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NI 43-101 Technical Report
Mt Todd Gold Project
50,000 tpd Preliminary Feasibility Study
Northern Territory, Australia

PROJECT NO.: 117-8348001

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FORWARD-LOOKING STATEMENTS

This Technical Report contains forward-looking statements within the meaning of the U.S. Securities Act of 1933, as amended, and U.S. Securities Exchange Act of 1934, as amended, and forward-looking information within the meaning of Canadian securities laws. All statements, other than statements of historical facts, included in this Technical Report that address activities, events or developments that Vista expects or anticipates will or may occur in the future, including such things as, the Company's continued work on the Mt Todd gold project; that process improvements will result in lower operating costs, reduced power consumption, increased gold recovery and higher gold production; estimates of mineral reserves and resources; projected project economics, including anticipated production, average cash costs, before and after-tax NPV, IRR, capital requirements and expenditures, gold recovery after-tax payback, operating costs, average tonnes per day milling, mining methods procedures, estimated gold recovery, project design, and life of mine; that the Project is an advanced stage development project; average annual production overtime; commencement of commercial production; timing for construction and commissioning; exploration of new deposits at Mt Todd and the surrounding exploration areas; size of final product through the high pressure grinding roll crusher; potential costs or savings related to gas price; ability to convert Quigleys estimated mineral resources to proven or probable mineral reserves; grade of minerals at the Quigleys deposit; ability to add higher grade feed from the Quigleys deposit to the Project in its mid years; timing for and completion of the NI 43-101 technical report for the PFS; and other such matters are forward-looking statements and forward-looking information. The material factors and assumptions used to develop the forward-looking statements and forward-looking information contained in this Technical Report include the following: the accuracy of the results of the PFS, mineral resource and reserve estimates, and exploration and assay results; the terms and conditions of our agreements with contractors and our approved business plan; the anticipated timing and completion of a feasibility study on the Project and permissions including approval of the MMP; the potential occurrence of certain threatened species of flora, vegetation, and fauna within the mine site; the anticipated receipt of required permits; no change in laws that materially impact mining development or operations of a mining business; the potential occurrence and timing of a production decision; the anticipated gold production at the Project; the life of any mine at the Project; all economic projections relating to the Project, including estimated cash cost, NPV, IRR, and initial capital requirements; and Vista's goal of becoming a gold producer. When used in this Technical Report, the words "optimistic," "potential," "indicate," "expect," "intend," "plans," "hopes," "believe," "may," "will," "if," "anticipate," and similar expressions are intended to identify forward-looking statements and forward-looking information. These statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of Vista to be materially different from any future results, performance or achievements expressed or implied by such statements. Such factors include, among others, uncertainty of mineral resource estimates, estimates of results based on such mineral resource estimates; risks relating to cost increases for capital and operating costs; risks related to the timing and the ability to obtain the necessary permits, risks of shortages and fluctuating costs of equipment or supplies; risks relating to fluctuations in the price of gold; the inherently hazardous nature of mining-related activities; potential effects on Vista's operations of environmental regulations in the countries in which it operates; risks due to legal proceedings; risks relating to political and economic instability in certain countries in which it operates; as well as those factors discussed under the headings "Note Regarding Forward-Looking Statements" and "Risk Factors" in Vista's Annual Report Form 10-K as filed in February 2019 and other documents filed with the U.S. Securities and Exchange Commission and Canadian securities regulatory authorities. Although Vista has attempted to identify important factors that could cause actual results to differ materially from those described in forward-looking statements and forward-looking information, there may be other factors that cause results not to be as anticipated, estimated or intended. Except as required by law, Vista assumes no obligation to publicly update any forward-looking statements or forward-looking information; whether as a result of new information, future events or otherwise.

Cautionary Note to United States Investors

The United States Securities and Exchange Commission ("SEC") limits disclosure for U.S. reporting purposes to mineral deposits that a company can economically and legally extract or produce. This Technical Report uses the terms "Proven reserves" and "Probable reserves". Reserve estimates contained in this Technical Report are made pursuant to NI 43-101 standards in Canada and do not represent reserves under the standards of the SEC's Industry Guide 7 and may not constitute reserves under the SEC's newly adopted disclosure rules to modernize mineral property disclosure requirements, which became effective February 25, 2019 and will be applicable to the Company in its annual report for the fiscal year ending December 31, 2021. Under the currently applicable SEC Industry Guide 7 standards, a "final" or "bankable" feasibility study is required to report reserves, the three-year historical average price is used in any reserve or cash flow analysis to designate reserves and all necessary permits and government approvals must be filed with the appropriate governmental authority. Additionally, this Technical Report uses the terms "Measured resources", "Indicated resources", and "Measured & Indicated resources". We advise U.S. investors that while these terms are Canadian mining terms as defined in accordance with NI 43-101, such terms are not recognized under SEC Industry Guide 7 and normally are not permitted to be used in reports and registration statements filed with the SEC. Mineral resources described in this Technical Report have a great amount of uncertainty as to their economic and legal feasibility. The SEC normally only permits issuers to report mineralization that does not constitute SEC Industry Guide 7 compliant "reserves" as in-place tonnage and grade, without reference to unit measures. The term "contained gold ounces" used in this Technical Report is not permitted under the rules of the SEC. "Inferred resources" have a great amount of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that any or all part of an Inferred resource will ever be upgraded to a higher category. **U.S. Investors are cautioned not to assume that any part or all of mineral deposits in these categories will ever be converted into SEC Industry Guide 7 reserves.**

NOTE

All references to the term "ore" contained in this Technical Report refer to mineral reserves, not mineral resources.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

ACRONYMS, ABBREVIATIONS AND SYMBOLS

"	second (plane angle)
%	percent
‘	minute (plane angle)
<	less than
>	greater than
°	degree
°C	degrees Celsius
°F	degrees Fahrenheit
µg	micrograms
µg/L	micrograms per liter or parts per billion
µm	microns
µS/cm	microsiemens per centimeter
3D	three-dimensional
A	ampere
a	annum (year)
ABA	acid base accounting
AD	annual deduction
ADWG	Australian Drinking Water Guidelines
AGR	Australian Gold Reagents Pty. Ltd.
ALS	Australian Laboratory Services
Alternate Case	33,000 tpd
AN	Ammonium nitrate
ANE	Ammonium nitrate emulsion
ANFO	Ammonium nitrate fuel oil
ANZECC	Australian and New Zealand Environment Conservation Council
ANZMARC	Australian and New Zealand Marketing Academy
AOM	Australian Ores and Minerals Limited
AP	aeration/settling ponds
APW	Aerobic Polishing Wetlands
ARD/ML	acid rock drainage and metal laden leachates
ARMCANZ	Agriculture and Resource Management Council of Australia and New Zealand
AStrk	Along Strike
Au	gold
AUD	dollar (Australian)
Ausenco	Ausenco Limited
B	billion
Base Case	50,000 tpd Case

BCR	biochemical reactor
BFA	Bench face angle
bgs	below ground surface
BH	Bench height
BKK	Bateman Kinhill and Kilborne
BP	Batman pit
Bt	billion tonnes
BWi	Bond Ball Mill work index
CAPEX	capital expenditure or capital expense
CCE	Capital Cost Estimate
CCI	Chamber of Commerce and Industry
CCTV	closed circuit television
CDN	Canadian dollar
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Standards	Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards
CIP	carbon-in-pulp
cm	centimeters
cm^2	square centimeter
cm^3	cubic centimeter
CoA	chart of accounts
CRD	capital recognition deduction
CV	Construction Verification
CWi	Crusher work index
d	day
d/a	days per year (annum)
D&C	Design and Construct
d/wk	days per week
DDH	Diamond drillhole core
DH	drillhole
dmt	dry metric ton
DO	Dissolved oxygen
DoR	Department of Resources
DRDPIFR	Department of Regional Development, Primary Industry, Fisheries and Resources
DC	Dry Commissioning
DUST	dust suppression
DWi	Drop Weight index
E&I	Electrical and Instrumentation
EEE	eligible exploration expenditure

EFCE	Enhanced Factored Cost Estimate
EHS	Environment, Health and Safety
EIS	Environmental Impact Statement
EL	exploration licenses
EMP	Environmental Management Plan
EPBC	Australian Environmental Protection and Biodiversity Conservation Act of 1999
EPCM	Engineering procurement construction management
EQP	equalization pond
F_{80}	80% feed passing size
FIS	Free In Store
FLS	FLSmidth
FS	Feasibility Study
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
g/L	grams per liter
g/m^3	gram per cubic meter
g Au/t	grams gold per tonne
g/t	grams per tonne
G&A	general and administrative
Ga	billion years ago
GCL	geosynthetic clay liner
General Gold	General Gold Resources Pty. Ltd.
GHD	GHD Pty Ltd.
GJ	Gigajoule
gpm	gallons per minute (US)
GR	gross realization
GW	gigawatt
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m2)
HAZOP	Hazard and Operability
HCL	Hydrochloric Acid
HHV	Higher Heating Value
HLP	heap leach pad
HNO_3	nitric acid

HPGR	high pressure grinding rolls
HQ	88.9 mm drill rod (outer diameter)
hr	hour
HSEC	Health, Safety, Environment and Community
HV	Heavy vehicles
HW	hanging wall
Hz	hertz
IBC	Intermediate bulk containers
ICP	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-OES	Inductively Coupled Plasma Optical Emission Spectroscopy
in	inch
in^2	square inch
in^3	cubic inch
IP	Internet Protocol
IRA	Inner-ramp angles
IRR	Internal Rate of Return
IR	Industrial Relations
IT	Information Technology
ITV	interim trigger values
JAAC	Jawoyn Association Aboriginal Corporation
k	kilo (thousand)
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square meter
kg/m^3	kilograms per cubic meter
km	kilometer
km/h	kilometers per hour
km^2	square kilometer
koz	kilo-ounce
kPa	kilopascal
kt	kilotonne
KV	Kriging variance
kV	kilovolts
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
kW/sec	Kilowatts per second

L	liter
L/m	liters per minute
lb	pound(s)
LGOS	low grade ore stockpile
LIMS	Laboratory information system
LLDPE	linear low-density polyethylene
LoM	life of mine
LPM	low-permeability material
m	meter(s)
M	million
m bgs	meters below ground surface
m/min	meters per minute
m/s	meters per second
m^2	square meter
m^3	cubic meter
m^3/hr	cubic meter(s) per hour
MARC	maintenance and repair contract
masl	meters above mean sea level
Mb/s	megabytes per second
Mbm^3	million bank cubic meters
Mbm^3/a	million bank cubic meters per annum
mbsl	meters below sea level
MCC	Motor Control Center
MDA	Mine Development Associates
µg/L	micrograms per liter
MGA	Map Grid of Australia
mg	milligram
mg/L	milligrams per liter or parts per million
mg/L	milligrams per liter
MIF	Measured, Indicated, inferred
min	minute (time)
mL	milliliter
MLN	Mineral License Number
mm	millimeter
MMP	Mining Management Plan
mo	month
Moz	million ounces
Mpa	megapascal
mPa·s	centipoise

MPU	Mobile processing unit
MRT	Mining & Resource Technology Pty Ltd
Mt	million tonnes
Mt/a	million tonnes per annum
MTO	material take-off
Mtpy	million tonnes per year
MVA	megavolt-ampere
MW	megawatt
MWH	Montgomery Watson Harza (now Stantec)
N/mm^2	Newtons per square millimeter
NAG	Net Acid Generation
NAL	Northern Australian Laboratories
NaOH	sodium hydroxide
NaSH	sodium hydrosulfide
NAPP	net acid production potential
NHMRC	National Health and Medical Research Council
NI	National Instrument
Nm^3/h	Normal meters cubed per hour
NOI	Notice of Intent
NP	neutralization potential
NPI	Non Process Infrastructure
NPR	neutralizing potential ratio
NPV	Net Present Value
NQ	69.9 mm drill rod (outer diameter)
NRETAS	Natural Resources, Environment, the Arts and Sport
NRMMC	Natural Resource Management Ministerial Council
NSR	Net Smelter Return
NT	Northern Territory
NTEL	NT Environmental Laboratories
NTEPA	Northern Territory Environmental Protection Authority
ø	diameter
OC	operating costs
OH&S	Occupational Health and Safety
OP	open rotary holes
OPEX	operating expenditure or operating expense
OPGW	optical ground wire
oz	ounce
oz/a	ounces/annum
oz/d	ounces/day

P_{80}	80% product passing size, in microns or µm
P&ID	piping and instrumentation diagram
Pa	Pascal
Pacific Gold Mines	Pacific Gold Mines NL
PAG	potentially acid generating
PAH	Pincock Allen and Holt
PbS	galena
PC	Prime Cost
PCG	Pine Creek Geosyncline
pcg	Porphyry copper gold
PER	Public Environmental Report
PFS	Preliminary Feasibility Study
PGM	plant growth medium
POWER	POWER Engineers, Inc.
PP	Process Plant
ppb	parts per billion
ppm	parts per million
Project	Mt Todd Gold Project
PRP	Process Plant Retention Pond
PWC	Power and Water Corporation
PWP	Process Water Pond
QA/QP	Quality Assurance/Quality Control
QP	Qualified Person
R&R	Rest and recreation
RDi	Resource Development Inc.
RKD	RKD (Company Name)
RL	Sample name
RO	runoff pond
RoM	Run of Mine
RP	retention pond
rpm	revolutions per minute
RVC	reverse circulation drilling method
RWD	raw water dam
s	second (time)
SAPS	Successive alkalinity producing systems
SG	specific gravity
SMBS	sodium metabisulfite
SMC	SAG mill comminution
SME	Society for Mining, Metallurgy, and Exploration, Inc.

SMP	Structural, Mechanical and Piping
SOCS	Site of Conservation Significance
SoW	Scope of Work
SPX	SPX company name
SRE	Soil and Rock Engineering
st	short ton (2,000 lb)
st/d	short tons per day
st/y	short tons per year
S.U.	Standard unit
SWWB	Site-wide water balance
t	tonne (1,000 kg) (metric ton)
t/a	tonnes per year
t/d	tonnes per day
t/m^3	tonnes per cubic meter
Technical Report	this Preliminary Feasibility Study
TEM	technical economic model
Tetra Tech	Tetra Tech, Inc.
TKI	Thyssen-Krupp Industries
tpd	tonnes per day
tph	tonnes per hour
TSF	tailings storage facility
TTP	Coffey Services Australia Pty Ltd (trading as Tetra Tech Proteus)
TUNRA	The University of Newcastle Research Associates
TV	Trigger value
TWC	The Winters Company
UCS	Unconfined compressive strength
US$	U.S. dollar
V	volt
Vista	Vista Gold Corp.
Vista Australia	Vista Gold Australia Pty Ltd
VoIP	voice over Internet protocol
w/v	weight/volume
w/w	weight/weight
WA	Western Australia
WAD	weak acid dissociable
WC	Wet Commissioning
WDL	Waste Discharge License
WGC	World Gold Counsel
wk	week

WRD	waste rock dump
WTP	water treatment plant
WWTP	waste water treatment plant
XRD	x-ray diffraction
yd^3	cubic yard
ZnS	Sphalerite

UNITS OF MEASURE

All dollars are presented in U.S. dollars (US$) 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 yard (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

ABBREVIATIONS OF THE PERIODIC TABLE

actinium = Ac	aluminum = Al	americium = 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	einsteinium = Es	erbium = Er	europium = Eu
fermium = Fm	fluorine = F	francium = Fr	gadolinium = Gd	gallium = Ga
germanium = Ge	gold = Au	hafnium = Hf	hassium = Hs	helium = He
holmium = Ho	hydrogen = H	indium = In	iodine = I	iridium = Ir
iron = Fe	joliotium = Jl	krypton = Kr	lanthanum = La	lawrencium = Lr
lead = Pb	lithium = Li	lutetium = Lu	magnesium = Mg	manganese = Mn
meitnerium = 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	praseodymium = 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	

1.0 SUMMARY

1.1 Introduction

Vista Gold Corp. (Vista) retained Tetra Tech, Inc., along with JDS Energy & Mining, Inc. (JDS), Mine Development Associates (MDA), Resource Development Inc. (RDi), Tetra Tech Proteus (TTP), and POWER Engineers, Inc. (POWER) to prepare this preliminary feasibility study (PFS) for its Mt Todd Gold Project (the Project) in Northern Territory (NT), Australia. The PFS (Technical Report) evaluates the Base Case, a development scenario of a 50,000 tonne per day (tpd) processing facility. In addition, an Alternate Case was considered at 33,000 tpd with higher grades presented under **Section 24.0 – Other Relevant Data and Information**.

Vista and its subsidiary, Vista Gold Australia Pty Ltd (Vista Australia) entered into an agreement to acquire an interest in the Project located in NT, Australia on March 1, 2006. The acquisition was completed on June 16, 2006 when the mineral leases comprising the Project were transferred to Vista Australia and funds held in escrow were released. Vista Australia is the operator of the Mt Todd property.

The Mt Todd property contains a number of known occurrences of gold, which have been explored and/or exploited to various degrees. The largest and best-known deposits are the Batman and Quigleys deposits, both of which have had historic mining by prior operators. The Batman deposit has produced and been explored more extensively than the Quigley deposit. Vista has reported mineral resource estimates in accordance with National Instrument (NI) 43-101 Standards of Disclosure for Mineral Projects and Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards (CIM) for Mineral Resources and Mineral Reserves (CIM Standards) for the Batman and Quigley deposits and a mineral reserve estimate in accordance with NI 43-101 and CIM Standards for only the Batman deposit.

The primary purpose of this **Technical Report** is to provide updated material, scientific, and technical information based on additional data obtained from extensive metallurgic test work conducted in 2018 and 2019. The recent metallurgic test programs have confirmed: (1) the efficiency of ore sorting across a broad range of head grades and the natural concentration of gold in the screen undersize material prior to sorting; (2) the efficiency of fine grinding and improved gold leach recoveries at an 80% passing grind size of 40 microns; and (3) the selection of FLSmidth's (FLS) VXP mill as the preferred fine grinding mill.

The highlights of the PFS are presented in **Table 1-1**.

Table 1-1: PFS Highlights

Description	Years 1-5		Life of Mine (LoM) (13 years)	
	Annual Average	Total	Annual Average	Total
Average Plant Feed Grade (g-Au/t)	0.96		0.82	
Payable Gold (koz)	495	2,476	413	5,305
Gold Recovery (%)	92.3%		91.9%	
Cash Costs (US$/oz)	$575		$645	
AISC ($/oz)	$688		$746	
Strip Ratio (waste:ore)	2.65		2.52	
Initial Capital (US$ millions)			$826	
After-tax Payback (Production Years)			2.9	
After-tax $NPV_{5\%}$ (US$ millions)			$823	
IRR (Pre-tax / After-tax) (%)			23.4%	

NOTE: Economics presented using US$1,350/oz gold and a flat US$0.70:AUD1.00 exchange rate and assumes deferral of NT Royalty payments prior to payback, and sale of excess electric power during reclamation and realization of salvage values at the end of the mine life.

This Technical Report notes material items which may, or may not, affect the market price of Vista's securities. Updated information contained herein includes scientific, technical and economic information deemed material to the Project.

1.2 Location

The Project is located 56 kilometers (km) by road northwest of Katherine, and approximately 290 km southeast of Darwin in NT, Australia (**Figure 1-1**). Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory.

1.3 Property Description

Vista Australia is the holder of four mineral licenses (ML) MLN 1070, MLN 1071, MLN 1127, and MLN 31525 comprising approximately 5,544 hectares (ha). In addition, Vista Australia controls exploration licenses (EL) EL 29882, EL 29886, EL 30898, and EL 28321 comprising approximately 153,700 ha. **Figure 1-1** illustrates the general location of the tenements and the position of the Batman deposit.

The general arrangement for the Project is shown on **Figure 1-3**, and landforms and impoundments are described in **Table 1-2**.

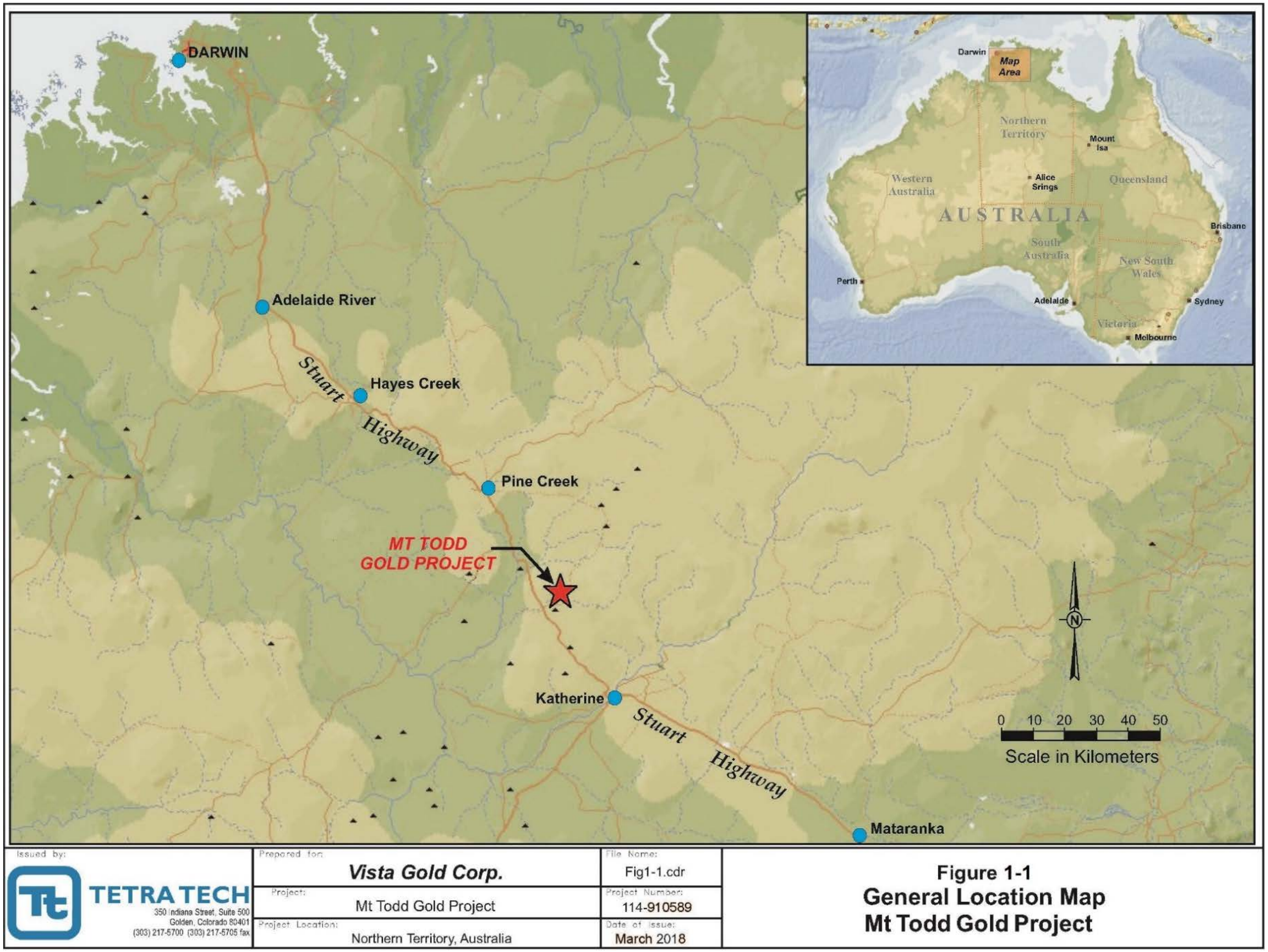
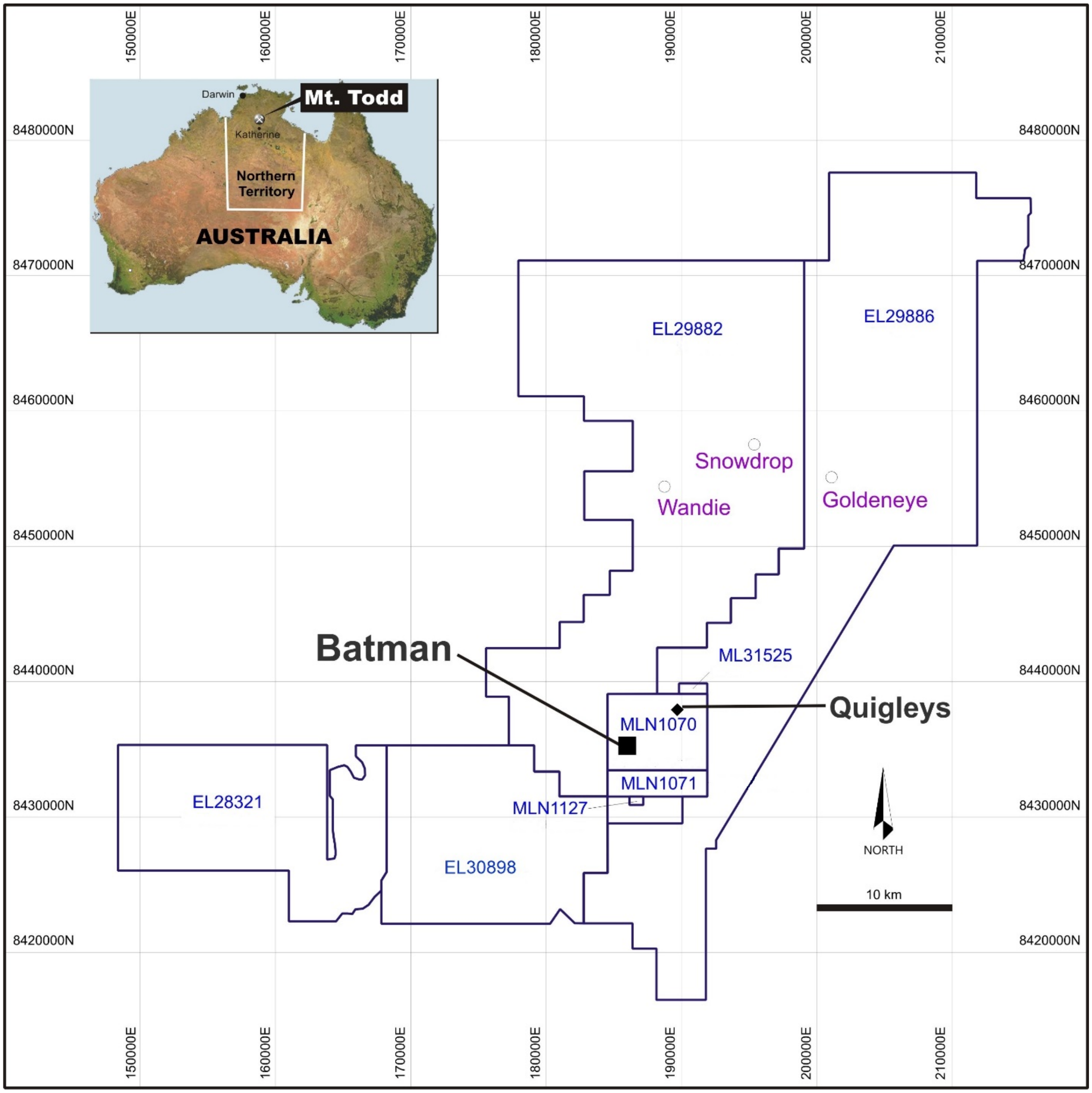


Figure 1-1: General Project Location Map

Table 1-2: Description of Landforms and Impoundments

Landform/Impoundment	Abbreviated Name
Tailings Storage Facility 1	TSF 1
Tailings Storage Facility 2	TSF 2
Raw Water Dam	RWD
Low Grade Ore Stockpile	LGOS
Low Grade Ore Stockpile Retention Pond	LGRP
Heap Leach Pad	HLP
Batman Pit	RP3
Process Plant Retention Pond	PRP
Waste Rock Dump	WRD
Waste Rock Dump Retention Pond	RP1
Process Water Pond	PWP
Water Treatment Plant	WTP
Process Plant	PP



NOTE: Prepared by Vista Gold Corp.; updated on February 23, 2018

Figure 1-2: Concessions

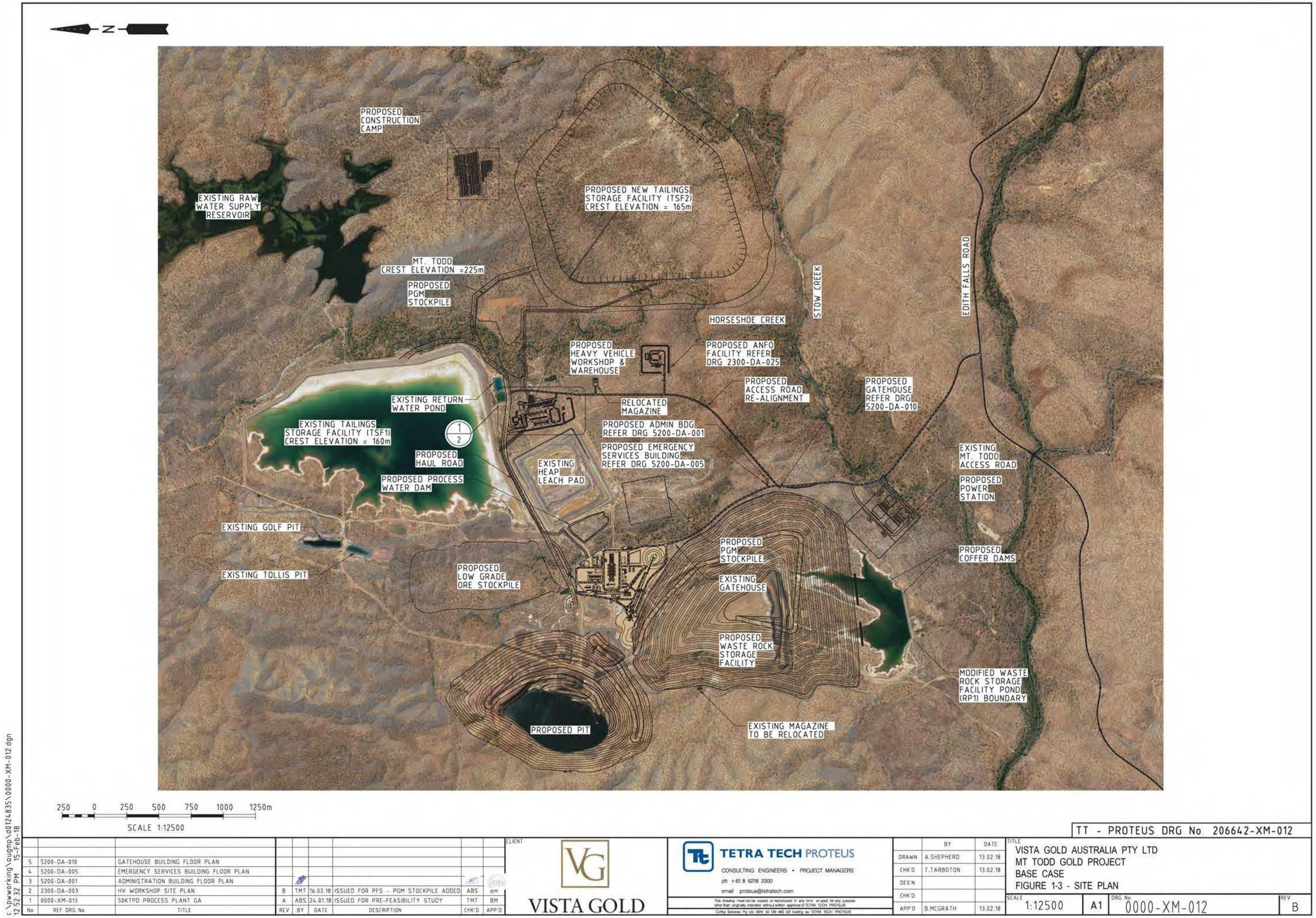


Figure 1-3: General Arrangement

1.4 Geology and Mineralization

The Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline (PCG). Meta-sediments, granitoids, basic intrusives, acid and intermediate volcanic rocks occur within this geological province.

The Batman deposit geology consists of a sequence of hornfelsed interbedded greywackes, and shales with minor thin beds of felsic tuff. Bedding is striking consistently at 325°, dipping at 40° to 60° to the southwest. Minor lamprophyre dykes trending north-south pinch and swell, crosscutting the bedding.

The deposits are similar to other gold deposits of the PCG and are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit shares some characteristics with intrusion-related gold systems, especially in terms of the association of gold with bismuth and reduced ore mineralogies. This makes the Batman deposit unique in the PCG. The mineralization within the Batman deposit is directly related to the intensity of the north-south trending quartz sulfide veining. The lithological units impact on the orientation and intensity of mineralization.

Sulfide minerals associated with the gold mineralization are pyrite, pyrrhotite and lesser amounts of chalcopyrite, bismuthinite and arsenopyrite. Galena and sphalerite are also present but appear to be post-gold mineralization and are related to calcite veining, bedding and the east-west trending faults and joints.

A variety of mineralization styles occur within the Project area. Of greatest known economic significance are auriferous quartz-sulfide vein systems. These vein systems include the Batman, Jones, Golf, Quigleys and Horseshoe prospects, which occur within a north-northeast trending corridor, and are hosted by the Burrell Creek Formation. Tin occurs in a north-northwest trending corridor. The tin mineralization comprises cassiterite, quartz, tourmaline, kaolin, and hematite bearing assemblages, which occur as bedding parallel to breccia zones and pipes. Polymetallic Au, W, Mo, and Cu mineralization occurs in quartz-greisen veins within the Yinberrie Leucogranite; a late stage highly fractionated phase of the Cullen Batholith. The Batman deposit extends approximately 2,200 meters (m) along strike, 400 m across dip and drill tested to a depth of 800 m. Drilling indicates the Batman mineralization to be open along-strike and down-dip.

To date, with regard to the exploration licenses (ELs), they represent an early-stage exploration program which has not produced an announceable discovery. While the work is promising and will be ongoing, there are no quantifiable resources or reserves on the ELs. Once an announceable discovery is made, Vista will detail that discovery according to all applicable disclosure regulations.

1.5 Mineral Resource Estimate

The following sections summarize the process, procedures, and results of Tetra Tech's independent estimate of the contained gold resources of the:

1) Batman deposit;
2) Existing heap leach pad; and
3) Quigleys deposit.

The resource estimate for the Batman deposit is updated from the July 7, 2014 Amended & Restated *NI 43-101 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia*, Technical Report prepared by Tetra Tech. This report includes an estimate of gold contained in a historic heap leach pad adjacent to the Batman deposit. Additionally, this report contains the resource estimation of the Quigleys deposit. The updated Project resource estimates are shown in **Table 1-3,** grade tonnage curve for the measured and indicated resource for the Batman deposit is presented in **Figure 1-4**.

Table 1-3: Statement of Mineral Resources Estimates

	Batman Deposit (August 2017)			Heap Leach Pad (May 2013)			Quigleys Deposit (August 2017)		
	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)
Measured (M)	77,725	0.88	2,191	-	-	-	457	1.27	19
Indicated (I)	200,112	0.80	5,169	13,354	0.54	232	5743	1.12	207
Measured & Indicated	**277,837**	**0.82**	**7,360**	**13,354**	**0.54**	**232**	**6,200**	**1.13**	**225**
inferred (F)	61,323	0.72	1,421	-	-	-	1,600	0.84	43

NOTES:

(1) Measured & indicated resources include proven and probable reserves.

(2) Batman and Quigleys resources are quoted at a 0.40g-Au/t cut-off grade. Heap Leach resources are the average grade of the heap, no cut-off applied.

(3) Batman: Resources constrained within a US$1,300/oz gold Whittle™ pit shell. Pit parameters: Mining Cost US$1.50/tonne, Milling Cost US$7.80/tonne processed, G&A Cost US$0.46/tonne processed, 50K TPD Ore, 355 Days/Yr., TPY Ore 17,750,000 TPY, G&A/Year 8,201 K US4, Au Recovery, Sulfide 85%, Transition 80%, Oxide 80%, 0.2g-Au/t minimum for resource shell. Selling Cost: US$/oz recovered US412.00.

(4) Quigleys: Resources constrained within a US$1200/oz gold Whittle™ pit shell. Pit parameters: Mining cost US$2.07/tonne, Milling Cost US$9.623/tonne processed, Sale Cost US$/oz US$15.18, Royalty 1% NPR , Gold Recovery All Types, 70%.

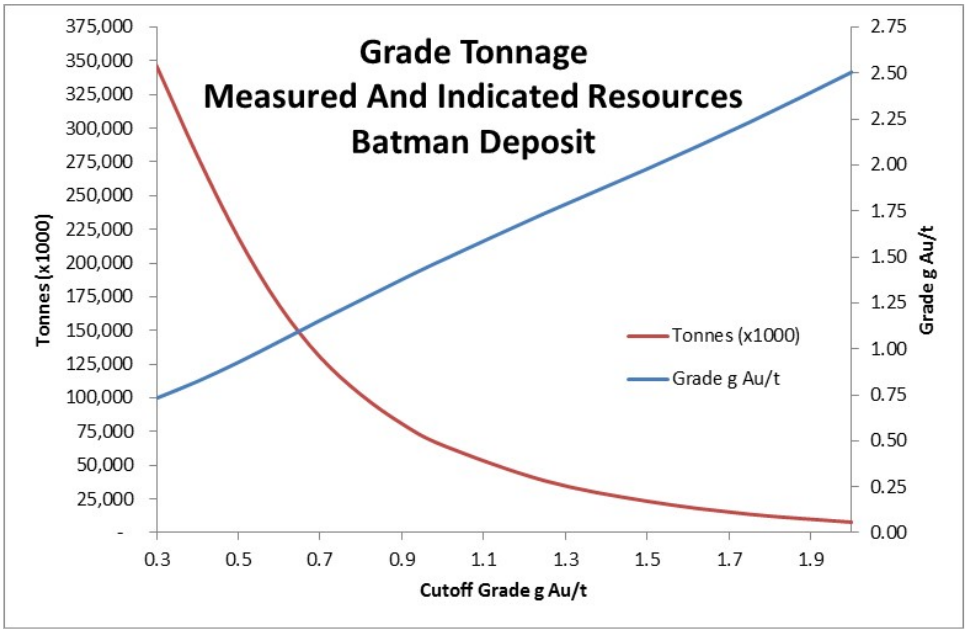
(5) Differences in the table due to rounding are not considered material

(6) Rex Bryan of Tetra Tech is the qualified person responsible for the Statement of Mineral Resources for the Batman, Heap Leach Pad and Quiqleys deposits.

(7) Thomas Dyer of Mine Development Associates is the qualified person responsible for developing the resource Whittle™ pit shell for the Batman Deposit.

(8) The effective date of the Batman and Quigleys resource estimate is August 2017, the effective date of the Heap Leach resource is May 2013.

(9) Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.



Source: Tetra Tech, Inc (August 2017).
NOTE: Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 1-4: Measured & Indicated Resource Estimates Grade Tonnage Curves – Batman Deposit

1.6 Mineral Reserve Estimates

Mine Development Associates (MDA) has used measured and indicated resources provided by Tetra Tech to estimate mineral reserves. Pit optimization was done using Geovia's Whittle^{TM} software to define pit limits with input for economic and slope parameters.

Optimization used only measured and indicated resources for processing. All inferred resource was considered as waste.

Varying gold prices were used to evaluate the sensitivity of the deposit to the price of gold as well as to develop a strategy for optimizing Project cash flow. To achieve cash-flow optimization, mining phases or push backs were developed using the guidance of Whittle^{TM} pit shells at lower gold prices.

The statement of mineral reserve estimates is shown in **Table 1-4**.

Table 1-4: Statement of Mineral Reserve Estimate

	Batman Deposit (January 2018)			Heap Leach Pad (May 2013)			Total P&P Reserves (January 2018)		
	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)
Proven	72,672	0.88	2,057	-	-	-	72,672	0.88	2,057
Probable	135,015	0.82	3,559	13,354	0.54	232	148,369	0.79	3,791
Proven & Probable	**207,687**	**0.84**	**5,616**	**13,354**	**0.54**	**232**	**221,041**	**0.82**	**5,848**

NOTES:
(1) Thomas L. Dyer, P.E., is the QP responsible for reporting the Batman Deposit proven and probable reserves.
(2) Batman deposit reserves are reported using a 0.40 g-Au/t cutoff grade.
(3) Deepak Malhotra is the QP responsible for reporting the heap-leach pad reserves.
(4) Because all of the heap-leach pad reserves are to be fed through the mill, these reserves are reported without a cutoff grade applied.
(5) The reserves point of reference is the point where material is fed into the mill.

1.6.1 Heap Leach Reserve Estimate

Existing heap leach pad (HLP) reserves are provided in **Table 1-4**, which are estimated to be 13.4 million tonnes (Mt). These reserves will be processed through the mill at the end of the mine life.

Previous test work indicated the following possible results :

- Cyanidation leach tests on “as is” material on the heap will extract ± 30% of the gold.
- CIP cyanidation tests at a grind size of P_{80} of 90 microns will extract on average 72% of gold (range: 64.14% to 80.37%) in 24 hours of leach time. The average lime and cyanide consumptions were 1.75 kg/t and 0.78 kg/t, respectively.

Vista is currently completing additional metallurgical test work at the target P_{80} 40 micron size. However, for the purposes of classifying the heap leach material as a reserve, the previous recovery values were used.

1.7 Mining Methods

The Project is designed to be a conventional, owner-operated, large open-pit mining operation that will use large- scale mining equipment in a drill/blast/load/haul operation. All dollar values in Section 1.7 are reported in US$.

A base gold price of US$1,250 per ounce was used for scenario analysis. However, various gold prices from US$300 to US$2,000 per ounce, in increments of US$20 per ounce, were used to determine different optimized pit shells. Economic parameters used for the pit designs are provided in **Table 1-5**.

Table 1-5: Initial Economic Parameters

Parameter	Base Case	Alternate Case
Gold Recovery	85% Sulfide 80% Transition 80% Oxide	85% Sulfide 80% Transition 80% Oxide
Payable Gold	99.9%	99.9%
Overall Mining Cost	US$1.90 per tonne mined	US$2.16 per tonne mined
Processing Cost	US$7.80 per tonne processed	US$8.65 per tonne processed
Tailings	US$0.90 per tonne processed	US$0.90 per tonne processed
General & Administrative	US$0.46 per tonne processed	US$0.77 per tonne processed
Water Treatment	US$0.09 per tonne processed	US$0.10 per tonne processed
Royalty	1% gross proceeds	1% gross proceeds

The mining costs used were varied by bench. An incremental cost of US$0.010 was added for each 6-meter bench below the 145 meter elevation. This represents the incremental increase in cost of haulage for both waste and ore for each bench that is to be mined. Reference mining costs of US$1.64 and US$1.86 were used for the Base and Alternate Cases, respectively. Additionally, an incremental cost was determined based on truck operating costs, truck cycle time to haul and return through a 6-meter gain in elevation, and truck capacity. The reference mining cost was determined using first principles from previous studies. The total mining cost (reference plus incremental) is US$1.90 and US$2.16 for the Base and Alternate Cases, respectively.

Processing, tailings construction, tailings reclamation, and water treatment costs were provided by Vista based on previous studies. Calculated cutoff grades based on the economic parameters are 0.38 and 0.33 g-Au/t for the Alternate and Base cases respectively. At Vista's request, MDA used a minimum cutoff grade of 0.40 g-Au/t for both the Alternate and Base cases. This was done to maintain higher grades with respect to material allowed to be processed.

Several iterations of pit optimizations were reviewed to determine the final pit limits. Ultimately, the pit limits were chosen to reflect the approximate total pit size from the previous PFS. For the Base Case a US$1,000/oz-Au pit shell was used to guide the ultimate pit design. For the Alternate Case the US$800/oz-Au pit shell was used to guide the ultimate pit design. **Table 1-6** shows the Whittle™ optimization results for the Base Case and **Table 1-7** shows the pit optimization results for the Alternate Case. Note that pit results for the base Au price used for pit optimization of US$1,250/oz-Au is highlighted in light green and the ultimate pit used for pit design is highlighted in orange.

Table 1-6: Whittle™ Pit Optimization Results – Base Case Using 0.40 g-Au/t Cutoff

Pit	Gold Price (US$)	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio
		K Tonnes	g-Au/t	K Ozs Au			
1	$ 300	3,282	1.77	186	2,797	6,078	0.85
6	$ 400	8,578	1.54	425	7,507	16,085	0.88
11	$ 500	15,988	1.34	686	15,740	31,728	0.98
16	$ 600	37,253	1.12	1,340	53,757	91,010	1.44
21	$ 700	89,301	0.99	2,855	171,617	260,918	1.92
26	$ 800	121,187	0.92	3,585	222,919	344,106	1.84
31	$ 900	159,485	0.87	4,442	316,889	476,374	1.99
36	$ 1,000	185,915	0.85	5,093	429,208	615,123	2.31
41	$ 1,100	212,340	0.84	5,741	566,907	779,247	2.67
46	$ 1,200	230,587	0.83	6,184	675,714	906,302	2.93
49	$ 1,250	234,858	0.83	6,278	700,519	935,376	2.98
51	$ 1,300	240,195	0.83	6,416	742,833	983,029	3.09
56	$ 1,400	243,306	0.83	6,498	771,190	1,014,497	3.17
61	$ 1,500	249,389	0.83	6,658	829,933	1,079,321	3.33
66	$ 1,600	254,050	0.83	6,779	880,583	1,134,633	3.47
70	$ 1,700	254,348	0.83	6,785	883,222	1,137,571	3.47
74	$ 1,800	259,140	0.83	6,908	943,012	1,202,152	3.64
78	$ 1,900	259,964	0.83	6,927	952,872	1,212,836	3.67
81	$ 2,000	260,099	0.83	6,929	953,985	1,214,083	3.67

Pit 36 was used for design purposes and Pit 49 illustrates the potential floating cone using a US$1,250/oz-Au price.

Table 1-7: Whittle^{TM} Pit Optimization Results – Alternate Case Using 0.40 g-Au/t Cutoff

Pit	Gold Price (US$)	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio
		K Tonnes	g-Au/t	K Ozs Au			
1	$ 300	966	1.88	59	731	1,697	0.76
6	$ 400	6,467	1.63	338	5,647	12,114	0.87
11	$ 500	11,133	1.46	523	10,299	21,432	0.93
16	$ 600	21,103	1.25	850	22,828	43,931	1.08
21	$ 700	60,702	1.06	2,078	115,883	176,585	1.91
26	$ 800	95,474	0.98	3,019	185,830	281,304	1.95
31	$ 900	121,301	0.92	3,586	222,805	344,106	1.84
36	$ 1,000	154,092	0.87	4,307	295,849	449,941	1.92
41	$ 1,100	185,525	0.85	5,085	427,833	613,358	2.31
46	$ 1,200	202,606	0.84	5,504	512,606	715,212	2.53
49	$ 1,250	212,904	0.84	5,753	569,384	782,289	2.67
52	$ 1,300	223,045	0.84	5,996	626,265	849,310	2.81
57	$ 1,400	234,858	0.83	6,278	700,519	935,376	2.98
62	$ 1,500	241,417	0.83	6,454	756,553	997,970	3.13
67	$ 1,600	247,350	0.83	6,605	809,868	1,057,218	3.27
71	$ 1,700	250,425	0.83	6,684	841,140	1,091,565	3.36
75	$ 1,800	254,050	0.83	6,779	880,583	1,134,633	3.47
80	$ 1,900	254,353	0.83	6,785	883,275	1,137,628	3.47
84	$ 2,000	259,140	0.83	6,908	943,012	1,202,152	3.64

Pit 26 was used for design purposes and Pit 49 illustrates the potential floating cone using a US$1,250/oz-Au price.

1.8 Metallurgy

The flowsheet consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using high pressure grinding rolls (HPGRs), ore sorting, two-stage grinding, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detoxification and disposal to conventional tailings storage facility (TSF).

Figure 1-5 provides the schematic diagram of the flowsheet.

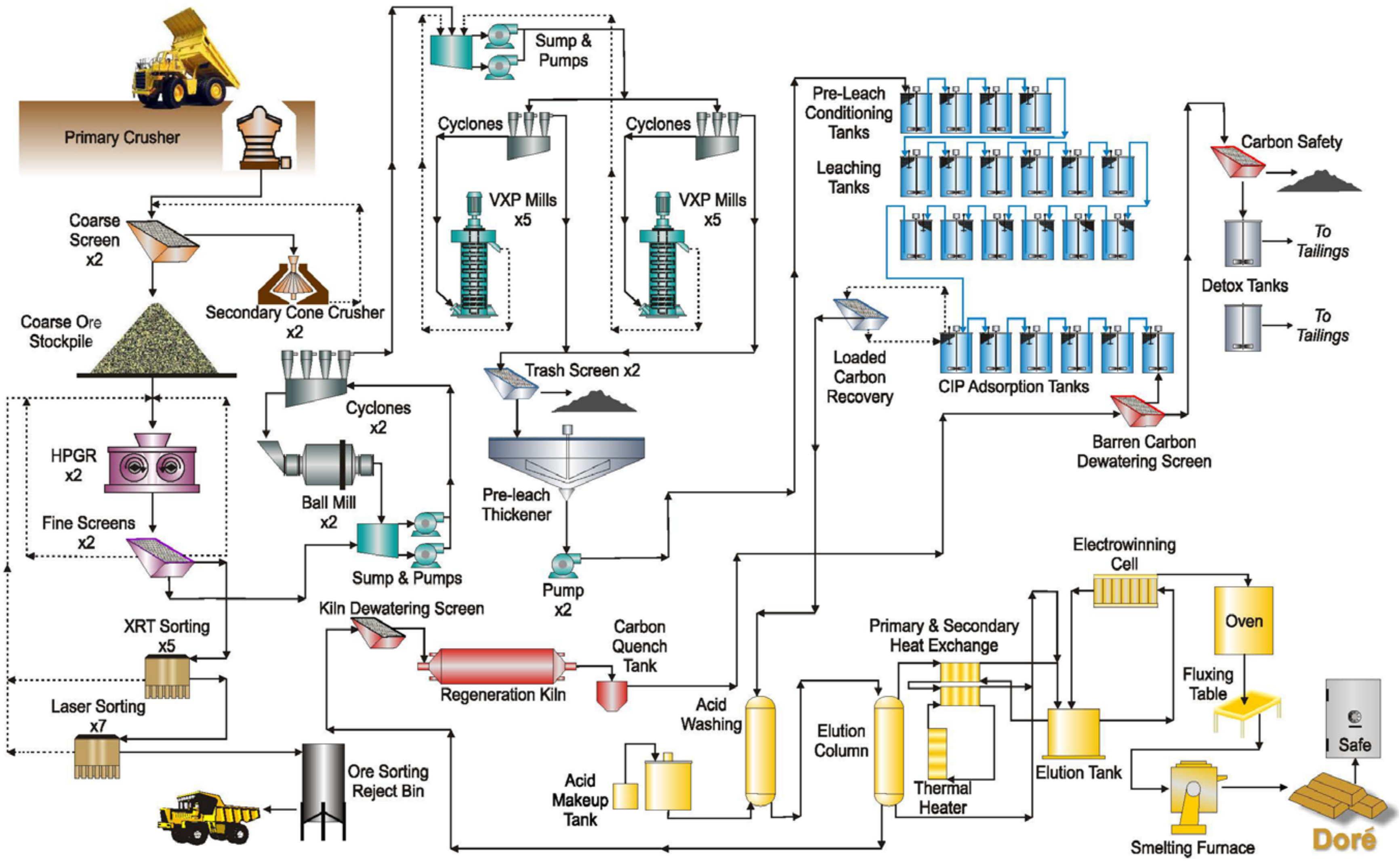


Figure 1-5: Mt Todd Flowsheet

1.9 Mineral Processing

Detailed design criteria have been developed for the process plant. The nominal headline design criteria are listed in **Table 1-8** below.

