suggest that numerous low angle structures are present in the Project area, some of which are mineralized. Late post-mineral north-south structures with normal motion further dissect the project. Airborne magnetic data in this part of the Fairbanks District indicate the presence of district scale east-west and northeast trending structures which appear to post-date N60-80E folding (DGGS, 1995). Gold mineralization on the Project post-dates regional and district scale folding and is contemporaneous with or slightly younger than district-scale northeast trending structures and plutonic activity. Excavations completed in the Cleary Hill area in 2006, 2007 and 2008 clearly indicate that the strike and/or dip of gold-bearing quartz veins were influenced by pre-existing fold geometry. This subject is discussed in more depth under **Section 9.0**.

7.1.3 Golden Summit Project Geology

The following summary of the Project general geology is derived in large part from Freeman (2009) and Adams and Giroux (2012).

Three main rock units underlie the Property, including rocks of the Fairbanks Schist, rocks of the Chatanika Terrane, and intrusive rocks (**Figure 7-2**). The Fairbanks Schist and Chatanika Terrane have both been subjected to one or more periods of regional metamorphism. The intrusive bodies are post-metamorphism. Chatanika Terrane rocks are found structurally above the Fairbanks Schist and north of the Chatanika Thrust fault and comprise the northernmost portion of the property. Intrusive rocks are relatively minor on the Property, and are primarily represented by the Dolphin stock, although small granitic dikes are known in several locations.

Most of the Property is underlain by the Fairbanks Schist. The Fairbanks Schist consists largely of quartz-mica schist and micaceous, massive to laminated quartzite, with lesser amounts of amphibolite, chlorite schist, calc-schist and marble. A unit within the Fairbanks Schist, referred to as the "Cleary Sequence", consists of three mappable sub-units containing distinctive and highly variable lithologies. The lower portion of the Cleary Sequence (~450 feet thick) consists of massive, mafic metavolcanic rocks (flows and tuffs), and minor actinolite schist, quartzite, and dolomite. The middle portion of the Cleary Sequence (~300 feet thick) consists of massive quartzite, feldspathic quartz schist, and quartz mica schist. The upper portion (~250 feet) is similar to the middle portion, but is distinguished by the presence of interlayered marble and minor amounts of garnet-bearing schist. Locally the Cleary Sequence is capped by a distinctive gray, sulfide-bearing marble unit up to 50 feet thick.

Chatanika Terrane rocks on the Property include muscovite-quartzite, coarse-grained muscovite schist, amphibolite, massive actinolite greenschist, chlorite schist, and local garnet-diopside eclogitic rocks (Swainbank, 1971). Chatanika Terrane mafic rocks are not readily discernible from mafic rocks of the Fairbanks Schist either in hand specimen or drill core. This has created difficulties with mapping, logging and establishing a stratigraphic section in the Tolovana Mine and Cleary Hill Mine areas. The Dolphin stock is located on the ridge between Bedrock and Willow Creek. Initial diamond core logging identified five intrusive phases within the Dolphin stock, including: 1) fine- to medium-grained, equigranular to weakly porphyritic biotite granodiorite; 2) fine- to medium-grained, equigranular to weakly porphyritic hornblende-biotite tonalite; 3) fine-grained biotite granite porphyry; 4) fine-grained biotite rhyolite to rhyodacite porphyry; and 5) rare fine-grained, chlorite-altered mafic dikes (Adams and Giroux, 2012).

Limited drill data suggests the north and west contacts of the Dolphin stock are fault contacts (Adams and Giroux, 2012). The south and east contacts are largely intrusive contacts with minor faulting.

Due to the paucity of radiometric age dates, limited outcrop, and limited observations of crosscutting relations, the crystallization and mineralization history of the Dolphin stock remain unknown. Small dikes of granodiorite cutting tonalite have been observed in core, and altered granitic dikes cut both altered and unaltered granodiorite and tonalite, suggesting multiple phases of intrusion and hydrothermal alteration. Two radiometric age dates, including two sericite Ar40/Ar39 plateau age dates (McCoy, 1996), place some constraints on the timing of crystallization and mineralization. The sericite ages were obtained from two different samples representing two distinctly different styles of gold mineralization. One sample, from stockwork style mineralization, was 90.1 Ma. Another sample, from a sericite shear-zone, was 88.3 Ma. These ages are quite similar to ages from Fort Knox (86.3-88.2 Ma). Due to age and chemical similarities, most workers associate the Dolphin and Fort Knox intrusive rocks with widespread intrusive-related gold deposits in the Tintina Gold Belt.

Nearly all rocks comprising the Property are highly deformed. Primary foliations (S_0) in the Fairbanks Schist generally dip north on the north half of the property and generally dip south on the south half of the property, defining the Cleary antiform, a large-scale northeast trending antiform. Deformation intensity increases further north, with proximity to the Chatanika Thrust fault. The Chatanika Thrust fault is thought to represent an ancient thrust event, and one of the earliest deformation events in the area.

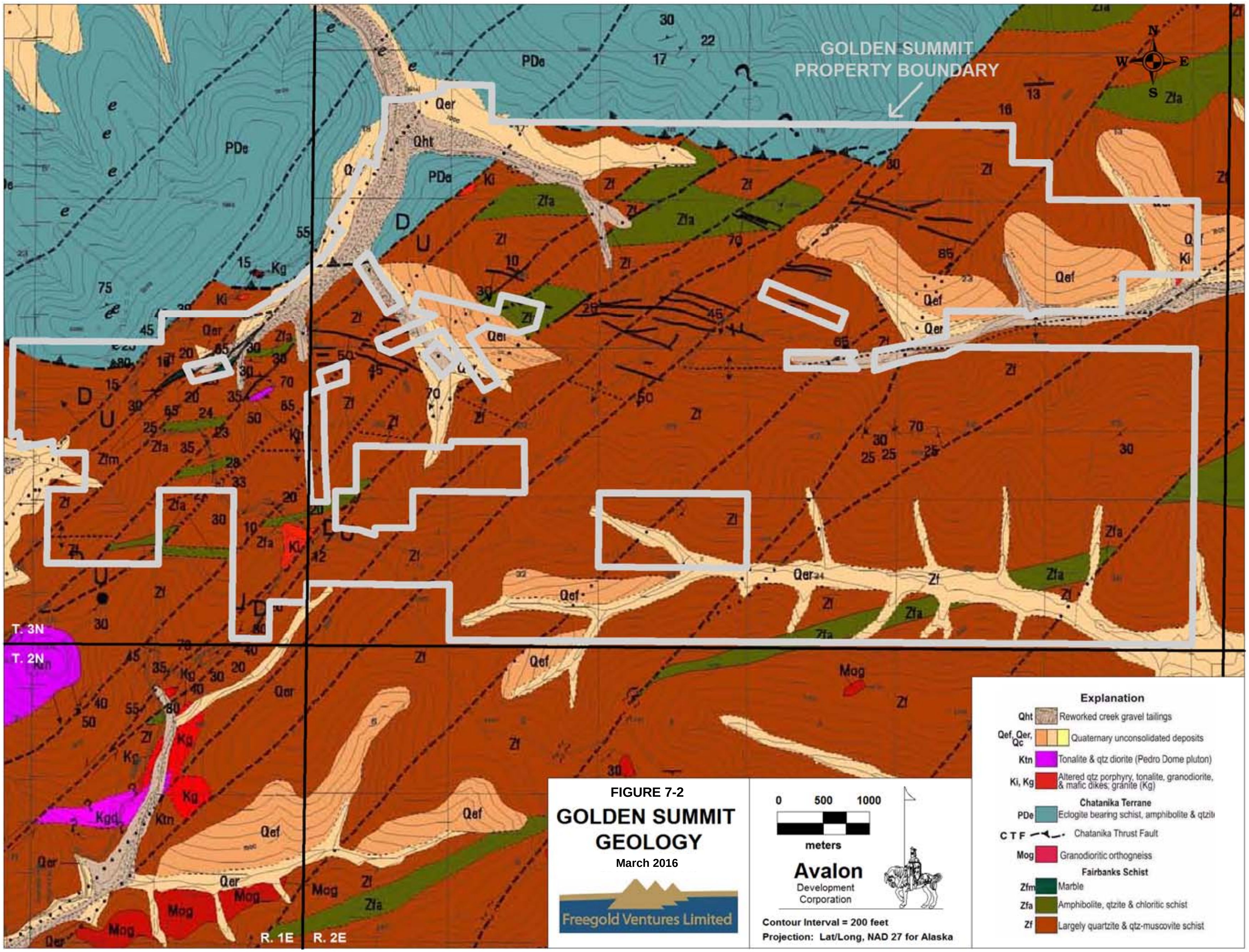


FIGURE 7-2
LOCAL GEOLOGY AND MAJOR PROSPECTS ON THE PROJECT (Geology from Newberry et al, 1996)

Rather than a simple fault contact as shown on published geologic maps of the district, the Chatanika Thrust fault is a complex thrust fault zone containing numerous thin thrust sheets or wedges emplaced above and in between layers of various Fairbanks Schist lithologies (Freeman, 2009). The Chatanika Thrust fault has been offset by numerous northeast-trending high angle faults. These types of faults are very common throughout the northern part of the Yukon Tanana Terrane, and typically represent a very late stage structural event. The Chatanika Thrust fault may also have been re-activated during later deformation events, or served as the focus of north-directed gravity or listric style fault activity. The next oldest structural event is thought to be represented by the high angle faults and shear zones which host the major auriferous quartz veins found at numerous locations on the property. These zones are largely oriented northwest-southeast, however, northeast-southwest oriented shear zones, which are otherwise very similar in terms of structural style and mineralization; occur to the west of the Dolphin deposit and at several other locations on the property. The veins most often dip steeply towards the south, but occasionally dip north. Field evidence for repeated veining, alternating with brecciation suggests the mineralization within these zones was largely syn-deformational. Short offsets (<20ft) of the veins occur along the youngest structures observed at the Property, along steep, north to northeast-trending normal faults.

7.2 Mineralization

Over 63,000 strike feet of mineralized shear zones have been identified within and immediately adjacent to the Project (Freeman and others, 1996). The majority of the mineralized shear zones on the eastern end of the project trend N60-80W and dip steeply to the southwest. Shear zones on the western end of the project area predominantly trend N60-80E and dip steeply north. Shear zones in the central portion of the project (centered on the Dolphin/Cleary Hill area trend closer to east-west with variable south dips and appear to mark a transition zone from primarily northwest trending, south dipping shears to the east to primarily northeast trending, north dipping shears to the west. Bulk sampling completed in 2006, 2007, and 2008 has exposed mineralized flat-lying (10-30 degrees (°)) structures dipping both north and south. The extent and economic significance of these flat-lying structures is uncertain. In addition, exploration activities conducted by Freegold have identified previously unrecognized shear zones trending N30-50W and due north-south (Freeman and others, 1998). These shear zones possess significantly different metal suites than flat-lying structures or N80W and N60E trending shears. These shear zone geometries and their distribution may represent sympathetic structures generated by regional scale shear couples related to Tertiary (post 55 Ma) motion of the Tintina and Denali faults (Flanigan and others, 2000).

Examination of the spatial arrangement of the +80 known gold occurrences in the Project area and the geometry of the +63,000 linear feet of documented gold-bearing quartz veins in the area suggest veins tend to cluster into discrete vein swarms. These vein swarms are controlled by a series of district-scale northeast-trending structures regularly spaced approximately 8,000 feet (2.4 km) apart in the Project area. These structures were first identified as district scale features evident on public airborne geophysical surveys conducted in the mid-1990's (DGGS, 1995). Their periodicity with respect to clusters of known gold occurrences was unrecognized prior to 2004 when it was recognized on the Project (Freeman, 2004). The Eldorado fault, which appears to control mineralization at both the Ryan Lode and the True North deposits, is the best documented of these district scale northeast structures. The Dolphin trend, located parallel to and 8,000 feet east of the Eldorado fault, is the next best-defined northeast-trending structure and probably is critical to the mineralization in the Newsboy, Tolovana, and 6 Moz Dolphin/Cleary Hill areas. Approximately 8,000 feet farther east, an unnamed northeast-trending structure passes through the Saddle zone where it may be integral to the formation of the highest known density of veins in the Fairbanks Mining District, including those which host gold mineralization at the historic McCarty, American Eagle, Pioneer and Pennsylvania mines. Eight thousand feet further east, another unnamed northeast-trending structure passes through the Hi Yu mine area and probably is key to the formation of multiple veins in this area of the Project. This 8,000-foot periodicity probably extends to the east where northeast structures may control mineralization on Coffee Dome and to the west of the Eldorado Creek fault where they may control gold mineralization in the Treasure Creek area and the Sheep Creek area of Ester Dome.

The other recently recognized feature of gold mineralization in the Project area is related to the structural relationship between "lower plate" rocks of the Fairbanks Schist – Cleary Sequence and "upper plate" rocks of the Chatanika Terrane. Published maps of the district (Robinson and others, 1990; Weber and others, 1992; Newberry and others, 1996) indicate that the contact between the overlying Chatanika Terrane and rocks of the lower plate are marked by a single north-dipping thrust plane that strikes northeast according to Robinson and others (1990) or east-west according to Newberry and others (1996). Douglas (1997 dated this thrust event at 130 Ma based on data derived from a single core hole drilled by Placer Dome on the south flank of Marshall Dome near the northwestern edge of the Project. The actual contact between upper and lower plate rocks is not exposed at surface anywhere along its mapped trace so the inferred motion direction (thrust versus low-angle gravity fault) remains uncertain. Regional scale kinematic evidence is permissible for the formation of either gravity or thrust faults. Douglas (1997) presents evidence of multiple low-angle fault events which structurally interpose thin (<250 feet) layers of upper and lower plate rocks over a +750-foot interval. Chemical evidence for structurally juxtaposed upper and lower plate rocks has also been documented in drilling in the Cleary Hill mine area (Freeman and others, 1998).

With the exception of gold and antimony mineralization in the vicinity of the True North deposit, published geologic maps of the district indicate that all of the historic lode gold, tungsten and antimony occurrences in the Project area are hosted in lower plate rocks. However, reinterpretation of the airborne magnetic data for the Project suggests rock with magnetic signatures identical to the Chatanika Terrane (variable but high magnetic susceptibilities) extend considerably farther south than current published geologic maps indicate. In the field, geological and multi-element geochemical data suggest that virtually all of the known lode gold occurrences on the Project are hosted in a zone containing structurally mixed lithologies derived from both upper and lower plate rocks. This mixed zone appears to be the result of multiple en-echelon low angle structures separating upper and lower plate rocks. If this interpretation is correct, the grade and geometry of gold mineralization in the Project area may be controlled in part by physical and/or chemical conditions that existed at the time of mineralization along or adjacent to en-echelon low-angle faults caused by emplacement of the Chatanika Terrane.

The major historic lode gold mines of the Project derived their production primarily from steeply dipping northwest and northeast trending high angle, low sulfide, gold-polymetallic quartz veins and shear zones which transect what is now thought to be the mixed upper plate - lower plate rock package at Golden Summit (Hill, 1933; Pilkington, 1969; Metz, 1991; Freeman and others, 1996). These shear zones are characterized by a metal suite containing free gold with variable amounts of tetrahedrite, jamesonite/boulangerite, arsenopyrite, stibnite and scheelite with minor base metal sulfides. Fluid inclusion data suggest mineralization was associated with high CO_2, low salinity fluids at temperatures averaging 350° Celsius (C). Lead and sulfur isotope data, tellurium geochemistry and tourmaline compositions suggest a strong plutonic component to the Golden Summit shear hosted mineralization (McCoy and others, 1997).

There are three styles of gold occurrences identified on the Property, including: 1) intrusive-hosted sulfide disseminations and sulfide-quartz stockwork veinlets (such as the Dolphin gold deposit); 2) auriferous sulfide-quartz veins; and 3) shear-hosted gold-bearing veinlets. All three types are considered to be part of a large-scale intrusive-related gold system on the Property.

7.2.1 Instrusive-Hosted Sulfide-Quartz Veinlets

Intrusive-hosted, auriferous sulfide disseminations and auriferous sulfide-quartz veinlets (0.1-5 mm) within the Dolphin stock are spatially associated with the highest gold grades within the Dolphin gold deposit (**Figure 7-3**). Gold also occurs with disseminated euhedral arsenopyrite (1 to 5 mm) which appear to be an earlier, higher temperature mineralization event (McCoy and Olson, 1997). Gold mineralization within the deposit also occurs as mineralized fault gouge enriched with sulfides, sulfide-rich veins, and locally as narrow sulfide-quartz veins <6 inches thick; however, these comprise a relatively small portion of the total gold resource.

Gold within the Dolphin gold deposit occurs largely as inclusions in sulfides, and locally as visible grains, within the sulfide-quartz veinlets. Pyrite and arsenopyrite is the most common sulfide mineral, although stibnite, lead-antimony sulfosalt minerals, tetrahedrite, scheelite, galena and sphalerite occur locally. McCoy and Olson (1997) identified two distinct varieties of arsenopyrite in the Dolphin gold deposit based on arsenopyrite geothermometry and age relations. Older arsenopyrite from quartz stockworks (90.1 Ma) formed at higher temperatures, whereas younger arsenopyrite from shear zones formed at lower temperatures (88.3 Ma). McCoy also noted that older "hotter" arsenopyrites were finer-grained compared to younger "cooler" arsenopyrites, which were generally coarse and bladey. Furthermore, the high-temperature arsenopyrite contains particulate inclusions of gold, whereas the low-temperature arsenopyrite contains maldonite (a gold-bismuth mineral). Although stibnite and antimony sulfosalts are not uncommon in the deposit, geochemical studies suggest that high antimony values are generally associated with very low gold values. Evidence suggests that the fluids evolved towards increasing base metals and antimony with time (**Figure 7-6**). For example, chalcopyrite embayments in pyrite were noted in thin section, and massive sulfide veins (jamesonite, galena, stibnite and/or sphalerite) cutting arsenopyrite-quartz veins are noted in several drill logs. In addition to sulfides, some portions of the Dolphin gold deposit contain abundant scheelite.

Several forms of alteration have overprinted the Dolphin intrusive rocks. The most common alteration types are chloritization, kaolinitization, silicification and sericitization. Carbonate alteration, as calcite or less commonly dolomite or iron carbonate, is found locally. Alteration can range from weak to intense, and is generally indicative of higher gold values, in particular, when strong silicification and sericitization are present. As mentioned, strong sericite alteration is characteristic of shear zones, but weak to moderate sericite alteration is ubiquitous throughout the deposit and appears to be one of the earliest phases of hydrothermal alteration in the Dolphin deposit. Detailed core logging suggests the paragenetic sequence of alteration and mineralization events at the Dolphin deposit range from early sericite alteration and disseminated arsenopyrite ± pyrite through sheeted auriferous quartz-sulfide veining to coarse grained pyrite-dominated ± base metal sulfide veining (no quartz associated).

7.2.2 Auriferous Quartz Veins

High grade auriferous quartz veins (2 cm to 3 m), hosted in metamorphic rocks, occur at numerous locations, and were the source of all previous gold production from the Property. A discussion of each occurrence is beyond the scope of this report; the general mineralogy, morphology and structural setting is summarized below. Detailed information for individual vein prospects on the Property can be obtained from previous reports (Freeman, 1992).

The auriferous quartz veins typically crosscut the host rock primary foliation at very high angles. A large number of these veins dip south, although some veins dip north. Vein thickness is quite variable, and can range from a few inches to several feet over short distances along both strike and dip. Pinch-and-swell features, bifurcations and splays are characteristic. Discrete auriferous quartz veins often have sharp wallrock contacts but can grade into shear zones suggesting a continuum between this type of gold quartz veining and shear-hosted gold described below (Brown and others, 2008a, 2008b). In contrast to the high grade quartz veins, barren, translucent or milky colored metamorphic quartz most often occurs as seams or boudinage sub-parallel to the primary foliation of the host rocks.

Auriferous quartz veins on the Property consist of hydrothermal quartz with minor to trace amounts of sulfides. The veins are opaque to milky white quartz and locally gray to mottled gray and white. Bands or laminations parallel to the vein walls are not uncommon, and vein centers often contain vuggy or comby quartz crystals. Silicified vein breccia is also common, and may comprise the entire vein or be restricted to bands within the banding sequence (Adams and Giroux, 2012). This suggests there were most likely multiple, possibly alternating episodes of silicification and deformation. Auriferous quartz veins seldom contain more than 5% total sulfides and average 1-3%. The most common sulfide is arsenopyrite, although other sulfides are locally present, including pyrite, stibnite, jamesonite, tetrahedrite, galena and sphalerite. Scheelite is present in a few specific veins (notably abundant in the Cleary Hill and Wyoming vein). Visible gold typically occurs as coarse flakes, filigree, or wires suspended in quartz or mingled with sparse, scattered sulfides. Locally the auriferous quartz veins may be accompanied by parallel stringers and pods of later massive stibnite. This massive stibnite occurs locally as <10 inch (<0.25 m) thick seams or pods parallel or adjacent to auriferous quartz veins, and also as veins up to 4 feet (1.3 m) thick along steep cross-faults which offset the auriferous quartz veins. This stibnite mineralization is thought to be formed as the last metal-bearing event at lower temperatures.

7.2.3 Shear-Hosted Veinlet Zones

Shear-hosted auriferous veinlet zones on the Golden Summit Property are found within some of the same shear zones which host major auriferous quartz veins and, as mentioned above, are likely parts of the same mineralization event. The key characteristic of these zones is that they may contain sufficient polyphase veinlet density and gold grade to justify bulk-mining methods. Several of these zones have been explored since about 1969, including the Too Much Gold prospect, the Circle Trail and Saddle prospects, and the Curry Zone. Most recently, several zones in the Cleary Hill Mine area have been targeted by Freegold and included in the resource estimate outlined in this report (refer to **Section 14**).

The shear-hosted veinlets consist largely of quartz with variable amounts of sulfides, although locally the veinlets may consist largely of sulfides with lessor amounts of quartz. Sulfide-quartz veins within the shear-hosted zones generally are less than a few centimeters in thickness. Locally these veins form vein sets with spacing of a few feet, resembling a sheeted vein system (vein swarm). The veins are discontinuous along strike and dip, and often grade into broken veins, vein breccia, or zones of sugary, granulated crush quartz material. Higher quartz vein and veinlet density is generally indicative of higher gold values.

The shear-hosted veinlet zones are characterized by pervasive sericite and clay alteration, as well as localized silicification and carbonate alteration. In addition, the zones are typically highly oxidized near the surface, and contain locally intense iron, arsenic or antimony oxides. The majority of the veinlets within the zones are sub-parallel to the strike and dip of the zone.

Host rocks for the veinlet zones are quite variable. Differences in rock competency appears to influence the geometry of mineralization within and adjacent to the deformation zone. For example, massive quartzite or greenstone units are more competent, and tended to propagate fractures where fluids were more restricted, resulting in the formation of thinner but often higher grade gold quartz veins. In comparison, thin-bedded units with higher pelitic, carbonaceous and calcareous components are more susceptible to shearing and widespread infiltration by metal-bearing fluids, resulting in stockwork of sheeted vein zones. Therefore, key factors are thought to be the right combination of host rock lithology, location within a major shear zone, and access to a hydrothermal fluid source. These zones are best developed where multiple shears or faults intersected and caused widespread fracturing and increase permeability within metamorphic host rocks.

Figure 7-3: Shear Hosted Breccia & Quartz Vein Zone – GSDL 12-10

Figure 7-4: Quartz Stockwork Zone in Granodiorite & Tonalite – GSDC 11-32

Figure 7-5: Intense Quartz Stockwork Zone (End of Hole) – GSDC 11-32

Figure 7-6: Intense Brecciation and Fractured Schist hosted Stockwork – GSDL 12-01

8.0 DEPOSIT TYPES

Recent discoveries in the Fairbanks District have outlined a series of distinctive mineral occurrences which appear to be genetically related to mid-Cretaceous plutonic activity which affected a large area of northwestern British Columbia, Yukon, Alaska and the Russian Far East (Flanigan and others, 2000). This work, based on extensive geologic and structural mapping and analytical studies (major and trace element analysis, fluid inclusion microthermometry, 40Ar/39Ar geochronology, and isotope analysis) has provided new information regarding gold metallogenesis in the Fairbanks District (Baker and others, 2006; Burns et al., 1991; Lelacheur et al., 1991; Hollister, 1991; McCoy et al., 1994; Newberry et al., 1995; McCoy et al., 1995). A synthesis of this information (Hart et al., 2002, McCoy et al., 1997, Lang and others 2001) suggests a deposit model in which gold and high CO_2 bearing fluids fractionate from ilmenite series, I-type mid- Cretaceous intrusions during the late phases of differentiation. The gold is deposited in anastomosing pegmatite and/or feldspar selvage quartz veins. Brittle fracturing and continued fluid convection and concentration lead to concentration of gold bearing fluids in intrusions and schist-hosted brittle quartz-sericite shear zones. Carbonate and/or calcareous metabasite horizons host W-Au skarns and replacement deposits. Structurally prepared calcareous and/or carbonaceous horizons may host bulk-minable replacement deposits. These occur most distal to the intrusions within favorable host rock in the Fairbanks Schist and Chatanika Terrane.

Seven different potentially economic gold deposit types have been identified in the Fairbanks District.

1. Gneiss or high-grade schist-hosted quartz veins or metasomatic replacement zones proximal to or within causative intrusives. Metals associated include Au, Bi, and As and possibly Cu. W. Pogo (+7 Moz) and Gil (+0.5 Moz) are examples of such mineralization.

2. Stockwork-shear style mineralization hosted in porphyritic intermediate to felsic intrusives. Mineralization contains Au with anomalous Bi, Te, W and trace Mo. There is a strong genetic relationship between host intrusion and gold mineralization. Examples include Fort Knox (10 Moz) and the Eagle (+3 Moz).

3. Porphyritic stockwork with intrusion/schist shear hosted Au-As-Sb with a strong genetic relationship between host intrusion and gold mineralization. Ryan Lode (2.4 Moz) and Dolphin area are examples of this type of mineralization.

4. Base metal ± Au, Ag and W intrusion hosted mineralization with a possible genetic relationship between precious metal mineralization and intrusion. Silver Fox prospect is an example.

5. Structurally controlled mineralization hosted by schist-only high angle shear zones and veins. Associated metals include Au, As, Sb, Ag, Pb and W in low-sulfide quartz-carbonate veins. Alteration adjacent to veins is pervasive quartz-sericite-sulfide alteration that can extend for up to one mile from the source structure. Deposits were mined heavily prior to World War II and are noteworthy because of their exceptional grades (+1 to +5,000 ounces per ton (opt) Au). Examples include Cleary Hill (281,000 oz production), Christina (20,000 oz production), American Eagle (60,000 oz production), Hi Yu (110,000 oz production) and Newsboy (40,000 oz production) veins.

6. Low angle, disseminated, carbonate-hosted Au-As-Sb mineralization associated with brittle thrust or detachment zones distal to generative intrusives. The True North deposit (1.3 Moz) is an example of this type of mineralization.

7. Shear-hosted monomineralllic massive stibnite pods and lenses. Trace As, Au, Ag and Pb but these prospects are noteworthy because they appear to represent the most distal end members of the intrusive gold hydrothermal systems. Examples include the past producing Scrafford and Stampede mines.

9.0 EXPLORATION

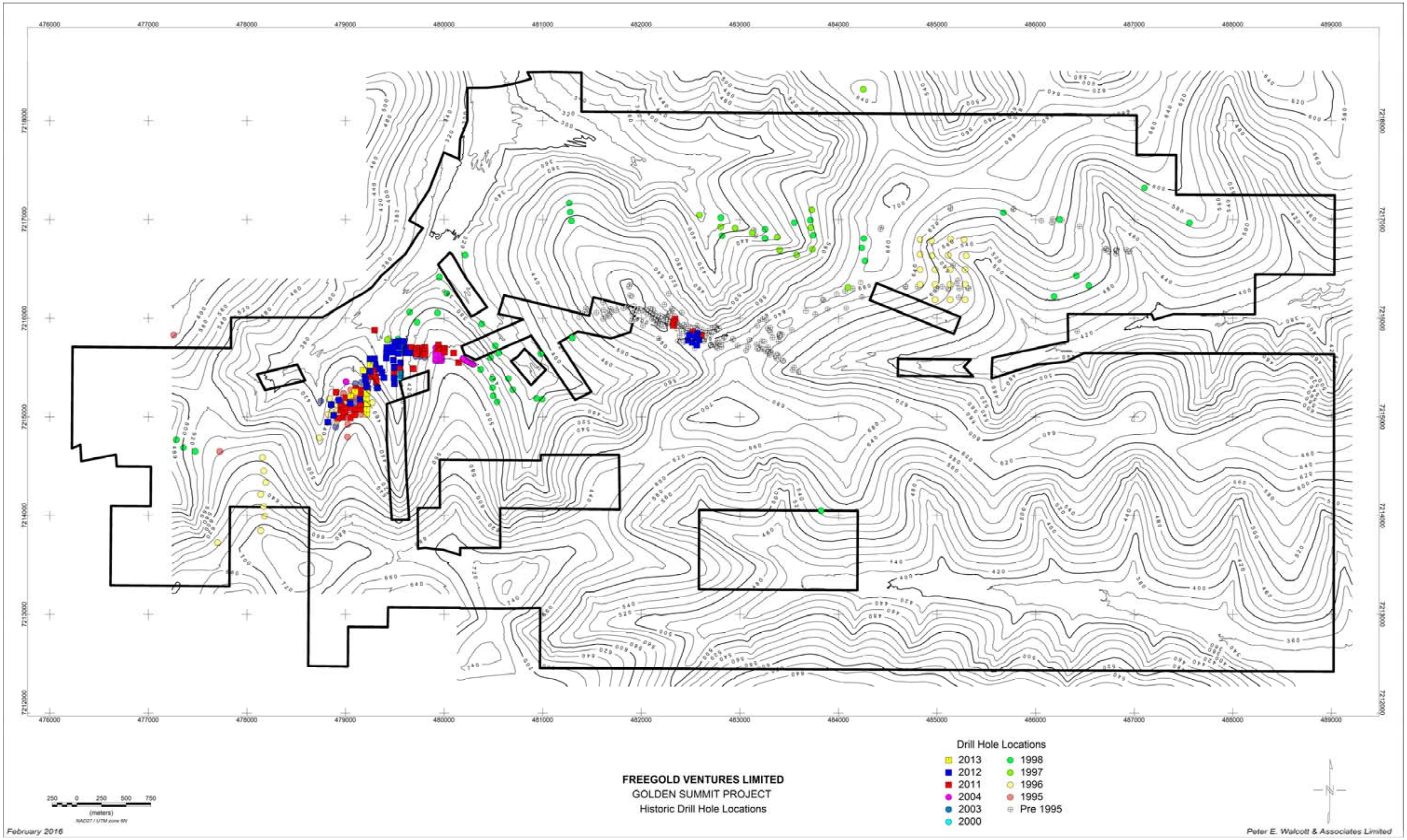
In 2010 Freegold commenced a comprehensive compilation program on the Project with a view to establish the potential for a bulk tonnage resource within the Dolphin/Cleary Hill area. Exploration commenced with an induced polarization survey in the Dolphin area which indicated a strong correlation between areas of known mineralization and potential area for expansion. Between 2011 and April, 2013, Freegold completed 102,183 feet (31,145 meters) of core drilling in 117 holes primarily in the Dolphin/Cleary Hill areas. In addition to the drilling induced polarization, limited trenching, shovel soil sampling, rock sampling and cultural resource activities were also undertaken. A summary of activities is presented in Adams and Giroux (2012) and Abrams and Giroux (2013) and is not repeated here and is filed on SEDAR. Drilling recommenced in July 2013 on the Dolphin area, with a total of 5,468 feet (1,666 m) in three holes completed. These holes were not included in the current resource however the results were determined not to have material effect on the resource. The three new holes are compared to the estimated blocks they pass through in **Section 14.8** were examined Exploration efforts since 2013 have included water quality sampling, cultural resource studies, metallurgical testing, as well as additional ground geophysical and geochemical surveys.

10.0 DRILLING

A summary of pre-2013 drilling activities is presented in Adams and Giroux (2012) and Abrams and Giroux (2012) and is not repeated here. A map showing all Freegold drilling is presented in **Figure 10-1**.

Drilling on the Golden Summit property during 2013 consisted of diamond core drilling in the Dolphin/Cleary Hill gold resource area. Freegold completed drilling a total of 16,860 feet (5,138 meters) of HQ (2.5 inch) and NQTW (1.995 inch) core in ten drillholes (**Table 10-1**; **Figure 10-2**). The locations of the 2013 drilling are shown in **Table 10-1**. **Figure 10-2** is a map showing the collar locations of the drillholes in the Dolphin/Cleary Hill gold resource. Significant assay results for all drillholes completed during 2013 are listed in **Table 10-2**.

All of the drilling was conducted with HQ sized core which resulted in excellent core recoveries in spite of difficult ground conditions, particularly within the schist and breccia zones. In addition to better recoveries it also provides for larger sample size which is normally more representative.



Source: Mark J. Abrams, Reno, NV

FIGURE 10-1
HISTORIC DRILLHOLE LOCATIONS – GOLDEN SUMMIT PROPERTY

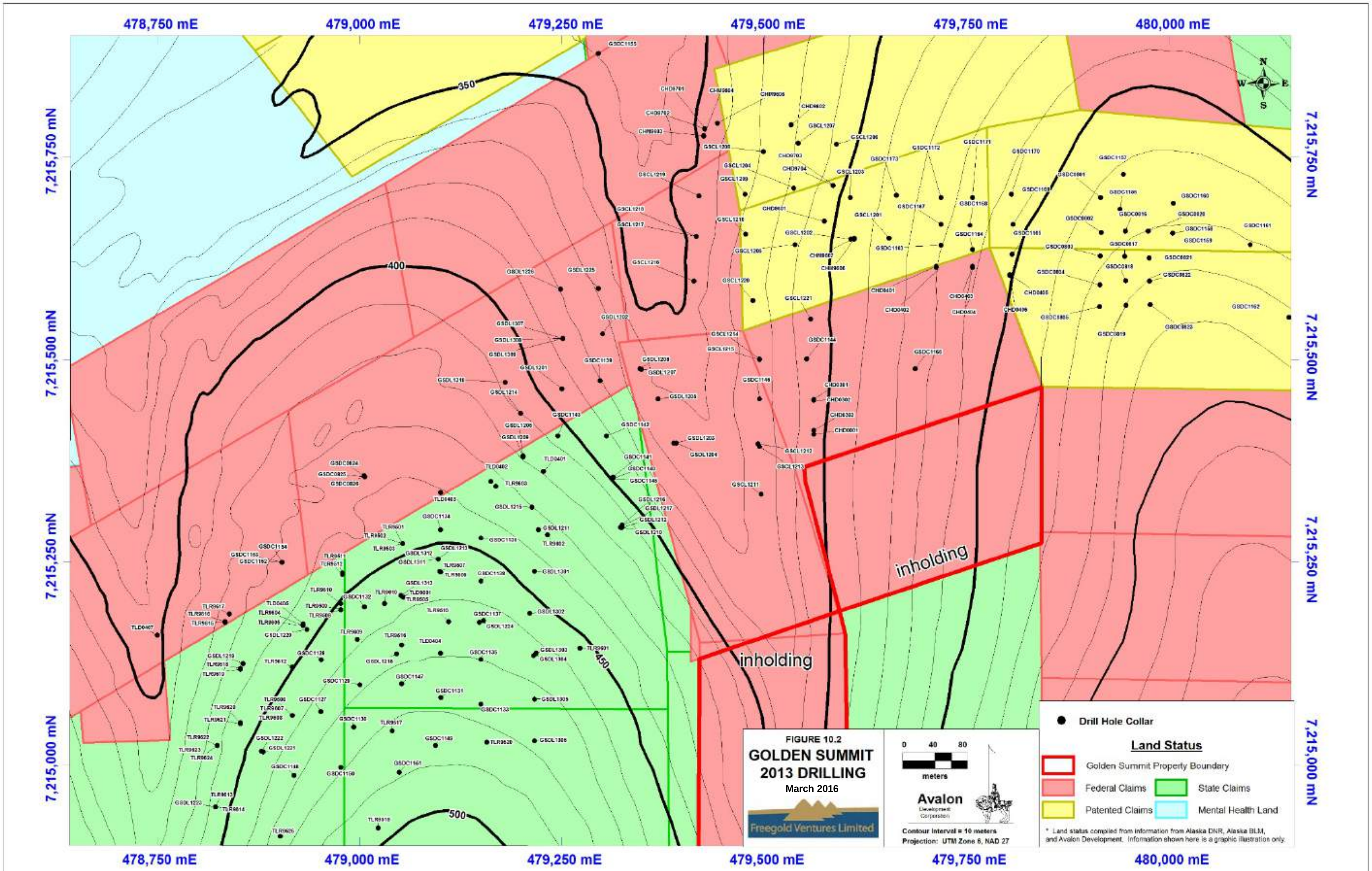


FIGURE 10-2
DRILLHOLE LOCATIONS INCLUDING 2013 DRILLING

Table 10-1: Drillholes Completed on the Property during 2013

Hole	Prospect	Easting	Northing	Elev. (ft)	Azimuth	Dip	TD (ft)
GSDL1301	Dolphin	479216	7215239	1473	360	-90	1489.5
GSDL1302	Dolphin	479210	7215187	1496	360	-90	2000
GSDL1303	Dolphin	479218	7215138	1526	360	-90	86.5
GSDL1304	Dolphin	479215	7215135	1526	360	-90	2000
GSDL1305	Dolphin	479216	7215081	1555	360	-90	2000
GSDL1306	Dolphin	479211	7215032	1572	360	-90	1597
GSDL1307	Dolphin	479252	7215527	1266	360	-55	232
GSDL1308	Dolphin	479252	7215527	1266	360	-55	161
GSDL1309	Dolphin	479252	7215527	1266	360	-62	917
GSDL1310	Dolphin	479173	7215468	1328	360	-55	907
GSDL1311	Dolphin	479097	7215154	1529	180	-75	1922
GSDL1312	Dolphin	479097	7215154	1529	360	-75	1832
GSDL1313	Dolphin	479051	7215209	1512	360	-70	1714.5

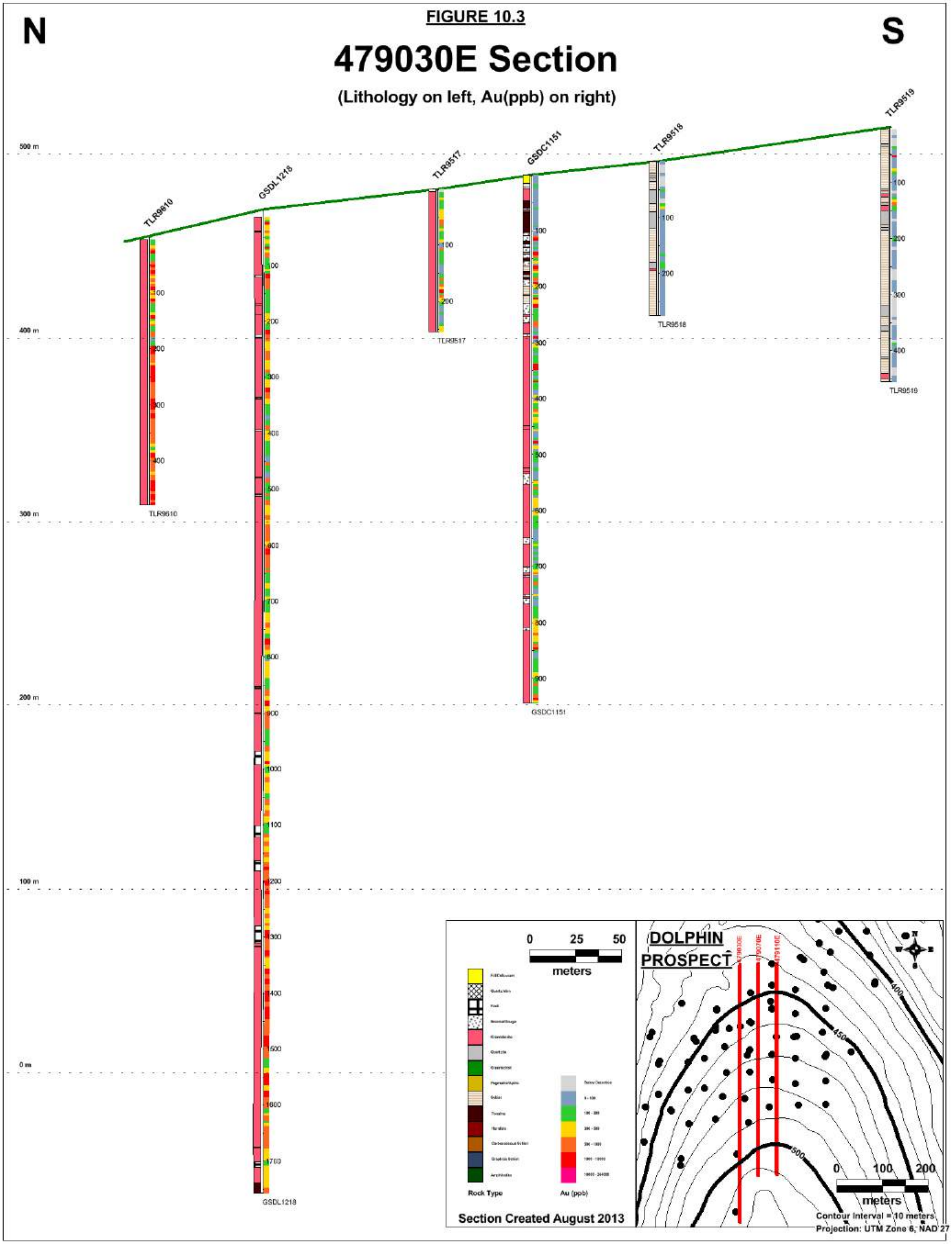
Holes GSDL 1311, GSDL 1312, and GSDL 1313 were not included in the current resource.

Table 10-2: Significant Core Drilling Assay Results for the 2013 Dolphin/Cleary Drillholes

Hole #	Hole Incl.	TD (ft)	From (ft)	To (ft)	Interval (ft)	Interval (m)	Au g/t
Dolphin Area							
GSDL1301	-90	1489.5	937	1489.5	552.5	168.4	0.64
including			1047	1489.5	442.5	134.9	0.71
GSDL1302	-90	2000	787	923.5	136.5	41.6	0.51
including			986.5	2000	1013.5	308.9	0.63
including			1259	1626.6	367.6	112.0	1.03
GSDL 1304	-90	2000	32	367	335	102.1	0.21
			648	2000	1352	412.1	0.58
including			678	848	170	51.8	0.61
including			1606	2000	394	120.1	0.87
GSDL 1305	-90	2000	12	232	220	67.1	0.23
			512	530	18	5.5	1.96
			619.5	2000	1380.5	420.8	0.46
including			1837	2000	163	49.7	1.02
GSDL 1306	-90	1597	118.5	252	133.5	40.7	0.40
			632	809	177	53.9	0.50
			977	1597	620	189.0	0.42
including			1247	1597	350	106.7	0.54
GSDL 1309	-62	917	64.5	98.5	34	10.4	0.69
			245.5	328	82.5	25.1	0.74
			512	612.5	100.5	30.6	0.96
			755.5	848	92.5	28.2	0.53

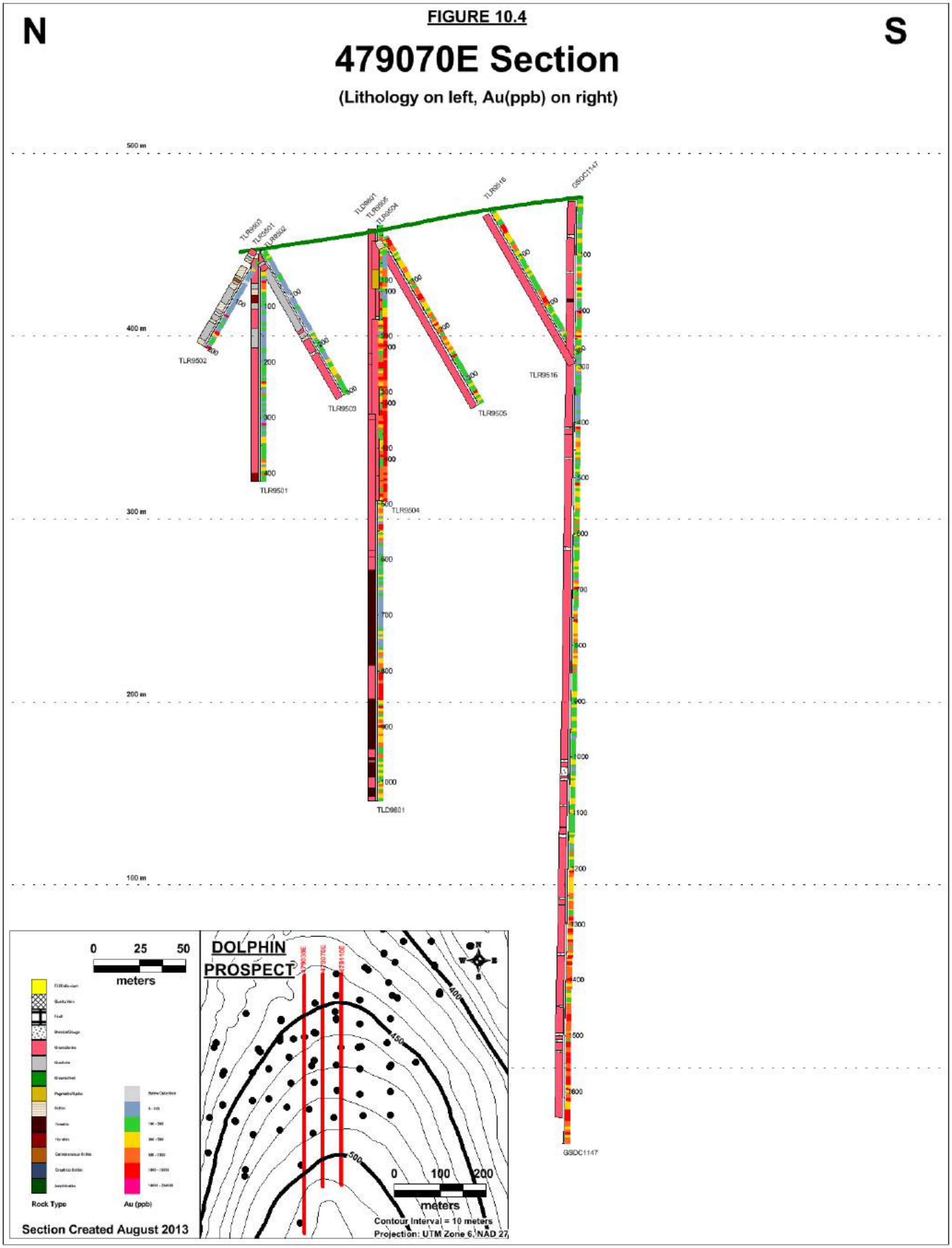
Hole #	Hole Incl.	TD (ft)	From (ft)	To (ft)	Interval (ft)	Interval (m)	Au g/t
GSDL 1310	-55	907	23	136.5	113.5	34.6	0.91
			275	302	27	8.2	3.29
			381	541	160	48.8	0.64
including			506	541	35	10.7	1.47
			654	890	236	71.9	0.56
including			782	890	108	32.9	0.83
GSDL 1311	-75	1922	37	1922	1885	574.52	0.82
		incl	37	78	41	12.50	2.61
		incl	243	347	104	31.70	1.48
		incl	377.5	558	180.5	55.01	0.75
		incl	667	798	131	39.93	0.62
		incl	1039.5	1628	588.5	179.67	1.13
		incl	1728	1922	194	59.13	0.87
GSDL1312	-75	1832	19	1832	1813	552.6	0.68
			19	88	69	21.03	0.54
			185	425.5	240.5	73.30	0.54
			507	596	89	27.13	3.00
			736	802	71	21.64	0.76
			1578	1795	217	66.14	1.76
		incl	1767	1795	28	8.53	7.49
GSDL1313	-70	1714.5	8	1714.5	1706.5	520.14	0.49
			13	178	165	50.29	0.62
			577	740	163	49.68	0.72
			811.5	883	71.5	21.79	1.15
			980	1068	88	26.82	1.39
			1340.5	1696.5	356	108.51	0.54

In the figures below are representative sections depicting geology and assay results through the central portion of the Dolphin Deposit. **Figure 10-3**, **Figure 10-4** and **Figure 10-5** are north south sections looking towards the east.



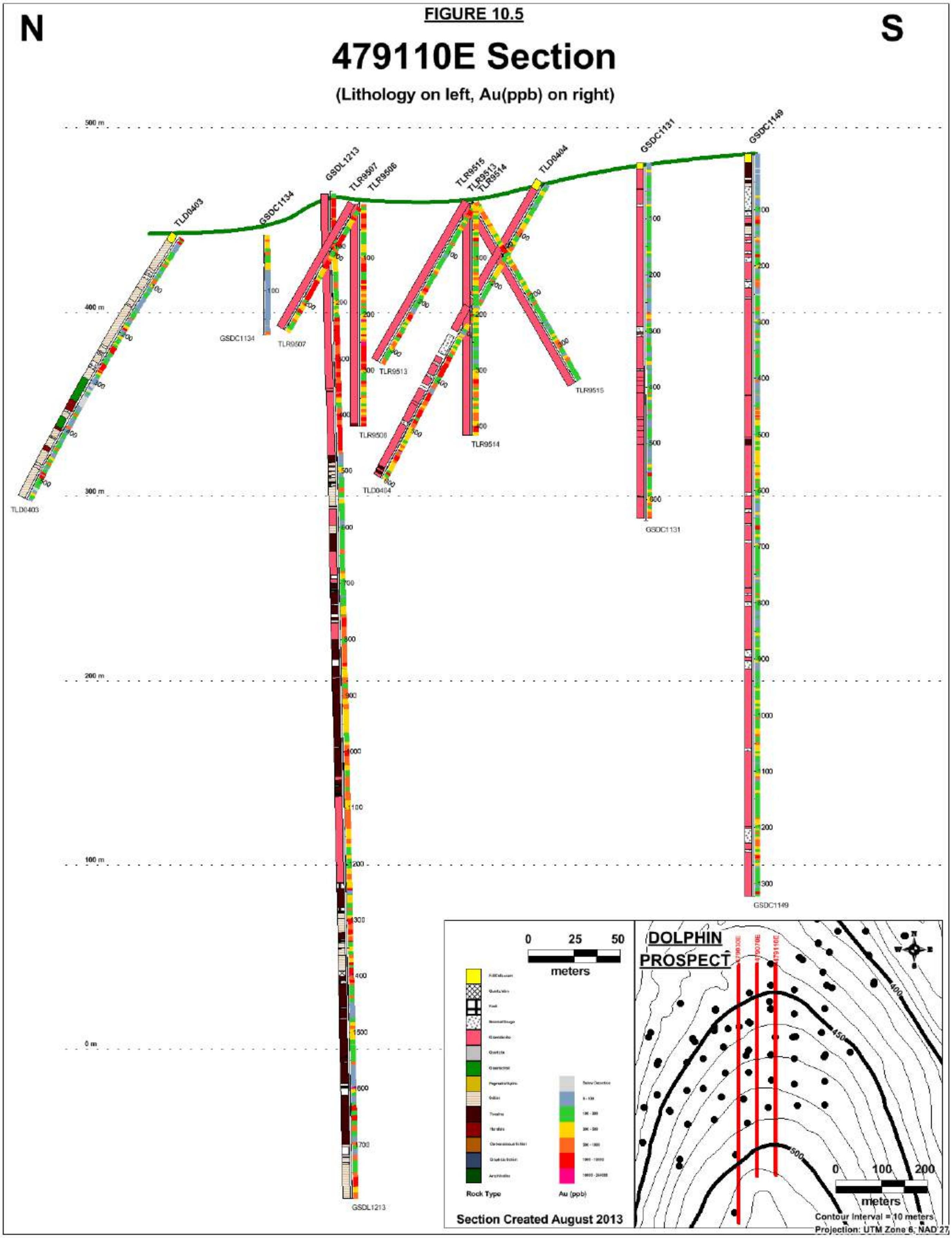
Source: Mark J. Abrams, Reno, NV

FIGURE 10-3
CROSS SECTION THROUGH DOLPHIN PROSPECT WITH LITHOLOGY AND GOLD ASSAYS



Source: Mark J. Abrams, Reno, NV

FIGURE 10-4
CROSS SECTION THROUGH DOLPHIN PROSPECT WITH LITHOLOGY AND GOLD ASSAYS



Source: Mark J. Abrams, Reno, NV

FIGURE 10-5
CROSS SECTION THROUGH DOLPHIN PROSPECT WITH LITHOLOGY AND GOLD ASSAYS

10.1 CHRISTINA PROSPECT

During 2011 and 2012 Freegold completed its first ever drilling in the Christina Prospect. Previous drilling at Christina (+70,000 feet from 1977 to 1988) was focused solely on outlining a high grade vein resource on the prospect (Freeman, 1992). No effort was made to explore for bulk tonnage mineralization associated with the Christina vein and Freegold conducted no other work on the prospect until 2011.

During 2011 and 2012 a total of 12 holes were drilled (15,058 feet) (4,589 meters). The holes were targeted on a combination of known geological structure and chargeability anomalies outlined by the induced polarization survey. Drilling has indicated a good correlation between chargeability and mineralization. The bulk of the mineralization is associated with quartz veins and quartz stockworks with associated pyrite and arsenopyrite. Host rocks are predominately chloritic schists. Several of the holes intersected broader zones of mineralization indicative of bulk tonnage potential. Additional drilling is contemplated, however the focus remains the Dolphin/Cleary area.