Table 1-8: Headline Design Criteria

	Unit	Base Case	Alternate Case
Annual Ore Feed Rate	Mt/a	17.75	11.72
Operating Days per Year	d/a	355	355
Daily Ore Feed Rate	t/d	50,000	33,000
Crushing Rate (6,637 hours per year availability)	tph	2,674	1,765
HPGR Rate (7,838 hours per year)	tph	2,264	1,495
Ore Sorting Rate (7,838 hours per year)	tph	408	318
Milling Rate (7,838 hours per year)	tph	2,055	1,332
Gold Head Grade[1]	g/t	0.82	0.82
Copper Head Grade	%	0.055	0.055
Cyanide Soluble Copper	%	0.0024	0.0024
Bulk Density	t/m^3	2.76	2.76
Primary Grind P_{80} to Secondary Grind	µm	250	250
Grind P_{80} to Leach	µm	40	40
Gold Recovery	%	91.9	91.9
Gold Production (average)[2]	oz/d	1,165	758
Gold Production (average)	oz/a	413,400	287,822

[1] *Weighted average between pit ore at 0.84 g Au/t and heap leach ore at 0.54 g Au/t]*
[2] *Based on block-by block total using constant tailing by specific feed grade range (Table 13-26)*

1.10 Project Infrastructure

Access to local resources and infrastructure is excellent. The Project is located sufficiently close to the city of Katherine to allow for an easy commute for workers. The area has both historic and current mining activity and therefore a portion of the skilled workforce will be sourced locally. In addition, Katherine offers the necessary support functions that are found in a medium-sized city with regard to supplies, accommodations, communications, etc.

The property has an existing high-pressure gas line and an electric power line that were used by previous operators. In addition, wells for potable water and a dam for process water are also located on or adjacent to the site. Finally, a side hill-type TSF is present on site.

Planned infrastructure for the site includes the following:

- Ammonium Nitrate and Fuel Oil (ANFO) Facility;
- Mine Support Facilities (Heavy Vehicle (HV) Workshop, Lube Farm, Washdown and Tire Change, Warehouse, Fuel Farm, Mining Offices, Core Storage Facility);
- Heap Leach Pad (existing);

- Accommodation Camp;
- Water Treatment Plant (WTP);
- Power Supply;
- Pit Dewatering;
- Mine Services;
- Communications;
- Gatehouse; and
- Expanded existing and additional TSF.

1.11 Market Studies and Contracts

1.11.1 Markets

Gold metal markets are mature, with many reputable refiners and brokers located throughout the world. The advantage of gold, like other precious metals, is that virtually all production can be sold in the market. As such, market studies, and entry strategies are not required.

Metallurgical process studies confirm that the Project will produce doré of a specification comparable with existing operating mines.

Demand is presently high with prices showing remarkable increases during recent times. The 36-month average London PM gold price fix through August 31, 2019 was US$1,279/oz.

1.11.2 Contracts

Currently there are no contracts in place for development and operations. However, Vista has obtained budgetary quotes, as is common for PFS level studies, for future materials and service needs. The following contracts are expected to be in place upon project commencement:

- Secure doré transportation to refinery;
- Doré refining;
- Supplier and service contracts including;
  - EPCM;
  - Equipment supply;
  - D&C;
  - Diesel and fuel oil;
  - Natural gas for the power plant;
  - Process reagents;
  - Equipment preventive maintenance and repair (MARC) services;
  - Site security services; and
  - Camp management, catering and support services.

1.12 Social and Environmental Aspects

1.12.1 Existing Environmental and Social Information

A number of environmental studies have been conducted at the Project to obtain environmental and operational permits. Studies conducted have investigated soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology, and water quality.

In January 2018, the “authorization of a controlled activity” was received for the Project as required under the Australian Environmental Protection and Biodiversity Conservation Act of 1999 (EBPC) as it relates to the Gouldian Finch, and as such has received approval from the Australian Commonwealth Department of Environment and Energy.

The Mt Todd Project Environmental Impact Statement (EIS) submitted June 28, 2013 to the Northern Territory Environment Protection Authority (NTEPA), approved in September 2014, provides an understanding of the existing environmental conditions and an assessment of the environmental impact of the Project.

1.12.2 Social or Community Requirements

Vista has a good relationship with the Jawoyn. Areas of aboriginal significance have been designated, and the mine plan has avoided development in these restricted works areas.

1.12.3 Approvals, Permits and Licenses

The Project will require approvals, permits and licenses for various components of the Project. **Table 1-9** includes a list of approvals, permits, and licenses required for the Project and their current status.

Table 1-9: Mt Todd Permit Status

Approval/ Permit/ License	Current Status	Approval/ Permit License Date	Expiration Date
Environmental Impact Statement	The NT Environmental Protection Authority provided its final assessment of the Project in June 2014.	Approved Sep. 2014	NA
Mining Management Act (or Plan) Approval from NT Department of Primary Industry and Resources	Mine operating permit request has been submitted. The MMP submitted in November 2018 is for 50kt/day operations.	Prior to commencing mine operations	NA
Heritage Act permit to destroy or damage archeological sites and scatters/ Aboriginal Areas Protection Authority Clearances	Authority Certificate Number 2011/15538 issued. This certificate defined restricted works areas and granted select clearances to allow for initial investigations. Additional clearances will be required for further investigations as well as prior to disturbance associated with mine development.	Aboriginal Areas Protection Authority dated Jul. 31, 2012	NA
Dangerous Goods Act (1988) permit for blasting activities	Waiting on final mine plan	NA	NA
Extractive Permit (under DME Guidelines) for development of borrow pits outside of approved mining areas	Would be required for PGM or LPM borrow areas. Permit application not yet in progress pending final selection of borrow areas	NA	NA
Waste Discharge License (under Section 74 of the Water Act 1992) for management of water discharge from the site	WDL 178-6 licensing discharge of waste water into the Edith River from the Mt Todd mine site, granted with conditions	Nov. 26, 2018	Nov. 30, 2020
Waste water treatment system permits under Public Health Act 1987 and Regulations	May be required for the waste water treatment system for the construction and operations accommodation village. Permit application not yet in progress pending design and siting of accommodation village.	NA	NA
Approval to Disturb Site of Conservation Significance (SOCS)	Batman pit expansion will disturb SOCS as breeding / foraging habitat for the Gouldian finch, pending determination on EIS.	Jan. 22, 2018	NA

1.13 Capital and Cost Estimates

1.13.1 Capital Cost Estimates – Base Case

LoM capital cost requirements are estimated at US$1,222 million as summarized in **Table 1-10**. Initial capital of US$826 million is required to commence operations. At the end of operations, the Project will receive an estimated US$140 million credit for asset sales and salvage.

Table 1-10: Estimated Capital Cost Summary (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
2000	Mining	7.3%	$121,239	$5,720	**$126,958**	$406,347	$32,677	**$439,024**	$527,586	$38,396	**$565,982**
3000	Process Plant	13.9%	$366,693	$51,073	**$417,766**	$17,027	$2,222	**$19,249**	$383,720	$53,295	**$437,016**
4000	Project Services	10.0%	$109,204	$12,681	**$121,885**	$72,448	$5,455	**$77,903**	$181,651	$18,136	**$199,787**
5000	Project Infrastructure	13.2%	$26,160	$3,463	**$29,623**	$0	$0	**$0**	$26,160	$3,463	**$29,623**
6000	Permanent Accommodation	10.0%	$60	$6	**$66**	$0	$0	**$0**	$60	$6	**$66**
7000	Site Establishment & Early Works	11.4%	$17,537	$1,995	**$19,532**	$0	$0	**$0**	$17,537	$1,995	**$19,532**
8000	Management, Engineering, EPCM Services	11.8%	$82,058	$9,721	**$91,779**	$0	$0	**$0**	$82,058	$9,721	**$91,779**
9000	Pre-Production Costs	12.3%	$16,121	$1,982	**$18,102**	$0	$0	**$0**	$16,121	$1,982	**$18,102**
10000	Asset Sale	0.0%	$0	$0	**$0**	($139,631)	$0	**($139,631)**	($139,631)	$0	**($139,631)**
	Capital Cost	**11.6%**	**$739,072**	**$86,641**	**$825,712**	**$356,191**	**$40,354**	**$396,545**	**$1,095,263**	**$126,994**	**$1,222,257**

1.13.2 Operating Cost Estimates – Base Case

LoM operating costs requirements are estimated to be US$15.18/t-milled as summarized in **Table 1-11**.

Table 1-11: Estimated LoM Operating Costs (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.10	0.03
Mine Maintenance	0.11	0.03
Engineering	0.05	0.01
Geology	0.03	0.01
Drilling	0.77	0.23
Blasting	1.17	0.35
Loading	0.60	0.18
Hauling	2.74	0.83
Mine Support	0.43	0.13
Mine Dewatering	0.01	0.004
Open Pit Mine	**6.02**	**1.82**
CIP PROCESS PLANT		
Labor	0.79	-
3100-Crush/Screen/Stockpile	0.18	-
3200-Reclaim & HPGR	0.44	-
3300-Classification & Grinding	3.14	-
3400-Pre-Leach,Thick/Aeration/CIP	0.13	-
3500-Desorption, Gold Room	0.02	-
3600-Detox & Tailings Pumping	0.06	-
3700-Reagents	2.98	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.01	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**7.88**	**-**
Project Services	$0.16	-
G&A	$1.11	-
Operating Costs	**$15.18**	**-**

1.13.3 Capital Cost Estimates – Alternate Case

LoM capital cost requirements are estimated at US$874 million as summarized in **Table 1-12**. Initial capital of approximately US$623 million is required to commence operations. At the end of operations, the Project will receive an estimated US$86 million credit for asset sales and salvage.

Table 1-12: Estimated Capital Cost Summary, Alternate Case (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
2000	Mining	8.3%	$58,218	$2,824	**$61,042**	$259,218	$23,424	**$282,641**	$317,436	$26,248	**$343,684**
3000	Process Plant	14.6%	$279,747	$40,780	**$320,527**	$7,886	$1,092	**$8,978**	$287,633	$41,872	**$329,505**
4000	Project Services	10.6%	$90,349	$10,859	**$101,208**	$42,256	$3,179	**$45,435**	$132,605	$14,038	**$146,643**
5000	Project Infrastructure	13.2%	$24,635	$3,246	**$27,881**	$0	$0	**$0**	$24,635	$3,246	**$27,881**
6000	Permanent Accommodation	10.0%	$60	$6	**$66**	$0	$0	**$0**	$60	$6	**$66**
7000	Site Establishment & Early Works	11.4%	$16,534	$1,889	**$18,423**	$0	$0	**$0**	$16,534	$1,889	**$18,423**
8000	Management, Engineering, EPCM Services	11.6%	$71,269	$8,279	**$79,549**	$0	$0	**$0**	$71,269	$8,279	**$79,549**
9000	Pre-Production Costs	11.4%	$13,224	$1,512	**$14,736**	$0	$0	**$0**	$13,224	$1,512	**$14,736**
10000	Asset Sale	0.0%	$0	$0	**$0**	($86,279)	$0	**($86,279)**	($86,279)	$0	**($86,279)**
	Capital Cost	**12.5%**	**$554,036**	**$69,396**	**$623,432**	**$223,080**	**$27,695**	**$250,775**	**$777,117**	**$97,091**	**$874,207**

1.13.4 Operating Cost Estimates — Alternate Case

LoM operating cost estimates are summarized in **Table 1-13**. The operating costs will average US$14.99 over the LoM.

Table 1-13: Estimated LoM Operating Costs, Alternate Case (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.12	0.05
Mine Maintenance	0.16	0.07
Engineering	0.07	0.03
Geology	0.05	0.02
Drilling	0.56	0.23
Blasting	0.83	0.35
Loading	0.47	0.20
Hauling	1.73	0.72
Mine Support	0.50	0.21
Mine Dewatering	0.02	0.007
Open Pit Mine	**4.52**	**1.88**
CIP PROCESS PLANT		
Labor	1.19	-
3100-Crush/Screen/Stockpile	0.23	-
3200-Reclaim & HPGR	0.50	-
3300-Classification & Grinding	3.21	-
3400-Pre-Leach,Thick/Aeration/CIP	0.15	-
3500-Desorption, Gold Room	0.03	-
3600-Detox & Tailings Pumping	0.07	-
3700-Reagents	3.00	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.01	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**8.51**	**-**
Project Services	$0.17	-
G&A	$1.79	-
Operating Costs	**$14.99**	**-**

1.14 Financial Analysis

1.14.1 Financial Analysis – Base Case

Estimated economic results are summarized in **Table 1-14**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,350:

- Mine Life: 13 years;
- Pre-Tax NPV5%: US$1,440 million, IRR: 30.4%;
- After-tax NPV5%: US$823 million, IRR: 23.4%;
- Payback (After-tax): 2.9 years;
- NT Royalty Paid: US$473 million;
- Australian Income Taxes Paid: US$553 million; and
- Cash costs (including JAAC Royalty): US$645.14/oz-Au.

Table 1-14: Estimated Technical-Economic Results (US$000s)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Gold Sales			
Gold Produced (koz)	5,305	-	-
Gold Price (US$/oz)	1,350	-	-
Gold Sales	**7,161,494**	**32.40**	**1,300**
Refining & Royalties			
Refinery Costs	(17,075)	(0.077)	(3.22)
JAAC Royalty	(71,615)	(0.324)	(13.50)
Gross Income from Mining	**7,072,805**	**31.998**	**1,333**
Operating Costs			
Open Pit Mine	(1,330,976)	(6.02)	(251)
CIP Process Plant	(1,742,519)	(7.88)	(328)
Project Services	(35,007)	(0.16)	(6.60)
G&A	(246,285)	(1.11)	(46.43)
Operating Costs	**(3,354,787)**	**(15.18)**	**(632.40)**
Power Sales Credit	21,156	0.096	3.99
Cash Cost of Goods Sold (COGS)	**(3,422,321)**	**(15.48)**	**(645.14)**
Operating Margin	**3,739,174**	**16.92**	**704.86**
Capital Costs			
Mining	565,982		
Process Plant	437,016		
Project Services	199,787		
Project Infrastructure	29,623		
Permanent Accommodation	66		
Site Establishment & Early Works	19,532		
Management, Engineering, EPCM Services	91,779		

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Pre-Production Costs	18,102		
Asset Sale	(139,631)		
Capital Costs	**1,222,257**		
Pre-Tax Cash Flow	2,511,917		
$NPV_{5\%}$	1,440,469		
IRR (%)	30.4%		
After-tax Cash Flow	1,439,863		
$NPV_{5\%}$	823,125		
IRR (%)	23.4%		
After-tax Payback (years)	2.9		

1.14.2 Financial Analysis – Alternate Case

Economic results are summarized in **Table 1-15**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,350:

- Mine Life: 11 years;
- Pre-Tax NPV5%: US$884 million, IRR: 25.7%;
- After-tax NPV5%: US$510 million, IRR: 19.8%;
- Payback (After-tax): 3.8 years;
- NT Taxes Paid: US$285 million;
- Australian Income Taxes Paid: US$316 million; and
- Cash costs (including JAAC Royalty): US$603.79/oz-Au.

Table 1-15: Estimated Economic Results, Alternate Case (US$000s)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Gold Sales			
Gold Produced (koz)	3,232	-	-
Gold Price (US$/oz)	1,350	-	-
Gold Sales	**4,363,271**	**34.08**	**1,350**
Refining & Royalties			
Refinery Costs	(10,641)	(0.083)	(3.292)
JAAC Royalty	(43,633)	(0.341)	(13.50)
Gross Income from Mining	**4,308,997**	**(33.661)**	**1,333**
Operating Costs			
Open Pit Mine	(578,421)	(4.52)	(179)
CIP Process Plant	(1,089,355)	(8.51)	(337)
Project Services	(21,777)	(0.17)	(7)
G&A	(228,808)	(1.79)	(71)
Operating Costs	**(1,918,361)**	**(14.99)**	**(594)**

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Power Sales Credit	21,156	0.17	**7**
Cash Cost of Goods Sold (COGS)	**(1,951,479)**	**(15.24)**	**(604)**
Operating Margin	**2,411,792**	**18.84**	**746**
Capital Costs			
Mining	343,684		
Process Plant	329,505		
Project Services	146,643		
Project Infrastructure	27,881		
Permanent Accommodation	66		
Site Establishment & Early Works	18,423		
Management, Engineering, EPCM Services	79,549		
Pre-Production Costs	14,736		
Asset Sale	(86,279)		
Capital Costs	**874,207**		
Pre-Tax Cash Flow	1,532,585		
$NPV_{5\%}$	884,337		
IRR (%)	25.7%		
After-tax Cash Flow	931,075		
$NPV_{5\%}$	509,611		
IRR (%)	19.8%		
Post- Tax Payback (years)	3.8		

1.15 Conclusions and Recommendations

1.15.1 Feasibility Study

A Feasibility Study (FS) should be completed to advance the Project and provide additional detailed information necessary to support capital and operating cost estimates for a potential project development decision.

The estimated budget for the FS is approximately US$2.5M – 4.0M.

1.15.2 Geology and Resources

1.15.2.1 Conclusions

- The Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline which is comprised of the Burrell Creek Formation, the Tollis Formation, and the Kombolgie Formation.

- Gold mineralization in this area is constrained to a single mineralization event and the deposits are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit has characteristics of an intrusion related gold system making it the primary resource.
- The Batman deposit is defined by approximately 7.4 million ounces (Moz) of gold within 278 Mt of measured and indicated resource at an average grade of 0.82 g-Au/t and a cutoff grade of 0.4 g-Au/t. as provided in **Table 1-3**.

1.15.2.2 Recommendations

- The Batman deposit potentially extends along strike both to the north and south. Step out drilling should be evaluated.
- Additional deep infill drilling should be used to help define potential deep mineralization not detected by early historical drillholes.
- Infill drilling within and exploration drillholes along the trend of the Quigleys deposit is recommended.
- Exploration of the exploration licenses, including work on geophysical and geochemical anomalies, should continue in a systematic manner.

The estimated budget for drilling within the MLs is US$500,000-1,000,000 and US$500,000-1,000,000 for initial drilling on the ELs.

1.15.3 Mineral Reserve and Mine Planning

- Pit designs were completed based on Whittle™ pit optimizations and are appropriate for metal prices of approximately US$800 per ounce Au for the Alternate Case and US$1,000 per ounce Au for the Base Case. The Mt Todd proven and probable reserve estimates have been defined using economics based on a gold price of US$1,250 per ounce and an elevated cutoff grade of 0.40 g-Au/t. The proven and probable reserve estimates were used to create a production schedule for mining, and a positive cash-flow analysis has been done based on the production schedule by Tetra Tech. The reserve estimates have reasonable economics with respect to the statement of reserves under NI 43-101 regulations.
- Mine production constraints were imposed to ensure that mining wasn't overly aggressive with respect to the equipment anticipated for use at Mt Todd. The schedule has been produced using mill targets and stockpiling strategies to enhance the project economics. The constraints and limits are reasonable to support the project economics which are used to justify the statement of reserve estimates.
- Pit designs use six-meter benches for mining. This corresponds to the resource model block heights, and MDA believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining. In areas where the material is consistently ore or waste so that dilution is not an issue, benches may be mined in 12 m heights.

1.15.4 Mineral Processing

1.15.4.1 Conclusions

The substantial quantity and quality of metallurgical test work data developed from Mt Todd drill core samples has led to the development of a robust, energy-efficient comminution circuit followed by a standard gold recovery process. Key conclusions drawn from the metallurgy studies are:

- Mt Todd (Batman) ore is among the hardest and most competent ore types processed for mineral recovery. The most energy efficient comminution circuit has been determined to be the sequence of primary crushing, closed circuit secondary crushing, and closed circuit HPGR tertiary crushing and ore sorting followed by two stages of grinding.
- The ore is free-milling, is not preg-robbing, and is amenable to gold extraction by conventional cyanidation processes.
- The ore has moderately high cyanide consumption, determined to be 0.876 kg of sodium cyanide per tonne of ore. This is largely due to the presence of sulfides and cyanide consuming copper and destruction of residual cyanide.
- The use of sorting has helped to decrease operational costs by removing portions of the uneconomic material mined.
- Achieving a finer grind with a two-stage grinding circuit resulted in significant improvement in gold extraction.

The equipment selection criteria for the Base Case operation has received considerable interaction with specialist vendors to the point where there is a reasonably high-degree of confidence in selected technology and process units at this preliminary feasibility study stage. The recommended flowsheet for a FS consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using HPGRs and ore sorting, two-stage grinding, cyclone classification, secondary grinding, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detox and disposal to conventional tailings storage facility.

1.15.4.2 Recommendations

The on-going testwork is directed towards optimization of the process parameters. The study would potentially lead to reduction of reagent usage thereby reducing operating costs. In addition, the potential of increasing leach pulp density is being evaluated which could result in reducing size of leach tanks and hence capital cost.

1.15.5 Infrastructure

- Bulk earthworks are designed to minimize the import of fill materials.
- Administration offices, gatehouse/security facilities, cribs/ablutions are planned to be transportable buildings.
- The process plant offices, workshop and warehouse are located inside the existing Flotation Building.
- Sample preparation and laboratory will have a purpose-built steel shed.
- The access road is based on the repaired existing road.

- Heavy cranage is allowed for all lifts greater than 50 t.
- All bulk transport will be weighed.
- Site-wide communication is based on a 50 m tall communication tower that will support eight channels.

1.15.6 Environmental and Social Impacts

1.15.6.1 Conclusions

A number of environmental studies have been conducted at the Project Site in support of the Environmental Impact Statement (EIS) and as required for environmental and operational permits. Studies conducted have investigated soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology, and water quality.

The draft EIS for the Project was submitted in June 2013. The document was prepared by independent consultants GHD Pty Ltd to identify potential environmental, social, transport, cultural and economic impacts associated with reopening and operating the mine. The Northern Territory Environmental Protection Authority (NTEPA) provided its final assessment of the Project in June 2014. Final approval was given in September 2014.

In January 2018, the “authorization of a controlled activity” was received for the Project as required under the Australian Environmental Protection and Biodiversity Conservation Act of 1999 (EBPC) as it relates to the Gouldian Finch, and as such has received approval from the Australian Commonwealth Department of Environment and Energy.

Areas of aboriginal significance have been designated, and the mine plan has avoided development in these restricted works areas.

1.15.6.2 Recommendations

Additional studies will be needed to further assess environmental baseline conditions to support feasibility level design, permitting, and closure planning for the Project, including:

- Erosion analyses;
- Waste and cover material hydraulic properties characterization and analysis;
- Ongoing aquatic, benthic, and wildlife studies;
- Comprehensive vegetation survey;
- Archaeological and historical assessments for all areas to be disturbed;
- Further hydrogeologic investigations and site-wide hydrogeologic characterization; and
- Continued precipitation, stream flow, and watershed data.

The estimated budget for this work is US$350,000.

1.15.7 Results of the Site-wide Water Balance Model

1.15.7.1 Conclusions

- The WTP rate of 500 m^3/hr and process water pond (PWP) sizing for 6 days of storage was determined to be appropriate for the 50,000 tpd production process water requirements.
- The greatest amount of make-up water required from the raw water dam (RWD) was quantified as 11,955 m^3/day. RWD requirements were found to be the most dependent upon TSF decant volumes.
- The Waste Rock Dump (WRD) retention pond (RP1), low grade ore stockpile retention pond (LGRP), process plant retention pond (PRP), and heap leach pad (HLP) were typically observed to overtop less than 1% of the time during the 13-year simulation.[1] LGRP storage may be optimized.

1.15.7.2 Recommendations

Recommended model improvements include:

- The site-wide water balance model is dependent upon the TSF water balance model, which provides decant water to the process facility, and the vadose and seepage models, which characterize seepage through the various rock piles on site (WRD, low grade ore stockpile (LGOS), and HLP). As such, completion of these models to the greatest detail practicable affects the overall quality of the site-wide water balance model results.
- Incorporate results of future Batman Pit potential groundwater inflow investigation.
- Optimize management of Batman Pit dewatering effluent and other contact water. This water may be of sufficient quality to be used as make-up water to the process circuit offsetting the water coming from the RWD. This water would also reduce the amount of water ultimately requiring treatment.
- In this iteration of the site-wide water balance model, the entirety of the WTP effluent is being used as dust suppression around the mine site during the dry season. Further investigation of other uses of the WTP effluent should be conducted.
- Further investigation of the adequacy of RP1 storage capacity is recommended, particularly within the early stages of the LoM when a larger fraction of the catchment reports to this pond.
- Incorporate RWD stage-storage relationship and catchment area into the site-wide water balance model such that it may be modeled as a reservoir, as opposed to an infinite source.
- Inclusion of process, fire, potable and raw water tanks. At present, only the dust suppression tank is modeled. The tanks above are currently modeled as drawing water directly from the RWD, rather than demands on discrete tanks.
- Review and update dust suppression requirements.
- Incorporate evolving Batman Pit shell geometry to more accurately model that facility.

The estimated budget for this work is US$200,000.

[1] A typical value is given. Separate model runs provide a range of overtopping events, due to the stochastic nature of the model.

1.15.8 Groundwater Hydrology and Mine Dewatering

The following work is recommended with respect to groundwater hydrology and mine dewatering:

- Additional hydrogeologic study should be completed in the vicinity of the Batman Pit to provide more detailed information on which to base calibration of the regional groundwater flow model and subsequent prediction of groundwater inflows to the pit and post-mining recovery of the groundwater system. The study should also include measurement of depth to water in any accessible existing borings or core holes within or immediately adjacent to the pit.
- Calibration of the regional groundwater flow model should be completed with the additional data, and the calibrated model should be used to refine the estimates of groundwater inflow to the pit and predictions of the hydrogeologic effects of pit dewatering.
- The post-mining version of the groundwater flow model should be updated with the calibrated model used as its basis. Output from the post-mining model should be incorporated into any geochemical modeling of post-mining pit lake formation and geochemistry.

The estimated budget for this work is US$400,000.

1.15.9 Process Plant Geotechnical Investigation

1.15.9.1 Conclusions

There are no conclusions with regard to the Process Plant geotechnical investigation.

1.15.9.2 Recommendations

Future geotechnical work is recommended during final engineering design, particularly for foundation design of the processing facilities.

The estimated budget for this work is US$150,000.

1.15.10 TSF Design

As part of advancing the TSF design in the FS, work should include optimization of the TSF construction schedule, geotechnical investigation and assessment of TSF 1 and TSF 2, TSF water balance update, and TSF consequence classification. The estimated budget for this work is included in the FS budget estimate.

1.15.11 Process

Two major items incurring operating costs are grinding media and reagents. Together these items make up 61% of the plant consumables operating costs. The FS should investigate options for reducing the consumption rate and the unit costs for these consumables.

The estimated budget for this work is included in the FS budget estimate.

1.15.12 Geochemical Analyses

Geochemical characterization will be updated to reflect the designations of Potentially Acid Forming, Potentially Acid Forming-Low Capacity, Non Acid Forming, Acid Consuming and Uncertain in accordance with DITR (2007) guidelines. Additionally, Tetra Tech recommends performing geochemical testing on the sorter reject material.

The estimated budget for this work is included in the Feasibility Study.

2.0 INTRODUCTION

Vista Gold Corp. and its subsidiaries (collectively, "Vista," the "Company," "we," "our," or "us") operate in the gold mining industry. We are focused on the evaluation, acquisition, exploration and advancement of gold exploration and potential development projects, which may lead to gold production or value adding strategic transactions such as earn-in right agreements, option agreements, leases to third parties, joint venture arrangements with other mining companies, or outright sales of assets for cash and/or other consideration. We look for opportunities to improve the value of our gold projects through exploration drilling and/or technical studies focused on optimizing previous engineering work. We do not currently generate cash flows from mining operations.

The Company's flagship asset is its 100% owned Mt Todd gold Project (Mt Todd) in the Northern Territory (NT) Australia. Mt Todd is the largest undeveloped gold project in Australia. The Company recently received authorization for the last major environmental permit and completed an updated Preliminary Feasibility Study (PFS) for Mt Todd, which confirms the project's robust economics at today's gold price. With these important milestones complete, Vista is in a position to actively pursue those strategic alternatives that provide the best opportunity to maximize value for the Company.

Vista Gold Corp. was originally incorporated on November 28, 1983 under the name "Granges Exploration Ltd." It amalgamated with Pecos Resources Ltd. during June 1985 and continued as Granges Exploration Ltd. In June 1989, Granges Exploration Ltd. changed its name to Granges Inc. Granges Inc. amalgamated with Hycroft Resources & Development Corporation during May 1995 and continued as Granges Inc. Effective November 1996, Da Capo Resources Ltd. and Granges, Inc. amalgamated under the name "Vista Gold Corp." and, effective December 1997, Vista continued from British Columbia to the Yukon Territory, Canada under the *Business Corporations Act* (Yukon Territory). On June 11, 2013, Vista continued from the Yukon Territory, Canada to the Province of British Columbia, Canada under the *Business Corporations Act* (British Columbia).

2.1 Background Information

Vista Gold Corp. (Vista) retained Tetra Tech, Inc., along with JDS Energy & Mining, Inc. (JDS), Mine Development Associates (MDA), Resource Development Inc. (RDi), Tetra Tech Proteus (TTP), and POWER Engineers, Inc. (POWER) to prepare this preliminary feasibility study (PFS) for its Mt Todd Gold Project (the Project) in Northern Territory (NT), Australia. The PFS (Technical Report) evaluates the Base Case, a development scenario of a 50,000 tonne per day (tpd) processing facility. In addition, an Alternate Case was considered at 33,000 tpd with higher grades presented under **Section 24.0 – Other Relevant Data and Information**.

Key differences between the Base Case and the Alternate Case include:

- A 33,000 tpd processing facility as compared to a 50,000 tpd facility with associated lower mining rates and a smaller mining fleet;
- Pit design is based on a pit shell calculated using a US$1,000/oz-Au and a US$800/oz-Au for the Base and Alternate Cases, respectively. The same cut-off grade of 0.40 g-Au/t was used; and
- Shorter operating life for the Alternate Case.

The Base Case includes:

- Estimated proven and probable reserves of 5.848 Moz of gold (221 Mt at 0.82 g-Au/t) at a cut-off grade of 0.40 g-Au/t;
- Average annual production of 381,211 ounces of gold per year over the mine life, including average annual production of 479,450 ounces of gold per year during the first five years of operations;
- LoM average cash costs of US$645 per ounce, including average cash costs of US$575 per ounce during the first five years of operations;
- A 13-year operating life;
- After-tax NPV5% of US$823 million and internal rate of return (IRR) of 23.4% at US$1,350 per ounce gold prices and US$0.70:AUD1.00 exchange rate, and
- Initial capital requirements of US$826 million.

The Alternate Case discussed in **Section 24.0 – Other Relevant Data and Information** includes:

- Estimated proven and probable reserves of 3.557 Moz of gold (121 Mt at 0.86 g-Au/t) at a cut-off grade of 0.40 g-Au/t;
- Average annual production of 273,000 ounces of gold per year over the mine life, including average annual production of 301,778 ounces of gold per year during the first five years of operations;
- LoM average cash costs of US$593 per ounce, including average cash costs of US$581 per ounce during the first five years of operations;
- An 11-year operating life;
- After-tax NPV5% of US$418 million and IRR of 17.8% at US$1,300 per ounce gold prices and US$0.70:AUD1.00 exchange rate; and
- Initial capital requirements of US$641 million.

2.2 Terms of Reference and Purpose of the Report

This Technical Report was prepared as a NI 43-101 Technical Report for Vista by Tetra Tech. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in Tetra Tech's services, 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.

This report provides mineral resource and mineral reserve estimates, and a classification of resources and reserves in accordance with the CIM Standards. The CIM Standards requires the completion of a PFS as the minimum prerequisite for the conversion of mineral resources to mineral reserves.

A preliminary feasibility study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method and the open pit configuration is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the modifying factors and the evaluation of any other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource may be converted to a mineral reserve at the time of reporting. 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.

A PFS is at a lower confidence level than a Feasibility Study (FS).

2.3 Sources of Information

The primary technical documents and files relating to the Project that were used in the preparation of this report are listed in **Section 27.0 – References**.

2.4 Units of Measure

The metric system has been used throughout this report. Tonnes are metric of 1,000 kilograms (kg), or 2,204.6 pounds (lb). Gold is reported in troy ounces (oz), equivalent to 31.1035 grams (g). Currency is in Q3 2019 U.S. dollars (US$) unless otherwise stated.

2.5 Detailed Personal Inspections

1) Rex Bryan last visited and inspected the property from June 28th through June 29th, 2017. Dr. Bryan spent time on site and reviewed the current database and archived supporting material, core logging, sampling procedures, handling and security measures, QA/QC procedures and inspected modern and historically collected core.
2) Anthony Clark last visited and inspected the property from June 28th through June 29th, 2017. Mr. Clark inspected the existing power infrastructure at the site, natural gas pipeline, and power rights-of-way.
3) Thomas Dyer last visited and inspected the property from June 28th through June 29th, 2017. Mr. Dyer toured the site along with geotechnical consultants and reviewed the pit, waste dump, tailings facility, and resource drilling sites. Previous mine production records held on site were also reviewed.
4) Chris Johns visited and inspected the property from June 28th through June 29th, 2017. Mr. Johns inspected the existing Tailings Storage Facility 1 (TSF 1) and the proposed site for Tailings Storage Facility 2 (TSF 2).
5) Zvonimir Ponos last visited and inspected the property from June 28th through June 29th, 2017. Mr. Ponos inspected the existing site infrastructure and process facility.
6) Vicki Scharnhorst visited and inspected the property from June 28th through June 29th, 2017. Ms. Scharnhorst inspected the infrastructure at site and reviewed the status of environmental permitting with site staff.

QPs not listed above have not visited or inspected the property. Personal inspections by these QPs are not required to complete their responsibilities.

3.0 RELIANCE ON OTHER EXPERTS

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report. 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 QPs do not consider them to be material.

The QPs relied upon the following experts:

- Environmental Impact Statement for the Project prepared by GHD (June 2013) to describe environmental matters (Tetra Tech, **Section 20.0 – Environmental Studies, Permitting, and Social or Community Impact**).

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project is located 56 kilometers (km) by road northwest of Katherine, and approximately 290 km southeast of Darwin in NT, Australia (**Figure 4-1**). Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory.

4.2 Property Description

Vista Australia is the holder of four mineral licenses (ML) MLN 1070, MLN 1071, MLN 1127, and MLN 31525 comprising approximately 5,544 hectares (ha). In addition, Vista Australia controls exploration licenses (EL) EL 29882, EL 29886, EL 30898, and EL 28321 comprising approximately 153,700 ha. **Figure 4-2** illustrates the general location of the tenements and the position of the Batman deposit. A general arrangement is provided in **Figure 4-3**.

4.3 Lease and Royalty Structure

Vista Australia entered into a lease agreement (the Lease Agreement) with the NT government for an initial term of five years commencing January 1, 2006, with an extension of five years at Vista Australia's option and three additional years upon the application of Vista Australia and with the approval of the NT government. Pursuant to the conditions of the first five-year term of the Lease Agreement, Vista Australia undertook a comprehensive technical and environmental review of the Project to evaluate site environmental conditions and developed a program to stabilize the environmental conditions and minimize offsite contamination. Vista also reviewed the water management plan and made recommendations and developed a Technical Report for the re-starting of operations. During the term of the Lease Agreement, Vista Australia was also required to examine all technical, economic, and environmental issues, estimate the cost to rehabilitate the site, explore and evaluate the potential of the Project, and prepare a technical and economic feasibility study for the potential development of the Project site.

Vista provided notice to the NT government in June 2010 that it wished to extend the Lease Agreement. In November 2010, the NT government granted the renewal and the Lease Agreement was extended for an additional five years to December 31, 2015. The NT government renewed the Lease Agreement by deed of variation in 2014 and again in May 2017, extending it to December 31, 2023.

Vista Australia paid the NT government's costs of management and operation of the Project Site up to a maximum of AUD375,000 during the first year of the term, and assumed site management and management and operation costs in the following years. In the agreement, the NT government acknowledges its commitment to rehabilitate the site and the Lease Agreement provides that Vista Australia has no rehabilitation obligations for pre-existing environmental conditions until it submits and receives approval of a Mining Management Plan (MMP) for the resumption of mining operations. The most recent MMP (Vista Gold Australia 2018) addresses activities undertaken by Vista with respect to site management, infrastructure maintenance and environmental management (**Section 20.3 – Permitting and Authorizations**).

Recognizing the importance placed by the NT government upon local industry participation, Vista Australia has agreed to use, where appropriate, NT-sourced labor and services during the period of the Lease

Agreement in connection with the Mt Todd property, and further, in connection with any proposed mining activities prepare and execute a local Industry Participation Plan.

Pursuant to an agreement (the JAAC Agreement) with the Jawoyn Association Aboriginal Corporation (JAAC), Vista was required to issue Vista common shares with a value of Canadian dollars (CAD) 1.0 million. as consideration for the JAAC entering into the JAAC Agreement and as rent for the use of the surface lands overlying the mineral leases during the period from the effective date of the agreement until a decision is reached to begin production. For rent of the surface rights from the current mining licenses, including the mining license on which the Batman deposit is located, the JAAC is entitled to an annual amount equal to 1% of the gross value of production with a minimum annual payment of AUD50,000. Vista also pays the JAAC AUD5,000 per month for consulting with respect to aboriginal, cultural, and heritage issues. If the Project proves feasible and subject to several conditions, Vista has agreed to offer the JAAC the opportunity to establish a joint venture company with Vista holding 90% and the JAAC holding a 10% participating interest, with each party being responsible to finance and provide funding for its respective develop costs of the Project.

There is also a royalty of 5% of based on the gross value of any gold or other metals that may be commercially extracted from certain mineral concessions (the Denehurst Royalty). The Denehurst Royalty would not apply to any presently identified mineralized zones at Mt Todd.

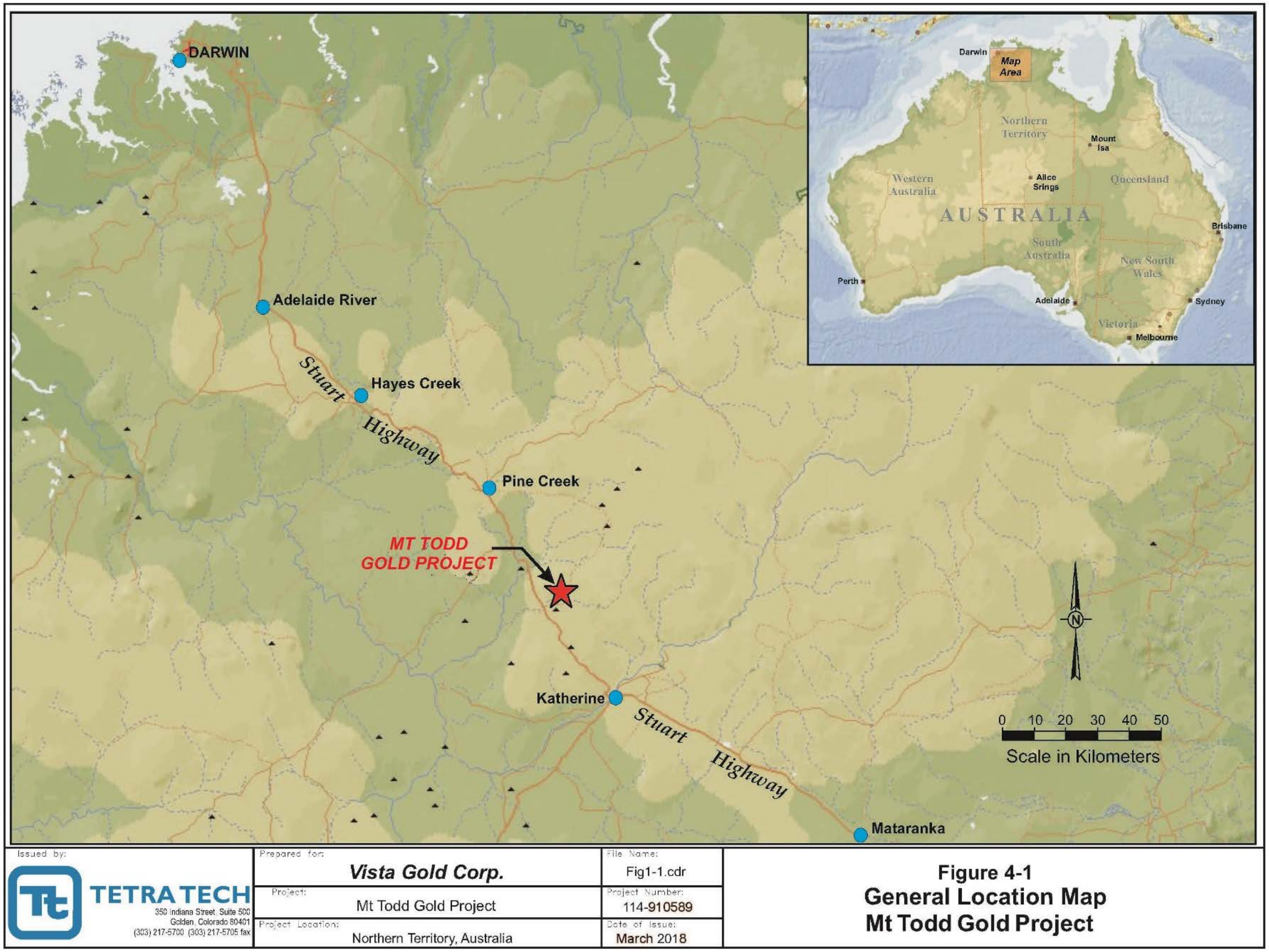
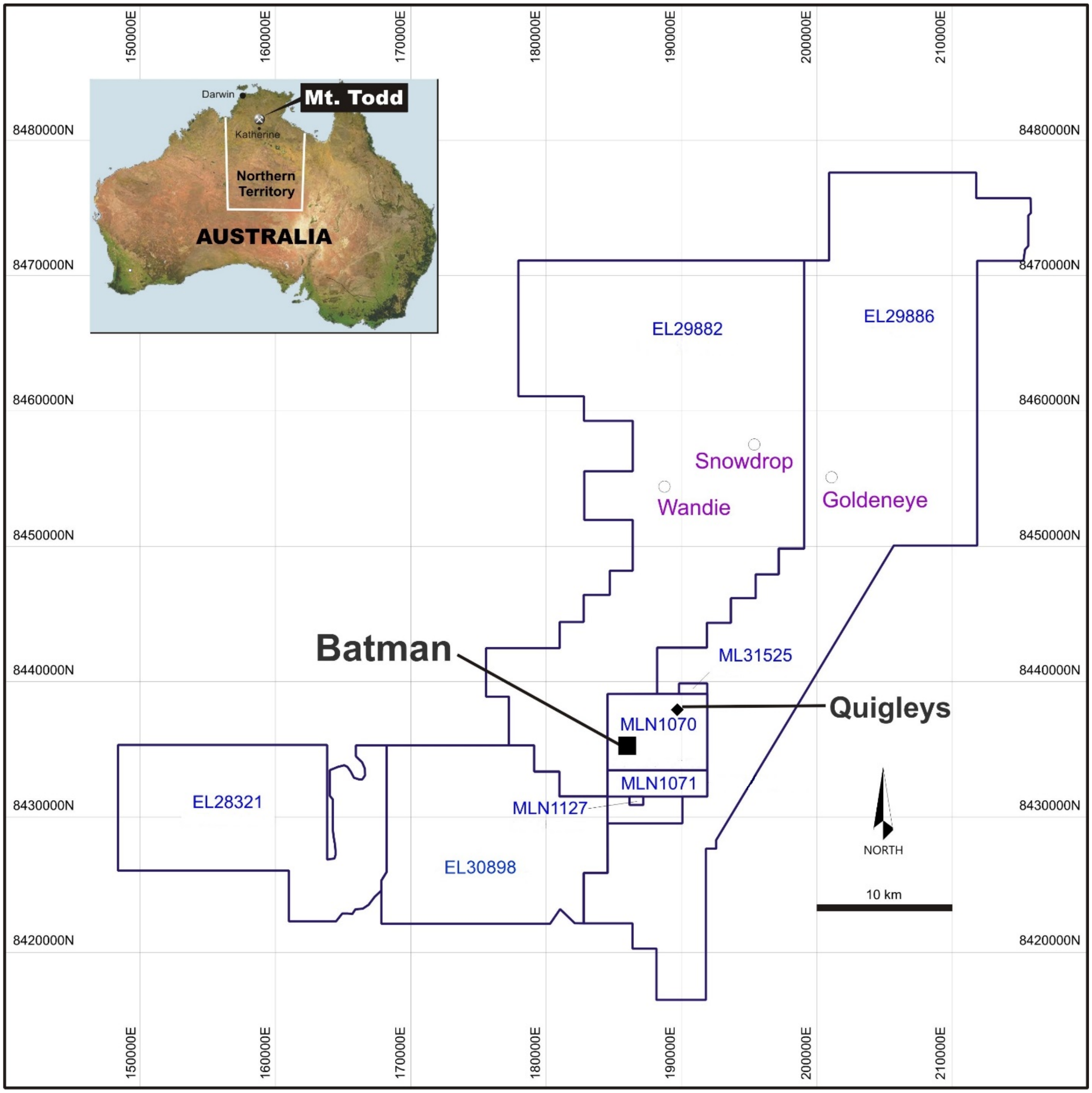


Figure 4-1: General Project Location Map



NOTE: Prepared by Vista Gold Corp.; updated on February 23, 2018

Figure 4-2: Concessions

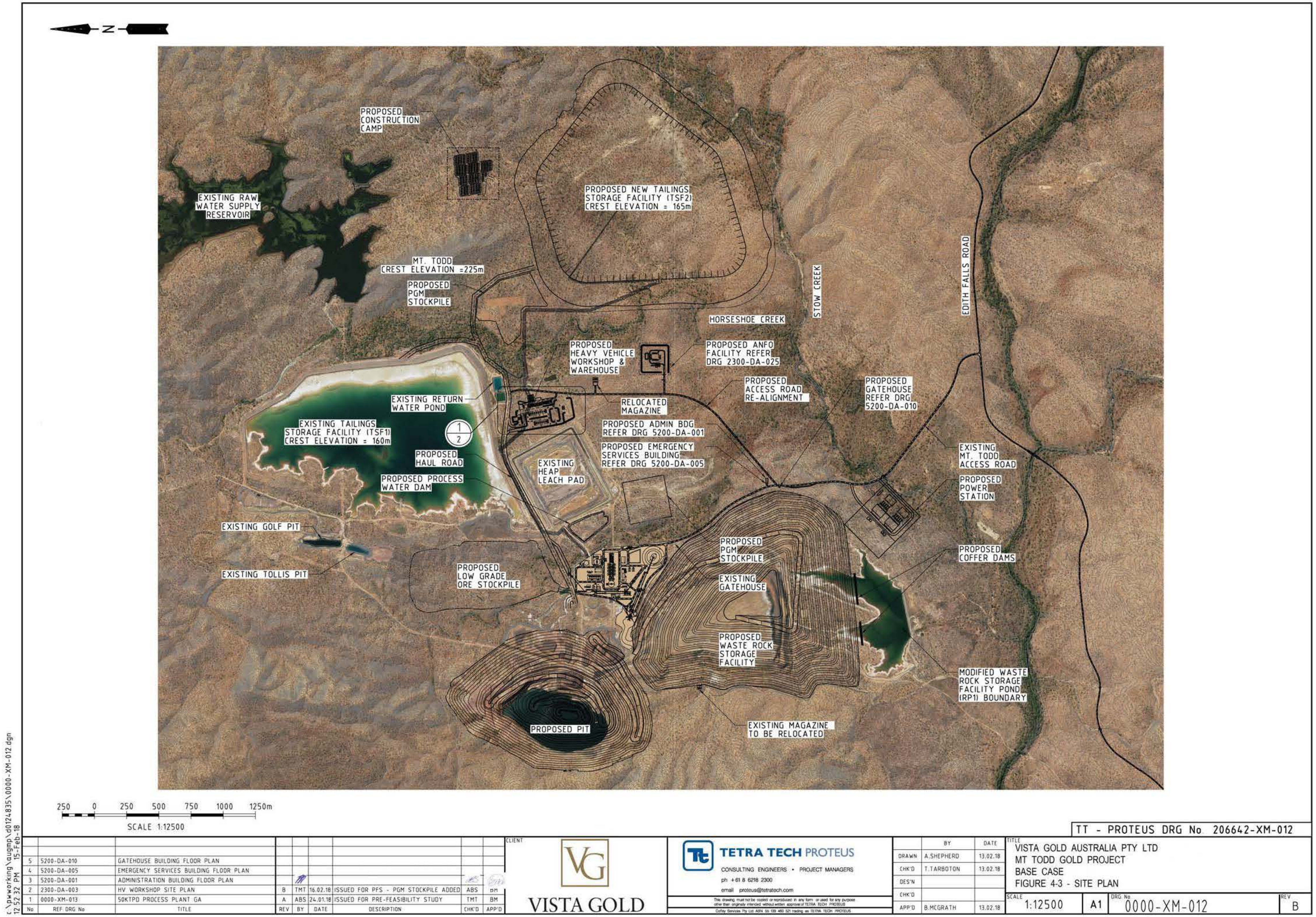


Figure 4-3: General Arrangement

4.4 Risks

Vista is in sole possession of the title and rights to perform work on the Project. Surface access is guaranteed through Vista's agreement with the JAAC. Exploration or other similar activities require a MMP to be submitted to the Department of Regional Development, Primary Industry, Fisheries and Resources (DRDPIFR) with approvals typically occurring in thirty or less days. Vista has been in sole possession of the site for approximately 13 years and no MMPs have ever been withheld with regard to exploration or other activities.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located 56 km by road northwest of Katherine, and approximately 290 km southeast of Darwin in the NT of Australia. Access to the mine is via high quality, two-lane paved roads from the Stuart Highway, the main artery within the territory.