Table 10-3: Significant Core Drilling Assay Results for the 2011 to 2012 Christina Drillholes

Hole #	Hole Incl.	TD (ft)	From (ft)	To (ft)	Interval (ft)	Interval (m)	Au g/t
GSDC 1175	-55	818.5	226.5	505	278.5	84.9	0.64
GSDC 1176	-55	736	402	614.5	212.5	64.7	1.75
GSDC 1178	-55	785	215	345	130	39.6	0.39
			544	615	71	21.6	0.38
GSCH 1201	-55	733.5	20	78	58	17.6	0.48
			243.5	383	139.5	42.5	0.32
GSCH1202	-50	748	135	204	69	21	0.56
			290	379	89	27.1	0.4
			679	748	69	21	0.35
GSCH1203	-50	810	118	205.5	87.5	26.7	0.3
			456	554.5	98.5	30	0.39
GSCH1204	-50	700	80	148.5	68.5	20.9	0.37
			514	629.5	115.5	35.2	0.35
GSCH1205	-50	863.5	240	310	70	21.3	0.42
			460	574.5	114.5	34.7	0.67
			702	753.5	51.5	15.7	1.07
			833.5	858.5	25	7.6	0.4
GSCH1206	-50	830	276.5	495	218.5	66.6	0.49
GSCH1207	-50	848	93.5	155	61.5	18.75	0.42
			339	448	109	33.22	0.78
including			426	433	7	2.13	7.9
			491	582	91	27.74	0.63
			789	838	49	14.94	0.36

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following summarizes the procedure used for sample preparation, analyses and security for drill samples collected during the drilling programs completed on the Golden Summit Project.

The database has been maintained by Freegold's prime geological contractor Avalon Development Corporation of Fairbanks, Alaska. Personnel from Avalon have been involved in each of the programs undertaken on the Golden Summit Project by Freegold. The author has had held numerous discussions with Avalon in regard to sampling protocol. A digital database has been maintained of all assay and geochemical work completed on the project, including results from all the drilling programs, both Reverse Circulation (RC) and diamond (Core); and RAB (Rotary Air Blast) as well as rock and soil sampling. Since 1997 all rock and soil geochemical samples collected were described in the field and located using hand-held global positioning system (GPS) methods. Data from each sample was then entered into a digital GIS-database for later interpretation. Channel samples collected on the project were taken along the trench floor or rib using a rock pick and chisel as required. Channel sampling using a power saw was attempted but abandoned due to the heavy weathering and penetrative cleavage of metamorphic rocks on the project, both of which made such sampling difficult and potentially unreliable.

The following is a summary of the methods and procedures employed for the various drill campaigns.

The bulk of the resource drilling, approximately 70%, that comprises the resource was completed during the 2011–2013 timeline, and accordingly, the discussion is heavily weighed to those programs.

11.1 1992–2004

Drilling completed on Golden Summit consisted of both diamond core and down-hole hammer reverse circulation drilling. The majority of the drilling conducted was RC. All drilling conducted during these programs was managed by Avalon Development and was conducted by local and national drilling contractors.

All reverse circulation and rotary air blast samples were quick-logged on-site by an experienced geologist and later detail logged using representative chip samples from each 2.5, 3 or 5 foot sample interval. Reverse circulation samples were one-eighth to one-quarter split, depending on hole diameter while 100% of RAB cuttings were collected, all core samples were sawed at variable intervals depending on visible geological criteria and shipped to the geochemical lab for analysis.

During all programs, Avalon Development collected, logged and retained the samples collected in the field until turned over to a commercial laboratory representative. Selected sample pulps were reanalyzed by metallic screen methods to quantify nugget effect in high-grade samples or where visible gold was noted during sampling.

All samples collected on the Golden Summit project were retained at Avalon's secure warehouse facility until picked up by Chemex or Bondar Clegg. Sample preparation was completed by Chemex or Bondar Clegg in their laboratories in Anchorage and/or Fairbanks, and analytical work was completed by Chemex Labs and Bondar Clegg Ltd. at their facilities in Vancouver, B.C, Analytical work consisted of a series of gold by fire assay plus multi-element inductively coupled plasma (ICP) analyses. Sample preparatory procedures employed by the laboratory at that time were not available to the author, but as Bondar Clegg, and subsequently ALS Chemex are well-recognized laboratories it was expected that sample preparation would have been conducted in line with industry accepted practices

Prior to 2000 all samples were prepared using two acid digestion procedures. Sampling conducted in 2000 through 2004 used four acid digestion procedures. In 1996, Quality assurance consisted of duplicate samples, which were inserted on a one for ten basis. During 1997-1998, additional Quality Assurance was added with the addition of blanks and standards. Blanks were inserted on a 1 for 25 basis from 2000-2004, and commercially prepared standards were introduced on a 1 to 50 basis during 2004.

11.2 2005–2011

Exploration during 2005 focused on a limited trenching program. During 2005, Alaska Assay Labs, a Fairbanks facility, prepared trench samples and ALS Chemex Labs completed sample analysis until August 2005. (See 2011-2013 discussion for general laboratory preparatory procedures).. Commercial standards containing 1.5 and 2.5 gpt gold were introduced on a 1:50 basis in 2005. Analyses of variance performed on samples analyzed by ALS Chemex indicated no unacceptable sample results in the standard submitted.

RAB (Rotary Air Blast) exploration drilling commenced in 2006. Samples were collected during the 2006 and 2007 RAB program and from January through June 2008 exploration programs. Sampling consisted of a 100% split of the drill cuttings. Samples were collected by Avalon Development personnel and weighed from 4 kilograms to 54 kilograms, averaging about 7 kilograms. The samples were weighed and logged on-site and transported daily to a locked warehouse at Avalon Development's office complex for subsequent pick-up, preparation and analysis by ALS Chemex and/or Alaska Assay Laboratories. A new sampling procedure was introduced as of June 2007, which consisted of collecting all samples on 2.5-foot intervals and passing 100% of the sample through a Jones-type splitter until the sample intended for analysis weighed between 250 and 500 grams. Depending on the volume of drill cuttings coming from the drill interval, this meant splitting the sample between 4 and 7 times (averaging 5 splits) to reach the desired sample weight. Results of RAB drilling have been viewed as a geochemical tool and have not been incorporated into the resource.

Commercial standards containing 0.627 ppm, 2.56 ppm, 4.46 ppm, or 11.33 ppm gold were included in sample streams for 2006 at a rate of 1 per 25 for rock and channel samples and 1 per rotary air blast drill hole (approx. 1 per 17-25 samples). No unacceptable analysis results were returned for these standards from either ALS Chemex or Alaska Assay Labs. During the program one duplicate sample was inserted per hole (average 45 feet) and a blank or standard was inserted every 10 samples.

Samples collected from September 2006 were prepared and analyzed entirely by Alaska Assay Laboratories, which was a member of the AHK Group and was fully accredited to ISO 17025. Sample procedures utilized by the laboratory include strict chain of custody, sample recording, preparation particle size, monitoring blanks, duplicates and blanks within given sample batches. Samples were crushed to 70% passing- 10 mesh, a 250 gram riffle split was taken, and then subsequently pulverized to 85% passing-200 mesh. The subsequent pulp was assayed utilizing Fire Assay with an AA finish. Samples in excess of >10,000 ppb gold would be automatically rerun with a gravimetric finish. . No unacceptable analysis results were returned for these standards and blanks from either ALS Chemex or Alaska Assay Labs.

11.3 2008 CORE DRILLING

Twenty-three core holes totaling 8,839.5 feet of drilling were completed, several of which were completed in the resource area. During the 2008 core drilling program, a total of 117 blank samples were inserted into the sample submittals. Sample blanks were inserted on a two per one hundred sample basis and consisted of Browns Hill Quarry basalt, an unmineralized Quaternary basalt flow from the Fairbanks Mining District, Alaska. Eight different commercial standards provided by Analytical Solutions were also used. Values in these standards raged from 0.627 ppm to 11.33 ppm gold. Whole core analyses was performed by Alaska Assay Labs, Fairbanks, Alaska (Subsequently acquired by Acme Laboratories). No unacceptable analysis results were returned for these standards and blanks from Alaska Assay Labs . Samples were crushed to 70% passing -10 mesh, and then a riffle split of 250 grams was taken. This split was subsequently pulverized to 85% passing -200 mesh. Analytical procedures included fire assay for gold using AA/Grav which had detection limits ranging from 10 ppb to 0.10/oz t.

The Core logging, chain of custody and sampling procedures employed were primarily the same as those in subsequent program.

The following summarizes the procedure used for sample preparation, analysis and security for drill samples collected in the Golden Summit drilling programs:

a. Core was moved by Avalon from the drill rig to the secure logging facilities at each shift change.

b. Core boxes were stacked in numerical order in the core logging area.

c. Core boxes were inspected for proper labeling and core in the boxes was inspected to insure that the core was placed in the boxes at the drill rig in the proper order with the proper footage markings on the core run blocks.

d. Core was moved to logging tables and placed in order by box number such that the lowest numbered box (with the shallowest drill core) was on the far left side of the logging bench and while the highest numbered box (with the deepest drill core) is on the far right side of the logging bench.

e. Core was washed with a spray bottle to remove polymer or other drill mud. Due to the presence of coarse free gold, core was not washed with a brush since this could smear coarse gold particles from a mineralized to an unmineralized interval.

f. Core recovery (ratio of core recovered in a given core run to the actual length of the core run) was calculated and marked on the logging sheet for each core run interval pulled by the drilling company. This information was entered in the logs as a percent- recovered.

g. The RQD, or Rock Quality Designation was calculated for each core run. The RQD is the combined length of all whole core segment in each core run that were greater than 10 cm (4 inches) or longer than twice the core diameter, divided by the total length of the recorded core run multiplied by 100 (expressed in % form). The total length of core includes all lost core sections. Breaks in the core that result from the drilling process or extraction of the core from the core barrel are usually fresh looking and have rough edges. These mechanical breaks were ignored while calculating RQD. For the NQ2 drill core drilled at Golden Summit (diameter 1.995 inches), samples qualifying for addition in the RQD calculation would be 4 inches or more in length. RQD information was recorded in percentage form on the logging sheet for each core run interval pulled by the drilling company.

h. The drill core was logged by a senior geologist with experience in the rock type, alteration and mineralization. Details relating to lithology, structure, alteration and mineralization were recorded systematically. Lithologic details were compiled on paper logs, and later converted to digital format. Structural details were measured and their angle to core axis recorded in the log. Details relating to the thickness, angle and other aspects were recorded in the log. Hydrothermal alteration features, such as quartz or sericite alteration, were noted in the logs and details relating to its extent and intensity were recorded. Hydrothermal mineralization was recorded in the log. Details recorded include morphology, mineralogy and color of quartz veins, sulfide mineralogy, form and abundance (in volume %), metallic oxide mineralogy, form and relative abundance, and any other feature related to gold, gold-pathfinder or other metallic mineralization. The geologist took close-up digital photographs of unique or otherwise significant features described above.

i. Following logging, the geologist selected sample intervals for geochemical analyses. Selection of sample intervals utilized all the visual rock information gathered by the logger as well as any information gathered through the use of additional tools such as an XRF hand held analyzer, hand held geophysical tools, ultraviolet lamp or any other analytical tool that provided additional information about the geologic environment and mineralization. Sample intervals did not cross core recovery block boundaries. Sample intervals were no longer than 5 feet in length and no shorter than 0.5 feet in length. The minimum core sample length was predicated on obtaining sufficient sample from which to create a 500 gram pulp. The selection of intervals for geochemical analysis focused on selecting the shortest sample interval that the accumulated logging information indicates was a unique zone, structure or area of mineralization. Similarly, wider zones that appear to be gold mineralized were all sampled as a unit. Wooden blocks, designating the sample number and starting footage mark, were placed in the core boxes to guide the sampler. These sample blocks were marked in red while core footage run blocks were marked in black. Care was taken in assigning sample numbers to allow for insertion of blanks and standards into the sample stream. Blanks and standards comprised approximately 10% of the samples submitted to the lab from any given drill hole.

j. The core was digitally photographed. During this process the core was wetted to enhance picture quality and photographed under high intensity electric lights utilizing plain light spectrum bulbs. Each core box was photographed with a placard denoting hole number and footage contained in the box. Core run block and sample interval blocks were plainly visible in the pictures. Digital resolution was +5 mega-pixels to insure extremely high quality results. In addition to photographing each core box, close-up or macro photos were taken by the core logger of any obviously mineralized intervals, significant alteration or textures, noteworthy lithologic contacts, distinctive structural zones, etc. The core logger kept an accurate written log of the footage and hole number of these macro photos were crossed referenced to the digital file name. Once a given hole was photographed completely, the file name of the macro photos was changed to reflect the hole number and footage of each macro photo.

k. Once all hole photos from a given hole or part of a hole was taken, they were checked for quality and completeness by Alina Wyatt, Avalon's QA/QC manager or Ken Wolf in the 2008 program. Unclear or incomplete photos were re-photographed, re-checked and added to the complete digital database for each hole.

l. The original hand-written drill core logs were scanned to a digital format (Adobe pdf) and the resulting scans were checked for clarity and completeness. Hard copy hand drill logs were converted to a digital drill log format (Excel format) to allow for their use in GIS and/or resource estimation software. The Excel file was checked for accuracy and completeness against the original hand written drill log by a third party and any discrepancies were rectified and errors or omissions corrected. Where necessary, the core logger referred to the core to make corrections, additions or other changes.

m. Once QA/QC checks were completed on core logs and core photos, a digital copy of the core logs and core photos was burned to a DVD and stored off-site. In addition, these data were stored on at least 2 computers in two separate buildings on Avalon's premises and were transmitted to Freegold via ftp or email.

n. Sampling Procedure: Once all of the above steps were completed and verified by the geologist, each marked geochemical sample interval was extracted from the core box.

    i. *2008 Sampling Procedure:* 100% of the core from each sample interval was placed in a canvas sample bag bearing the sample number on the sample interval block in the sample bag. Extra care was taken to insure that only rock and rock fragments from the proper interval were collected in the sample bag. This sampling was done by a two person team who cross-referenced sample numbers of intervals on the core logs to the sample blocks and the sample numbers on the sample bags. The individual sample bags were sealed and stored in Avalon's warehouse for subsequent batch shipping to the geochemical lab.

    ii. *2011–2013 Sampling Procedure:* Core was split in half length-wise using a tile saw fitted with a diamond blade. Every section of core drilled was then sampled by taking one half of the core drilled between each set of run blocks. Extra care was taken to ensure that only rock and rock fragments from the proper interval were collected in the sample bag. The individual sample bags were sealed and stored in Avalon's warehouse for subsequent batch shipping to the geochemical lab. The remaining half core is stored in the original boxes at Avalon's core logging facility.

o. Senior Avalon personnel and the core logger completed the geochemical laboratory submittal paperwork. Bagged and labeled samples were then loaded into large nylon poly- sacks capable of holding 2,000 pounds. Representatives of the geochemical lab collected the poly-sacks and handled all sample preparation and analysis from that point forward. The minimum instructions required for each sample shipment included:

    i. Project Name and client billing instructions.

    ii. Name or description for the sample preparation methods requested.

    iii. Name or description for the sample pulp size (500 grams).

    iv. Name or description of Au analysis procedure (Fire Assay, gravimetric finish) and description of over-limit condition and action required by laboratory.

v. Name or description of multi-element package analysis procedure (if any) and description of over-limit condition and action required by laboratory.

vi. Method for distribution of analytical results.

11.4 2011

A total of 10,790 samples were analyzed, including assay and QAQC samples. The types of QAQC samples used included standards, blanks and duplicates. Standards were inserted at a rate of approximately 7 standard samples per 100 assay samples (7%), blanks were inserted at a rate of approximately 2 blank samples per 100 assay samples (2.3%), and duplicates (a quarter-section of core) were inserted at a rate of approximately 1 duplicate sample per 100 assay samples (1%).

The standards used are commercially available from a reputable vendor (Analytical Solutions). The standards used had values ranging from 0.098ppm gold to 7.15ppm gold. An attempt was made to use lower gold value standards (with higher base metal values) in zones known to contain higher sulfide contents, and higher gold value standards were used where high gold values in the core were suspected. Seventeen different standards were used, with fifteen expected values, including: 7.15ppm Au, 0.334ppm Au, 0.527ppm Au, 1.02ppm Au, 1.81ppm Au, 2.57ppm Au, 3.63ppm Au, 0.885ppm Au, 0.098ppm Au, 0.841ppm Au, 0.627ppm Au, 1.52ppm Au, 4.76ppm Au, 1.24ppm Au, 2.0ppm Au. All except three standard samples returned acceptable values (within approximately 15% of the expected value, or approximately one standard deviation). Those standard samples which returned suspect values were re-run at Avalon's request, and in all cases the re-assay values fell within the acceptable range.

Blank samples consisted of Browns Hill Quarry basalt, an unmineralized Quaternary basalt flow from the Fairbanks Mining District, Alaska. Avalon Development has an extensive data base of assay values for this material which provides a reliable base-line for determining expected geochemical values. All except five blank samples returned acceptable values. Those blank samples which returned suspect values were re-run at Avalon's request, and in all cases the re- assay values fell within the acceptable range.

11.5 2012

QAQC samples were inserted into the drill sample strings on the basis of approximately 1 QAQC sample per 10 assay samples (approximately 10%). A total of 13,519 samples were analyzed, including assay and QAQC samples. The types of QAQC samples used included standards, blanks and duplicates. Standards were inserted at a rate of approximately 7 standard samples per 100 assay samples (7%), blanks were inserted at a rate of approximately 2 blank samples per 100 assay samples (2.3%), and duplicates (a quarter-section of core) were inserted at a rate of approximately 1 duplicate sample per 100 assay samples (1%).

Sixteen standards were used in the 2012 drill program. Four standards were obtained from Rocklabs and ranged in value from 0.203 ppm gold to 3.562 ppm gold. Twelve standards were obtained from Analytical Solutions and ranged in value from .334 ppm gold to 7.15ppm gold. An attempt was made to use lower gold value standards (with higher base metal values) in zones known to have a higher sulfide concentration, and higher gold value standards were used where high gold values in the core were suspected. Of the 941 standards used in the 2012 drill program, 11 returned values differing more than 15% from the expected value. Those standard samples which returned suspect values were re-run at Avalon's request along with core samples surrounding the standard in question, and in all cases the re-assay values fell within the acceptable range.

Blank samples consisted of Browns Hill Quarry basalt, an unmineralized Quaternary basalt flow from the Fairbanks Mining District, Alaska. Avalon Development has an extensive data base of assay values for this material which provides a reliable base-line for determining expected geochemical values. The Author reviewed the sample preparation, security and insertion of blanks and standards and is of the opinion the sampling was completed to industry standards.

11.6 2013

QAQC samples were inserted into the drill sample strings on the basis of approximately 1 QAQC sample per 10 assay samples (approximately 10%). A total of 2,448 samples were analyzed, including assay and QAQC samples. The types of QAQC samples used included standards, blanks and duplicates. Standards were inserted at a rate of approximately 7 standard samples per 100 assay samples (7%), blanks were inserted at a rate of approximately 2 blank samples per 100 assay samples (2.4%), and a duplicate sample was taken every 100 samples (1%). Standard and blank samples were analyzed in order of sample number by ALS Chemex along with the core samples. The coarse reject material to be used for the duplicate samples was returned to Avalon by ALS Chemex and will be sent to another lab for further quality assurance.

Thirteen standards were used in the 2013 drill program. Five standards were obtained from Rocklabs and ranged in value from .414 ppm gold to 3.562 ppm gold. Eight standards were obtained from Analytical Solutions and ranged in value from .334ppm gold to 7.15ppm gold. An attempt was made to use gold standards with higher base metal values in zones known to have a higher sulfide concentration, and higher gold value standards were used where high gold values in the core were suspected. Of the 71 standards used in the 2013 drill program, none returned values differing more than 15% from the expected value.

Blank samples consisted of Browns Hill Quarry basalt, an unmineralized Quaternary basalt flow from the Fairbanks Mining District, Alaska. Avalon Development has an extensive data base of assay values for this material which provides a reliable base-line for determining expected geochemical values.

Drill core from the 2011 – 2013 programs at Golden Summit were prepared at ALS Chemex in Fairbanks with pulps analyzed at either ALS Chemex's analytical facilities in Reno, Nevada or Vancouver, BC. Approximately half of the samples during the 2012 drilling campaign were sent to Acme Lab as ACME Lab had both prep and analysis laboratories in Fairbanks. ALS Chemex holds ISO 9001:2008 registration and an ISO 17025 accreditations for specific laboratory procedures. ACME was an ISO/IEC 17025 Accredited facility. There is no relationship between Freegold and any of the laboratories. Sample preparation procedures between the facilities has varied over time however, analytical work consisted of gold by fire assay with atomic absorption or gravimetric finish plus a variable multi-element suite analyzed by inductively coupled plasma emission spectroscopy (ICP) methods.

Laboratory Preparatory and Analytical Procedures have been largely derived from ALS Chemex and Acme Laboratories procedures that are publically available.

Samples Assayed by **ALS Chemex** generally underwent the following preparatory and assay procedures:

a. The sample was first logged in the tracking system, weighed, dried and finely crushed to better than 70 % passing a 2 mm (Tyler 9 mesh, US Std. No.10) screen. A split of up to 250 g was taken and pulverized to better than 85 % passing a 75-micron (Tyler 200 mesh, US Std. No. 200) screen. This method was utilized for rock chip or drill samples.

b. Excessively wet samples were dried in drying ovens. This is the default drying procedure for most rock chip and drill samples.

c. Fine crushing of rock chip and drill samples to better than 70% of the sample passing 2 mm. The sample was then split using a riffle splitter. The 250 g sample split was then pulverized to better than 85% of the sample passing 75 microns. In instance where gold only was required.: AA23 AU Atomic Absorption Spectroscopy (AAS) was performed.. A prepared sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead was digested in 0.5 mL dilute nitric acid in the microwave oven, 0.5 mL concentrated hydrochloric acid was then added and the bead was further digested in the microwave at a lower power setting. The digested solution was cooled, diluted to a total volume of 4 mL with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards. A 30 g sample weight was utilized. Detection Limits under this method are: 0.005 ppm to 10 ppm. Samples that returned greater that >10 ppm were automatically re-done using Au Grav 2 or Au-GRA22. Under this method the prepared sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents in order to produce a lead button. The lead button containing the precious metals was cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid, annealed and weighed as gold. Silver, if requested, is then determined by the difference in weights. Detection limits under this method ranged from 0.5 ppm to 1,000 ppm.

d. In the event multi-element analyses was requested generally the ME ICP61 was selected – Inductively Coupled Plasma – Atomic Emission Spectroscopy (ICP – AES). Under this method a prepared sample (0.25 g) was digested with perchloric, nitric, hydrofluoric and hydrochloric acids. The residue was topped up with dilute hydrochloric acid and the resulting solution analyzed by inductively coupled plasma-atomic emission spectrometry. Results are corrected for spectral interelement interferences. NOTE: Four acid digestions are able to dissolve most minerals; however, although the term “near- total ” is used, depending on the sample matrix, not all elements are quantitatively extracted.

e. Soil Sampling was also conducted on the project in 2011 and 2012. Soil samples were collected by digging a hole through the tundra mat cover down to the mineral soil layer and placing a sample of the soil into a marked bag. The clumps of moss and remaining soil were then returned and the hole was covered up. Sample weights were generally 250 – 500 grams. Samples were taken to ALS Chemex in Fairbanks for preparation and subsequent analysis at either their Vancouver, BC or Reno, Nevada analytical facilities. Multi-element analysis for gold and pathfinder elements was performed. Fire Assay for gold with an AA finish for the gold and four acid digestion was used for the 33 pathfinder elements. (ICP- AES). QA/QC was restricted to the laboratories internal QA/QC program for the soil sampling program.

Samples Assayed by **Acme Laboratories** generally underwent the following preparatory and assay procedures:

a. Excessive wet samples were first dried. The sample was crushed to 70% passing -10 mesh and then a 250 g split was taken. The 250g split was then pulverized to 85% passing 200 mesh. The preparation process and this split are subject to QA/QC control checks during the progression and prior to the submission to the analytical portion. A sieve test is used to monitor the process on select and random samples at the primary crushing stage and pulverization, which are recorded. In the event there is non-conformance the quality standard process is reviewed and corrected. In the instance where gold only assays were required Fire Assay 30 – with AAS Finish was selected. Detection limits ranges from 0.0005 to 10 ppm. Any assay that was greater than >10 ppm was automatically re-run employing a gravimetric finish.

Both ALS Chemex and Acme Laboratories have rigorous internal quality control standards, which utilize the use of their own standard, blanks and duplicates within the sample stream in addition to the standard, blanks and duplicates employed in the sample submittal process by Avalon.

It is the opinion of this author that the data collection, sampling, core recovery, chain of custody, preparation and analysis of the samples, and QA/QC protocol was conducted with a high level of due care, employing methods that meet or exceed industry standards.

12.0 DATA VERIFICATION

The following provides an overview of the Data Verification methods employed during the various exploration programs undertaken at Golden Summit.

Core photographs, and assay certificate as well as the database were reviewed. Spot checking of the assay database was also performed. During each exploration program Avalon Development undertook an evaluation of each sample batch as it was received and any spurious results were corrected by the analytical lab prior to the data being posted to the master geochemical database for the project. The Author has visited the Avalon Development logging facilities on multiple occasions and has noted the above detailed procedures being performed as described. The Author has also held several discussions with Avalon Development with regard to the QA/QC procedures employed and has not noted any areas of concern. It is the opinion of this author that the data collection, sampling, core recovery, chain of custody, preparation and analysis of the samples, and QA/QC protocol was conducted with a high level of due care, employing methods that meet or exceed industry standards.

13.0 MINERAL PROCESSING & METALLURGICAL TESTING

Metallurgical testing for the Project was initiated in 2012 with bottle roll tests being performed on 10 different drill samples. This testwork was performed by Kappes, Cassiday & Associates (KCA) with the final report dated March 21, 2012. The primary objective of this testwork was to obtain a preliminary indication of the cyanide leaching characteristics of the oxide mineralogy within the deposit.

A second set of process testwork was started in 2013 on five different mineralogical composites. These tests were performed by SGS Canada Inc. (SGS) with the final report dated May 21, 2014. This testwork primarily focused on investigation of various processing methods for the recovery of gold from sulfide materials.

Additional bottle roll and column leach testwork was performed in 2014 to investigate grind sensitivities in four drill core composites and to examine heap leach behavior in the oxide material. These tests were performed by McClelland Laboratories, Inc. with a final report dated January 9, 2015.

13.1 KCA TESTWORK

13.1.1 Bottle Roll Testwork

KCA received 13 drill interval samples on February 16, 2012 for preparation of ten separate bottle roll tests. The metallurgical testwork at KCA consisted of 120 hour bottle roll tests on seven individual samples as well as three composite samples.

The samples were first crushed in the lab and added to water to create a suitable slurry for testing. Sodium cyanide and hydrated lime were then added to the slurry to achieve 1.0 g/L NaCN at a pH between 10.5 and 11.0, additional reagents were added to maintain these values throughout the test period. The slurry was then agitated for two minutes every hour, with solution samples initially taken at two, four, eight, and 24 hours. After the initial 24 hours, samples were taken every 24 hours for four days.

Gold head grades for the ten samples ranged from 0.34 g/t to 1.4 g/t. Final soluble gold recoveries, after 120 hours, ranged from 38% to 73%, with no measurable correlation to head grade. The tests show that all of the samples have fast leaching kinetics, with over 60% of the total soluble recovery occurring in the first 24 hours. **Figure 13-1** shows the time vs recovery curve for each of the ten tests.

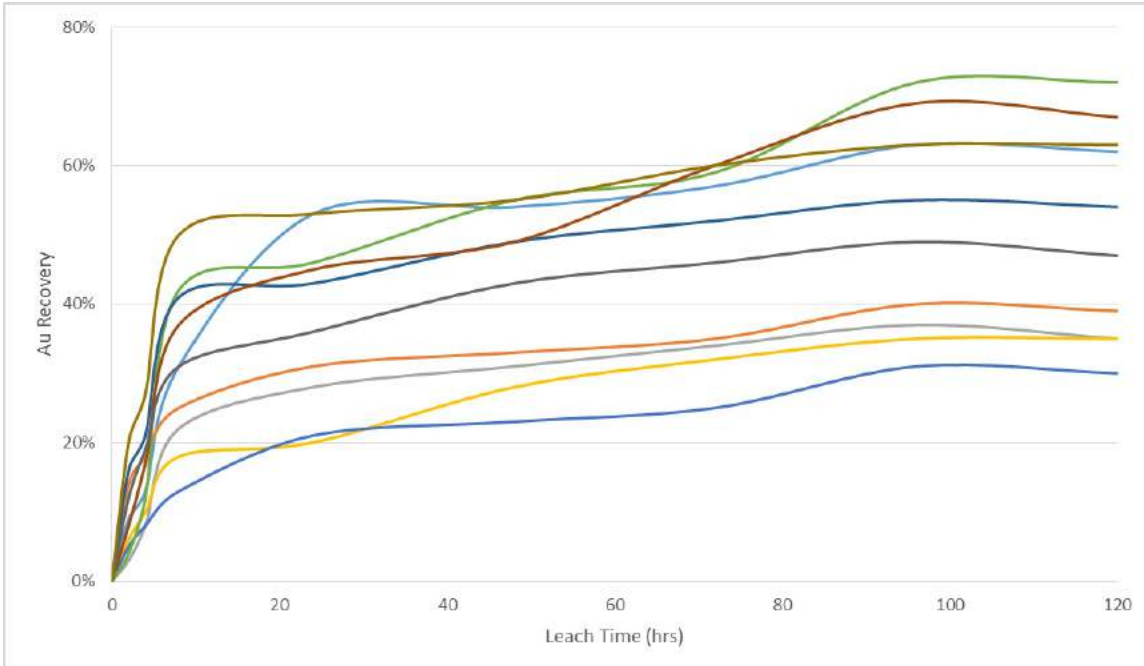


Figure 13-1: Gold Leaching Kinetics

13.2 SGS Process Flowsheet Testwork

SGS received 279 drill core samples that were composited into five different rock types: oxide, transition, hornfels sulfide, intrusive sulfide, and schist sulfide. All five composites were subjected to Bond Ball Mill Work Index and whole mineralized material cyanide leach testing. The four non-oxide composites were also subjected to additional sulfide recovery tests, including: whole mineralized material roasting, whole mineralized material pressure oxidation (POX), flotation, and flotation followed by pressure oxidation. A summary of the highest Gold recoveries is presented in **Table 13-1**.

Table 13-1: Summary of the Highest Leach Recoveries

Mineralized Material Type	Flowsheet	Gold Head Grade (g/t)	Gold Recovery (%)
Oxide	Whole Mineralized Material	0.94	89.3
	Coarse Mineralized Material	0.97	88.2
Transition	Whole Mineralized Material	0.66	75.6
	Coarse Mineralized Material	0.52	57.3
	Whole Mineralized Material POX	0.55	98.3
	Whole Mineralized Material Roast	0.57	85.4
	Flotation	0.66	74.8
	Flotation – POX	0.60	91.1
Hornfels Sulfide	Whole Mineralized Material	0.66	57.8
	Whole Mineralized Material POX	0.68	98.5
	Whole Mineralized Material Roast	0.63	81.5
	Flotation	0.78	57.0
	Flotation – POX	0.80	91.0
Intrusive Sulfide	Whole Mineralized Material	0.95	65.2
	Whole Mineralized Material POX	0.89	97.9
	Whole Mineralized Material Roast	0.94	84.0
	Flotation	1.02	66.6
	Flotation – POX	0.77	95.7
Schist Sulfide	Whole Mineralized Material	0.93	15.5
	Whole Mineralized Material POX	0.92	97.9
	Whole Mineralized Material Roast	1.13	68.4
	Flotation	0.91	14.1
	Flotation – POX	0.87	89.1

Results from process flowsheet testwork shows that the oxide and, to a lesser extent, the transition material are recoverable without any form of sulfide oxidation. Both the hornfels and intrusive sulfide material can be recovered with direct cyanidation, although at much lower recoveries. All of the sulfide containing material was shown to respond favorably to both POX and roasting.

13.2.1 Bond Ball Mill Work Index Testwork

All five composites were subjected to Bond Ball Mill Work Index testing. The composites were crushed to minus 6 mesh with the tests being conducted to a 150 mesh closing size. A summary of the test results is presented in **Table 13-2** indicating that the Project mineralized materials have a medium hardness.

Table 13-2: Bond Ball Mill Work Index

Mineralized Material Type	F_{80} (µm)	P_{80} (µm)	BWI (kWh/t)
Oxide	1484	81	12.5
Transition	1601	81	13.6
Hornfels Sulfide	1590	80	14.8
Intrusive Sulfide	844	77	13.7
Schist Sulfide	1485	79	12.8

13.2.2 Whole Mineralized Material Leaching

Whole mineralized material leaching testwork was performed on all five composites using standard bottle roll test procedures. The bottle roll tests were conducted for 48 hours at a range of target grind sizes, from P_{80} 20 µm to P_{80} 106 µm, with cyanide concentrations of 1.0 g/L.

Both the oxide and transition samples had recoveries that were slightly dependent on grind size. The oxide sample had gold recoveries between 85.2% at the coarsest grind to 89.3% at the finest grind. The transition sample had slightly lower gold recoveries than the oxide sample, recovering between 68.2% at the coarse size and 75.6% at the fine size.

The hornfels sulfide and intrusive sulfide samples had lower gold recoveries, with the hornfels sample recovery ranging between 47.9% and 57.8% and the intrusive sample recovery ranging from 57.8% to 65.2%. The schist sulfide sample had very low gold recoveries, ranging from 8.5% to 15.5%. All three sulfide composites were shown to have no measurable correlation between grind size and recovery at the tested grind sizes.

13.2.3 Whole Mineralized Material Pressure Oxidation and Leaching

Whole mineralized material POX testwork was performed on the four sulfide containing composites. Two samples from each of the sulfide containing composites were ground to P_{80} 75 µm and P_{80} 53 µm. All of the samples underwent 45 minutes of pre-acidification, to a pH of 2.0, prior to POX. The samples were then oxidized in an autoclave at 200°C with 100 psi of overpressure for 80 minutes. POX residue showed that over 97% of the sulfides in the samples were oxidized.

Residues of the POX tests were washed and neutralized prior to undergoing cyanidation bottle roll testing. Test parameters for the bottle roll tests were the same as those used in the whole mineralized material leaching testwork. The test results from the leaching show that gold recovery is insensitive to grind size in the ranges tested. Average gold recovery for the transition composite was 96.4%. The hornfels, intrusive, and schist sulfide samples had average gold recoveries of 97.1%, 97.2%, and 97.0%, respectively.

13.2.4 Whole Mineralized Material Roasting

Whole mineralized material roasting testwork was performed on the four sulfide containing composites. All of the samples were ground to P_{80} 75 µm and heated to 550°C for 90 minutes. The samples were then washed neutralized prior to leaching. Sulfide analysis on the roasted material showed that over 95% of the sulfides in the samples were oxidized.

The samples were then leached using the same standard bottle roll test procedures as the whole mineralized material leaching. All four samples showed increased gold recoveries compared to whole mineralized material leaching. The transition sample had the highest gold recovery, at 85.4%, an increase of approximately 15% compared to whole mineralized material leaching. The hornfels sample gold recovery increased to 81.5%, an increase of approximately 28% compared to whole mineralized material leaching. The gold recovery for the intrusive sample increased to 84.0%, an increase of approximately 25% compared to whole mineralized material leaching. The schist sample had the highest overall increase in gold recovery when compared to whole mineralized material leaching, an increase of approximately 57%, but had the lowest overall recovery, at 68.4%.

13.2.5 Sulfide Flotation & Leaching

Rougher kinetic flotation tests were performed on each of the four sulfide containing composites to determine flotation characteristics of the composites. Each composite had three tests performed at different grind sizes ranging between P_{80} 80 µm and P_{80} 130 µm. Copper sulfate was used to activate the sulfide minerals in the samples with potassium amyl xanthate (PAX) and Aero 407 being used as collectors. Gold recoveries into flotation concentrate are shown in **Table 13-3**.

Table 13-3: Flotation Concentrate Gold Recoveries

Composite Rock Type	Test #	Au Recovery (%)
Transition	R-04	85.2
Transition	R-08	88.1
Transition	R-12	95.9
Hornfels Sulfide	R-01	88.1
Hornfels Sulfide	R-05	83.9
Hornfels Sulfide	R-09	88.8
Intrusive Sulfide	R-02	92.8
Intrusive Sulfide	R-06	93.8
Intrusive Sulfide	R-10	96.1
Schist Sulfide	R-03	83.0
Schist Sulfide	R-07	91.4
Schist Sulfide	R-11	92.9

At the conclusion of the rougher kinetic tests, twelve batch flotation tests were performed to generate concentrate for downstream testing. The products from the twelve tests were combined to form composites for each of the four sulfide rock types.

Samples from each of the bulk flotation concentrates were ground for zero, 15, and 45 minutes and then subjected to leaching with a 5 g/L sodium cyanide solution. Gold recoveries for the transition sample averaged 74.8%. Gold recoveries for the hornfels, intrusive, and schist sulfide samples had recoveries averaging 58.0%, 69.0%, and 13.4%, respectively. These recoveries were similar to the recoveries seen in the whole mineralized material leaching testwork, indicating that oxidation of the sulfides is required to improve recoveries.

Additional cyanide leaching testwork was performed on flotation tailings to determine gold extractions of the tailings stream. Gold recoveries in the tailings streams ranged from 18.1% to 61.4%. The low recoveries reflect the low proportion of gold reporting to the flotation tailings.

13.2.6 Flotation Pressure Oxidation & Leaching

Flotation concentrate from the bulk flotation tests were subjected POX tests. Eight 80-minute POX tests were performed, two from each sulfide composite, utilizing an autoclave at 200°C and 100 psi oxygen overpressure. The residues from the POX tests indicated that sulfide oxidation was greater than 98% for all samples.

Residues of the POX tests were washed and neutralized prior to undergoing intense cyanidation bottle roll testing. Test parameters for the bottle roll tests were the same as those used in the flotation concentrate leaching testwork. Gold recoveries for the transition samples averaged 95.9%. Gold recoveries for the hornfels and schist Sulfide composites averaged 98.4% and 91.6%, respectively. One of the cyanidation tests performed on the intrusive sulfide composite achieved a gold recovery 83.8%. This result was likely erroneous due to poor solution chemistry. The second test performed on the intrusive sulfide composite achieved a much higher gold recovery of 97.1%.

13.2.7 Coarse Mineralized Material Cyanidation

Four coarse mineralized material bottle roll tests, two on each of the oxide and transition composites, were conducted to examine the sensitivity of gold recoveries to particle size. The samples were crushed to minus 6 mesh prior before the material was added to a 5 g/L sodium cyanide leach solution. The bottle roll tests were conducted by rotating the bottles for one minute every hour. Solution samples were taken at the two, six, and 24 hour marks and every 24 hours after, until the 120 hour mark.

The leaching kinetics for both of the samples were very fast, with greater than 95% of the total gold recoveries occurring in the first 24 hours. Overall gold recoveries for the oxide sample averaged 88.1%, only one percent lower than the best result from the whole mineralized material testwork ground to P_{80} 50 µm. The transition sample did not perform as well as the oxide sample when compared to the whole mineralized material testwork. The transition samples only achieved 57.3% gold recovery, compared to the 75.6% achieved for the whole mineralized material testwork ground to P_{80} 50 µm.

13.3 MCCLELLAND TESTWORK

Metallurgical testwork was performed on four drill core composites of different mineralogy from the Project. The different composites were designated as oxide, transition, intrusive sulfide, and hornfels sulfide. These composites were initially subjected to coarse bottle roll tests conducted at five different feed sizes. Due to poor recoveries on the non-oxide composites, additional bottle roll tests were performed at the finer grind sizes in attempt to increase recoveries.

One column leach test was performed on the crushed oxide composite to determine heap leaching characteristics of the material.

13.3.1 Bottle Roll Testwork

Bottle roll testwork was performed on four composites using standard bottle roll test procedures. The first set of bottle roll tests were ran for 120 hours, agitating for one minute every hour. Target grind size ranged from P_{80} 25 mm to P_{80} 1.7 mm, with cyanide concentrations of 1.0 g/L.

The oxide sample had gold recoveries between 77.2% and 81.3%. Grind size did not appear to have an appreciable effect on gold recoveries at the sizes tested. The transition sample had gold recoveries between 21.5% and 40.4%. Similar to the oxide sample, the grind size did not appear to have an appreciable effect on gold recoveries between 25 mm and 6.3 mm, as all four tests had recoveries between 21.5% and 29.4%. Grind size did appear to have an effect when going from 6.3 mm to 1.7 mm as gold recovery improved to 40.4%.

Both the intrusive sulfide and hornfels sulfide samples had low gold recoveries, with the intrusive sample recovery ranging between 17.9% and 41.5% and the hornfels sample recovery ranging from 12.3% to 27.9%. Finer grind sizes appeared to have a positive effect on recoveries. Recoveries increased at each finer grind size with the exception of the coarsest hornfels sample.

Due to the low recoveries achieved on the transition, hornfels, and intrusive samples, additional bottle roll tests were performed at P_{80} 212 µm and P_{80} 75 µm. The test procedures for the additional bottle rolls differed from the previous tests by decreasing the leach time to 96 hours and increasing the cyanide concentration to 5 g/L. All three samples had higher recoveries than the previous tests. Gold recoveries ranged from 57.9% to 65.8% in the transition sample, 54.7% to 63.9% in the intrusive sample, and 44.2% to 53.3% in the hornfels sample. Grind size did not appear to have an effect on recoveries between 212 µm to 75 µm.

Table 13-4 summarizes the gold recoveries for all bottle roll tests.

Table 13-4: Bottle Roll Test Results

Composite	Feed Size	Leach Time (hr)	NaCN Conc. (g/L)	Au Recovery (%)
Oxide	25 mm	5	1.00	79.8
Oxide	19 mm	5	1.00	79.2
Oxide	12.5 mm	5	1.00	77.8
Oxide	6.3 mm	5	1.00	77.2
Oxide	1.7 mm	5	1.00	81.3
Transition	25 mm	5	1.00	21.5
Transition	19 mm	5	1.00	29.4
Transition	12.5 mm	5	1.00	25.9
Transition	6.3 mm	5	1.00	26.7
Transition	1.7 mm	5	1.00	40.4
Transition	1.7 mm	5	1.00	36.6
Transition	1.7 mm	5	5.00	34.9
Transition	212 µm	4	5.00	65.8
Transition	212 µm	4	5.00	57.9
Transition	75 µm	4	5.00	57.8
Intrusive Sulfide	25 mm	5	1.00	17.9
Intrusive Sulfide	19 mm	5	1.00	25.3
Intrusive Sulfide	12.5 mm	5	1.00	29.7
Intrusive Sulfide	6.3 mm	5	1.00	31.9
Intrusive Sulfide	1.7 mm	5	1.00	41.5
Intrusive Sulfide	1.7 mm	5	1.00	36.4
Intrusive Sulfide	1.7 mm	5	5.00	39.5
Intrusive Sulfide	212 µm	4	5.00	63.9
Intrusive Sulfide	212 µm	4	5.00	54.7
Intrusive Sulfide	75 µm	4	5.00	60.2
Hornfels Sulfide	25 mm	5	1.00	23.6
Hornfels Sulfide	19 mm	5	1.00	12.3
Hornfels Sulfide	12.5 mm	5	1.00	15.4
Hornfels Sulfide	6.3 mm	5	1.00	18.9
Hornfels Sulfide	1.7 mm	5	1.00	26.5
Hornfels Sulfide	1.7 mm	4	1.00	27.9
Hornfels Sulfide	1.7 mm	4	5.00	26.7
Hornfels Sulfide	212 µm	4	5.00	47.8
Hornfels Sulfide	212 µm	4	5.00	44.2
Hornfels Sulfide	75 µm	4	5.00	53.3

13.3.2 Column Leach Testwork

Column leach testwork was performed on the oxide composite in a 15 cm diameter by 3 m high column. The material, crushed to a P_{80} 25 mm, was loaded into the column and subjected to cyanidation using a cyanide solution of 1.0 g/L sodium cyanide. The cyanide solution was applied at a rate of 12 Lph/m^2 with solution samples being collected every 24 hours for analysis. The total overall leach cycle for the test was 55 days, which included a 34 day primary leach cycle followed by a 14 day rest cycle and an additional 7 day secondary leach cycle. The leach cycle was followed by a nine day rinse cycle and a 10 drain-down test.

The test showed that the oxide composite had extremely fast leaching kinetics, achieving greater than an 80% gold recovery in 11 days with a total gold recovery of 87%. The gold recovery curve for the tests is presented in **Figure 13-2**.

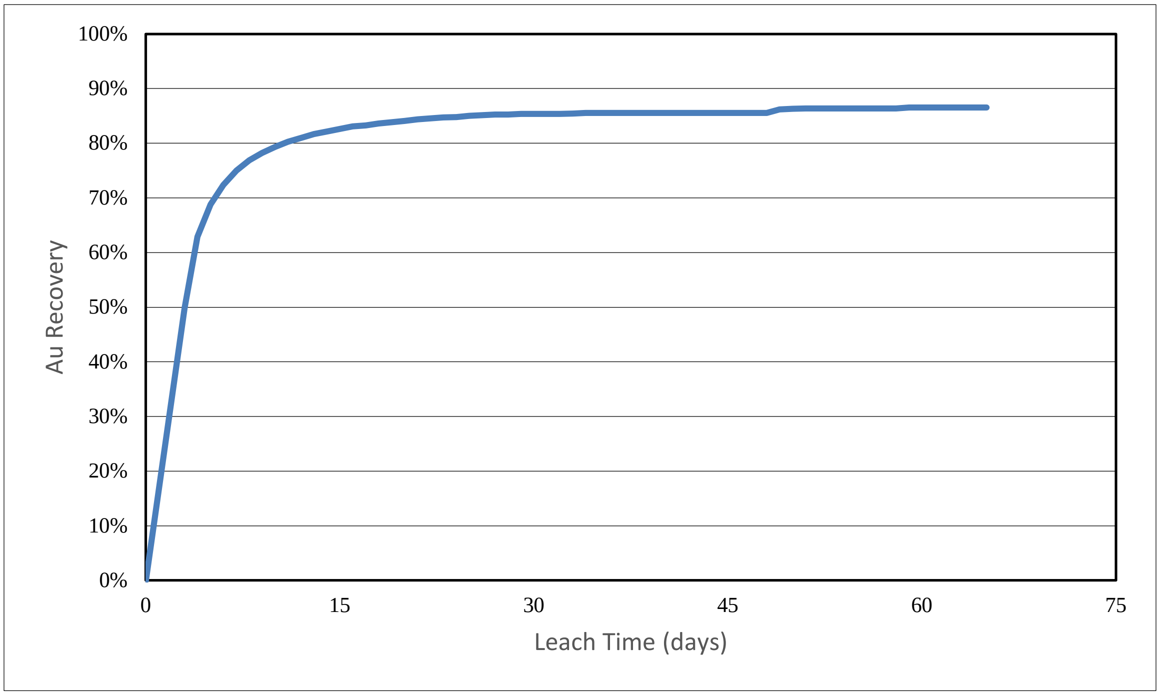


Figure 13-2: Gold Recovery Curve

14.0 MINERAL RESOURCE ESTIMATES

Freegold contracted Giroux Consultants to update the gold resource present on the Project. Gary Giroux was the Qualified Person responsible for the resource estimate. Mr. Giroux is a Qualified Person based on education, experience and his membership in a professional organization; criteria set out in NI 43-101. Mr. Giroux is also independent of Freegold.

This update of the NI 43-101 resource reported in December 2012 (Abrams and Giroux, 2012) was based on an additional 10 drillholes completed in 2013 and subdivides the resource into an oxide and sulfide portion. The effective date for this resource is May 31, 2013, the date that the data was received. There were 3 drillholes completed since this date which would not have a material effect on this resource and as result this resource remains current. The 3 new holes are compared to the estimated blocks they pass through in section 14.8 Model Verification.

14.1 DATA ANALYSIS

The data provided by Freegold consisted of 330 drillhole collars and 43,581 gold assays extending across the Property. Gold assays reported as less than the detection limit were replaced by a value of 0.5 that detection limit. Gold values reported as zero parts per billion (ppb) were also set to one ppb. A total of 306 gaps in the from-to record were found and values of one ppb Au were inserted to fill these gaps.

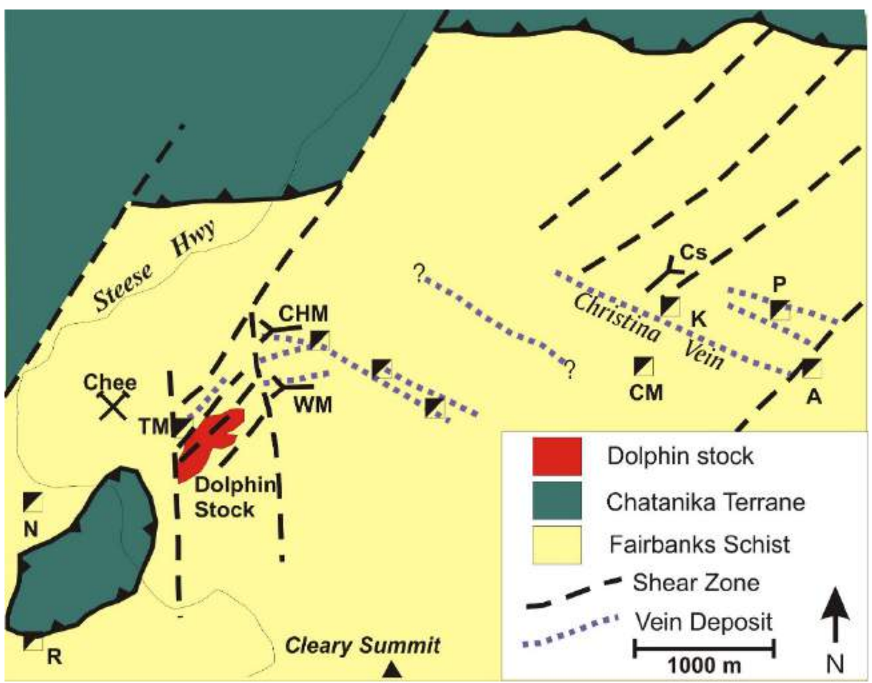


Figure 14-1: Local Geology of the Dolphin Stock Area (Adams, 2010)

The Dolphin stock is a multi-phase intrusive located on the ridge between Willow Creek and Bedrock Creek. The stock has been traced on surface by soil sampling and RC drill data and represents an area of 1,200 ft. by 2,000 ft. (366 x 610 m).

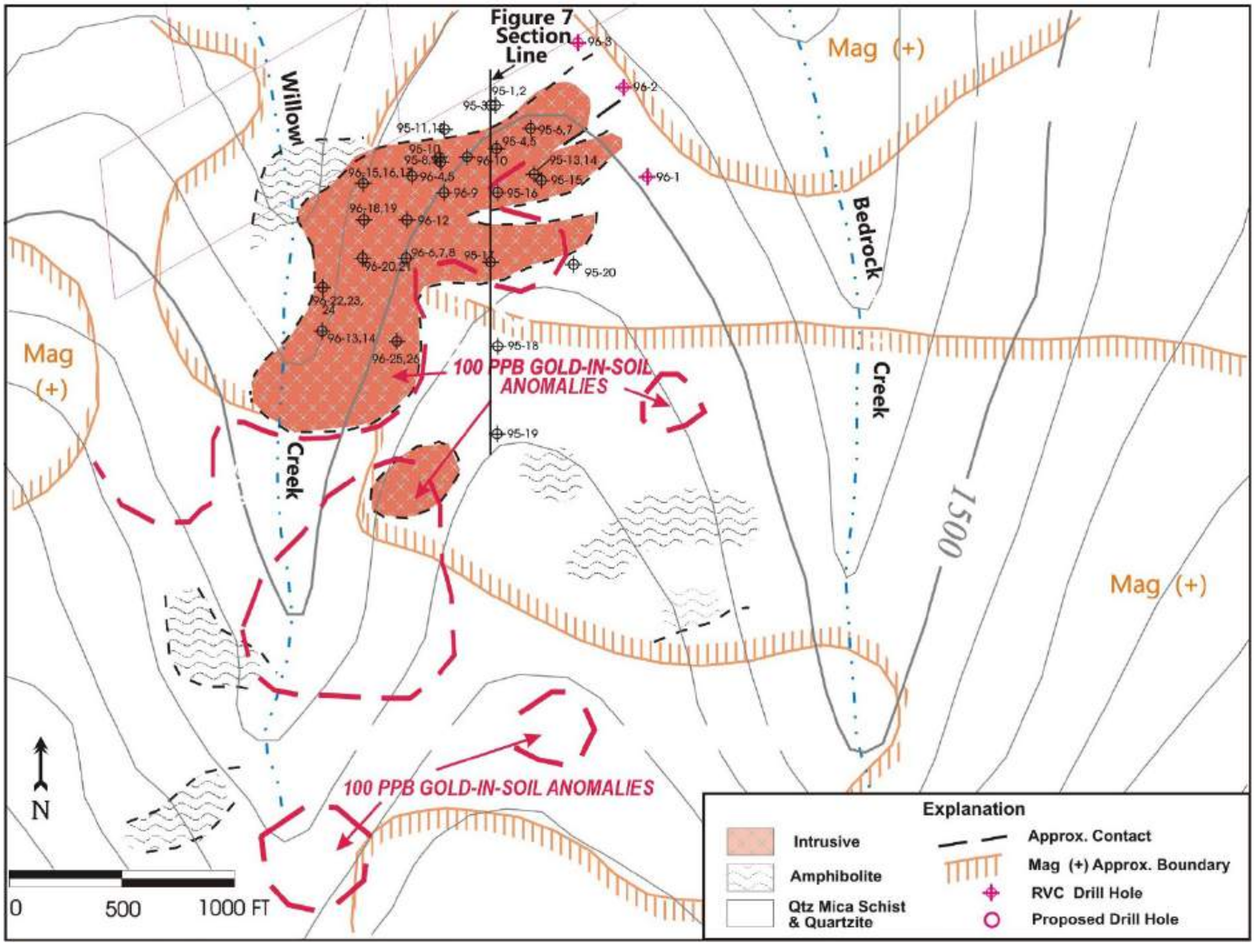


Figure 14-2: Dolphin Stock Area Geologic Map, Gold-Arsenic Soil Anomalies, Aeromagnetic Anomaly and Drillholes (Adams, 2010)

A three-dimensional mineralized solid was provided by Freegold to constrain the Dolphin Stock Zone Resource estimate.