5.2 Climate and Physiography

The Project area has a sub-tropical climate with a distinct wet season and dry season. The area receives most of its rainfall between the months of January and early March. During these months, the temperature usually ranges from 25° to 35°C, but temperatures can reach as high as 42°C. Winter temperatures in the dry season usually range from 14°C to 20°C, but can drop to as low as 10°C at night.

Mining and processing operations are planned year-round; however, pit dewatering will be required after large precipitation events.

5.3 Local Resources and Infrastructure

Access to local resources and infrastructure is excellent. The Project is located sufficiently close to the city of Katherine to allow for an easy commute for workers. The area has both historic and current mining activity and therefore a portion of the skilled workforce will be sourced locally. In addition, Katherine offers the necessary support functions that are found in a medium-sized city with regard to supplies, accommodations, communications, etc.

The property has an existing high-pressure gas line and an electric power line that was used by previous operators. In addition, wells for potable water and a dam for process water are also located on or adjacent to the site. Finally, a fully functioning tailings dam is present on site.

The concessions are within 2 to 3 km of the Nitmiluk Aboriginal National Park on the east. This National Park contains a number of culturally and geologically significant attractions. The proximity to the National Park has not historically yielded any impediments to operating. It is not expected to yield any issues to renewed operation of the property in the future. The Project is wholly contained within the Aboriginal Freehold Land and will require no additional acquisition of surface rights.

5.4 Topography, Elevation and Vegetation

The topography of the Project is relatively flat. The mineral leases encompass a variety of habitats forming part of the northern Savannah woodland region, which is characterized by eucalypt woodland with tropical grass understories. Surface elevations are on the order of 130 to 160 meters (m) above sea level in the area of the previous and planned site and waste dumps.

6.0 HISTORY

The Project area has significant gold deposits. It is situated in a well-mineralized historical mining district that supported small gold and tin operations in the past.

The Shell Company of Australia (Billiton), who was the managing partner in an exploration program in joint venture with Zapopan NL (Zapopan), discovered the Mt Todd mineralization, or more specifically the Batman deposit, in May 1988. Zapopan acquired Billiton's interest in 1992 by way of placement of shares to Pegasus Gold Australia Pty. Ltd. (Pegasus). Pegasus progressively increased their shareholding until they acquired full ownership of Zapopan in July 1995.

Feasibility studies (not NI 43-101 compliant) for Phase I, a heap leach operation which focused predominately on the oxide portion of the deposit, commenced during 1992 culminating in an engineering, procurement, construction management (EPCM) award to Minproc in November of that year. The Phase I project was predicated upon a 4 million tonnes per year (Mtpy) on an annualized basis heap leach pad, which came on stream in late 1993. The treatment rate was subsequently expanded to a rate of 6 Mtpy on an annualized basis in late 1994.

Historic production is shown in **Table 6-1**.

Table 6-1: Heap Leach – Historic Actual Production

Category	Historic Production Actual
Tonnes Leached (million)	13.2
Head Grade (g-Au/t)	0.96
Recovery (%)	53.8
Gold Recovered (oz)	220,755
Cost/t (AUD)	8.33
Cost/oz (AUD)	500

NOTE: All tonnages and grades are historic production numbers that pre-date Vista's ownership. The QPs and issuer consider historic estimates to be relevant but not current.

Phase II involved expanding to 8 Mtpy and treatment through a flotation and carbon-in-leach (CIL) circuit. The feasibility study was conducted by a joint venture between Bateman Kinhill and Kilborne (BKK, 1996) and was completed in June 1995.

The Pegasus board approved the project on August 17, 1995, and awarded an EPCM contract to BKK in October 1995. Commissioning commenced in November 1996. Final capital cost to complete the project were AUD232 million (US$181 million).

Design capacity was never achieved due to inadequacies in the crushing circuit. An annualized throughput rate of just under 7 Mtpy was achieved by mid-1997; however, problems with the flotation circuit which resulted in reduced recoveries necessitated closure of this circuit. Subsequently, high reagent consumption as a result of cyanide soluble copper minerals further hindered efforts to reach design production. Operating costs were above those predicted in the feasibility study.

The spot price of gold deteriorated from above US$400 in early 1996 to below US$300 per ounce during 1997. According to the 1997 Pegasus Annual Report, the economics of the project were seriously affected by the slump. Underperformance of the project and higher operating costs led to the mine being closed and placed on care and maintenance on November 14, 1997.

In February 1999, General Gold Resources Pty. Ltd. (General Gold) agreed to form a joint venture with Multiplex Resources Pty Ltd (Multiplex Resources) and Pegasus to own, operate, and explore the mine. Initial equity participation in the joint venture was General Gold 2%, Multiplex Resources 93%, and Pegasus 5%. The joint venture appointed General Gold as mine operator, which contributed the operating plan in exchange for a 50% share of the net cash flow generated by the project, after allowing for acquisition costs and environmental sinking fund contributions. General Gold operated the mine from March 1999 to July 2000.

6.1 History of Previous Exploration

The Batman gold prospect is part of a goldfield that was worked from early in the 20th century. Gold and tin were discovered in the Mt Todd area in 1889. Most deposits were worked in the period from 1902 to 1914. A total of 7.80 tonnes of tin concentrate was obtained from cassiterite-bearing quartz-kaolin lodes at the Morris and Shamrock mines. The Jones Brothers reef was the most extensively mined gold-bearing quartz vein, with a recorded production of 28.45 kg Au. This reef consists of a steeply dipping ferruginous quartz lode within tightly folded greywackes.

The Yinberrie Wolfram field, discovered in 1913, is located 5 km west of Mt Todd. Tungsten, molybdenum and bismuth mineralization was discovered in greisenized aplite dykes and quartz veins in a small stock of the Cullen Batholith. Recorded production from numerous shallow shafts is 163 tonnes of tungsten, 130 kg of molybdenite and a small quantity of bismuth.

Exploration for uranium began in the 1950s. Small uranium prospects were discovered in sheared or greisenized portions of the Cullen Batholith in the vicinity of the Edith River. The area has been explored previously by Esso for uranium without any economic success.

Australian Ores and Minerals Limited (AOM) in joint venture with Wandaroo Mining Corporation and Esso Standard Oil took out a number of mining leases in the Mt Todd area during 1975. Initial exploration consisted of stream sediment sampling, rock chip sampling, and geological reconnaissance for a variety of commodities. A number of geochemical anomalies were found primarily in the vicinity of old workings.

Follow-up work concentrated on alluvial tin and, later, auriferous reefs. Backhoe trenching, costeaning, and ground follow-up were the favored mode of exploration. Two diamond drillholes were drilled at Quigleys. Despite determining that the gold potential of the reefs in the area was promising, AOM ceased work around Mt Todd. The Arafura Mining Corporation, CRA Exploration, and Marriaz Pty Ltd all explored the Mt Todd area at different times between 1975 and 1983. In late 1981, CRA Exploration conducted grid surveys, geological mapping and a 14-diamond drillhole program, with an aggregate meterage of 676.5 m, to test the gold content of Quigleys Reef over a strike length of 800 m. Following this program CRA Exploration did not proceed with further exploration.

During late 1986, Pacific Gold Mines NL (Pacific Gold Mines) undertook exploration in the area which resulted in small-scale open cut mining on the Quigleys and Golf reefs, and limited test mining at the Alpha, Bravo, Charlie and Delta pits. Ore was carted to a carbon-in-pulp (CIP) plant owned by Pacific Gold Mines at Moline. This continued until December 1987. Pacific Gold Mines ceased operations in the area in February 1988 having produced approximately 86,000 tonnes grading 4 g-Au/t (historic reported production, not NI 43-101 compliant). Subsequent negotiations between the Mt Todd Joint Venture partners (Billiton and Zapopan) and Pacific Gold Mines resulted in the acquisition of this ground and incorporation into the joint venture.

Table 6-2 presents important historical events in a chronologic order.

Table 6-2: Property History

1986	
October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening (Higgins); Ground Acquisition, Zapopan N.L.
1987	
February: *June-July:* *October:*	Joint Venture finalized between Zapopan and Billiton. Geological Reconnaissance, Regional BCL, stream sediment sampling. Follow-up BCL stream sediment sampling, rock chip sampling and geological mapping (Geonorth)
1988	
Feb-March: *March-April:* *May:* *May-June:* *July:* *July-Dec:*	Data reassessment (Truelove) Gridding, BCL grid soil sampling, grid based rock chip sampling and geological mapping (Truelove) Percussion drilling Batman (Truelove) - (BP1-17, 1475m percussion) Follow-up BCL soil and rock chip sampling (Ruxton, Mackay) Percussion drilling Robin (Truelove, Mackay) – RP 1-14, (1584m percussion) Batman diamond, percussion and reverse circulation (RC) drilling (Kenny, Wegmann, Fuccenecco) - BP18-70, (6263m percussion); BD1-71, (8562m Diamond); BP71-100, (3065m R.C.)
1989	
Feb-June: *June:* *July-Dec:*	Batman diamond and RC drilling: BD72-85 (5060m diamond); BP101-208, (8072m RC). Penguin, Regatta, Golf, Tollis Reef Exploration Drilling: PP1-8, PD1, RGP1-32 GP1-8, BP108, TP1-7 (202m diamond, 3090m RC); TR1-159 (501m RAB). Mining lease application (MLA's 1070, 1071) lodged. Resource estimates; mining-related studies; Batman EM-drilling: BD12, BD8690 (1375m diamond); RC pre-collars and H/W drilling, BP209-220 (1320m RC); Exploration EM and exploration drilling: Tollis, Quigleys, TP9, TD1, QP1-3, QD1-4 (1141 diamond, 278m RC); Negative Exploration Tailings Dam: E1-16 (318m RC); DR1-144 (701. RAB) (Kenny, Wegmann, Fuccenecco, Gibbs).
1990	
Jan-March:	Pre-feasibility (PFS) related studies; Batman Inclined Infill RC drilling: BP222-239 (2370m RC); Tollis RC drilling, TP10-25 (1080m RC). (Kenny, Wegmann, Fuccenecco, Gibbs)
1993 - 1997	
	Pegasus Gold Australia Pty Ltd reported investing more than $200 million in the development of the Mt Todd mine and operated it from 1993 to 1997, when the project closed as a result of technical difficulties and low gold prices. The deed administrators were appointed in 1997 and sold the mine in March 1999 to a joint venture comprised of Multiplex Resources Pty Ltd and General Gold Resources Ltd.
1999 - 2000	
March - June	Operated by a joint venture comprised of Multiplex Resources Pty Ltd and General Gold Resources Ltd. Operations ceased in July 2000, Pegasus Gold Australia Pty Ltd., through the Deed Administrators, regained possession of various parts of the mine assets in order to recoup the balance of purchase price owed to it. Most of the equipment was sold in June 2001 and removed from the mine. The tailings facility and raw water facilities still remain at the site.
2000 - 2006	
	The Deed Administrators, Pegasus Gold Australia Pty Ltd, the government of the NT, and the Jawoyn Association Aboriginal Corporation held the property.

2006	
March	Vista Gold Corp. acquired mineral lease rights from the Deed Administrators.

6.2 Historic Drilling

The following discussion centers on the historic drillhole databases that were provided to Tetra Tech for use in this Technical Report. Based on the reports by companies, individuals and other consultants, it is The QPs' opinion that the drillhole databases used as the bases of this report contain all of the available data. Tetra Tech is unaware of any drillhole data that have been excluded from this report.

6.2.1 Batman Deposit

There are 730 historic drillholes in the Batman deposit assay database. **Figure 6-1** shows the drillhole locations for the Batman deposit. These drillholes include 225-diamond drill core (DDH), 435 reverse circulation holes (RVC), and 70 open rotary holes (OP). Nearly all of the DDH and RVC holes were inclined 60° to the west. Samples were collected in one-meter intervals. DDH holes included both HQ and NQ core diameters. Core recoveries were reported to be very high with a mean of 98%. The central area of the deposit was extensively core-drilled. Outside of the central area, most of the drillholes were RVC and OP holes. All drillholes collars were surveyed by the mine surveyor. Down-hole surveys were conducted on most drillholes using an Eastman single shot instrument. All drillholes were logged on site.

A series of vertical RVC infill holes were drilled on a 25 m x 25 m grid in the core of the deposit to depths between 50 m and 85 m below the surface. Zapopan elected to exclude these drillholes from modeling the Batman deposit because the assays from these drillholes seemed to be downwardly biased and more erratic compared to assays from inclined RVC holes. Of the possible reasons cited as to why vertical RVC holes might report lower grades and have a more erratic character, the 1992 Mining & Resource Technology Pty Ltd (Khosrowshahi et al. 1992 – MRT) report states that "*the orientation of vertical holes sub-parallel to mineralization caused preferential sampling of barren host rocks...*". This statement was, at least in part, borne out by the later sampling work done on the blast holes as it was credited with part of the reproducibility problems that were encountered when the Batman deposit was being mined.

6.2.2 Drillhole Density and Orientation

Pegasus was aware of the potential problem of drillhole density within the Batman deposit. The feasibility study prepared by BKK (BKK, 1996) indicates that the drilling density decreases with depth. In the central area oxide and transition zone spacing was generally 25 m by 25 m. The spacing was wider on the periphery of the mineralized envelope. The drilling density in the central area of the primary zone ranged from 50 m by 50 m, but decreased to 50 m by 100 m and greater at depth. At the time of that study, there were 593 drillholes in the assay database 531 of which RSG used in the construction of the MRT block model.

At the time of The Winters Company's (TWC) site visit in 1997, the drillhole database numbered 730 drillholes. It is not known if any drillholes were excluded from the Pegasus exploration models. Most of the new drilling that had been added since the 1994 MRT model was relatively shallow. TWC reviewed PGA's 50 m drill sections through the Batman deposit and saw that there was a marked decrease in drillhole spacing below 1,000 RL (the model has had constant 1,000 m added to it in order to prevent the reporting of elevations below 0 m and have been denoted as RL for relative elevation) and another sharp break below 900 RL. The drillhole spacing in the south of 1,000 N on the 954 RL bench plan approached 80 m x 80 m. Pegasus was able resolve this problem by using very long search ranges in its grade estimation. In the main ore zone, Pegasus used maximum search distances in the north and east directions of nearly 300 m.

Another potential problem related to drilling is the preferred orientation of the drillholes. Most of the drillholes in the assay database are inclined to the west to capture the vein set which strikes N10° to 20°E, dips east, and which dominates the mineralized envelope. This orientation is the obvious choice to most geologists since these veins are by far the most abundant. Ormsby (1996) discussed that while the majority of mineralization occurs in these veins, the distribution of gold mineralization higher than 0.4 g-Au/t is controlled by structures in other orientations, such as east-west joints and bedding. For this reason, Ormsby stated, *"[t]he result is that few ore boundaries (in the geological model) actually occur in the most common vein orientation.*" If this is truly the case, the strongly preferential drilling orientation has not crosscut the best mineralization and in cases may be sub-parallel to it.

Vertically oriented RVC holes were not included in the drillhole database for the 1994 MRT model because their assay results appeared to be too low compared to other drillhole orientations. If vertical drillhole orientations were actually underestimating the gold content during exploration drilling, the vertical and often wet blast holes, which are used for ore control, pose a similar problem and will need to be addressed prior to commencing any new mining on the site.

6.2.3 Quigleys

Table 6-3 details the Quigleys exploration database as of the time of this report. **Figure 6-1** also shows the drillhole locations for the Quigleys deposit.

Table 6-3: Summary of Quigleys Exploration Database

Category	Count	Min	Max	Average
Count	644	-	-	-
Depth	-	13	368	92
Collar Easting	644	187,067	190,023	189,484
Collar Northing	644	8,437,020	8,439,305	8,438,149
Collar Elevation	644	129	208	156
Survey Azimuth	2,057	0	359	87.36
Survey Dip	2,057	-90	-40	-60
Assay Au	54,073	0	36	0.241
Assay Interval	54,131	0.1	69	1.04

Snowden (1990) completed a statistical study of the Quigleys drillhole database in order to bias test it. A comparison of historic and recent data by Snowden suggested that a bias might exist. Further study concluded that a bias is not apparent where all drilling is oriented in a similar direction (and not clustered). This suggests the inclusion of assay data from all phases of drilling is reasonable. The March 2008 report entitled “Mt Todd Gold Project, Gold Resource Update” contains additional information regarding the Snowden findings.

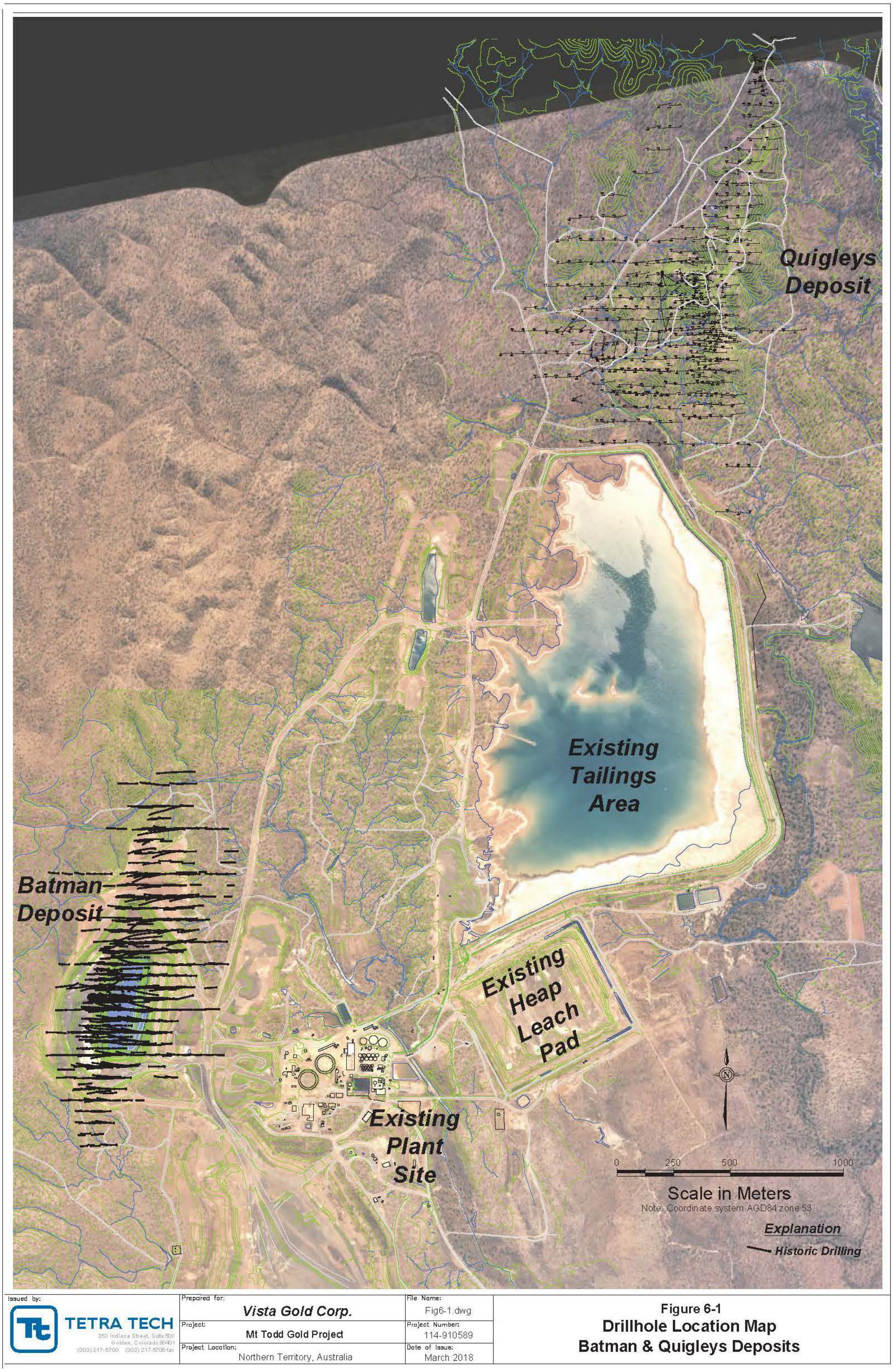


Figure 6-1: Drillhole Location Map – Batman and Quigleys Deposits

6.3 Historic Sampling Method and Approach

NQ core intervals were sawed lengthwise into half core. HQ core was quartered. RVC samples were riffle split on site and a 3- to 4-kg sample was sent to an assay lab. The 1992 MRT resource report commented that many of the RVC holes were drilled wet and that Billiton and Zapopan were aware of possible contamination problems. Oddly, in some comparison tests, DDH had averaged assays five percent to six percent higher than RVC holes; for that reason, MRT elected to exclude RVC holes from the drillhole database for grade estimation of the central area of the Batman deposit.

Since the property is currently not operating, Tetra Tech did not witness any drilling and sampling personally. We have taken the following discussion from reports by the various operators and more importantly, from reports by independent consultants that were retained throughout the history of the property to audit and verify the sampling and assaying procedures. It is the opinion of the QP for this section that the reports by the various companies and consultants have fairly represented the sampling and assaying history at the site and that the procedures implemented by the operators, most notably General Gold, have resulted in an assay database that fairly represents the tenor of the mineralization at Batman.

6.4 Historic Sample Preparation, Analysis and Security

The large number of campaigns and labs used in the Mt Todd drilling effort has resulted in a relatively complex sampling and assaying history. The database developed prior to August of 1992 was subjected to a review by Billiton, and has been subjected to extensive check assays throughout the project life. Furthermore, a number of consultants have reviewed the integrity of the database and have been content with the data for modeling purposes.

Drillhole samples were taken on one-meter intervals, though there are instances of two-meter intervals in the typically barren outlying drillholes. The procedure involved sawing the NQ core lengthwise in half. HQ core was quartered. RVC samples were riffle split on site and a 3- to 4-kg sample was sent to the laboratory for analyses. Pincock Allen and Holt (PAH) stated that they witnessed the sample preparation process at a number of steps and concurred with the methods in use (PAH, 1995).

Pegasus (and Zapopan, before) conducted a check assay program which is consistent with industry practice. Every 20th assay sample was subjected to assay by an independent lab. Standards were run periodically as well, using a non-coded sample number to prevent inadvertent bias in the labs.

6.4.1 Sample Analysis

According to reports by Pegasus, various consultants, and others, the early exploration assays were largely done at various commercial labs in Pine Creek Geosyncline (PCG) and Darwin. Later assays were done at the Mt Todd mine site lab. At least three different sample preparation procedures were used at one time or another. All fire assays were conducted on 50-gram charges. Based on these reports, it appears that the assay labs did use their own internal assay blanks, standards, and blind duplicates.

Assay laboratories used for gold analysis of the Batman drill data were Classic Comlabs in Darwin, Australia, Assay Laboratories in Pine Creek and Alice Springs and Pegasus site Laboratory.

The exploration data consist of 91,225 samples with an average and median length of 1 m. The minimum sample length is 0.1 m and the maximum sample length is 5 m. 137 samples are less than 1 m and 65 samples are over 1 m in length.

All exploration drill data were used for the resource estimate. Four-meter down hole composite samples were calculated down hole for the resource estimate. The assay composited data were tabulated in the database field called "Comp". The weighted average grades, the length, and the drillhole were recorded.

6.4.2 Check Assays

Extensive check assaying was carried out on the exploration data. Approximately 5% of all RVC rejects were sent as duplicates and duplicate pulps were analyzed for 2.5% of all DDH intervals. Duplicate halves of 130 core intervals were analyzed as well. Overall, Mt Todd's check assay work is systematic and acceptable. The feasibility study showed that the precision of field duplicates of RVC samples is poor and that high errors exist in the database. The 1995 study stressed that because of the problems with the RVC assays, the RVC and OP assays should be kept in a separate database from the DDH assays (PAH, 1995). However, since that time, the majority of the identified assaying issues have been corrected by General Gold based on recommendations of consultants. It is the opinion of the QP responsible for this section that the assay database used in the creation of the current independent resource estimation exercise is acceptable and meets industry standards for accuracy and reliability.

6.4.3 Security

The QP responsible for this section is unaware of any "special" or additional security measures that were in place and/or followed by the various exploration companies, other than the normal practices of retaining photographs, core splits, and/or pulps of the samples sent to a commercial assay laboratory.

6.5 Historic Process Description

The Mt Todd deposit is a large, but low-grade gold deposit. The average grade of the gold mineralization is approximately 1 g-Au/t. The gold mineralization occurs in a hard, uniform greywacke host and is associated with sulfide and silica mineralization which has resulted from deposition along planes of weakness that had opened in the host rock. Gold is very fine grained (<30 microns) and occurs with both silica and sulfides. The host rock is very competent with a Bond Ball Mill Work Index (BWi) of 23 to 30.

Pegasus and earlier owners did extensive metallurgical testing from 1988 to 1995 to develop a process flowsheet for recovering gold from low-grade extremely hard rock. The treatment route, based on the metallurgical studies, was engineered to provide for the recovery of a sulfide flotation concentrate which was subsequently reground and leached in a concentrate leach circuit. Flotation tailings were leached in a separate CIL circuit.

The historic design process flowsheet for the Project is given in **Figure 6-2**.

A brief description of the major unit operations is as follows:

- **Crushing:** Four stages of crushing were employed to produce a product having a P_{80} of 2.6mm. The primary crusher was a gyratory followed by secondary cone crushers in closed circuit. Barmac vertical shaft impact crushers were used for tertiary crushing in closed circuit and quaternary crushing stages. The crushed product was stored under a covered fine ore stockpile.

- **Grinding:** The crushed product was drawn from the fine ore stockpile into three parallel grinding circuits, each consisting of an overflow ball mill in closed circuit with cyclones to produce a grind with a P_{80} of 150 microns.
- **Flotation:** Cyclone overflow was sent to the flotation circuit where a bulk concentrate was supposed to recover seven percent of the feed with 65% to 70% of the gold.
- **CIL of Tailing:** The flotation tailing was leached in carbon-in-leach circuit. The leach residue was sent to the tailings pond. Approximately 60% of the gold in the flotation tailings was supposed to be recovered in the CIL circuit.
- **CIL of Flotation Concentrate:** The flotation concentrate was reground in Tower mills to 15 microns and subjected to cyanide leaching to recover the bulk of the gold in this product (94.5% of the flotation concentrate). The leach residue was sent to the tailings pond.
- **Process Recycle:** The process water was recycled to the milling circuit from the tailings pond. The overall gold recovery was projected to be 83.8% for the proposed circuit. However, during the initial phase of plant optimization, problems were encountered with high levels of cyanide in the recycled process water which, when returned to the mill, caused depression of pyrite and much lower recoveries to the flotation concentrate. As a result, the flotation plant was shut down and the ground ore was directly sent to the CIL circuit. The modified process flowsheet is given in **Figure 6-3**.
- Without the flotation circuit, the CIL plant recovered 72 to 75% of the gold.

The plant was shut down and placed on care and maintenance within one year of startup due to a collapse in gold price, under performance of the process plant and higher than projected operating costs.

6.6 Technical Problems with Historical Process Flowsheet

There were several technical problems associated with the design flowsheet. These technical problems have been documented by plant engineers, TWC, and other investigators. They are briefly discussed in this section.

6.6.1 Crushing

The four-stage crushing circuit was supposed to produce a product with P_{80} of 2.6mm. Also, historically the tonnage was projected to be 8 Mtpy on an annualized basis. The actual product achieved in the plant had a P_{80} of 3.2 to 3.5 mm and the circuit could handle a maximum of 7 Mtpy on an annualized basis. This resulted in an increased operating cost for gold production.

A four-stage crushing/ball mill circuit was selected over a SAG/ball mill/crusher circuit because crushers were available from the Phase I heap leach pad and could be used in the Phase II program. The use of this available equipment did reduce the overall capital cost.

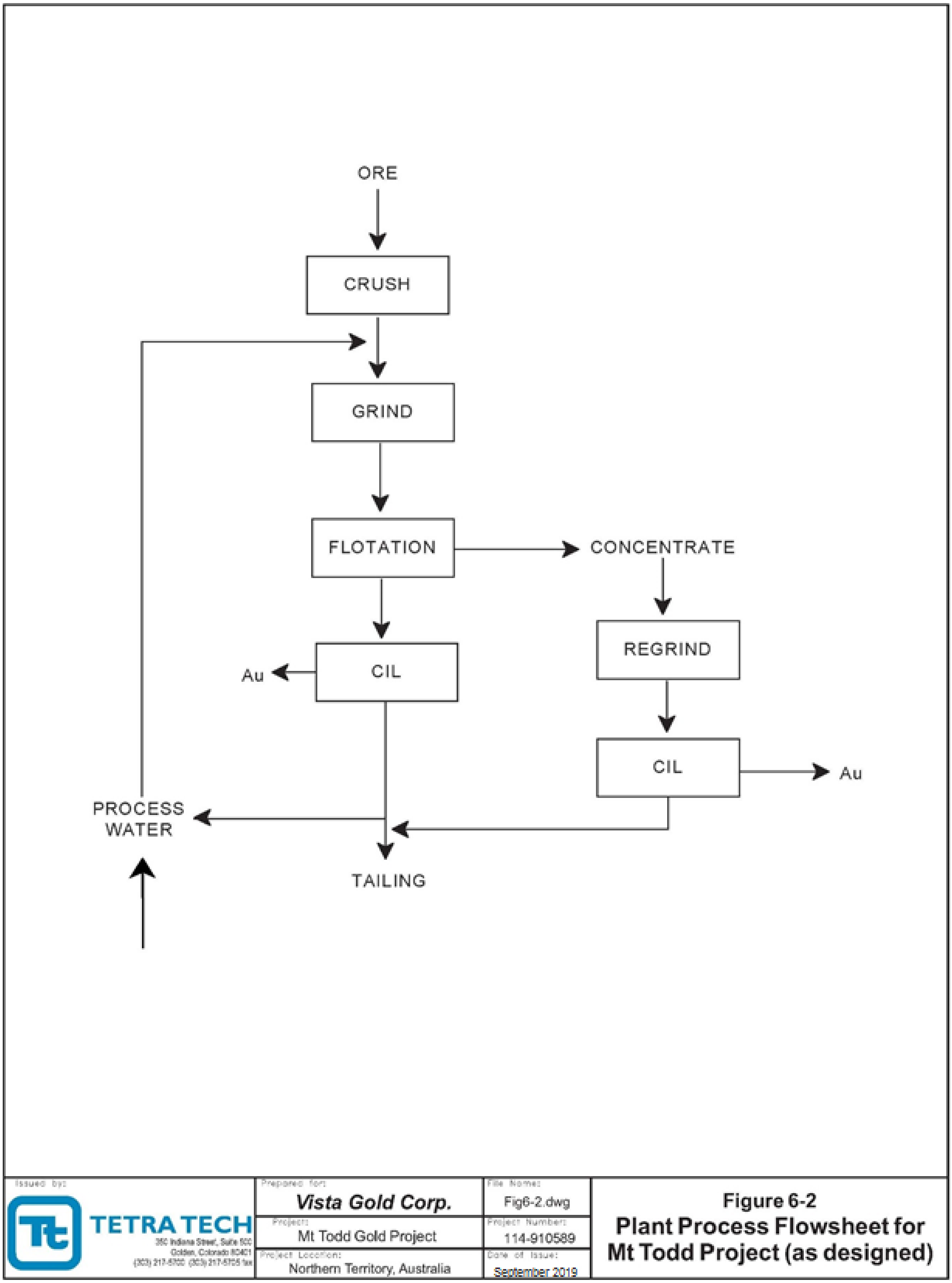


Figure 6-2: Plant Process Flowsheet for Project as Designed

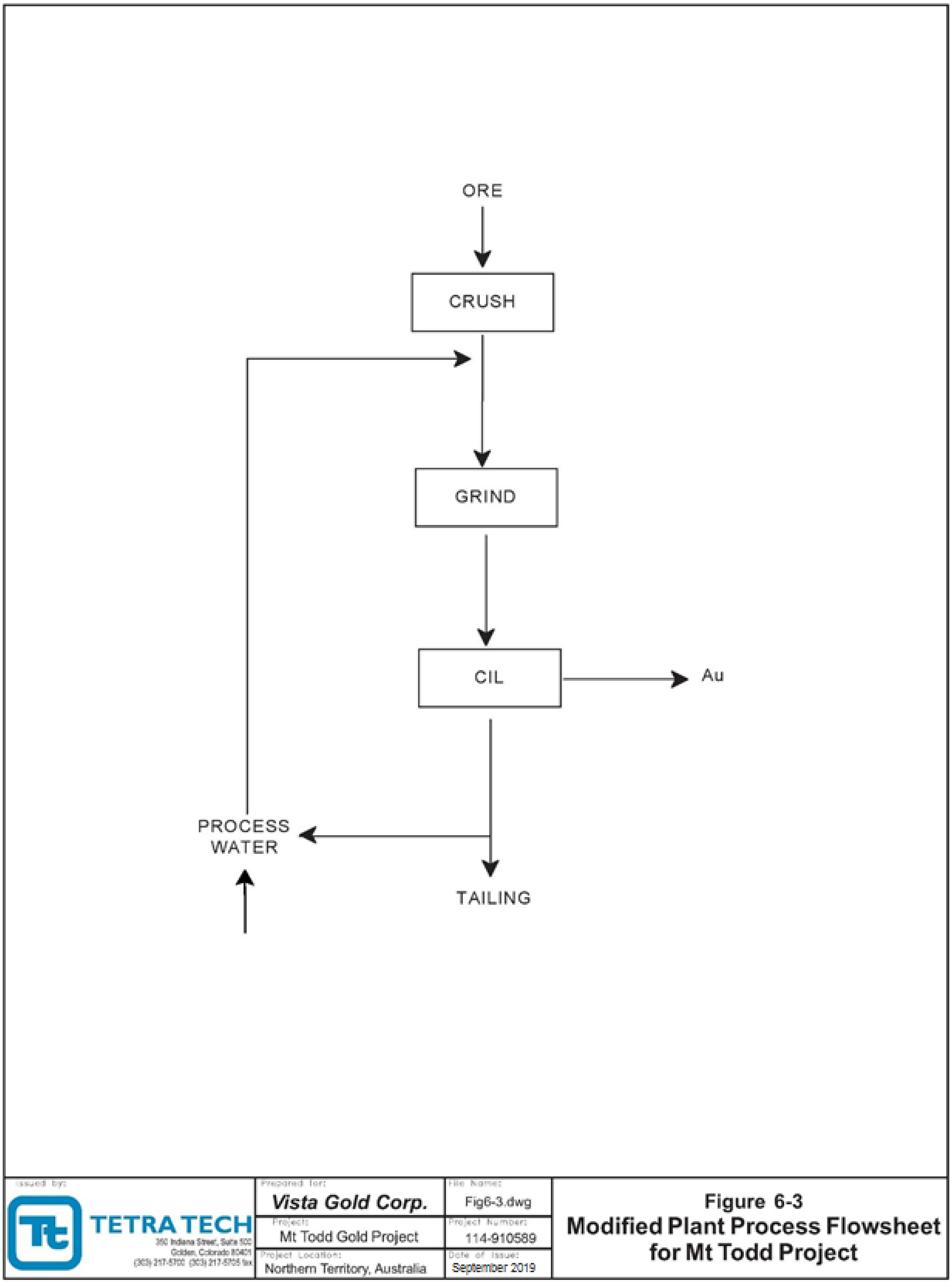


Figure 6-3: Modified Plant Process Flowsheet for Project

The following problems were encountered with the crushing circuit:

- The mechanical availability of the Barmac vertical shaft impact crushers was extremely poor.
- The Barmac crushers were not necessarily the best choice for the application. The three-stage crusher product could have been sent to the mills which would have had to have been larger size mills.
- The crushing circuit generated extreme amounts of fines and created environmental problems. The dust also carried gold with it. The dust levels increased the wear on machinery parts and were a potential long-term health hazard.
- The use of water spray to keep the dust down resulted in use of large amounts of fresh water. This was a strain on the availability of fresh water for the plant.

General Gold operated a whole-ore cyanide leach facility but no technical reports describing their process have been located by Vista to date.

6.6.2 Flotation Circuit

The flotation circuit was supposed to recover 60 to 70% of the gold in a bulk sulfide concentrate which was 7% of the feed material. The flotation circuit recovered ± 1% of the weight of material and less than 50% of the gold values. This was due to the significant amount of cyanide in the recycle process water which depressed the sulfide minerals in the flotation process. If the process water had been detoxified, the problems would not have occurred. This was not done because of the cost associated with a cyanide detoxification plant.

Additional problems which were overlooked during the testwork and design of the plant included the following:

- The presence of cyanide soluble copper was known but was not taken into consideration during the design of the process flowsheet; and
- Removal of copper from the bulk sulfide in the form of a copper concentrate would have reduced the consumption of cyanide as well as the amount of weak acid dissociable (WAD) cyanide in the recycled process water. Pilot plant testing was undertaken in the plant to produce copper concentrate. Documented results do indicate ± 60% of copper recovery at a concentrate grade of +10% Cu. Approximately 45% of the gold reported to this concentrate. However, from our discussions with the engineering contractors and the Pegasus staff running the pilot plant, a copper concentrate assaying over 20% was achieved in some of the later tests.

6.6.3 CIL of Flotation Concentrate and Tailings

A portion of the copper was depressed with cyanide with the recycled process water in the flotation process. Hence, the cyanide consumption was high even in the leaching of the flotation tailings. The availability of dissolved oxygen in leaching terms was very low thereby resulting in poor extraction of gold in the leach circuit. This resulted in an estimated reduction of 40% of gold recovery in the circuit.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological and Structural Setting

The Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline (**Figure 7-1**). Meta-sediments, granitoids, basic intrusives, acid and intermediate volcanic rocks occur within this geological province.

Within the Mt Todd region, the oldest outcropping rocks are assigned to the Burrell Creek Formation. These rocks consist primarily of interbedded greywackes, siltstones, and shales of turbidite affinity, which are interspersed with minor volcanics. The sedimentary sequence incorporates slump structures, flute casts and graded beds, as well as occasional crossbeds. The Burrell Creek Formation is overlain by interbedded greywackes, mudstones, tuffs, minor conglomerates, mafic to intermediate volcanics and banded ironstone of the Tollis Formation. The Burrell Creek Formation and Tollis Formation comprise the Finniss River Group.

The Finniss River Group strata have been folded about northerly trending F1 fold axes. The folds are closed to open style and have moderately westerly dipping axial planes with some sections being overturned. A later north-south compression event resulted in east-west trending open style upright D2 folds.

The Finniss River Group has been regionally metamorphosed to lower green schist facies.

Late and Post Orogenic granitoid intrusion of the Cullen Batholith occurred from 1,789 Ma to 1,730 Ma, and brought about local contact metamorphism to hornblende hornfels facies.

Unconformably overlying the Burrell Creek Formation are sandstones, shales and tuffaceous sediments of the Phillips Creek sandstone, with acid and minor basic volcanics of the Plum Tree Creek Volcanics. Both these units form part of the Edith River Group, and occur to the south of the Project Area.

Relatively flat lying and undeformed sediments of the Lower Proterozoic Katherine River Group unconformably overlie the older rock units. The basal Kombolgie Formation forms a major escarpment, which dominates the topography to the east of the Project area.

7.2 Local Geology

The geology of the Batman deposit consists of a sequence of hornfelsed interbedded greywackes, and shales with minor thin beds of felsic tuff. Bedding is striking consistently at 325°, dipping at 40° to 60° to the southwest. Minor lamprophyre dykes trending north-south pinch and swell, cross cutting the bedding.

Nineteen lithological units have been identified within the deposit and are listed in **Table 7-1** below from south to north (oldest to youngest).

Table 7-1: Geologic Codes and Lithologic Units

Unit Code	Lithology	Description
1	GW25	Greywacke
2	SH24	Shale
3	GW24A	Greywacke
4	SHGW24A	shale/greywacke
5	GW24	Greywacke
6	SHGW23	shale/greywacke
7	GWSH23	greywacke/shale
8	GW23	Greywacke
9	SH22	Shale
10	T21	felsic tuff
11	SH21	Shale
12	T20	felsic tuff
13	SH20	Shale
14	GWSH20	greywacke/shale
15	SH19	Shale
16	T18	felsic tuff
17	SH18	Shale
18	GW18	Greywacke
Int	INT	lamprophyre dyke

Bedding parallel shears are present in some of the shale horizons (especially in units SHGW23, GWSH23 and SH22). These bedding shears are identified by quartz/ calcite sulfidic breccias. Pyrite, pyrrhotite, chalcopyrite, galena and sphalerite are the main primary sulfides associated with the bedding parallel shears.

East west trending faults and joint sets crosscut bedding. Only minor movement has been observed on these faults. Calcite veining is sometimes associated with these faults. These structures appear to be post mineralization.

Northerly trending quartz sulfide veins and joints striking at 0° to 20°, dipping to the east at 60° are the major location for mineralization in the Batman deposit. The veins are 1 millimeter (mm) to 100 mm in thickness with an average thickness of around 8 mm to 10 mm. The veins consist of dominantly quartz with sulfides on the margins. The veining occurs in sheets with up to 20 veins per horizontal m. These sheet veins are the main source of mineralization in the Batman deposit.

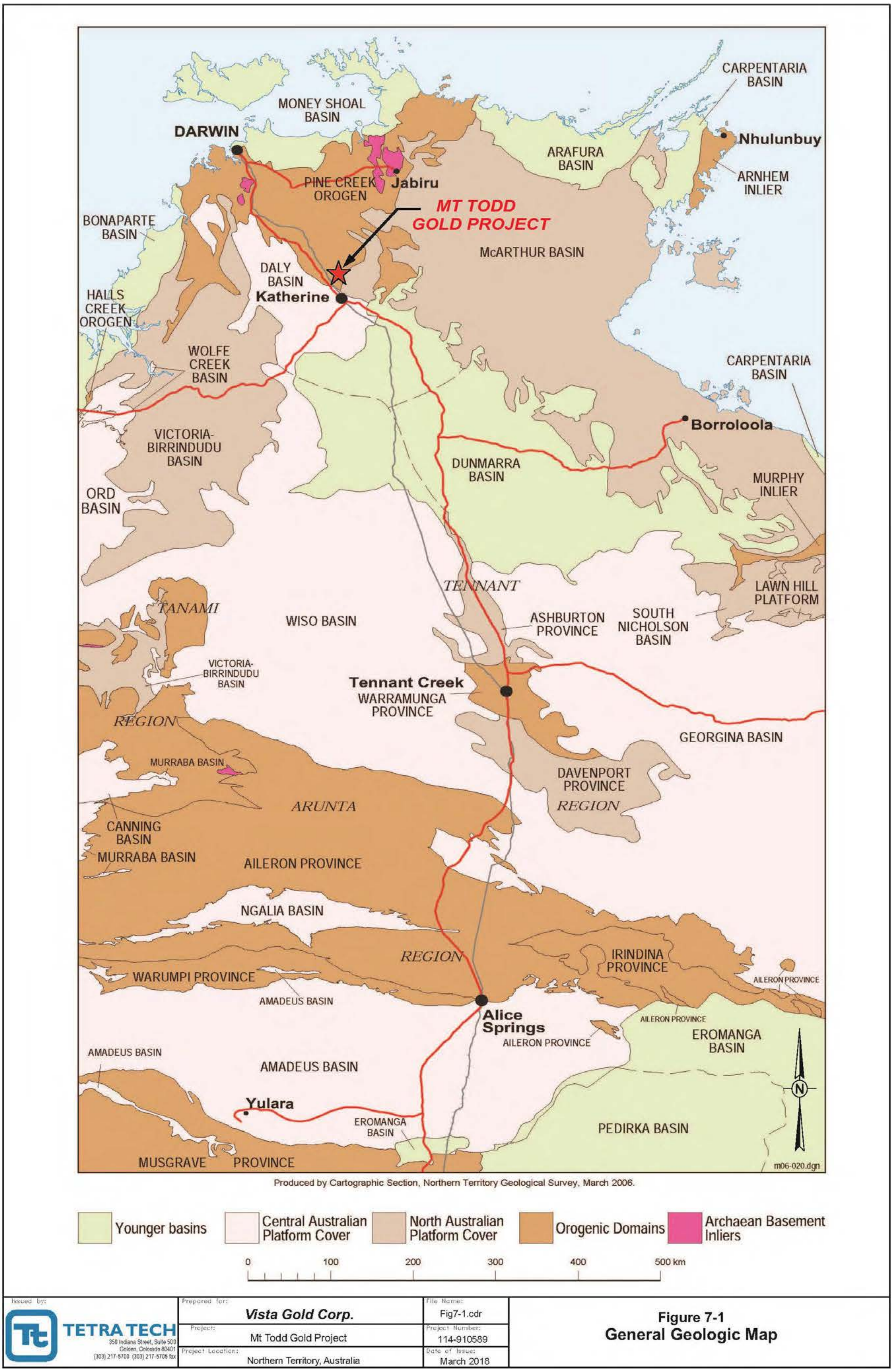


Figure 7-1: General Geologic Map

7.3 Mineralization

A variety of mineralization styles occur within the Mt Todd area. Of greatest known economic significance are auriferous quartz-sulfide vein systems. These vein systems include the Batman, Jones, Golf, Quigleys and Horseshoe prospects, which occur within a north-northeast trending corridor, and are hosted by the Burrell Creek Formation. Tin occurs in a north-northwest trending corridor. The tin mineralization comprises cassiterite, quartz, tourmaline, kaolin, and hematite bearing assemblages, which occur as bedding to parallel breccia zones and pipes. Polymetallic Au, W, Mo, and Cu mineralization occurs in quartz-greisen veins within the Yinberrie Leucogranite; a late stage highly fractionated phase of the Cullen Batholith. The Batman Deposit extends approximately 2,200 m along strike, 400 m across dip and drill tested to a depth of 800 m. Drilling indicates the Batman mineralization to be open along-strike and down-dip.

7.3.1 Batman Deposit

7.3.1.1 Local Mineralization Controls

The mineralization within the Batman deposit is directly related to the intensity of the north-south trending quartz sulfide veining. The lithological units impact on the orientation and intensity of mineralization.

Sulfide minerals associated with the gold mineralization are pyrite, pyrrhotite and lesser amounts of chalcopyrite, bismuthinite and arsenopyrite. Galena and sphalerite are also present, but appear to be post-gold mineralization, and are related to calcite veining in the bedding plains and the east-west trending faults and joints.

Two main styles of mineralization have been identified in the Batman deposit. These are the north-south trending vein mineralization and bedding parallel mineralization.

7.3.1.2 North-South Trending Corridor

The north-south trending mineralization occurs in all rock units and is most dominant in the shales and greywackes designated SHGW23. Inspection of grade control and exploration data, drill logs, diamond core and the pit has shown that the north-south trending mineralization can be divided into three major zones based on veining and jointing intensity.

CORE COMPLEX

Mineralization is consistent and most, to all, joints have been filled with quartz and sulfides. Vein frequency per meter is high in this zone. This zone occurs in all rock types.

HANGING WALL ZONE

Mineralization is patchier than the core complex due to quartz veining not being as abundant as the core complex. The lithology controls the amount of mineralization within the hanging wall zone. The hanging wall zone doesn't occur north of T21. South of reference line T21 to the greywacke shale unit designated GWSH23, the mineralization has a bedding trend. A large quartz/pyrrhotite vein defines the boundary of the hanging wall and core complex in places.

FOOTWALL ZONE

Like the hanging wall zone, the mineralization is patchier than the core complex and jointing is more prevalent than quartz veining. Footwall Zone mineralization style is controlled by the lithology and occurs in all lithological units.

Narrow bands of north-south trending mineralization also occur outside the three zones, but these bands are patchy.

BEDDING PARALLEL MINERALIZATION

Bedding parallel mineralization occurs in rock types SH22 to SH20 to the east of the Core complex. Veining is both bedding parallel and north south trending. The mineralization appears to have migrated from the south along narrow north-south trending zones and "balloon out" parallel to bedding around the felsic tuffs.

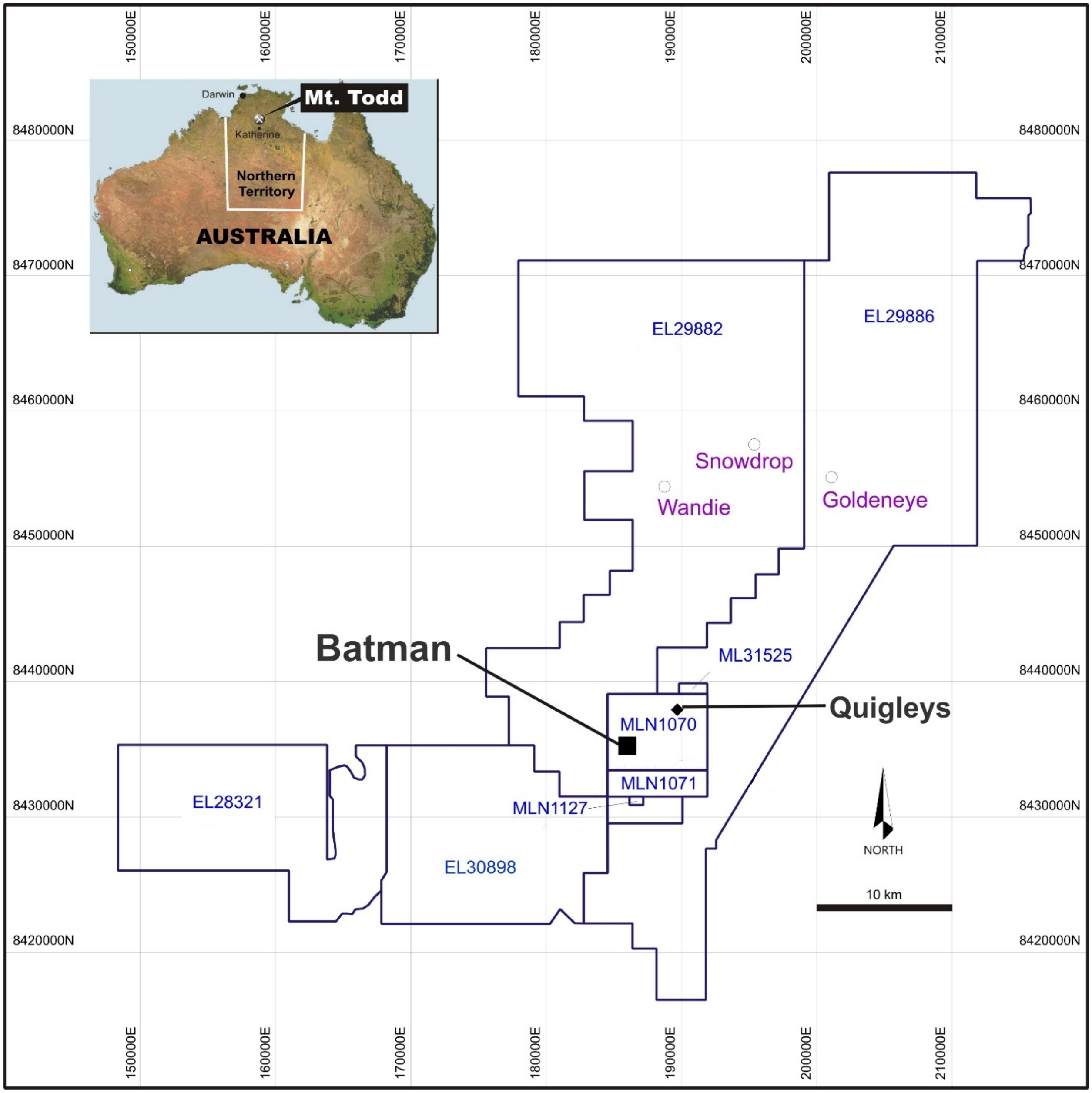
7.3.2 Quigleys Deposit

The Quigleys deposit mineralization was interpreted by Pegasus and confirmed by Snowden (1990) to have a distinctive high-grade shallow dipping 30°-35° northwest shear zone extending for nearly 1 km in strike and 230 m vertical depth within a zone of more erratic lower grade mineralization. The area has been investigated by RVC and diamond drilling by Pegasus and previous explorers on 50 m lines with some infill to 25 m.

Drillhole intersections generally revealed an abrupt change from less than 0.4 g-Au/t to high grade (>1 g-Au/t) mineralization at the hanging wall position of the logged shear, but also revealed a gradational change to lower grade mineralization with depth. Some adjacent drillholes were also noted with significant variation in the interpreted position of the shear zone, and some of the discrepancies appeared to have been resolved on the basis of selection of the highest gold grade. While the above method may result in a valid starting point for geological interpretation, the selection of such a narrow high grade zone is overly restrictive for interpretation of mineralization continuity and will require additional work prior to estimating any resources.

It was further thought that while the shear might be readily identified in diamond drillholes, interpretation in RVC drilling, and in particular later interpretation from previously omitted RVC holes, must invoke a degree of uncertainty in the interpretation. Snowden concluded that while the shear zone was identifiable on a broad scale, the local variation was difficult to map with confidence and therefore difficult to estimate with any degree of certainty at this time.

It is for these reasons that Vista has only drilled diamond drillholes. As reference above, the shears and other structural features are identifiable in drill core.



NOTE: Prepared by Vista Gold Corp.; updated on February 23, 2018

Figure 7-2: Concessions

8.0 DEPOSIT TYPES

According to Hein (2003), the Batman and Quigleys gold deposits of the Mt Todd Mine are formed by hydrothermal activity, concomitant with retrograde contact metamorphism and associated deformation, during cooling and crystallization of the Tennysons Leucogranite and early in D2 (Hein, submitted for publication). It is speculated that pluton cooling resulted in the development of effective tensile stresses that dilated and/or reactivated structures generated during pluton emplacement and/ or during D1 (Furlong et al., 1991, as cited in Hein, 2003), or which fractured the country rock carapace as is typical during cooling of shallowly emplaced plutons (Knapp and Norton, 1981, as cited in Hein, 2003). In particular, this model invokes sinistral reactivation of a northeasterly trending channelization basement strike–slip fault, causing brittle failure in the upper crust and/or dilation of existing north–northeasterly trending faults, fractures, and joints in competent rock units such as meta-greywackes and siltstones. The generation of dilatant structures above the basement structure (i.e., along a northeasterly trending corridor overlying the basement fault), coupled with a sudden reduction in pressure, and concomitant to brecciation by hydraulic implosion (Sibson, 1987; Je´brak, 1997; both as cited in Hein, 2003) may have facilitated channelization of predominantly metamorphic fluid in the intermediate contact metamorphic aureole (possibly suprahydrostatic-pressured) and into the upper crust (Furlong et al., 1991; Cox et al., 2001; both as cited in Hein, 2003). Rising fluids decompressed concurrent with mineral precipitation. Throttling of the conduit or fluid pathways probably resulted in over pressuring of the fluid (Sibson, 2001, as cited in Hein, 2003), this giving way to further fracturing, etc. Mineral precipitation accompanied a decrease in temperature although, ultimately, the hydrothermal system cooled as isotherms collapsed about the cooling pluton (Knapp and Norton, 1981).

Gold mineralization is constrained to a single mineralizing event that included:

- Retrogressive contact metamorphism during cooling and crystallization of the Tennysons Leucogranite;
- Fracturing of the country rock carapace;
- Sinistral reactivation of a NE-trending basement strike-slip fault;
- Brittle failure and fluid-assisted brecciation; and
- Channelization of predominantly metamorphic fluid in the intermediate contact metamorphic aureole into dilatant structures.

The deposits are similar to other gold deposits of the porphyry copper gold (PCG) and are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit shares some characteristics with intrusion-related gold systems, especially in terms of the association of gold with bismuth and reduced ore mineralogies. This makes the deposit unique in the PCG.

The mineral deposit types being investigated and the geological model being applied are described in **Section 9.0 – Exploration** and **Section 14.0 – Mineral Resource Estimates**, respectively.

9.0 EXPLORATION

Since acquiring the Mt Todd mining leases and exploration licenses, Vista has conducted an ongoing exploration program that includes prospecting, geologic mapping, rock and soil sampling, geophysical surveys and exploration drilling. Equipment and personnel were mobilized from the Mt Todd Mine site or from an exploration base camp established in the central part of the exploration licenses. The work was conducted by geologists and field technicians.

The exploration effort initially focused on follow up work on targets developed by Pegasus during their tenure on the property. These included the RKD target, Tablelands, and Silver Spray. During a review of Pegasus' airborne geophysical survey data, five distinct magnetic highs were observed located within sedimentary rocks that should have a low magnetic signature. These features are remarkably similar to those at the Batman deposit, which, as a result of the included pyrrhotite, exhibits a strong magnetic high. The geophysical targets were prioritized following review of historic work in the area and site visits. To date, two of the geophysical targets (Golden Eye and Snowdrop) have been grilled and a third has been covered by soil sampling (Black Hill).

Table 9-1 details soil geochemical samples collected on the exploration licenses (ELs) by year.

Table 9-1: Exploration Sampling

Year	Soils	Samples Collected
2008	0	164
2009	1,333	45
2010	3,135	224
2011	1,925	79
2012	2,312	295
2013	572	51
2014	2,601	143
2015	841	53
2016	241	27
2017	1,098	78
2018	341	132
2019		52
Total Samples	**14,399**	**1,395**

Table 9-2: Exploration Prospects

Year	Drill Hole	Location		Zone	GDA94 Coords		Tasks Completed		
		Prospect	Lease No		Easting	Northing	RL	Depth	Rehab Status
2010									
	GE10-001	Goldeneye	EL29886	53L	200220	8455415	184	252	Closed
	GE10-002	Goldeneye	EL29886	53L	200360	8455415	178	297	Closed
	GE10-003	Goldeneye	EL29886	53L	200340	8455495	189	194	Closed
	GE10-004	Goldeneye	EL29886	53L	200190	8455495	189	194	Closed
	RKD10-001	RKD	EL29882	53L	197400	8450650	201	201	Closed
	RKD10-002	RKD	EL29882	53L	197440	8450550	225	225	Closed
	RKD10-003	RKD	EL29882	53L	197440	8450550	291	291	Closed
	RKD10-004	RKD	EL29882	53L	197400	8450520	336	336	Closed
	RKD10-005	RKD	EL29882	53L	197530	8450450	183	183	Closed
	RKD10-006	RKD	EL29882	53L	197360	8450490	552	352	Closed
2011									
	SS11-001	Silver Spray	EL29882	53L	208572	8460026	217	369	Closed
	SS11-002	Silver Spray	EL29882	53L	208607	8459933	211	438	Closed
	LL11-001	Limestone Quarry	EL28321	52L	813950	8426350	95	60	Closed
	LL11-002	Limestone Quarry	EL28321	52L	813950	8426300	95	60	Closed
	LL11-003	Limestone Quarry	EL28321	52L	813950	8426250	95	60	Closed
	LL11-004	Limestone Quarry	EL28321	52L	814050	8426350	95	64	Closed
	LL11-005	Limestone Quarry	EL28321	52L	814050	8426300	95	61	Closed
	LL11-006	Limestone Quarry	EL28321	52L	814050	8426250	95	60	Closed
	GE11-001	Goldeneye	EL29886	53L	200300	8455555	177	195	Closed
	GE11-002	Goldeneye	EL29886	53L	200240	8455455	182	351	Closed
	GE11-003	Goldeneye	EL29886	53L	200350	8455455	182	241	Closed
	GE11-004	Goldeneye	EL29886	53L	200400	8455500	186	267	Closed
	GE11-005	Goldeneye	EL29886	53L	200400	8455555	186	240	Closed
2012									
	SD12-001	Snowdrop	EL29882	53L	195169	8457484	171	219	Closed
2015									
	SD15-001	Snowdrop	EL29882	53L	195164	8457302	170	250	Closed
	SD15-002	Snowdrop	EL29882	53L	195142	8457248	170	250	Closed
	SD15-003	Snowdrop	EL29882	53L	195305	8457599	170	250	Closed
	WD15-001	Wandie	EL29882	53L	190947	8455709	169	46	Closed
	WD15-002	Wandie	EL29883	53L	190920	8455696	168	100	Closed
	WD15-003	Wandie	EL29884	53L	190890	8455679	167	135	Closed
2016									
	WD16-001	Wandie	EL29882	53L	190859	8455663	166	204	Closed
								6,445	

Year	Drill Hole	Location		Zone	GDA94 Coords		Tasks Completed		
		Prospect	Lease No		Easting	Northing	RL	Depth	Rehab Status
2018									
	WD18-001	Wandie	EL29882	53L	190220	8456760	148	279.5	Open
	WD18-002	Wandie	EL29882	53L	190275	8456640	149	291.4	Open
								7,016	

9.1 Golden Eye Target

At Golden Eye, an initial 100m x 100m soil program identified 2 anomalous samples, one of 70PPB and one of 50ppb, follow-up rock chip sampling, in an area with limited exposure, returned a 25.0 g-Au/t sample from a small outcrop of Laminated Fe rich sediments. Further sampling returned 23.0 g-Au/t and 7.7 g-Au/t assays in vein and breccias located 15 m and 50 m, respectively, north of the original sample. Due to the sparse outcrop, the orientation and thickness of the mineralized zone is not currently known. An infill soil sampling program over the area was completed on a 20 m grid. The survey returned a strong coherent gold anomaly approximately 400 m in diameter with coincident anomalous base metals and arsenic.

In 2010 Vista completed four drillholes on the target. All four drillholes intersected strong sulfide mineralization associated with laminated Fe rich Burrell Creek Formation with interesting concentrations of copper, lead zinc and anomalous gold mineralization, with the best intercept occurring in drillhole GE10-003 and consisting of 1.1 m of 7.69 g-Au/t including 0.3 m of 26.7 g-Au/t.

Five additional drillholes were completed during the 2011 field season. Drilling intersected several narrow weakly mineralized zones; however, none that can yet be correlated with any confidence between different drillholes or between the drillholes and the mineralization identified on the surface. The most encouraging mineralization was intersected by GE11-002, consisting of a sheared, chloritic and broken sulfide-rich unit from 54.2 m to 55 m which assayed 1.41 g-Au/t and a siliceous lode from 162.07m to 162.82 m which assayed 1.86 g-Au/t. The remaining drillholes all intersected widespread quartz sulfide veining containing pyrrhotite, chalcopyrite, and arsenopyrite and contained anomalous gold, copper, bismuth, and arsenic. Although thin and patchy, this mineralization is at least a clear indication that there is a mineralized system at Golden Eye which is yet to be defined with confidence.

A detailed ground magnetic survey was completed over the area in 2012 and an airborne UTS geophysical survey was conducted in 2013. One IP line was conducted in 2017 to determine if a more extensive program would be helpful, this defined a thin target zone. The survey results, combined with detailed mapping and the drillhole data, have been reviewed and additional drilling is recommended.

9.2 RKD Target

Six drillholes totaling 1,587.4 m were completed on the target known as RKD during 2011. The drillholes intersected a NNW trending mineralized shear zone dipping steeply to the west. The best gold intercept was in drillhole RKD11-003 which contained 2.7 m of 2.3 g-Au/t. Drillhole RKD11-005 intersected 3 m of 3.4% copper and 50 ppm silver a chalcocite-rich part of the shear zone. All of the drillholes intersected anomalous gold with values up to 0.4 and 0.5 g-Au/t. Extensive surface mapping and rock-chip sampling indicates that RKD is likely to be thin and is strike constrained.

9.3 Silver Spray Target

Two drillholes totaling 806.8 m were completed at Silver Spray. The drillholes intersected strong chloritic alteration throughout both drillholes. Both drillholes intersected several 20-m zones of strong quartz veining with a thin (30 cm) zone of galena, pyrrhotite and arsenopyrite. These zones contained anomalous lead, zinc, and arsenic but only sporadic anomalous gold (up to 0.18 g-Au/t).

9.4 Snowdrop Target

In 2011, 100m x 100m soil geochemical lines were completed across the Snowdrop magnetic anomaly. These soils were later closed in on a 20-m spacing. The results confirmed and refined the gold-copper-arsenic-bismuth anomaly with 146 samples of 481 samples containing 100 ppm or greater copper and 60 samples containing greater than 5 ppb gold (high value 97 ppb). The onset of the wet season has suspended work on the target until next spring. A drill plan will be included in the updated mine management plan to permit drilling in 2012.

In 2012, the detailed 20 m by 20 m infill soil sampling program was continued. A total of 3,376 soils have been collected in the target area. Results show a coherent gold anomaly that is 200-m wide and at least 700-m long. It is oriented NE-SW and flanks a strong magnetic high. There is a strong correlation with As, Bi and Fe with zoned Cu and Zn on the margins. Rock chip sampling in the area has identified the highest grades within gossanous rocks associated with quartz float. Rock chip samples range up to 6 ppm.

In late November, 2012, a single diamond drillhole was completed on the target before the onset of the wet season. SD12-01 was drilled at an angle across the target zone to a depth of 219.1m. The drillhole intersected zones of intensely silicified greywackes and shales with minor sheeted quartz veins. The alteration and veining is notably similar to that observed at the Batman deposit in the vicinity of the core zone. The greywacke units are coarser grained than at Batman, but the frequency of lithological changes and alteration types are all very similar. Sulfides are present within the quartz veining and as disseminated blebs within intensely silicified siltstones. Common sulfide minerals include pyrite, pyrrhotite, chalcopyrite, and arsenopyrite with traces of galena, sphalerite and bornite. Veining has a steep dip to the east, similar to Batman, but appears richer in base metals. Disseminated sulfides are also more abundant, while the vein density is not as intense as Batman. Although the drillhole did not intersect significant ore grade mineralization, assay results were encouraging and additional drilling is warranted. The highest grade intercept was 0.90 g-Au/t with six intervals returning greater than 0.4 g-Au/t. In total, 80 intervals out of 272 samples contained detectable gold with two intervals greater than 30 m containing detectable gold. Two geochemical signatures are apparent in the assay data; one with gold associated with anomalous base metals and one with an association with As, Bi, Co, and Te.

To date, this early-stage exploration program has not produced an announceable discovery on the ELs. While the work is promising and will be ongoing, there are no quantifiable resources or reserves. Once an announceable discovery is made, Vista will detail that discovery according to all applicable disclosure regulations.

10.0 DRILLING

10.1 Drilling

The drilling discussed in this section is limited to the Batman deposit since the filing of the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, Amended & Restated; July 7, 2014. These drillhole data were used to complete the resource estimate in August 2017 as presented in this Technical Report. All of the drillhole data presented in the aforementioned 2014 Technical Report are still valid and have been incorporated as presented therein.

Between the fourth quarter of 2012 and the end of the first quarter of 2017, the Vista exploration program at the Batman deposit consisted of 22 diamond core drillholes containing 12,530 m that targeted both infill definitional drilling and step-out drilling. **Figure 10-1** contains information for the 22 drillholes completed.

Figure 10-1 is a plan map that details the locations of all exploration drillholes drilled at the Batman deposit up to and including VB18-003.

Table 10-1: Batman Deposit Drillholes Added for Resource Update

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB12-015	8434901.6	187446.7	144.4	268	-55	745.85	Diamond
VB12-016	8434703.6	187262.7	147.3	267	-61	713.5	Diamond
VB12-017	8435349.1	187391.2	150.8	277	-61	833.28	Diamond
VB12-018	8434849.2	187429.9	144.7	270	-56	177	Diamond
VB12-019	8434846.9	187429.4	144.8	269	-61	731.8	Diamond
VB12-020	8435852.4	187359.6	167.3	272	-67	611.9	Diamond
VB12-021	8435954.0	187378.8	149.9	271	-65	602.9	Diamond
VB12-022	8434453.4	187179.3	153.3	269	-57	647.9	Diamond
VB12-023	8435801.3	187371.0	161.3	265	-60	650.88	Diamond
VB12-024	8434482.1	187094.7	149.8	266	-58	460.14	Diamond
VB12-025	8435656.2	187344.7	158.6	261	-60	650.63	Diamond
VB12-026	8434393.4	187066.8	144.8	270	-59	378.9	Diamond
VB12-027	8435717.0	187259.7	169.8	291	-54	434.75	Diamond
VB15-001	187431	8434480	147	268.3	-75.812	455.5	Diamond
VB15-001W1	187431	8434480	147	268.3	-75.812	831.8	Diamond
VB15-001W2	187431	8434480	147	268.3	-75.812	746	Diamond
VB15-002	187277	8434703	147.268	266.07	-76.19	446.3	Diamond
VB15-002W1	187277	8434703	147.268	266.07	-76.19	705	Diamond
VB16-002*	187195	8434849	134.84	328.6	-64	485.7	Metallurgical Diamond
VB17-001*	187094	8435292	161.5	184.6	-55	166.6	Metallurgical Diamond
VB17-002*	187194	8434848	134.84	330.6	-64	485	Metallurgical Diamond
VB17-003*	187091	8435290	161.5	188.2	-55	568.9	Metallurgical Diamond
VB17-004*	187332	8435054	147.23	269	-58	509.41	Metallurgical Diamond

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB18-001*	187418	8434999	146.84	270	-50	586.5	Metallurgical Diamond
VB18-002*	187290	8435184	139	275	-58	409.7	Metallurgical Diamond
VB18-003*	187289.5	8435184	139	275	-54	394.9	Metallurgical Diamond

NOTE: Metallurgical drillholes are not used in the resource estimation.

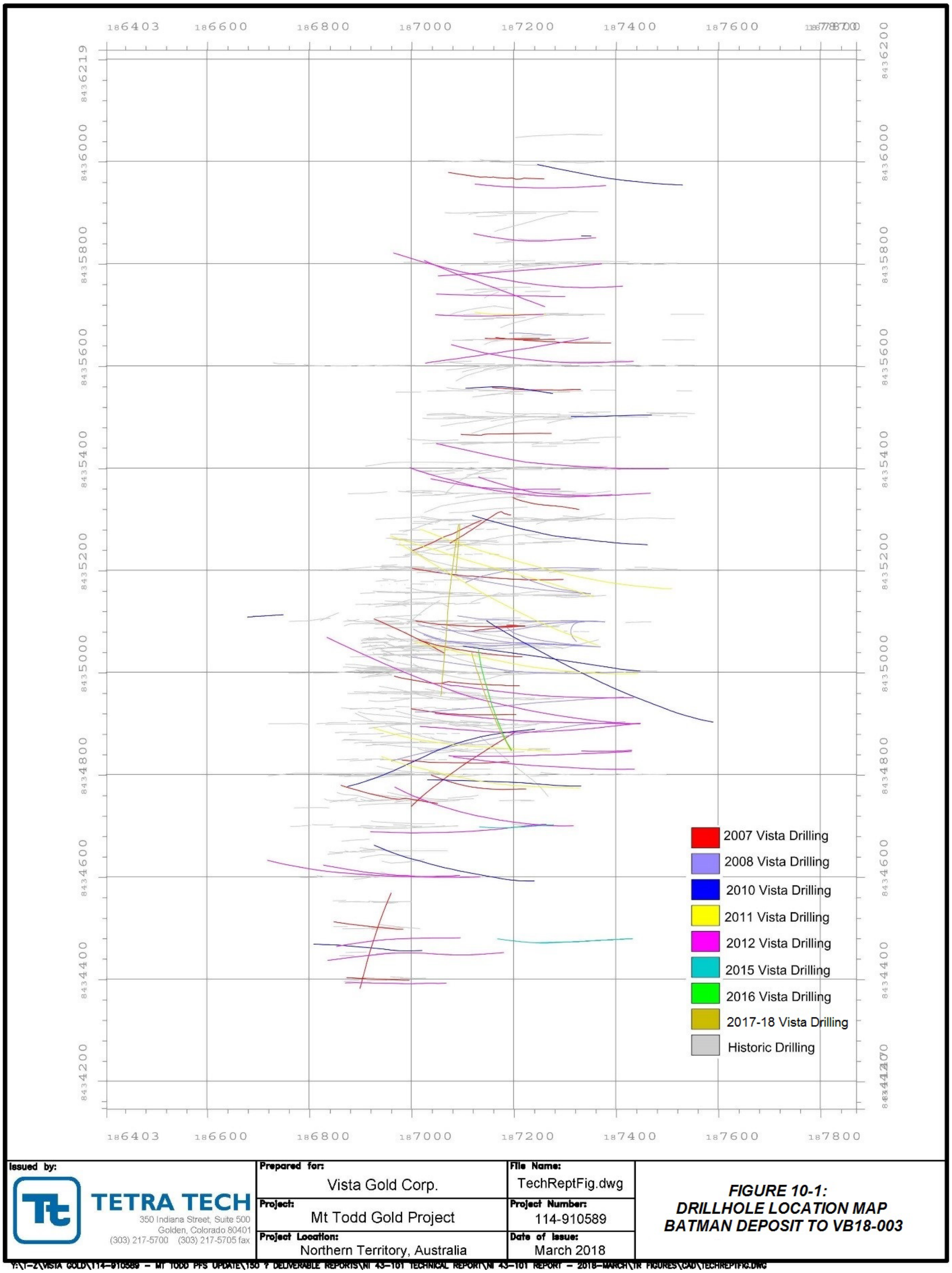


Figure 10-1: Drillhole Location Map Batman Deposit to VB18-003

10.2 Sampling

The sampling method and approach for drillholes completed between 2012 and 2018 was the same as has been used by Vista at the Batman Deposit for all of the Vista diamond drilling. The drill core, upon removal from the core barrel, is placed into plastic core boxes. The poly core boxes are transported to the sample preparation building where the core is marked, geologically logged, geotechnically logged, photographed, and sawn into halves. One-half is placed into sample bags as one-meter sample lengths, and the other half retained for future reference. The only exception to this is when a portion of the remaining core has been flagged for use in the ongoing metallurgical testwork.

The bagged samples have sample tags placed both inside and on the outside of the sample bags. The individual samples are grouped into "lots" for submission to Northern Analytical Laboratories for preparation and analytical testing. All of this work was done under the supervision of a Vista geologist.

Neither Vista nor Tetra Tech are aware of any drilling, sampling, or assaying issues that would materially impact the accuracy or the results presented in this Technical Report.

The QP has observed the sampling, statistically tested the approach, confirmed quality control procedures employed, and quality assurance actions taken for the Project, and is of the opinion that the data accurately represent the nature and extent of the deposit.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section describes the sample preparation, analyses and security undertaken by Vista through the March 2018 resource update.

All of the sample preparation, sample analysis and sample security information presented below is the same and unchanged from Section 11 of the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, Amended & Restated; July 7, 2014, with the exception that the same methodologies, practices, analytical preparation and security has been applied to the drillholes referenced in Section 10 of this Technical Report.

11.1 Sample Preparation

The diamond drilling program was conducted under the supervision of the geologic staff composed of a chief geologist, several experienced geologists, and a core handling/cutting crew. The core handling crew was labor recruited locally.

Facilities for the core processing included an enclosed logging shed and a covered cutting and storage area that was fenced in. Both of these facilities were considered to be limited access areas and kept secured when work was not in progress.

The diamond drill core was boxed and stacked at the rig by the drill crews. Core was then picked up daily by members of the core cutting crew and transported directly into the logging shed.

Processing of the core included photographing, geotechnical and geologic logging, and marking the core for sampling. The nominal sample interval was one meter. When this process was completed, the core was moved into the core cutting/storage area where it was laid out for sampling. The core was laid out using the following procedures:

- One meter depth intervals were marked out on the core by a member of the geologic staff;
- Core orientation (bottom of core) was marked with a solid line when at least three orientation marks aligned and used for structural measurements. When orientation marks were insufficient an estimation orientation was indicated by a dashed line;
- Geologic logging was then done by a member of the geologic staff. Assay intervals were selected at that time and a cut line marked on the core. The standard sample interval was one-m, with a minimum of 0.4 m and a maximum of 1.4 m;
- Blind sample numbers were then assigned based on pre-labeled sample bags. Sample intervals were then indicated in the core tray at the appropriate locations;
- Each core tray was photographed and restacked on pallets pending sample cutting and stored on site indefinitely; and
- 9,635 assays were added for the October 2012 resource update, an additional 7,601 assay intervals were added for the March 2013 resource, and 729 assay intervals were added for the 2017 model update.

The core was then cut using diamond saws with each interval placed in sample bags. At this time, the standards and blanks were also placed in plastic bags for inclusion in the shipment. A reference standard or a blank was inserted at a minimum ratio of 1 in 10 and at suspected high grade intervals additional blanks sample were added. Standard reference material was sourced from Ore Research & Exploration Pty Ltd and provided in 60 g sealed packets. When a sequence of five samples was completed, they were placed in a shipping bag and closed with a zip tie. All of these samples were kept in the secure area until crated for shipping.

Samples were placed in crates for shipping with 100 samples per crate (20 shipping bags). The crates were stacked outside the core shed until picked up for transport.

11.2 Sample Analyses

The following laboratories have been used for lab preparation, analyses, and check assays (**Table 11-1**).

Table 11-1: Assay and Preparation Laboratories

Laboratory	Address	Purpose	Abbreviation	Certifications
ALS | Minerals	31 Denninup Way Malaga, WA 6090	Main assay analyses	ALS	ISO:9001:2008 and ISO 17025 Certified
ALS | Minerals	13 Price St Alice Springs, NT 0870	Sample Preparation	ALS Alice Springs	ISO 9001:2008 and ISO 17025 Certified
Genalysis Laboratory Services (Intertek Group)	15 Davison St Maddington, WA 6109	Check Analyses	Genalysis	Unable to verify
North Australian Laboratories Pty Ltd	MLN 792 Eleanor Rd Pine Creek, NT 0847	Alternative assay analyses	NAL	ISO 10725 Certified
NT Environmental Laboratories (Intertek Group)	3407 Export Dr Berrimah, NT 0828	Check Analyses	NTEL	ISO 17025

Prior to the 2011 drilling campaign, the majority of samples were transported first to ALS in Alice Springs (NT) for sample preparation. After preparation, samples were then forwarded on to ALS in Malaga (WA) for assay analyses. One in every 20 pulp or reject was sent from ALS in Alice Springs to Northern Australian Laboratories (NAL), Vista was notified by email which samples were sent to NAL. For the 2011-2012 drilling campaign samples for assay were sent to NAL lab in Pine Creek, NT. Check assays on one in every 20 pulps or rejects were completed by NT Environmental Laboratories.

Following completion of assay results, all pulps and reject material was shipped back to the Project site and stored.

Vista is completely independent of any analytical testing entity presented in this Technical Report, other than they have engaged said entities as a customer.

11.3 Sample Security

NAL is the primary laboratory for the current drilling program. The NAL laboratory is located in the town of Pine Creek, approximately 100 km distant by road. Samples were picked up and transported by NAL employees.

Sample shipments were scheduled for approximately once a week. The crates were picked up on site by NAL for direct road transport to the assay lab. A sample transmittal form was prepared and included with each shipment and a copy was filed in the geologist office on site.

When the shipment left site, sample transmittals were prepared and e-mailed to NAL. When the shipment arrived at the preparation facility the samples were lined out and a confirmation of sample receipt was e-mailed back to Vista.

The QP is satisfied with the adequacy of sample preparation, security and analytical procedures employed by Vista given the fact that Vista has completed more than 50,000 m of core drilling in the Batman deposit, to verify the approximately 98,000 m of historic drilling. Statistical analysis of the various drilling populations and quality assurance/quality control (QA/QC) samples has not either identified or highlighted any reasons to not accept the data as representative of the tenor and grade of the mineralization estimated at the Batman deposit.

12.0 DATA VERIFICATION

12.1 Drill Core and Geologic Logs

Multiple site visits were performed by Mr. Rex Bryan, QP for the resource estimation portion of this Technical Report. During that visit, Tetra Tech found a comprehensive drillhole database comprised of drill core, photographs of the drill core, assay certificates and results, and geologic logs. All data were readily available for inspection and verification. In addition, most of the subsequent companies or their consultants that have examined the project have completed checks of the data and assay results. The author reviewed drill core, drill core logs and assay certificates and found a minimal number of errors (i.e., mislabeled intervals, number transpositions), which were corrected in development of the resource estimation. It is the opinion of the QP responsible for this section that the databases and associated data were of a high quality in nature and valid for use in mineral resource and reserve estimation.

The QP responsible for this section found no significant discrepancies with the existing drillhole geologic logs and is satisfied that the geologic logging, as provided for the development of the three-dimensional geologic models, fairly represents both the geologic and mineralogic conditions of each of the deposits that comprise the Project.

12.2 Topography

The topographic map of the project area was delivered electronically in an AutoCAD® compatible format and represents the topography in half-meter accuracy. The native coordinate system of the topography is GDA94 / Map Grid of Australia (MGA) zone 53, and for this resource update and as the Project goes forward GDA94 / MGA zone 53 will be the used coordinate system. The surveyed drillhole collar coordinates, once translated to GDA94 / MGA zone 53 agree well with the topographic map; it is the opinion of the QP for this section that the current topographic map is accurate and accurately represents the topography of the project area. In addition, it is suitable for the development of the geologic models, mineral resource estimates, and mineral reserve estimates.

12.3 Verification of Analytical Data

As part of the 2007 exploration program, an exercise to both verify the historic assay results and ensure that future analytical work meets current NI 43-101 standards for reporting of mineral resources was completed. This program consisted of two components; re-assaying of a portion of the historic drillholes, and assaying of the new core drillholes.

A multi-phase program evaluated the accuracy of gold assays generated by NAL on Mt Todd core samples. The test involved three phases including, 1) cross checking assay standards used in the program between NAL and ALS-Chemex, 2) preparing and assaying 30, one-m intervals of remaining half-core and detailed analysis of crushing and analytical performance between the two labs, and 3) screen sieve assay analysis of 45 coarse reject samples plus the 45 comparable remaining half core samples.

Analysis of the results from the two labs confirmed that finer material tends to be higher grade and that this fine material had been preferentially lost through the coarse-weave sample bags during storage and handling of the coarse reject samples. Vista now uses commercial polyester sample bags and loss of fines is no longer an issue. The test also showed good reproducibility between labs in all tests at grade ranges typical of the deposit. Greater variance, which is not unexpected, showed up in the few samples assaying in the 5-20 g-Au/t range.

Figure 12-1, **Figure 12-2** and **Figure 12-3** detail the results of the analytical check program that was completed on the 2007 exploration drillholes. The program was designed to check both internal laboratory accuracy and inter-laboratory accuracy. NAL was the primary laboratory for completion of the sample analyses. ALS-Chemex in Sydney, Australia performed the inter-laboratory analyses. As can be seen from the plots, the correlation coefficient was 0.997 for the re-splits of original assays, 0.992 for pulp repeats, and 0.986 for inter-laboratory analyses, respectively.

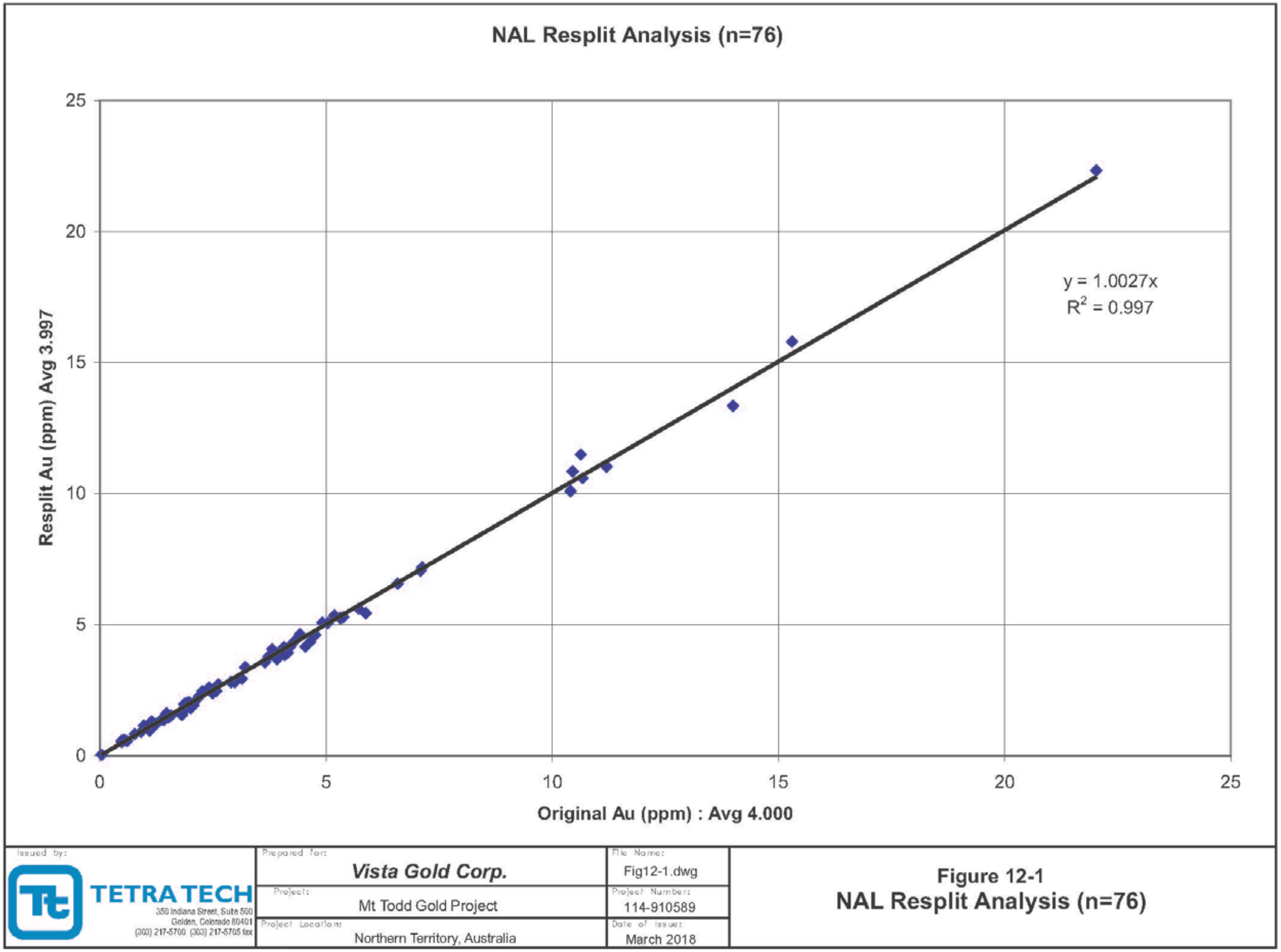


Figure 12-1: NAL Resplit Analyses

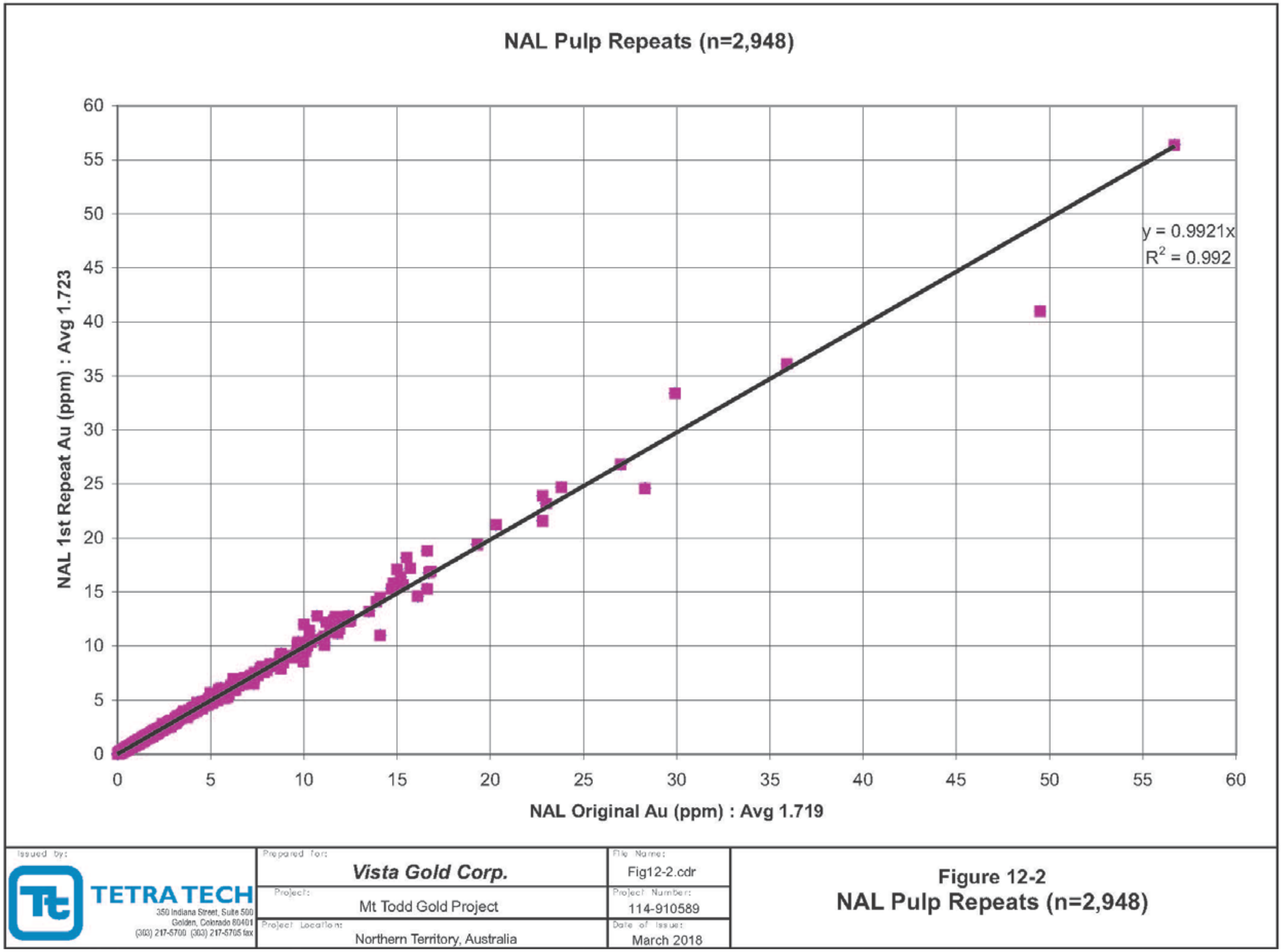


Figure 12-2: NAL Pulp Repeats

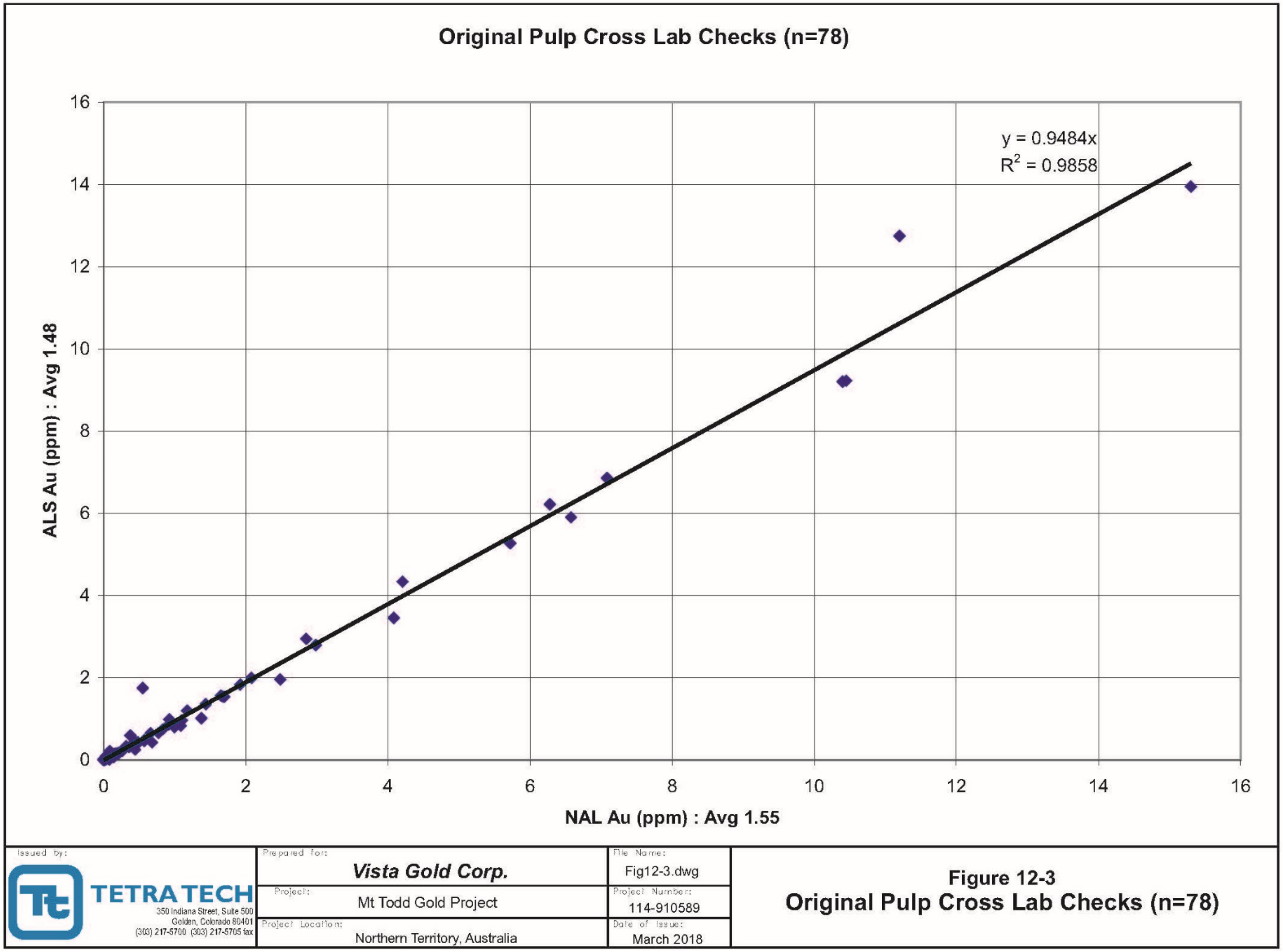


Figure 12-3: Original Pulp Cross Lab Checks

12.3.1 Latest Drilling Data Verification

For the March 2018 resource estimate, a detailed data verification procedure was undertaken by Tetra Tech which focused on two drilling campaigns (VB12-015 through VB17-003 inclusive). This verification was accomplished by: reviewing the assay database received from Vista, comparing results with laboratory certificates received directly from the laboratory and reviewing results of the field QA/QC samples. In April 2018, Tetra Tech verified that the latest four metallurgical drillholes (VB17-004, VB18-001, -002, and -003) followed Vista's drilling and sampling protocols.

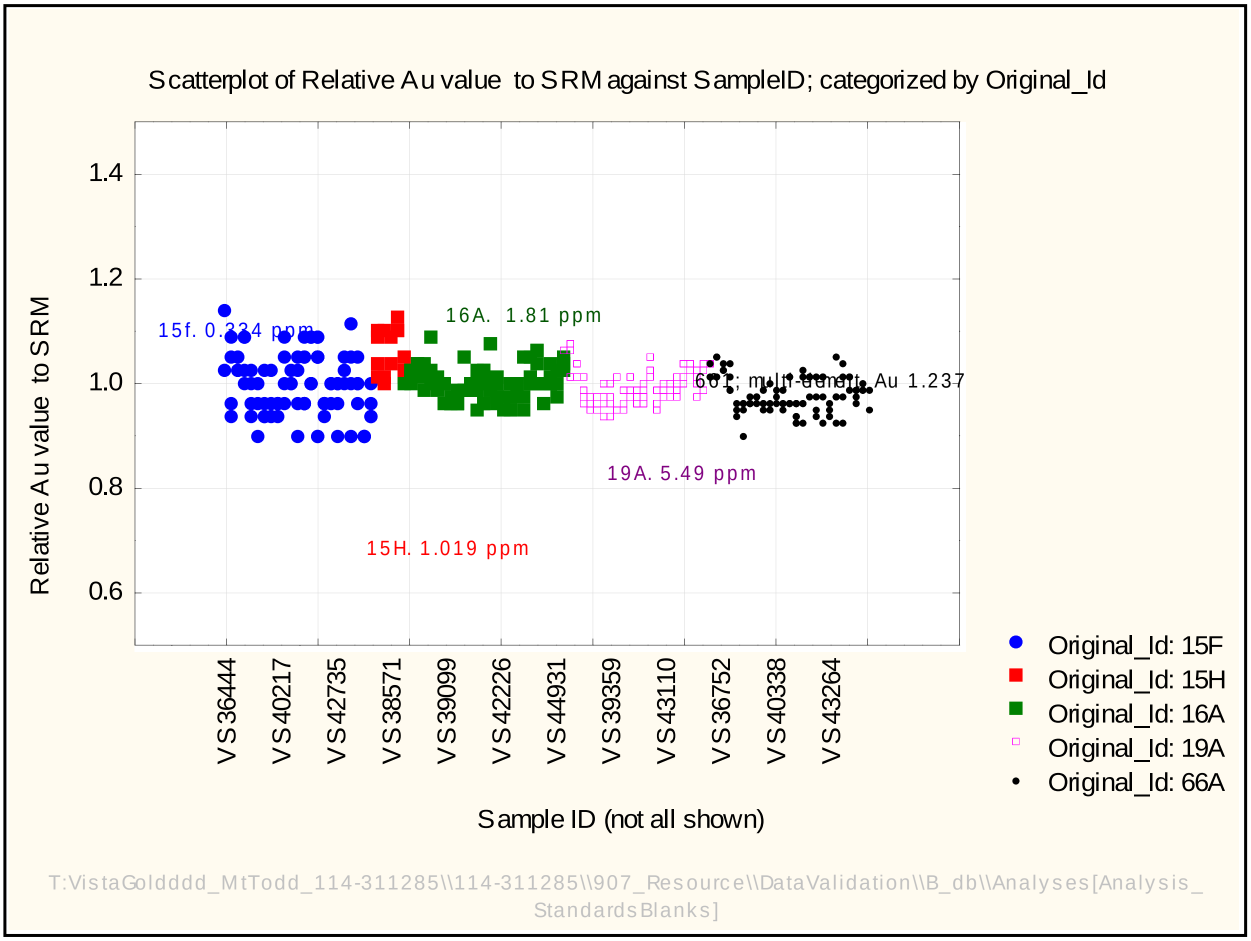
For the 13 drillholes from the 2012 exploration program, there were 7,601 intervals assayed. For the nine drillholes from the combined 2015-2017 exploration programs, there were 1,770 intervals. In addition to Au and other precious metals, most intervals had multi-element and environmental test results as well. Similar to previous work, the assay interval averaged one meter with a minimum interval of 0.4 m and a maximum interval of 1.4 m. No errors were noted in the assay data received other than selenium results for one drillhole that were erroneously entered. This was corrected by Vista. A spot-check of approximately 14% of the received database with laboratory certificates requested and received from NAL showed a 100% correct correlation of reported values.

Field QA/QC samples (those submitted with the drillhole samples to the laboratory) were also analyzed. Five standards (standard reference materials [SRMs]) were used by Vista with ranges of Au between 0.334 and 5.49 ppm of variable mineral/rock composition. Results of the SRMs were plotted as the relative difference to the average SRM certified Au concentration and are shown in **Figure 12-4**. Of the 385 results, no drift was noted over time and all but four were within 10% of the certified value. Of the four that fell outside that range the highest offset was 13.8%. One value was clearly a mislabeled sample and when plotted with the assumed correct standard fell within the 10% range. **Figure 12-4** demonstrates the variance is greatest at lower Au concentrations and this is normally seen with most Au analytical data.

Field blanks were also reviewed and found to be acceptable. Of 388 blank results, six blanks had Au concentrations greater than detection limit of 0.01. The maximum value was 0.11 ppm. Again, no drift was noted in the data over time.