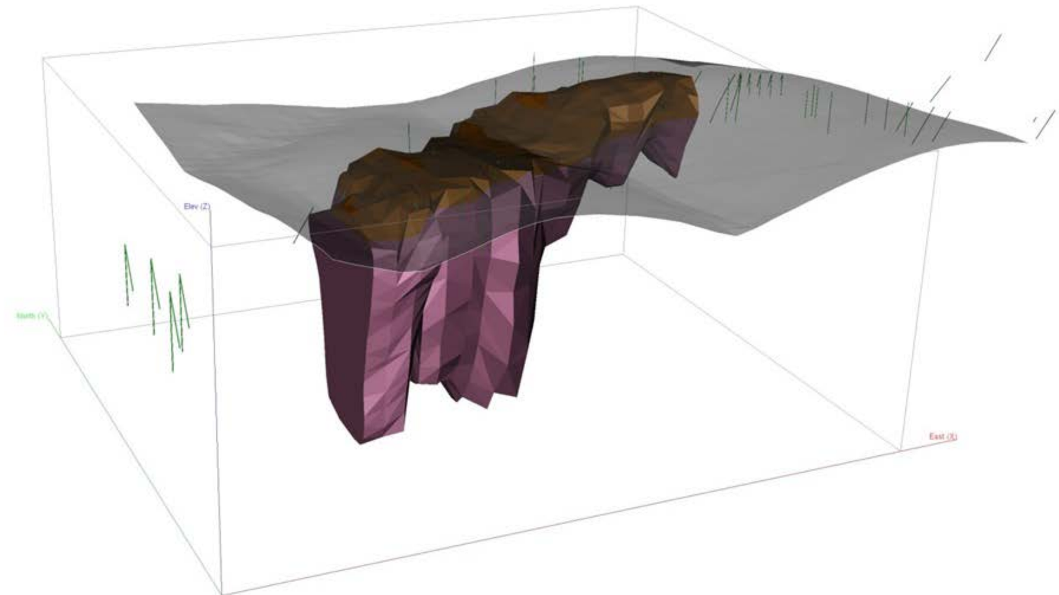


Figure 14-3: Isometric View Looking NE Showing the Mineralized Solid Purple, Oxides in Brown, Drillhole Traces and Surface Topography

Drillholes were "passed through" these solids with the point each hole entered and left the solid recorded. Individual assays were then tagged with a code of mineralized if inside solid and below oxide surface, oxide if inside the mineralized solid and above the oxide surface and waste if outside the mineralized solid. Of the supplied drillhole data, 185 drillholes were drilled in the mineralized Dolphin Stock totaling 39,301 m. Note that of the ten new drillholes provided for this update three were drilled from the same collar. Holes GSDL1307 and GSDL1308 were not used in the estimate as they were replaced by GSDL1309 which was drilled deeper.

To compare samples above and below the oxide surface the distribution of gold grades was examined using a lognormal cumulative frequency plot (**Figure 14-4**). The distributions of grade are almost identical with no differences shown that would indicate remobilization at the contact. As a result all assays were combined for estimation purposes.

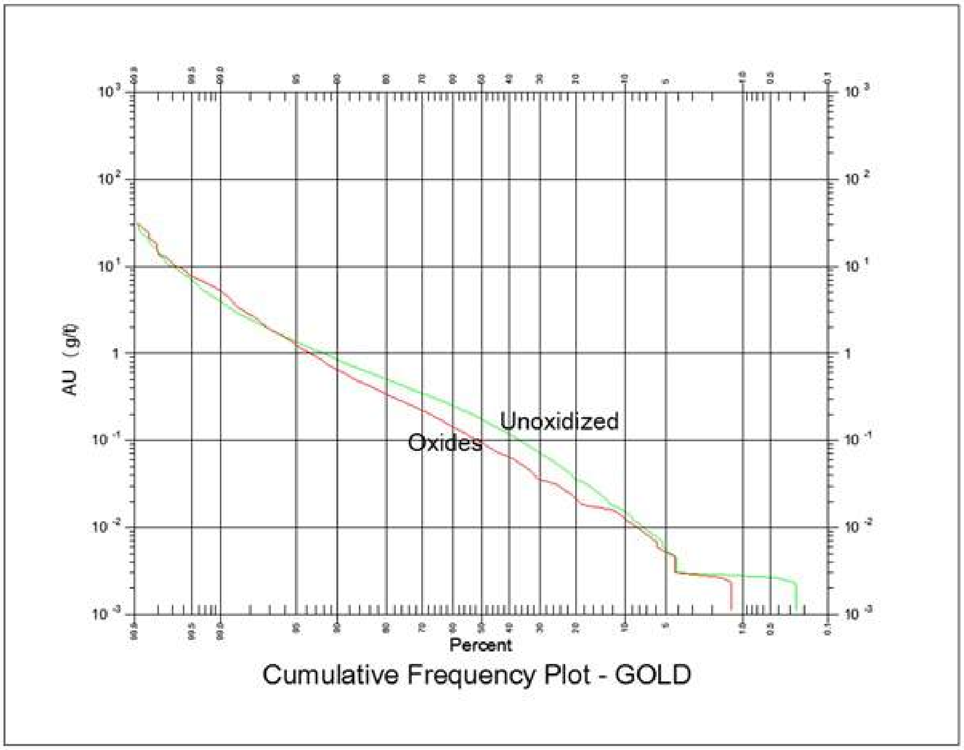


Figure 14-4: Cumulative Frequency Plot for Gold in Oxidized and Unoxidized Assays

The gold distribution, within the mineralized solid, was examined using a lognormal cumulative frequency plot to determine if capping was required and if so at what level. The procedure used is explained in a paper by Dr. A.J. Sinclair titled Applications of probability graphs in mineral exploration (Sinclair, 1976). In short the cumulative distribution of a single normal distribution will plot as a straight line on probability paper while a single lognormal distribution will plot as a straight line on lognormal probability paper. Overlapping populations will plot as curves separated by inflection points. Sinclair proposed a method of separating out these overlapping populations using a technique called partitioning. In 1993 a computer program called P-RES was made available to partition probability plots interactively on a computer (Bentzen and Sinclair, 1993). A screen dump from this program is shown for gold in **Figure 14-5**. On this plot the actual gold distribution is shown as black dots. The inflection points that separate the populations are shown as vertical lines and each population is shown by the straight lines of open circles. The interpretation is tested by recombining the data in the proportions selected and this test is shown as triangles compared to the original distribution.

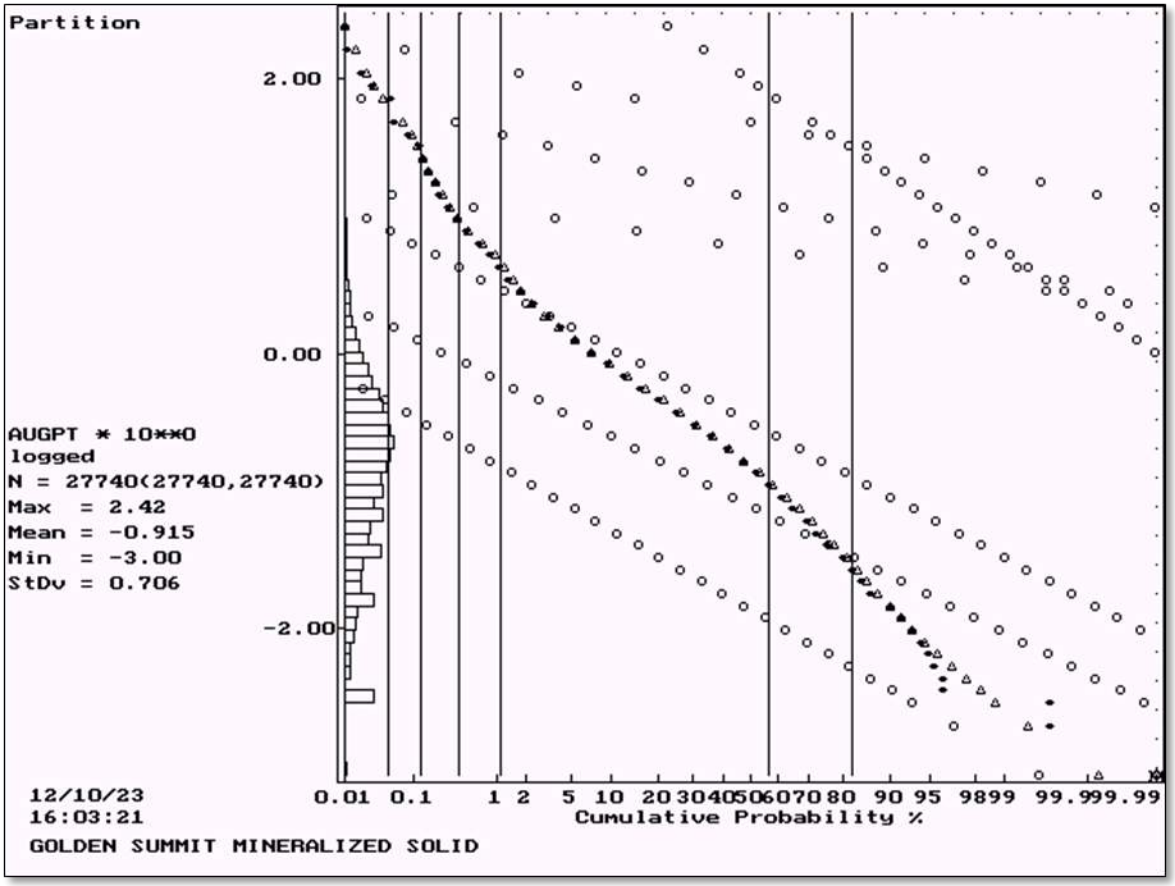


Figure 14-5: Lognormal Cumulative Frequency Plot for Gold Assays within Mineralized Solids.

A total of seven over-lapping lognormal populations are indicated (**Table 14-1**).

Table 14-1: Gold Populations Present within Mineralized Solid

Population	Mean Au (g/t)	Percentage of Total	Number of Assays
1	97.61	0.04 %	11
2	48.84	0.08 %	23
3	13.52	0.25 %	70
4	5.91	0.72 %	200
5	0.31	55.49 %	15,394
6	0.08	25.78 %	7,151
7	0.01	17.63 %	4,891

Population 1 represents erratic outlier grades and should be capped. An effective cap would be two standard deviations above the mean of Population 2, a value of 88 g/t Au. A total of seven assays were capped at 88 g/t Au. Populations 2, 3 and 4 might represent shear zone mineralization thought to strike to the north east and dip 40 to 50° to the northwest. Population 5 might represent the earlier stockwork style mineralization. Populations 6 and 7 could represent post mineral dykes and internal waste. Since there is insufficient data to model the higher grade shear zones an indicator approach was used.

Table 14-2: Statistics for Gold within the Mineralized Solid

Description	Assay Au (g/t)	Capped Au (g/t)
Number of Assays	30,152	30,152
Mean Au (g/t)	0.457	0.442
Standard Deviation	3.098	2.285
Minimum Value	0.001	0.001
Maximum Value	264.0	88.0
Coefficient of Variation	6.78	5.17

14.2 COMPOSITES

Uniform downhole three m composites were formed that honored the mineralized solid boundaries. Intervals less than 1.5 m at the boundary of the solid were combined with the adjoining sample to produce a composite file of uniform support, 3 ± 1.5 m in length. The statistics for three m composites are shown below.

Table 14-3: Statistics for Gold in Three m Composites within the Mineralized Solid

Description	Au (g/t)
Number of Composites	12,787
Mean Au (g/t)	0.417
Standard Deviation	1.256
Minimum Value	0.001
Maximum Value	52.47
Coefficient of Variation	3.01

A lognormal cumulative probability plot was again used to evaluate the mineralized populations within three m composites. **Figure 14-5** shows seven overlapping lognormal populations with the erratic outlier population gone after capping.

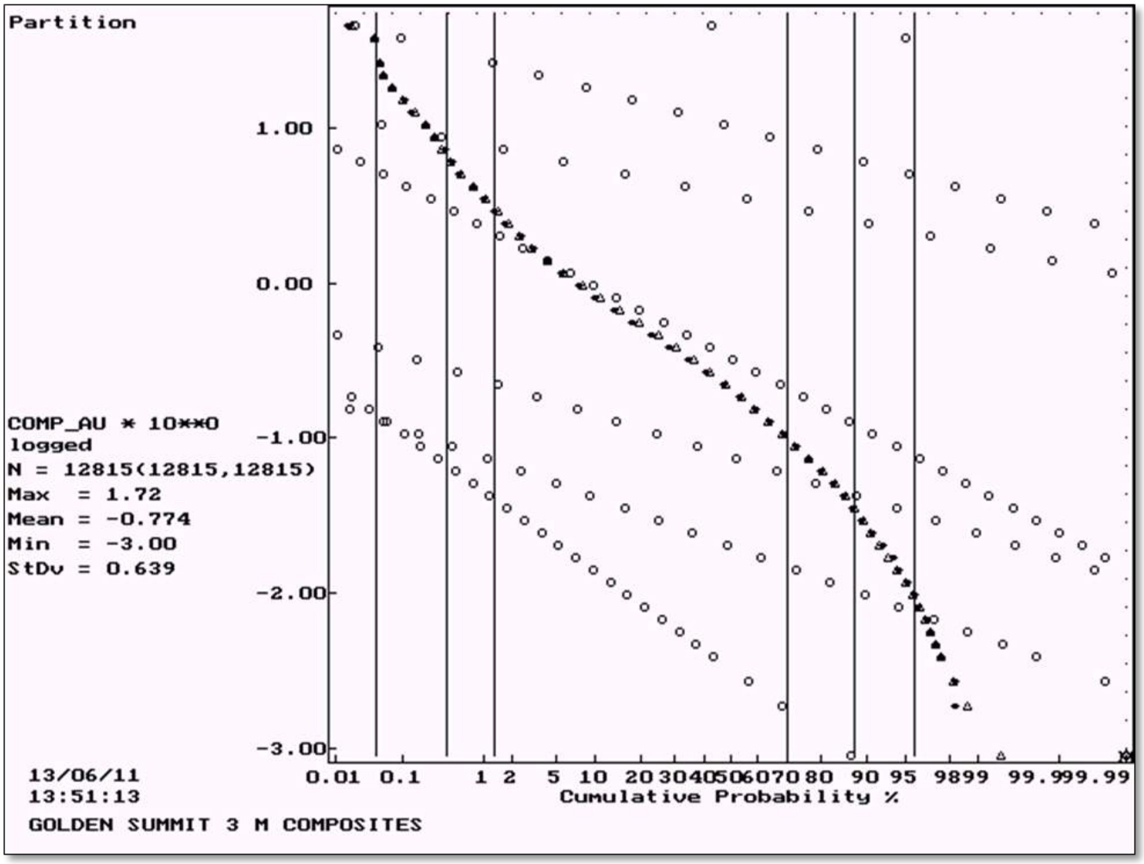


Figure 14-6: Lognormal Cumulative Frequency Plot for Gold Three m Composites within Mineralized Solids

Table 14-4: Gold Populations Three m Composites within Mineralized Solid

Population	Mean Au (g/t)	Percentage of Total	Number of Assays
1	45.00	0.04 %	5
2	10.25	0.33 %	42
3	3.67	0.93 %	119
4	0.33	68.93 %	8,833
5	0.07	17.50 %	2,243
6	0.02	8.05 %	1,032
7	0.003	4.21 %	541

Populations 1 to 3 might represent the higher grade shear hosted gold mineralization while Population 4 might represent the more pervasive stockwork style gold. Populations 5, 6 and 7 would represent post mineral dykes and other internal waste. A threshold that would separate Populations 1 to 3 from Population 4 would be two standard deviations above the mean of Population 4, a value of 1.0 g/t Au.

An indicator approach to modelling these two styles of mineralization would set up a single indicator variable for each composite. The indicator would be defined as follows.

Au IND = 0 if Au < 1.0 g/t Au (stockwork style mineralization)

Au IND = 1 if Au ≥ 1.0 g/t Au (shear zone mineralization)

In this manner the data base is reduced to zeros and ones for modelling.

14.3 Variography

Pairwise relative semivariograms were produced for gold in the low grade stockwork data (Au < 1.0 g/t) and for the higher grade shear zone indicator variable for composites with Au ≥ 1.0 g/t . The longest range and therefore best continuity within the stockwork mineralization was 120 m along azimuth 68°. The longest range for the higher grade shear zone indicator variable was 100 m along azimuth 90°. In all cases geometric anisotropy was demonstrated with nested spherical models fit to the data. The semivariogram parameters are tabulated below.

Table 14-5: Semivariogram Parameters

Variable	Az/Dip	Co	C1	C2	Short Range (m)	Long Range (m)
Au in LG	68 / 0	0.20	0.32	0.13	12.0	120.0
	158 / -73	0.20	0.32	0.13	50.0	110.0
	338/ -17	0.20	0.32	0.13	15.0	40.0
HG IND	90 / 0	1.40	0.31	0.19	30.0	100.0
	0 / -85	1.40	0.31	0.19	10.0	120.0
	180 / -5	1.40	0.31	0.19	12.0	30.0

14.4 Block Model

A block model containing blocks 10 x 10 x 5 m in dimension was superimposed over the Dolphin mineralized solid with the percentage of each block below surface topography and within the solid recorded. In addition the proportion of each block lying above the oxide surface was recorded. The block model origin is shown below.

Lower Left Corner		
Easting 478700 E	Column size = 10 m	145 Columns
Northing 7214700 N	Row size = 10 m	130 Rows
Top of Model		
Elevation 590	Level size = 5 m	155 Levels

No Rotation

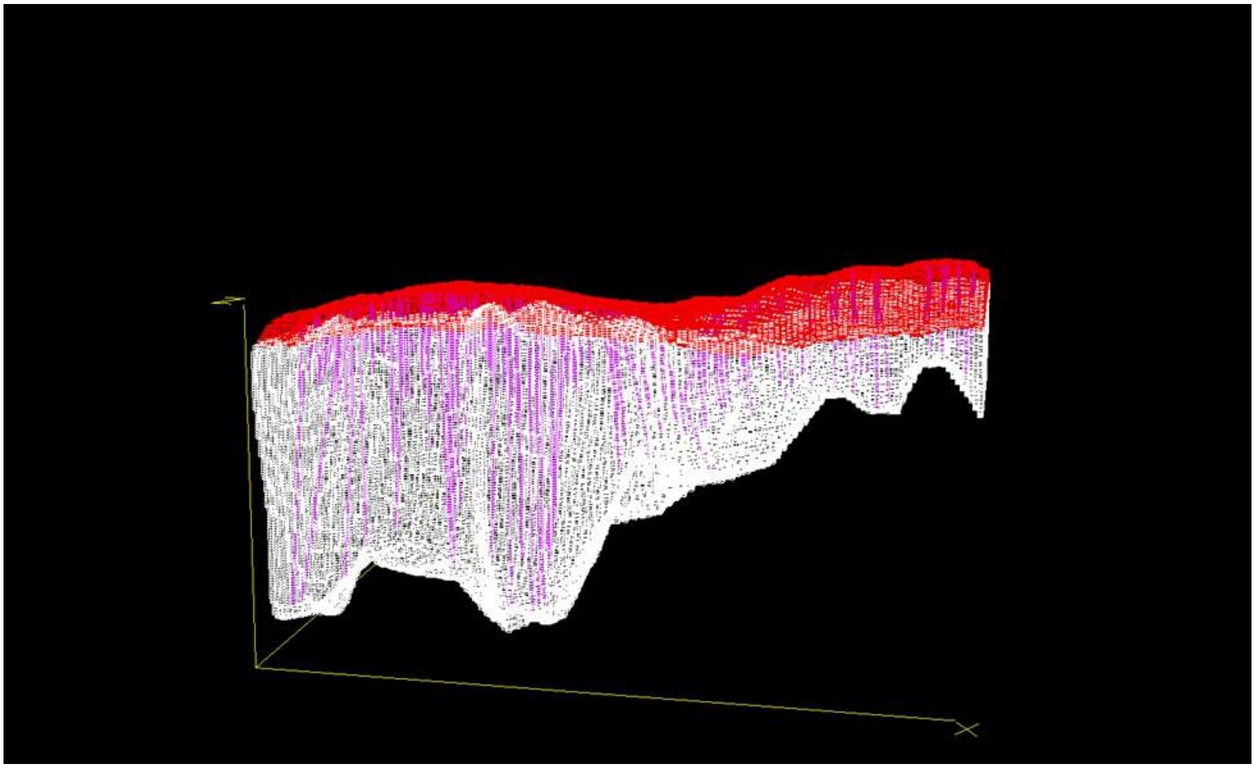


Figure 14-7: Isometric View of Block Model Looking N Showing Oxides Red, Mineralized Solid in White and Drillhole Traces in Purple

14.5 Bulk Density

A total of seven specific gravity (SG) determinations, using the weight in air/ weight in water methodology, were made in 2011 from drill core in holes GSDC1127 and GSDC1128. An additional 23 determinations were completed in 2011 from holes GSDC1128 to GSDC1131. In 2012 an additional 37 measurements were made. When the single measurement in massive sulfide is ignored, the other 66 had an average SG of 2.67.

Table 14-6: Specific Gravity Determinations Dolphin

Hole Number	Depth (ft)	Dry Weight Avg. (g)	Wet Weight Avg. (g)	SG	Rock Type
GSDC 1127	270.50	227.70	186.00	5.46	massive sulfide
GSDC 1130	594.00	391.40	231.70	2.45	AGRD
GSDC 1174	790.50	777.33	495.67	2.76	BqzS
GSDC 1174	802.00	1681.33	1062.33	2.72	BqzS
GSCH1205	637.00	1368.33	854.33	2.66	BqzS
GSCL1207	558.30	909.00	578.33	2.75	BqzS
GSDL1220	296.00	947.67	598.00	2.71	BqzS
GSDC 1176	590.60	522.00	328.00	2.69	CarbS
GSDC1165	390.60	707.00	440.00	2.65	CarbS
GSDC1167	91.00	690.67	403.00	2.40	CarbS
GSDC1169	396.50	567.00	356.00	2.69	CarbS

Hole Number	Depth (ft)	Dry Weight Avg. (g)	Wet Weight Avg. (g)	SG	Rock Type
GSCL1207	40.00	704.00	419.00	2.47	CarbS
GSDC 1130	545.00	486.30	294.70	2.54	CHL-GRD
GSDC 1176	509.00	730.00	455.33	2.66	ChlS
GSDC1165	123.40	521.00	326.00	2.67	ChlS
GSDC1165	968.00	566.00	361.67	2.77	ChlS
GSDC 1131	496.00	397.90	233.60	2.42	DAC PORPH
GSCL1202	777.50	1606.00	1029.00	2.78	Eco
GSCL1202	776.50	781.00	500.33	2.78	Eco
GSDC 1127	284.00	192.50	114.60	2.47	GRD
GSDC 1127	298.00	547.50	343.50	2.68	GRD
GSDC 1127	641.00	182.65	115.50	2.72	GRD
GSDC 1128	348.50	573.90	419.50	3.72	GRD
GSDC 1128	282.00	234.50	135.20	2.36	GRD
GSDC 1128	332.50	435.70	274.40	2.70	GRD
GSDC 1128	439.00	440.50	267.20	2.54	GRD
GSDC 1128	493.00	524.00	326.00	2.65	GRD
GSDC 1128	512.50	529.50	327.80	2.63	GRD
GSDC 1128	522.00	409.00	256.00	2.67	GRD
GSDC 1128	531.00	384.80	240.50	2.67	GRD
GSDC 1128	557.50	224.90	138.00	2.59	GRD
GSDC 1128	576.00	410.00	257.00	2.68	GRD
GSDC 1128	582.00	473.00	296.50	2.68	GRD
GSDC 1128	584.00	134.20	79.50	2.45	GRD
GSDC 1128	621.00	297.80	178.70	2.50	GRD
GSDC 1128	643.00	164.00	101.80	2.64	GRD
GSDC 1129	13.50	398.90	240.20	2.51	GRD
GSDC 1130	271.00	479.60	292.00	2.56	GRD
GSDL1211	155.00	1475.00	961.00	2.87	GRD
GSDL1220	224.00	746.00	464.67	2.65	GRD
GSDL1220	361.00	912.33	573.00	2.69	GRD
GSDL1220	411.00	670.00	423.00	2.71	GRD
GSDL1222	137.00	1177.00	709.00	2.51	GRD
GSDC1165	90.60	738.67	479.67	2.85	GS
GSDC1165	113.50	743.00	489.00	2.93	GS
GSDC1167	12.80	669.00	432.33	2.83	GS
GSDC1168	39.50	1138.33	742.33	2.87	GS
GSCL1212	122.00	689.00	437.67	2.74	HFS
GSDC1165	880.70	920.00	580.67	2.71	Mar
GSDC1165	886.00	884.00	526.00	2.47	Mar
GSDC1165	205.80	1020.33	633.67	2.64	QmiS
GSDC1167	95.50	507.00	309.67	2.57	QmiS
GSDC 1130	644.00	418.40	258.00	2.61	RHY PORPH

Hole Number	Depth (ft)	Dry Weight Avg. (g)	Wet Weight Avg. (g)	SG	Rock Type
GSDC1165	940.80	468.00	295.00	2.71	Sch
GSDC1165	954.20	727.00	448.33	2.61	Sch
GSDC1167	499.20	681.00	427.33	2.68	Sch
GSDC 1130	620.00	318.10	198.90	2.67	SGRD
GSDC 1131	528.00	424.50	263.90	2.64	SGRD
GSDC 1131	636.00	301.70	186.10	2.61	SGRD
GSDC 1127	651.50	179.30	111.60	2.65	TON
GSDC 1128	321.00	511.70	308.30	2.52	TON
GSDL1211	528.00	743.33	467.33	2.69	TON
GSDL1211	1068.50	995.00	630.00	2.73	TON
GSDL1212	777.00	1016.67	646.33	2.75	TON
GSDL1213	1661.50	991.00	635.33	2.79	TON
GSDL1220	533.00	1065.00	672.33	2.71	TON
GSDL1220	585.50	1066.33	673.67	2.72	TON
GSDC 1131	332.50	435.70	274.40	2.70	
Total = 66				2.67	

The relationship between SG and gold grade was examined by averaging the SG over a series of gold grade ranges in **Table 14-7**.

Table 14-7: Specific Gravity Sorted by Gold Grades

Au Grade Range (g/t)	Average Au (g/t)	Number of Samples	Minimum SG	Maximum SG	Average SG
0.001 – 0.01	0.005	11	2.47	2.93	2.74
0.01 – 0.05	0.027	13	2.40	2.78	2.63
0.05 – 0.10	0.063	10	2.51	2.87	2.69
0.10 – 0.50	0.228	22	2.45	2.79	2.63
> 0.5	1.086	11	2.36	3.72	2.70
TOTAL		67			2.67

Based on the samples to date there appears to be no correlation between gold grades and SG. When the oxide surface is considered the results can be subdivided as follows:

Table 14-8: Specific Gravity Sorted by Oxidation State

Oxidation State	Number of Samples	Minimum SG	Maximum SG	Average SG
Oxides	14	2.23	2.61	2.51
Sulfides	49	2.36	3.72	2.66
Waste	6	2.66	2.77	2.71

As a result an SG of 2.51 was used for oxide material while the average of 2.67 was applied to all blocks below the oxide surface. This is an increase from the average of 2.63 used in the 2011 estimate (Adams and Giroux, 2012).

During future drill campaigns every effort should be made to further quantify the SG value of oxide material.

14.6 Grade Interpolation

Grades for the lower grade stockwork style mineralization were first interpolated into blocks using only composites < 1.0 g/t Au. The interpolation was done by ordinary kriging in four passes. The first pass used a search ellipse with dimensions equal to 0.25 the semivariogram range for low grade Au. A minimum of four composites (from composites within the mineralized solid but less than 1.0 g/t Au), were required to estimate the block. For blocks not estimated in pass one a second pass using dimensions equal to 0.5 the semivariogram range was attempted. Again a minimum of four composites were required to make an estimate. For blocks not estimated a third pass using the full range and a fourth pass using twice the range completed the estimation process. In all passes a maximum of 12 composites were used with a maximum of three coming from any single drillhole. This exercise determined a grade for the low grade (stockwork) portion of the block.

A second kriging exercise was then completed estimating the high grade indicator or the probability of finding high grade within any given block. This estimation was completed using the zero or one indicator value for composites within the mineralized solid and resulted in a value between zero and one. Again ordinary kriging was used in a series of four passes with the search ellipse dimensions for each pass a function of the high grade indicator semivariogram.

Finally, for blocks with a kriged indicator value greater than zero, a high grade gold value was estimated from composites within the mineralized solid greater than or equal to 1.0 g/t Au. A similar four pass estimate was made with the search ellipse dimensions a function of the high grade gold indicator variogram. Blocks estimated for low grade Au but not estimated for HG IND were not included.

The final grade for each block was a weighted average of the two styles of mineralization.

Au Total = (LG Au * (1.0 – IND)) + (HG Au * IND)

Where:

- Au Total is the weighted average grade for the block;
- LG Au is the grade of the stockwork or low grade portion of block;
- HG Au is the grade for the shear zone or high grade portion of block; and
- IND is the probability between zero and one that high grade exists in the block.

The search parameters for the various kriging runs are tabulated below.

Table 14-9: Kriging Parameters

Variable	Pass	Number Estimated	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)
LG Au	1	17,888	68 / 0	30.0	158 / -73	27.5	338 / -17	10.0
	2	108,982	68 / 0	60.0	158 / -73	55.0	338 / -17	20.0
	3	214,728	68 / 0	120.0	158 / -73	110.0	338 / -17	40.0
	4	89,022	68 / 0	240.0	158 / -73	220.0	338 / -17	80.0
HG IND	1	7,239	90 / 0	25.0	0 / -85	30.0	180 / -5	7.5
	2	61,691	90 / 0	50.0	0 / -85	60.0	180 / -5	15.0
	3	193,861	90 / 0	100.0	0 / -85	120.0	180 / -5	30.0
	4	167,829	90 / 0	200.0	0 / -85	240.0	180 / -5	60.0
HG Au	1	408	90 / 0	25.0	0 / -85	30.0	180 / -5	7.5
	2	8,834	90 / 0	50.0	0 / -85	60.0	180 / -5	15.0
	3	66,706	90 / 0	100.0	0 / -85	120.0	180 / -5	30.0
	4	105,365	90 / 0	200.0	0 / -85	240.0	180 / -5	60.0

14.7 CLASSIFICATION

Based on the study herein reported, delineated gold mineralization of the Dolphin Zone at the Project is classified as a resource according to the following definitions from CIM NI 43-101:

> “In this Instrument, the terms "Mineral Resource", "Inferred Mineral Resource", "Indicated Mineral Resource" and "Measured Mineral Resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards (May 2014) on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended.”

The terms *Measured*, *Indicated* and *Inferred* are defined by CIM as follows:

> "A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."
>
> "The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing. Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time."
>
> **Inferred Mineral Resource**
>
> "An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."
>
> "An 'Inferred Mineral Resource' is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drillholes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101."
>
> "There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure

of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource."

Indicated Mineral Resource

"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve."

"Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions."

Measured Mineral Resource

"A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve."

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

Modifying Factors

> "Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors."

For the mineralized Dolphin zone the geological continuity has been established though surface mapping and diamond drillhole interpretation. Grade continuity can be quantified by semivariogram analysis. Blocks estimated in Pass 1 or Pass 2, using up to 0.5 the semivariogram range, during the low grade gold estimation, were classified as Indicated. All other blocks were classified as Inferred.

The results are tabulated (**Table 14-10** and **Table 14-11**) below assuming one could mine to the limits of the mineralized solids. At the time this resource was estimated (2013), no economic analysis had been completed for the Dolphin zone, and as a result the economic cut-off was unknown. In the author's judgement and experience the resource stated has reasonable prospects of economic extraction. The nearest analogous mine to the Dolphin would be the Fort Knox mine owned and operated by Kinross Gold Corporation. In their March 31, 2015 Technical Report, Kinross reports the mineral resource at a 0.16 g/t Au cut-off within a pit shell based on a $1400 Au price (Sims, 2015). A value of 0.3 g/t Au has been highlighted as a possible cut-off for open pit extraction on the Dolphin deposit.

As part of the 2015 PEA a conceptual open pit, based on $1300 Au, has been produced by Tetra Tech. As a result only blocks falling within this pit are reported as a Resource within the following Tables.

Table 14-10: Dolphin Zone Indicated Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	ozs Au
0.20	82,650,000	0.58	47,610	1,531,000
0.25	71,140,000	0.63	45,030	1,448,000
0.30	**61,460,000**	**0.69**	**42,410**	**1,363,000**
0.35	53,460,000	0.74	39,770	1,279,000
0.40	46,690,000	0.80	37,260	1,198,000
0.50	35,590,000	0.91	32,320	1,039,000
0.60	26,720,000	1.03	27,440	882,000
0.70	20,030,000	1.15	23,110	743,000
0.80	15,030,000	1.29	19,390	623,000
0.90	11,450,000	1.43	16,350	526,000
1.00	8,870,000	1.57	13,910	447,000
1.10	6,990,000	1.71	11,940	384,000
1.20	5,560,000	1.85	10,300	331,000
1.30	4,490,000	2.00	8,960	288,000

Table 14-11: Dolphin Zone Inferred Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	ozs Au
0.20	95,920,000	0.58	55,350	1,779,000
0.25	82,910,000	0.63	52,400	1,685,000
0.30	**71,500,000**	**0.69**	**49,260**	**1,584,000**
0.35	61,640,000	0.75	46,050	1,480,000
0.40	52,690,000	0.81	42,730	1,374,000
0.50	38,800,000	0.94	36,510	1,174,000
0.60	28,710,000	1.08	30,980	996,000
0.70	21,700,000	1.22	26,450	850,000
0.80	16,910,000	1.35	22,880	736,000
0.90	12,890,000	1.51	19,460	626,000
1.00	10,090,000	1.67	16,820	541,000
1.10	8,350,000	1.80	15,000	482,000
1.20	7,050,000	1.92	13,500	434,000
1.30	5,880,000	2.05	12,050	387,000

A second set of tables (**Table 14-12** and **Table 14-13**) show the resource present above the oxide surface, within the Conceptual Pit. A third set of tables (**Table 14-14** and **Table 14-15**) show the resource present below the oxide surface again within the Conceptual Pit.

Table 14-12: Oxide Zone Indicated Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	ozs Au
0.20	22,520,000	0.55	12,270	395,000
0.25	18,960,000	0.61	11,490	369,000
0.30	**16,180,000**	**0.66**	**10,730**	**345,000**
0.35	13,990,000	0.72	10,020	322,000
0.40	12,160,000	0.77	9,340	300,000
0.50	9,180,000	0.87	8,000	257,000
0.60	6,850,000	0.98	6,730	216,000
0.70	5,030,000	1.10	5,550	178,000
0.80	3,700,000	1.23	4,560	147,000
0.90	2,800,000	1.36	3,790	122,000
1.00	2,100,000	1.49	3,130	101,000
1.10	1,650,000	1.61	2,660	85,000
1.20	1,330,000	1.72	2,290	74,000
1.30	1,040,000	1.86	1,930	62,000

Table 14-13: Oxide Zone Inferred Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	ozs Au
0.20	14,660,000	0.47	6,950	223,000
0.25	11,810,000	0.53	6,310	203,000
0.30	**9,620,000**	**0.59**	**5,700**	**183,000**
0.35	8,120,000	0.64	5,220	168,000
0.40	6,910,000	0.69	4,770	154,000
0.50	4,940,000	0.79	3,890	125,000
0.60	3,360,000	0.90	3,020	97,000
0.70	2,330,000	1.01	2,360	76,000
0.80	1,690,000	1.11	1,880	61,000
0.90	1,160,000	1.23	1,430	46,000
1.00	720,000	1.41	1,020	33,000
1.10	510,000	1.57	800	26,000
1.20	360,000	1.75	630	20,000
1.30	270,000	1.91	510	17,000

Table 14-14: Sulfide Zone Indicated Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	ozs Au
0.20	60,130,000	0.59	35,360	1,137,000
0.25	52,180,000	0.64	33,550	1,079,000
0.30	**45,280,000**	**0.70**	**31,650**	**1,018,000**
0.35	39,470,000	0.76	29,800	958,000
0.40	34,530,000	0.81	27,930	898,000
0.50	26,410,000	0.92	24,300	781,000
0.60	19,870,000	1.04	20,720	666,000
0.70	14,990,000	1.17	17,550	564,000
0.80	11,330,000	1.31	14,820	476,000
0.90	8,650,000	1.45	12,550	404,000
1.00	6,770,000	1.59	10,780	347,000
1.10	5,340,000	1.74	9,280	298,000
1.20	4,230,000	1.89	8,010	257,000
1.30	3,450,000	2.04	7,030	226,000

Table 14-15: Sulfide Zone Inferred Resource within Conceptual Pit

Au Cut-off (g/t)	Tonnes> Cut-off (tonnes)	Grade > Cut-off		
		Au (g/t)	Contained	
			kg Au	ozs Au
0.20	81,260,000	0.60	48,350	1,554,000
0.25	71,100,000	0.65	46,070	1,481,000
0.30	**61,880,000**	**0.70**	**43,560**	**1,401,000**
0.35	53,520,000	0.76	40,840	1,313,000
0.40	45,780,000	0.83	37,950	1,220,000
0.50	33,860,000	0.96	32,610	1,048,000
0.60	25,360,000	1.10	27,970	899,000
0.70	19,360,000	1.24	24,080	774,000
0.80	15,210,000	1.38	20,990	675,000
0.90	11,730,000	1.54	18,040	580,000
1.00	9,370,000	1.69	15,810	508,000
1.10	7,840,000	1.81	14,200	456,000
1.20	6,700,000	1.93	12,900	415,000
1.30	5,610,000	2.06	11,530	371,000

14.8 MODEL VERIFICATION

In order to verify the block model results, three methods were used: swath plots, cross sections and a comparison of estimated block grades with new drillhole composites.

Swath plots take slices through the mineral deposit comparing average grades of blocks with the average grades of composites. The results are shown for east-west slices (**Figure 14-8**), for north-south slices (**Figure 14-9**) and for slices in the vertical plane (**Figure 14-10**). In general the block estimates match very well with the sample grades with the larger deviations occurring in areas with few sample points at the horizontal extremities of the zone and at the very bottom.

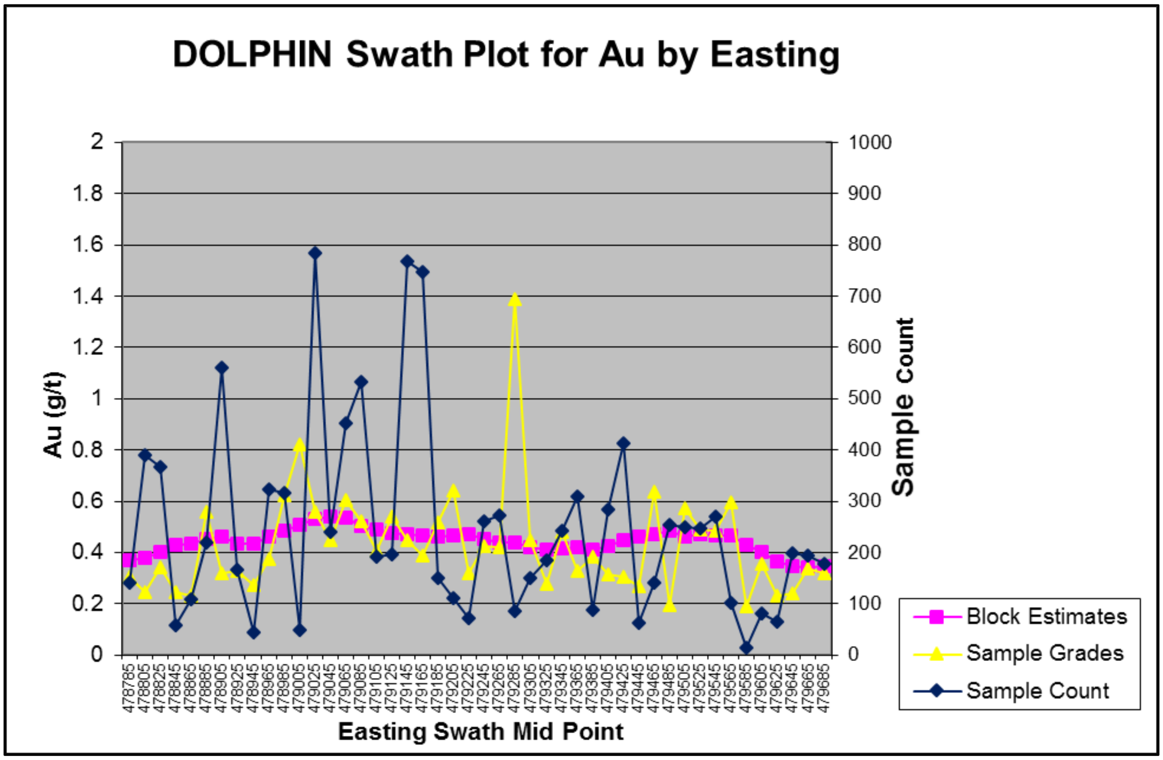


Figure 14-8: Swath Plot for Au along 20 m East-West Slices

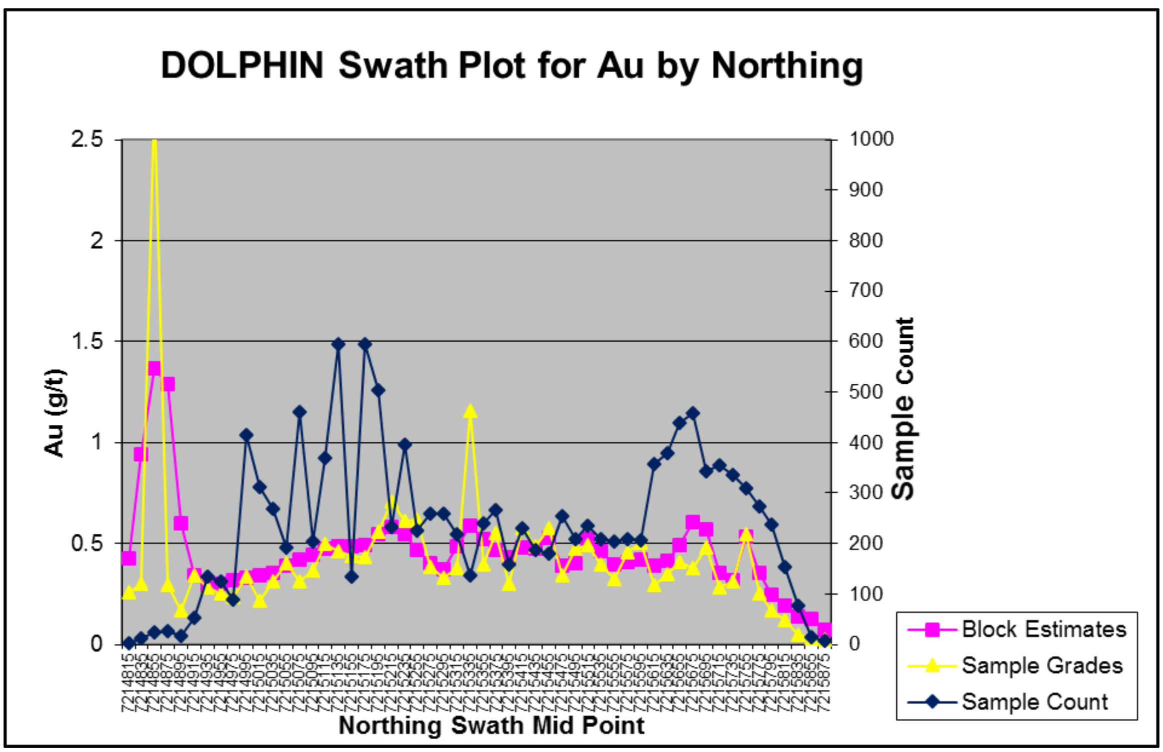


Figure 14-9: Swath Plot for Au along 20 m North-South Slices

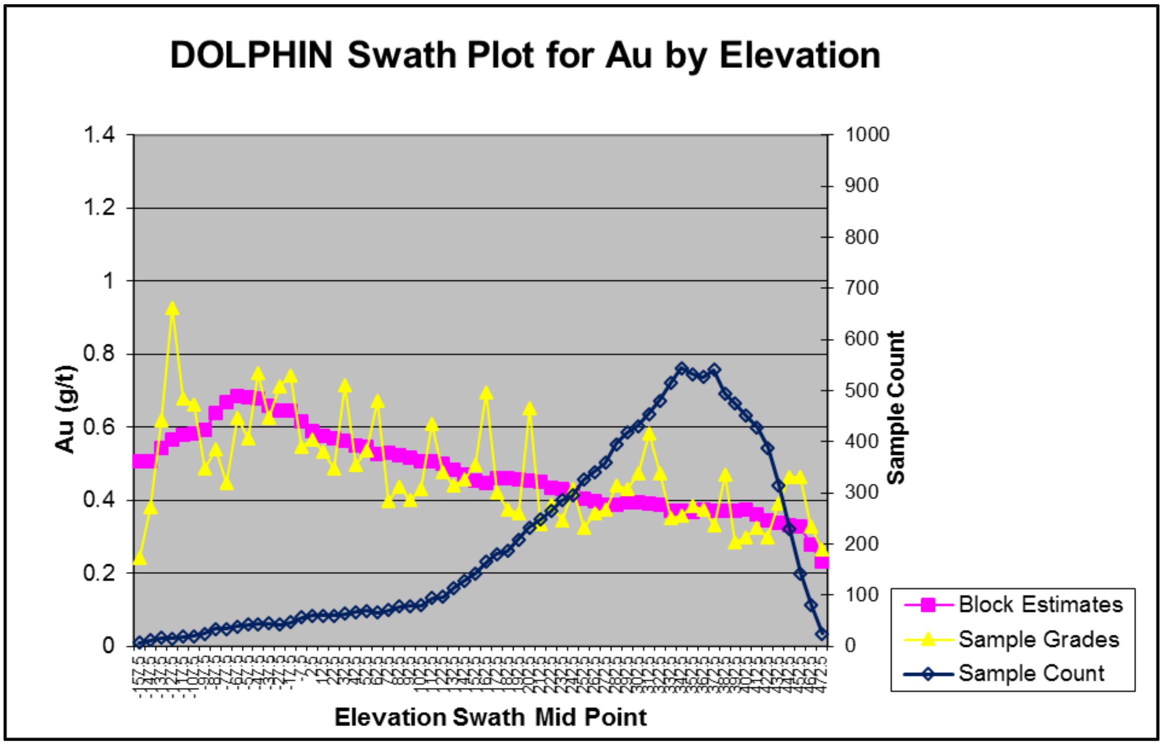


Figure 14-10: Swath Plot for Au along Vertical Slices

Cross sections were evaluated with block grades compared to composite grades with the results appearing reasonable. Three examples are shown as **Figure 14-11**, **Figure 14-12** and **Figure 14-13**.

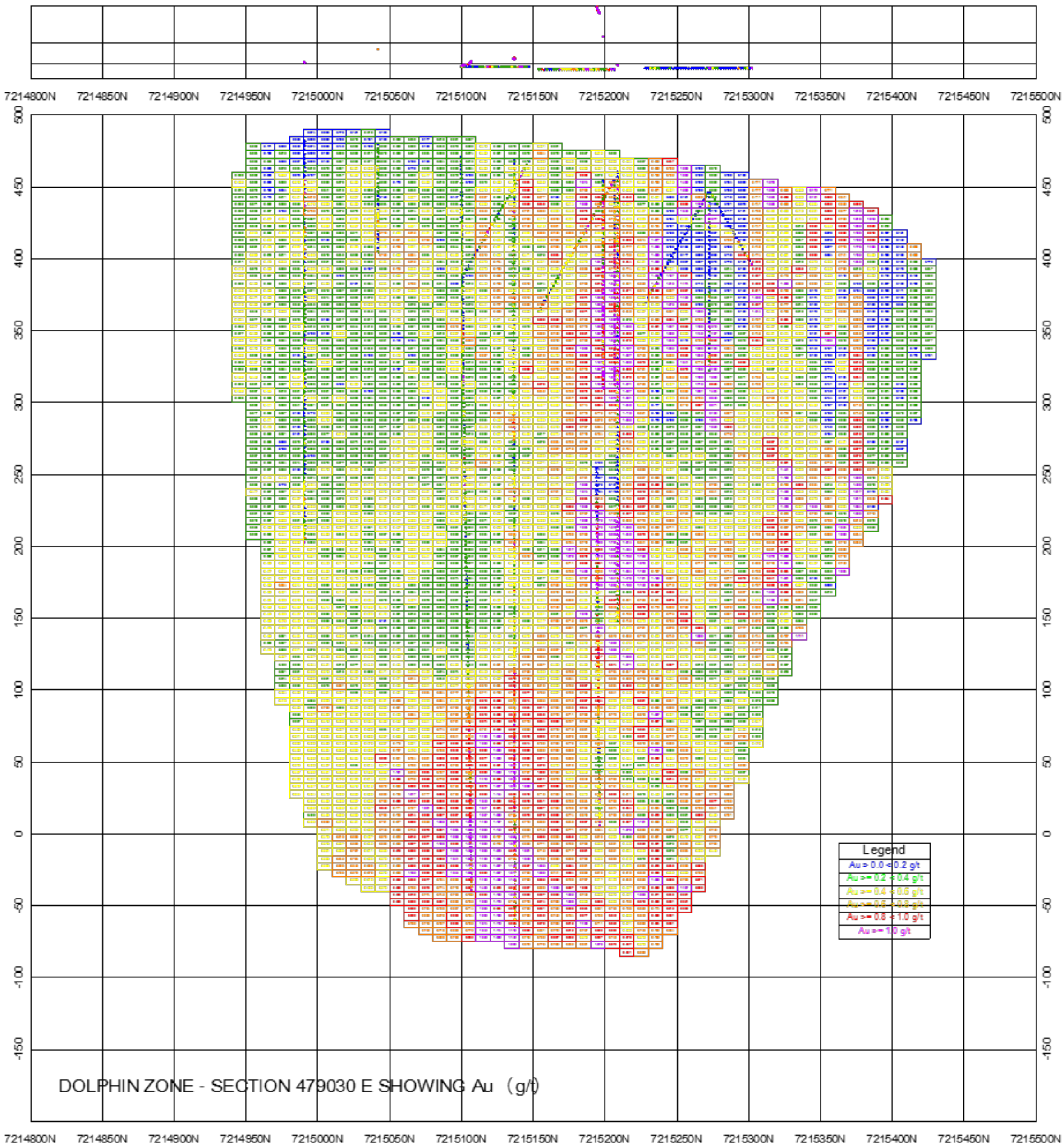


Figure 14-11: Dolphin Zone Section 479030 E

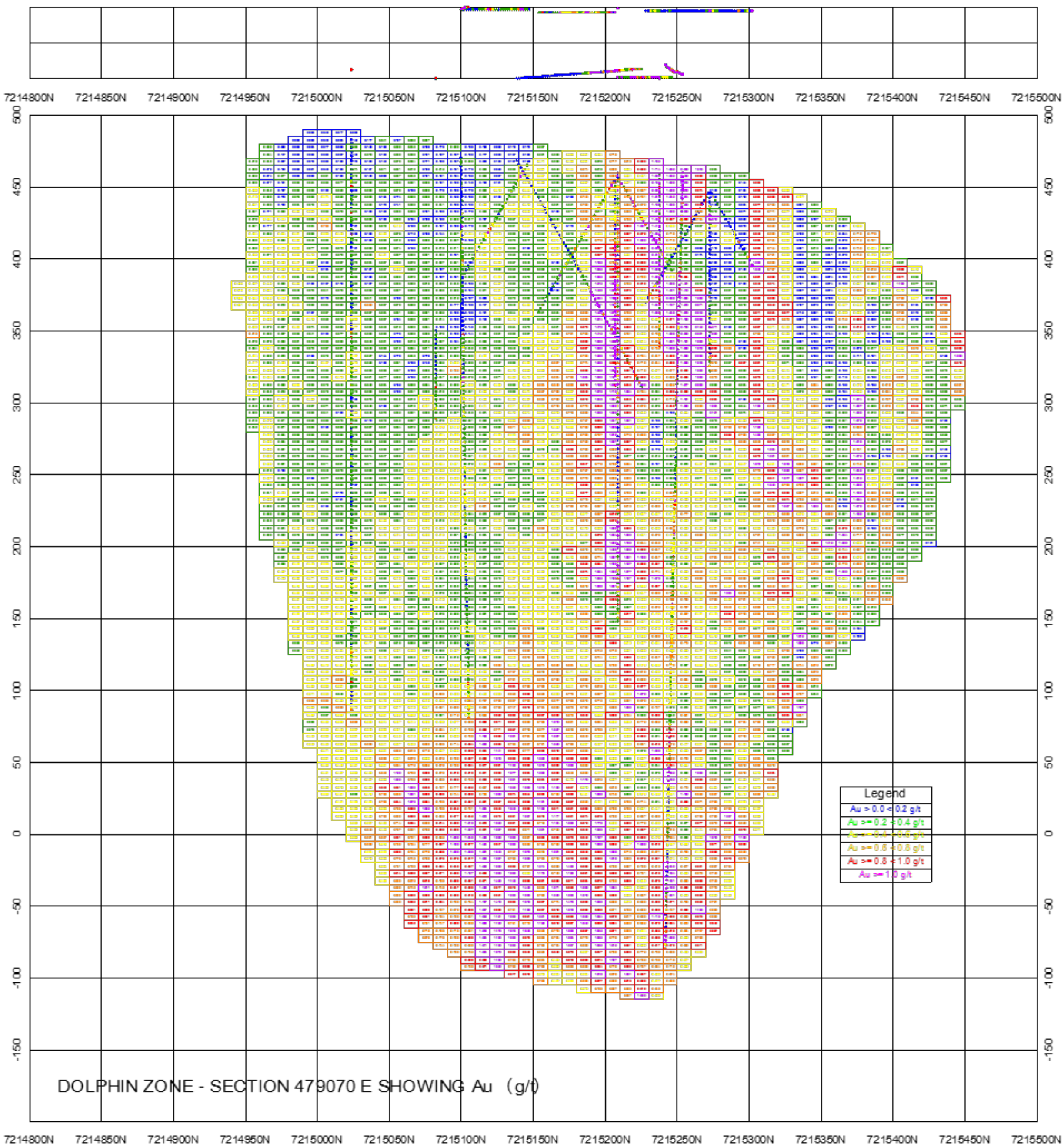


Figure 14-12: Dolphin Zone Section 479070 E

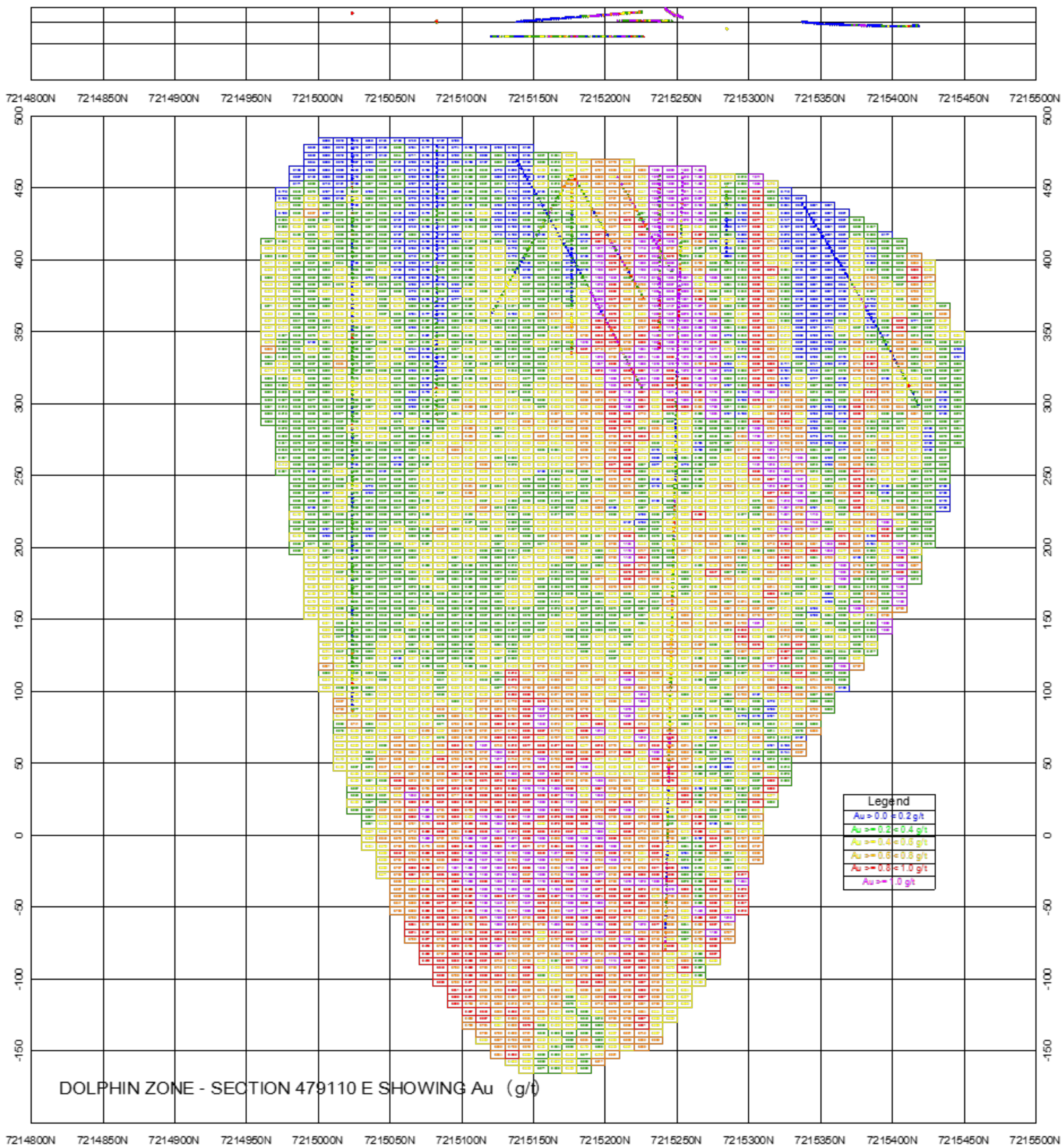


Figure 14-13: Dolphin Zone Section 479110 E

After the 2013 estimate was completed three additional drillholes were completed on the Dolphin Zone: GSDL1311, GSDL1312 and GSDL1313 (see **Figure 14-14**). As a test for the block model the gold assays from these three holes were composited and compared to the estimated gold grades of the blocks that contained them. A scatter plot showing the new hole composite gold grades vs. the estimated blocks is shown as **Figure 14-15**. There is no apparent bias with estimated grades matching new drillhole results reasonably well.

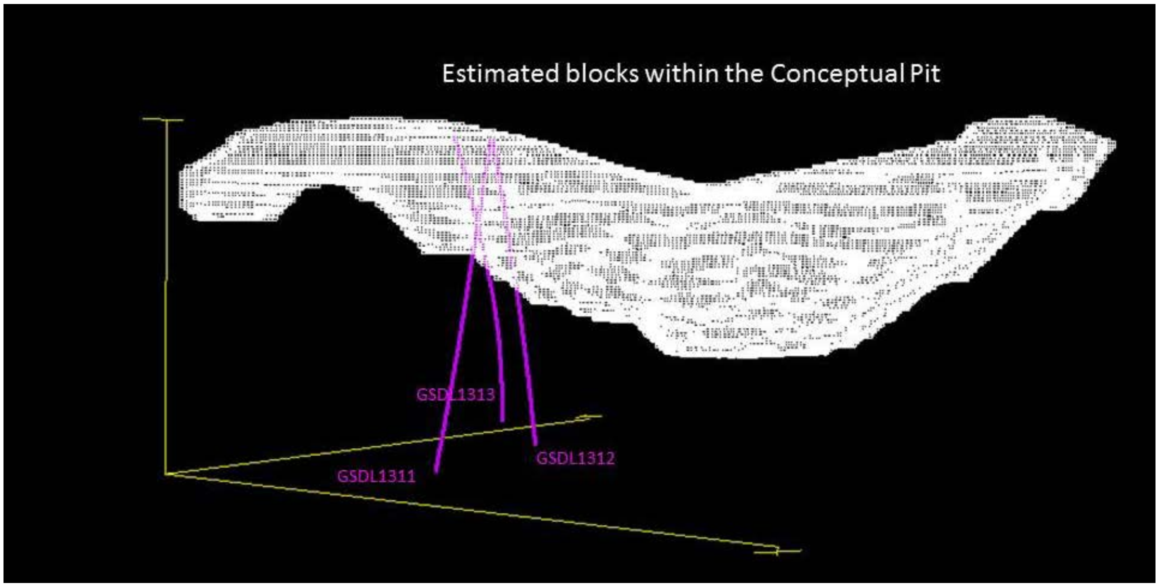


Figure 14-14: Dolphin Zone showing Conceptual Pit in White and 3 new Holes in Magenta

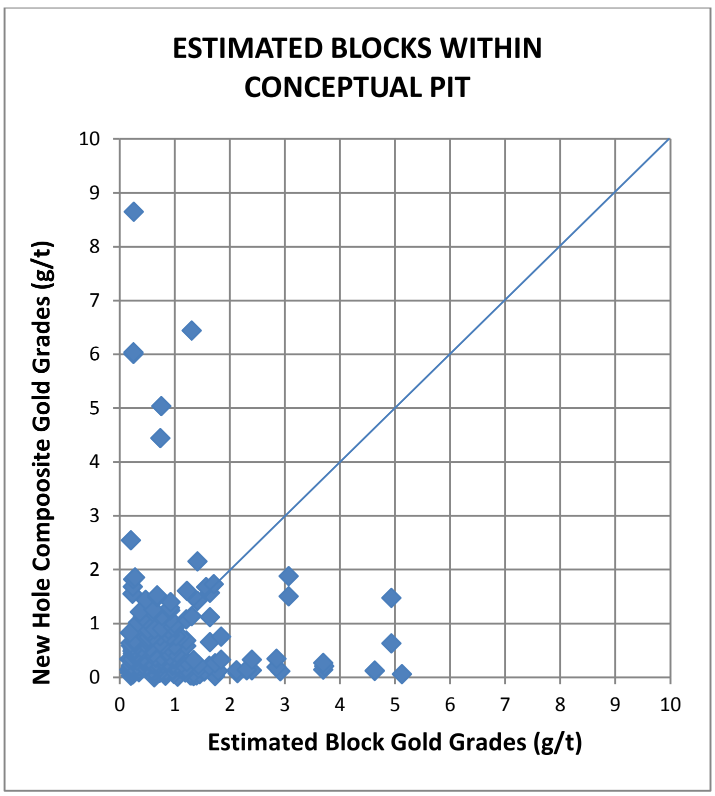


Figure 14-15: Scatter Plot for Gold in Estimated Blocks in Pit vs. New Hole Composite Gold Grades Within Blocks

15.0 MINERAL RESERVE ESTIMATES

Indicated and Inferred resources were used in the LoM plan. Mineral resources are not mineral reserves and have no demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of a positive preliminary feasibility study or feasibility study of a mineral project. Accordingly, at the present level of development, the Golden Summit Project has no mineral reserves.

16.0 MINING METHODS

16.1 CUT-OFF GRADE

Optimized pit cones were created in Vulcan 3D mining software. By using a minimum cut-off grade, the block model is queried by block to determine if the block has positive net block value using the assumed parameters. Blocks above the cut-off grade are flagged as mineralized blocks and assigned costs associated with the mineralized blocks, such as mining and processing costs. Waste blocks below the cut-off grade are assigned a mining cost. The combination of these factors provides an optimized (economic) pit design.

There are two cut-off grades typically used in the mining industry: breakeven and internal. The pit cones were generated using a breakeven cut-off grade, which implies that the mining cost was part of the cut-off calculation to burden every tonne of material mined. This method is used to produce a more conservative economic cone as a design guide. Once the breakeven pit has been designed, an internal cut-off grade is applied to the tonnes inside the pit. An internal cut-off grade removes the mining cost from the calculation thereby dropping the cut-off grade slightly and maximizing the tonnes of material to be processed while reducing the tonnes going to the Mine Rock Storage Facility (MRSF). The internal cut-off grade is applied to the Au grade. Furthermore, the pit optimization runs for the project use Indicated and Inferred Resources when developing pit shells.

The study consists of one ultimate open pit, which contains two types of material, oxide and sulfide. The process costs for the two types of materials differ. These costs were fed into the optimization process. The recovery for each material type is also different and were fed into the pit optimizer. Many factors can change the outcome of the cone analysis, such as the price of Au, cost of mining, and process recovery. The parameters used for the pit optimization are shown in **Table 16-1**.

Table 16-1: Pit Optimization Parameters

Parameters	Value
Gold Price	$1,300/oz
Oxide Recovery	80%
Sulfide Recovery	90%
Mining Cost	$1.65/tonne material mined
Oxide Process Cost	$3.50/tonne processed
Sulfide Process Cost	$20.00/tonne processed
Royalty	$0.72/tonne processed
General and Administrative Cost	$0.80/ton $/tonne processed
Freight/Smelting/Refining	$0.26/ton $/tonne processed

Another factor that can alter the outcome of the cone analysis is the pit slope used during the analysis. A default pit slope of 45°was used to run the pit optimization process.

Two sets of cut off values were calculated, one for the oxide material and one for the sulfide material. The oxide cone used a breakeven cut-off grade of 0.182 g/t Au, and an internal cut-off grade of 0.132 g/t Au. The sulfide cone used a breakeven cut-off grade of 0.611 g/t Au, and an internal cut-off grade of 0.566 g/t Au. Both the breakeven cut-off and the internal cut-off were calculated using

$1,300/oz Au price. Pit optimization results tabulated in this report are calculated on the internal cut off for each material.

16.2 Open Pit Mine Design

Due to the processing of both sulfide and oxide material, there would be two types of material provided for processing. The oxide would be processed via heap leach, while the sulfide would be processed through a plant. The mine has been scheduled to provide up to 3.5 million tonnes per year (Mtpy) of each material type. Oxide material is mined in the early years, as it forms a cap over the sulfide material. Years in the middle of the production schedule have an overlap of oxide and sulfide production prior to completion of oxide mining. A detailed pit design was created using the pit optimizer cones as guidelines. Items included in the design are ultimate pits, phased pit designs and annual pit designs. The ultimate pit was designed to allow mining of economic resources identified by pit optimization while providing safe access for personnel and equipment. The phases within the ultimate pit was developed to enhance the Project by scheduling higher-value material earlier in the mine life.

Oxide material is mined exclusively for the first eight years of the mine production. A small amount of sulfide material would be mined before Year Eight; the sulfide material (approximately 800,000 tonnes) will be stockpiled until the end of mine life. In Year Nine, sulfide material comes online for production. Mining of the oxide material continues through Year 14 of the 24 year mine life. Mining of sulfide material continues from Year Nine through the end of the 24 year mine life.

During production, material, both oxide and sulfide, is transported from the pit to the primary crusher located near the pit exit. After primary crushing, oxide and sulfide material would be transported by conveyor to its respective process area. The oxide leach would be processed in an area to the southeast of the pit, while the sulfide would be processed northwest of the pit. Waste is hauled by truck to the Mine Rock Storage Facility (MRSF). A summary of the open pit design criteria used is included in **Table 16-2**.

Table 16-2: Open Pit Design Criteria Summary

Input	Value
Mining Loss	Mine Plan Model - 5% Ave
Pit Design Parameters	
Benching	10m Single
Haul Roads	Two Way Roads - 27m
Primary Crushing	42 x60 Gyratory located on pit crest
Mine Fleet Parameters	
Loading	64 metric tonne payload Rope Shovels
Haulage	Haul Trucks - 227 metric tonne trucks
Drilling	Diesel Drills - 171mm bit diameter
Work Schedule	
Shifts / Day	2
Shift Length	12 hours
# of work crews	4
Operating Days per Year	365

Input	Value
Production	
Annual Outage Factor	5%
Shift Change Loss Factor	30 minutes/12 hour shift (87.5%)
Operator Efficiency Factor	50 minutes/ hour (84%)
Mechanical Availability Factor	Year 1 = 92% Year 2 = 90% Year 3 = 88% Year 4 = 86% Year 5 = 84% Year 6 = 82% Year 7 --->= 80%
Equipment Productivity Assumptions	
Moisture Content	4%
Swell Factor	40%
Truck Spot Time	90 seconds
Truck Dump @ Crusher	90 seconds
Truck Dump @ MRSF	60 seconds
Shovel Cycle Time per Pass	45 seconds
Rock Density	By block per block model
Truck Cycle Times	Vulcan Haul Profiler software used to develop cycle times per Block in Block Model using OEM Manufacturer provided rim pull data with local speed limits applied
Truck Speed Limits	50 kph (30 mph) - flat empty/loaded 50 kph (30 mph) - uphill empty 42 kph (25 mph) - downhill empty 25 kph (15 mph) - loaded up/downhill
Blasting Powder Factor Feed	0.237 kg/tonne
Blasting Powder Factor - Waste	0.222 kg/tonne
Average Drilling Penetration Rate	25.9 m/hr
Drilling bench – Feed and Waste	10 m
Sub-Drill – Feed and Waste	1.03 m
Stemming - Feed and Waste	3.5 m
Blasthole Diameter – Feed and Waste	171 mm

16.2.1 Pit Slope Constraints

Pit slope configurations used in designing the pit were based on the geologic information provided in the drill logs and physical inspection of the material during the site visit. Since no geotechnical pit slope analysis study has been conducted, a generic pit slope design consisting of 45° overall inter-ramp slope angles with 63° bench face angles were designed, using 10 m benches.

16.2.2 Bench Design

Pit designs were based on 10 m single benches for the rock units. This corresponds with the resource model block heights (10 m).

16.2.3 Haul Road Design

Haul-roads, in general, are designed to be inside of the pit where only one safety berm is required. Haul roads inside and outside of the pit have been designed at an average of 27 meters. This provides approximately 3.5 times the width of the planned trucks.

Ramps were designed to have a maximum centerline gradient of 10%. Switchbacks are designed with flat turnarounds. Once the switchback is complete, the ramp continues at 10%.

16.2.4 Dilution & Mining Loss

Dilution was not applied to the block model used for the pit optimization runs. Nor was a diluted model was used for the mine design. An overall 5% mining loss was applied to the block model mineralized block for design production purposes.

16.2.5 Ultimate Pit Design

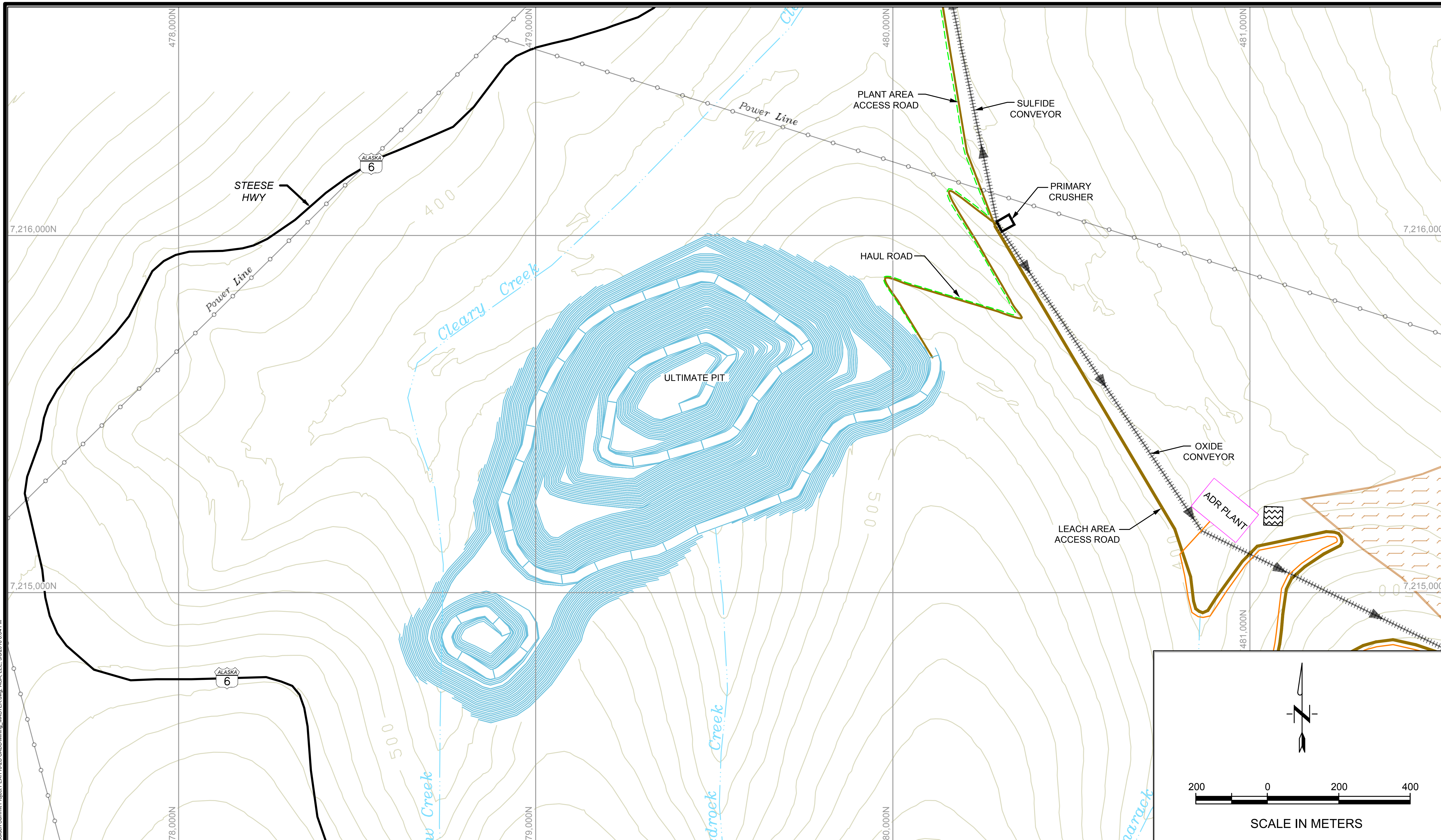
The ultimate pit design uses switchbacks to maintain the road and ramp for the entrance of the pit. This allows for better traffic flow between pit phases. The haul roads provide access to the Primary Crusher. The haul roads also provide access to the MRSF for placement of overburden and waste rock material.

The crest of the ultimate pit is at an elevation of about 460 meters above mean sea level (amsl), with a pit bottom of 80 meters amsl. The ultimate pit design is shown on **Figure 16-1**.

The analysis performed for the development of **Table 16-3** that was utilized for the economic model includes indicated and inferred mineral resources, of which 52% are indicated and 48% are inferred. Mineral resources are considered too speculative geologically to have economic considerations applied to them, and are therefore not categorized as mineral reserves. The reason there are no mineral reserves is that reserves require a positive pre-feasibility of the indicated resource estimates, and the project has not reached that level of advancement. There is no certainty that the preliminary economic assessment will be realized.

Table 16-3: Ultimate Pit Parameters

Description	Value	Unit
Gold Price	$1,300	USD
Waste	239	Mst
Oxide Tonnes	48	Mst
Sulfide Tonnes	50	Mst
Total	337	Mst
Stripping Ratio	2.45	waste:feed
Grade		
Oxide	0.54	g/t
Sulfide	1.14	g/t
Gold Ounces	2,660	koz



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REFERENCE

ENGINEER'S SEAL

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

Rev	Description	BY	Date

REVISIONS

Scale: As Noted

Designed by: K. JOHNSON
Drawn by: L. AGA
Checked by: E. LIPS
Approved by: V. SCHARNHORST

Issued for:

Freegold Ventures Limited

Issued by:

TETRA TECH

GOLDEN SUMMIT PROJECT
ULTIMATE PIT

A
REVISION

Project:
GOLDEN SUMMIT PROJECT
PRELIMINARY ECONOMIC ASSESSMENT

Project no.:
114-910054

Location:
FAIRBANKS MINING DISTRICT, ALASKA

Date:
Oct/2015

FIGURE 16-1

Table 16-4: Mine Production Schedule

		Units	Grand Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Grand Total
Oxide	Oxide Mined to Process	tonnes(000s)	47,864	3,800	3,800	3,550	3,666	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	1,300	58	155	36	0	0	0	0	0	0	0	**47,864**
	Oxide Volume	cubic meters	19,069	1,514	1,514	1,414	1,461	1,394	1,394	1,394	1,394	1,394	1,394	1,394	1,394	1,394	518	23	62	14	0	0	0	0	0	0	0	**19,069**
	Oxide Density		2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	0.000	0.000	0.000	0.000	0.000	0.000	0.000	**2.510**
	Oxide Au	gpt	0.435	0.453	0.500	0.530	0.506	0.493	0.515	0.337	0.369	0.508	0.444	0.381	0.318	0.342	0.331	0.299	0.273	0.290	0.000	0.000	0.000	0.000	0.000	0.000	0.000	**0.435**
		Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Grand Total
Sulfide	Sulfide Mined to Process	tonnes(000s)	48,791	0	0	0	0	0	0	0	0	1,020	1,808	2,598	3,300	3,500	3,500	3,500	3,500	3,500	3,500	3,481	3,412	3,500	3,500	3,500	1,672	**48,791**
	Sulfide Volume	cubic meters	18,274	0	0	0	0	0	0	0	0	382	677	973	1,236	1,311	1,311	1,311	1,311	1,311	1,311	1,304	1,278	1,311	1,311	1,311	626	**18,274**
	Sulfide Density		2.670	0.000	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	2.670	**2.670**
	Sulfide Au	gpt	1.030	0.000	0.586	0.713	0.894	1.034	0.925	0.948	0.987	1.095	1.103	0.977	0.982	0.972	1.081	1.058	1.236	1.266	0.987	0.980	0.941	0.910	0.855	1.057	1.049	**1.030**
Waste	Waste	tonnes(000s)	239,170	3,861	3,230	3,115	3,139	2,719	2,413	4,000	4,000	4,000	9,955	16,055	25,000	24,882	20,665	16,000	16,000	16,000	16,792	12,639	11,814	9,148	8,075	3,950	1,719	**239,170**
	Waste Volume	cubic meters	89,592	1,462	1,241	1,199	1,209	1,053	934	1,505	1,506	1,525	3,742	6,020	9,340	9,286	7,717	5,977	5,965	5,960	6,250	4,716	4,414	3,422	3,024	1,479	644	**89,592**
	Waste Density		2.670	2.640	2.603	2.597	2.597	2.581	2.582	2.657	2.655	2.624	2.660	2.667	2.677	2.680	2.678	2.677	2.682	2.684	2.687	2.680	2.677	2.673	2.671	2.670	2.670	**2.670**
		Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Grand Total
Combined	Mined to Process	tonnes(000s)	96,655	3,800	3,800	3,550	3,666	3,500	3,500	3,500	3,500	4,520	5,308	6,098	6,800	7,000	4,800	3,558	3,655	3,536	3,500	3,481	3,412	3,500	3,500	3,500	1,672	**96,655**
	Volume	cubic meters	37,343	1,514	1,514	1,414	1,461	1,394	1,394	1,394	1,394	1,776	2,072	2,368	2,630	2,705	1,829	1,334	1,372	1,325	1,311	1,304	1,278	1,311	1,311	1,311	626	**37,343**
	Density	tonnes/cu.M	2.588	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.510	2.544	2.562	2.576	2.585	2.588	2.625	2.667	2.663	2.668	2.670	2.670	2.670	2.670	2.670	2.670	2.670	**2.588**
	Au Grade	gpt	0.735	0.453	0.500	0.530	0.506	0.493	0.515	0.337	0.369	0.640	0.669	0.635	0.640	0.657	0.878	1.046	1.196	1.256	0.987	0.980	0.941	0.910	0.855	1.057	1.049	**0.735**
	Waste	tonnes(000s)	239,170	3,861	3,230	3,115	3,139	2,719	2,413	4,000	4,000	4,000	9,955	16,055	25,000	24,882	20,665	16,000	16,000	16,000	16,792	12,639	11,814	9,148	8,075	3,950	1,719	**239,170**
	Waste Volume	cubic meters	89,592	1,462	1,241	1,199	1,209	1,053	934	1,505	1,506	1,525	3,742	6,020	9,340	9,286	7,717	5,977	5,965	5,960	6,250	4,716	4,414	3,422	3,024	1,479	644	**89,592**
	Waste Density		2.670	2.640	2.603	2.597	2.597	2.581	2.582	2.657	2.655	2.624	2.660	2.667	2.677	2.680	2.678	2.677	2.682	2.684	2.687	2.680	2.677	2.673	2.671	2.670	2.670	2.670
		Units	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Grand Total
Open Pit Mine, Mill and Stockpile Information	Tonnes Processed	tonnes(000s)	97,483	3,800	3,800	3,550	3,666	3,500	3,500	3,500	3,500	4,520	5,308	6,098	6,800	7,000	4,800	3,558	3,655	3,536	3,500	3,481	3,412	3,500	3,500	3,500	2,500	**97,483**
	Waste Tonnes	tonnes(000s)	239,170	3,861	3,230	3,115	3,139	2,719	2,413	4,000	4,000	4,000	9,955	16,055	25,000	24,882	20,665	16,000	16,000	16,000	16,792	12,639	11,814	9,148	8,075	3,950	1,719	**239,170**
	Total Tonnes Moved	tonnes(000s)	336,653	7,661	7,030	6,665	6,805	6,219	5,913	7,500	7,500	8,520	15,263	22,153	31,800	31,882	25,465	19,558	19,655	19,536	20,292	16,120	15,225	12,648	11,575	7,450	4,219	**336,653**
	Tonnes mined	tonnes(000s)	97,483	3,800	3,800	3,590	3,796	3,732	3,703	3,587	3,634	4,520	5,308	6,098	6,800	7,000	4,800	3,558	3,655	3,536	3,500	3,481	3,412	3,500	3,500	3,500	1,672	**97,483**
	Au	gpt	0.735	0.453	0.500	0.532	0.520	0.526	0.537	0.352	0.392	0.640	0.669	0.635	0.640	0.657	0.878	1.046	1.196	1.256	0.987	0.980	0.941	0.910	0.855	1.057	1.049	**0.735**
	Sulfide Tonnes to Stockpile	tonnes(000s)	828	0	0	40	130	232	203	87	134	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**828**
	Au	gpt	0.953	0.000	0.586	0.713	0.894	1.034	0.925	0.948	0.987	1.095	1.103	0.977	0.982	0.972	1.081	1.058	1.236	1.266	0.987	0.980	0.941	0.910	0.855	1.057	1.049	**0.953**
	Sulfide Tonnes from Stockpile	tonnes(000s)	828	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	828	**828**
	Au	gpt	0.953	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.953	**0.953**
	Sulfide Tonnes in Stockpile	tonnes(000s)	0.000	0	0	40	171	403	606	694	828	828	828	828	828	828	828	828	828	828	828	828	828	828	828	828	0	0
	Au	gpt	0.000	0.000	0.586	0.712	0.851	0.957	0.946	0.946	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.953	0.000	**0.000**
	Oxide Tonnes to Pad	tonnes(000s)	47,864	3,800	3,800	3,550	3,666	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	3,500	1,300	58	155	36	0	0	0	0	0	0	0	47,864
	Au	gpt	0.435	0.453	0.500	0.530	0.506	0.493	0.515	0.337	0.369	0.508	0.444	0.381	0.318	0.342	0.331	0.299	0.273	0.290	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.435
	Sulfide Tonnes to Mill	tonnes(000s)	49,619	0	0	0	0	0	0	0	0	1,020	1,808	2,598	3,300	3,500	3,500	3,500	3,500	3,500	3,500	3,481	3,412	3,500	3,500	3,500	2,500	**49,618.581**
	Au	gpt		0.000	0.586	0.713	0.894	1.034	0.925	0.948	0.987	1.095	1.103	0.977	0.982	0.972	1.081	1.058	1.236	1.266	0.987	0.980	0.941	0.910	0.855	1.057	1.017	

NOTE: Oxide and Sulfide grades displayed in this table include process recovery.

16.2.6 Pit Phases

Pit phases were used to create a design work flow to assist with better annual pit development thus improving the Project NPV by extracting higher-value material in the early years of the Project. Phase 1 includes mining of the oxide material, which would be produced first via heap leach and must be mined to uncover the sulfide material.

Two criteria were used to establish the best pit-phasing strategy. First, the pit optimizer "nested shells" were used for phase creation. By examining shells with a lower Au sell price, the most profitable material can be targeted for early exploitation. Secondly, the chosen "nested shells" were selected to allow for the creation of push backs with appropriate work areas between phases. From the pit optimizer cones a series of fully designed (including haul roads) pit phases were developed for the life of the mine. The pit phases are shown in **Figure 16-2** through **Figure 16-4**. The final phase (ultimate pit) is shown in **Figure 16-1**.

16.2.7 Annual Pit Designs

Annual pit designs complete with haul roads and slope constraints were designed to meet the annual processing plant requirements while removing the necessary quantity of waste rock material.

16.2.8 Surge Stockpile

A small amount of sulfide material would be mined before Year Eight, but not an amount large enough to justify constructing the process facility earlier. These tonnes (approximately 800,000 tonnes) would be stockpiled until the end of mine life and added to the feed tonnes during the final year of sulfide processing.

16.2.9 Mine Rock Storage Facility Design

A MRSF has been designed to permanently contain the overburden and waste material associated within the pit. The ultimate design incorporates an overall slope angle of 3:1 with catch benches of10 meters on 20 meter lifts. **Figure 18-1** shows the MRSF location. The current MRSF design, located to the northeast of the pits, is built around the hill. The MRSF was designed with a buffer around the nearby creeks.

A 40% swell factor and densities specific to the rock being hauled were used in the volume calculations for the design of the MRSF. The average specific gravity (SG) of the MRSF material (before swell) is estimated to be approximately 2.65. The total MRSF design would contain 100% of the expected waste material planned to be generated - approximately 239 million tonnes of swelled material.

16.2.10 Production Schedule

A mining schedule was developed based on sequencing the pit phases, starting with the Phase 1 and finishing with Phase 3, which is the Ultimate Pit. Scheduling was accomplished using Vulcan Haul Profiler and MineMax Scheduler. Production and waste removal were scheduled to maximize revenue while minimizing yearly production fluctuations.

A summary of mined primary material and waste material was generated for each period. A plant feed schedule was then prepared from the open pit mine material movement schedule.

In-pit material was used to schedule mine production. The final mining production schedule is shown in **Table 16-4**. The mining production schedule, together with the plant feed and projected grades, were used to drive the economic model developed for this study.

16.2.11 Equipment Selection & Productivities

The open pit mine has been planned using diesel blasthole drills, large haul trucks and rope shovels. Production blasthole drilling for both mineralized and waste material would use a DM-45 type diesel drill. Primary mine production is achieved using P&H 2800 rope shovels along with Cat 793 type haul trucks. The shovels have a nominal rated payload of 64 metric tonnes; due to the average density of the material buckets sized at 31 m^3 throughout the life of mine (LoM) were used. The haul trucks have a nominal rated payload of 227 metric tonnes. The drills, shovels and haul trucks selected for the Project are scheduled to operate around the clock and require four crews on 12-hour shifts for complete shift coverage.

The production rate for each truck varies through the life of the mine since productivity is based on the density of the material being loaded and the distance to the destination. The following factors were used in determining the truck and shovel productivities. Mechanical availability was based on age of equipment:

- Year 1 mechanical availability = 92%;
- Year 2 mechanical availability = 90%;
- Year 3 mechanical availability = 88%;
- Year 4 mechanical availability = 86%;
- Year 5 mechanical availability = 84%;
- Year 6 mechanical availability = 82%; and
- Year 7 (and older) mechanical availability = 80%.

Other factors affecting productivity include:

- An operator efficiency factor of 50 minutes per operating hour (84%) was used on all production equipment;
- An annual outage factor of 5% was used - maximum hours available per year is 8,322; and
- A shift change loss factor to account for the time lost in changing crews, breaks, and lunch was used. The shift change factor is 1 hour and 30 minutes lost per shift change (87.5% on a 12 hour shift).

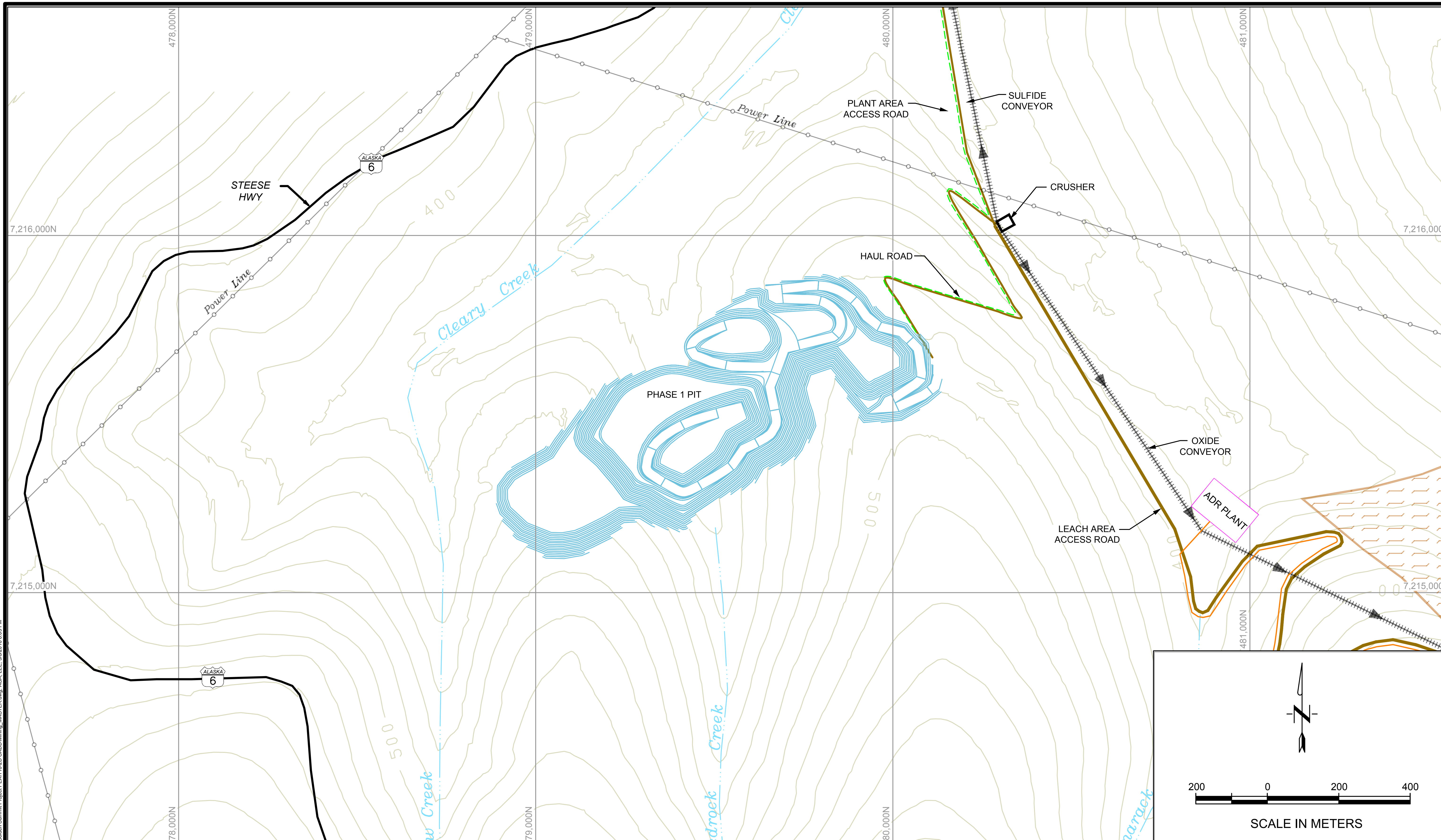
The truck productivity for each block profile was estimated by a haul profile simulator (Vulcan Haul Profiler) which estimates the haul and return times for each block in the block model. Truck cycle-times are based on weighted-average truck-cycle times for resource and waste by period to either a pre-determined primary crusher location or a waste dump location. The destinations include the primary crusher and various MRSF locations depending on material type and period. Each production period has a weight-averaged cycle time estimated for each period's destination. The estimated haul times are shown in **Table 16-6**. Truck fleets were determined based on total operating hours required for resource and waste. Due to a peak in required truck hours during a three year period (Years 12-14) a mining contactor would be used to support the owner-operated trucks. A summary of the estimated maximum owner open pit equipment is shown in **Table 16-7**.

Table 16-5: Production Equipment

Description	Maximum
Drills	
DM-45	7
Shovels	
Rope Shovel	3
Production Support	
Loader Caterpillar 992	1
Wheel Dozer Cat 854	1
Dozer Caterpillar D10	3
Grader Caterpillar 16	2
Water Wagon	1
Haul Trucks	
Haul Truck Cat 793	20

16.2.12 Mine Personnel

Mine personnel estimates include both hourly and salaried staff personnel. Hourly personnel is estimated as the number of people required to operate trucks, loading equipment, and support equipment to achieve the production schedule. Mine staffing is based on the personnel required for supervision and support of mine production. The estimated maximum number of mine personnel required to achieve the mine plan is shown in **Table 16-7**. Hourly wages for each position were estimated based on information estimated from the 2014 CostMine Wage and Salary Survey for an Alaskan mine similar to the Golden Summit Project. Salaries include an allowance for benefits of the base salary for each position.



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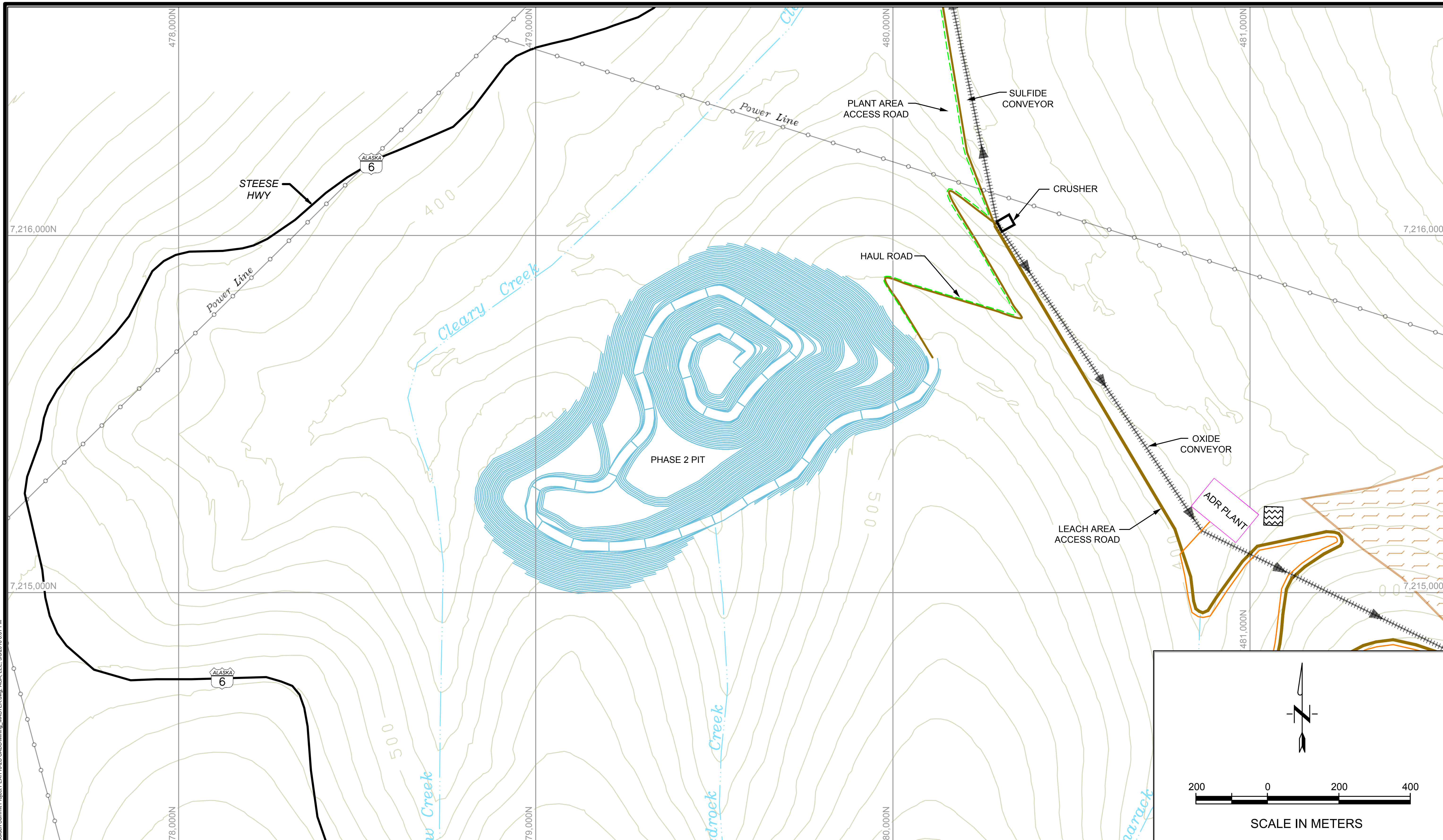
GOLDEN SUMMIT PROJECT
PHASE 1 PIT

REVISION A

Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT
Project no.: 114-910054
Location: FAIRBANKS MINING DISTRICT, ALASKA
Date: Oct/2015

FIGURE 16-2



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GOLDEN SUMMIT PROJECT
PHASE 2 PIT

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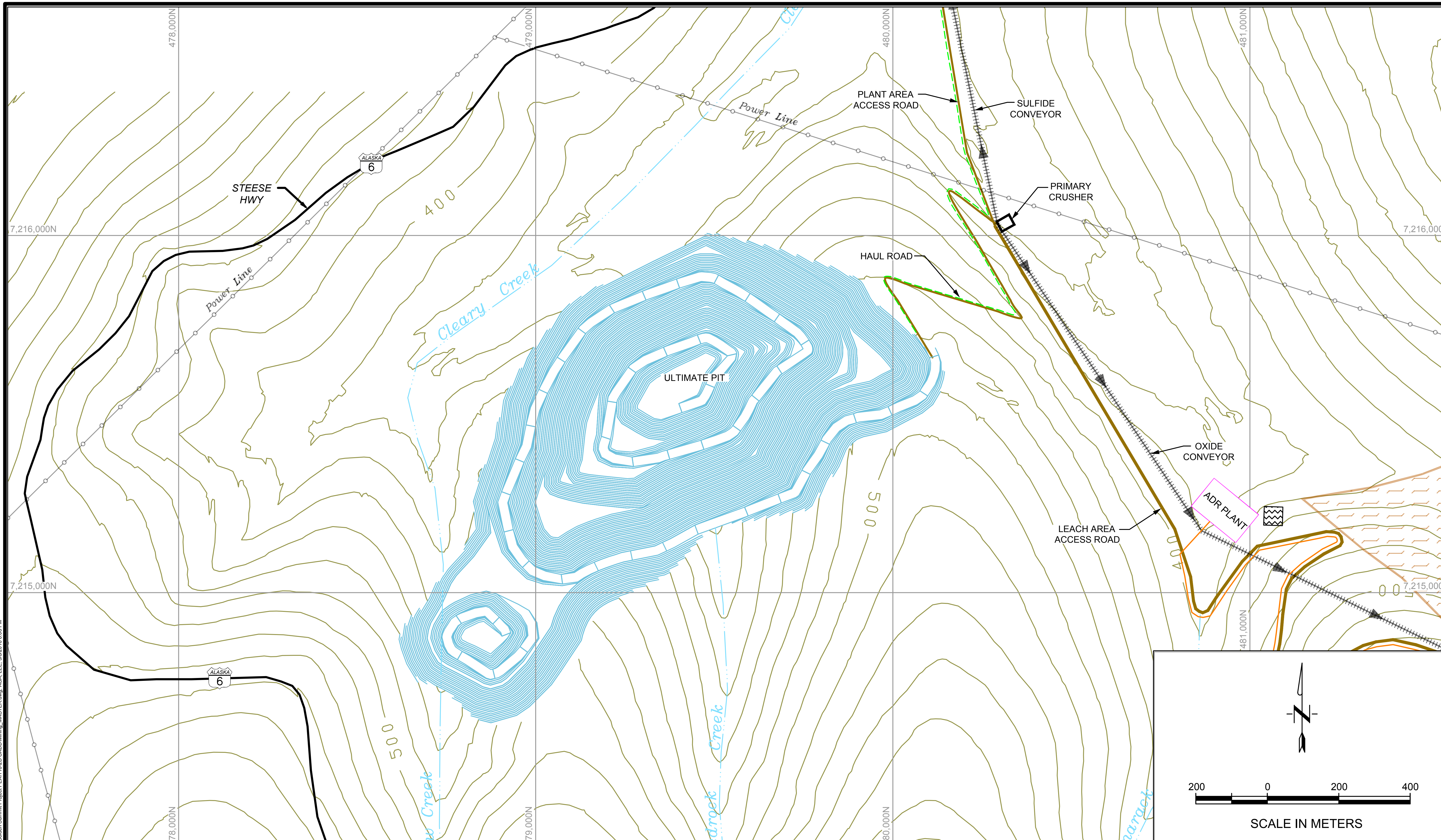
Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT

Project no.: 114-910054

Location: FAIRBANKS MINING DISTRICT, ALASKA

Date: Oct/2015

FIGURE 16-3



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GOLDEN SUMMIT PROJECT
PHASE 3/ULTIMATE PIT

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Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT
Project no.: 114-910054
Location: FAIRBANKS MINING DISTRICT, ALASKA
Date: Oct/2015

FIGURE 16-4

Table 16-6: Haul Time Estimates

Description		Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Yr 22	Yr 23	Yr 24
RESOURCE																									
Load		4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50
Haul		22.54	21.71	19.71	19.05	18.00	16.84	14.06	14.71	16.11	17.02	17.37	19.13	17.71	20.39	24.35	24.88	27.44	30.07	33.77	28.24	30.68	33.28	36.44	39.15
Dump		1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50
Total Cycle Time	Minutes	28.54	27.71	25.71	25.05	24.00	22.84	20.06	20.71	22.11	23.02	23.37	25.13	23.71	26.39	30.35	30.88	33.44	36.07	39.77	34.24	36.68	39.28	42.44	45.15
WASTE																									
Load		4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50
Haul		23.15	33.50	32.62	32.38	31.48	30.46	26.79	28.79	31.55	32.34	33.41	42.70	43.78	45.04	46.02	46.82	46.93	47.33	50.95	53.62	56.82	59.82	63.12	67.14
Dump		1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Total Cycle Time	Minutes	28.65	39.00	38.12	37.88	36.98	35.96	32.29	34.29	37.05	37.84	38.91	48.20	49.28	50.54	51.52	52.32	52.43	52.83	56.45	59.12	62.32	65.32	68.62	72.64

Table 16-7: Open Pit (Maximum) Manpower Table

Mine Operations	Hourly Rate	Benefit Load	Total Hourly Rate	Maximum Manpower
Driller, blasthole	$ 33.80	50.9%	$ 51.01	24
Driller Helper, blasthole	$ 26.84	50.9%	$ 40.51	8
Blaster	$ 33.19	50.9%	$ 50.09	2
Blaster Helper	$ 27.18	50.9%	$ 41.02	5
Shovel Operator	$ 33.80	50.9%	$ 51.01	11
Wheel Dozer Operator	$ 26.84	50.9%	$ 40.51	6
Truck Driver	$ 23.30	50.9%	$ 35.17	71
Track Dozer Operator	$ 26.84	50.9%	$ 40.51	11
Loader Operator	$ 26.84	50.9%	$ 40.51	6
Grader Operator	$ 26.84	50.9%	$ 40.51	11
Water Truck Driver	$ 26.84	50.9%	$ 40.51	3
Dispatcher	$ 33.96	50.9%	$ 51.26	4
Laborer/Trainee	$ 19.96	50.9%	$ 30.13	3
VSA Operator*	$ 26.84	50.9%	$ 40.51	9
VSA Laborer/Trainee**	$ 19.96	50.9%	$ 30.13	4
			Subtotal =	**178**
Maintenance				
Heavy Equip. Mechanic	$ 36.05	50.9%	$ 54.41	15
Welder/Mechanic	$ 25.77	50.9%	$ 38.89	8
Electrician/Instrumentman	$ 33.10	50.9%	$ 49.96	8
Lubeman/PM Mechanic	$ 25.77	50.9%	$ 38.89	8
Tireman	$ 25.77	50.9%	$ 38.89	4
Machinist	$ 36.05	50.9%	$ 54.41	4
Crusher/Belt Operator	$ 23.72	50.9%	$ 35.80	8
Utilityman	$ 33.96	50.9%	$ 51.26	3
Laborer/Trainee	$ 19.96	50.9%	$ 30.13	2
VSA Mechanic*	$ 25.77	50.9%	$ 38.89	3
VSA Laborer**	$ 19.96	50.9%	$ 30.13	2
			Subtotal =	**64**
Salary				
Production Superintendent	$ 126,900	43.0%	$ 181,467	1
Mine Foreman	$ 98,600	43.0%	$ 140,998	17
Maintenance Superintendent	$ 126,200	43.0%	$ 180,466	1
Maintenance Foreman	$ 98,600	43.0%	$ 140,998	8
Maint. Planner	$ 86,200	43.0%	$ 123,266	4
Chief Engineer*	$ 120,600	43.0%	$ 172,458	1
Sr. Mine Engineer*	$ 110,000	43.0%	$ 157,300	1
Mine Engineer	$ 86,200	43.0%	$ 123,266	2
Chief Geologist	$ 115,600	43.0%	$ 165,308	1
Geologist	$ 79,000	43.0%	$ 112,970	2
Equipment Trainer	$ 100,000	43.0%	$ 143,000	1
Surveyor	$ 86,200	43.0%	$ 123,266	2
Surveyor Ass't	$ 47,500	43.0%	$ 67,925	2
Sampler	$ 47,500	43.0%	$ 67,925	2
			Subtotal =	**57**
			Total (Maximum) Manpower needed =	**299**

** 5% of total for Vacations, Sickness, and Absenteeism (VSA)*

*** 2% of total for Vacations, Sickness, and Absenteeism (VSA)*

Note: Benefits listed include scheduled/planned overtime but excludes bonus pay

17.0 RECOVERY METHODS

Gold recovery from the Project deposit would be accomplished in two separate processing operations for oxide and sulfide mineralized materials. Gold from oxide material in Phase 1 production would be recovered by crushing run-of-mine (RoM) material prior to loading onto a heap leach pad. The crushed oxide material would then be leached with a sodium cyanide solution to recover the soluble gold. Gold from the pregnant leachate solution would then be recovered onto activated carbon and further refined in an elution/electrowinning (EW) circuit. The product from the EW cells would be further refined into gold doré. For the purpose of this report, an oxide gold recovery of 80% was used in all calculations based on the available metallurgical testwork.