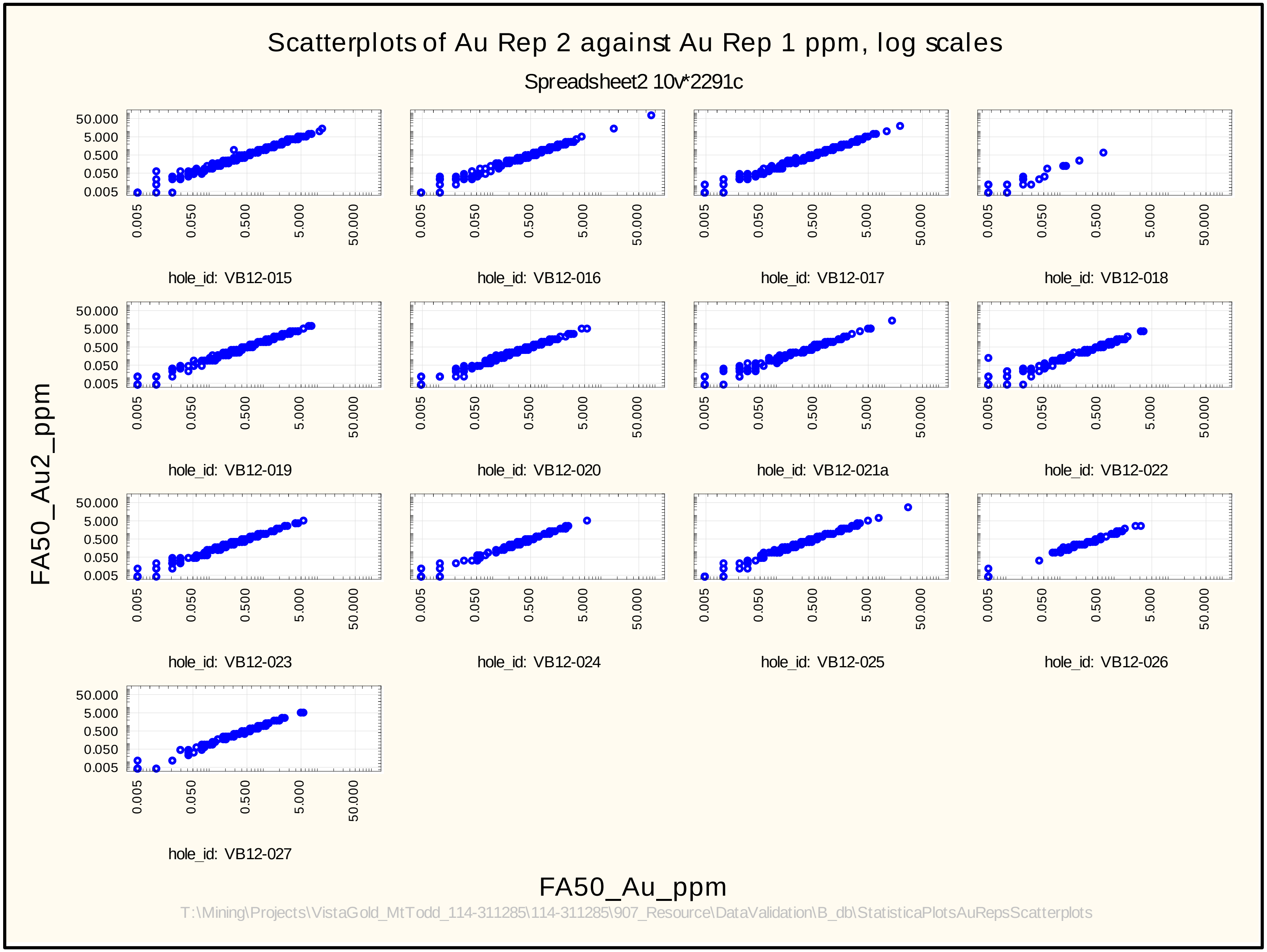
Because the current drilling campaign uses core, a regular program of field duplicates is not instituted at this time, but approximately 30% of samples have at least one replicate assay performed and an additional 3% of these have a second replicate assay. Replicates are taken from pulp when the primary sample is taken and run in the same analytical "batch." Variability is highest at concentrations near detection limit, but overall trends are very good for the drillholes. **Figure 12-5** shows the first replicate value against the primary value by drillhole. Equally good correlation is seen for the second replicates against the original and against the first replicate value.

The author is of the opinion that the current field QA/QC program and results meet industry standards and that the assay database adequately reflects values reported from the laboratory and is suitable for use in mineral resource and mineral reserve estimation.



Source: Tetra Tech, Inc (August 2017)

Figure 12-4: Scatterplot of Relative Au Value to Certified Standard Reference Material Value



Source: Tetra Tech, Inc (August 2017)

Figure 12-5: Scatterplots (Log Scale) of Replicates by Drillhole

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section reports on the work done to develop the understanding of the metallurgical characteristics of the remaining ore in the Batman deposit. This understanding contributes to the design of a technically effective and economically efficient gold recovery operation.

13.1 Summary

Key conclusions drawn from the metallurgy studies to date are:

- Mt Todd (and in particular the Batman deposit) ore is among the hardest and most competent ore types processed for mineral recovery. The most energy efficient comminution circuit has been determined to be the sequence of primary crushing, closed circuit secondary crushing, and closed circuit HPGR tertiary crushing and ore sorting, followed by two stages of grinding.
- The ore is free-milling, is not preg-robbing, and is amenable to gold extraction by conventional cyanidation processes.
- The ore has moderate to high cyanide consumption, determined to be 0.876 kg of sodium cyanide per tonne of ore. This is largely due to the presence of sulfides, cyanide-consuming copper, and not recycling cyanide from leach residue prior to cyanide destruction.
- The ore requires a P_{80} grind of 40 µm and 30-hour leach residence time to achieve a nominal 91.9% gold recovery net of solution loss from ore with a pit head grade of 0.84 g-Au/t.

13.2 Historic Metallurgical Test Programs

The Mt Todd deposit is a large low-grade gold deposit. The average grade of the gold mineralization is approximately 1 g-Au/t. The gold mineralization occurs in a hard, uniform greywacke host and is associated with sulfide and silica mineralization which has resulted from deposition along planes of weakness that had opened in the host rock. Gold is fine grained (<30 µm) and occurs with both silica and sulfides. The host rock is very competent with a Bond Ball Mill Work Index (BWi) of 23 to 30 kWh/t.

A substantial body of knowledge has been accumulated for the metallurgy of the Mt Todd ore, some from the historical operation of the mine, but more importantly, detailed information has been developed from recent sampling of the remaining ore body. Observations are as follows:

- 1988-1997 metallurgical studies by previous owners (Pegasus) led to the design and construction of a treatment plant comprised of crushing, milling to a P_{80} of 150 µm, sulfide flotation, concentrate regrind and cyanidation, and, separate CIL cyanidation of flotation tailings. Operational efficiencies were lower than planned due to ore hardness, presence of cyanide-soluble copper minerals, and inefficient flotation performance resulting from the presence of free cyanide in the process water (from recycled tailings decant water). One could reasonably state that these operational challenges were the result of inadequate design and equipment selection, in part due to an incomplete understanding of the deposit. These process difficulties together with the collapse of the gold price led to the cessation of operations in November of 1997.
- In 2006, Vista acquired the Project with the belief that each of these challenges could be overcome through the use of current technology, adequate metallurgical testing and higher gold prices. Vista's consultant, Resource Development Inc. (RDi), completed a study using

historical metallurgical data and test results from transition ore samples. RDi proposed a flowsheet consisting of crushing and grinding followed by rougher flotation to produce a sulfide concentrate containing 85% of the gold. Rougher tailings, substantially barren of gold and sulfides, would be discarded to a benign tailings dam. Rougher concentrate would be reground to enable upgrading in a cleaner flotation circuit to produce a saleable copper concentrate containing 50% of the gold. Cleaner tailings would be cyanide leached in a CIL circuit for gold recovery. The cleaner tailings would be subjected to cyanide destruction and stored in a separate sulfide tailings dam.

- The design incorporated energy efficient HPGR technology in the comminution circuit to handle the hard ore. These processing advantages combined with a higher gold price significantly improved the viability of the proposed operation. It then became necessary to confirm if the remaining ore had the same metallurgical characteristics as the historically processed ore.

In 2007/2008 two exploration drilling programs were completed focusing on the deeper ore beneath the existing Batman pit. The following composites/samples were prepared for RDi's testwork conducted on the samples of the deeper Batman ore from the 2007/2008 drilling program:

- Composite 1 – 1,200 kg composite sample made up from 2007 drill core. The composite consisted of samples from five drillholes selected to be representative of a cross section of the deposit. The head assay was 1.3 g-Au/t, 0.92% S and 447 ppm Cu. The sequential copper analysis indicated 80.4% of the copper in the sample was primary copper. The dominant sulfide in the sample was pyrrhotite.
- Composite 2 – 140 kg composite sample made up from 2008 drill core. The head assay was 0.89 g-Au/t and 450 ppm Cu. The sequential copper analysis indicated 80.3% of the copper in the sample was primary copper. The dominant sulfide in the sample was pyrrhotite.
- Drillhole 41 sample was sourced from the oxide and transitional zones (depth of 0–65 m). The head assay was 1.78 g-Au/t, 1.42% S, 448 ppm Cu.
- The new cores were more representative of the remaining resource and samples were selected for confirmatory metallurgical test work. It was confirmed that the ore was extremely hard but it was not possible to repeat the flotation results previously achieved. The tests indicated that gold recovery into the rougher flotation concentrate was ± 80% at a grind P_{80} of 74µm but copper could not be upgraded to saleable concentrate grade of ± 20% Cu. The best results were ± 6% Cu using the same test procedure as employed for earlier core testing (2006).
- Investigations revealed that the historical core tested in 2006 was transition zone material containing copper minerals predominantly as secondary copper which is known to be a major consumer of cyanide. The major sulfide mineral was pyrite. However, the 2007 and 2008 drill core had primary copper as predominant copper species and pyrrhotite as the major sulfide mineral. Pyrrhotite is known to float more readily as compared to pyrite and is significantly more difficult to depress in the flotation process. It was difficult to selectively float copper minerals and produce a copper concentrate without the dilutive effect of pyrrhotite and other gangue minerals. Consequently flotation was dropped from the flow sheet and replaced with whole ore leach.

In 2010/2011 a confirmatory drilling campaign and metallurgical test program was conducted on the remaining Batman resource. The objective was to validate the findings of the 2007/2008 programs and to expand the level of understanding of variability of metallurgical performance within the Batman ore body. Samples used for the 2011 metallurgical testwork program were sourced from eight drillholes drilled 2010/2011. The drillholes were orientated to intersect the main Batman ore body beneath the existing pit and are representative of the ore within the Technical Report pit shell.

All samples from drillholes labeled VB11 were drilled in 2011, logged, packaged then shipped directly to the laboratory for processing. Drillholes labeled MHT were drilled and logged during 2010 and were stored in cold storage before being transported to the laboratory in 2011.

- The test program was designed by Vista, supervised by Ausenco Limited (Ausenco), and executed by ALS Ammtec in Perth, Western Australia. There were a total of ninety-nine composited gold ore drill core intervals originating from the Project area. The metallurgical testwork included head analyses, crushing tests (HPGR and conventional crush), comminution testing, mineralogical analyses, leaching tests, cyanide detoxification and thickening and rheology testing. The test results confirmed that gold recovery by whole ore leach was the appropriate approach to process design.
- Vista had additional testwork undertaken in 2016 at RDi on the 2011 drilling samples. The test results indicate that the recovery was independent of the ore types but was somewhat dependent on the content of quartz in the ore. Also testing of the HPGR product indicated that the plus 5/8-in material had the potential to be treated by ore sorting to reject non-sulfide material. Since this was undertaken in small-scale tests, it provided incentive to undertake large scale tests to improve the process flowsheet and economics of gold production.

13.3 2017 Metallurgical Testwork

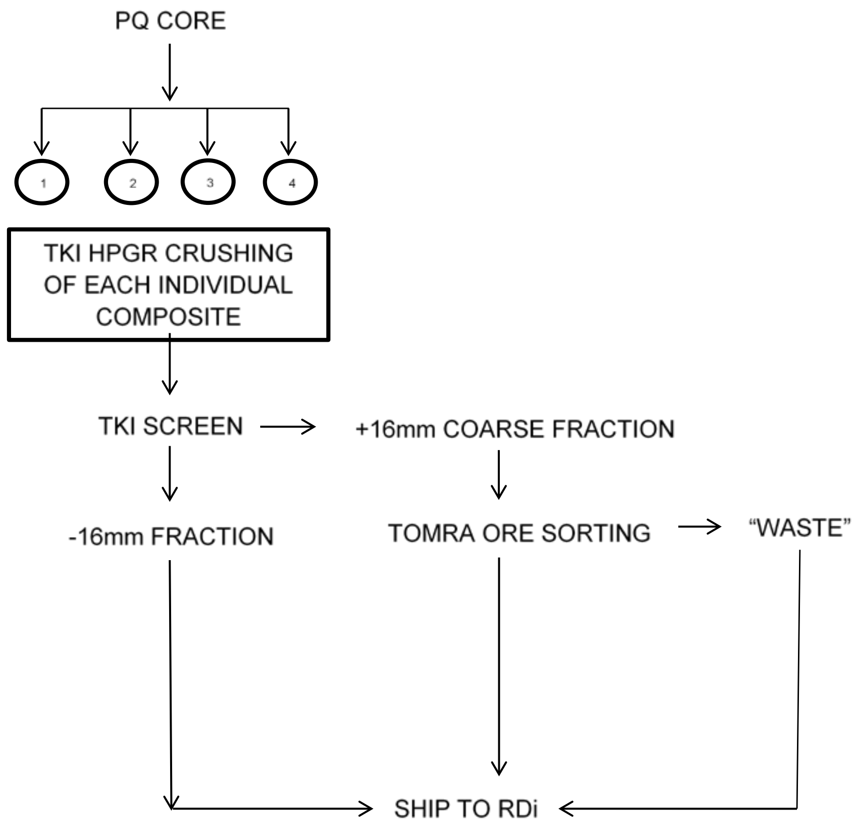
During January and February 2017 Vista completed drilling and logging of approximately 1,700 m of PQ (3.75 in diameter) core to obtain four 5-tonne bulk samples of ore representing different parts of the deposit. These composites were selected to represent both near-term and longer-term mining and are spatially located to provide variability both horizontally and vertically.

The primary objective of this phase of the test program was to perform sufficient metallurgical testwork to confirm the preferred process flowsheet developed during the last two years and associated reagent consumptions.

13.3.1 HPGR Testing at Thyssen-Krupp Industries (TKI)

The four composite samples were sent to TKI (formerly Polysius) in Germany for the HPGR crushing component of the test program. The material was crushed in a one meter diameter HPGR unit. The material was subjected to a single pass through the HPGR and then screened on 16mm (5/8 inch) and each composite had the coarse fraction weighed and placed into a drum. The fine fraction was weighed and placed into several drums. The coarse fractions were sent to Tomra Sorting Solutions/Outotec for ore sorting.

The test protocol is given in **Figure 13-1**. The weights of the plus and minus 16mm fractions for each composite are given in **Table 13-1**.



Source: Resource Development Inc, September 2019

Figure 13-1: Protocol for HPGR/Ore Sorting

Table 13-1: Material Balance for HPGR Tests

Composite No.	Sample Weight, Kgs	HPGR Products %	
		+16 mm	-16 mm
1.	4399.9	17.5	82.5
2.	4977.7	17.8	82.2
3.	4370.7	16.6	83.4
4.	4317.3	18.7	81.3

13.3.2 Tomra/Outotec Ore Sorting Testwork

Each plus 16mm sample was weighed at the Tomra sorting facility. Each composite was split into three parts. Each split sample was subjected to a two-step automated sorting test designed to separate the gold-bearing sulfide minerals and quartz veining from non-gold bearing waste material. The first step (XRT) sorts the material by measuring differences in density to target the gold-bearing sulfide material. Three different sensitivities (1%, 2% and 5%) were tested. The X-ray Transmission (XRT) material was then washed to remove the fines which could interfere with the laser ore sorting. The second step (laser) separates the gold-bearing, quartz-veining material.

The test results, summarized in **Table 13-2**, indicate the following:

- Open-circuit HPGR produced approximately 18% of the feed as a plus 16mm fraction.
- The ore sorting rejected approximately 10% of the run-of-mine feed as below cut-off grade material. Approximately 1.3% of the gold was rejected with the waste fraction.
- Removal of waste resulted in approximately 8% improvement in estimated mill feed grade (average life-of-mine grade of 0.91 g/t Au compared to 0.84 g/t Au reserve grade).

Table 13-2: Tomra Sorting Test Results

		Product (XRT + Laser + Wash)					Final Reject					Head Grade of +16mm To Sorting				
	Test	Wt	Au (g/mt)	Ag (g/mt)	Sulfur (%)	CN Soluble Cu (ppm)	Wt	Au (g/mt)	Ag (g/mt)	Sulfur (%)	CN Soluble Cu (ppm)	Wt	Au (g/mt)	Ag (g/mt)	Sulfur (%)	CN Soluble Cu (ppm)
Composite # 1																
XRT Sensitivity at 5%	1.1	190.2	0.817	0.7	1.09	45.6	125.5	0.103	0.2	0.24	10.0	315.7	0.533	0.6	0.89	30.65
	Distribution	60.2%	92.3%	83.7%	87.3%	87.0%	39.8%	7.7%	16.3%	12.7%	13.0%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 2%	2.1	101.5	0.541	1.0	1.12	38.9	118	0.110	0.4	0.23	11.2	219.53	0.309	0.9	0.83	23.21
	Distribution	46.2%	80.9%	68.8%	80.8%	74.1%	53.8%	19.1%	31.2%	19.2%	25.9%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 1%	3.1	71.4	0.758	2.2	0.94	42.4	124.5	0.086	0.2	0.27	12.5	195.9	0.331	1.7	0.80	22.94
	Distribution	36.4%	83.5%	86.0%	66.7%	65.4%	63.6%	16.5%	14.0%	33.3%	34.6%	100.0%	100.0%	100.0%	100.0%	100.0%
Composite # 2																
XRT Sensitivity at 5%	4.1	193.2	0.365	2.1	0.73	27.8	117.5	0.075	0.2	0.23	6.5	310.7	0.255	1.5	0.55	19.25
	Distribution	62.2%	88.9%	94.6%	83.9%	87.2%	37.8%	11.1%	5.4%	16.1%	12.8%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 2%	5.1	138.4	0.449	11.8	1.03	40.7	114.5	0.106	0.2	0.18	8.1	252.86	0.294	10.1	0.90	25.26
	Distribution	54.7%	83.6%	98.6%	87.3%	85.5%	45.3%	16.4%	1.4%	12.7%	14.5%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 1%	6.1	132.9	0.566	35.7	0.86	32.4	151.5	0.185	0.2	0.22	10.7	284.4	0.363	23.6	0.45	20.23
	Distribution	46.7%	72.9%	99.4%	77.4%	71.9%	53.3%	27.1%	0.6%	22.6%	28.1%	100.0%	100.0%	100.0%	100.0%	100.0%
Composite # 3																
XRT Sensitivity at 5%	7.1	110.3	0.255	1.0	0.51	64.1	94	0.072	0.4	0.12	23.2	204.3	0.171	0.9	0.41	43.12
	Distribution	54.0%	80.6%	75.0%	83.4%	75.2%	46.0%	19.4%	25.0%	16.6%	24.8%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 2%	8.1	106.4	0.570	3.6	0.64	94.7	139.5	0.233	0.4	0.13	36.7	245.87	0.379	2.7	0.62	59.23
	Distribution	43.3%	65.1%	87.2%	78.9%	64.8%	56.7%	34.9%	12.8%	21.1%	35.2%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 1%	9.1	86.2	0.282	13.5	0.58	96.7	153.5	0.055	0.2	0.11	33.6	239.7	0.136	10.6	0.69	54.55
	Distribution	36.0%	74.2%	97.4%	74.8%	60.5%	64.0%	25.8%	2.6%	25.2%	39.5%	100.0%	100.0%	100.0%	100.0%	100.0%

		Product (XRT + Laser + Wash)					Final Reject					Head Grade of +16mm To Sorting				
	Test	Wt	Au (g/mt)	Ag (g/mt)	Sulfur (%)	CN Soluble Cu (ppm)	Wt	Au (g/mt)	Ag (g/mt)	Sulfur (%)	CN Soluble Cu (ppm)	Wt	Au (g/mt)	Ag (g/mt)	Sulfur (%)	CN Soluble Cu (ppm)
Composite # 4																
XRT Sensitivity at 5%	10.1	148.0	0.901	1.4	0.99	43.0	98	0.192	0.4	0.23	18.9	246.0	0.619	1.3	0.88	32.67
	Distribution	60.2%	87.6%	83.8%	86.7%	77.0%	39.8%	12.4%	16.2%	13.3%	23.0%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 2%	11.1	127.2	0.933	2.3	1.32	46.4	136	0.127	0.4	0.21	13.8	263.17	0.516	2.1	1.18	28.88
	Distribution	48.3%	87.3%	84.3%	85.5%	75.2%	51.7%	12.7%	15.7%	14.5%	24.8%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at 1%	12.1	112.9	1.005	17.0	1.67	44.3	161.5	0.113	0.4	0.26	9.9	274.4	0.480	15.3	1.70	23.61
	Distribution	41.1%	86.1%	96.7%	81.8%	75.3%	58.9%	13.9%	3.3%	18.2%	24.7%	100.0%	100.0%	100.0%	100.0%	100.0%

13.3.3 Preparation of Composites for Metallurgical Testwork

The HPGR product (minus 16mm) and the ore sorting products were weighed for each composite. This was followed by blending of the minus 16mm product and splitting using a cone and quarter process to obtain quarter portions of the material for each composite. The ore sorting product was proportioned into the split samples to prepare the composite samples, representing the product of crushing with sorting as feed to leach processing.

The composite samples were stage crushed to nominal 6 mesh. However, the required samples were split out at 3/4 inch material for abrasion testing. The minus 6 mesh material was thoroughly blended and split into 1 kg and 10 kg charges, and approximately half the material was stored in drums.

13.3.4 Mineralogical Study

The four prepared composite samples were submitted for mineralogical study with emphasis on gold, silver, and speciation of pyrrhotite. Each sample was prepared as a standard polished thin section for study by transmitted/reflected light microscopy.

The highlights of the study indicate the following:

- The mineralogy of the four composites was very similar.
- Quartz was the primary phase in all samples and accounts for over 60% of the volume.
- Quartz occurs as very fine mosaic grains (5 to 10 µm) or as angular to rounded grains in sizes from 5 to 125 µm. Some very course fragments of quartz up to several millimeters were also present in all samples.
- The coarse quartz was commonly associated with coarse grain sulfides.
- Other silicate minerals identified in the samples were biotite, muscovite, chlorite and plagioclase feldspar.
- Sulfide minerals represented 2% to 3% in each composite. Pyrite was common in all samples and occurred as euhedral cubes and anhydral grains (3 to 300 µm).
- Pyrite concentration was highest in Composites 1 and 2. It was intermixed with marcasite and arsenopyrite.
- Arsenopyrite was most prominent in Composite 3 with a grain size of up to 100 µm.
- Other sulfide minerals present included chalcopyrite, sphalerite and galena.
- Pyrrhotite was identified in all four composites. It was determined to have monoclinic structure.
- Most of the gold grains identified were associated with pyrite and ranged in sizes from 3 to 28 µm.
- No discrete silver minerals were identified in any of the composite samples.

13.3.5 Head Analyses

The composite samples were submitted for head analysis. The test results are summarized in **Table 13-3**, **Table 13-4,** and **Table 13-5**. The results indicate the following:

- The samples assayed from 0.348 g/t Au to 0.760 g/t Au.
- The total sulfur content ranged from 0.43% to 1.26%.
- The copper valves ranged from 241 ppm to 467 ppm.
- The samples contained significantly lower gold values than projected from the drilling data as shown in **Table 13-5**.

Table 13-3: Head Analyses of Composite Samples

Element	Composite			
	1	2	3	4
Au, g/t	0.679	0.350	0.350	0.699
Assay 1				
Assay 2	0.672	0.346	-	0.713
Average	0.675	0.348	0.350	0.706
Ag, g/t	1.6	3.7	1.2	0.8
S_{Total}, %	1.26	0.67	0.43	0.76

Table 13-4: Whole Rock Analyses of Composite Samples

Element Percent	Composite			
	1	2	3	4
Al	7.33	7.65	7.44	6.97
Ca	0.33	0.32	0.17	0.37
Fe	5.48	5.02	5.44	4.97
K	3.59	3.63	3.03	3.06
Mg	1.14	1.23	1.26	1.16
Na	0.29	0.36	0.50	0.36
Ti	0.19	0.21	0.20	0.22
ppm				
As	50	103	403	113
Ba	579	622	574	548
Bi	<10	<10	<10	<10
Cd	8	9	7	7
Co	21	22	22	18
Cr	83	97	111	88
Cu	467	285	241	384
Mn	352	372	360	368
Mo	<175	<1	<1	<1

Element Percent	Composite			
	1	2	3	4
Ni	213	72	78	74
Pb	17	81	302	222
Sr	83	23	17	20
V	<10	86	100	92
W	575	11	<10	<10
Zn		240	392	421

Table 13-5: Assayed vs. Projected Head Analyses

	g/t Au	
	Assayed	Projected
Composite 1	0.675	1.54
Composite 2	0.348	0.99
Composite 3	0.350	0.74
Composite 4	0.706	0.56

13.3.6 Abrasion Indices

The samples were submitted for Bond abrasion index determination. The test results are summarized in **Table 13-6**. The test results indicate that the material is low to moderately abrasive.

Table 13-6: Abrasion Indices for the Various Composite Samples

Sample	A_i, g
Composite CC ¾ X ½ in	0.1603
Composite 1 – 16mm	0.2278
Composite 2 – 16mm	0.1616
Composite 3 – 16mm	0.2006
Composite 4 – 16mm	0.2250

13.3.7 Bond Ball Mill Work Indices

Bond ball mill work indices (BWi) were determined at a grind size of P_{80} of 100 mesh for the various products, namely HPGR, ore-sorting, composite samples and waste material. The results are summarized in **Table 13-7** and **Table 13-8**.

The test results indicate the following:

- The BWi for the plus 16mm sorted product was higher than the composite samples prepared from the crushed products. Hence, it is reasonable to conclude that the uncrushed material in the HPGR is harder than the crushed product.
- The rejected plus 16mm material has a BWi harder than the composite sample and harder than the plus 16mm sorted product.
- The BWi for the products ranged from 23.1 to 24.28. A BWi of 24.5 was selected for the design of the primary ball mill circuit.

Table 13-7: Bond Ball Mill Work Indices for Composite Samples

Composite	BW_i (kwh/mt)
1	23.10
2	24.41
3	23.79
4	24.48

Table 13-8: Bond Ball Mill Work Indices for Ore Sorting Products and Wastes

No.	Composite	Sample	BW_i (kwh/mt)	Average BW_i
1	1	1.1 XRT Product	23.0	
2	1	2.1 XRT Product	25.15	24.71
3	1	3.1 XRT Product	25.98	
4	2	4.1 XRT Product	26.55	
5	2	5.1 XRT Product	26.91	26.63
6	2	6.1 XRT Product	26.44	
7	3	7.1 XRT Product	24.54	
8	3	8.1 XRT Product	24.63	24.87
9	3	9.1 XRT Product	25.44	
10	4	10.1 XRT Product	25.37	
11	4	11.1 XRT Product	25.89	25.62
12	4	12.1 XRT Product	25.61	
13	2	4.2 Laser Waste	26.34	
14	4	10.2 Laser Waste	23.89	
15	Composite Sample (before HPGR		25.01	

13.3.8 Leach Tests

Several series of leach tests were performed to evaluate the effect of grind size, leach pulp density, cyanide concentration and two-stage grind on the gold extraction and reagent consumption.

The test procedure consisted of grinding the ore to the desired particle size in a single stage or two stages as would be done in the plant and the ground pulp was transferred to a bottle. The pulp density was adjusted to the desired level and then the pH was adjusted to 11 with hydrated lime. The slurry was pre-aerated for 4 hours with 50ppm lead nitrate. Sodium cyanide was then added to a calculated level of cyanide concentration. The pH and cyanide concentration were determined at 6 and 24 hours and a sample of solution was taken and assayed for gold and silver. Activated carbon was added at 24 hours at a level of 20g/L. After 30 hours, the solution was measured to determine pH, free cyanide, and gold and silver content. The carbon was screened and dried. The slurry was filtered, washed and dried. The products were prepared and assayed for gold and silver.

The test results are summarized in **Table 13-9** to **Table 13-13**. The test results indicate the following:

- The gold extraction is size dependent. The finer the grind size, the higher the gold extraction.
- The gold extraction for average grade composites 1 and 4 were 82.8% to 87.6% at a P_{80} of 46 µm in a single-stage grind. However, for two-stage grind to P_{80} of 53 µm, the gold extraction improved from 86.4% to 89.7%.
- The NaCN consumption in the two-stage grind tests was also lower by ± 20% as compared to single-stage grind.
- The preliminary optimization study indicated that the leach circuit could potentially operate at higher pulp density (± 50% solids) and lower cyanide concentration (750 ppm initial concentration) without impacting gold extraction.

Table 13-9: Gold Extraction vs. Grind Size for the Four Composites

Test No.	Composite	P_{80}, mesh	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
						NaCN	Lime
1	1	200	84.4	0.12	0.75	0.515	3.782
2	1	200	84.9	0.10	0.68	0.512	3.000
3	1	230	85.1	0.10	0.65	0.471	3.351
4	1	230	85.4	0.10	0.66	0.514	2.987
5	1	325	85.1	0.10	0.66	0.516	3.578
6	1	325	87.6	0.10	0.77	0.515	3.446
13	2	200	77.0	0.10	0.42	0.336	3.743
14	2	200	76.4	0.10	0.44	0.393	3.460
15	2	230	77.3	0.10	0.45	0.393	3.533
16	2	230	75.1	0.11	0.44	0.394	3.493
17	2	325	68.3	0.16	0.50	0.453	3.631
18	2	325	75.5	0.12	0.48	0.453	3.678
19	3	200	65.2	0.10	0.30	0.456	4.554
20	3	200	64.0	0.10	0.27	0.397	4.545
21	3	230	66.5	0.10	0.29	0.454	4.555

Test No.	Composite	P_{80}, mesh	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
						NaCN	Lime
22	3	230	69.8	0.09	0.29	0.396	4.678
23	3	325	69.8	0.08	0.27	0.454	4.700
24	3	325	70.0	0.08	0.27	0.454	4.632
7	4	200	80.0	0.13	0.65	0.551	3.237
8	4	200	79.7	0.14	0.71	0.516	2.992
9	4	230	81.8	0.14	0.75	0.576	2.980
10	4	230	82.9	0.12	0.72	0.513	3.008
11	4	325	82.8	0.12	0.72	0.575	3.458
12	4	325	84.1	0.10	0.66	0.576	2.939

NOTE: Lime Consumption was assumed to be the same as lime addition to the test.

Table 13-10: Gold Extraction at P_{80} of 270 mesh (53µm) with Two-stage Grind for the Four Composites

Test No.	Composite	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
					NaCN	Lime
25	1	86.6	0.09	0.67	0.393	4.972
26	1	86.2	0.09	0.67	0.336	4.866
27	2	85.8	0.06	0.44	0.398	4.446
28	2	85.2	0.07	0.44	0.458	4.529
29	3	80.1	0.06	0.31	0.514	4.773
30	3	80.5	0.06	0.32	0.513	4.930
31	4	86.1	0.10	0.69	0.392	4.521
32	4	86.4	0.09	0.68	0.397	4.501

NOTE: Lime Consumption was assumed to be the same as lime addition to the test.

Table 13-11: Effect of Pulp Density and NaCN Concentration on Gold Extraction for Composite No. 1 at P_{80} of 270 mesh (53µm) with Two-stage Grinding

Test No.	Composite	NaCN g/t	Pulp Density % Solids	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
							NaCN	Lime
62	1	1.0	40	87.8	0.08	0.65	0.399	3.010
63	1	1.0	40	88.8	0.08	0.67	0.399	3.003
64	1	1.0	45	89.1	0.07	0.66	0.273	3.008
65	1	1.0	45	88.7	0.07	0.64	0.271	3.011
66	1	0.75	45	87.5	0.08	0.63	0.270	3.028
67	1	0.75	45	88.4	0.07	0.62	0.221	3.024
68	1	0.5	45	88.8	0.07	0.64	0.210	3.007
69	1	0.5	45	88.4	0.08	0.65	0.212	3.007

Test No.	Composite	NaCN g/t	Pulp Density % Solids	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
							NaCN	Lime
70	1	1.0	50	89.5	0.07	0.66	0.305	3.021
71	1	1.0	50	89.7	0.07	0.63	0.344	3.015

NOTE: Lime Consumption was assumed to be the same as lime addition to the test.

Table 13-12: Effect of Pulp Density and NaCN Concentration on Gold Extraction for Composite No. 3 at P_{80} of 270 mesh (53µm) with Two-stage Grinding

Test No.	Composite	NaCN g/t	Pulp Density % Solids	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
							NaCN	Lime
82	3	1.0	40	84.7	0.04	0.25	0.460	3.011
83	3	1.0	40	84.9	0.04	0.25	0.272	3.011
84	3	1.0	45	84.7	0.04	0.25	0.271	3.010
85	3	1.0	45	84.8	0.04	0.25	0.372	3.010
86	3	0.75	45	83.2	0.04	0.24	0.269	3.017
87	3	0.75	45	86.3	0.03	0.25	0.322	3.010
88	3	0.50	45	83.8	0.04	0.25	0.211	3.011
89	3	0.50	45	84.4	0.04	0.24	0.211	3.016
90	3	1.0	50	85.0	0.04	0.25	0.347	3.011
91	1	1.0	50	84.9	0.04	0.25	0.346	3.011

NOTE: Lime Consumption was assumed to be the same as lime addition to the test.

Table 13-13: Effect of Pulp Density and NaCN Concentration on Gold Extraction for Composite No. 4 at P_{80} of 270 mesh (53µm) with Two-stage Grinding

Test No.	Composite	NaCN g/t	Pulp Density % Solids	Extraction % Au (30 hrs.)	Residue g/t Au	Cal. Head g/t Au	Consumption Kg/t	
							NaCN	Lime
72	4	1.0	40	86.7	0.08	0.62	0.337	3.014
73	4	1.0	40	86.8	0.08	0.62	0.275	3.012
74	4	1.0	45	85.9	0.09	0.61	0.315	3.024
75	4	1.0	45	86.8	0.08	0.62	0.270	3.017
76	4	0.75	45	86.4	0.08	0.60	0.222	3.013
77	4	0.75	45	86.0	0.09	0.62	0.270	3.018
78	4	0.50	45	86.5	0.09	0.64	0.210	3.015
79	4	0.50	45	86.1	0.09	0.62	0.210	3.022
80	4	1.0	50	88.4	0.09	0.63	0.264	3.014
81	4	1.0	50	86.0	0.09	0.64	0.263	3.023

NOTE: Lime Consumption was assumed to be the same as lime addition to the test.

13.3.9 Cyanide Destruction

The cyanide leach residue for composites No. 1 and No. 4 were subjected to cyanide destruction tests using the air/SO_2 method. Approximately 1.5 liters of leach residue at 50% solids was agitated with sodium meta-bi-sulfite (SMBS) three times the stoichiometric amount of free cyanide and copper sulfate. Samples were taken every hour and free cyanide determined. Though no free cyanide was detected after one hour, the test was run for four hours.

The cyanide specification before and after destruction for the two tests are given in **Table 13-14**. The test results indicate the following:

- The air-SO_2 process successfully reduced CN_{WAD} to levels of <10 ppm.
- There is sufficient dissolved copper in solution for precipitation of copper iron cyanide compounds in the earlier years of operation. Hence, addition of copper sulfate may not be needed.
- One hour of detox residence time is sufficient for the process.

Table 13-14: Cyanide Destruction Test Results

Forms of Cyanide ppm	Composite 1		Composite 4	
	Before	After	Before	After
Free	600	6.3	590	4.0
Total	587	3.6	615	2.2
WAD	590	5.0	560	2.6

13.3.10 Thickening Tests

Thickening tests on leach residue having a grind size of P_{80} of 53 µm generated in two-stages of grinding were performed for the four composites. The test results, given in **Table 13-15**, indicate the following:

- Approximately 8 g/t of high molecular weight low anionic acrylamide/sodium acrylate flocculant will be required for the settling of the slurry.
- Unit area required to settle the slurry to 45% solids ranges from 0.044 to 0.182 m^2/mt/day.
- The unit area requirements increase significantly if the desired underflow solids is 50%.

Table 13-15: Unit Area Requirements for Thickener for Composite Samples

Composite	P_{80}, µm	pH	Flocculent	Feed % Solids	Unit Area Required m^2/mt/day			
					40%	45%	50%	55%
1	53	11	8 g/t DAF-10	25	0.031	0.044	0.164	2.41
2	53	11	8 g/t DAF-10	25	0.050	0.069	0.150	2.448
3	53	11	8 g/t DAF-10	25	0.042	0.081	0.191	2.436
4	53	11	8 g/t DAF-10	25	0.083	0.182	0.650	2.425

13.4 2018/2019 Metallurgical Test Work

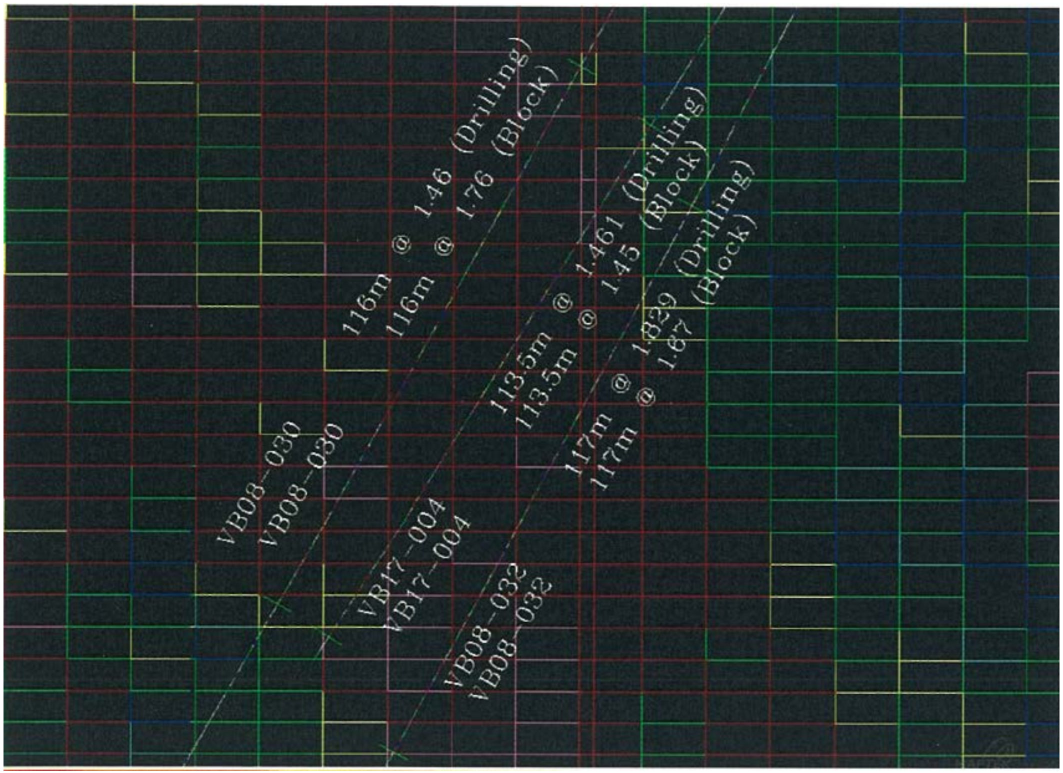
The gold grades of the initial composites tested in 2017 metallurgical program were lower than the projected grades for the samples based on the grades being projected from the 3D resource model. Vista engaged in a detailed review to determine why the grade difference existed and found that by drilling the core zone at an oblique angle too few veins were intersected to provide a representative sample and, therefore, provided a biased result. The following table presents the average vein intercept angles for each of the drill holes completed.

BHID	Average intercept angle
VB17-001	10 degrees
VB17-003	20 degrees
VB17-002	40 degrees
VB16-002	40 degrees

Vista initiated an additional drilling program to address two specific questions, namely, whether the geological model is correct or not and how would higher grade material perform in the proposed process flowsheet.

The drilling program was initiated in December 2017 and completed in January 2018. The 2017/2018 PQ metallurgical drill holes VB17-004, VB18-001, -002 and-003 were drilled approximately perpendicular to the mineralized host orientation and targeted similar locations to the 2016/2017 metallurgical samples. In addition, in order to test the accuracy of the resource model, the drill holes were drilled between known resource model drill holes. The following table details the results of this drilling as compared to the existing drilling that was on either side of the new metallurgical drill hole. The figure illustrates the relationship of the resource model estimated grades nearest existing drill hole intercept grades and the grades of the 2017/18 drill holes for one of these drill holes.

DH	Drill Hole ID	HG Core Length (m)	Composite (g Au/t)	Block Model (g Ault)
Existing	VB08-030	116	1.46	1.76
New Met	VB17-004	113.5	1.461	1.45
Existing	VB08-032	117	1.829	1.67
Existing	VB07-001	126	1.879	1.44
New Met	VB18-001	132	1.13	1.52
Existing	VB08-028	129	1.739	1.59
Existing	VB07-018	111	1.935	1.58
New Met	VB18-002	110.7	1.499	1.56
New Met	VB18-003	141	1.1	1.13
Existing	VB07 -018	135	1.72	1.55
	Total/Avg	1,231.20	1.57	1.52



A quarter split of the PQ core was assayed generally in one-meter lengths per the approved assay procedure. Based on the assay results, the following composites were prepared targeting the grade ranges that Vista desired for test work:

- 2.5 metric tons of composite sample designated "Big Yellow" and assaying 1.7 g/t Au.
- 2.5 metric tons of composite sample designated "Big Blue" and assaying 1.4 g/t Au.
- 1.0 metric ton of composites sample designated "Weir" and assaying 0.99 g/t Au.
- 40 kgs each of composite samples designated "small yellow", "small blue" and "small red" assaying 1.27 g/t Au, 0.84 g/t Au and 1.02 g/t Au, respectively.

The Big Yellow and Big Blue composites were subjected to HPGR crushing and ore sorting whereas the Weir composite was subjected to only HPGR crushing. All the products from the HPGR and ore sorting tests were shipped to RDi for subsequent metallurgical test work. The remaining three samples were shipped to RDi and were not subjected to HPGR crushing or sorting.

The samples from 2017 drilling, namely Composites 1 to 4, were also utilized in the 2018/2019 metallurgical test program.

13.4.1 HPGR Testing at Thyssen-Krupp Industries (TKI)

The two 2.5 mt composite samples, Big Yellow and Big Blue, were sent to TKI in Germany for the HPGR crushing component of the test program. The test program was identical to that performed in 2017 and produced similar results. The samples were jaw crushed followed by HPGR. The material balance is given in **Table 13-16**. The specific throughput rate was ±300 ts/hm^3.

Table 13-16: Material Balance for HPGR Tests at TKI

Composite	Sample Weight, kg	HPGR Products, %	
		+16mm	-16 mm
Biq Yellow	2400	18.6	81.4
Biq Blue	2370	17.8	82.2

13.4.2 HPGR Testing at WEIR Minerals

Approximately 1 mt of drill core was also sent to WEIR minerals for evaluating the WEIR Enduron HPGR for Mt. Todd ore. The drill core was pre-crushed with a jaw crusher and fed to HPGR in three batches and screened at 16 mm. The three HPGR runs delivered consistent and repeatable results. The specific energy showed little variation around the average of 1.94 kwh/t and the average specific throughput was 254 ts/hm^3. The average mass oversize at 16 mm screen was 17.3%. The results were similar to the HPGR testing at TKI.

13.4.3 Tomra/Outotec Ore Sorting Test Work

The plus 16 mm screened samples from TKI were sent to Tomra for ore sorting test work. The sorting tests were completed on the same XRT and laser equipment as the tests completed in 2017 (Section 13.3.2).

The test results are given in Table 13.17. The test results indicate the following:

- The calculated head analyses of the plus 16 mm fraction for both composites were almost identical (0.731 g/t Au and 0.737 g/t Au). This has been determined to be due to the "softer" vein material preferentially crushing into finer material leaving the same approximate grades for the material with vein selvages on them going to sorting.
- The final rejection fraction was 54.5% for blue composite and 47.2% for yellow composite.
- Based on the assays of the various products, ore sorting rejected 8.7% and 7.9% of the feed for Big Yellow and Big Blue samples, respectively. The corresponding rejection of gold in the waste material was 0.9% and 0.7%. The gold loss was lower than 1.3% which was achieved in the 2017 test program

Table 13-17: Tomra Ore Sorting Test Results

				XRT Cut								Laser Cut							
	Test	Units	Total Mass	Wt	Au (g/mt)	Ag (g/mt)	Cu (mg/kg)	CN Soluble Au (ppm)	CN Soluble Ag (ppm)	CN Soluble Cu (ppm)	Sulfur (%)	Wt	Au (g/mt)	Ag (g/mt)	Cu (mg/kg)	CN Soluble Au (ppm)	CN Soluble Ag (ppm)	CN Soluble Cu (ppm)	Sulfur (%)
Blue Composite																			
XRT Sensitivity at X%	1.1	kg	193.6	98	1.262	0.6	716	0.78	0.28	253	1.49	4.6	0.734	0.6	504	0.48	0.28	179	0.48
		%	100%	50.6%	87.4%	60.0%	77.1%	90.9%	84.8%	78.8%	80.7%	2.4%	2.4%	2.8%	2.5%	2.6%	4.0%	2.6%	1.2%
XRT Sensitivity at X%	2.1	kg	167.1	56	2.599	1.0	794.0	1.8	0.5	307 .0	1.72	5.6	1.454	0.8	636	1.2	0.38	286	0.71
		%	100%	33.5%	78.1%	54.5%	62.4%	78.6%	74.1%	64.7%	67.6%	3.4%	4.4%	4.4%	5.0%	5.2%	5.2%	6.0%	2.8%
Blue Comp Total kg		**kg**	**361**	**154**								**10**							
		%	**100%**	**42.7%**								**2.80%**							
Yellow Composite																			
XRT Sensitivity at X%	3.1	kg	249.6	132.5	1.255	0.8	540	0.92	0.44	236	1.5	6.1	0.898	1	586	0.84	0.52	299	0.63
		%	100%	53.1%	90.4%	59.3%	71.9%	91.2%	77.9%	73.5%	80.8%	2.4%	3.0%	3.4%	3.6%	3.8%	4.2%	4.3%	1.6%
XRT Sensitivity at X%	4.1	kg	161.1	73.5	0.905	0.8	664	0.8	0.5	312	2.02	4.6	2.257	1.4	672	2.12	0.84	404	0.65
		%	100%	45.6%	52.4%	51.1%	71.8%	55.1%	57.1%	72.3%	80.8%	2.9%	8.2%	5.6%	4.5%	9.1%	6.5%	5.9%	1.6%
Yellow Comp Total kg		**kg**	**411**	**206**								**11**							
		%	100%	**50.2%**								**2.60%**							

				Final Rejects								Sum - Head Grade							
	Test	Units	Total Mass	Wt	Au (g/mt)	Ag (g/mt)	Cu (mg/kg)	CN Soluble Au (ppm)	CN Soluble Ag (ppm)	CN Soluble Cu (ppm)	Sulfur (%)	Wt	Au (g/mt)	Ag (g/mt)	Cu (mg/kg)	CN Soluble Au (ppm)	CN Soluble Ag (ppm)	CN Soluble Cu (ppm)	Sulfur (%)
Blue Composite																			
XRT Sensitivity at X%	1.1	kg	193.6	91	0.158	0.4	204	0.06	0.04	64.1	0.36	193.6	0.731	0.5	470.3	0.4	0.2	162.45	0.93
		%	100%	47.0%	10.2%	37.2%	20.4%	6.5%	11.2%	18.5%	18.1%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at X%	2.1	kg	167.1	105.5	0.309	0.4	220	0.2	0.08	73.6	0.4	167.1	1.115	0.6	426.3	0.8	0.2	158.94	0.85
		%	100%	63.1%	17.5%	41.1%	32.6%	16.3%	20.7%	29.2%	29.6%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
Blue Comp Total kg		**kg**	**361**	**197**								**361**							
		%	**100%**	**54.5%**								**100.0%**							
Yellow Composite																			
XRT Sensitivity at X%	3.1	kg	249.6	111	0.11	0.6	220	0.06	0.12	85.2	0.39	249.6	0.737	0.7	398.8	0.5	0.3	170.48	0.99
		%	100%	44.5%	6.6%	37.3%	24.5%	5.0%	17.8%	22.2%	17.6%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
XRT Sensitivity at X%	4.1	kg	161.1	83	0.604	0.6	194	0.46	0.26	83.3	0.39	161.1	0.789	0.7	422.1	0.7	0.4	196.8	1.14
		%	100%	51.5%	39.5%	43.3%	23.7%	35.8%	36.4%	21.8%	17.6%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
Yellow Comp Total kg		**kg**	**411**	**194**								**411**							
		%	**100%**	**47.2%**								**100.0%**							

13.4.4 Steinert Ore Sorting Test Work

RDi recombined the ore sorting samples from 2017 study discussed in Section 13.3.2 for evaluation at Steinert. Three samples (Composite1, 3 and 4) were sent to Steinert in Walton, Ky with the objective of evaluating the STEINERT combined sensor sorter (KSS FLI XT) for separating ore and waste. The test results, summarized in **Table 13-18**, were similar to those obtained at Tomra test facility in 2017.

Table 13-18: Steinert Sorting Results for Composites 1, 3 and 4

Sample	Wt (kg)	Individual Wt%	Cumulative Wt%	Au Assay (g/mt)	Individual Au Distribution %	Cumulative Au Distribution %	Ag Assay (g/mt)	Individual Ag Distribution %	Cumulative Ag Distribution %	CN Soluble Cu Assay (ppm)	Individual CN Cu Distribution %	Cumulative CNCu Distribution %	S Assay (%)	Individual S Distribution %	Cumulative S Distribution %
Composite # 1															
Product 1.1	3.8	3.2	3.2	3.711	45.7	45.7	2.0	12.9	12.9	428	14. 7	14 .7	6.25	32.2	32.2
Product 2.1	4.5	3.7	6.9	0.823	11.9	57.5	1.0	7.6	20.5	277	11.1	25.8	1.71	10.3	42.6
Product 3.1	11.1	9.3	16.2	0.322	11.5	69.1	0.6	11.3	31.8	141	14.0	39.8	0.70	10.5	53.1
Product 4.1	23.0	19.1	35.2	0.151	11.1	80.2	0.4	15.5	47.3	73.0	15.0	54.8	0.49	15.2	68.2
Product 5.1	32.0	26.6	61.9	0.075	7.7	87 .9	0.4	21.7	69.0	61.4	17.6	72.4	0.32	13.8	82.1
Waste 5.2	45.9	38.1	100.0	0.082	12.1	100.0	0.4	31.0	100.0	67.2	27.6	100.0	0.29	17.9	100.0
Total	**120.4**	**100.0**		**0.259**	**100.0**		**0.5**	**100.0**		**92.9**	**100.0**		**0.62**	**100.0**	
Composite # 3															
Pro duct 1.1	2.1	1.7	1.7	3.999	51.5	51.5	2.2	7.6	7.6	468	10 .9	10.9	5.54	28.7	28.7
Product 2.1	2.8	2.3	4.1	0.912	16.0	67.5	1.4	6.6	14.2	220	7.0	17.9	1.39	9.8	38.6
Product 3.1	8.6	7.2	11.3	0.185	10.0	77.5	1.0	14.5	28.7	129	12.6	30.S	0.68	14.8	53.3
Product 4.1	20.2	16.9	28.2	0.034	4.3	81.8	0.4	13.6	42.3	50.2	11.5	41.9	0.14	7.1	60.5
Product 5.1	30.7	25.6	53.8	0.034	6.5	88.3	0.4	20.6	62.9	61.6	21.4	63.3	0.17	13.1	73.6
Waste 5.2	55.2	46.2	100.0	0.034	11.7	100.0	0.4	37.1	100.0	58.8	36.7	100.0	0.19	26.4	100.0
Total	**119.6**	**100.0**		**0.134**	**100.0**		**0.5**	**100.0**		**74.0**	**100.0**		**0.33**	**100.0**	
Composite # 4															
Product 1.1	4.1	3.2	3.2	3.992	35.4	35.4	2.6	16.3	16.3	589	19.2	19.2	5.89	35.8	35.8
Product 2.1	5.1	4.1	7.3	0.85 7	9.5	45.0	1.0	7.9	24.1	306	12.5	31.8	1.20	9.1	44.9
Product 3.1	13.2	10.4	17.7	0.487	13.9	58.9	0.6	12.1	36.2	121	12.7	44.5	0.55	10.7	55.7
Product 4.1	23.9	18.8	36.5	0.322	16.6	75.5	0.4	14.6	50.8	56.8	10.8	55.3	0.31	11.0	66.6
Product 5.1	34.3	27.1	63.6	0.062	4.6	80.1	0.4	21.0	71.8	70.4	19.3	74.6	0.32	16.3	82.9
Waste 5.2	46.1	36.4	100.0	0.199	19.9	100.0	0.4	28.2	100.0	69.2	25.4	100.0	0.25	17.1	100.0
Total	**126. 7**	**100 .0**		**0.364**	**100.0**		**0.5**	**100.0**		**99.0**	**100.0**		**0.53**	**100.0**	

13.4.5 Preparation of Composites for Metallurgical Test Work and Head Analyses

The samples from HPGR and ore-sorting test work were prepared using the same protocol as used in 2017 study and discussed in Section 13.3.3.

All the samples were submitted for head analyses. The test results, summarized in **Table 13-19**, indicate the following:

- Head analyses of some of the composite were close to expected values whereas for other samples, the assays were significantly different.
- The assayed values covered a range from 0.5 g Au/t to 2.95 g Au/t.

Table 13-19: Head Analyses of Composite Samples

Sample	Expected Head Grade, g/tAu	Multiple Head Grade Analyses, g/t
Big Blue	1.39	0.91, 1.31
Biq Yellow	1.70	0.83, 1.68
Weir	1.00	1.05
Small Blue	0.84	2.60, 2.62, 2.95
Small Yellow	1.27	1.48, 0.67, 0.72
Small Red	1.02	0.44, 0.51, 0.65

13.4.6 Bond's Ball Mill Work Indices

A Bond's ball mill work index (BWi) was determined at a grind size of P_{80} of 100 mesh for each of the three large samples (Big Yellow, Big Blue and Weir). The ore sorting waste was removed from the Big Yellow and Big Blue samples. The results are summarized in **Table 13-20**. The test result indicates the following:

- The BWi's for Big Yellow and Big Blue samples following the rejection of ore sorting waste were lower than Weir sample which represented the run-of-mine ore.
- The average BWi of the two composites (Big Yellow and Big Blue) was 24.3 which is similar to the valve selected for mill design.

Table 13-20: Bond's Ball Mill Work Indices for Composite Samples

Composite	BWi (kwh/mt)
Big Yellow	25.08
Biq Blue	23.41
Weir	25.81

13.4.7 Primary Grind

Earlier studies had indicated that the selected circuit would require three of the largest-size manufactured ball mills to achieve a targeted grind of P_{80} of 90 microns.

The concept of two stage grinding was developed with the idea of using the HPGR crushers to generate a smaller product size. This allowed the three large ball mills to be replaced by two smaller ball mills for the first stage of grinding and to produce a product with a P_{80} of 250 microns. This first stage of grinding could then be followed by removal of finished product and regrinding the coarse material to the desired product size in a stirred media mill.

The primary grind size in the present study remained the same as the 2017 study (P_{80} of 250 microns).

13.4.8 Fine Grind

The 2017 study confirmed that gold extraction was size dependent, as also observed in historic metallurgical work. The finer the grind size, the higher the gold extraction.

Fine grind testing had been initiated to evaluate ISA mills and FLS VXP mills for the January 2018 Technical Report. However, since the results of the test work was not available until March 2018, ISA mills were selected for the PFS study.

The test results for Composites 1 to 4 indicated that FLS VXP mills used significantly less energy (±15 kwh/t) to achieve P_{80} of 60 microns as compared to ISA mills that require ±28 kwh/t.

Several additional studies were undertaken at FLS facilities for VXP testing and Core labs in Australia and SGS Canada for ISA mill testing. The targeted grind size was reduced to 40 microns in 2019 study.

The following conclusions were drawn from the fine grind studies at the above-mentioned laboratories and RDi:

- The Malvern particle size analyzer did not provide an accurate analysis of the particle size distribution for the ground products. Hence, additional testing was undertaken on both machines, and products were screened in order to obtain accurate energy requirements and product for cyanide leach testing.
- FLS estimated specific energy requirements between 16.7 and 17.4 kwh/t to achieve P_{80} of 40 microns.
- SGS signature plots for the same samples tested at FLS facility indicated specific energy requirements between 26 and 34 kwh/t.

The specific energy requirements for VXP mill are significantly lower because the mill is vertical and the flow of material upward through the mill results in the finer material being carried up and out of the mill more quickly, while the coarser particles remain subject to additional grinding. In contrast, the IsaMill is a horizontal mill and the flow of material is more homogeneous and of a more fixed duration. This helps explain the IsaMill being more commonly used to produce a finer product than Vista is targeting.

Due to the significantly lower power requirement, the ISA Mills were replaced with FLS VXP mills in the present study.

13.4.9 Leach Feed Thickener

Since the leach feed size was changed from P_{80} of 60 microns to 40 microns, additional thickening tests were undertaken at Pocock Industrial and RDi. Based on the test results, the thickener size was changed from 45 meter diameter to 67 meter diameter in the process flowsheet.

13.4.10 Leach Agitator Design and Power Requirements

SPX Flow Lightnin performed test work on the ground slurry to determine full scale sizing for the leach conditioning and leach tank agitators in April 2018 Their recommendations were incorporated into the process flowsheet.

13.4.11 Leach Tests

Several series of leach tests were performed with the six samples in the present study. The test protocol was the same as used in the 2018 Technical Report (Section 13.3.8).

The primary objective of the leach tests was to evaluate the effect of feed grade on gold extraction at grind sizes of P_{80} of 53 microns and finer. The feed gold grades were divided into the following ranges:

- Greater than 1.5 g/t Au
- 1.0 to 1.5 g/t Au
- 0.8 to 1.0 g/t Au
- 0.6 to 0.8 g/t Au
- 0.4 to 0.6 g/t Au
- Less than 0.4 g/t Au

The test results for 71 leach tests are summarized in **Table 13-21** to **Table 13-26**. The test results indicate the following:

- Gold extraction of over 90% was obtained for feed grades of 0.6 g/t Au or higher.
- The higher the feed grade, the higher the gold extraction.

Table 13-21: Leach Results for Feed Grade >1.5 g/t Au

Test#	P_{80} Particle Size (µm)	% Recovery (Au)	Calc. Head Grade (g Au/t)	Residue Grade (g Au/t)
+1.5g Au/t				
BR113	101	86.1	1.77	0.25
BR114	101	85.4	1.77	0.26
BR119	91	87.6	1.82	0.23
BR120	91	88.9	1.74	0.19
BR117	76	87.3	1.74	0.22
BR118	76	87.0	1.70	0.22
BR116	74	87.0	1.70	0.22
BR115	74	86.4	1.67	0.23
BR153(1)	**53**	**93.6**	**1.96**	**0.12**
BR154(1)	**53**	**93.6**	**1.90**	**0.12**

Test#	P_{80} Particle Size (µm)	% Recovery (Au)	Calc. Head Grade (g Au/t)	Residue Grade (g Au/t)
BR196	**31**	**90.3**	**1.73**	**0.17**
BR195	**31**	**90.4**	**1.69**	**0.16**
BR204	**22**	**93.1**	**1.70**	**0.12**
BR205	**22**	**93.0**	**1.63**	**0.11**
BR201	**19**	**91.8**	**1.56**	**0.13**
<53 micron average values		**92.3**		**0.13**

Table 13-22: Leach Results for Feed Grade of 1.0 to 1.5 g/t Au

Test#	P_{80} Particle Size (µm)	% Recovery (Au)	Calc. Head Grade (g Au/t)	Residue Grade (g Au/t)
>=1.0g Au/t < 1.5g Au/t				
BR122	97	84.6	1.24	0.19
BR121	97	86.6	1.20	0.16
BR123	74	89.1	1.26	0.14
BR124	74	87.5	1.21	0.15
BR144	59	84.9	1.21	0.18
BR143	59	84.8	1.17	0.18
BR197	**29**	**90.4**	**1.21**	**0.12**
BR198	**29**	**90.1**	**1.16**	**0.11**
BR206	**20**	**92.7**	**1.10**	**0.08**
BR207	**20**	**92.7**	**1.09**	**0.08**
<53 micron average values		**91.5**		**0.10**

Table 13-23: Leach Results for Feed Grade of0.8 to 1.0 g/t Au

Test#	Particle Size (Pso µm)	% Recovery (Au)	Calc. Head Grade (g Au /t)	Residue Grade (g Au /t)
>=0.8g Au/t < 1.0g Au/t				
BR126	87	85.5	0.88	0.13
BR125	87	86.5	0.87	0.12
BR128	79	88.4	0.89	0.10
BR127	79	87.4	0.86	0.11
BR147	69	85.5	0.95	0.14
BR148	69	85.0	0.91	0.14
BR130	69	86.9	0.86	0.11
BR129	69	89.1	0.83	0.09
BR158	59	87.4	0.93	0.12

Test#	Particle Size (Pso µm)	% Recovery (Au)	Calc. Head Grade (g Au /t)	Residue Grade (g Au /t)
BR199	35	89.5	0.9	0.09
BR200	35	89.6	0.85	0.09
BR209	22	91.8	0.88	0.07
BR208	22	91.9	0.84	0.07
<53 micron average values		90.7		0.08

Table 13-24: Leach Results for Feed Grade of 0.6 to 0.8 g/t Au

Test#	Particle Size ($P_{80\ \mu m}$)	% Recovery (Au)	Calc. Head Grade (g Au /t)	Residue Grade (g Au /t)
>=0.6g Au/t < 0.8g Au/t				
BR104	70	85.3	0.63	0.09
BR105	70	84.9	0.61	0.09
BR106	70	84.1	0.61	0.10
BR157	59	88.S	0.77	0.09
BR162(1)	52	92.3	0.73	0.06
BR161(1)	52	91.4	0.72	0.06
BR96	49	89.9	0.68	0.07
BR95	49	89.6	0.66	0.07
BR97	49	89.5	0.66	0.07
BR101	39	90.5	0.65	0.06
BR102	39	90.9	0.64	0.06
BR103	39	90.4	0.64	0.06
BRlO0	36	92.1	0.79	0.06
BR98	36	88.3	0.70	0.08
BR99	35	89.7	0.73	0.08
BR109	18	94.0	0.69	0.04
BR107	18	89.4	0.68	0.07
BR108	18	93.8	0.66	0.04
BRlll	15	91.0	0.61	0.06
BRll0	15	92.0	0.60	0.05
BR112	15	90.9	0.60	0.06
<53 micron average values		90.9		0.06

Table 13-25: Leach Results for Feed Grade of 0.4 to 0.6 *g/t* Au

Test#	P_{80} Particle Size (µm)	% Recovery (Au)	Calc. Head Grade (g Au /t)	Residue Grade (g Au /t)
>=0.4g Au/t < 0.6g Au/t				
BR131	59	84 .8	0.46	0.07
BR132	59	86 .2	0.46	0.06
BR165	56	83.6	0.52	0 .08
BR166	56	85 .0	0.52	0.08
BR210	**22**	**88.S**	**0.42**	**0.05**
BR211	**22**	**89.0**	**0.41**	**0.05**
<53 micron average values		**88.8**		**0.05**

Table 13-26: Leach Results for Feed Grade of <0.4 *g/t* Au

Test#	P_{80} Particle Size (µm)	% Recovery (Au)	Calc. Head Grade (g Au /t)	Residue Grade (g Au /t)
< **0.4g Au/t (below cutoff)**				
BR167	60	81.6	0.18	0.03
BR212	60	80.6	0.18	0.03
BR213	**49**	**85.8**	**0.32**	**0.05**
BR168	**49**	**78.S**	**0.21**	**0.04**
BR133	**21**	**87.S**	**0.26**	**0.03**
BR134	**21**	**86.9**	**0.26**	**0.03**
<53 micron average values		**84.7**		**0.04**

- The average gold extraction, irrespective of the feed grade, at P_{80} of 53 microns or fine was 90.4% on a non-weighted average basis. The actual final recovery was determined on a weighted average basis.
- The cyanide consumption for all tests with particle size of 59 microns or finer averaged 0.636 kg/t (47 tests). Assuming a residual cyanide of 200 ppm and leach tests at 45% solids, the total cyanide consumption would be 0.876 kg/t. This assumes no cyanide recycle in the process.
- The average lime consumption in the 47 leach tests was 4.64 kg/t. Assuming that once the tailing pond stabilizes, the lime consumption will only be 60% of the consumption with tap water. Hence, the lime consumption is reduced to 2.8 kg/t after 3 months of operation.
- The fine grind products received from FLS and Core Laboratories that did not meet the targeted size were reground in ball mill with steel media at RDi. The cyanide consumption for samples ground with steel media was significantly higher than those ground with ceramic media. Hence, ceramic media is recommended for regrind mills in the flowsheet.

- The average leach residue assay for the different range of ore grades is given in **Table 13-27**. This data can be used by the process engineer to predict gold extraction in the plant.

Table 13-27: Leach Residue Assay Versus Ore Feed Grade

Ore, g Au/t	Leach Residue, g Au/t
>1.5	0.13
1.0-1.5	0.10
0.8-1.0	0.08
0,6-0.8	0.06
0.4-0.6	0.05
<0.4	0.04

13.4.12 Thickening Tests on Leach Residue

Thickening tests were performed at Pocock Industrial Inc. on leach residue having a P_{80} of 53 microns and 37 microns. The test results indicated that the maximum underflow density of 55% could be achieved but would require a significantly larger size thickener than determined in the previous study.

A trade-off study between savings in recycling cyanide and Capex required for larger thickener was undertaken. A decision was made not to have a thickener for densifying leach residue in the circuit.

13.4.13 Cyanide Destruction

The cyanide leach residue having a P_{80} of 45 micrometer and free cyanide of 200 ppm was subjected to cyanide destruction using the air/SO_2 method discussed in Section 13.3.9.

The forms of cyanide before and after destruction for the test is given in **Table 13-28**. The test results indicate that the air/SO_2 process will reduce the cyanide to below environmentally acceptable levels.

Table 13-28: Cyanide Destruction Test Results

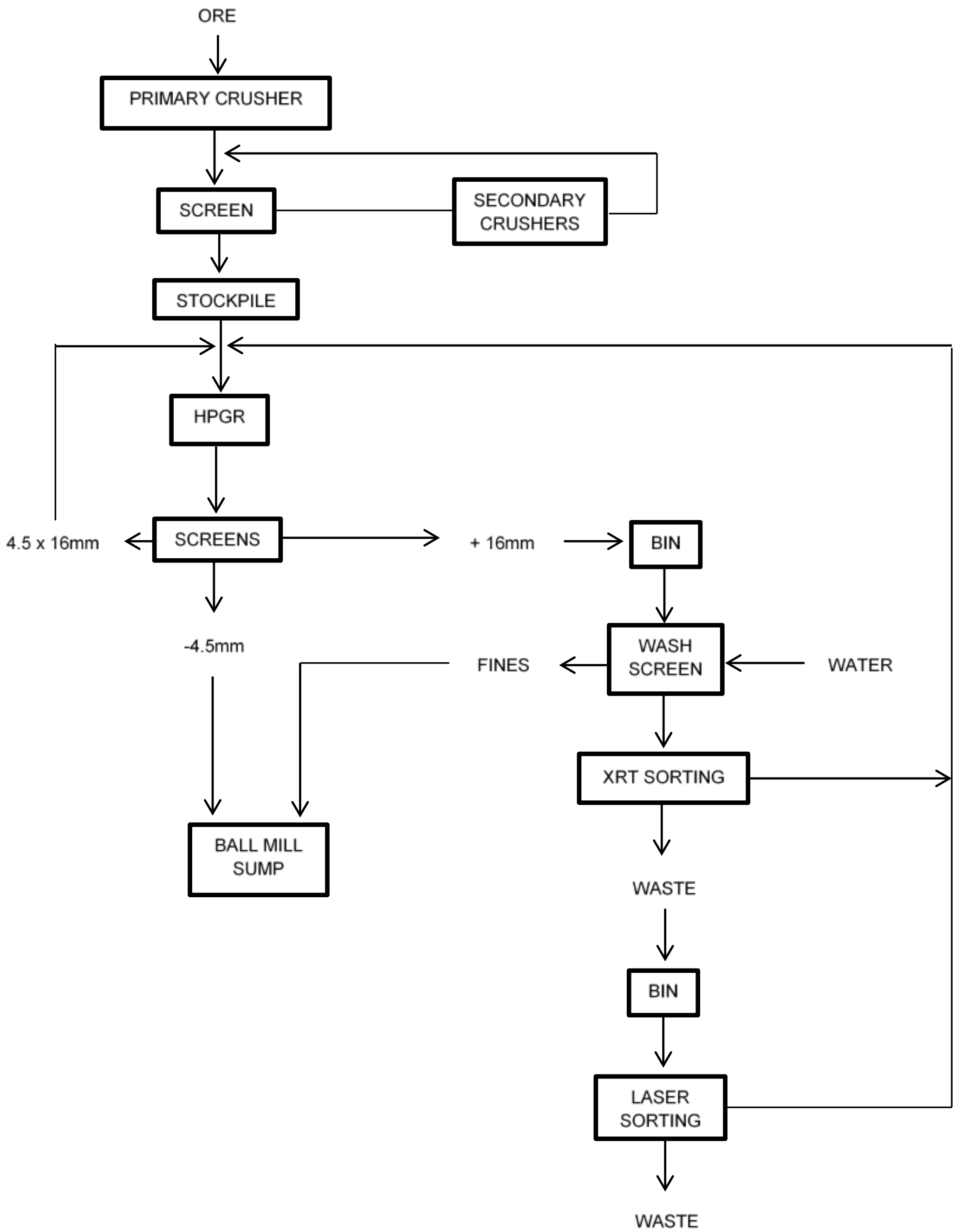
Forms of Cyanide	Before	After
Free, ppm	130	0.036
Total, ppm	124	0.062
WAD, ppm	132	0.048

13.5 Process Flowsheet

The process flowsheet given in **Figure 13-2** and **Figure 13-3** has significant advantages over the process flowsheet provided in the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia; March 2, 2018. The samples tested in the 2017 and 2018/2019 programs are believed to be representative of the deposit as stated in **Section 13.3 and 13.4** . These composites were selected to represent both near term and longer-term mining and spastically located to provide variability both horizontally and vertically. The recovery was estimated from the leach test results for the various composites given in **Table 13-20** to **Table 13-25** based on the proportion of each composite over the life of the mine. The only deleterious element in the deposit will be oxide and secondary copper which increases the cyanide consumption but will not impact gold extraction. Though the amount of oxide and secondary copper decreases with depth, the cyanide consumption was not corrected for it. Therefore, the cyanide consumption is conservatively estimated. These results have not been compiled in the final metallurgical report which will be issued by RDi later this year. The significant changes confirmed in the recent testwork and maintained in the process flowsheet include the following:

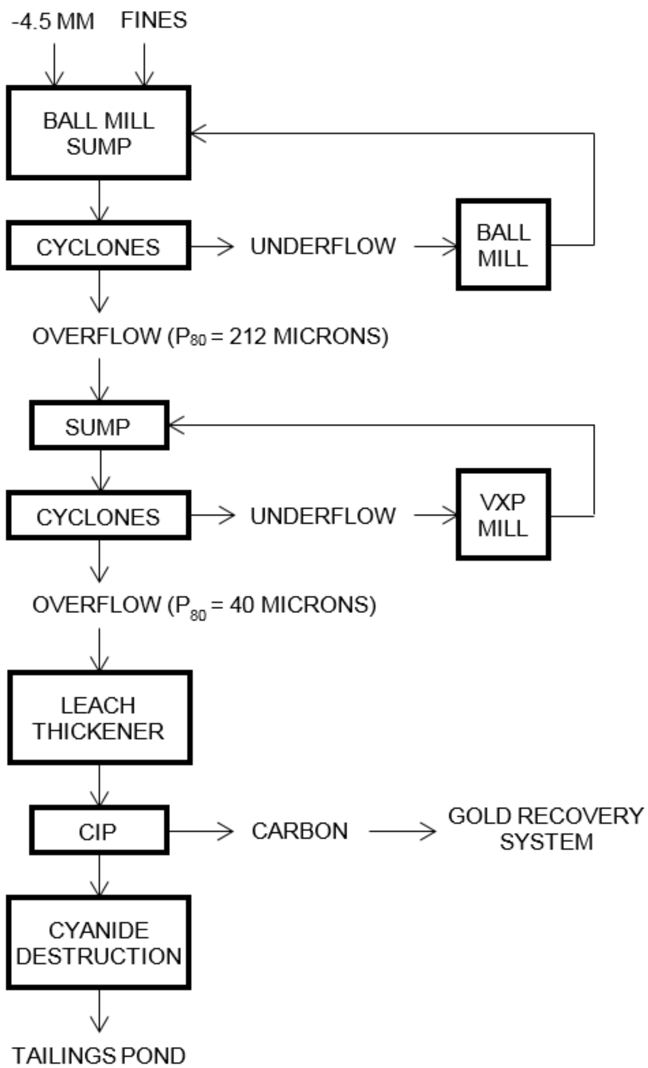
- Ore sorting of the coarse HPGR product which rejects ± 10% of the feed as waste product.
- Fine crushing and two-stage grinding with classification at each stage which reduces the quantity of material that needs to be ground in the second stage.

These modifications, along with finer grind of P_{80} of 40 microns, have resulted in producing much finer product to the leach circuit. This has resulted in enhancing gold extraction by ± 5.5%.



Source: Resource Development Inc, September 2019

Figure 13-2: Conceptual Process Flowsheet for Mt Todd Ore (1/2)



Source: Resource Development Inc, September 2019

Figure 13-3: Conceptual Process Flowsheet for Mt Todd Ore (2/2)

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following sections summarize the thought process, procedures, and results of Tetra Tech's independent estimate of the contained gold resources of the:

1) Batman Deposit,
2) The Quigleys Deposit, and
3) The Heap Leach Pad.

Only these three deposits currently have resource estimates classified in accordance with CIM Standards. Each of the mineral resources for the Batman and Quigleys deposits have been reported within a shell generated using Whittle™, 4-D Lerchs-Grossman algorithm. Mineral resources within such a shell are not mineral reserves and do not demonstrate economic viability.

It is the opinion of the QP for this section that the reported mineral resource classifications comply with current CIM definitions for each mineral class.

Geostatistics resource estimation and 3-D visualization was done with various mining software. The primary software used were MicroModel®, MicroMine®, Vulcan®, GemCom® and Whittle™. Additional statistical analysis was done with Statistica® and Excel®.

Figure 14-1 shows the relative locations of the three resource estimations for the Project. The Batman deposit is located approximately 500 meters west of the original plant site, the Quigleys deposit and the Heap Leach Pad are north and south of the existing tailings area respectively. **Table 14-1** summarizes the resources of each.

> *Cautionary statements regarding mineral resource estimates:*
>
> *Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves. Inferred resources are 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. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.*
>
> *All references to the term "ore" contained in this Technical Report refer to mineral reserves, not mineral resources.*

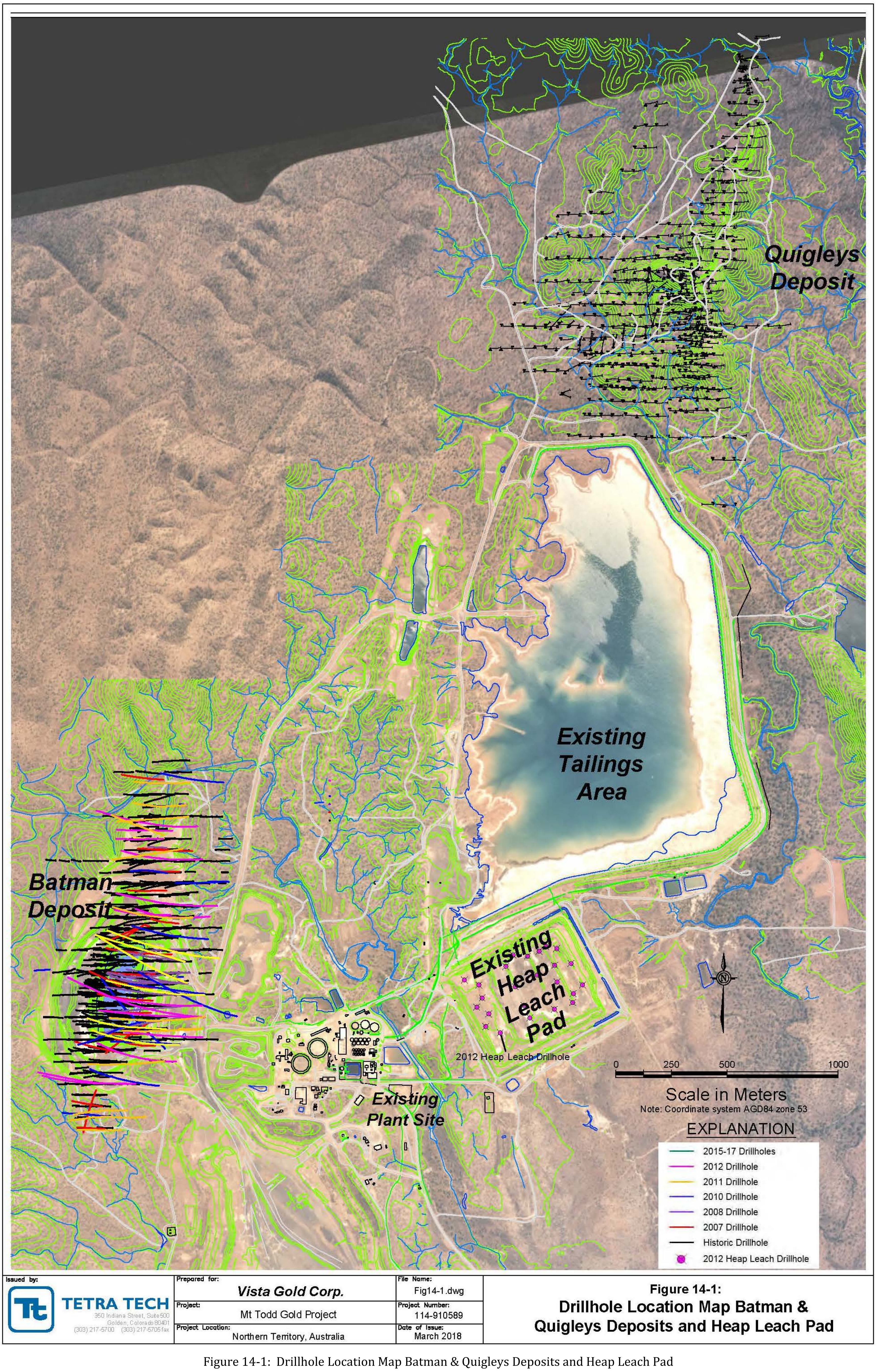


Figure 14-1: Drillhole Location Map Batman & Quigleys Deposits and Heap Leach Pad

Table 14-1: Summary of the Batman, Heap Leach Pad and Quiqleys Deposits

	Batman Deposit (August 2017)			Heap Leach Pad (May 2013)			Quigleys Deposit (August 2017)		
	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)
Measured (M)	77,725	0.88	2,191	-	-	-	457	1.27	19
Indicated (I)	200, 112	0.80	5,169	13,354	0.54	232	5743	1.12	207
Measured & Indicated	**277,837**	**0.82**	**7,360**	**13,354**	**0.54**	**232**	**6,200**	**1.13**	**225**
Inferred (F)	61,323	0.72	1,421	-	-	-	1,600	0.84	43

NOTES:
1. *Measured & indicated resources include proven and probable reserves.*
2. *Batman and Quigleys resources are quoted at a 0.40g-Au/t cut-off grade. Heap Leach resources are the average grade of the heap, no cut-off applied.*
3. *Batman: Resources constrained within a US\$1,300/oz gold Whittle™ pit shell. Pit parameters: Mining Cost US\$1.50/tonne, Milling Cost US\$7.80/tonne processed, G&A Cost US\$0.46/tonne processed, 50K TPD Ore, 355 Days/Yr., TPY Ore 17,750,000 TPY, G&A/Year 8,201 K US4, Au Recovery, Sulfide 85%, Transition 80%, Oxide 80%, 0.2g-Au/t minimum for resource shell. Selling Cost: US\$/oz recovered US412.00.*
4. *Quigleys: Resources constrained within a US\$1200/oz gold Whittle™ pit shell. Pit parameters: Mining cost US\$2.07/tonne, Milling Cost US\$9.623/tonne processed, Sale Cost US\$/oz US\$15.18, Royalty 1% NPR, Gold Recovery All Types, 70%.*
5. *Differences in the table due to rounding are not considered material*
6. *Rex Bryan of Tetra Tech is the qualified person responsible for the Statement of Mineral Resources for the Batman, Heap Leach Pad and Quiqleys deposits.*
7. *Thomas Dyer of Mine Development Associates is the qualified person responsible for developing the resource Whittle™ pit shell for the Batman Deposit.*
8. *The effective date of the Batman and Quigleys resource estimate is August 2017, the effective date of the Heap Leach resource is May 2013.*
9. *Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.*

14.2 Geologic Modeling of the Batman Deposit (2017)

Gold mineralization in the Batman deposit at the Project occurs in sheeted veins within silicified greywackes/shales/siltstones. The Batman deposit strikes north-northeast and dips steeply to the east. Higher grade zones of the deposit plunge to the south. The core zone is approximately 200-250 meters wide and 1.5 km long, with several hanging wall structures providing additional width to the orebody. Mineralization is open at depth as well as along strike, although the intensity of mineralization weakens to the north and south along strike.

The Batman deposit contains 94% of the gold resources classified as measured and indicated within the Project. Only the Batman resources have been further converted to classified reserves of Proven and possible ore.

Over several drilling campaigns, the shape of the mineralized shear zone has been adjusted and resized to accommodate this new data. Deeper step-out drilling by Vista indicated that the lower footwall of the core complex was previously not drill tested. The additional drilling confirmed the previously indicated higher grade plunge of the core complex. The new data was used to re-define the granite contact that constrains the lower footwall of the core complex. The granite contact is a mineral exclusionary zone and has been modeled as a triangulated surface, which can be seen in **Figure 14-2.**

In addition to resizing the core complex wireframe solid, three structures paralleling the core complex to the east were also resized and constructed into wireframe solids and used for this resource estimate. The interpreted parallel structures represent an echoing of the main mineralization controls of the core complex nearer the surface and to the east. Wireframe solids for the parallel structures were interpreted on sections using Au mineralization, veining percentage, visual sulfide percentages, structural orientations and multi-element data. Deep drilling conducted in 2011 and through 2012 confirmed the existence of these structures and indicates a possible increasing definition and grade at depth.

The Batman Deposit resource was updated to reflect the increase in available data provided by drilling conducted in 2015 through 2017. A redefinition of the geometry of a granite contact reducing primarily inferred resources at depth. A Whittle^{TM} pit further constrained the reported 2017 resources.

Figure 14-2 is a schematic of domain designations and crucial parameters used in the resource model. The figure lists the resource classification codes, the rock codes, density assignments. Also schematically shown are the constraining surfaces for current topography, levels of oxidation, granite basement and the US$1,300/oz gold pit shell constrain reported resources.

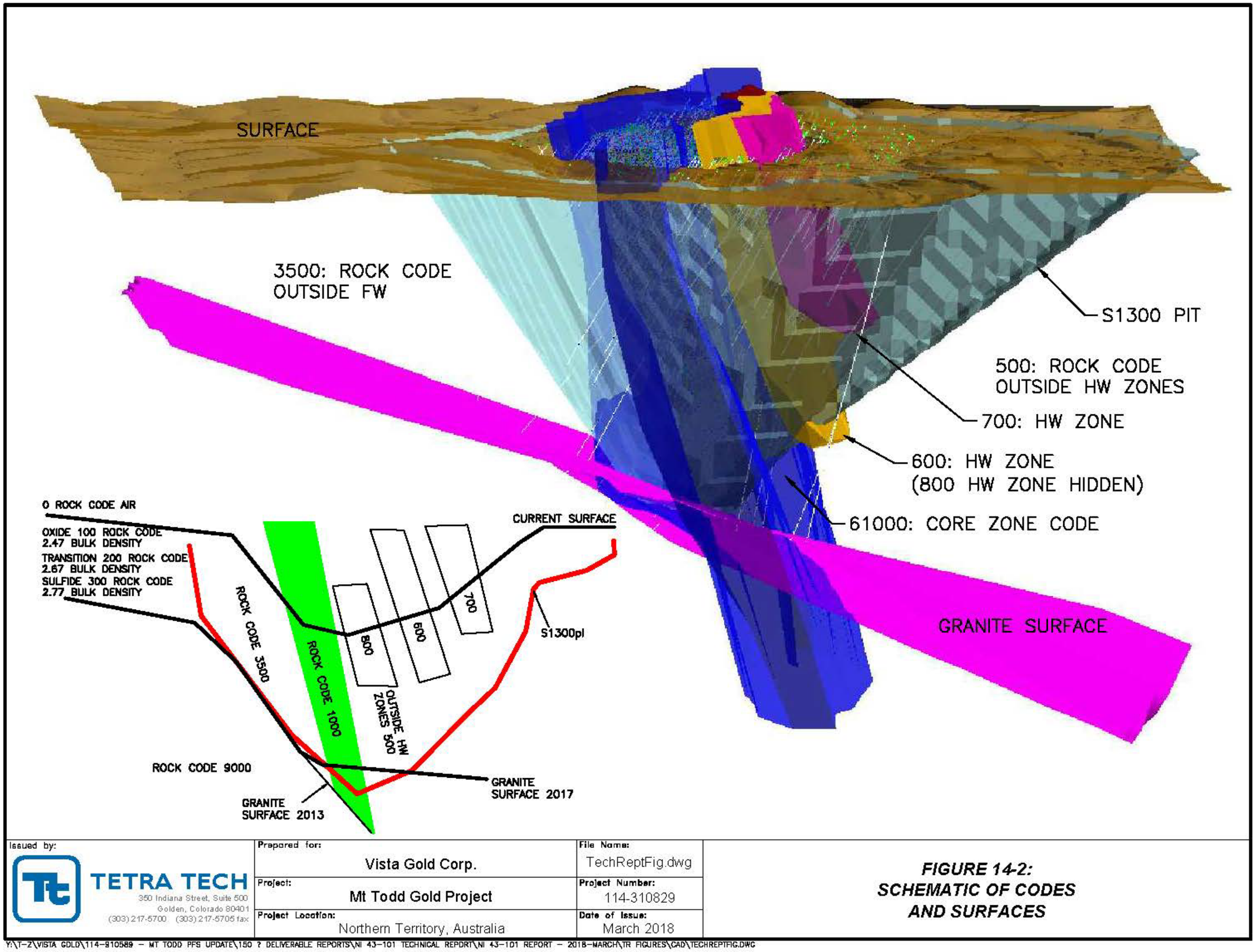


Figure 14-2: Schematic of Codes and Surface Designations (Looking North)

Figure 14-3 shows a sectional view of the drillhole data as of June 2013 unchanged in 2017 due to the limited amount of new drilling. The predominate direction of Batman Deposit drilling is dipping at approximately 45-degrees to the west. The figure shows the original 2013 and new 2017 granite contact surfaces. Yellow highlight shows the 2017 truncation of the core zone.

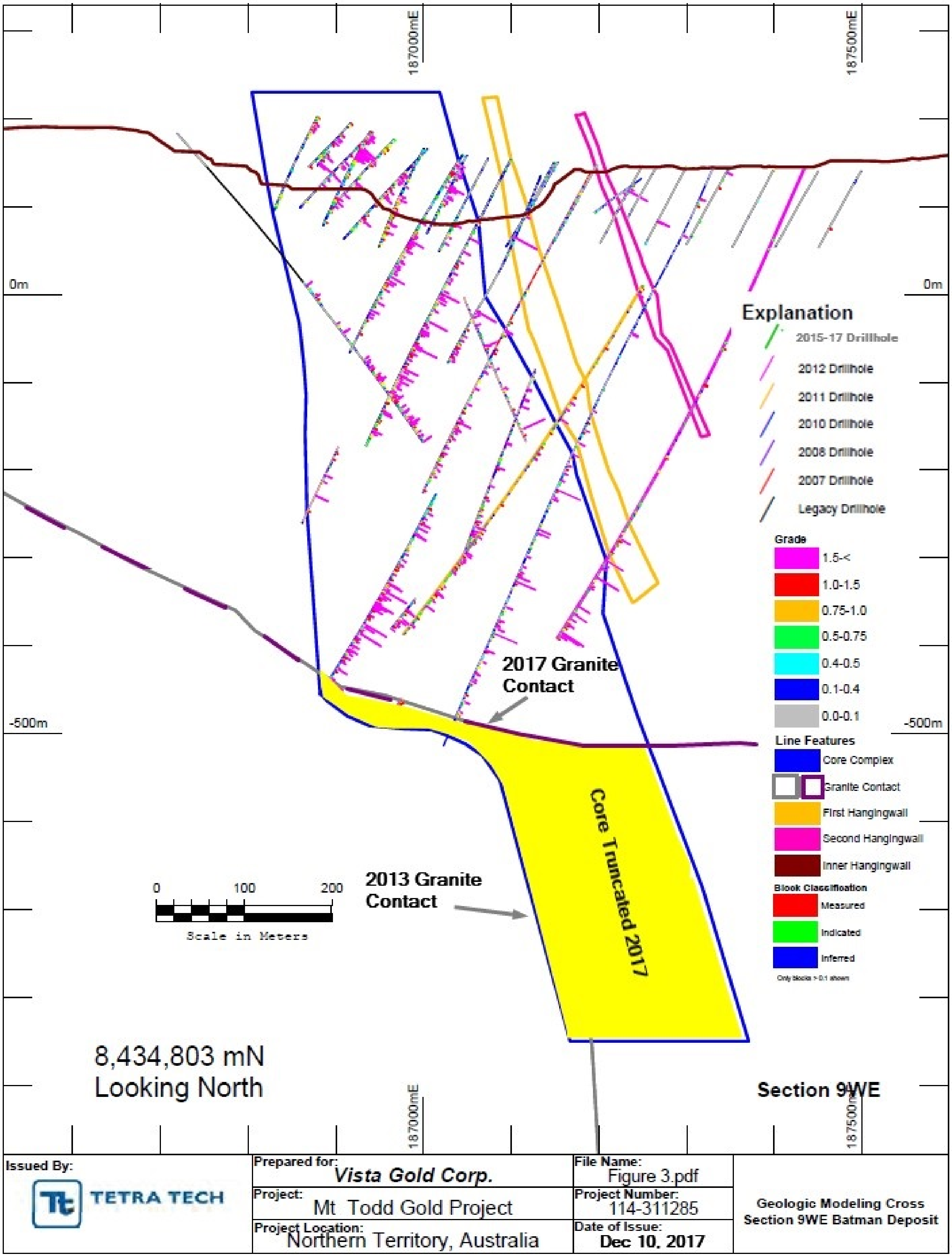


Figure 14-3: Sectional View of Drillhole Data 8,434,803 mN (Looking North)

14.2.1 Batman Deposit Density Data

Drillhole data through 2012 for a total of 16,373 samples were tested for bulk density (diamond core). These bulk densities were carried out on a 10 to 15 centimeters (cm) piece of core from a 1-m sample. Based on this work, the bulk densities applied to the resource model are presented in **Table 14-2**.

Table 14-2: Summary of Batman Bulk Density Data by Oxidation State

Oxidation	No. of Samples	Min	Max	Mean	Variance	CV
Oxide	2,341	1.77	3.28	2.47	0.04	0.08
Transitional	1,316	2.07	3.55	2.67	0.01	0.04
Primary	12,716	1.58	3.90	2.77	0.006	0.03

Since then, an additional 3,370 samples have confirmed these results for Primary material bulk density.

14.2.2 Grade Capping

Review of the log probability plot of the composited gold grades shows that there is a distinct break in the distribution at 50 g-Au/t. All gold composites were capped at this value. Inspection of the cumulative frequency plot of data from the core domain codes 600, 700, 800 and 1000 suggest that the 1m assay values when composited to 4 m limits the higher gold grades to a maximum value of 10.9 g-Au/t.

14.3 Batman Block Model Parameters

Table 14-3 details the physical limits of the Batman deposit block model utilized in the estimation of mineral resources.

Table 14-3: Block Model* Physical Parameters – Batman Deposit

Direction (dir)	Minimum (m) MGA94 z53	Maximum (m) MGA94 z53	Block Size	#Blocks
y-dir	8,433,801 mE	8,436,213 mE	12 m	201
x-dir	185,999 mN	187,931 mN	12 m	161
z-dir	-994 m	224 m	6 m	203

** Model changed from previous Tetra Tech estimates to reflect the 2011 drillhole locations and depths.*

14.3.1 Geostatistics of the Batman Deposit

Geology of the Batman Deposit consists of a sequence of hornfelsed interbedded greywackes, and shales with minor thin beds of felsic tuffs. Minor lamprophyre dykes trending north-south crosscut the bedding. The mineralized lithologic package consists of a tabular deposit striking at 325^o with a dip of 40^o to 60^o to the southeast. The majority of drilling slants at a dip of approximately 65^o with an azimuth of 270^o.

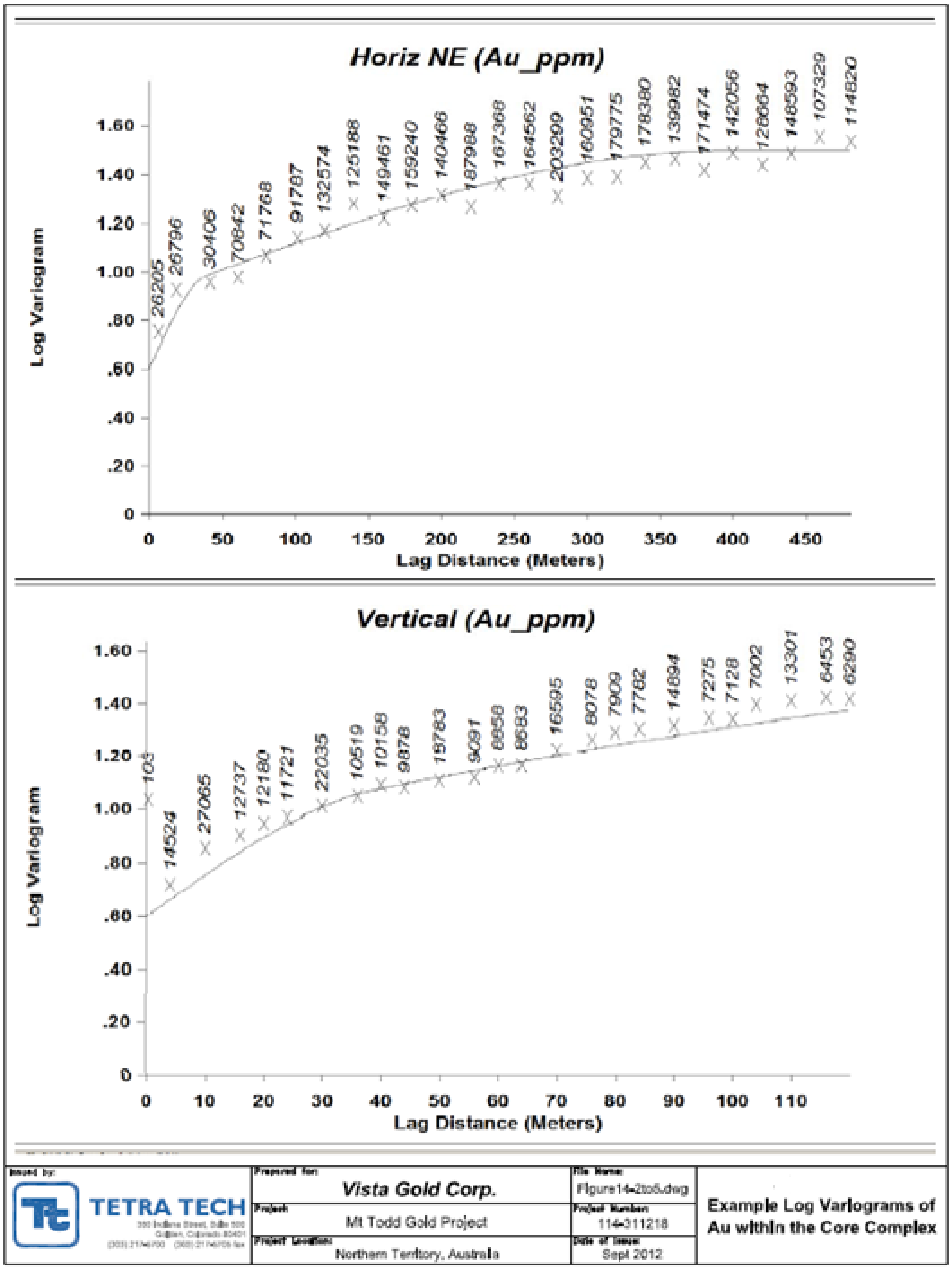
Bedding parallel shears are present in some of the shale horizons (especially in lithologic units SHGW23, GWSH23, and SH22). These bedding shears are identified by quartz/calcite sulfidic breccias. Pyrite, pyrrhotite, chalcopyrite, galena, and sphalerite are the main primary sulfides associated with the bedding parallel shears.

NE-SW trending faults and joint sets crosscut bedding. Only minor movement has been observed on these faults. Calcite veining is sometimes associated with these faults. These structures appear to be post mineralization.

Northerly trending quartz sulfide veins and joints striking at 0^o to 20^o, dipping to the east at 60^o, are the major location for mineralization in the Batman Deposit. The veins are 1 to 100 mm in thickness with an average thickness of around 8 to 10 mm. The veins consist of dominantly quartz with sulfides on the margins. The veining occurs in sheets with up to 20 veins per horizontal meter. These sheet veins are the main source of mineralization in the Batman Deposit.

The mineralization within the Batman Deposit is directly related to the intensity of the north-south trending quartz sulfide veining. The lithological units impact on the orientation and intensity of mineralization. Sulfide minerals associated with the gold mineralization are pyrite, pyrrhotite and lesser amounts of chalcopyrite, bismuthinite and arsenopyrite. Galena and sphalerite are also present but appear to be post gold mineralization and are related to calcite veining bedding and the east-west trending faults and joints.

Multiple directional variograms explored the best continuity of mineralization given the combination of control by bedding and sulfide veining. **Figure 14-4** is an example of two log variograms in the core complex.



Source: Tetra Tech, prepared in August 2017

Figure 14-4: 2012 Example Log Variograms of Gold within the Core Complex

Table 14-4 shows the resource classification criteria and variogram for the Batman resource model.

Table 14-4: Batman Resource Classification Criteria and Variogram

Category	Search Range & Kriging Variance	No. of Sectors/ Max Points per DH	Search Anisotropy	Min Points	Composite Codes	Block Codes	CORE
Indicated	Core Complex: 150 m & KV < 0.45 Pass 1	4/2	(1.0:0.7:0.4) [110:80:0]	2	1000	1000	CORE COMPLEX
Measured	Core Complex: 60 m & KV < 0.**30**) Pass 2 (overwrite Pass 1)	4/3	(1.0:0.7:0.4) [110:80:0]	4	1000	1000	
inferred	Core Complex KV >= 0.34 Classification Step	4/2	(1.0:0.7:0.4) [110:80:0]	2	1000	1000	
inferred	Outside Core Complex: 150 m & KV <= 0.45 Pass 3	4/3	(1.0:0.7:0.4) [110:80:0]	3	500/3500	500/ 3500	OUTSIDE CORE COMPLEX
inferred	Outside Core Complex: 50 m & KV > = 0.45 Pass 4 (overwrite Pass 3)	4/3	(1.0:0.7:0.4) [110:80:0]	8	500/3500	500/ 3500	
inferred	Primary Satellite Deposit: 150 m & KV >= 0.45 Pass 5	4/3	(1.0:0.7:0.4) [110:80:0]	3	600	600	
Indicated	Primary Satellite Deposit: 50 m & KV < 0.45 Pass 6 (overwrite Pass 5)	4/3	(1.0:0.7:0.4) [110:80:0]	8	600	600	
inferred	Secondary Satellite Deposit: 150 m & KV >= 0.45 Pass 7	4/3	(1.0:0.7:0.4) [110:80:0]	3	700	700	
Indicated	Secondary Satellite Deposit: 50 m & KV < 0.45 Pass 8 (overwrite Pass 7)	4/3	(1.0:0.7:0.4) [110:80:0]	8	700	700	
inferred	Tertiary Satellite Deposit: 150 m & KV >= 0.45 Pass 9	4/3	(1.0:0.7:0.4) [110:80:0]	3	800	800	
Indicated	Tertiary Satellite Deposit: 50 m & KV < 0.45 Pass 10 (overwrite Pass 9)	4/3	(1.0:0.7:0.4) [110:80:0]	8	800	800	
VARIOGRAM FOR ALL CATEGORIES							
Type: Spherical First Rotation (Azimuth: 110) Second Rotation (Dip: 80) Third Rotation (Tilt: 0)	Primary Axis: 150m Secondary Axis: 105m Tertiary Axis: 60m	Nugget: 0.6 Sill 1: 0.3 Sill 2: 0.2	Range 1: 40m Range 2: 500m				

INDEX		
Zone Codes	**Zone Names**	**Notes**
3500	Footwall	**Ranges** In meters (m) **KV** = kriging variance, Passes refer to multiple re-estimations of blocks with greater constraints (minimum points, search ranges, etc.) imposed. **Core and Satellites** have more consistent gold grades, while the Footwall and Hanging Wall have patchy gold grades, **Search Ranges (a:b:c)** Proportion of Maximum Range for: a. Primary Axis Length: b. Secondary Axis Length: c. Tertiary Axis Length **Orientation of Ellipse [1:2:3]** 1. Azimuth of Primary Axis : 2. Dip of Primary Axis: 3. Rotation (Tilt) around Primary Axis
1000	Core Complex	
800	Tertiary Satellite (between 600 and 700)	
700	Secondary Satellite (in HW farthest from Core)	
600	Primary Satellite (in HW nearest to Core)	
500	Hanging Wall Area	

Figure 14-5 through **Figure 14-10** are a series of sections and plan views of the Batman deposit block model from 2013. An insert showing the 2017 block model within the core and outlier zones has been added. Note that the granite layer that truncates the lowest portion of the deposit was re-modelled from a steeply plunging surface in 2013 to more horizontal surface in 2017.