Gold from the sulfide materials would be recovered by crushing and grinding the material prior to bio-oxidation of the sulfide minerals. The oxidized slurry would be sent to a carbon-in-leach (CIL) circuit for cyanide leaching and recovery onto activated carbon. Gold would be loaded onto the activated carbon and then recovered in the same elution circuit as the oxide material to produce gold doré. For the purpose of this report, a sulfide gold recovery of 90% was used in the calculations. Additional metallurgical testwork is needed to confirm the gold recovery rate.

17.1 SULFIDE MATERIAL PROCESSING TRADEOFF STUDY

Metallurgical testwork (SGS, 2014) on the Project deposit showed that sulfide oxidation would likely be necessary to achieve acceptable gold recoveries in the non-oxide feed material. An economic tradeoff study was performed between three options in order to confirm the need for oxidation as well as to determine a preferred processing method for the sulfide feed material. The three options investigated were:

1) Heap leaching of sulfide material,
2) Whole material pressure oxidation (POX) followed by CIL, and
3) POX treatment of sulfide flotation concentrate followed by CIL.

All three options include heap leaching of oxide material. Due to the lower recoveries observed in the metallurgical testwork, flowsheets for the leaching of either roasted material or non-oxidized flotation concentrate were not considered in the tradeoff.

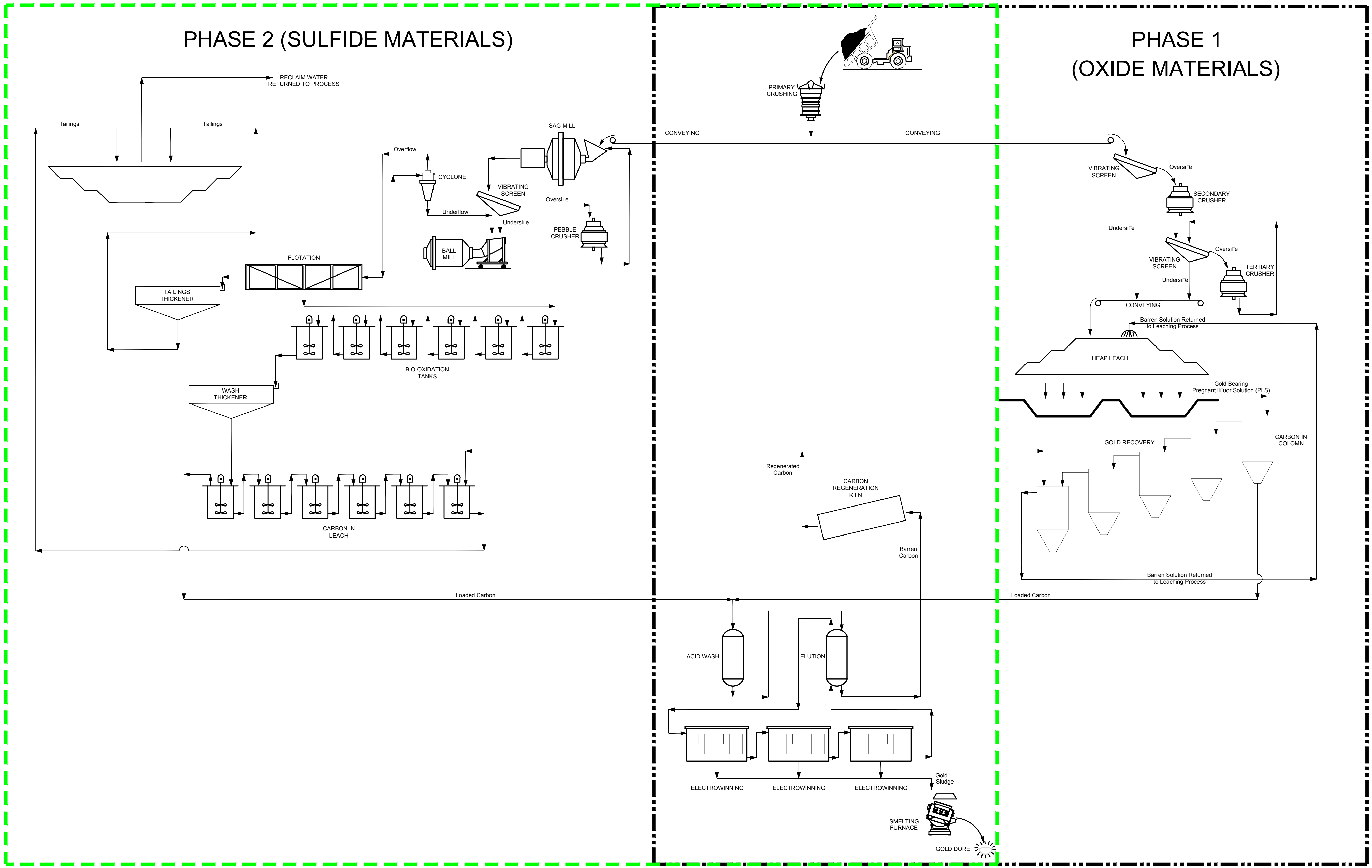
The whole sulfide material POX-CIL option provided negative economic results and was eliminated as a processing option. Both the float-POX-CIL option and sulfide heap leaching provided positive economic results with the float-POX-CIL option having a higher relative NPV, higher capital cost, and longer payback period. At the conclusion of the tradeoff study, it was determined that neither of these two options were economically-preferred processing methods as the float-POX-CIL option was determined to have a relatively high capital cost and the sulfide heap leaching option only processes a small portion of the sulfide material due to low recoveries.

17.2 Bio-Oxidation of Sulfide Materials

Once the tradeoff study concluded that the aforementioned processing options examined in the metallurgical test program were economically advantageous for the processing of the sulfide material, it was determined that bio-oxidation of sulfides would be a possible alternative processing method. While no metallurgical bio-oxidation testwork has been performed on the deposit, the success of other oxidation methods would indicate that bio-oxidation of the sulfide material is feasible. Additionally, benchmarking of bio-oxidation plants around the world indicated that the Project would be similar in size and cost to existing operating projects.

17.3 Processing Flowsheet

Processing of mineralized materials at the Project facility would consist of two separate phases, termed Phase 1 for oxide materials and Phase 2 for sulfide materials. The Phase 1 production would process oxide materials at nominal rate of 10,000 tpd by heap leaching. The Phase 2 production would provide bio-oxidation and leaching of sulfide materials in a nominal 10,000 tpd processing facility, starting in production year nine of the mine life. Heap leaching of oxide materials would continue throughout Phase 2 until the end of the mine life. A simplified flowsheet for both processing circuits is shown in **Figure 17-1**.



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PROCESS FLOWSHEET

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Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT

Project no.: 114-910054

Location: FAIRBANKS MINING DISTRICT, ALASKA

Date: August/2015

FIGURE 17-1

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17.3.1 Oxide Heap Leach

Crushed oxide material would be received from the gyratory crusher located at the mine and conveyed to secondary and tertiary crushing circuit to reduce the size to a nominal minus one-inch product. The crushed material would be placed on a nominal 10,000 tpd lined heap leach pad via conveyors. After the material is prepared for leaching, barren leach solution containing sodium cyanide would be applied to the heap leach surface using buried drip irrigation lines. Pregnant leach solution would percolate through the heap and would be collected in the drainage overliner and gravity flow into pregnant solution pond. Pregnant solution would be pumped from the pregnant solution pond to carbon adsorption columns (CIC). Additional sodium cyanide would then be added to the barren leach solution to maintain reagent concentrations and pumped back to the heap leach. Heap leaching of fresh oxide material would occur seasonally with new oxide material being added to the pad as weather allows. During the cold weather months, leach solution would be recirculated within the pad, but no fresh leaching would occur. The designed primary leach cycle is 90 days with secondary leaching occurring on subsequent lifts.

Loaded carbon from the CIC would be transported to the elution circuit where it would be acid washed prior to stripping. After acid washing, the carbon would be neutralized with caustic and transferred to a stripping vessel. Carbon stripping would use a pressurized Zadra method to desorb the gold from the carbon. Stripped carbon would be transferred to a rotary kiln for thermal reactivation prior to being returned to CIC.

Effluent solution from the stripping vessel would be circulated through EW cells to precipitate gold into a concentrated sludge. Solution from the discharge of the EW cells would be recirculated back to the elution circuit. Gold-bearing sludge from the EW cells would be periodically collected for smelting into gold doré.

Major equipment planned for the oxide leach process is presented in **Table 17-1**.

Table 17-1: Oxide Equipment List

Equipment Description	Number	Size
Secondary Cone Crusher; Standard	1	7 ft diameter; 800 HP
Tertiary Cone Crusher; Shorthead	1	7 ft diameter; 800 HP
Vibrating Screens	2	8 ft by 16 ft; double-deck; inclined; 40 HP
Grasshopper Conveyors	10	36 inch width by 100 ft long; 20 HP Each
Carbon Columns	5	16 ft diameter
Submersible Solution Pumps (8 operating; 2 standby)	10	1,100 gpm, 400 ft head: 150 HP
Centrifugal Solution Pumps (2 operating; 2 standby)	4	10,000 gpm, 250 ft head; 1,000 HP
Carbon Stripping Circuit	1	4 ton capacity; 3 HP
Carbon Reactivation Kiln	1	4 ft diameter by 25 ft long; 15 HP; propane fueled
EW Cells	3	16 cubic ft capacity
Gold Furnace	1	285 lb capacity; propane fueled

17.3.2 Sulfide Bio-Oxidation & Leaching

Phase 2 of the project would use the existing primary crushing circuit from Phase 1 to provide primary crushed sulfide mineralized material to a crushed coarse material stockpile at the process plant site. Crushed sulfide material would be reclaimed by apron feeders and conveyed to the primary grinding circuit. The primary grinding circuit would use a SAG mill in closed circuit with a pebble crusher to grind the material to an acceptable size for the secondary grinding circuit. The secondary grinding circuit would use a ball bill operating in closed circuit with hydrocyclones to produce material suitable for rougher flotation assumed at P_{80} 100-200 microns for this study (to be confirmed by additional test work).

Ground material from the cyclone overflow would then be sent to a flotation circuit to recover gold-bearing sulfide mineralization. Flotation concentrate would then be pumped to bio-oxidation tanks for sulfide oxidation. The oxidized residue would be pumped to acid neutralization circuit to increase the pH of the slurry to acceptable levels for cyanide leaching. Sodium cyanide would then be added to the neutralized slurry and be sent to Carbon-in-Leach (CIL) tanks to recover the gold onto activated carbon. Tailings from the CIL circuit would then be treated for cyanide detoxification and sent to a tailings storage facility. Loaded carbon from the CIL circuit would be transported to the shared elution circuit of the oxide circuit from Phase 1 where the gold would be stripped from the carbon, recovered by EW cells, and smelted into gold doré.

Table 17-2 lists the major equipment items for the Phase 2 sulfide process.

Table 17-2: Sulfide Equipment List

Equipment Description	Number	Size
SAG Mill	1	26 ft by 12 ft; 4,500 HP
Vibrating Screen	1	8 ft by 16 ft; double-deck; inclined; 40 HP
Pebble Shorthead Crusher	1	5 ft diameter; 500 HP
Cyclone Feed Pumps (1 operating; 1 standby)	2	10,000 gpm; 500 HP
Cyclones	5	26 inch diameter
Ball Mill	1	16 ft by 28 ft; 4,500 HP
Rougher Flotation Cells	5	3,500 cubic ft; 125 HP each cell
Flotation Concentrate Thickener	1	25 ft diameter; 2 HP
Biox Tanks	6	35 ft diameter by 35 ft high; agitated; 150 HP Each
Biox Wash Thickener	1	25 ft diameter; 2 HP
CIL Tanks	6	35 ft diameter by 35 ft high; agitated; 150 HP Each
Tailings Thickener	1	50 ft diameter; 5 HP
Slurry Pumps (10 operating; 10 standby)	20	2,500 gpm; 100 HP Each

18.0 PROJECT INFRASTRUCTURE

18.1 SITE LAYOUT

The proposed on-site and off-site infrastructure for the Project will include:

- Process Plants
- Truck Shop
- Administration Building
- Process/Mine Warehouse
- Substation and power distribution
- Mine Rock Storage Facility
- Tailings Storage Facility
- Water Treatment Facility
- Wastewater Treatment Facility
- Access and site roads

The general arrangement for the site is provided in **Figure 18-1**.

18.2 PROCESS PLANTS

The oxide Adsorption Desorption Recovery **(**ADR**)** plant is shown in **Figure 18-1**. The following supporting infrastructure for the Oxide process facility includes

- Three stage crushing and conveying circuit
- Heap leach pad and solution storage
- Carbon adsorption columns
- Carbon stripping circuit
- Carbon reactivation kiln
- Electrowinning cells
- Gold smelting furnace
- Reagent handling
- Maintenance/Warehouse
- ADR Building and Operations Office

The sulfide processing facility, shown on **Figure 18-2**, includes:

- Primary crushing circuit
- Primary and secondary grinding circuits
- Sulfide flotation
- Bio-oxidation tanks and wash thickener
- CIL leaching circuit
- Tailings thickener
- Tailing storage facility
- Reagent handling
- Carbon stripping circuit (shared with oxide process)
- Carbon reactivation kiln (shared with oxide process)

- Electrowinning cells (shared with oxide process)
- Gold smelting furnace (shared with oxide process)
- Assay and metallurgical laboratory Carbon adsorption columns

18.3 Project Logistics

The Property has direct access to Fairbanks via paved state highways (reference Figure 4-1). The City of Fairbanks serves as the region service and supply center for Interior Alaska. It serves as the seat of government for the Fairbanks North Star Borough, where the Property is located, which comprises a total population of approximately 100,000.

Fairbanks has excellent labor and services infrastructure, including rail and international airport access. The Fairbanks International airport is served by several major airlines with numerous scheduled daily flights. The main campus of the University of Alaska is located in Fairbanks in addition to numerous State and federal Offices. Major employers within the Fairbanks Area include Fort Knox, Fort Wainwright (US Army), the University of Alaska as well as numerous state and federal agencies. Exploration and development costs in the Fairbanks area are at or below those common in the western United States.

18.4 Roads & Rail

From Fairbanks, the Property lies approximately 29 km (18 miles) northeast via State Hwy 2 and State Hwy 6 (the Steese Highway). The site holds a series of gravel roads which allow access to most areas of the property on a year-round basis.

Fairbanks is served by the Alaska Railroad, and is connected to Anchorage and Whitehorse, Canada by well-maintained paved highways.

General corridor and road sections are provided in **Figure 18-3** to **Figure 18-5**.

18.5 Buildings & Facilities

The main entrance to the project would be constructed on the northwest side of the property with the main access road coming from the Steese Highway. The administration building, parking lot and fuel farm would be just inside the ADR Plant area compound surrounded by an 8-ft chain link fence. Just past the administration building would be a security gate with an armed guard controlling access to the mineralized process portion of the plant area.

The supporting buildings, and a description of purpose and phase are included in the **Table 18-1.** For Phase 2 facilities, the maintenance shop areas for the sulfide process plant will be contained within the mill building.

Table 18-1: Buildings and Facilities

Building/Structure	Description
	PHASE 1
Mine Entrance	Located at project entrance gate; includes reception; security; gate bar; desks
Administrative Building	Reception; offices; conference room; communications center; dining/kitchen area; all office equipment and furnishings
Laboratory	Metallurgical lab; sample preparation; assay laboratory; offices, sample storage; assay equipment; office furnishings
Change House	Showers, toilets, lockers/change areas (separate for work and street clothes); security
	HEAP LEACH AREA:
Operations Office	Offices for operations and maintenance staff; lunch room area
Maintenance/Warehouse	Closed area for shelving for spare parts and equipment; outside fenced-area for large equipment such as crusher liners
	PHASE 2: *The maintenance shop areas for the sulfide process plant will be contained within the mill building.*
Mine Entrance	Located at project entrance gate; includes reception; security; gate bar
Administrative	Reception; offices; conference room; communications center; dining/kitchen area
Laboratory	Metallurgical lab; sample preparation; assay laboratory; offices, sample storage
Change House	Expanded for additional personnel in mine and sulfide process plant
HEAP LEACH AREA:	*For Phase 2, assumes there would not be any additional building/structures required for heap leach area.*
Note 1: assumes no additional building construction for Phase 2 production.	

18.6 Power Supply

Power would be supplied to the Project by two 3-MW diesel generators in Phase 1. Once the Project ramps up to Phase 2, Freegold would upgrade the system with supply from Golden Valley Electric Association (GVEA), who provide electric power in the Fairbanks area and along the Steese Highway. The estimated peak electrical load for the Project is estimated at approximately 15MW. In order to serve this load, the existing 138kV transmission line currently terminating at the Ft. Knox Mine would be extended to the Project. This would include construction of a 138kV switching station near the existing 138kV transmission line to provide a connection point, and a 138kV substation.

18.7 Communications

Existing telephone lines run along the Steese Highway and there is currently cellular phone coverage servicing the property.

On-site communication systems would include a voice over internet protocol (VoIP) telephone system, a local area network (LAN) with wired and wireless access points, and hand-held very high frequency (VHF) radios. Telecommunications for the Project would be provided by Summit Telephone Company.

The estimate includes providing telephony for approximately 20 personnel in the administration office and sites within the sulfide plant area and oxide leach facility. The phone system and internet access would be provided by Alaska Communications.

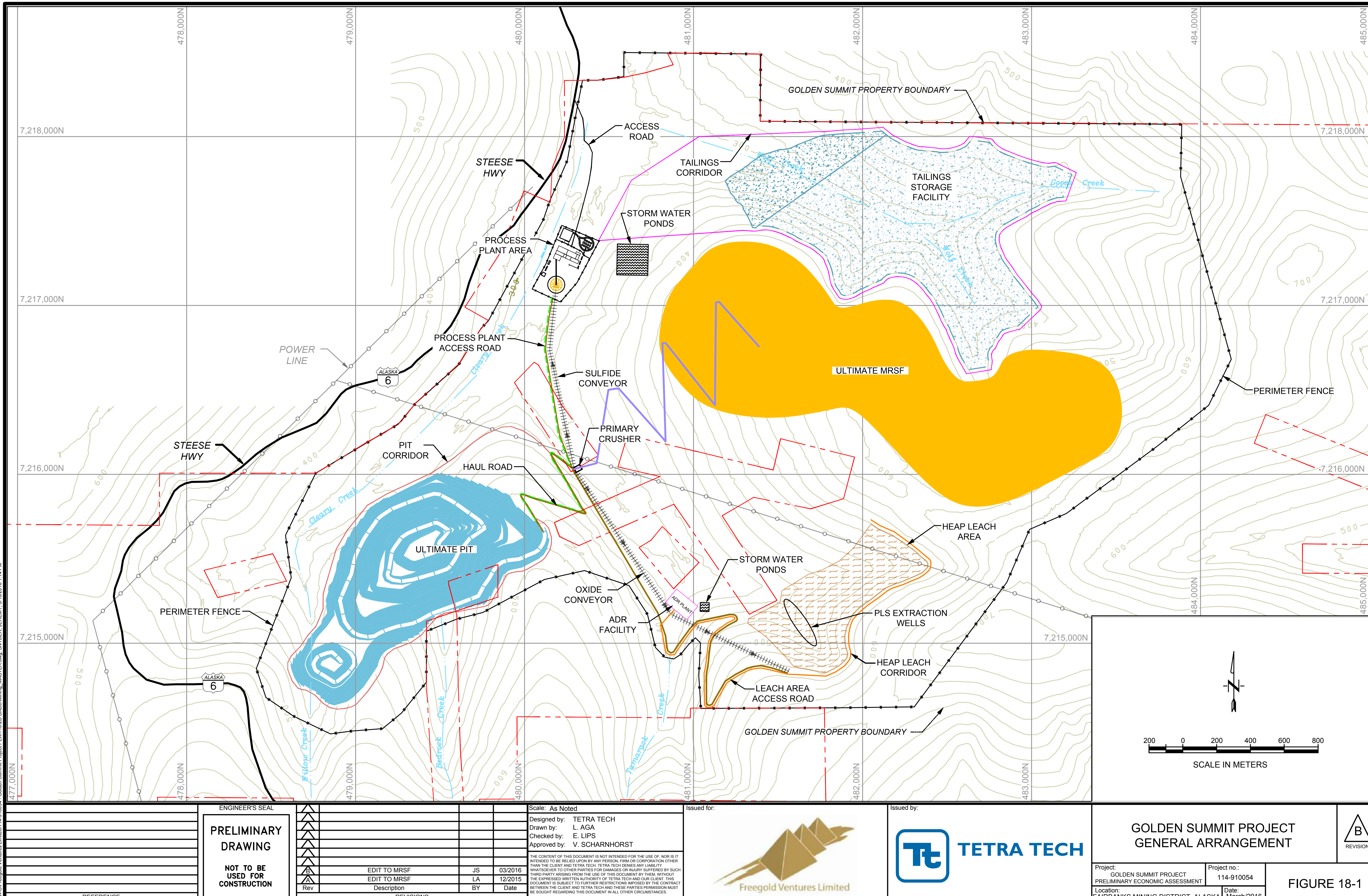
18.8 WATER MANAGEMENT

This section describes water management and required infrastructure for the Project. There are four types of water sources at the Property that would require management:

- Groundwater reporting to the open pit mine;
- Precipitation that would contact material associated with engineered facilities;
- Small streams to be diverted around the footprint of the facilities; and
- Stormwater runoff from surface disturbance areas.

Section 24 includes the methodology and analysis for surface water and groundwater hydrology; water balance; and geochemistry as well as design criteria. This section provides an overview of required facilities, and includes water supply, process water, fire/potable water, treatment of site wastewater and dewatering requirements.

The groundwater reporting to the open pit mine and the contact precipitation would be collected, treated and recycled for use in the processing facilities. It is expected, based on the preliminary assessment, that excess water would need to be released back into the environment. This expectation requires that the PEA include capital and operating costs for a wastewater treatment plant. The need and type of treatment facility would be determined during the feasibility study stage.



GOLDEN SUMMIT PROPERTY BOUNDARY
ACCESS ROAD
STEESE HWY
TAILINGS CORRIDOR
TAILINGS STORAGE FACILITY
STORM WATER PONDS
PROCESS PLANT AREA
PROCESS PLANT ACCESS ROAD
POWER LINE
SULFIDE CONVEYOR
ULTIMATE MRSF
PERIMETER FENCE
PRIMARY CRUSHER
PIT CORRIDOR
HAUL ROAD
HEAP LEACH AREA
ULTIMATE PIT
STORM WATER PONDS
OXIDE CONVEYOR
PLS EXTRACTION WELLS
ADR FACILITY
ADR PLANT
HEAP LEACH CORRIDOR
LEACH AREA ACCESS ROAD
GOLDEN SUMMIT PROPERTY BOUNDARY
Cleary Creek
Willow Creek
Bedrock Creek
Tamarack Creek
Wolf Creek
Cook Creek
ALASKA 6
7,218,000N
7,217,000N
7,216,000N
7,215,000N
477,000N
478,000N
479,000N
480,000N
481,000N
482,000N
483,000N
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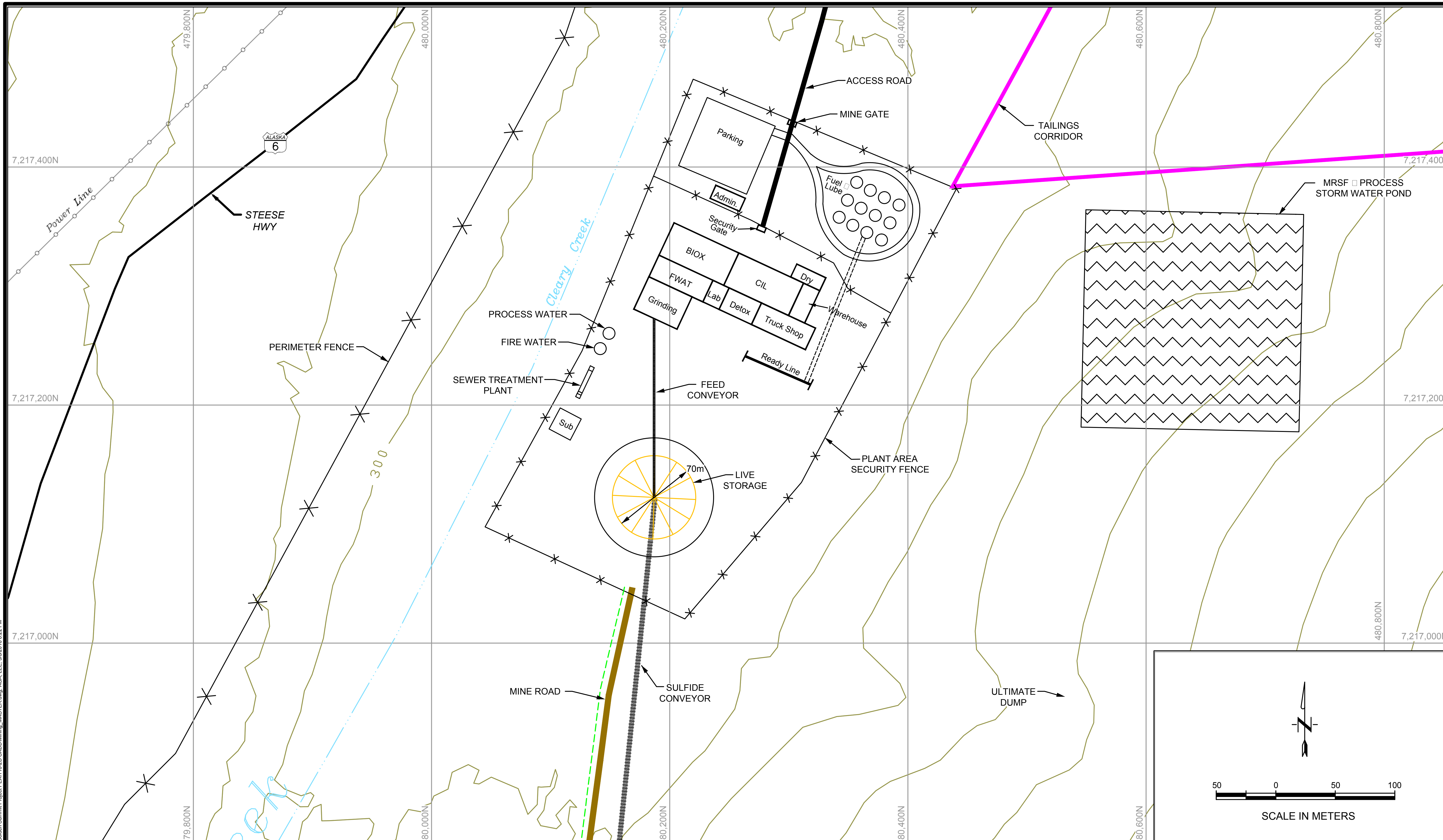
SCALE IN METERS
ENGINEER'S SEAL
PRELIMINARY DRAWING
NOT TO BE USED FOR CONSTRUCTION
B EDIT TO MRSF JS 03/2016
A EDIT TO MRSF LA 12/2015
Rev Description BY Date
REFERENCE
REVISIONS
Scale: As Noted
Designed by: TETRA TECH
Drawn by: L. AGA
Checked by: E. LIPS
Approved by: V. SCHARNHORST
Issued for:
Freegold Ventures Limited
Issued by:
TETRA TECH
GOLDEN SUMMIT PROJECT GENERAL ARRANGEMENT
B REVISION
Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT
Project no.: 114-910054
Location: FAIRBANKS MINING DISTRICT, ALASKA
Date: March/2016
FIGURE 18-1



REFERENCE	ENGINEER'S SEAL	Rev	Description	BY	Date
	PRELIMINARY DRAWING NOT TO BE USED FOR CONSTRUCTION	B	Edit to MRSF	L. AGA	Dec/2015

REVISIONS

Scale: As Noted

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Drawn by: L. AGA
Checked by: E. LIPS
Approved by: V. SCHARNHORST

Issued for: Freegold Ventures Limited

Issued by: TETRA TECH

GOLDEN SUMMIT PROJECT
SULFIDE PLANT AREA LAYOUT

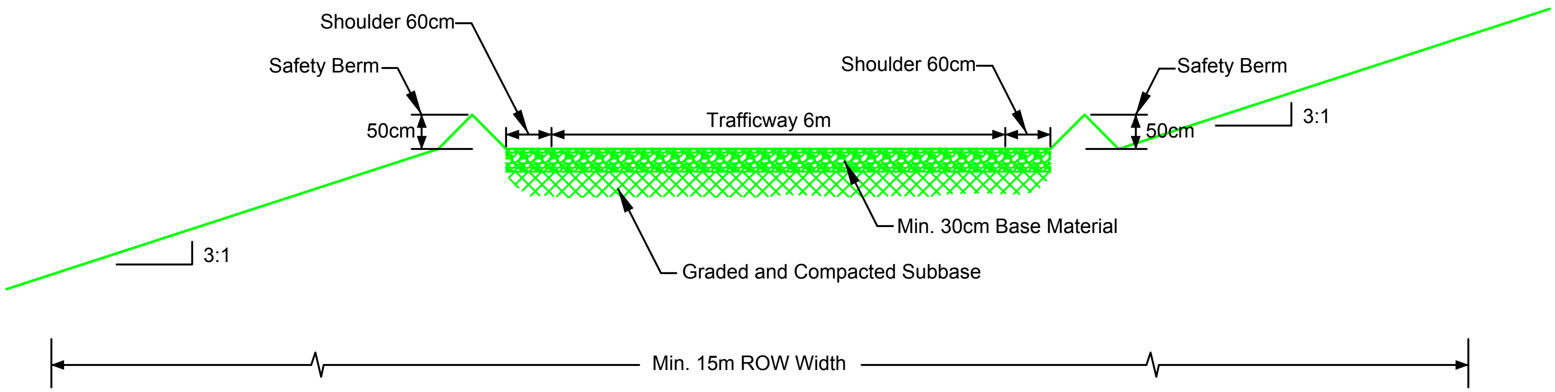
B
REVISION

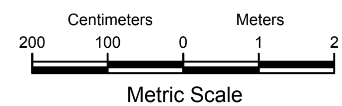
Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT
Project no.: 114-910054
Location: FAIRBANKS MINING DISTRICT, ALASKA
Date: Oct/2015

FIGURE 18-2

LOCAL ROAD DESIGN NOTES

- *Minimum Right Of Way = 15m*
- *Minimum Trafficway = 6m*
- *Minimum Shoulder = 60cm*
- *Minimum Base Material = 30cm*
  - *Replace top 10cm with surface course on grades >7%, 7.6cm on grades ≤7%*
- *Add 50cm tall safety berm*

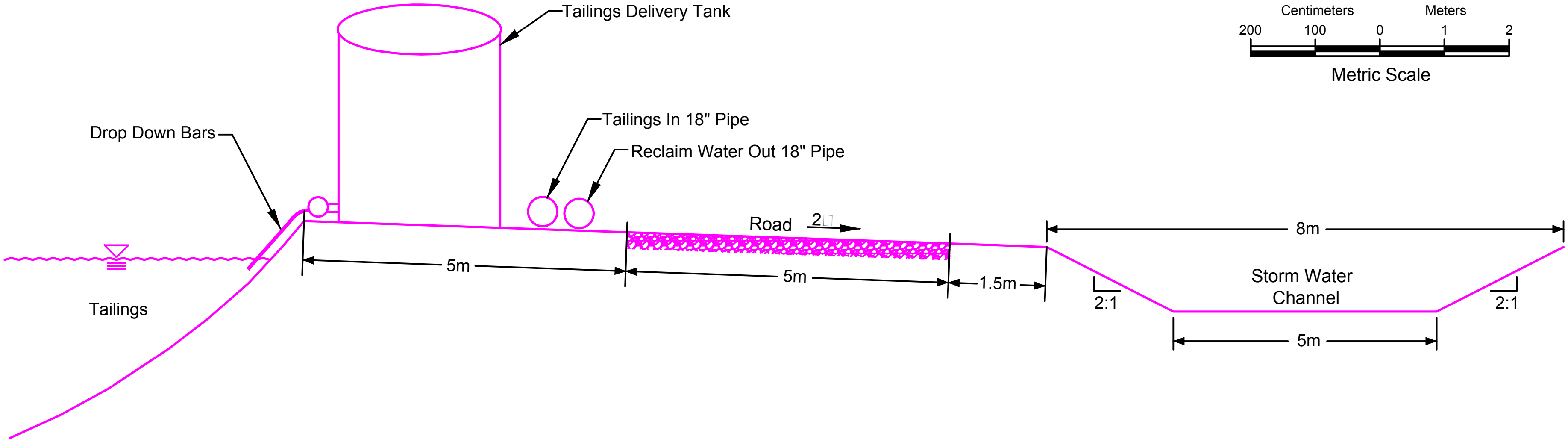




Designed by: TETRA TECH Drawn by: L. AGA Checked by: E. LIPS Approved by: V. SCHARNHORST	Issued for: Freegold Ventures Limited	Issued by: TETRA TECH	GOLDEN SUMMIT PROJECT LOCAL ROAD DESIGN SECTION A REVISION

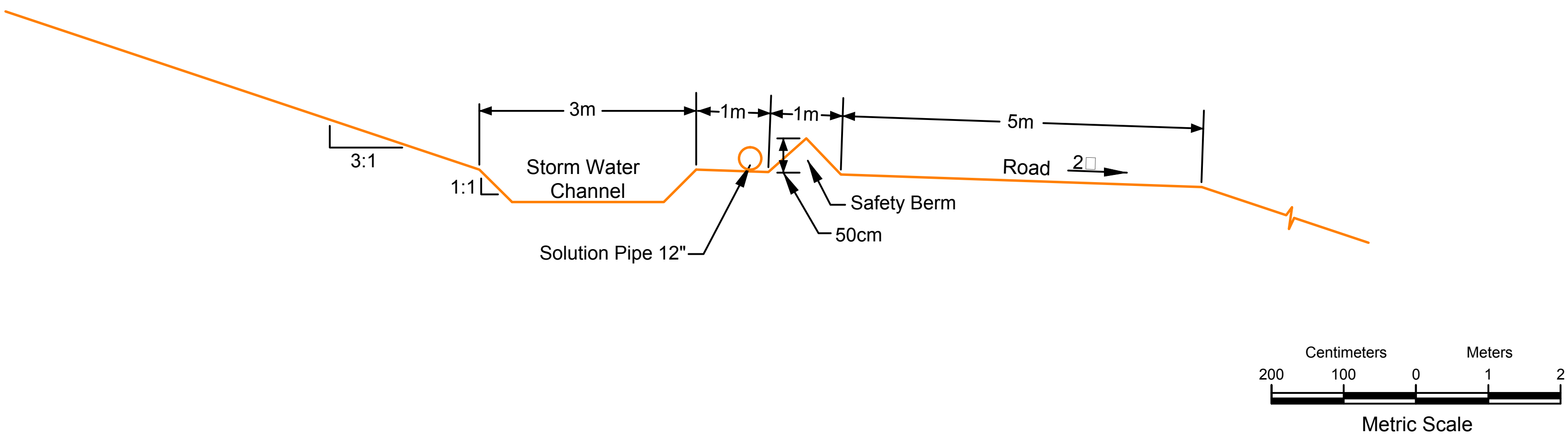
Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT	Project no.: 114-910054	FIGURE 18-3
Location: FAIRBANKS MINING DISTRICT, ALASKA	Date: Oct/2015	

TAILINGS CORRIDOR



Note: shifted/relocated depending on deposit strategy

HEAP LEACH CORRIDOR



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Checked by: E. LIPS
Approved by: V. SCHARNHORST

Issued for:

Freegold Ventures Limited

Issued by:

TETRA TECH

GOLDEN SUMMIT PROJECT
TAILINGS □ HEAP CORRIDOR SECTIONS

A
REVISION

Project: GOLDEN SUMMIT PROJECT PRELIMINARY ECONOMIC ASSESSMENT

Project no.: 114-910054

Location: FAIRBANKS MINING DISTRICT, ALASKA

Date: Oct/2015

FIGURE 18-4

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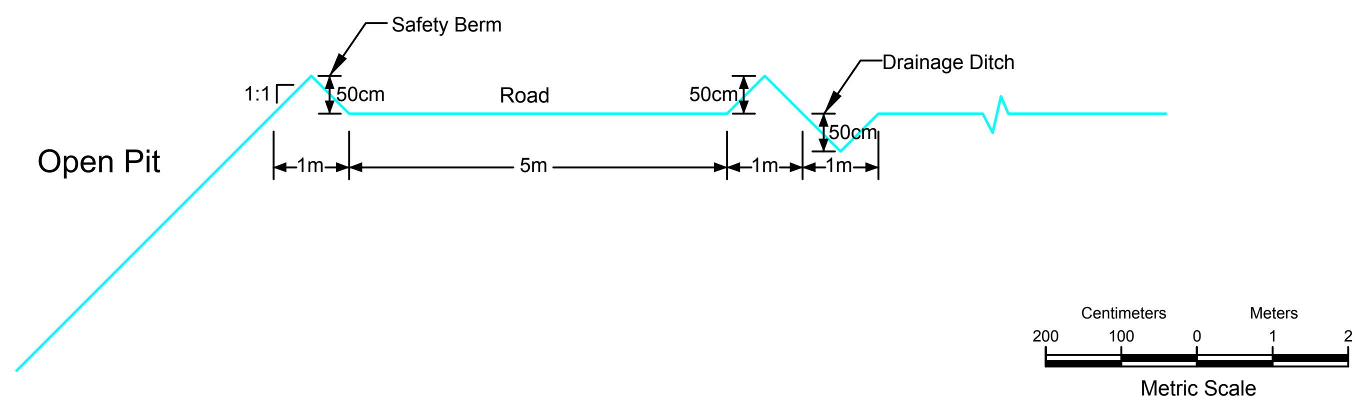
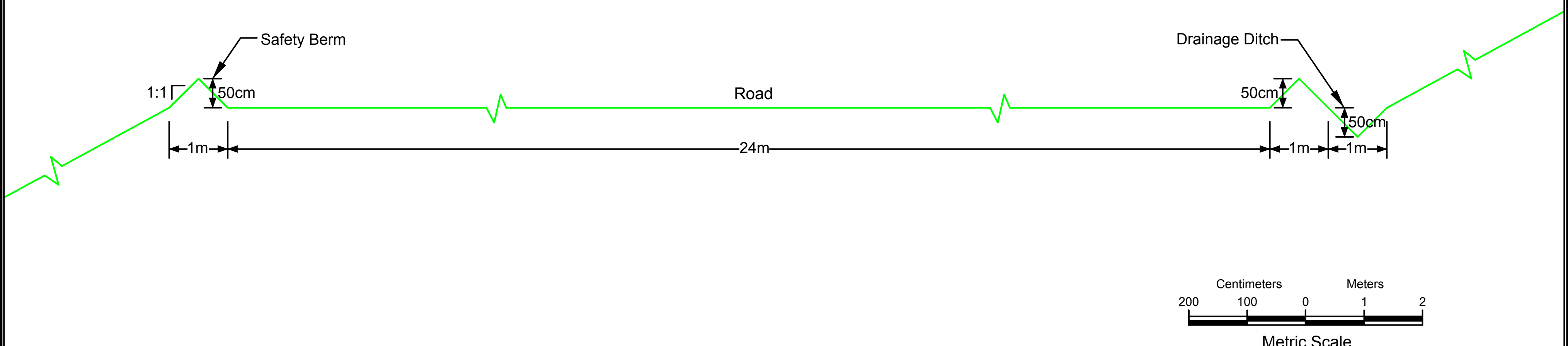
PIT CORRIDOR



Note: shifted/relocated depending on deposit strategy

HAUL ROAD CORRIDOR



Note: shifted/relocated depending on deposit strategy

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Drawn by:	L. AGA
Checked by:	E. LIPS
Approved by:	V. SCHARNHORST

Issued for:

Freegold Ventures Limited

Issued by:

TETRA TECH

GOLDEN SUMMIT PROJECT
PIT □ HAUL ROAD CORRIDOR SECTIONS

A
REVISION

Project:
GOLDEN SUMMIT PROJECT
PRELIMINARY ECONOMIC ASSESSMENT

Project no.:
114-910054

Location:
FAIRBANKS MINING DISTRICT, ALASKA

Date:
Oct/2015

FIGURE 18-5

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18.8.1 Surface Water Management

Surface water is divided into runoff from mine affected surfaces such as the TSF, MRSF, HLP and process plant (contact water), and runoff from natural surfaces (non-contact water). Non-contact water also includes management of natural streams entering or exiting the property. Non-contact water may be collected or diverted and released directly into natural systems downstream whereas contact water may require treatment prior to release. The overall goal of water management is to maintain separation between the two types of water so that treatment volumes are minimized and to protect the environment and site facilities.

Non-contact water would be diverted prior to encountering site facilities by constructed channels that would ultimately report to natural systems downstream. Similarly, contact water would be collected by a separate channel system but would report to ponds for detention, evaluation and possible treatment. Contact water may also be recycled back to the mill, for use as process water.

Channel and pond design is based on site rainfall and runoff evaluation and regulatory design basis. Given the lack of design criteria specific to gold mining operations, Alaska coal mining regulations were used as a basis of design.

Additional information regarding the methodology for storm water management is included in **Section 24.2.**

18.8.2 Water Supply

Raw water for processing would primarily be required for start-up and emergency purposes, gland seal water, reagent, and process water makeup.

Potable and fire water supply would be from groundwater wells.

A water use authorization would be required from the State of Alaska Department of Natural Resources, Division of Mining, Land and Water. Fairbanks Creek and Too Much Gold Creek are authorized for water appropriation of 8,000 gallons per minute (gpm) for placer mining under water right Certificate of Appropriation ADL 46157. Wolf Creek is authorized for water use of 10,000 gpm for placer mining under TWUP F2011-48. The Property would be required to coordinate water withdrawals with other companies and placer mine operations in the area that may be withdrawing water.

18.8.3 Process Water

Mineral processing requires an estimated 500 gpm of make-up water. Groundwater wells (described above), water from the pit, or diverted run-on would be used as sources for makeup water.

18.8.4 Fire / Potable Water

Fire demand, storage tank and distribution lines at the Property are estimated based upon the pressure, flow rates and volumes required for fire suppression as defined by the International Fire Code and NFPA 122. The water tank would either be stored within the process building or insulated if located outside, to prevent freezing. Duration of fire water use would be dependent upon the area of the process facility. Based on required water supply for fire suppression at the largest building (Process Facility), a minimum of 330,000 gallons of water would be maintained in the potable water supply tank to ensure a flow rate of 2,750 gpm for two hours of fire suppression. Water must not be used to suppress petroleum or chemical based fires.

Sprinkler systems would be required in facilities with areas greater than 12,000 square feet and/or heights of more than three stories. The process facility and mine truck shop would be constructed with automatic sprinkler systems designed to provide 0.18 gpm/ft^2 for fire suppression.

It is estimated that approximately 15,000 gallons of water would be required daily to satisfy potable water demand. A potable water tank (500,000 gallons was sized to allow for ample fire flow requirements) and a hydro-chlorination unit would be provided. The chlorination system was estimated with a flow rate of 500 gpm.

18.8.5 Waste Water Treatment

Sewage treatment and disposal for the estimated 500 site employees would consist of a packaged wastewater treatment facility. The plant would be manufactured off-site and containerized for simple connection to the collection system on site.

This plant would be sized to treat domestic wastewater as well as excess water from the pit (after pre-treatment as required). The plant would meet secondary treatment requirements for the State of Alaska Department of Environmental Conservation (ADEC), 18 AAC Chapter 70 - Water Quality Standards.

For the purposes of this study, a treatment plant was sized for a flow of 25,000 gallons per day (gpd) and the following secondary treatment effluent limits (**Table 18-2**).

Table 18-2: Secondary Treatment Effluent Limits (Excerpt from State of Alaska, as described above)

Table B-1: Secondary Treatment Effluent Limits [18 AAC 72.990(59) and 40 CFR §133.102 as adopted by reference at 18 AAC 83.010(e)]

Parameter	Average Monthly Limit	Average Weekly Limit	Maximum Daily Limit	Range
BOD_5	30 mg/L	45 mg/L	60 mg/L	---
TSS	30 mg/L	45 mg/L	60 mg/L	---
Removal Rates for BOD_5 and TSS	85% (minimum)	---	---	---
pH	---	---	---	6.0 – 9.0 s.u.

In addition, the regulations in 18 AAC 83.540 require that effluent limits meet mass-based limits for copper, lead and ammonia. The regulation at 18 AAC 83.520 requires that effluent limits be calculated based on the design flow of the facility.

Once treated, the plant effluent would be discharged to Cleary Creek (considered a “non-salmon-bearing stream” in the regulations) in accordance with an Alaska Pollution Discharge Elimination System (APDES) permit.

18.8.6 Dewatering

Hydrogeologic conditions at the Project site are discussed in **Section 24.1**.

Planned open pit mining at the property would extend below the water table, and dewatering would be required for maintaining pit wall stability and dry conditions within the pit. Considering winter temperatures, dewatering by means of wells would be the most feasible strategy. Data from the dewatering well system at the Fort Knox mine were used to estimate dewatering requirements for open pit mining at the property.

Specific capacity is a term used to denote the relationship between pumping rate and water-level drawdown in a well. If a constant drawdown is maintained in the well, the pumping rate needed to maintain that drawdown, and thus specific capacity, would decrease gradually with time. This concept can be applied to a dewatering system. The "specific capacity" of the dewatering system at the Fort Knox mine was estimated from reported pumping rates for the dewatering system (FGMI 2006, 2008, 2011, 2012, 2013, 2014) and estimated depths of the mine pit at various points in time. The time-varying "specific capacity" estimated for the Fort Knox mine dewatering system was applied to the Project mine plan, based on the changing depth of the planned open pit below an assumed water table elevation. The initial water table elevation in the planned pit area was assumed to be approximately 450 m (1,475 ft) amsl, the approximate elevation at which the floor of the Willow Creek valley intersects the planned mine pit.

The estimated pumping rates and number of dewatering wells that would be required to depress groundwater levels to below the pit floor are summarized in **Figure 18-6**. The number of wells is shown for two scenarios, the first based on an average pumping rate of 545 m^3/day (100 gpm) per well, plus one backup well for every five dewatering wells, and the second based on 218 m^3/day (40 gpm) per well, plus one backup well for every ten dewatering wells. The 545 m^3/day (100 gpm) rate is based on typical well yields listed on Fort Knox well construction and testing records obtained from ADNR (2014) for Fort Knox dewatering wells; the 218 m^3/d (40 gpm) rate is based on the annual pumping rate and number of wells listed in the Fort Knox 2010 annual activity report (FGMI, 2011), the year of highest reported annual inflow to the pit. The Project mine pit would intersect the water approximately six months to one year after the start of mining, but dewatering would need to start earlier in order for the pumping effects to extend throughout the required area. The estimated annual average pumping rate was approximately 410 m^3/day (75 gpm) initially, increased to approximately 4,460 m^3/day (818 gpm) by the third year of mining, declined slightly through the eighth year of mining, and then increased gradually to approximately 6,600 m^3/day (1,210 gpm) near the end of the mine life.

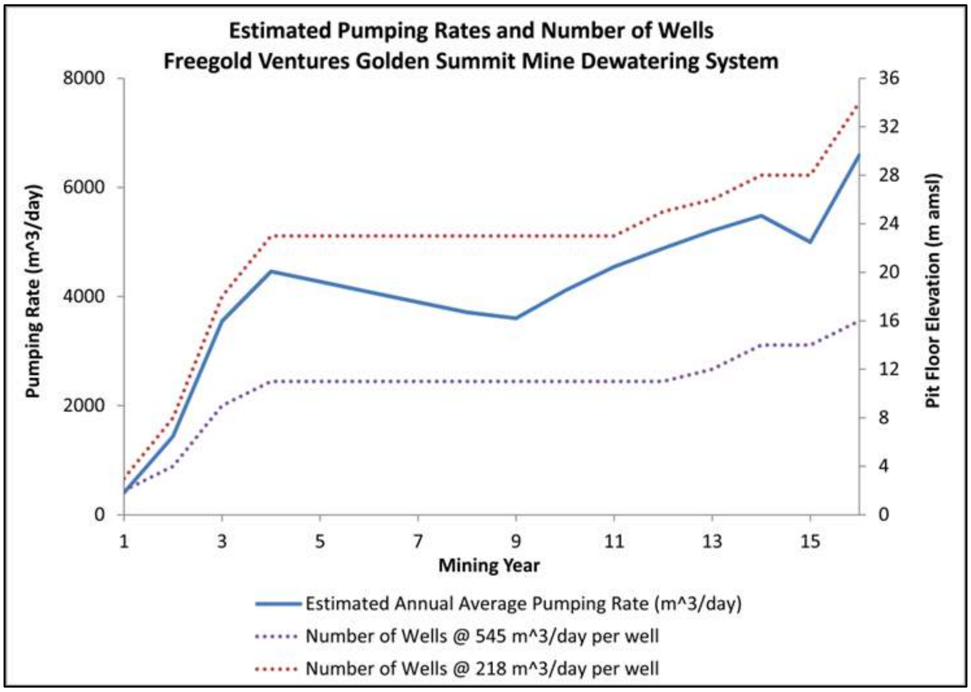


Figure 18-6: Estimated Pumping Rate and Number of Dewatering Wells

The number of wells that would be required for effective dewatering through the LoM was estimated based on a combination of the total estimated pumping rate, the length of pit perimeter, and the average pumping capacity of a dewatering well at the Fort Knox mine. The number of wells would increase as the pit is enlarged and deepened. Two wells would be required initially. That number would increase to 11 by the second year of mining, remain steady through the tenth year of mining, and then increase to 16 by the final year of mining. The number of wells includes at least one backup well throughout the mine life.

The cost of dewatering was estimated based on the cost of a typical dewatering well 200 m (656 ft) in total depth, cased with 20.3-cm (8-inch) diameter steel casing and mill-slotted well screen and equipped with a submersible pump capable of pumping approximately 550 m^3/day (100 gpm) from the total depth of the well. The average total depth of the wells assumes that wells would be installed on benches within the mine pit whenever possible, thereby limiting the required drilling depth.

18.9 TAILINGS STORAGE FACILITY

18.9.1 Design Requirements and Concept

The Tailings Storage Facility (TSF) was designed to accommodate the nominal 49.6 Mt of sulfide material to be processed between Year 9 through Year 24. The mill throughput would ramp up to a nominal rate of 3.5 Mtpa.

A cross valley type conventional slurry TSF concept was adopted based on the mine plan and assessment of the site topography. The TSF was sited north of the proposed waste dump location in the Wolf Creek valley.

The valley storage design was established to permit storage of 38 Mm^3 assuming an average settled tailings dry density of 1.3 t/m^3 and including an allowance for freeboard and tailings beach slopes.

The TSF embankment would be raised in three stages and constructed of mine waste rock material. The total volume of the embankment at full capacity is 13.3 Mm^3. The final crest elevation of the TSF would be 405 m.

A summary of TSF design requirements and characteristics is provided in **Table 18-3**.

Table 18-3: TSF Requirements and Characteristics

TSF Feature	Value
Tailings Storage Capacity	38 Mm^3
Tailings Storage Capacity	49.6 Mt
Tailings Storage Capacity	16 years up to 3.5 Mtpa
Embankment Crest Elevation (Final)	405 m
Embankment Volume (Final)	13.3 Mm^3
Maximum Embankment Height	100 m

18.9.2 Design and Construction

The TSF embankment would be constructed in stages by downstream methods using mine waste rock and select borrow material as required. The staged raises would be constructed to accommodate storage requirements and mine waste rock production. The zoned embankment would include a low permeability compacted clayey zone keyed into competent and low permeability foundation, a random fill zone, and rockfill blanket drain. The clayey zone would be protected from freeze-thaw cycles by the random fill zones and suitable cover at each stage of construction. A liner is then placed on top of the upslope interior random fill zone.

The crest width of 15 m was adopted to accommodate maintenance equipment access, windrows, and the tailings slurry pipeline. The adopted embankment design slope is 3H:1V downstream and 2.5H:1V upstream to suit typical stability and closure requirements.

A nominal 3m thick rockfill drainage blanket shall be installed below the downstream portion of the embankment to improve downstream drainage and maintain a low phreatic surface in the embankment.

Additionally, a geomembrane liner with an underliner and overliner drain system will be installed in order to collect ground water seepage and TSF seepage, respectfully.

Surface water diversion ditches will be required to divert surface water from the storage area.

The TSF footprint would be grubbed and topsoil stripped and stockpiled for future reclamation. The TSF basin area would be ripped, moisture conditioned, and compacted in place to create a low permeability layer and reduce infiltration to groundwater.

An access road would be constructed around the facility perimeter to facilitate installation of the tailings slurry pipeline and provide access to the water return.

18.9.3 Operation

The tailings slurry would be deposited from the embankment and along the perimeter of the storage area. This would optimize tailings storage capacity while reducing the risks associated with embankment stability and seepage. Tailings deposition would be undertaken to maintain the decant water return pond adjacent to the south valley wall. Decant water would be returned to the process plant for re-use. Heat traced and insulated pipelines and storage tanks would be required to mitigate cold weather operation risks.

The following outlines practices important in the optimization of the TSF:

1. The water pond size shall be kept to a minimum by optimizing water return.

2. Deposition should be cycled in such a manner as to concentrate and maintain the water pond around the water recovery point located in the valley area of the storage.

3. The supernatant water should not be allowed to pond against the embankment.

4. TSF Monitoring.

The TSF monitoring program would include the embankment stability, tailings storage management, and groundwater quality.

Embankment stability would be monitored by routine visual inspections and periodic measurements of slope inclinometers, survey stakes, and standpipe and/or vibrating wire piezometers.

Tailings management would be monitored by routine visual inspection by operations and management staff as well as annual audits by geotechnical specialists.

Piezometers would be installed to permit monitoring of groundwater flow and quality.

18.9.4 Closure

The conceptual closure plan involves covering the top surface of the TSF with overburden and revegetating the surface and embankment. The revegetation technique that is adopted would be based on site specific trials and experience.

A spillway would be required to facilitate controlled release of surface runoff from significant storm events.

18.10 Heap Leach Facility

18.10.1 Design Requirements and Concept

The proposed heap leach facility is a valley fill concept located adjacent to the Mine Waste Rock Facility (MRSF) and in the Chatham Creek valley. The heap leach facility was designed to accommodate the 47,864 Mt of oxide material to be treated in the current mine plan. This translates to a required minimum capacity of 27,350 million m^3 of mineralized material assuming an average dry density of 1.75 t/m^3 in the heap.

The heap leach facility would utilize an in-heap storage pond for the collection of the pregnant solution. This approach was selected to reduce operational risks in the cold environment.

The facility design incorporates surface water diversion features and a lined containment system with a network of overdrainage pipework for solution collection. The facility would be constructed in stages to suit the production schedule. The footprint was selected to suit site geometry with consideration of lease limits and proposed mine infrastructure. An underdrain system would be in place to collect any groundwater seepage.

Material for the heap leach would be transported from the open pit to the heap leach facility via conveyors. Material would be loaded on the pad in lifts of nominal 12 m thickness at a rate of 3.5 Mt per annum.

The design was developed based on the environmental setting, state of practice design requirements, and similar operations close to the Project site. Geotechnical and environmental site investigation was not undertaken as part of this preliminary design.

18.10.2 Design and Construction

18.10.2.1 IN-HEAP STORAGE POND

The heap leach facility would include an in-heap storage pond to eliminate surface exposure of the process solution. The in-heap storage pond would be sized to contain:

1. Solution storage for operations;
2. Solution from a 24-hour draindown;
3. Runoff from design storm event; and
4. Freeboard as per Alaska dam safety requirements.

Earthworks for the in-heap storage would include a toe embankment and contouring of the storage pond basin to promote drainage towards the proposed return pump wells. The in-heap embankment would be constructed of mine waste and select borrow as required, with slopes of 3H:1V with a 15m wide crest.

The basin fill would be moisture conditioned and compacted. The surface would be contoured for a minimum 2% slope toward the proposed return pump wells.

18.10.2.2 COLLECTION AND CONTAINMENT DESIGN

The leach pad would be fully lined to facilitate effective pregnant solution recovery and mitigate impact to the environment.

Topsoil in the basin would be stripped and stockpiled for future reclamation. Any soils deemed unsuitable for the foundation of the facility would also be removed and stockpiled. This would include removal of any local and discrete permafrost zones that may be present to mitigate risks associated with differential settlement due to permafrost melt. Following stripping, a subbase soil layer would be prepared. This subbase would be ripped, moisture conditioned and compacted to create a low permeability layer. The compacted surface would then be covered with a Linear Low Density Polyethylene (LLDPE) liner. The LLDPE liner was selected for its strength, chemical resistance and performance in cold environments.

The area below the in-heap storage pond would include a double synthetic liner system with a Leakage Collection and Recovery System (LCRS) between the liners. The double synthetic liner would be comprised of two LLDPE liners with a geocomposite drainage layer in between. Any seepage collected by the LCRS would report to a sump at the upstream toe of the in-heap embankment. It would then be pumped back to the storage pond.

Above the liner system, the entire footprint of the leach pad would be overlain with 1 meter of crushed mill reject material. This 1 meter of crushed material (the overliner) would consist of less than 1" size rock with a network of perforated piping. The purpose of this overliner is to convey collected pregnant solution to the storage pond and solution collection wells. It would also serve to protect the synthetic liner from damage during material loading on the heap. The piping network would consist of perforated corrugated double walled collection pipes.

18.10.2.3 UNDERDRAIN AND DIVERSION DITCHES

A network of underdrains would be installed to capture and transport flow from seepage areas below the heap leach facility. The underdrains are designed with a primary function of removing seepage from below the liner system, therefore process solution is not anticipated to drain to these underdrains. Flow would be released unless indications of process solution are identified through monitoring. Monitoring of the underdrain would provide a performance review of the lining system.

Diversion ditches would be constructed and lined with run of mine rock around the active stages of the heap leach facility to convey any surface water runoff around the facility.

18.10.3 Operation

18.10.3.1 HAULING AND LOADING

The heap leach pad would be constructed over a period of 17 years (Mine Year 1 to Year 17). Loading would occur in 12 m lifts. The operation of mining and hauling material to the pad is anticipated to occur year-round at a rate of 3.5 Mt per annum. The material is to be conveyed from the open pit to the heap leach pad.

The bench face angle of the heap leach facility would be 37.5^{o} while the overall slope is to be 18.5^{o}.

18.10.3.2 SOLUTION MANAGEMENT

The solution would be applied via drip emitters, drop emitters, or sprinklers. The method used would depend on the season. The drip emitters would be utilized during the cold winter months and would be buried under 5 feet of material. In the summer months either drop emitters or sprinklers would be used. The solution applied would then flow through the material to the in-heap storage pond.

Once the solution is applied and allowed to flow through the heap to the in-heap storage pond, the pregnant solution would then be recovered via collection wells. There would be five pregnant solution collection wells located at the lowest portion of the in-heap storage pond. Three of the five wells would be in use at any given time with the others on standby. The pumping rate would closely match the application rate. The wells can be run simultaneously during storm events. However, the application rate of the barren solution would be reduced during these wet conditions to mimic the typical operational levels.

Barren and Pregnant solution would be pumped between the heap leach pad and the plant in double lined pipes. The requirements for pumping and maintaining the required head to pump the barren solution to the top of the heap leach pad would vary as the size of the heap leach facility grows. Pumps would need to be added over the life of the mine.

18.10.4 Closure

The Heap Leach Facility closure concept would involve residual leaching until uneconomic recovery is achieved, followed by solution recirculation/rinsing to destroy cyanide and meet compliance standards.

At closure, the facility would be re-graded and growth media placed as required to create a stable landform. The seepage and quality of minor long-term seepage would be monitored.

18.10.5 Cold Weather Considerations

Year-round leaching operations in the cold climate are considered feasible assuming design provisions are incorporated for adding and maintaining heat in the process solutions applied to the heap and adequate operational methods are adopted. This would include adjustments to the heap loading schedule based on air and rock temperature monitoring to prevent formation of ice lenses within the heap, cross ripping of cells before leaching to break up frozen and/or compacted ground, burial of solution emitter lines, and heat tracing and insulation of solution tanks and pipelines as required.

These measures are adopted because frozen material on a heap leach pad is detrimental to the operation. This is due to the loss of effective percolation resulting in reduced recovery and possible heap instability from lateral solution flows to the heap slopes.

The proposed valley fill heap leach construction with internal pond and pump recovery wells has the advantage of limited heat loss from solution as compared to a design with an external pregnant solution pond. The approach may result in relatively higher construction cost and reduced operational control as compared to a design with an external solution pond.

19.0 MARKET STUDIES AND CONTRACTS

There were no market studies conducted and no contracts in place between Freegold and refiners at this time. Freegold plans to establish refining agreements with third-party entities for refining of doré.

In the future, Freegold will negotiate refining contracts and sales agreements that are typical and consistent with standard industry practice, and similar to contracts for doré elsewhere in the global market.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section characterizes the existing environmental baseline data for the Project area, makes suggestions for additional studies that would provide a basis for the mine permitting efforts, describes the major environmental permits that would likely be required for the Project, and identifies potential significant social or community impacts.

20.1 ENVIRONMENTAL STUDIES

The Project area lies within the Cleary Creek watershed and in addition to Cleary Creek, includes the drainages of Willow Creek, Bedrock Creek, Chatham Creek, Fairbanks Creek, Too Much Gold Creek, and Wolf Creek. The Cleary Creek basin is tributary to the Chatanika River. To date, a limited amount of baseline environmental data have been collected in the Project area to characterize water resources, water quality, wetlands, aquatic resources, on-site meteorology, subsistence use and cultural resources. An evaluation under Section 106 of the National Historic Preservation Act (NHPA) concerning the historic status of a former ski area within the Project area has been conducted by the Alaska State Historic Preservation Office (SHPO). Additionally, an initial evaluation of waste rock geochemistry has also been conducted in support of this PEA.

20.1.1 Historic Evaluation of Former Ski Area

The Cleary Summit ski area was located in the Willow Creek watershed and operated between 1949 and 1993. The area, which extended both above and below the Steese Highway, consisted of several buildings, lodges, and ski tows that were associated with the ski area over the years of operation. The buildings, debris and equipment were removed or sold after the area closed in 1993. The State of Alaska SHPO performed a NHPA Section 106 review of the ski area and former operations to determine if it would be eligible for listing under the National Register of Historic Places. Alaska SHPO found that the ski area was not eligible for listing as an historic place. As a result of this finding, the Project will not need to consider avoidance or mitigation the former property in locating facilities or for mine operations.

20.1.2 Initial Geochemical Characterization of Waste Rock

The mineralization described in **Section 7.2** indicates the presence of sulfide material, including arsenopyrite. Sulfide waste material has the potential to generate acid upon exposure to both oxygen and water. Acid drainage is often associated with metal leachate generation that requires management to reduce impact to water resources. As the sulfidic material is arsenopyrite, arsenic is predicted to be released upon weathering of the waste rock. A geochemical testing program was initiated to evaluate the potential for acid drainage and metal leachate to be generated.

Twenty one (21) representative waste rock samples were selected from available core for initial geochemical characterization of waste rock. The samples represented spatially and vertically distributed drill core primarily obtained from the Dolphin Deposit. Seven samples were analyzed for mineralogical quantification. Twenty-one samples were analyzed for constituent mobility using the Synthetic Precipitation Leaching Procedure (SPLP), acid generating and neutralization potential using acid-base accounting (ABA), and net acid generating (NAG) testing to determine the resulting pH after complete oxidation of sulfide minerals. Results from these initial evaluations are as follows:

- Two of the seven samples had measurable pyrite based on mineralogical quantification with percentages of 0.2 percent (%) and 1.4 %. Acid neutralizing minerals, primarily calcite, were observed in three of the seven samples with concentrations ranging from 4.4 % to 20.4 %. Ankerite, another acid neutralizing mineral, was present in two of the seven samples with concentrations of 0.3 % and 1.3 %.
- Total elemental arsenic in excess of 1,000 parts per million (ppm) was reported in six of the 21 samples across various lithologies. Total elemental lead in excess of 1,000 ppm is reported in two of the 21 samples from the granodiorite rock type.
- Predominately, the pH of leachate generated by the SPLP was slightly alkaline to alkaline. In all, 20 of the 21 samples report values above the upper Reference Value pH threshold of 8.5. A total of six samples report arsenic concentrations above the Reference Value of 0.15 mg/L. A correlation with total element arsenic concentrations (exceedences of 1,000 ppm) is observed in four of the six samples. A total of five of the 21 samples reported iron concentrations above the Reference Value of 1.0 mg/L. Isolated exceedences of copper, lead, and zinc were also reported from the granodiorite rock type.
- A total of six of the 21 samples were classified as potentially acid generating (PAG) across a range of rock types. NAG pH results show a wide range of values between 2.8 to 11.0, with three of the 21 samples reporting a value less than 4.5. Insufficient acid neutralization capacity exists in the majority of the represented samples to counteract acidity that may theoretically be generated as a result of weathering processes. Appreciable acid neutralizing potential is observed in only two of the 21 samples.

20.1.3 Further Environmental Study Requirements

Development of the Project would require extensive environmental baseline analyses, assessment of environmental impacts and evaluation, and associated permitting requirements reflective of the direct, indirect and cumulative impacts associated with full project build-out, and the environment in which it would be constructed. Development of the Project would entail significant infrastructure development including the mine, mill, tailings impoundment and ancillary facilities, as well as any off-site infrastructure such as power transmission and road improvements. The complexity of the environmental impact review and permitting of the various facilities would be dependent on siting of facilities in relationship to the various creeks and valleys surrounding the Project development target areas. This PEA provides preliminary siting information of facilities such as the open pit, tailings disposal, waste rock, and leach pad.

Baseline environmental data would be required for this Project including on-the-ground studies to delineate jurisdictional wetlands. These data would be required to meet a number of needs including permitting and mine design and location of facilities, mine construction and operations. Freegold has initiated consultation with the State's Large Mine Permitting Team (LMPT) to begin the process of project planning, development and environmental permitting. Through this process, the LMPT would assist in developing a broader environmental baseline program.

Owing to the long lead time to collect data, it is important that the baseline program generates adequate data in terms of type, quality and quantity. For example, defined baseline needs would include characterization of surface water resources, including type, flow, and water quality. Groundwater baseline sampling down gradient from proposed tailings and waste rock storage facilities would be important and should be initiated as soon as those areas have been tentatively identified through the feasibility process. Groundwater pump tests should be performed within the proposed limits of the open pit. Collection of meteorological data would need to be implemented to delineate local variations in wind and precipitation and for air quality permitting. These data may need to be initially complemented by snow surveys, depending on the site wide water balance which would be better defined as the feasibility study process continues. Other surveys or studies would also be required including birds and wildlife. Permafrost studies would likely be required for foundation and dam designs.

The characterization program would need to extend beyond the anticipated footprint of the Project to provide hydrologic, hydrogeologic and water quality data that is representative of background conditions downstream of proposed operations. Monitoring of established sites would be required by regulatory agencies both during mine operations and after closure. Agencies often require evaluations of alternative sites for waste rock and tailings storage, so hydrology and water quality at feasible alternative sites should also be characterized. All of these data are important to the development of an accurate environmental baseline and water balance for the Project area.

20.2 PERMITTING

20.2.1 Exploration Permit

Freegold performs mineral exploration at the Project under State of Alaska Land Use Permit #9726. The permit covers exploration activities through 2016. These activities are bonded through the State Reclamation Bond Pool under bond Application for Permits to Mine in Alaska (APMA) #9726 for a proposed disturbance area of 65 acres. A portion of this bond may be returned after reclamation of disturbed areas is completed and approved by the Alaska Department of Natural Resources (ADNR) Division of Mining, Land, and Water.

20.2.2 Temporary Water Use Permit

Freegold currently conducts site activities and exploration under a State of Alaska Temporary Water Use Permit (TWUP F2011-133). The permit defines allowable takeout points on Cleary Creek, Chatham Creek, Wolf Creek, Fairbanks Creek, and Too Much Gold Creek. The current permit expires in 2016.

20.2.3 Wetlands Permit

Exploration activities are also permitted under a federal Clean Water Act Section 404 Permit (POA-2007-510-M1, Cleary Creek) authorized by the United States Army Corps of Engineers (USACE) for the discharge of approximately 76,450 cubic meters (100,000 cubic yards) of gravel fill into 17 hectares (41.8 acres) of jurisdictional wetlands. The permit was required for anticipated impacts to waters of the U.S. that would result from the exploration activities, including the development of access roads, drill pads, and drillholes. The permit has been issued through 2018.

20.2.4 Required Major Mining Permits

ADNR requires approval of a Reclamation & Closure Plan and bond assessment, prior to mine construction. ADNR also grants certificates to construct and then operate a dam (tailings and water storage) and issues Water Use Authorizations. ADNR further requires approval of a Plan of Operations (PoO) which is normally required when a mine project is situated at least partially on State lands. Typically the PoO consists of the project description, reclamation & closure plan, water, waste rock and tailings management plans and monitoring plans, and may contain additional information such that it may simultaneously satisfy the application requirements of other permits.

The Alaska Department of Environmental Conservation (ADEC) would require an Integrated Waste Management Permit, air permits for construction, then operations, and an Alaska Pollution Discharge Elimination System (APDES) Permit for the discharge of wastewater. Discharges of stormwater to surface waters would be regulated under the state Multisector Stormwater General Permit (MSGP). ADEC would also be required to provide a federal Clean Water Act (CWA) Section 401 Certification for the CWA Section 404 permit (see further discussion below).

The Alaska Department of Fish & Game would require permits for any culverts that need to be placed in fish-bearing streams or other impacts to fish-bearing streams and fishery habitats.

An underground injection control (UIC) permit from ADEC and U.S. Environmental Protection Agency (USEPA) would be needed if underground injection would be used to dispose of wastewater.

USACE would require a CWA Section 404 permit for dredging and filling activities in "waters of the U.S.," including jurisdictional wetlands. This federal permitting action requires USACE to comply with the National Environmental Policy Act (NEPA) and, for a project of this magnitude, the preparation of an Environmental Impact Statement (EIS) would likely be required. The USACE would likely serve as the lead agency for the NEPA process. The NEPA process would require consultation and coordination with additional federal agencies, such as the U.S. Fish and Wildlife Service and EPA, as well as with Alaska SHPO and Tribal Governments under Section 106 of the Historical and Cultural Resources Protection Act. Additional studies or surveys would be required to support preparation of an EIS, including traditional knowledge and subsistence use, noise, visual resources, and socioeconomics. A more detailed discussion of permitting requirements under CWA Section 404 and NEPA is provided in **Section 20.3.5** and **Section 20.3.6**, respectively.

The overall timeline required for permitting would largely be driven by the time required for the NEPA process, which would be triggered by the submission of the Section 404 permit application to the USACE. The NEPA process is completed with a Record of Decision, following publication of the final EIS. In Alaska, the EIS and permitting processes are generally coordinated so that permitting and environmental review under NEPA occurs in parallel.

A list of major potential permits that could be required is provided in **Table 20-1** below. Several other minor federal, state, and local permits would be required depending on specific facility operations.

Table 20-1: Potential Required Major Permits for the Project

Government Entity	Permit
State of Alaska	Plan of Operations Approval/Reclamation Plan Approval
	Upland Mining Lease
	Water Use Authorization (Water Right)
	Reclamation Bond
	Certificates to Construct/Operate Dam(s)
	Fish Passage and Habitat Permits
	Discharge Permit (treated waste water) to surface water (APDES)
	Stormwater Management Permit (MSGP)
	Integrated Waste Management Permit (Tailings and Waste Rock)
	Air Quality Permit, both during construction and then operation under Title V of the Clean Air Act (CAA)
Federal Government	CWA Section 404 Dredge and Fill Permit (Wetlands)
	Spill Prevention Control, and Countermeasures Plan under the CWA
Local – Fairbanks North Star Borough	Master Plan Approval

20.2.5 Clean Water Act Section 404 Wetlands Permit

The major environmental driver for the Project would be the issuance of a CWA Section 404 Permit, issued by USACE for the purpose of authorizing the placement of fill into wetlands and Waters of the U.S. The permit authorizes the placement of "clean fill" for use as necessary in Waters of the U.S. for the construction of facilities, bridges, roads, or for the storage of wastes such as tailings. The Section 404 permit application is required to include the following information: a description of the activities that require Section 404 permitting, description of the fill material, a determination of impacts on the aquatic ecosystem, information on alternative disposal sites and locations, and a Mitigation Plan. The Mitigation Plan defines how the Project has avoided or minimized impacts to waters of the U.S. and wetlands to the extent practicable and identifies the mitigation proposed for wetland impacts which are unavoidable. The plan would need to cross reference the Reclamation and Closure Plan of the PoO as part of the minimization and mitigation discussion. Depending on the nature of the proposed mitigation, a wetlands monitoring plan may also need to be developed. Mitigation is driven to an extent by the functions and values provided by the existing wetland types. The USACE requires a functions/values assessment as part of the baseline data collection. Recent guidance in the USACE's approach to mitigation calls for a focus on in lieu fee programs where fees are paid to wetland managers who then obtain deed restrictions or conservation easements to protect wetlands from development pressures.

The USACE cannot issue the Section 404 permit for the Project until the Project attains NEPA compliance. When the USACE is a lead (or cooperating) agency, it develops the EIS in parallel with the Section 404 permit. Typically the USACE requires a draft 404 permit application to trigger the NEPA process. A final application is not desired since it is anticipated that the actions requiring permitting and/or the impact analysis supporting the application would be modified through the NEPA process.

20.2.6 Air Quality Permit

The Project is located outside the $PM_{2.5}$ nonattainment area boundary for the Fairbanks area. Regardless of the project's final potential-to-emit (PTE) and source classification, ADEC retains the discretionary authority to require an applicant to conduct an Air Quality Impact Analysis (AQIA) (i.e. dispersion modeling) as part of the permitting process. In order to conduct the AQIA, meteorological and ambient air quality data are required, in addition to the engineering and emissions data that is obtained from the facility design. The adequacy of any meteorological and ambient air quality monitoring and permitting program is determined by ADEC who requires approval of the location and type of monitoring equipment installed, as well specifying Data Quality Objectives (DQO) for meteorological and background pollutant data.

In general, ADEC requires at least 12 months of continuous monitoring with a DQO of data logging of 90 percent of the time within each quarter prior to construction (80 percent of each quarter for background pollutants). It is not uncommon for meteorological and pollutant monitoring programs in Alaska to have difficulty meeting these DQOs at remote sites, often resulting in project delays while the necessary data are obtained for permitting. In addition, obtaining more than a single continuous year of data can result in less stringent permit requirements generated through the AQIA. Consultation with ADEC is required to obtain approval of siting a proposed met station, the equipment used as well as approval of a Quality Assurance Project Plan (QAPP). The goal is to obtain multiple years of applicable data to be used in permitting.

Should generator sets be used as a power source for a period of time, they would be considered for inclusion in the air permitting process and regulatory compliance for the project. Based on the potential size of the generator sets, 2 MW, specific federal requirements for the generator sets themselves are likely to be required. These requirements may include:

- Code of Federal Regulations Title 40 Part 60 Subpart IIII—Standards of Performance for Stationary Compression Ignition Internal Combustion Engines, and
- Code of Federal Regulations Title 40 Part 63 - Subpart ZZZZ—National Emissions Standards for Hazardous Air Pollutants for Stationary Reciprocating Internal Combustion Engines.

Specific requirements will depend on variety of factors including the fuel burned (i.e., diesel or LNG) and the anticipated operating characteristics and power generation needs of the mine. Operating generators of this size could produce a significant amount of emissions (particularly if fueled by diesel), and could trigger federal operating permit requirements, commonly referred to as Title V. In addition, this alternative might potentially trigger federal prevention of significant deterioration (PSD) permitting requirements as a major source. As the project further advances through the prefeasibility and feasibility process, a regulatory applicability assessment can be conducted to iteratively compare power generation needs and mine operations with potential emission inventories to further evaluate regulatory drivers (i.e. major versus minor emitters) and assess financial impact.

20.2.7 National Environmental Policy Act

NEPA and the Council of Environmental Quality (CEQ) Regulations at 40 CFR 1500-15008 would govern the federal environmental permitting process for the Project. Before the USACE makes a decision on whether or not to issue a CWA Section 404 permit for the Project, the Project would need to comply with NEPA, including preparation of an EIS or an Environmental Assessment (EA).

The EIS/EA evaluates and discloses the projected impacts of the Project across a reasonable range of alternatives. It is likely that USACE would serve as the lead NEPA agency as the primary federal action would include issuance of a CWA 404 permit. The EIS/EA would also serve as a vehicle to support a required CWA Section 404(b)(1) analysis. Under Section 404(b)(1), Freegold Ventures would be required to demonstrate through the PoO how project planning:

1. First considered avoidance of wetlands altogether;
2. Second, that the project alternatives considered minimization of impacts to wetlands, this includes potential alternative placement of facilities, such as tailings impoundments; and
3. Third, how impacts would be mitigated.

Other agencies would have roles in reviewing the EIS as cooperating agencies. For large mining projects in Alaska, the state and federal agencies are proficient at coordinating the baseline data and analysis requirements for NEPA and permitting.

There are no standard guidelines regarding the specific the amount of baseline information and analysis needed to prepare an EIS/EA. However, much of the data required for permitting and approvals described above are also required and used to support the preparation of the EIS or EA.

20.3 Social or Community Requirements

The Project is located approximately 18 miles northeast of the city of Fairbanks and is within the Fairbanks North Star Borough (FNSB). The population of Fairbanks is approximately 32,324 and the population of the FNSB as a whole is approximately 100,000 (2013 Census). The Project has the potential to positively impact work opportunities and socioeconomics in the area and provide economic growth within the interior of Alaska.

Potential impacts, real or perceived, to hunting, fishing, and recreational opportunities for the local population would likely result in some public opposition to the Project. It is anticipated that there would be concerns voiced by local environmental groups and the operators of the Skiland ski area (Mount Aurora) which is located immediately south of the Project. Local community concerns would be formally recognized during the scoping stage at the beginning of the NEPA process. At that time, the lead federal agency would hold scoping meetings and record concerns in order to address significant issues during the preparation of the EIS. Early and continued community engagement and government affairs programs by Freegold would aid in minimizing these concerns.

20.4 Mine Closure and Reclamation

Mine reclamation and closure would be conducted pursuant to reclamation and closure plans developed as regulated by ADNR Office of Project Management and Permitting (OPMP) under the permitting-related requirements of state law AS 27.19. Preliminary reclamation and closure plans would serve as the basis for Freegold's financial assurance obligations to the DNR Mining, Land and Water (MLW) Mining Section.

The preliminary reclamation plan evaluates the necessary reclamation measures that would be conducted on-site during and after mining to minimize impacts to the surrounding area.

20.4.1 Plant Growth Medium Salvage

The Project mine site is composed of Ester-Gilmore Complex, Steese, Ester Peat, and existing mine disturbance (NRCS, 2015) and thus provides varying levels of salvageable plant growth medium (PGM) for future use in reclamation of mine features. To the extent practicable, up to approximately 0.3 meters (12 inches) of PGM would be salvaged from the mine site prior to mining for use as seed bedding material during reclamation. Sensitive vegetation would be transplanted for use during revegetation.

PGM salvage would consist of scraping and excavating any salvageable PGM from disturbance footprints prior to construction of mine features. PGM would be salvaged from the pit Mine Rock Storage Facility (MRSF), Tailings Storage Facility (TSF) and Heap Leach Pad (HLP) disturbance areas and stockpiled at the toe of the MRSF. Approximately 0.3 meters (12 inches) of PGM would be salvaged from mine feature disturbance areas.

The Project mill site is composed of primarily disturbed soils from previous mining activities (NRCS, 2015). As a result no surface soil would be salvaged in the disturbed areas at the mill site. Any sensitive vegetation would be transplanted during PGM salvage activities for use during revegetation.

20.4.2 Revegetation

Project site vegetation is characterized primarily by forested land. Areas disturbed by previous mining activity are sparsely vegetated. Disturbed areas would be revegetated during reclamation and closure to prevent erosion, and improve soil and slope stability. Disturbed areas to be reclaimed include the MRSF, TSF, HLP, and process facility yard.

To the extent practicable, sensitive species of vegetation would be collected prior to and during PGM salvage activities. Salvaged vegetation would be transplanted to PGM stockpiles for preservation until reclamation commences at each facility. Transplanted vegetation would serve to prevent erosion of the PGM stockpiles during mining operations, and reclaimed mine facilities after reclamation.

Revegetation would be accomplished with a native seed mix applied by approved methods. The seed mix is to be genetically pure and certified from a source adapted to the project area. Acceptable species of vegetation include grass species native to the area such as Artared Fescue, Guening Alpine Bluegrass, Tundra Glaucous Bluegrass, and Nortran Tufted Hairgrass. Seeding would be completed during the late spring months through about mid-July. Due to the generally steep slopes of the mine site, hydroseeding inclusive of a tackifier would be the recommended seeding method.

20.4.3 Erosion Control

Erosion from bare and disturbed areas would be minimized during reclamation activities. To prevent erosion and sedimentation caused by surface water runoff, silt fence and hay bales would be installed perpendicular to slopes along down-gradient edges of disturbed areas. Additional berms and diversions would be constructed as necessary to manage surface water during precipitation events. Surface water management is discussed in **Sections 18** and **24**.

Water trucks would be used to spray disturbed areas during reclamation, minimizing the potential for wind erosion. Dust suppression would likely be a continuous need throughout reclamation.

20.4.4 Mine Rock Storage Facility

A MRSF would be constructed during pre-stripping and pit development. The MRSF would primarily contain surface stripping waste generated during pit development. The MRSF would be constructed to a height of approximately 200 feet at 3(H):1(V) slopes and cover approximately 310 hectares during life of mine operations. PGM would be salvaged from the MRSF footprint to the extent practicable prior to construction, as previously discussed.

No regrading of the MRSF slopes would be required during reclamation because the MRSF would be constructed at a 3(H):1(V) slope during mining operations. Minor regrading may be required to create uniform slopes on the MRSF. Because the MRSF would be composed primarily of stripping waste from pit development, the MRSF surface is assumed to be sufficiently coarse with large cobbles to protect against erosion both during mining and after reclamation regrading. Upon closure the MRSF surface would be covered by approximately 0.3 meters (12 inches) of PGM salvaged from the pit and MRSF footprint. Placed PGM would then be ripped to prepare a suitable seed bedding surface and revegetated using native seed mix as previously discussed.

20.4.5 Open Pit

The Open Pit would cover a disturbance footprint of approximately 100 hectares during the life of mine. PGM would be salvaged from the pit footprint to the extent practicable prior to construction. Discussion of the open pit development is provided in **Section 16**.

After mining, the pit area would be protected with the construction of an access prevention berm around the pit perimeter. The pit perimeter berm would also serve to prevent surface water drainage from entering the pit, directing surface water around the berm and down native slopes. The berm would be constructed to a height of approximately 3 meters with a crest width of approximately one meter and side slopes of approximately 2(H):1(V). The berm would be covered with approximately 0.3 meters (12 inches) of PGM and revegetated as discussed in **Section 20.4.2**.

20.4.6 Access and Service Roads

Mine site access roads and facility service roads would not be reclaimed upon the cessation of mining activities. These roads would be left in place to allow access to the site and to reclaim mine features for post-closure monitoring and maintenance activities. Maintenance activities may be required on access and service roads throughout the monitoring and maintenance period to ensure vehicle access to reclaimed mine features.

20.4.7 Process Facility

The Process Facility would be located within a fenced yard area at the mine site. Process equipment is estimated to cover a disturbance area of approximately 28 hectares. The process area would include the mill building, administration building, mine truck shop, change house, warehouses, material stockpile, parking lots, and laboratory. No PGM would be salvaged from the process yard area due to its location within areas of historic mine disturbance.

Reclamation of the Process Facility would include decommissioning of all processing equipment. To the extent feasible, used equipment would be salvaged and sold. In the estimate it was assume that costs for equipment decommissioning would be offset by the salvage value of the used equipment.

Walls of process facility structures would be demolished and demolition debris would be disposed of as at the MRSF or a landfill as appropriate. Concrete from conveyor and building foundations and slabs would be demolished and rubblized. Rubblized concrete would be hauled from the mill site to the MRSF prior to MRSF closure.

Fences installed around the process area would be left in place to prevent access to the site and provide continued security to minimize public health and safety risks.

The mill site area would be regraded, covered with a 0.3 meter (12 inch) PGM cover, ripped to prepare a suitable seed bedding surface, and revegetated using a native seed mix.

20.4.8 Tailings Storage Facility

Tailings produced at the mill site would be stored in an impounded tailings storage facility (TSF) at the north end of the mine site. The TSF embankment would be constructed at a slope of approximately 3(H):1(V) to a height of approximately 100 meters, and would cover approximately 140 hectares during operations. PGM would be salvaged from the MRSF footprint to the extent practicable prior to construction. Further details regarding TSF construction are provided in **Section 18**.

At closure the following steps would be completed to minimize erosion potential at the TSF.

1. Minor grading of the embankment slope to create uniform 3(H):1(V) slopes.
2. Bridging of impounded tailings with approximately 0.67 meters (2 feet) of non-PAG rock material
3. Cover of rock bridging material in the impoundment and the entire embankment area with approximately 0.3 meters (12 inches) of salvaged PGM
4. Ripping of PGM to prepare suitable seed bedding surface
5. Revegetation of placed PGM with native seed mix.

20.4.9 Heap Leach Pad

Oxide material at the project would be placed in a Heap Leach Pad (HLP) as discussed in Section 18. The HLP would be constructed to a height of approximately 140 meters and cover a disturbance footprint of approximately 55 hectares. The HLP would be constructed with slopes of approximately 1.3(H):1(V) with benches at 12 meter intervals. Upon cessation of mining activities the following steps would be completed to reclaim and minimize erosion potential at the TSF.

1. Regrading the HLP slopes to uniform slopes of approximately 3(H):1(V)
2. Placement of PGM to a thickness of approximately 0.3 meters (12 inches) of salvaged PGM
3. Ripping of PGM to prepare suitable seed bedding surface
4. Revegetation of placed PGM with native seed mix.

20.4.10 Process Water Ponds

Process water storage ponds would be constructed to contain contact water from the TSF, HLP, and MRSF. Design details for these process water ponds are discussed in Section 24. Closure of process water ponds would be conducted upon cessation of mining activities. Closure activities would include cutting and folding of HDPE liner, burial of liner in place, and backfilling of the process water pond depressions using bermed material from pond excavation. Backfilled material would be ripped to prepare a suitable seed bedding surface and would be revegetated with a native seed mix as previously discussed.

20.4.11 Tailings Slurry, TSF Reclaim, Barren Solution and PLS Pipelines

Pipelines would be installed to direct tailings slurry from the process facility to the TSF, tailings reclaim water to the TSF contact water pond, and tailings reclaim water from the pond to the process facility. Additional pipelines would be installed at the HLP to direct barren solution from the ADR facility to the HLP and PLS from the HLP to the ADR. All TSF and HLP pipelines would be removed upon cessation of mining activities. Reclamation activities would include excavation and removal of buried pipelines and removal of pipelines installed at the surface. Pipe segments would be tested for contamination levels and disposed of at the MRSF or at a hazardous waste landfill as determined by contamination characterization.

20.4.12 Storm Water Ponds

Stormwater ponds would be constructed throughout the facility to manage non-contact surface water and run-on stormwater. Stormwater and non-contact surface water pond design details are provided in **Section 24**. Non-contact surface water ponds would be left in place upon reclamation and closure of the mine site to provide continued surface water management at the site.

20.4.13 Monitoring and Maintenance

Ongoing monitoring and maintenance would be conducted at the Project to ensure effective implementation of reclamation construction. A comprehensive site monitoring and maintenance plan would be developed and implemented as future work at the Project. Revegetation progress, cover stability, and erosion control measures would be routinely inspected, and maintenance actions would be taken in areas appearing vulnerable to erosion and instability. Maintenance and monitoring actions would continue at the site until reclamation metrics are achieved.

21.0 CAPITAL AND OPERATING COSTS

All costs and economic results are presented in Q4 2015 U.S. dollars. Quantities and values are presented using metric units unless otherwise specified. No escalation has been applied to capital or operating costs.

21.1 PRINCIPAL ASSUMPTIONS

Parameters used in the Technical Economic Model (TEM) analysis are shown in **Table 21-1**. These parameters are based upon current market conditions, vendor quotes, design criteria developed by Tetra Tech, and benchmarks against similar existing projects.

The construction schedule accounts for four years of pre-production activities. The operations are planned to operate at a rate of 10 ktpd for oxide material and 10 ktpd for sulfide materials; where sulfide mining starts in year nine of mine operations.

Table 21-1: TEM Principal Assumptions

Description	Parameter	Unit
General Assumptions		
Pre-Production Period	4	years
Mine Life	24	years
Operating Days	365	days/year
Production*	3,650	ktpy
Market Assumptions		
Price		
Gold	$1,300	$/oz
Payable Metal		
Gold	2,308	koz
Deductions		
Gold Deduction	0.1%	
Transport & Insurance	$4.00	$/oz
Refining	$3.00	$/oz
Financial Assumptions		
Private Royalty	2.0%	
Federal Income Tax	35.0%	
State Income Tax	9.4%	
Property Tax	1.3%	
Mining License Tax	7.0%	
Alaska Production Royalty	3.0%	
Technical Assumptions		
Diesel	$3.00	$/gal
Electric	$0.13	$/kWh
Recovery		
Heap Leach	80%	
Bioxidation	90%	

*Applicable to each phase individually (oxide and sulfide).

Projected revenue from the sale of gold is based upon a market price of $1,300/oz-Au. There is a gold deduction of 0.1% to recovered gold, transport and insurance is $4.00/oz-doré, refining is $3.00/oz-doré and a private royalty of 2.0%.

The Project will be subject to a 35% federal income tax, a 9.4% Alaska state income tax, a property tax of 1.3%, a mining license tax of 7%, and an Alaska production royalty of 3%.

Diesel fuel price used is $3.00/gal. Electric power costs are $0.130/kWh, which include provision for demand and energy charges.

Metallurgical testwork supports the assumed oxide and sulfide material recovery rates of 80% and 90%, respectively.

21.2 Life of Mine Production

21.2.1 Open Pit Mining

Mining will commence with the open pit production of the gold deposit. The RoM open pit production totals 97 Mt. Open pit production will have a 2.45:1 strip ratio over the 24-year LoM. Production over the LoM is summarized in **Table 21-2**.

Table 21-2: LoM Production

Description	Value	Unit
Waste	239	Mst
Oxide Tonnes	48	Mst
Sulfide Tonnes	50	Mst
Total	337	Mst
Stripping Ratio	2.45	waste:feed
Grade		
Oxide	0.54	g/t
Sulfide	1.14	g/t
Contained Metal (Au)	2,660	koz

21.2.2 *Processing*

The Project process oxide material by heap leach and sulfide material with bioxidation. See **Section 17.0** for specific process procedures. Each recovery method is designed to process 10 ktpd. LoM mill feed is shown in **Table 21-3**. Over the Project life, 2,358 koz of doré will be produced.

Table 21-3: RoM Mill Feed

RoM Feed	Value	Unit
Contained Metal		
Oxides	837	koz
Sulfides	1,823	koz
Total	2,660	koz
Recovered Gold		
Heap Leach	670	koz
Bioxidation	1,641	koz
Total	2,310	koz
Doré Produced	2,358	koz

21.3 CAPITAL COSTS

LoM capital cost requirements are estimated at $437 million as summarized in **Table 21-4**. Initial capital of $88 million is required to commence operations and a sustaining capital of $348 million.

Table 21-4: LoM Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
Direct Costs				
10	Mining	$39,744	$110,784	$150,528
20	Crushing & SAG Mill Circuits	$3,921	$9,884	$13,805
30	Heap Leach (Oxide)	$11,410	$23,723	$35,133
40	Process Plant (Sulfide)	$0	$27,894	$27,894
50	Tailings Storage Facility	$0	$67,774	$67,774
60	Infrastructure	$10,131	$11,000	$21,131
70	Construction	$12,095	$56,903	$68,998
	Direct Costs	**$77,301**	**$307,962**	**$385,263**
Indirect Costs				
800	Construction Indirects	$456	$2,232	$2,688
810	Spares & Inventory	$342	$1,674	$2,016
820	First Fills	$342	$1,674	$2,016
830	Freight & Logistics	$799	$2,789	$3,588
840	Commissioning & Start-Up	$342	$1,674	$2,016
850	EPCM	$1,369	$4,184	$5,553
860	Vendor & Consulting Assistance	$228	$1,116	$1,344
	Indirect Costs	**$3,879**	**$15,342**	**$19,221**
90	Owner's Costs	$7,240	$24,984	$32,224
	Total Capital	**$88,420**	**$348,288**	**$436,708**

The open pit mine utilizes some leased mobile equipment. Leases are capitalized during the preproduction period, then reported in the operating costs during the production.

21.3.1 10 – Open Pit Mining

LoM open pit capital cost requirements are estimated at $151 million as summarized in **Table 21-5**. Initial capital of $40 million is required to commence operations and a sustaining capital of $111 million.

Table 21-5: Open Pit Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
100	Capitalized Costs	$14,360	$2,578	$16,938
110	Mobile Equipment	$6,906	$103,820	$110,726
120	Facilities	$2,225	$0	$2,225
130	Mine Services	$16,252	$4,387	$20,639
	Total	**$39,744**	**$110,784**	**$150,528**

21.3.2 20 – Crushing Circuit

LoM crushing circuit capital cost requirements are estimated at $14 million as summarized in **Table 21-6**. Initial capital of $4 million is required to commence operations and a sustaining capital of $10 million.

Table 21-6: Crushing Circuit Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
200	Heap Leach	$3,921	$0	$3,921
210	Sulfide Plant	$0	$9,884	$9,884
	Total	**$3,921**	**$9,884**	**$13,805**

21.3.3 30 – Heap Leach (Oxide)

LoM heap leach capital cost requirements are estimated at $35 million as summarized in **Table 21-7**. Initial capital of $11 million is required to commence operations and a sustaining capital of $24 million.

Table 21-7: Heap Leach Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
300	Site Preparation	$3,253	$23,723	$26,976
310	HLP Equipment	$3,773	$0	$3,773
320	ADR Plant	$2,278	$0	$2,278
330	Plant Services	$2,106	$0	$2,106
	Total	**$11,410**	**$23,723**	**$35,133**

21.3.4 40 – Process Plant (Sulfide)

LoM process plant capital cost requirements are estimated at $28 million as summarized in **Table 21-8**, and are only sustaining capital.

Table 21-8: Process Plant Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
400	Site Preparation*	$0	$0	$0
410	Grinding	$0	$5,719	$5,719
420	Flotation	$0	$2,551	$2,551
430	Bioxidation	$0	$4,140	$4,140
440	CIL Plant	$0	$4,296	$4,296
450	Ancillary Equipment	$0	$1,332	$1,332
460	Cyanide Detoxification	$0	$1,718	$1,718
470	Plant Services	$0	$2,098	$2,098
480	Structures	$0	$6,040	$6,040
	Total	**$0**	**$27,894**	**$27,894**

Accounted for in infrastructure.

21.3.5 50 – Tailings Storage Facility

LoM tailings storage facility capital cost requirements are estimated at $68 million as summarized in **Table 21-9**, and are only sustaining capital.

Table 21-9: Tailings Storage Facility Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
500	Earthworks	$0	$64,549	$64,549
510	Tailings Pumping & Piping	$0	$3,226	$3,226
	Total	**$0**	**$67,774**	**$67,774**

21.3.6 60 – Infrastructure

LoM infrastructure capital cost requirements are estimated at $21 million as summarized in **Table 21-10**. Initial capital of $10 million is required to commence operations and a sustaining capital of $11 million.

Table 21-10: Infrastructure Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
610	Structures	$1,186	$0	$1,186
620	Power Lines/Substations	$1,990	$11,000	$12,990
630	Water Management	$6,922	$0	$6,922
640	Communications	$33	$0	$33
650	Mobile Equipment*	$0	$0	$0
	Total	**$10,131**	**$11,000**	**$21,131**

*Accounted for in mining and process.

21.3.7 70 – Construction

LoM construction capital cost requirements are estimated at $69 million as summarized in **Table 21-11**. Initial capital of $12 million is required to commence operations and a sustaining capital of $57 million.

Table 21-11: Construction Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
700	Construction Labor	$3,423	$11,158	$14,581
710	Piping	$1,712	$11,158	$12,869
720	Electrical & Instrumentation	$2,510	$10,321	$12,831
730	Concrete	$1,712	$8,368	$10,080
740	Structural Steel	$2,054	$12,552	$14,606
750	Painting & Insulation	$685	$3,347	$4,032
	Total	**$12,095**	**$56,903**	**$68,998**

21.3.8 80 – Indirects

LoM indirect capital cost requirements are estimated at $19 million as summarized in **Table 21-12**. Initial capital of $4 million is required to commence operations and a sustaining capital of $15 million.

Table 21-12: Indirect Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
800	Construction Indirects	$456	$2,232	$2,688
810	Spares & Inventory	$342	$1,674	$2,016
820	First Fills	$342	$1,674	$2,016
830	Freight & Logistics	$799	$2,789	$3,588
840	Commissioning & Start-Up	$342	$1,674	$2,016
850	EPCM	$1,369	$4,184	$5,553
860	Vendor & Consultant Assistance	$228	$1,116	$1,344
	Total	**$3,879**	**$15,342**	**$19,221**

21.3.9 90 – Owner's Costs

LoM owner's capital cost requirements are estimated at $32 million as summarized in **Table 21-13**. Initial capital of $7 million is required to commence operations and a sustaining capital of $25 million.

Table 21-13: Owner's Capital Costs

Description		Initial ($000s)	Sustaining ($000s)	LoM ($000s)
900	Project Management	2,000	0	2,000
910	Environmental & Permitting	600	0	600
920	Mine Closure & Reclamation	0	17,834	17,834
930	Exploration & Infill Drilling	1,000	1,000	2,000
940	Engineering Studies	3,000	0	3,000
950	Legal	500	0	500
960	Insurance	140	6,150	6,290
	Total	**7,240**	**24,984**	**32,224**

21.4 Operating Costs

LoM operating costs are summarized in **Table 21-14**. Open pit mining costs, as reported in this table, do not include the lease costs. Lease unit costs are shown separately.

Table 21-14: LoM Operating Costs

Description	$/t-moved	$/t-Feed	$/oz-gold
Mining	$3.04	$10.56	$441.68
Mining Lease	-	$1.06	$44.53
Crushing Circuit	-	$0.91	$38.10
Heap Leach (Oxide)	-	$1.20	$50.18
Process Plant (Sulfide)	-	$4.44	$185.59
Tailings Storage Facility	-	$0.12	$4.96
Infrastructure	-	$0.31	$13.09
Direct Operating Cost	**-**	**$18.60**	**$778.13**
Property Tax	-	$0.15	$6.10
Mining License Tax	-	$0.57	$23.74
Operating Cost	**-**	**$19.31**	**$807.97**

Refining charges, transportation, and royalties are not included in the operating cost estimate.

21.4.1 General Operating Assumptions

The following operating parameters were used for development of operating hours, presented in **Table 21-15**.

Table 21-15: Operating Parameters

Description	Units	Feed	Waste	Mill
Annual Production Rate	kt/yr	3,650	10,950	3,650
Operating Days	day/yr	350	350	365
Shifts	shifts/day	2	2	3
Shift Length	hours/shift	12	12	8
Annual Hours	hours/year	8,400	8,400	8,760
Availability	%	85%	85%	85%
Available Hours	hours/year	7,140	7,140	7,446
Production	st/day	511	1,534	490

21.4.2 Labor Assumptions & Wages

Table 21-16 presents G&A labor assumptions and wages.

Table 21-16: G&A Labor

Area	Classification	Salary ($/yr)	Base Rate ($/hr)	Base Hours	Burden (%)	Burdened Rate ($/hr)	Annual Pay ($000s)
Project Support Functions							
65100	General Manager	$250,000	-	-	30%	-	$325,000
65100	Sr. Accountant	$110,000	-	-	30%	-	$143,000
65100	Accountant	$79,000	-	-	30%	-	$102,700
65100	Clerk	$47,500	-	-	30%	-	$61,750
65100	HR/HSE Manager	$86,200	-	-	30%	-	$112,060
65100	Purchasing Agent	$90,000	-	-	30%	-	$117,000
65100	Environmental Manager	$110,000	-	-	30%	-	$143,000
65100	Environmental Technician	$79,000	-	-	30%	-	$102,700
65100	IT Coordinator	$90,000	-	-	30%	-	$117,000
65100	IT Technician	-	$21.96	2,080	30%	$28.55	$59,380
65100	Training Coordinator	$79,000	-	-	30%	-	$102,700
65100	Administrative Support	-	$19.96	2,080	30%	$25.95	$53,972
65100	Security	-	$21.96	2,080	30%	$28.55	$59,380

Table 21-17 presents mining labor assumptions and wages.

Table 21-17: Mining Labor

Area	Classification	Salary ($/yr)	Base Rate ($/hr)	Base Hours	Burden (%)	Burdened Rate ($/hr)	Annual Pay ($000s)
Operations							
10200	Driller, blasthole	-	$33.80	2,080	30%	$43.94	$91,395
10200	Driller Helper, blasthole	-	$26.84	2,080	30%	$34.89	$72,575
10200	Blaster	-	$33.19	2,080	30%	$43.15	$89,746
10200	Blaster Helper	-	$27.18	2,080	30%	$35.33	$73,495
10200	Shovel Operator	-	$33.80	2,080	30%	$43.94	$91,395
10200	Wheel Dozer Operator	-	$26.84	2,080	30%	$34.89	$72,575
10200	Truck Driver	-	$23.30	2,080	30%	$30.29	$63,003
10200	Track Dozer Operator	-	$26.84	2,080	30%	$34.89	$72,575
10200	Loader Operator	-	$26.84	2,080	30%	$34.89	$72,575
10200	Grader Operator	-	$26.84	2,080	30%	$34.89	$72,575
10200	Water Truck Driver	-	$26.84	2,080	30%	$34.89	$72,575
10200	Dispatcher	-	$33.96	2,080	30%	$44.15	$91,828
10200	Laborer/Trainee	-	$19.96	2,080	30%	$25.95	$53,972
10200	VSA Operator	-	$26.84	2,080	30%	$34.89	$72,575
10200	VSA Laborer/Trainee	-	$19.96	2,080	30%	$25.95	$53,972
Maintenance							
10300	Heavy Equip. Mechanic	-	$36.05	2,080	30%	$46.87	$97,479
10300	Welder/Mechanic	-	$25.77	2,080	30%	$33.50	$69,682
10300	Electrician/Instrumentman	-	$33.10	2,080	30%	$43.03	$89,502
10300	Lubeman/PM Mechanic	-	$25.77	2,080	30%	$33.50	$69,682
10300	Tireman	-	$25.77	2,080	30%	$33.50	$69,682
10300	Machinist	-	$36.05	2,080	30%	$46.87	$97,479
	Crusher / Belt / Pump crew	-	$23.72	2,080	30%	$30.84	$64,139
	Utilityman	-	$33.96	2,080	30%	$44.15	$91,828
	Laborer/Trainee	-	$19.96	2,080	30%	$25.95	$53,972
	VSA Mechanic*	-	$25.77	2,080	30%	$33.50	$69,682
	VSA Laborer**	-	$19.96	2,080	30%	$25.95	$53,972
Maintenance							
10300	Production Superintendent	$126,900	-	-	30%	-	$164,970
10300	Mine Foreman	$98,600	-	-	30%	-	$128,180
10300	Maintenance Superintendent	$126,200	-	-	30%	-	$164,060
10300	Maintenance Foreman	$98,600	-	-	30%	-	$128,180
10300	Maint. Planner	$86,200	-	-	30%	-	$112,060
10300	Chief Engineer*	$120,600	-	-	30%	-	$156,780
10300	Sr. Mine Engineer*	$110,000	-	-	30%	-	$143,000
10300	Mine Engineer	$86,200	-	-	30%	-	$112,060
10300	Chief Geologist	$115,600	-	-	30%	-	$150,280
10300	Geologist	$79,000	-	-	30%	-	$102,700
10300	Equipment Trainer	$100,000	-	-	30%	-	$130,000
10300	Surveyor	$86,200	-	-	30%	-	$112,060
10300	Surveyor Asst.	$47,500	-	-	30%	-	$61,750
10300	Sampler	$47,500	-	-	30%	-	$61,750

Table 21-18 presents process and infrastructure labor assumptions and wages.