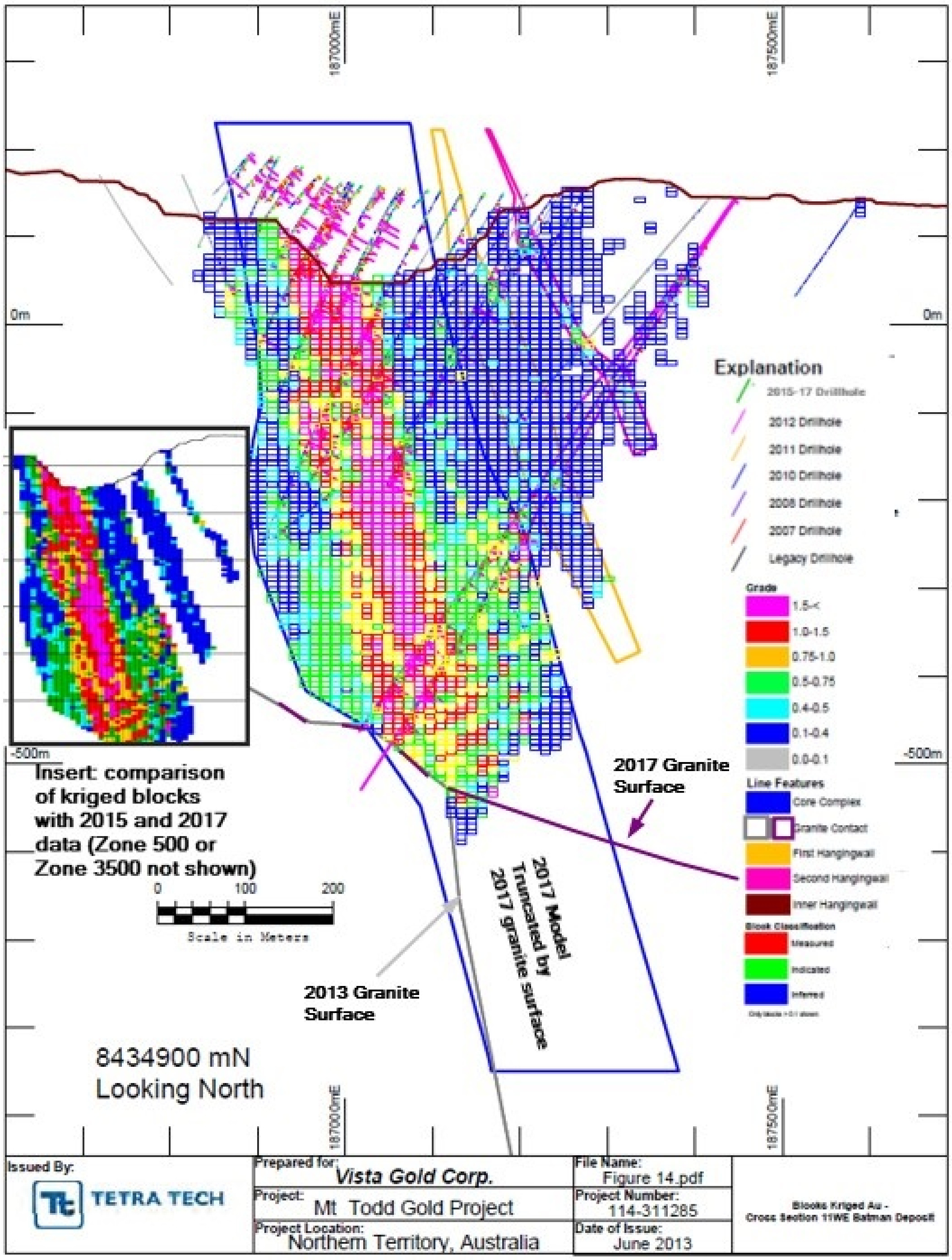


Figure 14-5: 2013 Study – Blocks Kriged Au – Cross-section 8,434,900 mN Looking North, Batman Deposit

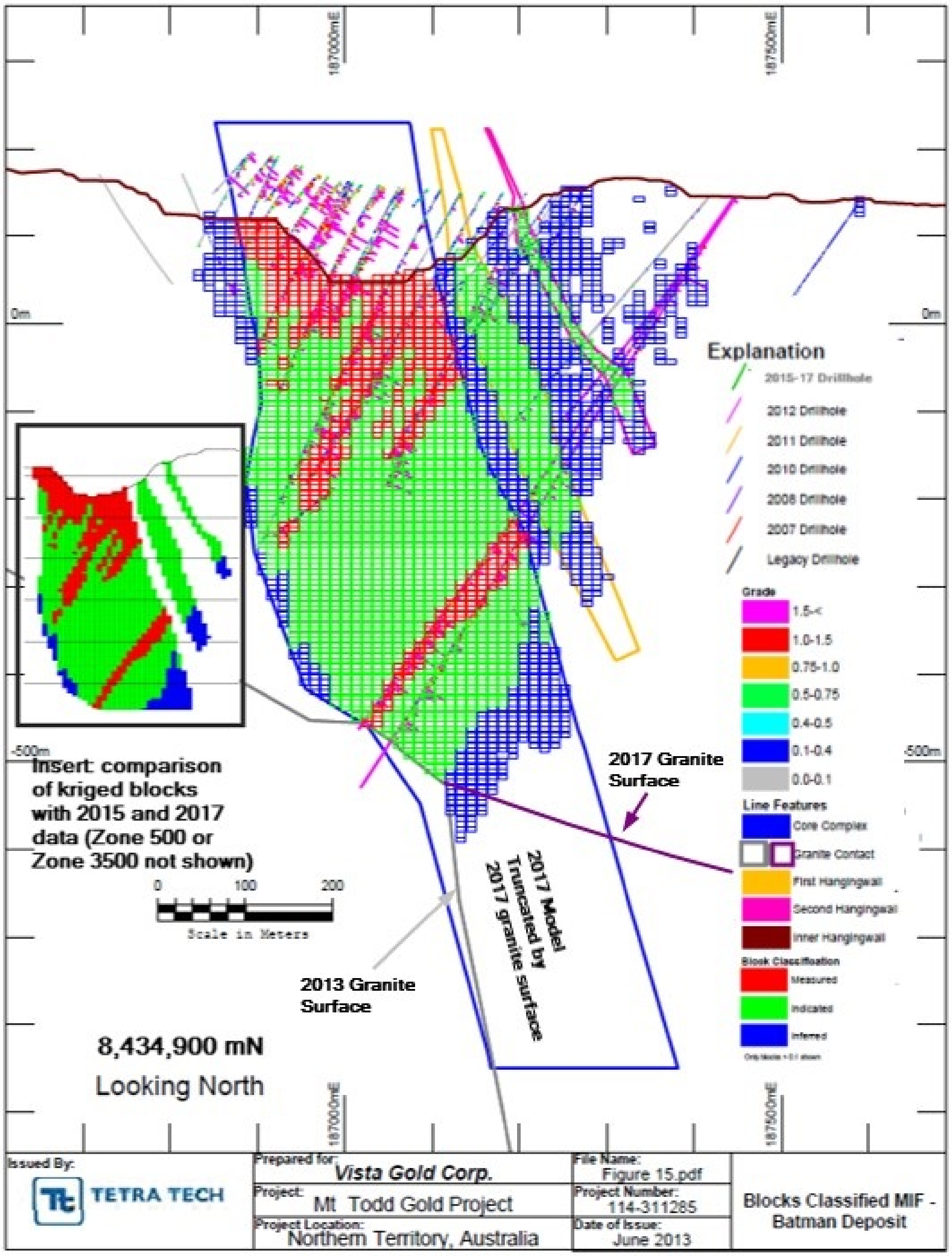


Figure 14-6: 2013 Study – Classified Blocks Measured, Indicated, and Inferred – Cross-section 8,434,900 mN Looking North, Batman Deposit

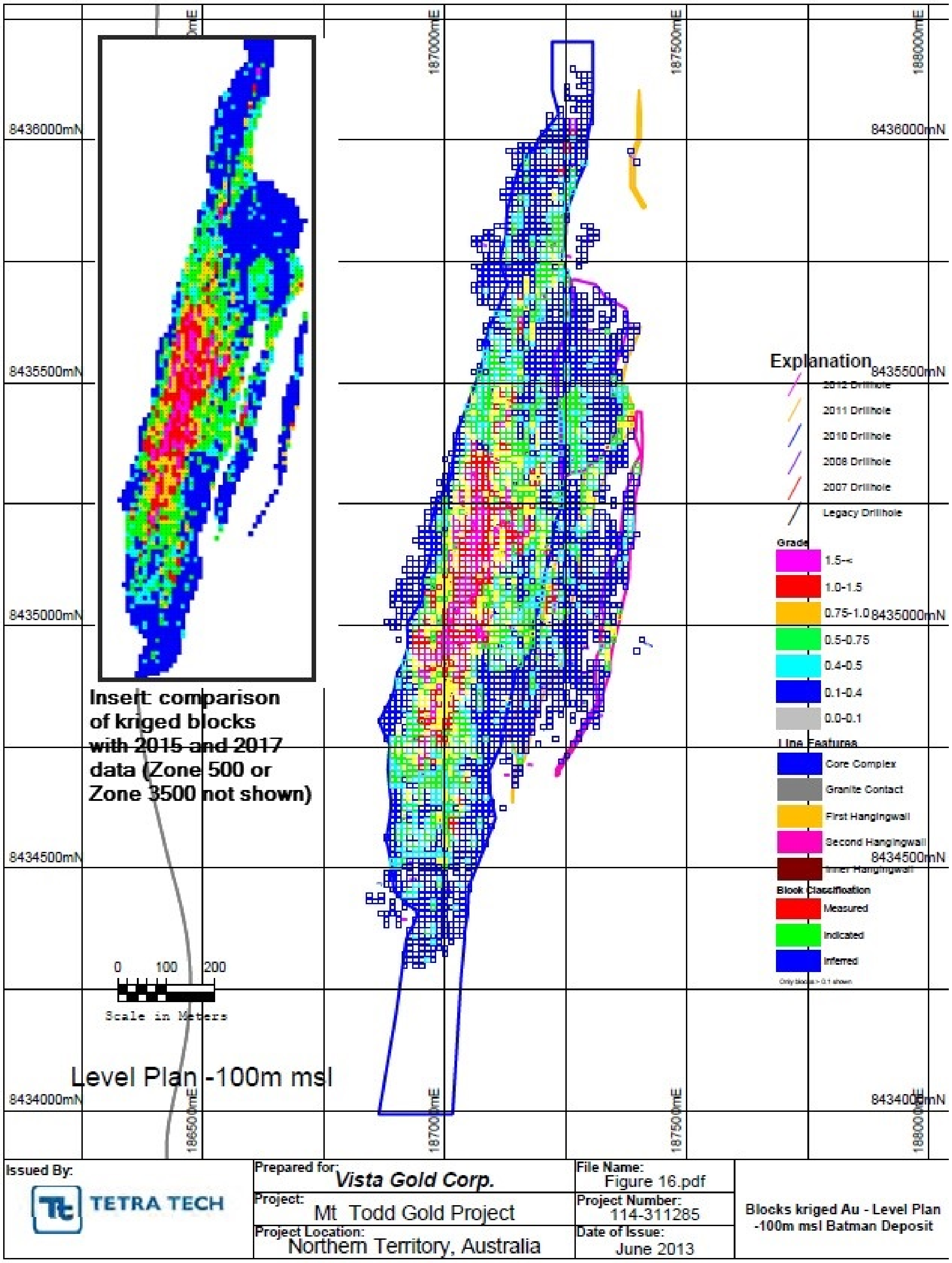


Figure 14-7: 2013 Study – Blocks Kriged Au – Level Plan -100m msl Batman Deposit

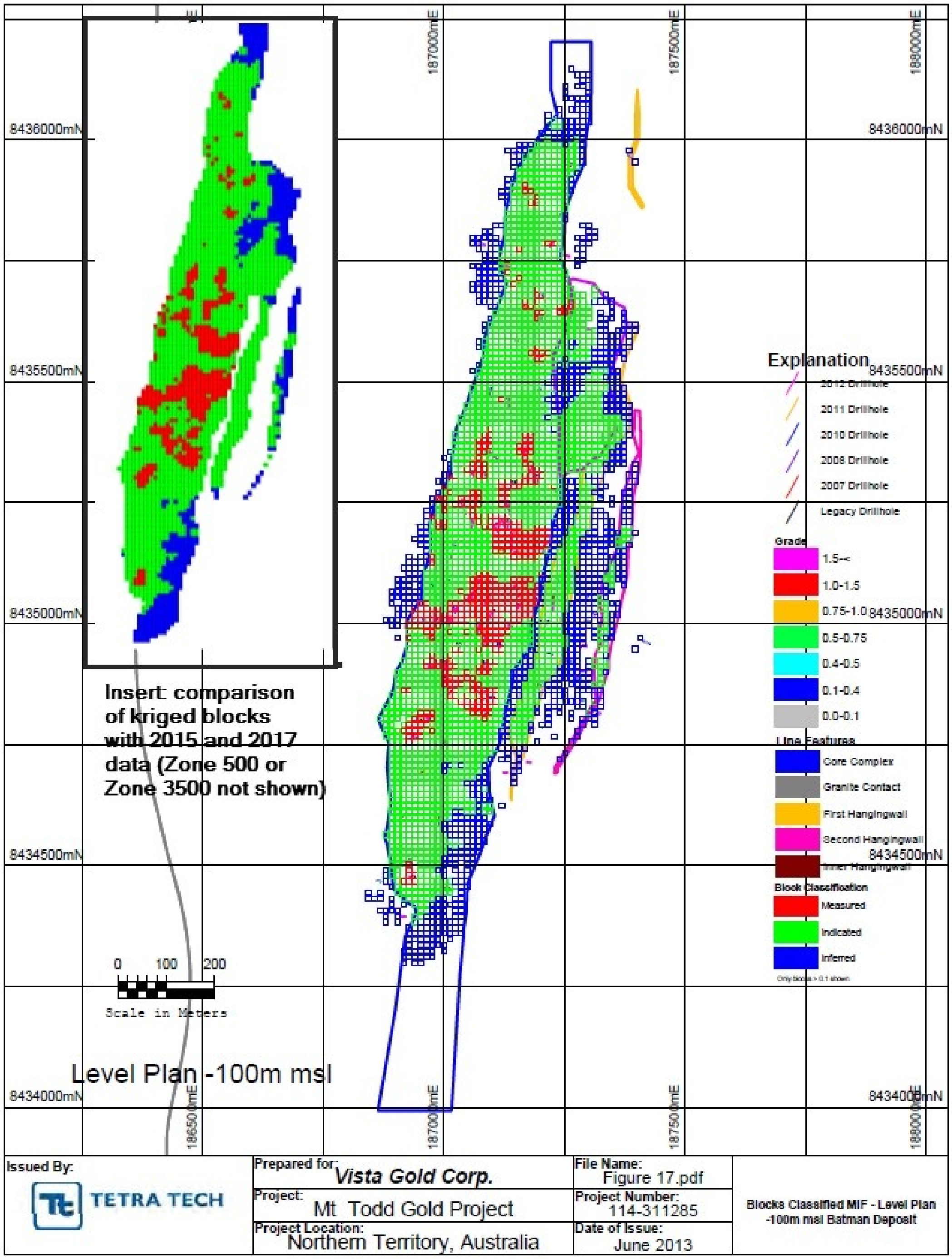


Figure 14-8: 2013 Study – Classified Blocks Measured, Indicated, and Inferred – Level Plan -100m msl Batman Deposit

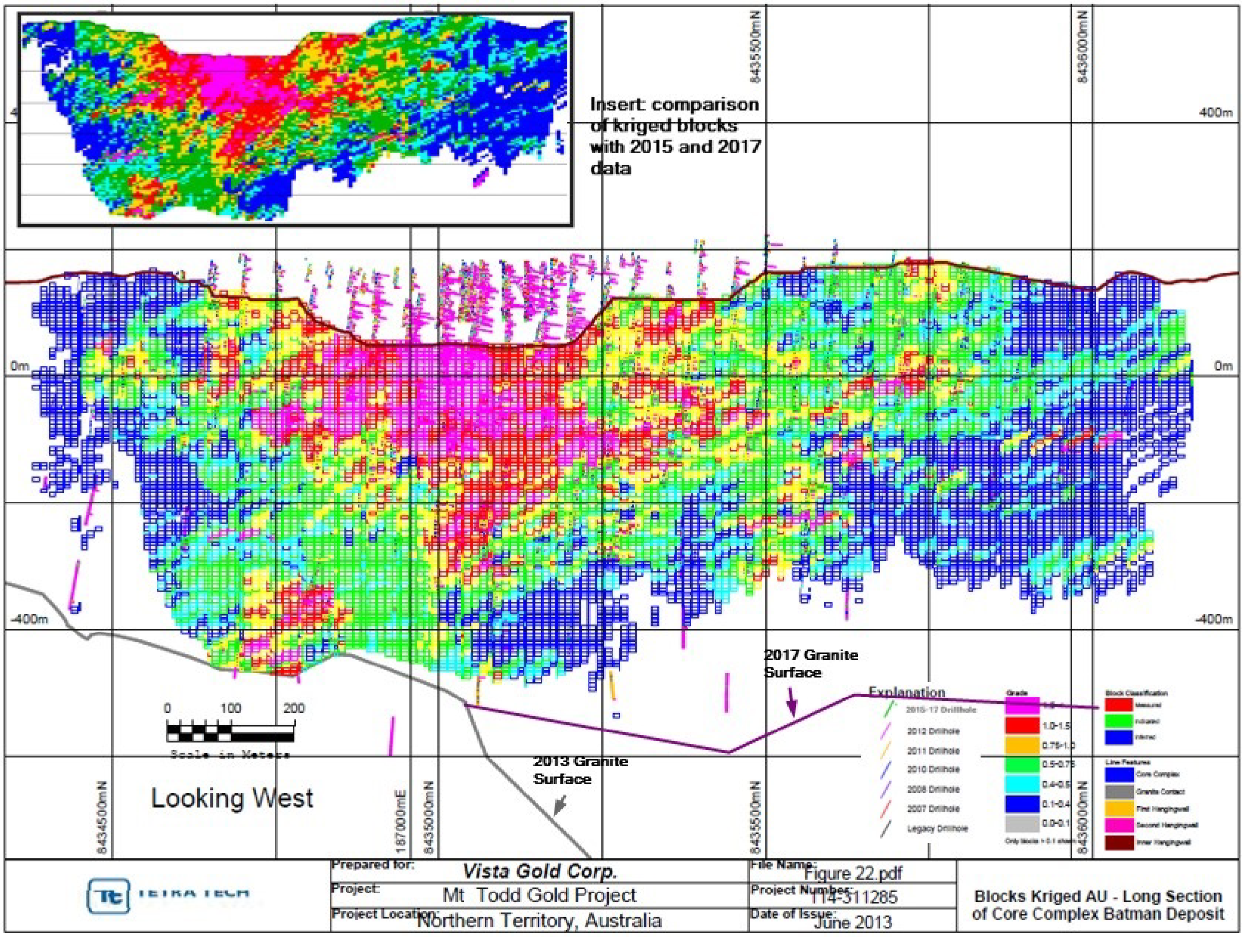


Figure 14-9: 2013 Study – Blocks Kriged Au – Long Section of the Core Complex Looking West

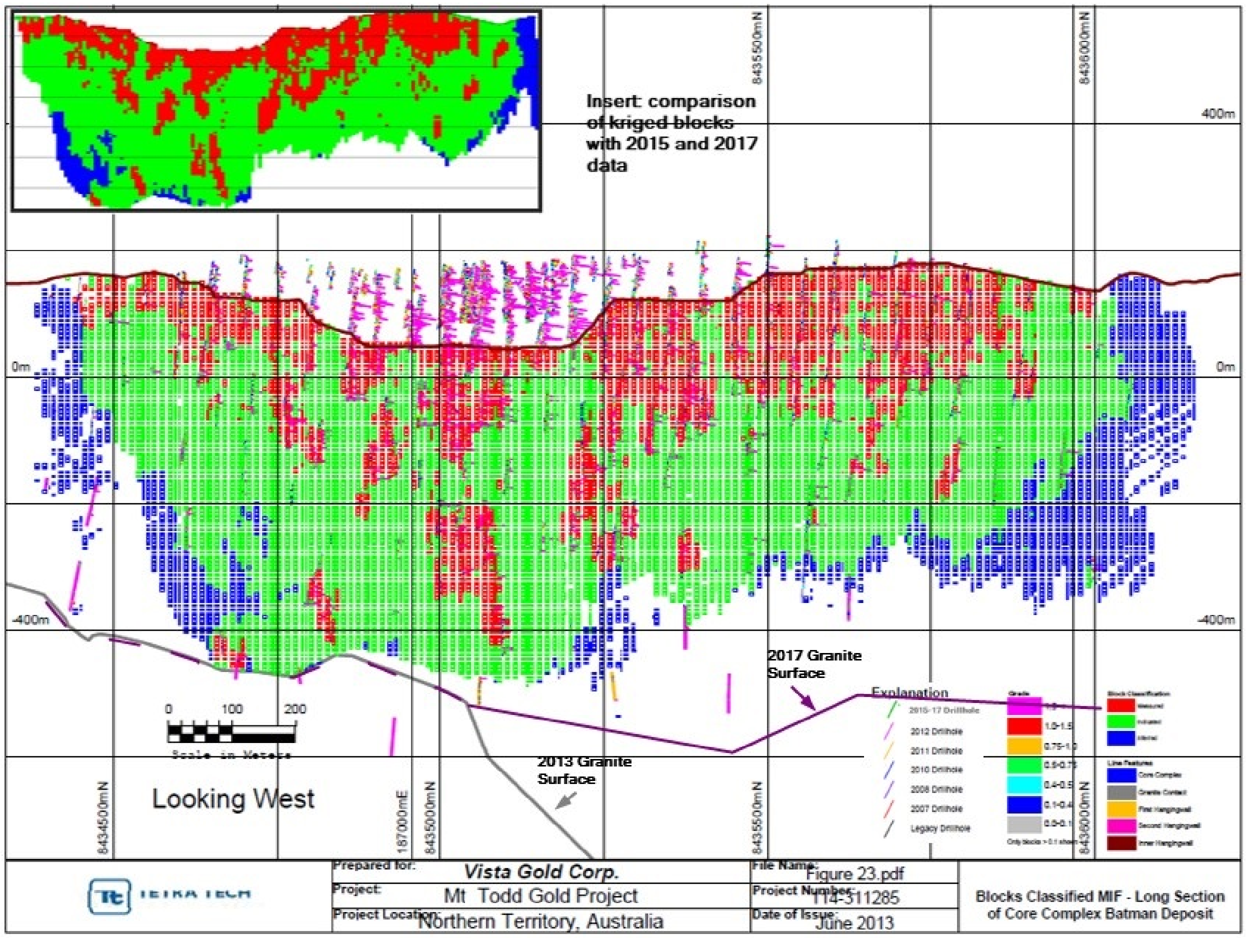


Figure 14-10: 2013 Study – Classified Blocks Measured, Indicated, and Inferred – Long Section of the Core Complex Looking West

Table 14-5 lists the current 2017 Batman measured and indicated resource estimates at cutoff grades ranging from 0.3 g-Au/t to 2.0 g-Au/t. **Table 14-6** lists the current 2017 Batman inferred resource estimates at cutoff grades ranging from 0.3 g-Au/t to 2.0 g-Au/t**.**

Figure 14-11 graphically shows the grade-tonnage for Measured plus Indicated classified Batman deposit resources.

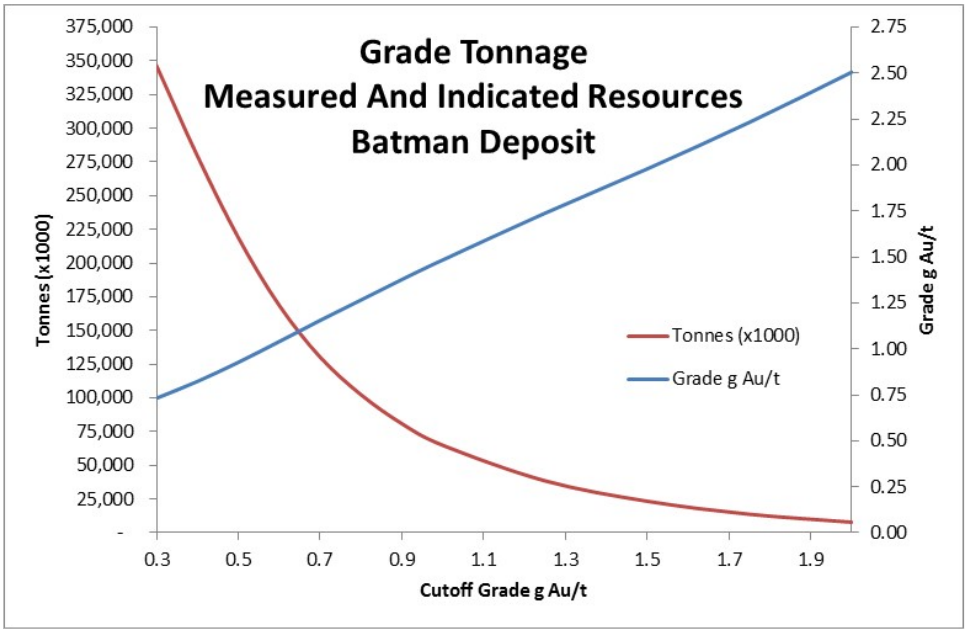
Table 14-5: Batman Deposit Measured and Indicated Gold Resource Estimate

	Cutoff Grade g-Au/t	Tonnes (x1000)	Average Grade g-Au/t	Total Au Ounces (x1000)
MEASURED	2.00	2,474	2.40	191.1
	1.75	4,616	2.15	319.4
	1.50	8,186	1.92	505.1
	1.25	13,205	1.71	725.8
	1.00	21,512	1.48	1,024.0
	0.90	26,481	1.38	1,175.2
	0.80	33,167	1.27	1,357.2
	0.70	41,594	1.17	1,560.0
	0.60	52,492	1.06	1,787.3
	0.50	64,597	0.96	2,001.1
	0.40	**77,725**	**0.88**	**2,191.1**
	0.3	90,719	0.80	2,337.8
	2.00	5,413	2.55	443
INDICATED	1.75	9,124	2.27	666
	1.50	15,194	2.01	982
	1.25	25,183	1.75	1,420
	1.00	43,059	1.49	2,057
	0.90	54,104	1.38	2,394
	0.80	68,845	1.26	2,796
	0.70	88,256	1.15	3,262
	0.60	115,528	1.03	3,830
	0.50	153,278	0.91	4,494
	0.40	**200,112**	**0.80**	**5,169**
	0.30	253,187	0.71	5,765
	2.00	7,887	2.50	634
MEASURED + INDICATED	1.75	13,740	2.23	985
	1.50	23,380	1.98	1,487
	1.25	38,387	1.74	2,145
	1.00	64,571	1.48	3,081
	0.90	80,585	1.38	3,569
	0.80	102,012	1.27	4,153
	0.70	129,850	1.16	4,822
	0.60	168,021	1.04	5,617
	0.50	217,875	0.93	6,495
	0.40	**277,837**	**0.824**	**7,360**
	0.30	343,906	0.73	8,104
	0.30	343,906	0.73	8,104

NOTE:

(1) The measured and indicated resource estimates presented in this table include the proven and probable reserves presented in Section 15 of this Technical Report.

(2) Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.



Source: Tetra Tech, Inc (August 2017)
NOTE: Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 14-11: Grade Tonnage Curve of Measured and Indicated Resource for the Batman Deposit

Table 14-6: Batman Deposit Inferred Gold Resource Estimate

	Cutoff Grade g-Au/t	Tonnes (x1000)	Average Grade g-Au/t	Total Au Ounces (x1000)
INFERRED	2.00	1,664	2.94	157
	1.75	2,196	2.68	189
	1.50	2,975	2.40	230
	1.25	4,532	2.05	298
	1.00	7,914	1.65	419
	0.90	10,170	1.49	487
	0.80	14,327	1.30	601
	0.70	19,576	1.15	726
	0.60	27,798	1.00	897
	0.50	40,964	0.86	1,128
	0.40	**61,323**	**0.72**	**1,421**
	0.30	94,532	0.59	1,790

NOTE:
(1) Resources constrained within a US$1,300/oz gold Whittle™ Pit Shell. Pit parameters: Mining Cost US$1.50/tonnes, Milling Cost US$7.80/tonnes processed, G&A Cost US$0.46/tonnes processed, 50K TPD Ore, 355 Days/Yr., TPY Ore 17,750,000 TPY, G&A/Year 8,201 K US$, Au Recovery, Sulfide 85%, Transition 80%, Oxide 80%, 0.2g-Au/t minimum for resource shell. Tonnage, grades and totals may not total due to rounding. The reported resources at a cutoff of 0.4 g/t is highlighted.
(2) Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

14.4 Batman Estimation Quality

Several methods were used to validate the block model to determine the adequacy of the Batman deposit resource. Confirmatory drilling was used to ascertain the general good quality of the model within the core zone. In addition, overlaid cumulative frequency plots of blocks, composites, and assays were used. The three overlaid plots showed the expected decrease in the variability of the gold distributions going from assays to assay composites and then to kriged blocks. In addition:

- Jackknife studies were employed to determine the optimum kriging search parameters and the overall quality of the estimation as required by classification. **Figure 14-12** shows the Jackknife results for the measured class.
- Numerous swath plots were analyzed in the direction of rows and columns were used to verify that composite and block gold grades are spatially in sync. Several examples of these swath plots are shown in **Figure 14-13**.
- The use of visual inspection of the kriged blocks models in section and plan and the inspection of gold histograms of assays, composites and blocks.

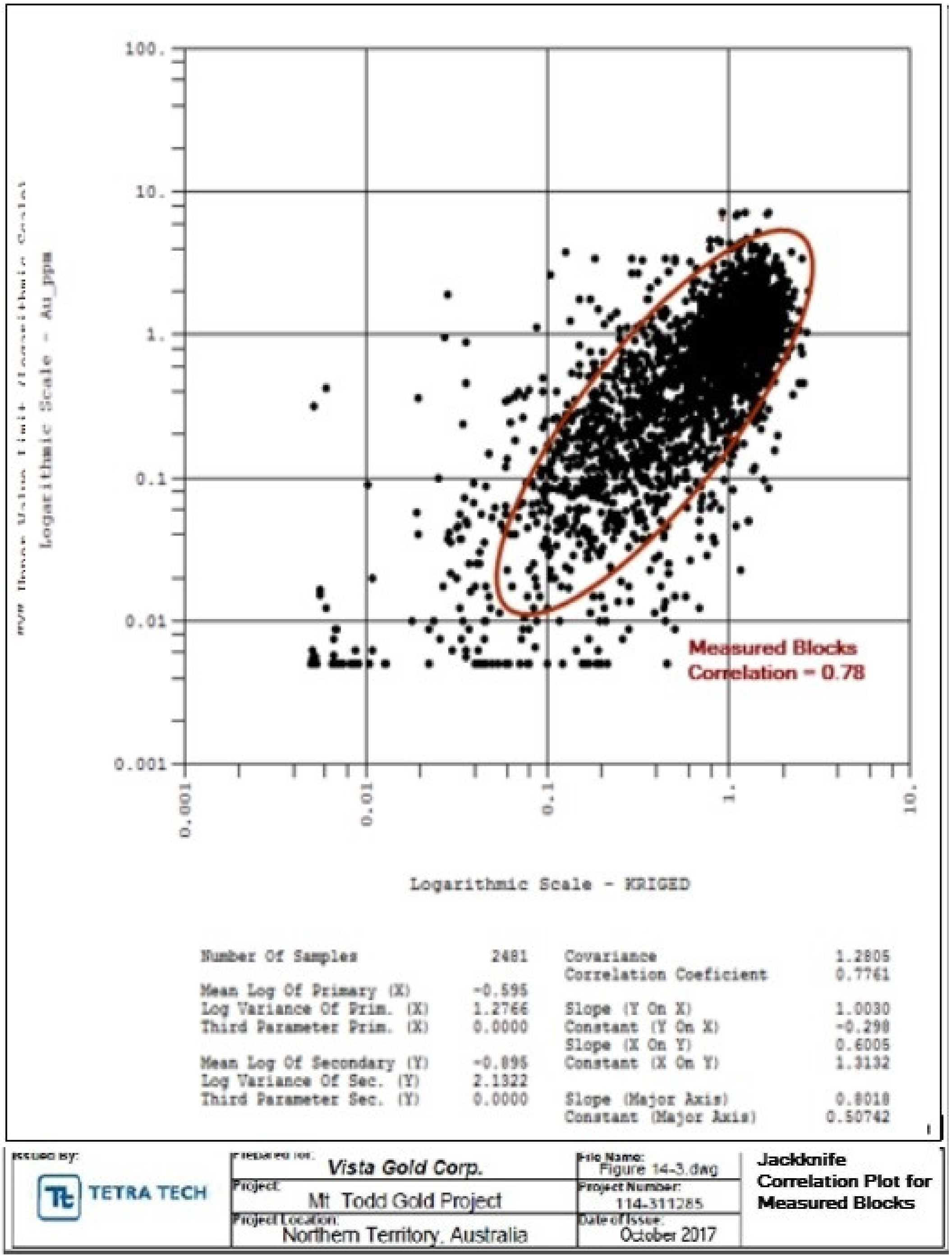
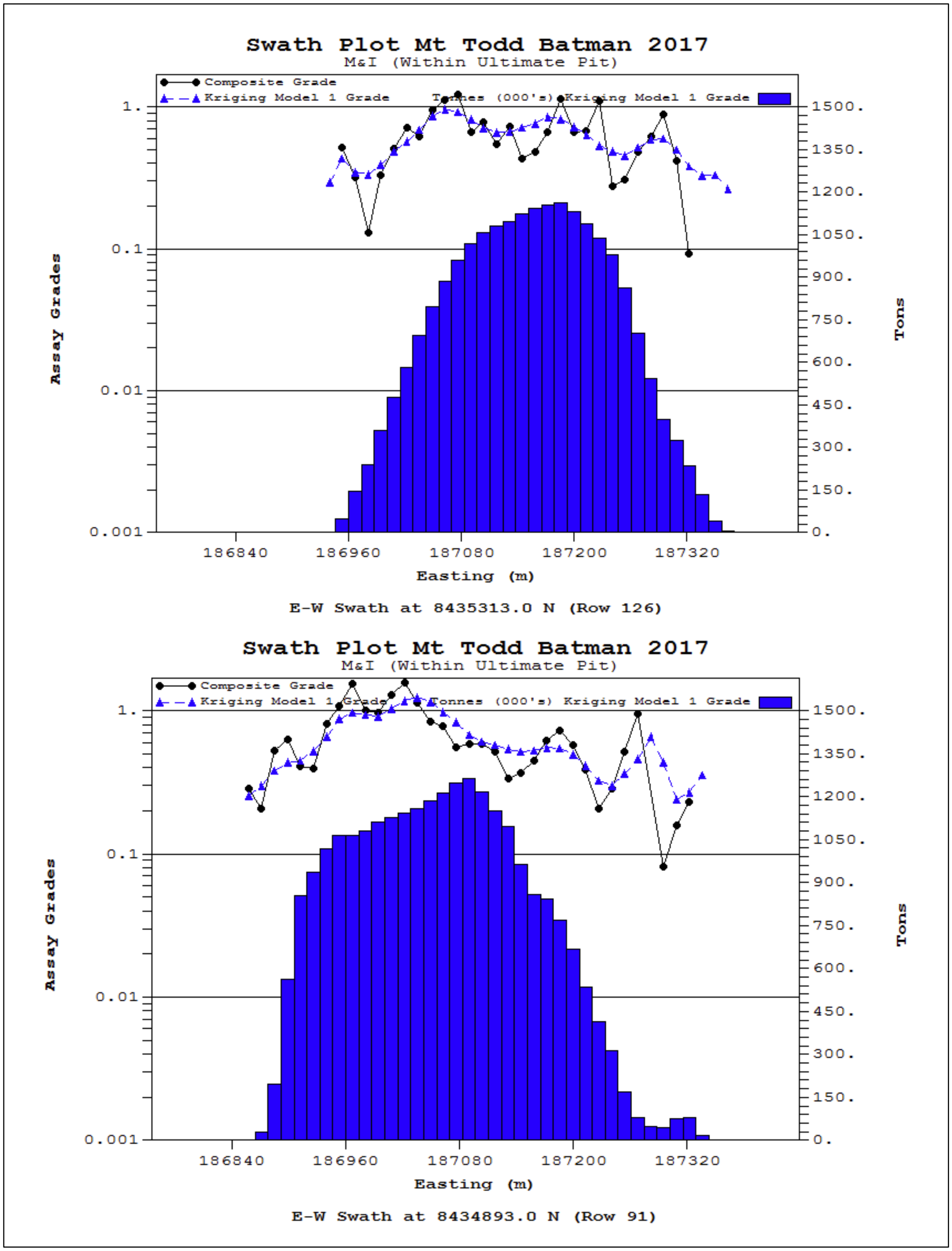


Figure 14-12: Jackknife Correlation Plot for Measured Blocks



Source: Tetra Tech, Inc (August 2017)

Figure 14-13: Jackknife Correlation Plot for Inferred Blocks

14.5 Modeling of the Quigleys Deposit

The Quigleys Deposit is located approximately 3.5 km northeast of the Batman Deposit. The deposit is not as deep as the Batman deposit; it reaches a maximum depth of approximately 200 m. The deposit has been sampled with 57,600 m of drilling by 631 drillholes, with the majority reaching a depth of 100m at a 60 degree dip; oriented 83 degrees azimuth. Assays were taken at a nominal one meter interval. Geologic interpretation in section produced wireframes modeling thin ore zones dipping west. Material inside the wire frames was given a code of 1. Outside the mineralization zones, the material was given a code of 9999.

Bulk density data were supplied by Pegasus for two ore types and waste within the oxide, transition and primary zones, based on a total of 39 samples collected from RC drilling. The two densities supplied were for stockwork and shear, with the density of the shear material substantially higher, particularly in the transition and primary zones. These samples were over one-m to two-m intervals and thus selected the narrow high grade portion of the shear zone as originally interpreted by Pegasus. The final mineralization envelope was much broader than this, and the bulk density was therefore estimated by assuming the final envelope contained 15% shear and 85% stockwork and weighting the density values accordingly. **Table 14-7** shows the specific gravity data assigned to the Quigleys area according to oxidation state.

Table 14-7: Quigleys Deposit Specific Gravity Data

Oxide within modeled shear (t/cm)	2.60
Oxide Waste (t/cm)	2.62
Transition within modeled shear (t/cm)	2.65
Transition Waste (t/cm)	2.58
Primary within modeled shear (t/cm)	2.70
Primary Waste (t/cm)	2.61

14.5.1 Quigleys Exploration Database

Table 14-8 summarizes the Quigleys exploration database.

Table 14-8: Summary of Quigleys Exploration Database

Drillhole Statistics						
	Northing (m) AMG84 z53	Easting (m) AMG84 z53	Elevation (m)	Azimuth	Dip	Depth (m)
Minimum	8,430,1876	188,445.7	129.7	0	45	0
Maximum	8,432,290	189,746.5	209.0	354.0	90	330.5
Average	8,431,129.5	189,230.8	155.9	83.4	62.5	91.3
Range	2,104.0	1,300.8	79.3	354.0	45.0	330.5
Cumulative Drillhole Statistics						
Total Count	631					
Total Length (m)	57,821					
Assay Length (m)	1 (approx.)					
Drillhole Grade Statistics	**Number**	**Average**	**Std. Dev.**	**Min.**	**Max.**	**Missing**
Au (g/t)	52,152	0.2445	0.8764	0	36.00	82
Cu (%)	40,437	0.0105	0.0305	0	2.98	11,897

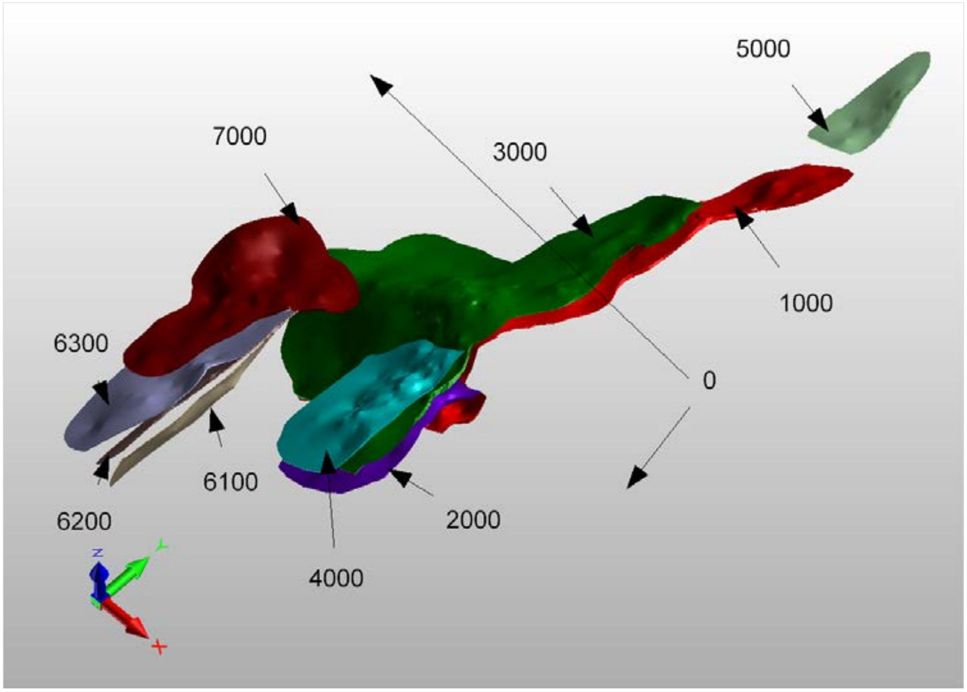
14.5.2 Quigleys Block Model Parameters

Quigleys' block model parameters are shown in **Table 14-9**. The model consisted of 37,082 blocks within the modeled mineralized zones (blocks within the modeled grade zones are coded as 1). Each of the blocks is 250 m^3 (5x25x2m) with a defined density of 2.77 g/cm (692.5 tonnes).

Table 14-9: Block Model Physical Parameters – Quigleys Deposit

Direction	Minimum (m) AMG84 z53	Maximum (m) AMG84 z53	Block Size	# Blocks
x-dir	188,250 mE	189,900 mE	5m	330
y-dir	8,430,337.5 mN	8,432,487.5mN	25m	86
z-dir	-200 m	208m	2m	204

Figure 14-14 shows the rock codes used for the Quigleys estimation.



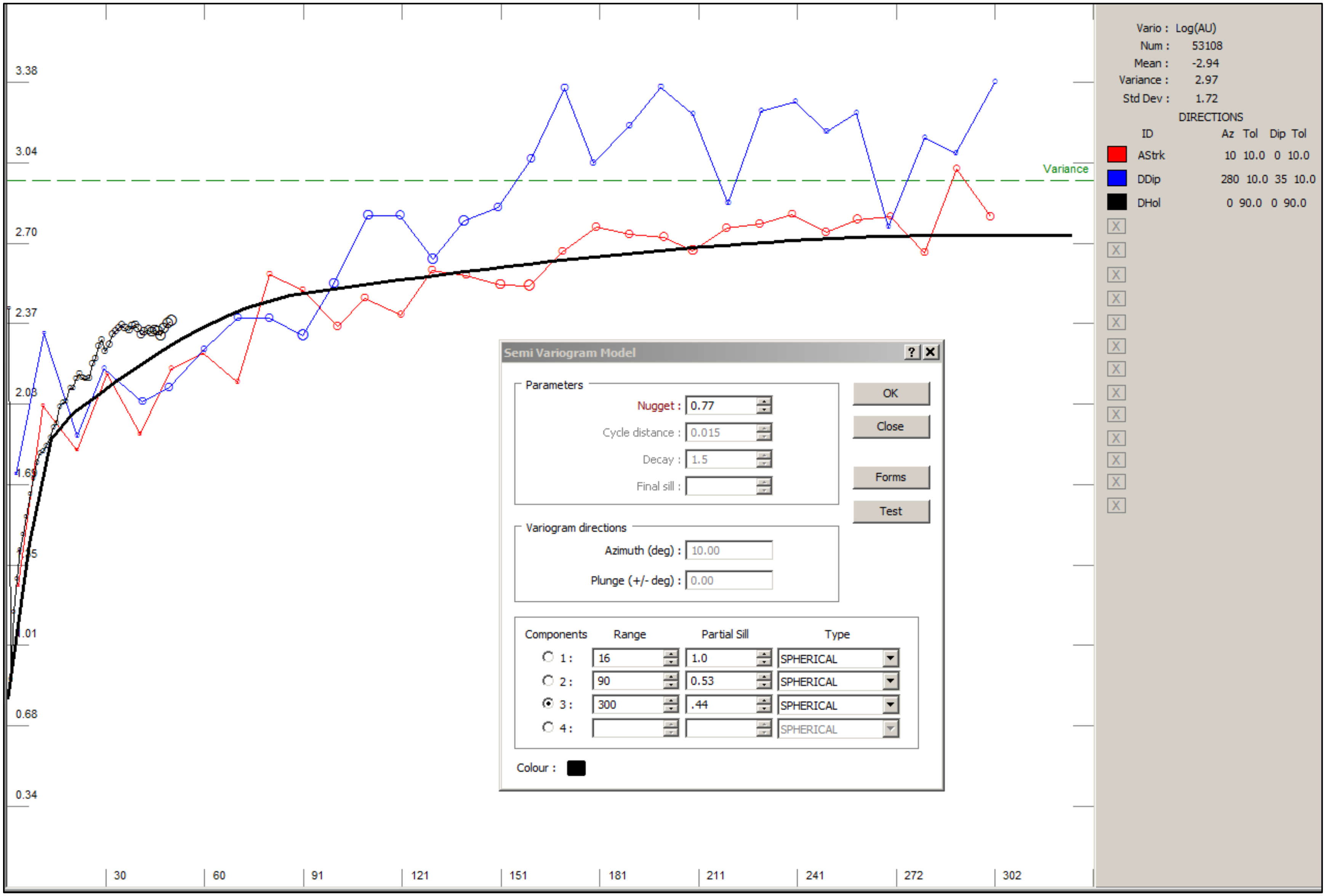
Source: Tetra Tech, Inc (August 2017)

Figure 14-14: 3-D Visualization of the Quigleys Deposit Mineralized Zone Positions with Wireframe Codes

The cap value of 12.0 g-Au/t has been chosen based on review of natural log transformed histograms, cumulative frequency and probability plots. Review of the log probability plot of the composited gold grades shows that there is a distinct break in the distribution at 12 g-Au/t. All gold composites were capped at this value.

Two surfaces were generated based on historic downhole logging of drill holes. The first surface represents the boundary between weathered mineral type (oxide) and transition mineral type (mixed), and the second surface represents the boundary between transition mineral type and fresh mineral type (sulfide).

Figure 14-15 shows the log (Au) variogram for along strike, down dip and down hole coded as AStrk, DDip, Dhole respectively. These variograms have a nugget of 0.77, with an ultimate sill of 2.74. The ranges are 90 meters Along Strike (AStrk) and 30 m Down Dip (DDip). **Table 14-10** shows the search parameters selected for each domain.



Source: Tetra Tech, Inc (August 2017)

Figure 14-15: Quiqleys Median Indicator Variogram

Table 14-10: Search Parameters for each Domain

Code	Azimuth	Dip	Axis1 m	Axis2 m	Axis3 m
0	280	35	90	90	30
1000	266	26	90	90	30
2000	273	35	90	90	30
3000	266	26	90	90	30
4000	273	35	90	90	30
5000	275	30	90	90	30
6100	280	35	90	90	30
6200	280	55	90	90	30
6300	280	70	90	90	30
7000	300	25	90	90	30

Table 14-11 lists the resource classification criteria. The classification was accomplished by a combination of search distance, kriging variance, number of points used in the estimate, and number of sectors used. The block model was estimated using ordinary kriging. The estimation searched for four composites in a sector, allowing a maximum of three composites per drillhole. Inside the ore zone (blocks coded as "1"); composites were selected only if they also were coded as "1". Separate kriging passes were done at increasing search distances. The first pass and second pass restricted points to be within 30 m and 90 m as defined by the search ellipsoid axis to produce provisional resources classes of measured and indicated. Review of the kriging error plotted as a log-probability graph indicated that the gold estimates were particularly poor when kriging variances were greater than 1.0 and 1.55 for the measured and indicated classes respectively. Hence the provisional Measured, Indicated, inferred (MIF) codes were then adjusted to a more restricted class when a blocks kriging error exceeded this value.

Table 14-11: Search Parameters and Sample Restrictions

Domain	Class	Drill Holes	Max Sample Per Drill Hole	Search Major	Search Semi-major	Search Minor	Kriging Error
1000 to 7000	Measured	>= 3	4	30	30	10	<=1.00
1000 to 7000	Indicated	>=2	4	90	90	30	<=1.55
1000 to 7000	inferred	>=1	4	90	90	30	NA
0	inferred	>=2	2	30	30	10	NA

For the outside zone, a two-stage kriging for MIF class 3 was done inside and outside of modelled wireframes with a maximum search ellipse range of 90 m and 30 m respectively.

Each domain was assigned a unique search orientation; however, kriging parameters were the same for all domains. Blocks with a given domain code were estimated only by composites of the same code.

Several methods used to validate the block model were used to determine the adequacy of the Quigleys resource. Cumulative frequency plots of blocks, composites, and assays were overlaid. The three overlaid plots showed the expected decrease in the variability of the gold distributions going from assay to assay composites and then to kriged blocks. Additional verification of the block model was completed by the use of jackknife studies (model validation) where known assays were estimated using surrounding samples, visual inspection of the kriged blocks models in section and plan and the inspection of gold histograms of assays, composites and blocks.