Table 21-18: Process & Infrastructure Labor

Area	Classification	Salary ($/yr)	Base Rate ($/hr)	Base Hours	Burden (%)	Burdened Rate ($/hr)	Annual Pay ($000s)
Heap Leach Operations							
	Plant Operator	-	$33.80	2,080	30%	$43.94	$91,395
	Plant Helper	-	$27.18	2,080	30%	$35.33	$73,495
	Refiner Operator	-	$33.80	2,080	30%	$43.94	$91,395
	Laborer	-	$19.96	2,080	30%	$25.95	$53,972
	Pad Operator	-	$30.42	2,080	30%	$39.55	$82,256
	Pad Helper	-	$27.18	2,080	30%	$35.33	$73,495
	Assayer	$56,500	-	-	30%	-	$73,450
	Sample Prep	$47,500	-	-	30%	-	$61,750
	Mechanic/Welder	-	$25.77	2,080	30%	$33.50	$69,682
	Mechanic Helper	-	$19.96	2,080	30%	$25.95	$53,972
	Electrician	-	$33.10	2,080	30%	$43.03	$89,502
	Plant Superintendent	$126,900	-	-	30%	-	$164,970
	Plant General Foreman	$98,600	-	-	30%	-	$128,180
	Shift Foreman	$98,600	-	-	30%	-	$128,180
	Clerk	-	$19.96	2,080	30%	$25.95	$53,972
	Plant Maintenance Superintendent	$126,900	-	-	30%	-	$164,970
	Maintenance General Foreman	$98,600	-	-	30%	-	$128,180
	Foreman	$98,600	-	-	30%	-	$128,180
	Metallurgist	$115,600	-	-	30%	-	$150,280
Plant Operations							
	Control Room Operators	-	$27.18	2,080	30%	$35.33	$73,495
	Crusher Operator	-	$27.18	2,080	30%	$35.33	$73,495
	Grinding Operators	-	$27.18	2,080	30%	$35.33	$73,495
	Flotation Operators	-	$27.18	2,080	30%	$35.33	$73,495
	Filter Operators	-	$27.18	2,080	30%	$35.33	$73,495
	Dryer Operators	-	$27.18	2,080	30%	$35.33	$73,495
	Assayers	$56,500	-	-	30%	-	$73,450
	Samplers	$47,500	-	-	30%	-	$61,750
	Laborers	-	$19.96	2,080	30%	$25.95	$53,972
	Mechanics	-	$25.77	2,080	30%	$33.50	$69,682
	Electricians	-	$33.10	2,080	30%	$43.03	$89,502
	Mill Superintendent	$126,900	-	-	30%	-	$164,970
	General Foreman	$98,600	-	-	30%	-	$128,180
	Maintenance Foreman	$98,600	-	-	30%	-	$128,180
	Plant Foreman	$98,600	-	-	30%	-	$128,180
	Senior Metallurgist	$115,600	-	-	30%	-	$150,280
	Metallurgist	$98,600	-	-	30%	-	$128,180
	Process Technician	-	$30.39	2,080	30%	$39.51	$82,175
	Instrument Technician	-	$33.10	2,080	30%	$43.03	$89,502
	Process Foreman	$98,600	-	-	30%	-	$128,180
Tailings Dam							
	TSF Operator	-	$27.18	2,080	30%	$35.33	$73,495

21.4.3 Mining Assumptions

Table 21-19 presents drilling and blasting parameters.

Table 21-19: Drilling & Blasting Parameters

Description	Units	Value
ANFO Price	$/kg	$1.23
Booster Price	$/hole	$5.32
Cap/Cord/Line	$/hole	$10.89

21.4.4 Process Assumptions

Table 21-20 presents heap leach process reagents.

Table 21-20: Heap Leach Process Reagents

Description	Consum. (lb/t Feed)	Unit Cost (US$/lb)	Total Cost (US$/year)	Unit Cost (US$/t Feed)
Lime	3.000	$0.15	$1,642,500	$0.45
Cyanide	0.350	$1.50	$1,916,250	$0.53
Carbon	0.010	$2.00	$73,000	$0.02
Total			**$3,631,750**	**$1.00**

Table 21-21 presents sulfide process reagents.

Table 21-21: Sulfide Process Reagents

Description	Consum. (lb/t Feed	Unit Cost (US$/lb)	Total Cost (US$/year)	Unit Cost (US$/t Feed
CuSO4	0.400	$2.00	$2,920,000	$0.80
Collector - AERO 407	0.100	$1.50	$547,500	$0.15
Collector - PAX	0.240	$1.50	$1,314,000	$0.36
MIBC	0.100	$1.75	$638,750	$0.18
Lime	4.000	$0.15	$2,190,000	$0.60
Cyanide	0.350	$1.50	$1,916,250	$0.53
Carbon	0.080	$2.00	$584,000	$0.16
NaOH	0.200	$0.30	$219,000	$0.06
Flocculant	0.100	$1.90	$693,500	$0.19
Total			**$11,023,000**	**$3.02**

Table 21-22 presents process supplies.

Table 21-22: Process Supplies

Description	Consum. (lb/t Feed	Unit Cost (US$/lb)	Total Cost (US$/yr)	Unit Cost (US$/t Feed)
SAG Mill Balls	1.00	$0.58	$2,104,225	$0.58
SAG Mill Liners			$300,000	$0
Cone Crusher Liners			$86,493	$0
Ball Mill Balls	0.75	$0.58	$1,591,856	$0.44
Ball Mill Liners			$300,000	$0
Total			**$4,382,574**	**$1.20**

21.5 Taxes & Royalties

21.5.1 Royalties

Royalties are estimated on NSR at a rate of 2%.

21.5.2 Taxes

<u>Federal Tax</u>

Corporate federal income tax is determined by computing and paying the higher of a regular tax or a Tentative Minimum Tax (TMT). If the TMT exceeds the regular tax, the difference is called the Alternative Minimum Tax (AMT). Regular tax is computed by subtracting all allowable operating expenses, overhead, depreciation, amortization and depletion from current year revenues to arrive at taxable income. The tax rate is then determined from the published progressive tax schedule (35% for the Project). An operating loss may be used to offset taxable income, thereby reducing taxes owed. A 3.5 year tax holiday on the State of Alaska Production royalty is currently in place.

<u>State Income Tax</u>

State income tax is calculated in the same manner as federal tax, however it takes 9.4% of taxable income after the deduction of federal income tax.

<u>Property Tax</u>

Property tax is calculated using an estimated 1.3% of the net of gross income and direct operating costs.

<u>Mining License Tax</u>

Mining License Tax is 7% of taxable income.

<u>Production Royalty</u>

The Alaska Production Royalty is 3% of net income.

22.0 ECONOMIC ANALYSIS

The following preliminary economic assessment analysis includes inferred mineral resources which are considered too speculative geologically to have economic considerations applied to them, and are therefore not categorized as mineral reserves. There is no certainty that the preliminary economic assessment will be realized.

Project cost estimates and economics are prepared on an annual basis. Based upon design criteria presented in this report, the level of accuracy of the estimate is considered ±35%.

Project economics are based primarily on inputs developed in the preliminary economic assessment. Economic results suggest the following conclusions:

- Mine Life: 24 years;
- Pre-Tax $NPV_{5\%}$: $213 million; IRR: 20.0%;
- Post-Tax $NPV_{5\%}$: $188 million; IRR: 19.6%;
- Payback (Post-Tax): 3.3 years;
- Federal Income Taxes Paid: $58 million;
- State Income Tax Paid: $21 million;
- Mining License Tax Paid: $55 million;
- Cash costs of $842/oz; and
- Initial project capital of $88 million, sustaining project capital of $348 million, and total project capital of $437 million.

Technical economic tables and figures presented require subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding. Where these occur they are not considered to be material.

22.1 SMELTER SCHEDULE

The estimate of smelter revenue is summarized in **Table 22-1**. Technical parameters supporting these estimates are described in **Section 21.0**. Gold deductions are assume to be 0.1% off recovered gold, transport and insurance is estimated at $4.00 per ounce of doré produced, and refining is estimated at $3.00 per ounce of payable gold.

Table 22-1: LoM Revenues & Costs

Description	LoM Cost ($000s)
Gross Sales	$3,003,548
Metal Deduction	($3,004)
Transport & Insurance	($9,430)
Refining Charge	($6,924)
Net Smelter Return	**$2,984,190**
Private Royalty	($59,684)
Gross Income from Mining	**$2,924,506**

22.2 ECONOMIC RESULTS

Technical economic results for the Project are presented in **Table 22-2** and **Table 22-3**.

Table 22-2: Technical-Economic Results

Cost Category	Unit Cost $/oz-recovered	LoM Cost ($000s)
Gross Sales	**$1,300**	**$3,003,548**
Deductions	($8.38)	($19,358)
Royalty	($25.83)	($59,684)
Gross Income	**$1,266**	**$2,924,506**
Operating Costs		
Open Pit Mining	($441.68)	($1,020,468)
Mining Lease	($44.53)	($102,877)
Crushing Circuit	($38.10)	($88,038)
Heap Leach (Oxide)	($50.18)	($115,929)
Process Plant (Sulfide)	($185.59)	($428,803)
Tailings Storage Facility	($4.96)	($11,464)
Infrastructure	($13.09)	($30,236)
Property Tax	($6.10)	($14,094)
Mining License Tax	($23.74)	($54,845)
Total Operating	**($807.97)**	**($1,866,753)**
Operating Profit	**$457.82**	**1,057,753**
Capital Costs		
10 Open Pit Mining	-	($150,528)
20 Crushing & SAG Mill Circuits		($13,805)
30 Heap Leach (Oxide)	-	($35,133)
40 Process Plant (Sulfide)	-	($27,894)
50 Tailings Storage Facility	-	($67,774)
60 Infrastructure	-	($21,131)
70 Construction	-	($68,998)
Indirect Costs	-	($19,221)
Owner's Costs		($32,224)
Total Capital	-	**(436,708)**
Pre-Tax Cash Flow		$613,168
$NPV_{5\%}$		$212,603
IRR		20.0%
Post-Tax Cash Flow		$533,613
$NPV_{5\%}$		$187,742
IRR		19.6%
Payback (years)		3.3

Table 22-3
Freegold Ventures Limited
Golden Summit Project PEA
Estimate of Cash Flow

>Pre-Production (years -4 to -1) END< | >Oxide Mining & Processing (years 1 to 17) END< | >Sulfide Mining & Processing (years 9 to 24) END<

Description			Units	Total or Avg.	-4 / 1	-3 / 2	-2 / 3	-1 / 4	1 / 5	2 / 6	3 / 7	4 / 8	5 / 9	6 / 10	7 / 11	8 / 12	9 / 13
Production Summary																	
Waste Mined	-	-	kt	239,170	0	0	0	0	3,861	3,230	3,115	3,139	2,719	2,413	4,000	4,000	4,000
Oxide Resource	-	-	kt	47,864	0	0	0	0	3,800	3,800	3,550	3,666	3,500	3,500	3,500	3,500	3,500
Sulfide Resource	-	-	kt	48,791	0	0	0	0	0	0	0	0	0	0	0	0	1,020
RoM Resource	-	-	kt	96,655	0	0	0	0	3,800	3,800	3,550	3,666	3,500	3,500	3,500	3,500	4,520
Material Moved	-	-	kt	335,826	0	0	0	0	7,661	7,030	6,665	6,805	6,219	5,913	7,500	7,500	8,520
Recovered Gold	-	$96.27	koz	2,310	0	0	0	0	50	61	61	60	56	58	40	41	56
Dore Produced	-	-	koz	2,358	0	0	0	0	51	62	62	61	57	59	41	42	57
Estimate of Cash Flow																	
Gross Income from Mining	price option	$/oz															
Market Price	1	-	$/oz	1	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Gold Sold	-	-	koz	2,310	0	0	0	0	50	61	61	60	56	58	40	41	56
Gross Sales	-	$1,300	$000s	3,003,548	0	0	0	0	64,727	78,699	78,731	77,681	72,622	74,953	51,854	53,503	72,253
NSR:																	
Gold Deduction	0.1%		koz	(2.3)	0.00	0.00	0.00	0.00	(0.05)	(0.06)	(0.06)	(0.06)	(0.06)	(0.06)	(0.04)	(0.04)	(0.06)
Payable Gold	-	-	koz	2,308	0	0	0	0	50	60	61	60	56	58	40	41	56
Deductions:																	
Metal Deduction	-	($1.30)	$000s	(3,004)	0	0	0	0	(65)	(79)	(79)	(78)	(73)	(75)	(52)	(54)	(72)
Transport & Insurance	$4.00	($4.08)	$000s	(9,430)	0	0	0	0	(203)	(247)	(247)	(244)	(228)	(235)	(163)	(168)	(227)
Refining	$3.00	($3.00)	$000s	(6,924)	0	0	0	0	(149)	(181)	(182)	(179)	(167)	(173)	(120)	(123)	(167)
Deductions	-	($8.38)	$000s	(19,358)	0	0	0	0	(417)	(507)	(507)	(501)	(468)	(483)	(334)	(345)	(466)
Net Smelter Return		$1,292	$000s	2,984,190	0	0	0	0	64,310	78,192	78,223	77,180	72,154	74,470	51,520	53,158	71,787
Royalty:																	
Private Royalty (NSR)	2.0%	($25.83)	$000s	(59,684)	0	0	0	0	(1,286)	(1,564)	(1,564)	(1,544)	(1,443)	(1,489)	(1,030)	(1,063)	(1,436)
Gross Income from Mining		$1,266	$000s	2,924,506	0	0	0	0	63,023	76,628	76,659	75,636	70,710	72,981	50,489	52,095	70,352
Operating Costs																	
Mining	$3.039	$441.68	$000s	1,020,468	0	0	0	0	21,423	21,357	20,356	20,576	19,289	18,370	20,576	21,137	23,961
Mining Lease	-	$44.53	$000s	102,877	0	0	0	0	8,199	8,199	8,199	8,199	8,199	8,199	8,199	8,199	3,107
Crushing Circuit	-	$38.10	$000s	88,038	0	0	0	0	1,407	1,407	1,314	1,357	1,295	1,295	1,295	1,295	2,765
Heap Leach (Oxide)	-	$50.18	$000s	115,929	0	0	0	0	8,231	8,231	7,929	8,069	7,868	7,868	7,868	7,868	6,757
Process Plant (Sulfide)	-	$185.59	$000s	428,803	0	0	0	0	0	0	0	0	0	0	0	0	16,072
Tailings Storage Facility	-	$4.96	$000s	11,464	0	0	0	0	491	491	491	491	491	491	491	491	491
Infrastructure	-	$13.09	$000s	30,236	0	0	0	0	1,318	1,318	1,318	1,318	1,318	1,318	1,318	1,318	1,331
Direct Operating Cost	-	$778.13	$000s	1,797,814	0	0	0	0	41,069	41,003	39,606	40,010	38,461	37,541	39,747	40,308	54,485
Property Tax	-	$6.10	$000s	14,094	0	0	0	0	289	468	487	463	403	442	133	144	193
Mining License Tax	-	$23.74	$000s	54,845	0	0	0	0	0	0	0	376	1,620	1,823	639	866	1,174
Operating Cost	$19.31	$807.97	$000s	1,866,753	0	0	0	0	41,357	41,472	40,093	40,850	40,483	39,806	40,519	41,318	55,852
Cash Cost	-	0.000	$/oz	$842	$0	$0	$0	$0	$865	$719	$696	$718	$759	$725	$1,050	$1,038	$1,039
Operating Profit	-	$457.82	$000s	1,057,753	0	0	0	0	21,666	35,157	36,566	34,786	30,227	33,174	9,970	10,777	14,500
Capital Costs																	
Project Capital:																	
10 Mining			$000s	150,528	0	0	9,131	30,613	0	0	0	0	0	344	0	0	8,282
20 Crushing & SAG Mill Circuits			$000s	13,805	0	0	1,569	2,353	0	0	0	0	0	0	0	9,884	0
30 Heap Leach (Oxide)			$000s	35,133	0	0	0	11,410	1,977	1,977	1,977	1,977	1,977	1,977	1,977	1,977	1,977
40 Process Plant (Sulfide)			$000s	27,894	0	0	0	0	0	0	0	0	0	0	0	27,894	0
50 Tailings Storage Facility			$000s	67,774	0	0	0	0	0	0	0	0	0	0	0	13,394	0
60 Infrastructure			$000s	21,131	0	0	0	10,131	0	0	0	0	0	0	0	11,000	0
70 Construction			$000s	68,998	0	0	0	12,095	0	0	0	0	0	0	0	56,903	0
Indirect Costs			$000s	19,221	0	0	0	3,879	0	0	0	0	0	0	0	15,342	0
Owner's Costs			$000s	32,224	1,810	2,810	1,310	1,310	150	150	150	150	150	150	650	800	300
Project Capital			$000s	436,708	1,810	2,810	12,010	71,790	2,127	2,127	2,127	2,127	2,127	2,471	2,627	137,193	10,559
Working Capital:																	
Beginning Balance			$000s	389,159	0	0	0	0	0	8,612	8,709	8,433	8,579	8,479	8,356	8,377	8,545
Ending Balance	20%		$000s	389,159	0	0	0	0	8,612	8,709	8,433	8,579	8,479	8,356	8,377	8,545	11,551
Required Working Capital			$000s	0	0	0	0	0	8,612	96	(276)	146	(100)	(123)	21	168	3,005
Total Capital	-		$000s	436,708	1,810	2,810	12,010	71,790	10,739	2,223	1,851	2,273	2,027	2,348	2,648	137,361	13,565
Cumulative	-		$000s	-	1,810	4,620	16,630	88,420	99,159	101,383	103,234	105,507	107,534	109,882	112,530	249,891	263,456
Estimate of Cash Flow																	
Operating Profit			$000s	1,057,753	0	0	0	0	21,666	35,157	36,566	34,786	30,227	33,174	9,970	10,777	14,500
Project Capital			$000s	(436,708)	(1,810)	(2,810)	(12,010)	(71,790)	(10,739)	(2,223)	(1,851)	(2,273)	(2,027)	(2,348)	(2,648)	(137,361)	(13,565)
Federal Tax			$000s	(58,386)	0	0	0	0	0	0	0	0	0	0	0	0	0
State Tax			$000s	(21,169)	0	0	0	0	0	0	0	0	0	0	0	0	0
Alaska Production Royalty			$000s	(7,877)	0	0	0	0	0	0	0	0	0	(494)	(56)	0	0
Post-Tax Cash Flow			$000s	533,613	(1,810)	(2,810)	(12,010)	(71,790)	10,927	32,933	34,714	32,513	28,200	30,333	7,266	(126,584)	935
Cumulative			$000s	-	(1,810)	(4,620)	(16,630)	(88,420)	(77,493)	(44,560)	(9,846)	22,668	50,868	81,201	88,466	(38,118)	(37,183)
Present Value	5%		$000s	187,742	(1,810)	(2,676)	(10,893)	(62,015)	8,990	25,804	25,904	23,107	19,087	19,553	4,460	(74,011)	520
NPV			$000s	-	(1,810)	(4,486)	(15,380)	(77,395)	(68,405)	(42,601)	(16,697)	6,410	25,497	45,050	49,510	(24,501)	(23,981)
IRR			-	20%													
Payback			years	3.3													

Description	10 / 14	11 / 15	12 / 16	13 / 17	14 / 18	15 / 19	16 / 20	17 / 21	18 / 22	19 / 23	20 / 24	21 / 25	22 / 26	23 / 27	24 / 28	25 / 29	26 / 30	27 / 31
Production Summary																		
Waste Mined	9,955	16,055	25,000	24,882	20,665	16,000	16,000	16,000	16,792	12,639	11,814	9,148	8,075	3,950	1,719	0	0	0
Oxide Resource	3,500	3,500	3,500	3,500	1,300	58	155	36	0	0	0	0	0	0	0	0	0	0
Sulfide Resource	1,808	2,598	3,300	3,500	3,500	3,500	3,500	3,500	3,500	3,481	3,412	3,500	3,500	3,500	1,672	0	0	0
RoM Resource	5,308	6,098	6,800	7,000	4,800	3,558	3,655	3,536	3,500	3,481	3,412	3,500	3,500	3,500	1,672	0	0	0
Material Moved	15,263	22,153	31,800	31,882	25,465	19,558	19,655	19,536	20,292	16,120	15,225	12,648	11,575	7,450	3,391	0	0	0
Recovered Gold	111	158	148	148	136	121	137	142	117	111	107	103	97	115	79	0	0	0
Dore Produced	113	161	151	151	139	123	139	144	119	114	109	105	99	118	81	0	0	0
Estimate of Cash Flow																		
Gross Income from Mining																		
Market Price	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Gold Sold	111	158	148	148	136	121	137	142	117	111	107	103	97	115	79	0	0	0
Gross Sales	143,960	205,099	192,197	192,327	176,466	157,319	177,675	184,084	151,684	144,933	139,060	134,061	126,591	149,928	103,142	0	0	0
NSR:																		
Gold Deduction	(0.11)	(0.16)	(0.15)	(0.15)	(0.14)	(0.12)	(0.14)	(0.14)	(0.12)	(0.11)	(0.11)	(0.10)	(0.10)	(0.12)	(0.08)	0.00	0.00	0.00
Payable Gold	111	158	148	148	136	121	137	141	117	111	107	103	97	115	79	0	0	0
Deductions:																		
Metal Deduction	(144)	(205)	(192)	(192)	(176)	(157)	(178)	(184)	(152)	(145)	(139)	(134)	(127)	(150)	(103)	0	0	0
Transport & Insurance	(452)	(644)	(603)	(604)	(554)	(494)	(558)	(578)	(476)	(455)	(437)	(421)	(397)	(471)	(324)	0	0	0
Refining	(332)	(473)	(443)	(443)	(407)	(363)	(410)	(424)	(350)	(334)	(321)	(309)	(292)	(346)	(238)	0	0	0
Deductions	(928)	(1,322)	(1,239)	(1,240)	(1,137)	(1,014)	(1,145)	(1,186)	(978)	(934)	(896)	(864)	(816)	(966)	(665)	0	0	0
Net Smelter Return	143,033	203,777	190,958	191,088	175,329	156,305	176,530	182,898	150,706	143,999	138,164	133,197	125,775	148,961	102,477	0	0	0
Royalty:																		
Private Royalty (NSR)	(2,861)	(4,076)	(3,819)	(3,822)	(3,507)	(3,126)	(3,531)	(3,658)	(3,014)	(2,880)	(2,763)	(2,664)	(2,515)	(2,979)	(2,050)	0	0	0
Gross Income from Mining	140,172	199,702	187,139	187,266	171,822	153,179	173,000	179,240	147,692	141,119	135,401	130,533	123,259	145,982	100,428	0	0	0
Operating Costs																		
Mining	39,107	55,609	91,818	93,093	76,372	58,971	59,953	60,417	63,984	53,221	51,226	44,522	42,433	28,114	14,584	0	0	0
Mining Lease	3,107	3,107	3,107	3,107	3,107	3,107	3,107	3,107	3,107	3,107	3,107	0	0	0	0	0	0	0
Crushing Circuit	3,901	5,040	6,052	6,340	5,526	5,066	5,102	5,058	5,045	5,017	4,917	5,045	5,045	5,045	2,410	0	0	0
Heap Leach (Oxide)	6,866	6,996	7,112	7,145	4,480	1,697	1,532	1,351	1,197	1,194	1,182	1,197	1,197	1,197	896	0	0	0
Process Plant (Sulfide)	20,387	24,711	28,552	29,646	29,646	29,646	29,646	29,646	29,646	29,541	29,164	29,646	29,646	29,646	13,558	0	0	0
Tailings Storage Facility	491	491	491	491	491	491	491	491	491	491	491	491	491	491	160	0	0	0
Infrastructure	1,331	1,331	1,331	1,331	1,331	1,331	1,331	1,331	1,331	1,331	1,331	1,331	1,331	703	352	0	0	0
Direct Operating Cost	75,191	97,287	138,464	141,154	120,954	100,310	101,163	101,403	104,802	93,903	91,420	82,233	80,144	65,196	31,960	0	0	0
Property Tax	811	1,276	598	566	629	655	896	973	533	589	549	604	539	1,005	849	0	0	0
Mining License Tax	3,309	5,357	3,192	3,061	3,033	3,049	3,718	3,868	2,378	2,422	2,241	2,363	2,144	4,331	3,882	0	0	0
Operating Cost	79,311	103,921	142,254	144,781	124,616	104,014	105,776	106,243	107,712	96,914	94,209	85,200	82,826	70,532	36,691	0	0	0
Cash Cost	$750	$693	$996	$1,013	$952	$894	$808	$785	$957	$903	$915	$860	$885	$646	$497	$0	$0	$0
Operating Profit	60,861	95,781	44,885	42,485	47,206	49,164	67,223	72,996	39,980	44,205	41,192	45,333	40,434	75,450	63,737	0	0	0
Capital Costs																		
Project Capital:																		
10 Mining	46,768	24,254	23,837	4,515	673	1,606	344	0	26	0	0	135	0	0	0	0	0	0
20 Crushing & SAG Mill Circuits	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
30 Heap Leach (Oxide)	1,977	1,977	1,977	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
40 Process Plant (Sulfide)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
50 Tailings Storage Facility	0	15,449	0	0	0	38,932	0	0	0	0	0	0	0	0	0	0	0	0
60 Infrastructure	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
70 Construction	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect Costs	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner's Costs	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	17,834	0	0
Project Capital	49,045	41,980	26,114	4,815	973	40,838	644	300	326	300	300	435	300	300	300	17,834	0	0
Working Capital:																		
Beginning Balance	11,551	16,620	21,864	29,462	29,969	25,852	21,631	22,090	22,218	22,341	20,146	19,574	17,746	17,231	14,896	7,881	0	0
Ending Balance	16,620	21,864	29,462	29,969	25,852	21,631	22,090	22,218	22,341	20,146	19,574	17,746	17,231	14,896	7,881	0	0	0
Required Working Capital	5,069	5,244	7,599	506	(4,117)	(4,221)	460	127	123	(2,195)	(572)	(1,828)	(514)	(2,336)	(7,015)	(7,881)	0	0
Total Capital	54,114	47,224	33,713	5,321	(3,143)	36,617	1,103	427	449	(1,895)	(272)	(1,393)	(214)	(2,036)	(6,715)	9,953	0	0
Cumulative	317,570	364,793	398,506	403,827	400,684	437,300	438,404	438,831	439,280	437,385	437,113	435,720	435,506	433,470	426,755	436,708	436,708	436,708
Estimate of Cash Flow																		
Operating Profit	60,861	95,781	44,885	42,485	47,206	49,164	67,223	72,996	39,980	44,205	41,192	45,333	40,434	75,450	63,737	0	0	0
Project Capital	(54,114)	(47,224)	(33,713)	(5,321)	3,143	(36,617)	(1,103)	(427)	(449)	1,895	272	1,393	214	2,036	6,715	(9,953)	0	0
Federal Tax	0	0	0	0	0	0	(3,370)	(7,491)	(3,160)	(4,367)	(4,181)	(4,983)	(4,314)	(13,315)	(13,205)	0	0	0
State Tax	0	0	0	0	0	0	(1,222)	(2,716)	(1,146)	(1,583)	(1,516)	(1,807)	(1,564)	(4,827)	(4,788)	0	0	0
Alaska Production Royalty	(494)	(934)	(93)	(77)	(222)	(262)	(529)	(642)	(271)	(374)	(358)	(427)	(370)	(1,141)	(1,132)	0	0	0
Post-Tax Cash Flow	6,254	47,622	11,080	37,086	50,128	12,286	60,999	61,720	34,954	39,775	35,409	39,510	34,399	58,202	51,327	(9,953)	0	0
Cumulative	(30,929)	16,693	27,773	64,859	114,987	127,272	188,271	249,991	284,945	324,719	360,128	399,638	434,037	492,240	543,566	533,613	533,613	533,613
Present Value	3,317	24,053	5,329	16,990	21,871	5,105	24,139	23,262	12,546	13,597	11,528	12,251	10,158	16,369	13,748	(2,539)	0	0
NPV	(20,664)	3,389	8,718	25,708	47,578	52,683	76,822	100,084	112,630	126,227	137,756	150,006	160,164	176,533	190,281	187,742	187,742	187,742
IRR																		
Payback																		

22.3 SENSITIVITY

Project Sensitivity at a post-tax basis is shown in **Figure 22-1**. As shown below, the Project is most sensitive to revenue. Sensitivity to operating and capital costs is closely matched, with the project being more sensitive to operating costs. These results are typical of similar projects.

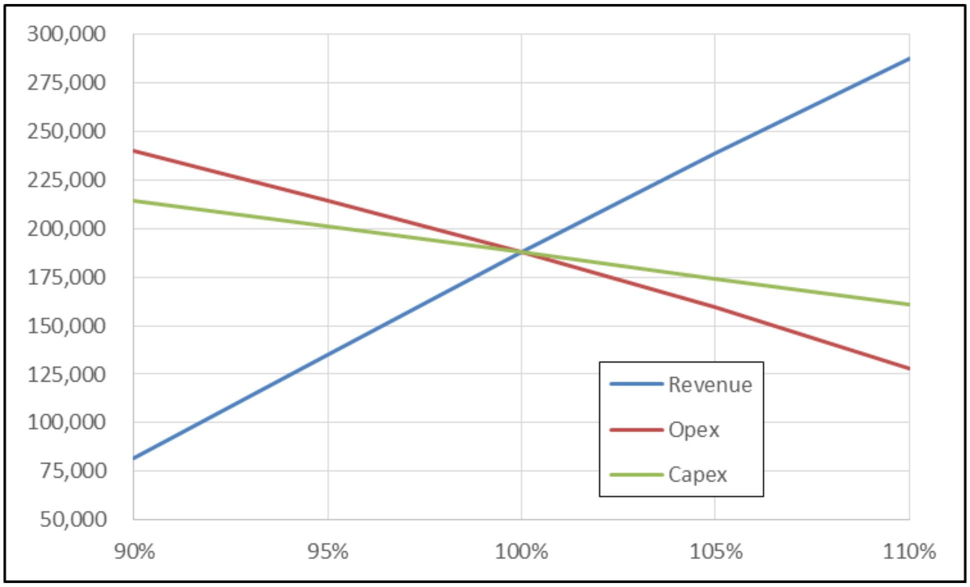


Figure 22-1: Project Sensitivity – Post-Tax NPV (5%)

23.0 ADJACENT PROPERTIES

In addition to several nearby exploration properties and gold-bearing prospects the Project is adjacent to leases controlled by Kinross Gold Corporation (Kinross) on the southern border of Section 32 of Township 3 North 2 East that are associated with the Fort Knox Mine.

The qualified person has not independently verified the past production, resources or reserve estimates of any adjacent properties. Results from adjacent properties are not necessarily indicative of the mineralization on the property that is the subject of the technical report.

23.1 FORT KNOX MINE

The Fort Knox Mine is located nine km to the southwest of the Project. The Fort Knox Mine is an operating mine that includes an open pit, carbon-in-pulp mill, heap leach, and a tailings storage facility. As of year-end 2015, the mine has produced approximately 6.8 million ounces of gold since commencing commercial production in 1997. The remaining reserves stated in Kinross's 2015 technical report are shown in **Table 23-1.**

Table 23-1: Fort Knox Reserve Estimate March 31, 2015

Classification	Tonnes M	Grade Au g/t	Contained Au Ounces M
Proven	24.0	0.56	0.435
Proven Stockpiles	43.9	0.31	0.437
Subtotal Proven	67.9	0.40	0.872
Probable	96.0	0.49	1.527
Total Proven and Probable	**163.8**	**0.46**	**2.398**

Notes:
1) The cutoff grades are based on a gold price of US$1,200/oz Au
2) Proven Reserve contains stockpiles.
3) Mineral Reserves are reported to a cutoff grade of 0.35 g/t Au for A-ore (mill), 0.29 g/t for B-ore (stockpile), and 0.19 g/t C-ore (leach)

23.2 TRUE NORTH MINE

The True North Mine, part of the greater Fort Knox Mine project, is 6 km to the west of the Dolphin deposit and is currently under post-closure monitoring. In 1997 estimated resources were 18.2 M tons grading 0.072 Au opt containing 1.3 million ounces of gold (La Teko Resources Ltd. June, 1997). The True North Mine achieved commercial production in early April 2001 and closed in 2004. While in production, 11,026,772 tons of ore were delivered to the Fort Knox Mine for processing (USGS Alaska Resource Data File).

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 GROUNDWATER HYDROLOGY

Hydrogeologic conditions at the Project site are expected to be similar to those at the nearby Fort Knox mine, based on the proximity of the two sites and the similarity of geologic conditions. The Fort Knox mine has been operating since 1996, and valuable groundwater-related data from annual operations reports and various permitting documents were available from those documents.

A search of State of Alaska, federal records and publicly-available literature was conducted to locate information regarding hydrogeologic conditions at or near the Project property. The following information was identified.

- Records of three water wells on the property, nine wells within approximately six km (3.7 miles) northwest, southwest and south of the property, and four wells at the Fort Knox mine were obtained from ADNR.
- Several United States Geological Survey (USGS) reports regarding groundwater quality in the vicinity of Fairbanks, but not within the property, were obtained.
- Numerous documents regarding the Fort Knox mine, including permit information and annual reports, were obtained from the ADNR.
- No other publicly-available literature or hydrogeologic data specific to the Project property were identified for use in this study.

24.1.1 Hydrogeologic Setting

The Project area topography is characterized by low, rounded hills cut by steep valleys. Land surface elevations range from approximately 305 to 670 m (1,000 to 2,200 ft) amsl. The lower reaches of the valleys typically hold perennial streams. Unconsolidated alluvial deposits or dredge tailings are present along the valley floors, and a variably-thick layer of eolian silt covers most of the property (Abrams and Giroux, 2013). Permafrost occurs in small, discontinuous lenses on steep, poorly-drained north-facing slopes.

The geology of the Project is described in Abrams and Giroux (2013); the following description is taken from that document. The majority of the Project is underlain by the Fairbanks Schist; rocks of the Chatanika Terrane are present beneath the northern edges of the property. Both rock units are comprised primarily of schist and are similar both lithologically and in appearance. The two units are in contact across the east-northeast-trending Chatanika Thrust Fault, which carries the Chatanika Terrane southward over the Fairbanks Schist. The thrust fault itself is offset by a series of northeast-trending, steeply-dipping normal faults. In addition, a number of shorter, more closely-spaced, normal faults trend east-west through the north-central part of the property. Mineralization is hosted in shear zones and auriferous quartz veins oriented northwest-southeast in the eastern part of the property and east-west in the western part of the property.

24.1.2 Groundwater Conditions

Based on data obtained from records of water wells within approximately 6 km (3.7 miles) of the Property (**Table 24-1**) (ADNR, 2014) and conditions reported at the Fort Knox mine site (FGMI, 2006), groundwater is expected to be present in two units: unconsolidated deposits consisting of alluvium and dredge tailings along the valley floors, and fractured bedrock throughout the property. Logs of water wells in the Project area indicate that alluvial deposits and dredge tailings may be 10 m (33 ft) or more in thickness. FGMI (2006) reported that the upper portion of the bedrock (up to about 30 m (100 ft) in thickness) is highly weathered, with variable degrees of fracturing, and that movement of groundwater in the bedrock occurs in open fractures. The fractures in the bedrock provide essentially all of the permeability, as the schistose rock mass has very low permeability. The hydraulic conductivity provided by the fractures is directly related to the degree of fracturing and is expected to be highly variable. Aquifer testing results reported in FGMI (2006) indicated that the alluvial deposits and the bedrock each exhibit hydraulic conductivities in the range of 0.0086 to 8.6 meters per day (m/d) (0.028 to 28 ft/d).

The water table is anticipated to reflect the topography, but with subdued relief. Reported depths to static water levels in wells at and near the Project were reported to range from 2.1 m (6.9 ft) below the land surface in the valley bottoms to 68.6 m (225 ft) below the land surface in upland areas. Reported yields of water supply wells ranged from 16 to 491 m^3/day (3 to 90 gallons per minute (gpm)), and dewatering wells at the Fort Knox mine were reported to have capacities up to approximately 1,000 m^3/day (183 gpm). Groundwater flow on a local scale is anticipated to be from bedrock in the upland areas toward the valleys and thence down-valley in the alluvial deposits or dredge tailings. Regional-scale groundwater flow cannot be determined from available data.

Table 24-1: Water Well Data Summary

Township	Range	Section	Quarter Sections	Hole Depth (m)	Depth to Static Water Level (m)	Pumping Rate (m^3/day)	Comments
003N	001E	2	--	61.0	--	--	Domestic
003N	001E	12	--	82.3	--	--	Stanford Research Institute
003N	001E	12	--	27.4	--	--	Domestic
003N	001E	36	SW NE	54.9	37.5	16.4	Domestic*
003N	002E	31	NW NE SE	121.9	68.6	45.2	Domestic*
003N	002E	31	SE NE NW SE	61.0	21.3	--	Fairbanks Creek Lodge*
002N	001E	2	--	13.7	11.6	490.6	Public water supply
002N	001E	10	SE SE SE SE	28.7	9.1	163.5	Domestic
002N	001E	15	NE NE	29.9	2.1	136.3	Domestic
002N	001E	15	SW	164.6	--	--	Domestic
002N	001E	15	SW	22.9	13.7	43.6	Domestic
002N	001E	15	--	12.5	2.4	16.3+	Domestic
002N	002E	16	--	182.9	--	--	Dewatering, Fort Knox
002N	002E	16	--	150.9	--	27.3	Dewatering, Fort Knox
002N	002E	16	--	18.3	7.3	218.1	Dewatering, Fort Knox
002N	002E	16	SW NE NE	125.0	40.8	81.8+	Mill house water supply, Fort Knox

*Denotes well located within Project property.

Groundwater quality is anticipated to be similar to that observed at the Fort Knox mine. Average groundwater quality in samples collected from dewatering wells there in 2011, as reported by Schlumberger Water Services (SWS, 2013), is summarized in (**Table 24-2**). That water could be considered representative of groundwater from bedrock; water quality in unconsolidated alluvial deposits and dredge tailings would likely differ. Overall, the predominant ions were calcium, sulfate and bicarbonate. Groundwater was slightly basic and contained a moderately small concentration of total dissolved solids (TDS) (approximately 220 milligrams per liter [mg/L]).

Table 24-2: Summary of Groundwater Quality for Fort Knox Mine Dewatering Wells

Parameter	Average Concentration in 2011 Dewatering Well Samples (mg/L)	Parameter	Average Concentration in 2011 Dewatering Well Samples (mg/L)
pH, std. units	8.0	Lead	0.00026
Alkalinity as CaCO3	74	Magnesium	5.4
Ammonia	<MDL	Manganese	0.014
Antimony	0.0032	Mercury	<MDL
Arsenic	0.0247	Nitrate, as N	0.54
Barium	0.0014	Nitrite, as N	0.063
Cadmium	<MDL	Phosphorus	0.001
Calcium	40.0	Potassium	0.99
Chloride	0.3	Selenium	0.00095
Chromium	<MDL	Silver	<MDL
Copper	<MDL	Sodium	16.1
WAD-cyanide	0	Sulfate	78.2
Fluoride	0.36	Zinc	0.013
Iron	0.028		

Less than method detection limit (<MDL)

FGMI (2008) reported baseline (pre-mining) concentrations of TDS, iron, manganese, arsenic and antimony for 43 groundwater samples from alluvial wells and 46 samples from bedrock wells in the Fish Creek valley near the current Fort Knox tailings storage facility. The results are summarized in **Table 24-3**. That report also noted that concentrations of iron and manganese were elevated after placer mining in that area but before initiation of mining at Fort Knox. Lang Farmer et al. (1998) reported that arsenic concentrations in the Fairbanks area are highly variable both spatially and between wells, and relatively high concentrations of arsenic in water reflect a naturally high regional background.

Table 24-3: Summary of Baseline Groundwater Quality for Fort Knox Alluvial and Bedrock Wells

Parameter	Alluvial Wells		Bedrock Wells	
	Minimum	Maximum	Minimum	Maximum
TDS, mg/L	114	366	84	357
Iron, mg/L	0.164	58.2	0.017	23.9
Manganese, mg/L	0.384	155	0.016	1.61
Arsenic, mg/L	0.001	0.034	0.002	0.026
Antimony, mg/L	0.005	0.008	0.004	0.025

24.2 Surface Water

24.2.1 Surface Water Hydrology

The project is located within the Cleary Creek watershed (**Figure 24-1**). Numerous creeks exist on the property including Willow Creek, Tamarack Creek, Bedrock Creek, Chatham Creek, Fairbanks Creek, Wolf Creek and Goose Creek. Cleary Creek discharges into the Chatanika River. Dominant soil types are Ester-Gilmore complex, with Steese-Gilmore, Steese Loam and Ester Peat in the vicinity of the proposed tailings storage facility. These soils are of hydrologic class D, which exhibit low infiltration rates. Precipitation for nearby Fairbanks, Alaska, is reported to be approximately 11 inches annually. Monthly totals are presented in **Table 24-4**.

Table 24-4: Monthly Precipitation

Month	Precipitation (in)/(mm)	Precipitation Days	Snowfall (in)/(mm)
Jan	0.59/15	9	10/254
Feb	0.43/11	7	8/203
Mar	0.24/6	7	5/127
Apr	0.31/8	4	3/76
May	0.59/15	8	1/25
Jun	1.38/35	11	0
Jul	2.17/55	13	0
Aug	1.89/48	15	0
Sep	1.1/28	10	2/51
Oct	0.83/21	11	11/280
Nov	0.67/17	9	13/330
Dec	0.63/16	8	12/305

The manner in which precipitation becomes runoff is a function of site topography, soil characteristics, vegetative cover and geology. Surface water management designs were developed by evaluation of typical meteorological conditions (monthly precipitation totals) and statistical design storms, which are discussed in greater detail below.

Water management design criteria were developed using Alaska coal mining regulations as a basis. Design storm selection for sizing of facilities was dependent upon the characteristics of the water being stored or conveyed. Stated differently, design criteria were based on whether water contacted mine affected surfaces (runoff) or undisturbed areas (run-on). A summary of the design storms and their basis is presented below.

Table 24-5: Design Storm Basis for Water Management Facilities

Design Storm	Rainfall Depth (in)/(mm)	Water Management Facility
100-year, 24-hour	3.83/96	Runoff from Tailings, Waste Rock or Heap Leach Facilities. Pit dewatering.
100-year, 6-hour	2.15/54	Run-on diversion of perennial streams
10-year, 24-hour	2.18/55	Sediment Ponds
10-year, 6-hour	1.31/33	Run-on diversion of overland flow

Design storm depths were defined using the National Oceanic and Atmospheric Administration's (NOAA) point precipitation frequency data server. Temporal distributions were selected from the All Cases data set and 10 percent exceedence was assumed, for conservatism.

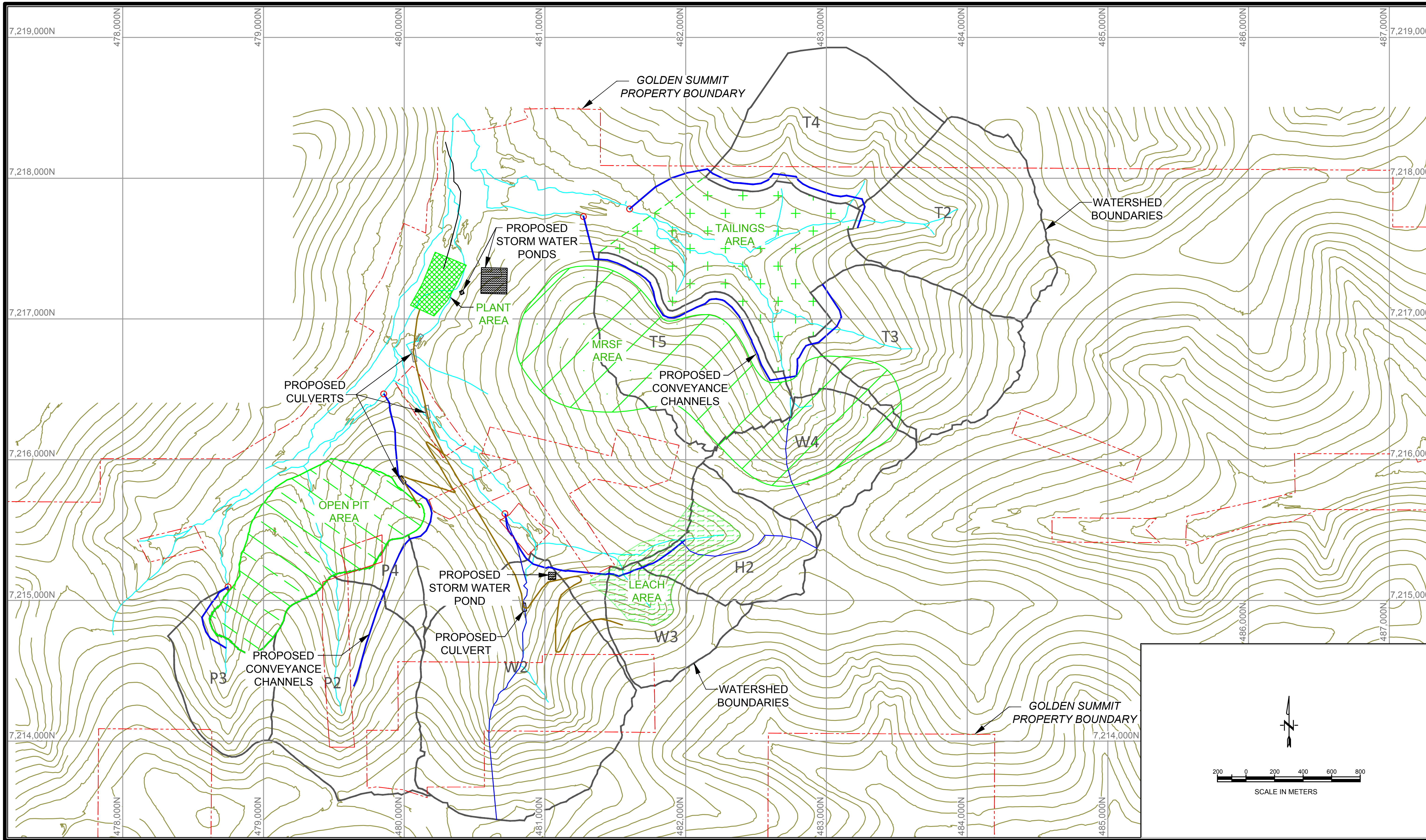
Basin areas were delineated within ArcGIS using LiDAR topography. Areas were exported for further evaluation within AutoCAD Civil 3D and to define hydrologic modeling parameters.

Hydrologic modeling was performed using HEC-HMS software to determine peak flow rates and volumes for each basin and facility. Within HEC-HMS, losses to soil were characterized using the Soil Conservation Service (SCS) curve number method and direct runoff was calculated using the SCS unit hydrograph. A curve number of 80 was assigned to the upland areas, to represent a combination of evergreen and aspen trees, with grass understory. Curve number estimates for the facilities ranged between 84 and 88 and may be adjusted in future studies as material properties become available. The SCS unit hydrograph is based on time of concentration, or the time necessary for water to travel from the hydraulically most remote point to the basin outlet. The unit hydrograph is further defined by the lag time parameter. Assuming uniform distribution of runoff, time of concentration (Tc) and lag time (Tlag) are related based on the following equation:

$$Tlag = 0.6*Tc$$

Time of concentration calculations were performed for the largest basins: W2, W4, T2 and T3 and found to be very near the minimum value of 6.0 minutes. Lag times were therefore 3.6 minutes for all basins. The singular exception to the above is the time of concentration for the MRSF was calculated as 10 minutes, resulting in a lag time of 6 minutes.

Rainfall and snowmelt would accumulate on site as either run-on from undisturbed areas or contact water runoff from mine-affected surfaces. These waters would be managed by a series conveyance channels and detention basins. Run-on channels would divert non-mine affected waters around facilities and confluence with existing natural channels downstream. Runoff channels would direct mine-affected water from the MRSF and HLP into dedicated stormwater ponds. Waters associated with the TSF would be contained within the TSF embankment. All contact water would either be incorporated into the process circuit or treated and released.



B	EDIT TO MRSF	JS	03/2016
A	EDIT TO MRSF	LA	12/2015
Rev	Description	BY	Date

REVISIONS

REFERENCE

ENGINEER'S SEAL

PRELIMINARY DRAWING

NOT TO BE USED FOR CONSTRUCTION

Scale: As Noted

Designed by: K. JOHNSON
Drawn by: L. AGA
Checked by: E. LIPS
Approved by: V. SCHARNHORST

Issued for:

Freegold Ventures Limited

Issued by:

TETRA TECH

GOLDEN SUMMIT PROJECT
WATERSHED MAP

B
REVISION

Project:
GOLDEN SUMMIT PROJECT
PRELIMINARY ECONOMIC ASSESSMENT

Project no.: 114-910054

Location: FAIRBANKS MINING DISTRICT, ALASKA

Date: March/2016

FIGURE 24-1

24.2.2 Surface Water Management

Surface water management addresses the protection of the natural environment and site facilities by means of conveyance and detention structures. Site facilities include the Tailings Storage Facility (TSF), the Heap Leach Pad (HLP), the Mine Rock Storage Facility (MRSF) and the Open Pit.

Run-on from areas upland of mine facilities would be diverted to protect infrastructure and maintain pre-development hydrology to the greatest extent practicable. Areas associated with the mine facilities and upland zones are presented in **Table 24-6**. For calculation purposes, each basin or facility was assigned a unique ID, presented in the table below and depicted in **Figure 24-1.**

Table 24-6: Facility Areas

Facility and Catchment Identification	Area (acres)/(sq. km)
TSF	
T1 - TSF Embankment	295/1.19
T2 - Goose Diversion	324/1.31
T3 - Wolf Diversion	300/1.21
T4 - North Side Upslope Diversion[1]	284/1.15
T5 - South Side Diversion	284/1.15
HLP	
H1 – HLP Perimeter & Basin	27/0.11
H2 - Chatham Diversion	170/0.69
Pit[2]	
P2 - Bedrock Diversion	333/1.35
P3 - Willow Diversion	133/0.54
P4 - Overland from Ridge	47/0.19
MRSF	
W1 – MRSF Perimeter and Basin	718/2.91
W2 - Tamarack Diversion (South)	527/2.13
W3 - Chatham Tributary Diversion	134/0.54
W4 - Wolf Diversion	209/0.85

(1) Some uncertainty due to limits of LiDAR
(2) Runoff from disturbed pit surfaces excluded from this study. Pump and pipe infrastructure for pit dewatering from groundwater inflows assumed suitable to accommodate additional flows from runoff.

Conveyance channel dimensions were determined using Manning's equation based on the peak flow rates from HEC-HMS. For diversions serving multiple basins, no attenuation, or diminishing of hydrograph peaks during travel was assumed for conservatism. Minimum longitudinal grades were assumed to be 2%. Conveyance channels and stormwater ponds that serve the MRSF and HLP were assumed to be lined with geosynthetic material. Channels conveying run-on from undisturbed uplands were assumed to be earthen with rock and gravels sized similarly with existing, natural channels on site. The TSF embankment, HLP and MRSF runoff conveyance channels and stormwater ponds are designed as zero discharge facilities under the Solid Waste Permit. As such, each was designed using the 100-year, 24-hour storm criteria. Design criteria for diversions and mine facilities are summarized in **Table 24-7** and **Table 24-8**, respectively.

Table 24-7: Diversion Design Criteria

Diversions	Peak Flow (cfs)/(m^3/s)	Approximate Channel Length (ft)/(m)	Liner Material
MRSF Runoff Conveyance	191/5.4	31,820/9,700	Geosynthetic
HLP Runoff Conveyance	9/0.25	4,265/1,300	Geosynthetic
T2/T4 Diversion	467/13.2	6,889/2,100	Earthen/Gravel/Rock
T3/W4/T5 Diversion	658/18.6	10,497/3,200	Earthen/Gravel/Rock
H2/W3/W2 Diversion	693/19.6	5,249/1,600	Earthen/Gravel/Rock
P3 Diversion	138/3.9	1,968/600	Earthen/Gravel/Rock
P2/P4 Diversion	236/6.7	7,738/2,360	Earthen/Gravel/Rock

Table 24-8: Mine Facility Design Criteria

Facilities	Volume (acre-ft)/(m^3)	Liner Material
MRSF Stormwater Pond	69.5/85,730	Geosynthetic
HLP Stormwater Pond	2.8/3,450	Geosynthetic
Facility Stormwater Pond	4.7/5,800	Earthen
Bedrock Detention Pond	27.5/34,000	Earthen
TSF Embankment	33/40,580	Geosynthetic

Design of the box culverts to be placed at an estimated six locations to protect haul road and stream crossings are not included in this study. However, these items are included as a cost contingency.

Run-on diversion channels would follow side slope contours around the facilities they serve and connect with existing natural channels at a downstream endpoint. It is assumed that the strip of land that separates the south side of the TSF from the north side of the MRSF would be maintained throughout the LoM, allowing the diversion channel to remain functional (not to be encroached upon by the ultimate MRSF footprint). MRSF and HLP runoff channels would terminate into stormwater ponds that would be pumped back into process or to treatment prior to release. The MRSF and HLP runoff channels and stormwater ponds would allow an additional 1-foot of freeboard. The TSF embankment would be designed to store tailings solids, supernatant water and the storm volume specified above. Additional freeboard is recommended to accommodate wave run-up and ice formation on the supernatant pond. In addition to the runoff within the TSF, seepage would be collected from both the overdrain and underdrain systems; seepage would need to be diverted back into the TSF embankment, or stored within a dedicated pond. Project cost estimates assume a dedicated pond.

24.3 Water Balance

Simple schematics showing the Project site wide water balance are presented in **Figure 24-2** and **Figure 24-3** for site wide and camp specific areas, respectively.

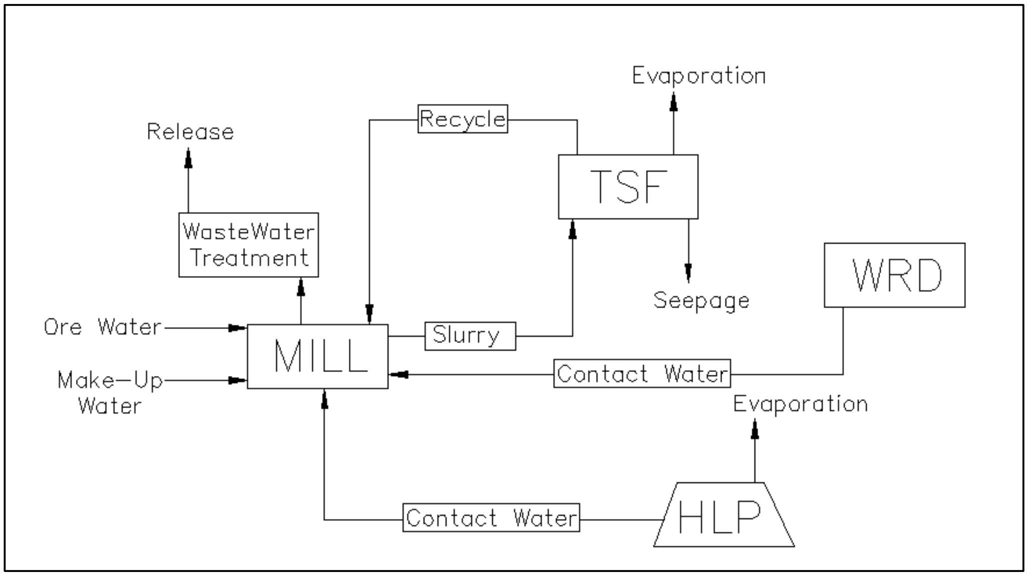


Figure 24-2: Site Wide Water Balance

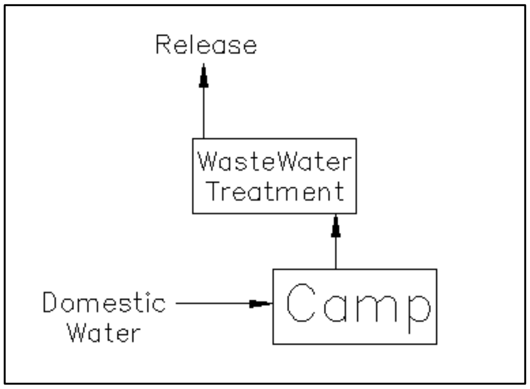


Figure 24-3: Domestic Water Balance

Water sources for the Project are identified as:

- Mine dewatering (estimated as 4,000 to 6,000 m^3/day) (Reference the hydrogeology section);
- Contact water from MRSF, Pit faces, TSF and HLP;
- Run-on from basins undisturbed by mining;
- Well water and
- Mineralized Material moisture.

Site water demands include:

- Process water requirement (raw water and make-up water); and
- Domestic water requirements.

Finally, water exits the system as follows:

- Water entrained within the tailings slurry (a fraction of this water is recovered via decant processes and recycled back into the process circuit);
- Evaporative and seepage losses from the TSF;
- Evaporative loss from other contact or stormwater ponds; and
- Treatment and release processes (e.g. treatment and release of wastewater).

Site water would be managed to minimize the volumes requiring treatment prior release by utilizing dewatering and contact water within the process circuit and by separating run-on and contact water streams to the greatest extent practicable. Run-on from basins undisturbed by mining would be diverted around mine facilities to confluence with existing natural channels below. Considering the ultimate configuration of mine affected surfaces, approximately 524 hectares (1,294 acres) of the site would contribute roughly 1.13 million cubic meters (m^3) (915 acre-feet) of runoff each year near the end of the LoM that must either be retained, recycled or treated. (SCS method applied to annual total precipitation applied to ultimate mine affected surfaces; conservative, as some precipitation would occur as snowfall).

Process water requirements and tailings characteristics are not well defined at the time of this Report. However, the initial estimate for make-up water required for process is approximately 0.03 m^3/s (500 gallons per minute). Weighing this requirement against the flow rates associated with the various sources on site, a net surplus of water is assumed for the Project site. This estimate is consistent with similar mining operations located nearby.

24.4 GEOCHEMISTRY

A site visit was conducted between May 6 and May 7, 2014 to look over the surface geology of the mine site and inspect diamond rock core samples as an integral part of the geochemical waste rock characterization. The core laboratory was located on the north corner of the junction of Alaska Highway 2 and Goldstream Road in Fox, Alaska. Core was stored under water proof tarps on pallets outside with core cutting and viewing facilities available in the shed.

A total of 23 samples representing the spatial, lithologic, and oxide/sulfide range of the site geology were viewed and selected for analysis. **Figure 24-4** depicts the surface locations of available cores from the Project. These bores appear in **Figure 24-5** projected in 3D over the oxide/sulfide block model as supplied by the client. As shown in **Figure 24-5**, borehole distribution appears to adequately penetrate and represent the current mineralized Dolphin body.

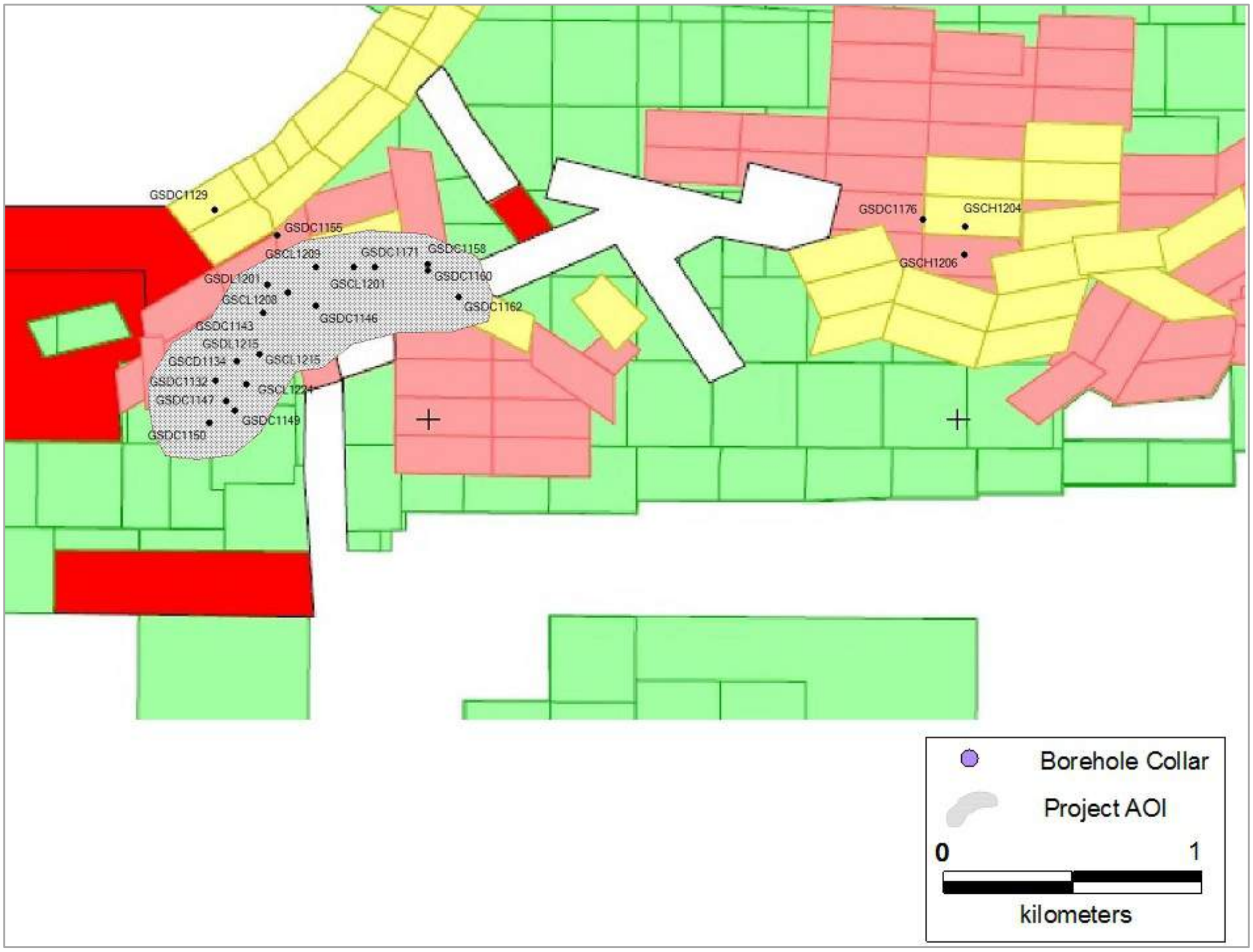


Figure 24-4: Static Geochemical Boreholes Sample Locations

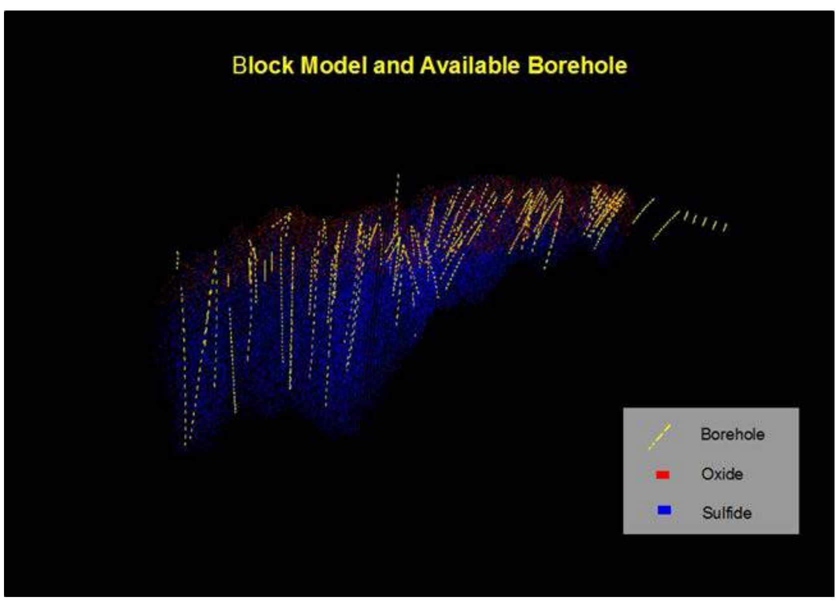


Figure 24-5: Available Boreholes Relative to the Dolphin Mineralized Body

A total of 21 representative waste rock cores were selected for geochemical characterization consisting of spatially and vertically distributed drill core samples primarily from within the current Dolphin Deposit. From these rock cores, 23 total samples were selected that are representative of the Dolphin Deposit located in the Project area. These materials predominantly represent the lower to middle Paleozoic metavolcanic and metasedimentary rocks of the Cleary Sequence and Fairbanks Schist. Lithologies within the Cleary Sequence include quartzite, massive to finely laminated mafic to intermediate flows and tuffs, calc-schist, black chloritic quartzite, quartz-sericite schist of hydrothermal origin and impure marble. Lithologies in the Fairbanks Schist include quartz muscovite schist, micaceous quartzite and biotite quartz mica schist. These lithologies have been metamorphosed to the lower amphibolite facies.

Table 24-9 lists the core samples and intervals used in the geochemical characterization.

Table 24-9: Core Samples and Intervals for Waste Rock Characterization

Sample ID	Rock Type	Borehole	Interval	
			From	To
GSDC_4	Granodiorite	GSDC1129	121.2	127.5
GSDC_8		GSDC1149	60.5	69.5
GSDC_20		GSDC1132	31.3	42
GSDC_21		GSDC1150	60.6	69.8
GSDC_22		GSDC1132	37.5	45
GSDC_24		GSDC1174	779	785
GSDC_11	Granodiorite/Quartzite	GSCL1224	218	224.5
GSDC_19	Tonalite	GSDC1132	294.7	310.5
GSDC_3	Schist	GSDC1160	30.5	39.5
GSDC_7		GSDC1155	140.5	150.5
GSDC_14		GSCL1201	132.7	146.5
GSDC_18		GSDL1215	425	436.7
GSDC_16	Albitic Greenschist	GSDC1162	56	70
GSDC_17		GSDC1171	54.4	58.2
		GSDC1171	59.5	65.8
GSDC_13	Graphitic Schist	GSCL1209	42.9	45.5
		GSCL1209	48.3	51
GSDC_15		GSDC1158	151	155
GSDC_23	Marble	GSDC1156	546	629
GSDC_1	Skarn/Hornfels	GSDC1143	160.6	168.5
GSDC_12		GSCD1134	37.5	51
GSDC_5	Breccia Gouge/Granodiorite	GSDC1147	380	389
GSDC_2	Fault/Vein	GSCH1204	160.6	164.5

Each rock type was chosen to allow for a representative sampling as shown in **Table 24-10**. Two samples were composited (Albitic Greenschist, and Graphitic Schist) to allow for sufficient sample for the geochemical analysis.

Table 24-10: Geochemical Sample Summary

Lithology	Number of Samples	Percentage of Samples
Granodiorite	6	28.6
Granodiorite/Quartzite	1	4.8
Tonalite	1	4.8
Schist	4	19.0
Albitic Greenschist	2	9.5
Graphitic Schist	2	9.5
Marble	1	4.8
Skarn/Hornfels	2	9.5
Breccia Gouge/Granodiorite	1	4.8
Fault/Vein	1	4.8

A total of seven samples underwent mineralogical quantification. All 21 samples underwent water leachability testing to estimate constituent mobility upon meteoric water contact (by Synthetic Precipitation Leaching Procedure (SPLP)), acid-base accounting (ABA) to assess their potential to generate and neutralize acid, and net acid generating (NAG) pH testing to determine the pH upon complete oxidation of sulfide minerals. Results from this stage of testing show:

- Two of the seven samples have measurable pyrite based on mineralogical quantification with percentages of 0.2% and 1.4%. Acid neutralizing minerals are present as calcite, observed in three of the seven samples with concentrations ranging from 4.4% to 20.4%. Ankerite, another acid neutralizing mineral, is present in two of the seven samples with concentrations of 0.3% and 1.3%.
- Total elemental arsenic in excess of 1,000 parts per million (ppm) is reported in six of the 21 samples across various lithologies. Total elemental lead in excess of 1,000 ppm is reported in two of the 21 samples from the granodiorite rock type.
- The pH of leachate generated by the SPLP is predominantly slightly alkaline to alkaline. In all, 20 of the 21 samples report values above the upper Reference Value threshold of 8.5. A total of six samples report arsenic concentrations above the Reference Value of 0.15 mg/L. A correlation with total element arsenic concentrations (exceedences of 1,000 ppm) is observed in four of the six samples. A total of five of the 21 samples reported iron concentrations above the Reference Value of 1.0 mg/L. Isolated exceedences of copper, lead, and zinc were also reported from the granodiorite rock type.
- A total of six of the 21 samples are classified as potentially acid generating across a range of rock types. Furthermore, NAG pH results show a wide range of values between 2.8 to 11.0, with a total of three of the 21 samples reporting a value less than 4.5. Insufficient acid neutralization capacity exists in the majority of the represented samples to counteract acidity that may theoretically be generated as a result of weathering processes. Appreciable acid neutralizing potential is observed in only two of the 21 samples.

25.0 INTERPRETATION & CONCLUSIONS

25.1 GEOLOGY

Three main rock units underlie the Property, including rocks of the Fairbanks Schist, rocks of the Chatanika Terrane, and intrusive rocks (**Figure 7-2**). The Fairbanks Schist and Chatanika Terrane have both been subjected to one or more periods of regional metamorphism. The intrusive bodies are post-metamorphism. Chatanika Terrane rocks are found structurally above the Fairbanks Schist and north of the Chatanika Thrust fault and comprise the northernmost portion of the property. Intrusive rocks are relatively minor on the Property, and are primarily represented by the Dolphin stock, although small granitic dikes are known in several locations.

The Dolphin stock is located on the ridge between Bedrock and Willow Creek. Initial diamond core logging identified five intrusive phases within the Dolphin stock, including: 1) fine- to medium-grained, equigranular to weakly porphyritic biotite granodiorite; 2) fine- to medium-grained, equigranular to weakly porphyritic hornblende-biotite tonalite; 3) fine-grained biotite granite porphyry; 4) fine-grained biotite rhyolite to rhyodacite porphyry; and 5) rare fine-grained, chlorite-altered mafic dikes (Adams and Giroux, 2012).

Limited drill data suggests the north and west contacts of the Dolphin stock are fault contacts (Adams and Giroux, 2012). The south and east contacts are largely intrusive contacts with minor faulting

25.2 MINING

Mine production constraints were imposed to ensure that mining wasn't overly aggressive with respect to the equipment anticipated for use at the Project. The schedule has been produced using mill targets and stockpiling strategies to enhance the project economics. The constraints and limits used are reasonable to support the project economics.

Pit designs were created using 10 m benches for mining with a catch bench every level. This corresponds to the resource model block heights, and Tetra Tech believes this to be reasonable with respect to mining loss and the equipment anticipated to be used in mining.

25.3 GROUNDWATER HYDROGEOLOGY

Estimates of groundwater conditions at the project site are based on records from existing groundwater wells at and near the Project site and on conditions observed at the Fort Knox mine, which is approximately 5 km (3 mi) to the south of the project site and is considered to provide a good representation of the conditions at the project site.

Groundwater is expected to be present in two units: unconsolidated deposits consisting of alluvium and dredge tailings along the valley floors, and fractured bedrock throughout the property. The degree of bedrock fracturing, and therefore the hydraulic conductivity, are expected to be highly variable. Reported depths to groundwater in nearby water wells ranged from 2.1 m (6.9 ft) below the land surface in the valley bottoms to 68.6 m (225 ft) below the land surface in upland areas. Reported yields of water supply wells ranged from 16 to 491 m^3/day (3 to 90 gpm), and dewatering wells at the Fort Knox mine were reported to have capacities up to approximately 1,000 m^3/day (183 gpm). Groundwater flow on a local scale is anticipated to be from bedrock in the upland areas toward the valleys and thence down-valley in the alluvial deposits or dredge tailings. Regional-scale groundwater flow cannot be determined from available data.

Planned open pit mining at the property would extend below the water table, and dewatering would be required for maintaining pit wall stability and dry conditions within the pit. Because of weather conditions, a well system would likely be the most feasible dewatering method. The mine pit would intersect the water approximately six months to one year after the start of mining, but dewatering would need to start earlier in order for the pumping effects to extend throughout the required area. The estimated annual average pumping rate was approximately 410 m^3/day (75 gpm) initially, increased to approximately 4,460 m^3/day (818 gpm) by the third year of mining, declined slightly through the eighth year of mining, and then increased gradually to approximately 6,600 m^3/day (1,210 gpm) near the end of the mine life. The number of wells required for dewatering is estimated to range from two initially to 16 later in the mine life.

Data would need to be collected to characterize the site-specific hydrogeologic conditions and develop site-specific designs for dewatering.

25.4 METALLURGY & PROCESS

Sufficient metallurgical testwork has been completed on samples from the Project deposit to determine the preferred processing methods to recover gold from oxide and sulfide materials at a PEA level study. The oxide material was shown to be highly amenable to heap leaching. The testwork showed that oxidation of the sulfide material was needed to achieve acceptable gold recoveries. The oxidation methods tested were able to achieve acceptable recoveries, but high capital cost requirements made those methods un-feasible for this PEA study. The processing method chosen for the sulfide material was bio-oxidation followed by cyanide leaching. While bio-oxidation testwork has not been performed on the deposit material, the high recoveries achieved throughout the testwork indicate that the sulfide material would be amenable to bio-oxidation.

25.5 ENVIRONMENTAL

Development of the project will require extensive environmental baseline analyses, assessment of environmental impacts and evaluation, and associated permitting requirements reflective of the direct, indirect and cumulative impacts associated with full project build-out, and the sensitive environment in which it is to be constructed. The complexity of the environmental impact review and permitting of the various facilities will be dependent on siting of facilities in relationship to the various creeks and valleys surrounding the project development target areas. This PEA provides preliminary siting information of facilities such as tailings disposal, waste rock, and leach pads. Baseline and environmental studies that will be required to move the project toward permitting can now be planned, implemented, and modified as necessary as the project progresses through the prefeasibility and feasibility planning process.

Required environmental data for this Project will include on-the-ground studies to delineate jurisdictional wetlands. These data will be required to meet a number of needs including permitting, mine design, location of facilities, mine construction and operations. Freegold Ventures, Ltd. has initiated consultation with the State's Large Mine Permitting Team (LMPT) to begin the process of project planning, development and environmental permitting. Through this process the LMPT will assist in developing a broader environmental baseline program.

26.0 RECOMMENDATIONS

Based on the results of the PEA, and the resultant economic evaluation, it is recommended that this study be followed by a preliminary feasibility study in order to further assess the economic viability of the Project. Additional drilling, metallurgical testing, environmental analyses, other permitting and property confirmation activities will need to be undertaken as part of this next level of study. The approximate cost of this study is estimated at $700,000.

The recommendations are designed to further advance the Project and as such should be undertaken independently of each part of the program. Total recommended program is budgeted at $8.5 million.

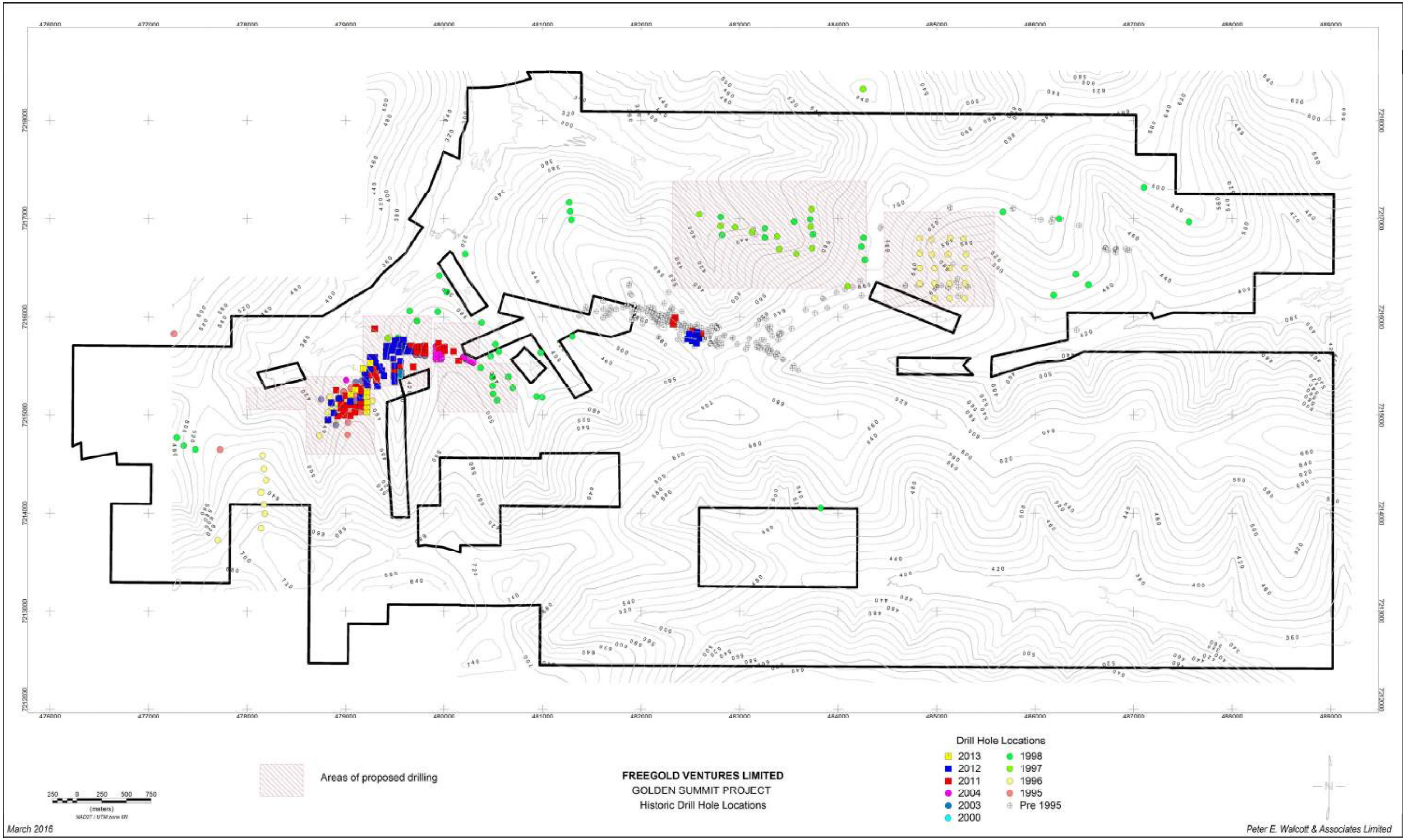
Table 26-1: Project Recommendations

Task	Total Recommended Costs ($000s)
Geology & Resources	$3,000
Mining	$1,000
Groundwater Hydrogeology	$500
Water Management	$120
Metallurgy & Process	$250
Tailings Storage Facility	$250
Heap Leach Facility	$150
Geochemical	$400
Environmental Permitting & Regulatory Compliance	$2,800
Total	**$8,470**

26.1 GEOLOGY

It is recommended that the following actions be initiated to further understanding of the resource:

- Increase the Dolphin/Cleary Hill gold resource by a) drilling shallow to moderate depth holes in un-tested areas adjacent to the southwest, north and west portions of the deposit, b) drilling a limited number of exploration drill holes in locations more distal to the resource, and c) drilling strategically located infill drill holes to move more ounces into the drill indicated category. Exploration drill holes should target areas where gold-bismuth anomalous soils are known to the south of the deposit and on the west side of Willow Creek, and areas where IP/resistivity survey data suggests the presence of possible shallow intrusive rocks to the southwest of the deposit. Approximately an additional 15,000 meters of drilling recommended for the Dolphin/Cleary area - approximate cost of this program is $3,000,000 (**Figure 26-1**).
- It is recommended that during future drill campaigns more attention is paid to getting representative specific gravity determinations from oxide material.



Source: Mark J. Abrams, Reno, NV

FIGURE 26-1
AREAS OF PROPOSED DRILLING

Geophysics has proven to be an effective tool in the Dolphin/Cleary Area. The resistivity lows track the alteration extremely well and match well with the coincident gold geochemistry. A significant gold geochemical anomaly has been delineated on the newly acquired Mental Health Trust Land. A ground geophysical survey should be carried out over that portion of the property. RAB drilling completed to the north of the current resource outlined an area of potential gold mineralization. Additional drilling is warranted to the west, north and southwest to expand upon the oxide portion of the current resource.

Additional drilling, metallurgical testing, environmental analyses and studies, and other and property acquisition activities will need to be undertaken as the project moves toward preliminary feasibility.

26.2 Mining

- The current pit slope geotechnical is based on review of drill logs and site observations. A full geotechnical pit slope investigation should be conducted using the latest drilling information, the latest pit designs and a geotechnical borehole drilling/lab analysis program. The estimated budget for the geotechnical investigation is $500,000.
- In order to ensure that there is no economic resource under the planned MRSF and/or leach areas, a comprehensive condemnation drilling program should be instituted. The estimated budget for the condemnation drilling investigation is $400,000.
- Perform lab testing/analysis on the mineralized rock types to determine if a jaw crusher could be used instead of a gyratory crusher to potentially lower capital and operating costs. The estimated budget for the crushing investigation is $50,000.

26.3 Groundwater Hydrogeology

A hydrologic investigation and monitoring plan capable of providing baseline data for mine permitting and dewatering system design should be developed. The plan should be based on basic data requirements for developing a conceptual site model for groundwater flow and quality as well as in consideration of dewatering design, water supply and permitting needs. Baseline hydrogeologic data collection would include on-site testing to determine hydraulic properties of the rock units and geologic structures, as well as groundwater monitoring for water levels and quality. Such a system should also allow ongoing monitoring during the mine life.

A basic groundwater monitoring system should include monitoring wells in bedrock and alluvium upgradient and downgradient from planned disturbance areas. Initially, all wells would provide data on background conditions unaffected by mining activity associated with the Project. This should include baseline groundwater quality in areas previously disturbed by placer mining. During the Project life, wells upgradient of planned disturbance areas would monitor groundwater quality conditions unaffected by the Project activities, while wells downgradient from planned disturbance areas would monitor for water quality changes in areas potentially affected by the Project activities.

For dewatering system design, at least one aquifer test within the proposed pit footprint would be required. Hydrogeologic investigation plans should be incorporated into the ongoing exploration drilling program. Cost savings could be realized in that manner. Exploration core holes often can provide valuable hydrogeologic information if routine but specific efforts are made to 1) collect and record appropriate data (such as groundwater occurrences, static water levels, factures, and other high-permeability zones as indicated by drilling fluid loss or lost circulation) while drilling is in progress, and 2) utilize exploration boreholes for hydrologic testing and piezometer or monitoring well construction.

When site-specific hydrogeologic information becomes available, mine dewatering plans and cost estimates based on the site-specific data should be developed.

The cost of developing and implementing a hydrologic investigation and monitoring plan as described above is estimated to be $500,000. The cost estimate is based on the assumptions that eight monitoring wells (three to approximately 75 m [250 ft], two to approximately 30 m [100 ft] and three to approximately 15 m [50 ft]) and one pumping well (to 150 m [500 ft]) would be installed and slug-tested, two 1-day and one 7-day aquifer tests would be conducted on selected wells, water-level data loggers would be installed for continuous monitoring of water levels in all the wells, and water-quality samples would be collected quarterly for one year. Water-quality samples would be analyzed for the following parameters:

- Metals (dissolved and total): aluminum, antimony, arsenic, cadmium, calcium, chromium, cobalt, copper, iron, lead, magnesium, manganese, molybdenum, nickel, potassium, selenium, silver, sodium, and zinc;
- Metals (total): barium, beryllium, boron, lithium, mercury (low-level), thallium, and vanadium; and
- General Chemistry: pH (field and lab), specific conductance (field and lab), temperature (field and lab), hardness, total alkalinity, total acidity, total dissolved solids, chloride, sulfate, carbonate, bicarbonate, fluoride, ammonia as N, nitrate plus nitrite as N, total organic carbon, dissolved organic carbon, cyanide (total and WAD).

26.4 WATER MANAGEMENT

Tetra Tech recommends the following to improve accuracy, performance and overall quality of water management and infrastructure for the Project site:

- Development of a detailed site-wide water balance based on the production process, expectations for tailings and waste rock characteristics, advancement of the heap leach operation and greater understanding of camp and domestic water usage ($15,000);
- Performing environmental base line studies to characterize site specific meteorology, soils and hydrology ($55,000); and
- Installation of instrumentation to collect site specific meteorological data ($50,000).

26.5 METALLURGY & PROCESS

In the ongoing effort to progress the project, Tetra Tech recommends that the following metallurgical testwork be performed on representative samples for subsequent engineering studies:

- Bench scale bio-oxidation testwork on all identifiable sulfide material types. The oxidation testwork should explore the following:
  - Grind size vs. recovery relationship;
  - Comparative tests on different bacteria types;
  - Reagent dosages and consumptions;

- Temperature;
- Acid generation; and
- Oxidation kinetics.

- Leaching testwork subsequent to oxidation should follow the same protocols as previous testwork.
- Heap leach column tests on the oxide material. This testwork should expand upon the existing oxide testwork by testing multiple areas of the deposit. The column tests should be conducted using larger scale columns than previous testwork. The tests should also be conducted at different ambient temperatures to determine the effect of the sub-arctic conditions on leaching kinetics.
- Additional comminution testwork on the various different sulfide material types. These tests should include Bond Ball Mill Grindability, Bond Abrasion Index, JKTech Drop Weight, and JKTech SMC tests.

The cost for the testwork programs described is approximately $250,000.

26.6 Tailings Storage Facility

The following items are recommended to advance the preliminary design of the TSF as part of a Preliminary Feasibility Study:

- A trade-off study of alternate tailings storage methods should be undertaken that includes consideration of thickened and dry stack approaches.
- A subsurface geotechnical investigation including materials characterization via field and laboratory testing should be performed at the proposed footprint to assess foundation conditions and potential construction materials. The assessment of potential permafrost conditions should be undertaken as part of this investigation. Geotechnical characterization of tailings samples should be undertaken.
- Geotechnical stability and seepage assessment of select stages of the TSF development should be undertaken that include thermal analyses and potential ice entrainment considerations.
- Geochemical assessment of tailings and mine waste.
- The design of containment features should be developed based on seepage and stability assessments that consider material properties, site conditions, and regulatory requirements. Contaminant fate and transport modelling should be undertaken to support determination of containment requirements.
- Design of water management features, including diversion size and alignment, and incorporating seasonal climate and mine site water balance considerations.
- A staged construction plan should be developed with consideration of site features, climate, and the mine schedule.
- Geotechnical and environmental monitoring plan developed that includes consideration of monitoring instrument type and position, and locations of groundwater monitoring wells.

It is estimated that $250,000 would be required for the PFS design and tailings tradeoff study.

26.7 HEAP LEACH FACILITY

The following items are recommended to advance the preliminary design of the Heap Leach Facility as part of a Preliminary Feasibility Study:

- A subsurface geotechnical investigation including materials characterization via field and laboratory testing should be performed at the proposed footprint of the facility to assess foundation conditions and potential construction materials. The assessment of potential permafrost conditions should be undertaken as part of this investigation. Geotechnical characterization of heap leach samples should be undertaken.
- Geotechnical stability and seepage assessment of select stages of the Heap Leach Facility development should be undertaken that include thermal analyses considerations.
- The design of containment features should be developed based on seepage and stability assessments that consider material properties, site conditions, and regulatory requirements.
- Design of water management features, including diversion size and alignment, and incorporating seasonal climate and mine site water balance considerations.
- A preliminary stacking plan and proposed haul road and pipeline alignments should be developed with consideration of site features and the mine schedule. The following provisions for seasonal stacking should be considered:
    - Sizing of the crushing operation and haul fleet to allow increased production rate during warm months;
    - Sizing of the starter heap leach pad to accommodate more than 1 year of mineralized material production to allow advanced stacking for the first winter season;
    - Provision for ripping frozen material prior to leaching; and
    - Provision for temporary over-irrigation to melt potential ice layers in the heap.
- Geotechnical and environmental monitoring plan developed that includes consideration of monitoring instrument type and position, and locations of groundwater monitoring wells.
- The closure and reclamation plan should be developed in accordance with design guidelines and regulatory requirements.
- Seepage flow model for heap leaching process that considers temperature effects on leaching, mineralized material placement schedule.

It is estimated that $150,000 would be required for the PFS design.

26.8 Geochemical Recommendations

Preliminary geochemical testing indicates that some of the waste rock is likely to generate acid drainage and metal leachate. As mine planning progresses, additional geochemical testing is required to support waste rock management to minimize the generation of deleterious drainage that may require water treatment through both operational and closure phases. The following testing is recommended to support a PFS-level study:

- Additional static testing to reflect the waste proportions and tonnages of rock type that will comprise the waste rock facility. The geochemistry team will evaluate the proposed proportional tonnages of each rock type and then calculate the number of samples required to support decision making at the PFS level. The available core data will then be reviewed and representative samples will be selected for static testing.
- Kinetic testing involves weathering tests to aid prediction of drainage quality from mine wastes. Two rates are obtained from kinetic testing - Weathering Rate (rate (mass per unit time) at which a primary mineral is transformed into a secondary product (soluble species or insoluble mineral) and Release Rate (the mass efflux (per unit mass of bulk rock) of an element or species away from a unit mass of rock, per unit time). As there is no single test that produces all of the chemical information required to evaluate all mine wastes under all conditions of disposal, the objectives of the testing will be discussed with the mine planners and an appropriate kinetic test method will be selected to best simulate site conditions. It is recommended that kinetic testing be undertaken on individual lithologies as well as lithology combinations to understand the interaction of acid generation and neutralization to minimize deleterious drainage generation.

Approximate costs to do this work will be $400,000.

26.9 Environmental Permitting & Regulatory Compliance

Baseline and environmental studies that would be required to move the Project toward permitting can now be planned, implemented, and modified as necessary as the Project progresses through the preliminary feasibility and feasibility planning process estimated a total cost of $2,800,000. The following items are recommended to advance the Project as part of a Preliminary Feasibility Study:

- Freegold has initiated consultation with the State's Large Mine Permitting Team (LMPT) to begin the process of project planning, development and environmental permitting. Through this process, the LMPT would assist in developing a broader environmental baseline program.
- Owing to the long lead time for data collection, it is important that the baseline program generates adequate data in terms of type, quality and quantity. For this reason, baseline studies to support environmental impact assessments under NEPA and environmental permitting should be initiated. Primary initial studies should include both desk-top and ground verification wetlands delineation studies to support CWA Section 404 permitting, meteorological monitoring for air quality permitting, characterization of site ground and surface water hydrology and water quality, and flora and fauna studies. As previously described, the major environmental driver for the Project would be the issuance of a CWA Section 404 permit (wetlands) which will require an impacts assessment under NEPA. Several years of environmental baseline studies are required in order to support an EA or EIS.

- Additional samples would verify the preliminary geochemical results from this study and assist in better understanding the potential for acid generation and metal leaching. The additional sampling is necessary to reflect the waste rock proportions and tonnages of rock types that would comprise the mine rock storage facility.
- As the program advances, a need to focus on acquiring representative rock type samples should be undertaken. By acquiring an accurate representative lithology apportionment of the waste rock storage facility a more thorough classification and understanding of waste rock leaching chemistry and acid generation can be realized. Further, subsequent waste rock handling can be better planned and executed. Using the planned mine schedule in association with the tonnages and proportion of waste rock types a more representative sample set would be selected for the next phase of testing.
- The cost of initiating a meteorological monitoring program is approximately $150,000 in the first year followed by approximately $30,000 per year following. Wetland delimitation studies are estimated at $100,000 but could vary depending on available aerial photographic data. The estimated costs of groundwater characterization studies were previously discussed and estimated to be $464,000. Surface water characterization studies are estimated to be approximately $80,000 per year and initial flora and fauna studies could be initiated for approximately $30,000.
- The costs of preparing an Environmental Impact Study for recent similar mining projects in Alaska have ranged between $1 million and $2 million. However, it is estimated that with adequate baseline characterization studies, the project impact assessment would be at the lower end of this range. Final permitting is estimated to be approximately $500,000.
- Reclamation and closure costs will be developed as more detailed engineering and study work is completed that is sufficient to support a detailed reclamation and closure plan. Both ADNR and ADEC require financial assurance in conjunction with approval and issuance of large mine permits. ADNR, under authority of Alaska Statute 27.19, requires a reclamation and closure plan as well as financial assurance to assure reclamation of the site prior to construction. ADEC requires 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 must also include the property holding costs for a one-year period. The financial assurance amount is calculated through the process of reviewing and approving the Project reclamation and closure plan during the permitting process. Current financial assurance amounts for Alaska's six operating metal mines range from $6 million to $305 million; however, for comparison, the required financial assurance amount for the near-by open pit Fort Knox mine is $68 million.

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28.0 CERTIFICATE OF QUALIFIED PERSON

28.1 QUALIFICATIONS OF CONSULTANTS

The Consultants preparing this Technical Report are specialists in the fields of geology, exploration, Mineral Resource estimation and classification, mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in Freegold. The Consultants are not insiders, associates, or affiliates of Freegold. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Freegold and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions.

This Technical Report was prepared by the following QPs, certificates for whom are contained herein:

Name	Title, Company	Responsible for Sections
Mark J. Abrams, C.P.G.	Certified Professional Geologist	1.3, 1.4, 1.5, 2.0, 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, and 12.0
Jackie A. Blumberg, P.E.	Surface Water Hydrologist Tetra Tech, Inc.	18.8, 18.8.1, 18.8.2, 18.8.3, 18.8.4, 18.8.5, 24.2, 24.3, and 26.4
Gary H. Giroux, P.Eng.	Consulting Geological Engineer Giroux Consultants, Ltd.	1.6, 1.13.1, 14.0, 25.1, and 26.1
Chris Johns, M.Sc., P.Eng.	Geological Engineer Tetra Tech, Inc.	18.9, 18.10, 26.6, and 26.7
Edwin C. Lips, P.E.	Principal Mining Engineer Tetra Tech, Inc.	1.8, 1.13.2, 15.0, 16.0, 25.2, and 26.2
Nick Michael, QP	Principal Mineral Economist Tetra Tech, Inc.	1.11, 1.12, 19.0, 21.0, and 22.0
Dave M. Richers, PhD, PG	Geochemist / Geologist Tetra Tech, Inc.	24.4 and 26.8
Vicki J. Scharnhorst, P.E.	Principal Consultant Tetra Tech, Inc.	1.1, 1.2, 1.9, 1.10, 1.13.5, 1.14, 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.7, 20.0, 23.0, 25.5, 26.0, 27.0, and 28.0
D. Erik Spiller, QP	Principal Metallurgist Tetra Tech, Inc.	1.7, 1.13.4, 13.0, 17.0, 25.4, and 26.5
Keith Thompson, CPG, PG	Hydrogeologist Tetra Tech, Inc.	1.13.3, 18.8.6, 24.1, 25.3, and 26.3

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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Mark J. Abrams, C.P.G. of Reno, Nevada do hereby certify that:

1. I am a consulting geologist with an office at 604 Elko Summit Drive, Elko, Nevada 89801, USA.
2. I am a graduate of Eastern Washington University in 1978 with a B.S. degree; and in 1980 with a M.S. degree, both in Geology. I am a member in good standing of the American Institute of Professional Geologists #11451. I have practiced my profession continuously since 1979. I have 35 years of experience in all phases of mineral exploration and economic geology. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
3. I visited the property on May 25 and 26, 2012.
4. I am responsible for Sections 1.3, 1.4, 1.5, 2.0, 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, and 12.0 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Elko, Nevada.

Original document dated, signed and sealed by Mark J. Abrams, C.P.G.

Mark J. Abrams, C.P.G.
Consulting Geologist

CERTIFICATE OF QUALIFIED PERSON

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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Jackie A. Blumberg, P.E. of Golden, CO, do hereby certify that:

1. I am a Water Resource Engineer at Tetra Tech, Inc. located at 350 Indiana Street, Suite 500, Golden, CO 80401, USA.
2. I have a Master of Science degree in Civil Engineering from Utah State University (2000). I hold a Colorado Professional Engineering License (#43184). My relevant experience is that I have practiced my profession as a Water Resource Engineer for 15 years. I have practiced my discipline within the mining engineering framework for the past 3 years. I have provided engineering services for numerous mine projects in arid environments: Nevada, Arizona, Utah, New Mexico, Mexico and the Pilbara region of Australia. I am a "Qualified Person" for purposes of National Instrument 43-101.
3. I have not visited or inspected the property.
4. I am responsible for Sections 18.8, 18.8.1, 18.8.2, 18.8.3, 18.8.4, 18.8.5, 24.2, 24.3, and 26.4 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Golden, Colorado.

Original document dated, signed and sealed by Jackie A. Blumberg, P.E.

Jackie A. Blumberg, P.E.
Water Resource Engineer
Tetra Tech, Inc.

CERTIFICATE OF QUALIFIED PERSON

Gary H. Giroux, P.Eng.
Consulting Geological Engineer
982 Broadview Drive
North Vancouver, British Columbia, Canada
Telephone: 604 684-0899
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Gary H. Giroux, P.E. of North Vancouver, British Columbia, do hereby certify that:

1. I am a consulting geological engineer with an office at 1215-675 West Hastings Street, Vancouver, British Columbia.
2. I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia. I have practiced my profession continuously since 1970. I have had over 40 years experience calculating mineral resources. I have previously completed resource estimations on a wide variety of intrusive hosted gold deposits, including Brewery Creek, Kisladag and Red Mountain. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a profesional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
3. I have not visited or inspected the property.
4. I am responsible for Sections 1.6, 1.13.1, 14.0, 25.1, and 26.1 of the Technical Report.
5. I satisfy all the requirements of independence according to section 1.5 of NI 43-101.
6. Before being retained by Freegold Ventures, I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Vancouver, British Columbia.

Original document dated, signed and sealed by Gary H. Giroux, P.Eng.

Gary H. Giroux, P.Eng.
Consulting Geological Engineer

CERTIFICATE OF QUALIFIED PERSON

Chris Johns, P.Eng.
Geological Engineer
Tetra Tech, Inc. EBA
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Email: Chris.Johns@tetratech.com

To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Chris Johns, P.Eng. of Kelowna, British Columbia, do hereby certify that:

1. I am a Geological Engineer at Tetra Tech, Inc. EBA located at 150-1715 Dickson Avenue, Kelowna, British Columbia, Canada.
2. I am a graduate of Queen's University, Ontario with a Bachelor of Science degree in Geological Engineering (1994) and of the University of Alberta with a Master of Science degree in Environmental Engineering (1999). My relevant experience includes 18 years of geological engineering on projects involving design of waste containment facilities. I have been involved with tailings storage facility design from scoping study through feasibility and construction stage. I am a registered Professional Engineer in the Provinces of Alberta and British Columbia, and a Chartered Professional Engineer with the Institution of Engineers Australia. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
3. I have not visited or inspected the property.
4. I am responsible for Sections 18.9, 18.10, 26.6, and 26.7 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Kelowna, British Colombia.

Original document dated, signed and sealed by Chris Johns, P.Eng.

Chris Johns, P.Eng
Geological Engineer
Tetra Tech EBA, Inc.

CERTIFICATE OF QUALIFIED PERSON

Edwin C. Lips, P.E.
Principal Mining Engineer
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Edwin C. Lips, P.E. of Phoenix, Arizona do hereby certify that:

1. I am a Principal Mining Engineer at Tetra Tech, Inc. located at 350 Indiana Street, Suite 500, Golden, CO 80401, USA.
2. I am a graduate of Montana Tech (Bachelor of Science degree in Mining Engineering, 1982). I am a licensed Professional Engineer in good standing in the State of Nevada, license number 022863. My relevant experience is that I have practiced my profession as a mining engineer continuously since graduation, for a total of 33 years. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
3. I have visited and inspected the property on May 6, 2014.
4. I am responsible for Sections 1.8, 1.13.2, 15.0, 16.0, 25.2, and 26.2 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Phoenix, Arizona.

Original document dated, signed and sealed by Edwin C. Lips, P.E.

Edwin C. Lips, P.E.
Principal Mining Engineer
Tetra Tech, Inc.

CERTIFICATE OF QUALIFIED PERSON

Nick Michael, QP
Principal Mineral Economist
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Nick Michael, QP of Lakewood, CO, do hereby certify that:

1. At the time this report was prepared, I was a Principal Mineral Economist at Tetra Tech located at 350 Indiana Street, Suite 500, Golden, CO 80401.
2. I am a graduate of the Colorado School of Mines in Golden, Colorado USA in mining engineering (1983) and received a MBA from Willamette University (1986). I have practiced my profession continuously since 1987. Since 1990, I have completed valuations, evaluations (technical-economic models), and have audited a variety of projects including exploration, pre-production (feasibility-level), operating and mine closure projects. I have also served as expert witness with respect to technical-economic issues. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument"). I am also a Registered Member of the Society of Mining, Metallurgy, and Exploration (# 4104304) in good standing.
3. I have not visited or inspected the property.
4. I am responsible for Sections 1.11, 1.12, 19.0, 21.0, and 22.0 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Lakewood, Colorado.

Original document dated, signed and sealed by Nick Michael, QP

Nick Michael, QP
Principal Mineral Economist
Golder Associates Inc.

CERTIFICATE OF QUALIFIED PERSON

David M. Richers, QP, P.G.
Geochemist / Geologist
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, David M. Richers, QP, P.G. of Golden, CO, do hereby certify that:

1. I am a Geochemist/Geologist at Tetra Tech, Inc. located at 350 Indiana Street, Suite 500, Golden, CO 80401, USA.
2. I have been practicing my profession as a geologist/geochemist for over 41 years since receiving my BS degree in Geology from Pennsylvania State University in 1974. I also received an MS degree in Geology/Geochemist in 1977 from University of Kentucky, and a PhD degree in Geology/ Geochemistry from University of Kentucky in 1980. My relevant experience as a geologist and geochemist includes geochemical site characterization services and mine geology. I have worked on mining projects in the United States, Australia, Spain, and Canada including both surface and underground operations. My duties routinely included participation in geochemical studies and programs aimed at protecting the environment including quantification of geochemical processes for engineering design, closure planning and impact analysis. My background also includes extensive work with acid rock drainage and metal leaching (ARD/ML) and the associated fate and transport. I also have expertise in geologic computer mapping and 3D GIS. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument"). I am a Registered Member of the Society of Mining, Metallurgy, and Exploration (# 4174527) in good standing.
3. I have completed a personal inspection of the Property on May 6, 2014.
4. I am responsible for Sections 24.4 and 26.8 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Golden, Colorado.

Original document dated, signed and sealed by David M. Richers, QP, P.G.

David M. Richers, QP, P.G.
Geochemist / Geologist
Tetra Tech, Inc.

CERTIFICATE OF QUALIFIED PERSON

Vicki J. Scharnhorst, P.E., LEED AP
Principal Consultant
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Vicki J. Scharnhorst, P.E., LEED AP, of Golden, CO, do hereby certify that:

1. I am a Principal Consultant at Tetra Tech, Inc. located at 350 Indiana Street, Suite 500, Golden, CO 80401, USA.
2. I am a graduate of Kansas State University with a Bachelor of Science degree in Civil Engineering (1982). My relevant experience includes 30 years of civil engineering on infrastructure and water resource projects inclusive of water quality programs, environmental impact studies, permitting and civil works. I am a licensed Engineer in the states of Nevada, Michigan, Missouri and Colorado; a water right surveyor in the State of Nevada; a LEED Accredited Professional with the U.S. Green Building Council; and have served on the Nevada State Board of Professional Engineers and Land Surveyors. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
3. I have completed a personal inspection of the Property on May 6, 2014.
4. I am responsible for Sections 1.1, 1.2, 1.9, 1.10, 1.13.5, 1.14, 18.1, 18.2, 18.3, 18.4, 18.5, 18.6, 18.7, 20.0, 23.0, 25.5, 26.0, 27.0, and 28.0 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Golden, Colorado.

Original document dated, signed and sealed by Vicki J. Scharnhorst, P.E., LEED AP

Vicki J. Scharnhorst, P.E., LEED AP
Principal Consultant
Tetra Tech, Inc.

CERTIFICATE OF QUALIFIED PERSON

D. Erik Spiller, QP
Principal Metallurgist
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Telephone: 303-217-5700
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, D. Erik Spiller, QP of Golden, CO, do hereby certify that:

1. I am a Principal Metallurgist at Tetra Tech, Inc. located at 350 Indiana Street, Suite 500, Golden, CO, 80401, USA.
2. I am a graduate of the Colorado School of Mines, (Bachelor of Science degree in Metallurgical Engineering, 1970). I am a Qualified Professional (QP) member of the Mining and Metallurgical Society of America (MMSA #01021QP). In addition, I am a Registered (QP) member of Society for Mining, Metallurgy, and Exploration, Inc. (SME #3051820RM). My relevant experience is that I have worked as a metallurgical engineer in the mineral resource industry for more than 40 years. During this career I held responsible positions in process research, process development, engineering, and senior management. In addition, I have served as an Adjunct instructor (22 years) and as an appointed Research Professor (9 years) in the Metallurgical and Materials Engineering Department at the Colorado School of Mines, where I lecture in mineral beneficiation and direct graduate students conducting metallurgical research in my area of expertise.
3. I have not visited or inspected the property.
4. I am responsible for Sections 1.7, 1.13.4, 13.0, 17.0, 25.4, and 26.5 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016, at Golden, Colorado.

Original document dated, signed and sealed by D. Erik Spiller, QP

D. Erik Spiller, QP
Principal Metallurgist
Tetra Tech, Inc.

CERTIFICATE OF QUALIFIED PERSON

Keith Thompson, C.P.G.
Senior Hydrogeologist
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To accompany the Report Entitled: "NI 43-101 Technical Report, Golden Summit Project, Preliminary Economic Assessment, Fairbanks North Star Borough, Alaska, USA" (Technical Report), Resource Effective Date: May 31, 2013, Report Effective Date: January 20, 2016, Issue Date: March 10, 2016, and amended and restated May 11, 2016.

I, Keith Thompson, C.P.G., of Greeley, CO, do hereby certify that:

1. I am a Senior Hydrogeologist at Tetra Tech, Inc. located at 3801 Automation Way, Suite 100, Fort Collins, CO 80525, USA.
2. I am a graduate of Youngstown State University (Bachelor of Science degree in Geology, 1975). I am also a graduate of the University of Wyoming (Master of Science degree in Geology, specialization in Hydrogeology, 1979). I am an active member of the American Institute of Professional Geologists (C.P.G. #6005). I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
3. I have not visited or inspected the property.
4. I am responsible for Sections 1.13.3, 18.8.6, 24.1, 25.3, and 26.3 of the Technical Report.
5. I satisfy all the requirements of independence according to NI 43-101.
6. I have had no prior involvement with the property that is the subject of this Technical Report.
7. I have read National Instrument 43-101, Form 43-101F1, Companion Policy 43-101CP, and the Technical Report. The portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument, form, and companion policy.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated May 11, 2016 at Greeley, Colorado.

Original document dated, signed and sealed by Keith Thompson, C.P.G.

Keith Thompson, C.P.G.
Senior Hydrogeologist
Tetra Tech, Inc.