Table 14-12 lists the parameters used to generate a Whittle^{TM} pit shell for reporting the measured plus indicated resource in **Table 14-13** and the inferred resource in **Table 14-14**.

Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 14-16 shows the grade tonnage relations for measured plus indicated classified resource.

Table 14-12: Whittle^{TM} Pit Shell Parameters

Item	Input
Gold Price	US$1,200 per troy ounce
Gold Recovery	82% Sulfide 78% Transition 78% Oxide
Payable Gold	99.90%
Overall Mining Cost	US$1.90 per tonne
Processing Cost	US$9.779 per tonne processed
Tailings	US$0.985 per tonne processed
Water Treatment	US$0.09 per tonne processed
Royalty	1% NPR
Sell Cost	US$3.19

Table 14-13: Quigleys Deposit Measured and Indicated Gold Resource Estimate within M&I Whittle^{TM} Shell (August 2017)

	Cutoff Grade g-Au/t	Tonnes (x1000)	Average Grade g-Au/t	Total Au Ounces (x1000)
Measured + Indicated	2.0	700	3.04	68
	1.5	1,200	2.51	97
	1.0	2,300	1.86	139
	0.4	**6,200**	**1.13**	**225**
	0.0	6,400	1.11	228

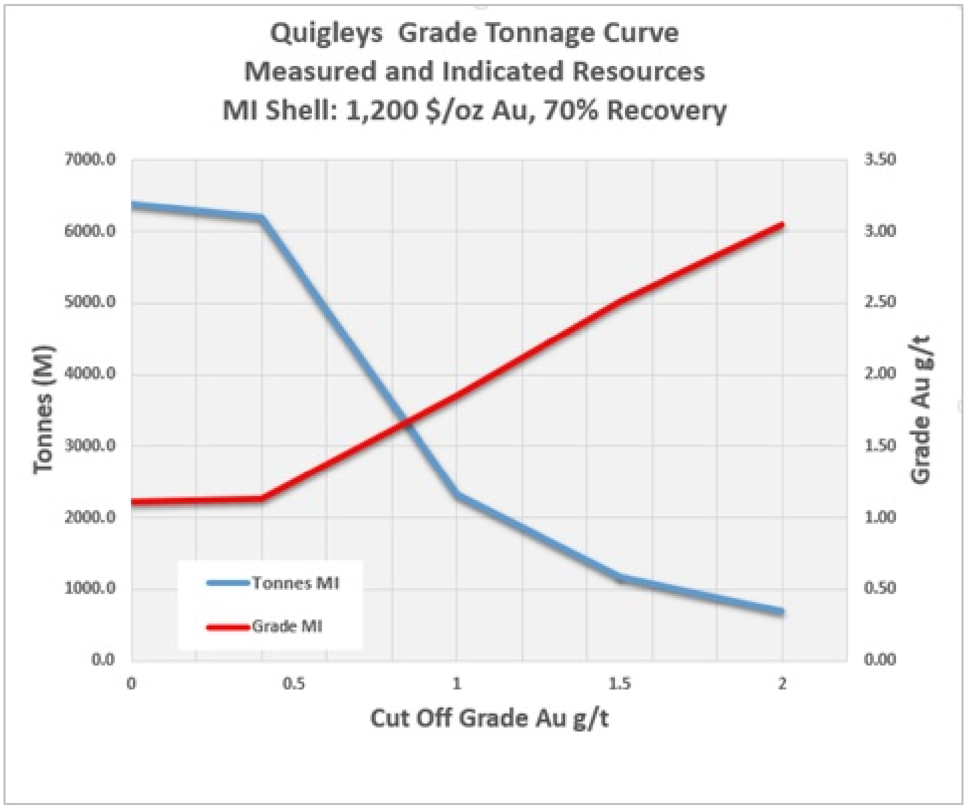
NOTE:

(1) Resources constrained within a US$1,200/oz gold Whittle^{TM} Pit Shell. Pit parameters: Mining cost US$2.07/tonnes, Milling Cost US$9.623/tonnes processed, Sale Cost US$/oz US$15.18, Royalty 1% NPR, Gold Recovery All Types, 70%.

(2) Tonnage, grades and totals may not total due to rounding. The reported resources at a cutoff of 0.4 g/t is highlighted.

(3) There are no mineral reserves at the Quigleys deposit at this time.

(4) For measured and indicated defined at the chosen cutoff grade, reference Table 14-1.



Source: Tetra Tech, Inc (August 2017)
NOTE: Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 14-16: Measured and Indicated Grade Tonnage Curve for the Quigleys Deposit

Table 14-14: Quigleys Deposit Inferred Gold Resource Estimate within M&I Whittle™ Shell (August 2017)

	Cutoff Grade g-Au/t	Tonnes (000s)	Average Grade g-Au/t	Total Au Ounces (000s)
inferred	2.0	91	2.74	8
	1.5	204	2.19	14
	1.0	338	1.79	19
	0.4	**1,600**	**0.84**	**43**
	0.0	20,400	0.18	120

NOTE:
(1) Resources constrained within a US$1,200/oz gold Whittle™ Pit Shell. Pit parameters: Mining cost US$2.07/tonnes, Milling Cost US$9.623/tonnes processed, Sale Cost US$/oz US$15.18, Royalty 1% NPR , Gold Recovery All Types, 70%.
(2) Tonnage, grades and totals may not total due to rounding. The reported resources at a cutoff of 0.4 g/t is highlighted.
(3) There are no mineral reserves at the Quigleys deposit at this time.

14.6 Existing Heap Leach Gold Resource

In addition to the in-situ gold resource for the Batman Deposit, a historical heap leach pad (HLP) adjacent to the current Mt Todd pit was analyzed for gold. The HLP is a remnant of the Pegasus operation, pre-2006. The HLP's geometry was analyzed using historical maps to determine the pile bottom and current surveys of the present day surface. This work produced two surfaces which were used to calculate the volume of the pile. The concentration of gold was analyzed with 24 vertical drillholes separated by an approximately 100 meters. Drilling depth was terminated 5-meters before the final depth of the heap to keep from piercing the bottom liner. The 363 assays from 1-m composites were analyzed for gold and copper grade. Density of the pile was estimated from 11 drillholes using 1,162 dual density sidewall gamma probe technology. Note that the probe uses a gamma source and a scintillation detector to estimate density via the Compton Effect.

A nearest neighbor (polygon) method was employed to estimate grades within the heap leach pad since there is no apparent spatial correlation between samples. The existing heap leach pad is estimated to contain 230,000 ounces of gold within 13.4 Mt of indicated mineral resource at an average grade of 0.54 g-Au/t. It is the opinion of Tetra Tech that the heap leach resource can be classified as an indicated mineral resource as the surveyed volume, the tonnage derived from density measurements, and grade assays from drillhole sampling reconciles with Pegasus' original reported values.

Table 14-15 lists the indicated mineral resources for the existing heap leach pad. Note the grade-tonnage plot in **Figure 14-17**. The majority of tonnage has an average grade of 0.5 g-Au/t as indicated by the flat portion of the tonnage curve. Note too that no cutoff grade was applied to the heap leach pad resource as all material will be processed as part of the site rehabilitation process. Copper was also estimated, but the copper results are not presented here.

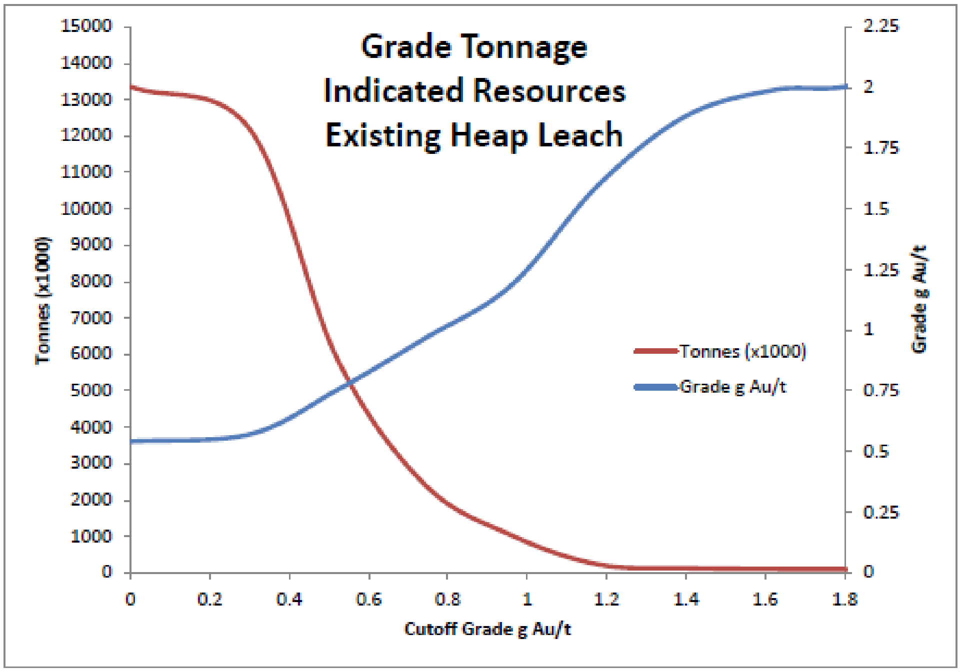
Table 14-15: Existing Heap Leach Indicated Gold Resource Estimate (May 2013)

	Cutoff Grade g-Au/t	Tonnes (000s)	Average Grade g-Au/t	Total Au Ounces (000s)
INDICATED	0	13,400	0.541	230

NOTE:

(1) No cutoff grade is technically applied due to all heap leach material will be re-processed. Resources are reported at 0.4 g/t cutoff gold grade to be consistent with the reported Batman and Quigleys resource is Resource is defined by the geometry of the existing heap leach pad.

(2) Resource & reserve estimates for the heap leach materials are the same because 100% of the heap leach material is processed at the conclusion of mining the Batman Pit.



Source: Tetra Tech, Inc (August 2017)
NOTE: Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 14-17: Inferred Resource Grade Tonnage Curve for the Quigleys Deposit

14.7 Relevant Factors Affecting Resource Estimates

There are currently no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could affect the mineral resource estimate.

15.0 MINERAL RESERVES

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

The measured and indicated resource estimates as of August 2017 were used to estimate reserves.

Reserve definition is done by first identifying ultimate pit limits using economic parameters and pit optimization techniques. The resulting optimized pit shells were then used for guidance in pit design to allow access for equipment and personnel. Several phases of mining were defined to enhance the economics of the project, and MDA used the phased pit designs to define the production schedule to be used for cash-flow analysis for the preliminary feasibility study.

The following section details the definition of reserves used for the production scheduling. Later sections detail the production schedule and the mining costs used in the Tetra Tech cash-flow model.

15.1 Pit Optimization

Pit optimization was done using Geovia's Whittle™ software (version 4.7) to define pit limits with input for economic and slope parameters. The optimization used parameters provided by Vista and their consultants based on current and previous studies.

Optimization used only measured and indicated material for processing. All inferred material was considered as waste.

Varying gold prices were used to evaluate the sensitivity of the deposit to the price of gold, as well as to develop a strategy for optimizing project cash flow. To achieve cash-flow optimization, mining phases or push backs were developed using the guidance of Whittle™ pit shells at lower gold prices.

15.1.1 Economic Parameters

Initially, several iterations of pit optimizations were reviewed for the final determination of pit limits based on the Base Case parameters.

Initial mining cost parameters were based on the economic parameters provided in **Table 15-1.** The final mining costs from this study have turned out to be lower than those in **Table 15-1,** thus making the pit optimization conservative with respect to costs and resulting reserves.

Table 15-1: Initial Economic Parameters

Parameter	Base Case
Gold Recovery	85% Sulfide 80% Transition 80% Oxide
Payable Gold	99.9%
Overall Mining Cost	US$1.90 per tonne
Processing Cost	US$7.80 per tonne processed
Tailings	US$0.90 per tonne processed
General & Administrative	$0.46 per tonne processed

Parameter	Base Case
Water Treatment	US$0.10 per tonne processed
JAAC Royalty	1% gross proceeds

The mining costs used were varied by bench. An incremental cost of US$0.010/tonne was added for each 6-meter bench below the 145 meter elevation. This represents the incremental increase in cost of haulage for both waste and ore for each bench that is to be mined. The incremental cost was determined based on truck operating costs, truck cycle time to haul and return through a six-meter gain in elevation, and truck capacity. The reference mining cost was determined using first principles from previous studies. Reference mining costs of US$1.64/tonne for the Base case. The total mining cost (reference plus incremental) is US$1.90 for the Base Case.

Processing, tailings construction, tailings reclamation, and water treatment costs were provided by Vista and their consultants. Calculated cutoff grades based on the economic parameters are 0.38 and 0.33 g-Au/t for the Alternate and Base cases respectively. At Vista's request, MDA used a minimum cutoff grade of 0.40 g-Au/t for the Base Case. This was done to maintain higher grades with respect to material to be processed.

A base gold price of US$1,250 per ounce was determined by Vista for use in scenario analysis. However, various gold prices from US$300 to US$2,000 per ounce, in increments of US$20 per ounce, were used to determine different optimized pit shells.

Final recoveries were estimated using a constant tail by range of grades for the processed material. The equation used to calculate the recovery based on the constant tail is:

$$\frac{Au_{grade} - Const.Tail_{grade}}{Au_{grade}}$$

The ranges for the constant tail, based on model grade input in g Au/t are:

- 0.20 to 0.40 = 0.04 g Au/t tail
- 0.40 to 0.60 = 0.05 g Au/t tail
- 0.60 to 0.80 = 0.06 g Au/t tail
- 0.80 to 1.00 = 0.08 g Au/t tail
- 1.00 to 1.50 = 0.10 g Au/t tail
- 1.50 and above = 0.13 g Au/t tail

The use of the constant tails resulted in higher final back calculated recoveries of ~92% for sulfide material, ~91% for transition material, and ~90% for oxide material.

Final mining and process operating costs are somewhat higher; however, the imposition of an elevated cutoff grade and selection of pit limits using a lower gold price pit shell create additional conservatism that offset the processing cost and recovery adjustments.

15.1.2 Slope Parameters

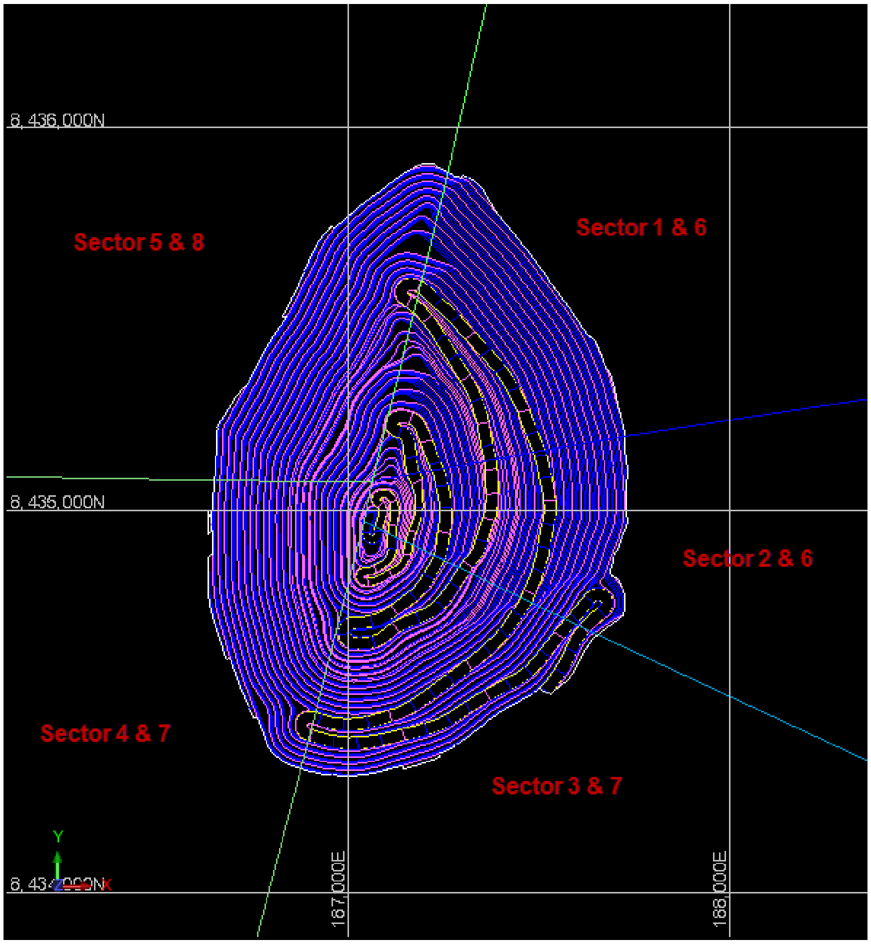
The slope parameters were based on studies provided by Golder Associates and Ken Rippere as detailed in a Golder memo dated September 13, 2011 (*"Mt Todd Gold Project: Batman Pit Slope Design Guidance in Support of the Definitive Feasibility Study"*). Minor modifications were made based on comments from Call & Nicholas, Inc. (2016). The Golder parameters suggested that the catch benches would not be maintainable on the east side of the pit and that these catch benches should not be placed in the design. For safety, the roads on the east side were widened to allow a berm to be maintained along the road to contain any rock that would slough off of the wall.

The primary change suggested by Call & Nicholas (2016) is to either place catch benches in the high wall on the east side, or bolt and mesh the high wall. In both cases the ramp along the wall would be reduced to a normal width.

For this study, catch benches were inserted in preliminary pit phases. However, the ultimate pit used a flat slope with bolting and mesh. This helps to improve the overall slope and the reduce the resulting stripping. **Figure 15-1**. Each sector was modeled into a zone resulting in eight zones. Slopes on the eastern side of the pit were reduced to account for ramps in the high wall. The recommended and adjusted inner-ramp angles are shown in **Table 15-2**.

Table 15-2: Slope Angles for Pit Optimization

Zone	Sector	Slope Angle (°)	Adjusted Angle (°)
1	Northeast	36	33
2	East	40	36
3	South	55	50
4	Southwest	55	55
5	Northwest	51	51
6	Northeast & East Weathered	33	33
7	South & Southwest Weathered	45	45
8	Northwest - Weathered	45	45



From Golder Associates, Technical Memorandum, 9/13/2011: "Mt Todd Gold Project: Batman Pit Slope Design Guidance in Support of the Definitive Feasibility Study"

Figure 15-1: Mt Todd Geotechnical Sectors

15.1.3 Pit-Optimization Results

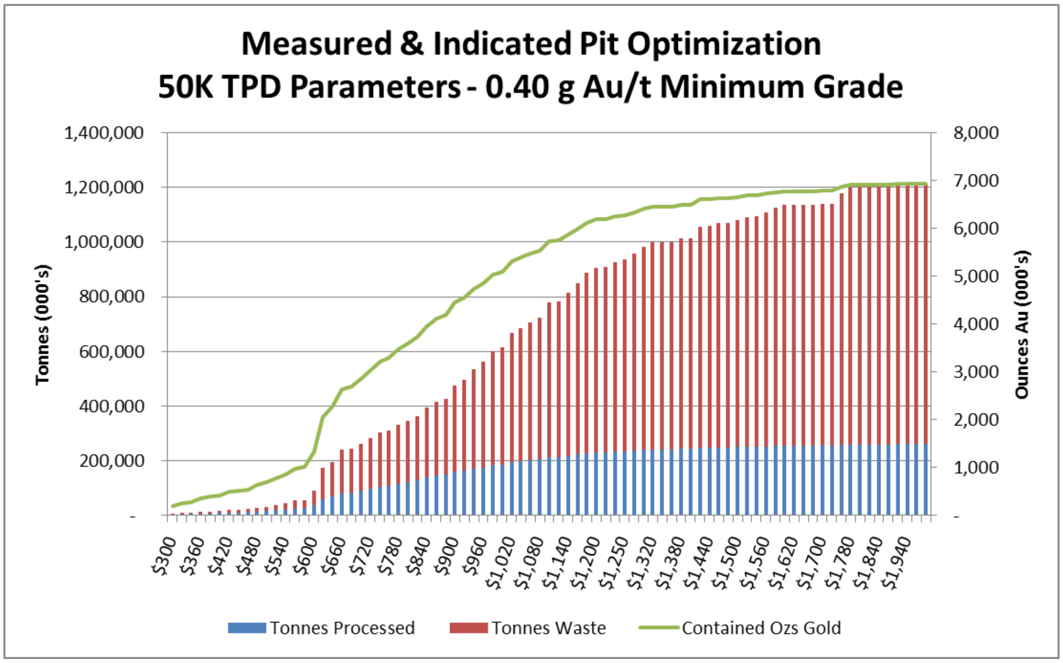
Whittle™ pit optimizations were run using the economic and slope parameters described in previous sections. Pit optimizations were completed using prices of US$300 to US$2,000 per ounce gold with increments of US$20 per ounce. One additional pit shell was created using US$1,250 per ounce gold price. These pits were used to assess the deposit's sensitivity to gold prices for both scenarios. Results for US$100 per ounce increments, from US$300 to US$2,000 per ounce of gold, are shown in **Table 15-3**, with a highlighted price of US$1,000/oz-Au as the pit shell used to guide the ultimate pit design and US$1,250/oz-Au used as the base price for Whittle™ analysis. The pit optimizations only used measured and indicated resources. Inferred materials are considered waste.

Graphs of the tonnes and contained ounces from the Whittle™ results are shown in **Figure 15-3.**

Table 15-3: Whittle™ Pit Optimization Results – Base Case using 0.40 g-Au/t Cutoff

Pit	Gold Price (US$)	Material Processed K Tonnes	g-Au/t	K Ozs Au	Waste Tonnes	Total Tonnes	Strip Ratio
1	$ 300	3,282	1.77	186	2,797	6,078	0.85
6	$ 400	8,578	1.54	425	7,507	16,085	0.88
11	$ 500	15,988	1.34	686	15,740	31,728	0.98
16	$ 600	37,253	1.12	1,340	53,757	91,010	1.44
21	$ 700	89,301	0.99	2,855	171,617	260,918	1.92
26	$ 800	121,187	0.92	3,585	222,919	344,106	1.84
31	$ 900	159,485	0.87	4,442	316,889	476,374	1.99
36	$ 1,000	185,915	0.85	5,093	429,208	615,123	2.31
41	$ 1,100	212,340	0.84	5,741	566,907	779,247	2.67
46	$ 1,200	230,587	0.83	6,184	675,714	906,302	2.93
49	$ 1,250	234,858	0.83	6,278	700,519	935,376	2.98
51	$ 1,300	240,195	0.83	6,416	742,833	983,029	3.09
56	$ 1,400	243,306	0.83	6,498	771,190	1,014,497	3.17
61	$ 1,500	249,389	0.83	6,658	829,933	1,079,321	3.33
66	$ 1,600	254,050	0.83	6,779	880,583	1,134,633	3.47
70	$ 1,700	254,348	0.83	6,785	883,222	1,137,571	3.47
74	$ 1,800	259,140	0.83	6,908	943,012	1,202,152	3.64
78	$ 1,900	259,964	0.83	6,927	952,872	1,212,836	3.67
81	$ 2,000	260,099	0.83	6,929	953,985	1,214,083	3.67

Pit 36 was used for design purposes and Pit 49 illustrates the potential floating cone using a US$1,250/oz-Au price.



NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia – May 29, 2013

Figure 15-2: Measured and Indicated Graph of Whittle™ Results – Base Case using 0.40 g-Au/t Cutoff

15.1.4 Ultimate Pit Limit Selection

The ultimate pit limit was determined based on various iterations analyzed by MDA, Vista, and Tetra Tech. A lower gold price pit shell was used as a guide for the ultimate pit design. This decision was made to reduce the project footprint while still capturing the most valuable material in the pit optimizations.

For consistency, pit shells that closely replicate the tonnage from the previous NI 43-101 Technical Report (Tetra Tech, 2014) were selected for guidance in pit design. MDA selected US$1,000 and US$1,020 gold pit shells for the Base Case.

15.2 Pit Designs

Detailed pit designs were completed, including an ultimate pit and three internal pits for the Base Case. The ultimate pits were designed to allow mining economic resources identified by Whittle™ pit optimization, while providing safe access for people and equipment. Internal pits or phases within the ultimate pits were designed to enhance the project by providing higher-value material to the processing plant earlier in the mine life.

15.2.1 Bench Height

Pit designs used six-meter benches for mining. This corresponds to the resource model block heights, and the QP for this section believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining. In areas where the material is consistently ore or waste, so that dilution is not an issue, benches may be mined in 12-meter heights.

15.2.2 Pit Design Slopes

The slope parameters were based on geotechnical studies by Golder Associates and Ken Rippere (Golder, 2011). These were reviewed by Ross Barkley of Call & Nicholas (Barkley, 2016), and the slope parameters were modified based on his recommendations. The largest difference between the previous and 2017 slopes is in the use of catch benches on the eastern walls. The previous parameters specified the use of a flat wall on the east without any catch benches. To keep rocks from rolling down on trucks, the ramps were designed to be 28 m wide (total width of 50 m), so that a berm could be placed and rock would collect at the base of the slope behind the berm.

Call & Nicholas (2016) commented that *"for interim phases, assuming a 47- to 50-degree bedding dip the interramp angles should be 37- to 39-degrees in order to maintain a 9+ meter wide catch bench"*. For the final walls in the northeast area, the recommendations continue to state *"… the walls will be smooth and excavated to the bedding dip"* and *"To mitigate the rock fall risk in the final wall, it is recommended that mesh be installed over the interramp slopes between the ramps."*

The recommended slopes are developed around five different sectors in fresh rock and three sectors in weathered rock as shown in **Table 15-4**. The design parameters used are shown in **Table 15-4** for the ultimate pit and **Table 15-5** shows the sector 1 and 2 slope parameters for interim pit designs. The parameters are applied based on height between catch benches in meters (BH), safety berm widths in meters (berms), bench face angles in degrees (BFA) and inner-ramp angles also in degrees (IRA).

Table 15-4: Pit Design Slope Parameters

	Due North	Sector 1	Sector 2	Sectors 3 & 4	Sector 5	Sector 6	Sector 7	Sector 8
BH (m)	24	24	24	24	24	30	30	30
BFA (°)	61	47	49	73	68	35	60	60
Berm (m)	9.5	-	-	9.5	9.5	12.0	12.0	12.0
Net IRA (°)	46.5	47.0	49.0	54.9	51.3	28.7	45.7	45.7

In the northern direction the slope azimuth must be 205 degrees or better.

Table 15-5: Interim Pit Slope Parameters (Sectors 1 & 2)

	Northeast	East
BH (m)	24	24
BFA (*)	48	49
Berm (m)	9.5	9.5
Net IRA (*)	37.6	38.3
Zone	1	2

For design purposes, weathered material is considered to be the top 30 meters from the surface.

15.2.3 Haulage Roads

Ramps were designed to have a maximum centerline gradient of 10%. In areas where the ramps may curve along the outside of the pit, the inside gradient may be up to 11% or 12% for short distances. Designs use switchbacks to maintain the ramp system on the east side of the pit. This is done to better match the dip of the deposit and also allows better traffic connectivity between pit phases. In areas where switchbacks are employed, a maximum centerline gradient of 8% is used.

Ramp width was determined as a function of the largest haul truck width to be used. Mine plans use 226-tonne capacity trucks with operating widths of 8.30 meters. For haul roads inside of the pit, a single safety berm on the inside of the roadway will be required to be at least half the height of the largest vehicle tire that uses the road. MDA has designed safety berms with a 1.5 horizontal to 1 vertical slope using run-of-mine material, and a height of 1.97 meters, which provides half of the truck tire height plus 10% for the haul trucks. The 10% addition is used to ensure that the berm height exceeds half of the truck tire height in all cases. The resulting base width of safety berms is 5.9 meters.

Haul-roads inside of the pit, where only one safety berm is required, are designed to be 32 meters wide for two-way traffic. Subtracting berm widths, this provides 3.14 times the width of haul trucks for running width.

In lower portions of the pit, where haulage requirements allow use of one-way traffic, haul roads are designed to have a width of 20 meters. This provides 1.7 times the width of haul trucks for running width.

Haul roads outside of pit designs have been designed to be 42 meters wide to account for an additional safety berm.

15.2.4 Ultimate Pit

The final ultimate pit design uses switchbacks to maintain the ramp system on the east side of the pit. This allows for better traffic flow between pit phases and allows the west side of the pit to best follow the dip of the deposit. In all, there are four switchbacks in the ultimate pit design and the lower portions of the pit have spirals to achieve the ultimate pit design.

The ultimate pit design, along with the ultimate dump and stockpile designs, and planned infrastructure, are shown in **Figure 15-3**.

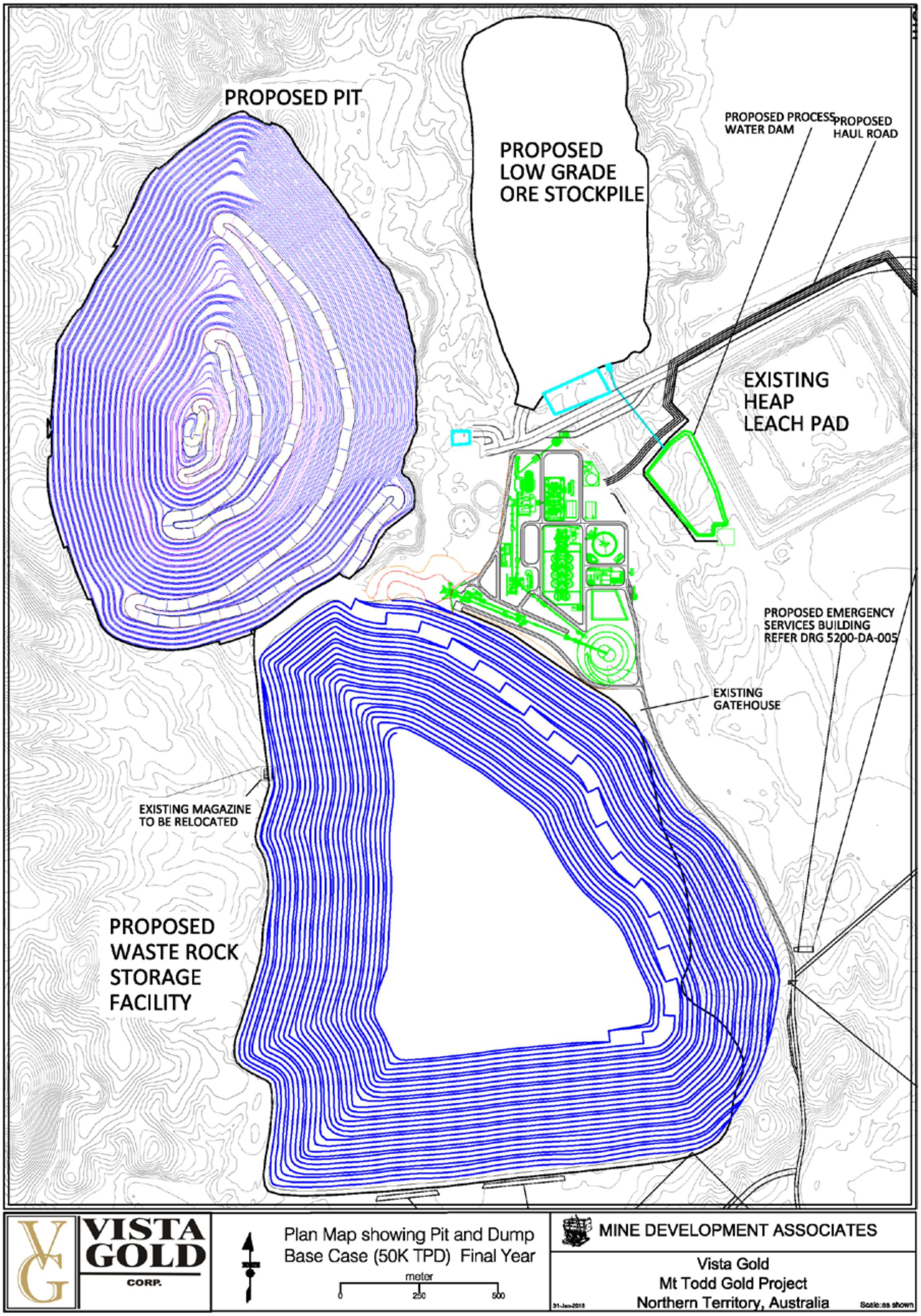


Figure 15-3: Mt Todd Ultimate Pit Design – Base Case (October 4, 2019)

15.2.5 Pit Phasing

Phase 1 continues the western wall down from that done by prior operators, and wraps the ramp around the pit clockwise from the south. Phase 2 expands the pit to the east, north, and south, maintaining a portion of the phase 1 west wall. The phase 2 ramp is placed on the east wall and has a total of five switchbacks located in the north and south ends of the pit. Phase 3 will be mined to the final wall on the western side of the pit. Phase 4 expands the pit to the north, east, and south and mines under the phase 3 pit to the ultimate pit limits. The first three phases, prior to the ultimate pit, are presented in **Figure 15-4** to **Figure 15-6**.

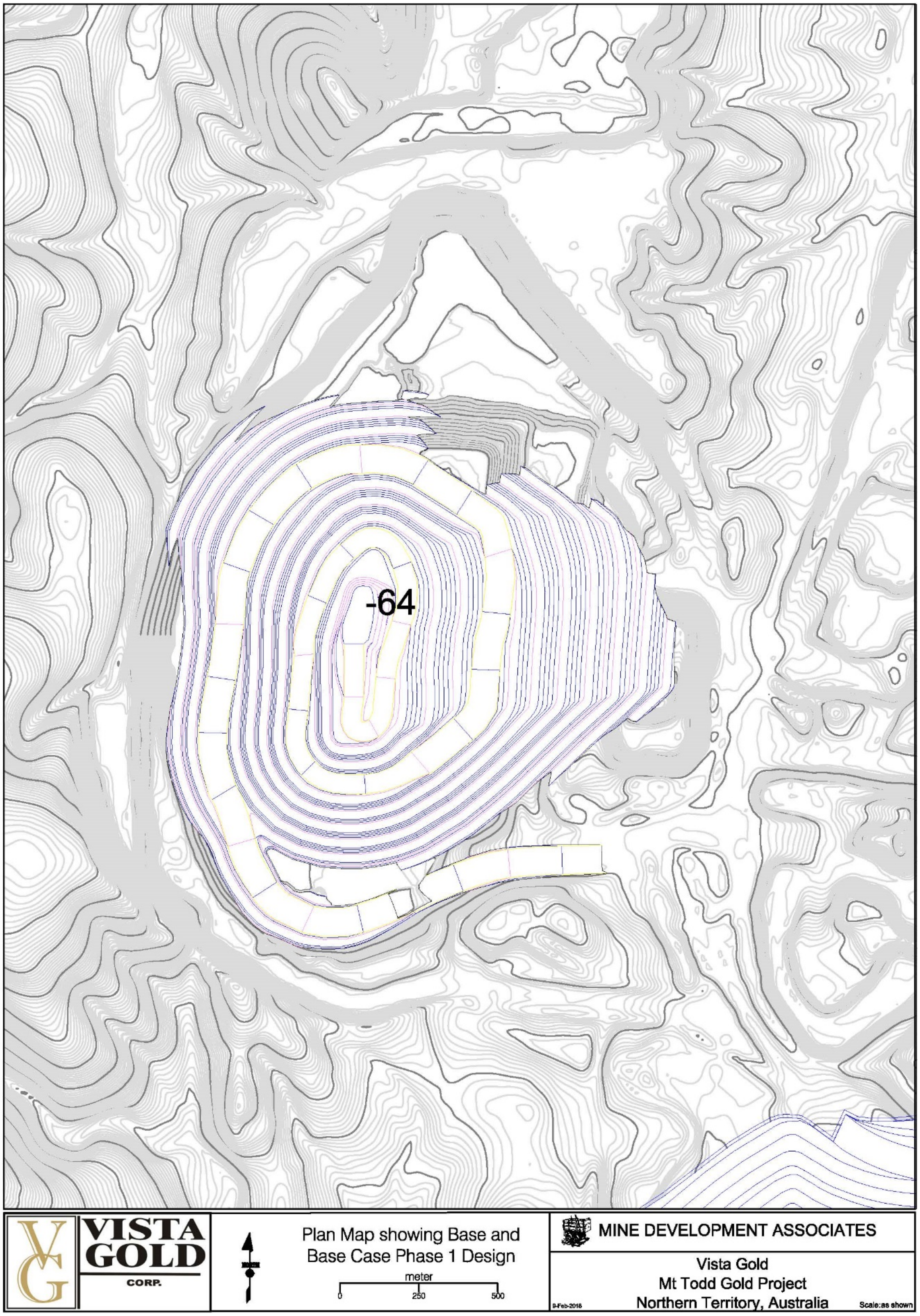


Figure 15-4: Base Case Phase 1 Design (February 12, 2018)

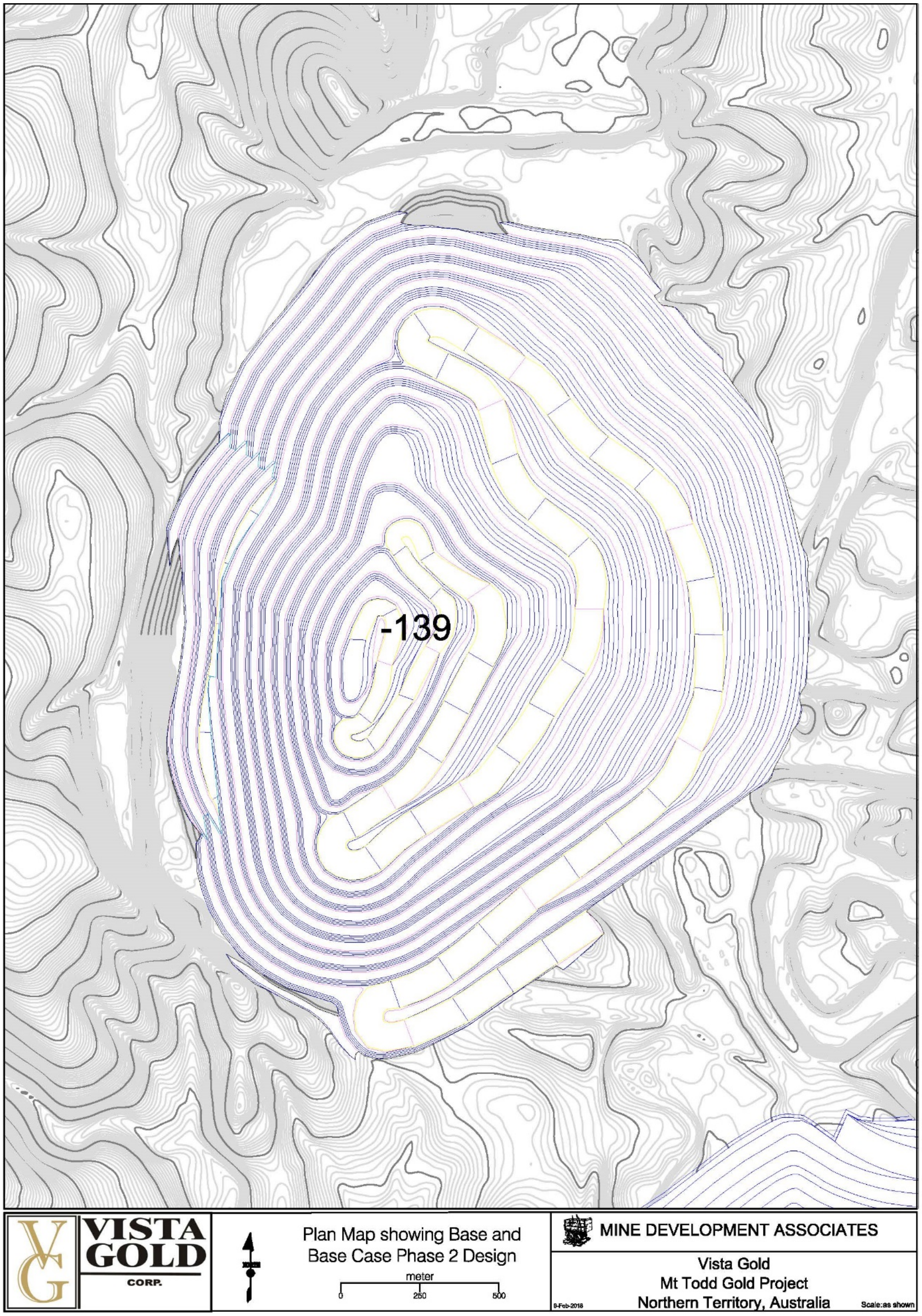


Figure 15-5: Base Case Phase 2 Design (February 12, 2018)

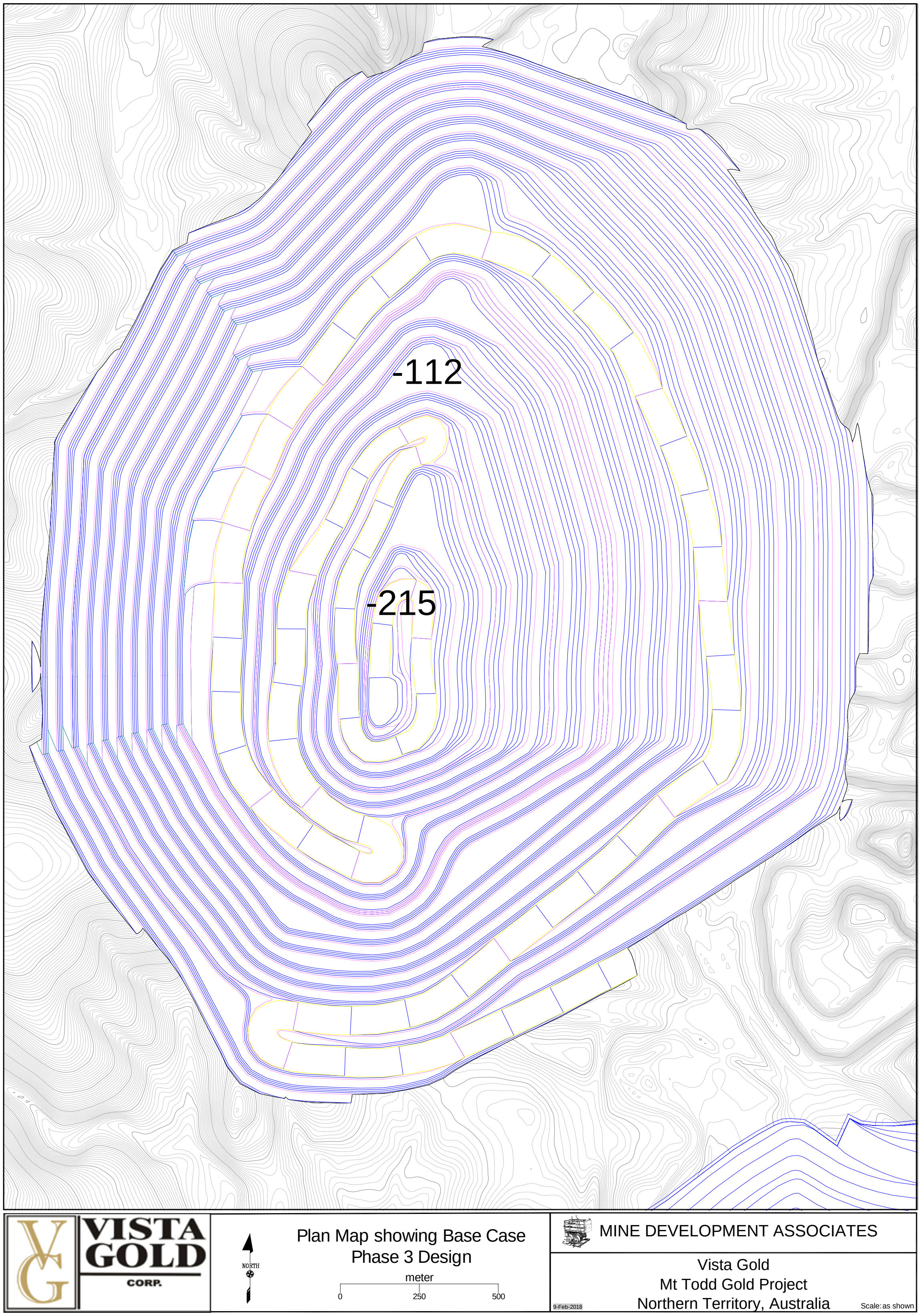


Figure 15-6: Base Case Phase 3 Design (February 12, 2018)

15.3 Cutoff Grade

The breakeven and internal cutoff grades calculated using the economic parameters are shown in **Table 15-6**. The internal cutoff grade assumes that mining is constrained to an economic pit and does not include the mining cost.

To enhance projects economics, Vista has decided to use an elevated cutoff grade for reserves and scheduling. Reserves are reported using 0.40 g-Au/t cutoff grades for the Base Case.

Table 15-6: US$1,250 Gold Price Cutoff Grades (g-Au/t)

	Base Case		
	Sulfide	Transition	Oxide
Breakeven	0.33	0.35	0.35
Internal	0.28	0.29	0.29
Cutoff Grade Used	0.40	0.40	0.40

For purposes of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used the reserve cutoff grade (0.40 g-Au/t). Medium-grade and high-grade cutoffs used were 0.55 and 0.85 g-Au/t, respectively.

15.4 Dilution

The resource block model was estimated with block sizes of 12m by 12m by 6m, and this model was used to define the ultimate pit limit, and to estimate proven and probable reserves. The QP responsible for this section considers the 12m by 12m by 6m block size to be reasonable for open pit mining of the deposit and believes that this represents an appropriate amount of dilution for statement of reserves.

15.5 Reserves

Mineral reserves for the project were developed by applying relevant economic criteria (modifying factors) in order to define the economically extractable portions of the estimated resources. MDA developed the reserves to be in accordance with NI 43-101, which is based on the CIM Standards. CIM Standards define modifying factors as:

> *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.*

CIM Standards define mineral reserves as:

> *Mineral reserves are sub-divided in order of increasing confidence into probable mineral reserves and proven mineral reserves. A probable mineral reserve has a lower level of confidence than a proven mineral reserve.*
>
> *A mineral reserve is the economically mineable part of a measured and/or indicated mineral resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include*

application of modifying factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which mineral reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a mineral reserve must be demonstrated by a pre-feasibility study or feasibility study.

Mineral reserves are those parts of mineral resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the qualified person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant modifying factors. Mineral reserves are inclusive of diluting material that will be mined in conjunction with the mineral reserves and delivered to the treatment plant or equivalent facility. The term 'mineral reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

'Reference point' refers to the mining or process point at which the qualified person prepares a mineral reserve. For example, most metal deposits disclose mineral reserves with a "mill feed" reference point. In these cases, reserves are reported as mined ore delivered to the plant and do not include reductions attributed to anticipated plant losses. The qualified person must clearly state the 'reference point' used in the mineral reserve estimate.

Probable Mineral Reserve

A probable mineral reserve is the economically mineable part of an indicated, and in some circumstances, a measured mineral resource. The confidence in the modifying factors applying to a probable mineral reserve is lower than that applying to a proven mineral reserve.

The qualified person(s) may elect, to convert measured mineral resources to probable mineral reserves if the confidence in the modifying factors is lower than that applied to a proven mineral reserve. Probable mineral reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.

Proven Mineral Reserve

A proven mineral reserve is the economically mineable part of a measured mineral resource. A proven mineral reserve implies a high degree of confidence in the modifying factors.

Application of the proven mineral reserve category implies that the qualified person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven mineral reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a pre-feasibility study. Within the CIM Definition standards the term proved mineral reserve is an equivalent term to a proven mineral reserve.

Proven and probable reserves are stated based on the Base Case pit designs. **Table 15-7** reports the proven and probable reserves, by pit phase, along with waste material for the pit designs discussed in previous sections.

RDi is responsible for reporting of the heap-leach pad reserves (**Section 15.7 – Heap Leach Reserve**). This is based on the tonnage and grade of heap-leach material that was loaded onto a heap-leach pad by a historical operator. The tonnes and grades are well known based on record keeping of the historical operator as discussed in **Section 14.0 – Mineral Resource Estimate**. The heap-leach reserves are shown with the Batman reserves in **Table 15-8**.

The Base Case reserves are shown to be economically viable based on cash flow analysis provided by Tetra Tech. The QP responsible for this section has reviewed the cash flow and believes that they are reasonable for the statement of proven and probable reserves.

Table 15-7: Base Case Proven and Probable Reserves by Pit Phase

	Proven			Probable			Total P&P			Waste	Total	Strip
	K Tonnes	g Au/t	K Ozs Au	K Tonnes	g Au/t	K Ozs Au	K Tonnes	g-Au/t	K Ozs Au	K Tonnes	K Tonnes	Ratio
Ph_1	13,551	1.09	473	6,245	1.10	221	19,796	1.09	694	19,312	39,109	0.98
Ph_2	18,980	0.80	490	19,008	0.88	538	37,988	0.84	1,028	59,167	97,155	1.56
Ph_3	20,356	0.87	571	27,471	0.85	747	47,827	0.86	1,318	122,372	170,198	2.56
Ph_4	19,785	0.82	523	82,291	0.78	2,052	102,076	0.78	2,576	322,138	424,215	3.16
Total	**72,672**	**0.88**	**2,057**	**135,015**	**0.82**	**3,559**	**207,687**	**0.84**	**5,616**	**522,990**	**730,677**	**2.52**

NOTES:
(1) Proven and probable mineral reserves are reported using a cutoff grade of 0.40 g-Au/t.
(2) The reserves point of reference is the point where material is fed into the mill.

Table 15-8: Total Batman Project Reserves (Base Case plus Heap Leach)

	Batman Deposit (January 2018)			Heap Leach Pad (May 2013)			Total P&P Reserves (January 2018)		
	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)
Proven	72,672	0.88	2,057	-	-	-	72,672	0.88	2,057
Probable	135,015	0.82	3,559	13,354	0.54	232	148,369	0.79	3,791
Proven & Probable	**207,687**	**0.84**	**5,616**	**13,354**	**0.54**	**232**	**221,041**	**0.82**	**5,848**

NOTES:
(1) Thomas L. Dyer, P.E., is the QP responsible for reporting the Batman deposit proven and probable reserves.
(2) Batman deposit reserves are reported using a 0.40 g-Au/t cutoff grade.
(3) Deepak Malhotra is the QP responsible for reporting the heap-leach pad reserves.
(4) Because all of the heap-leach pad reserves are to be fed through the mill, these reserves are reported without a cutoff grade applied.
(5) The reserves point of reference is the point where material is fed into the mill.

15.6 In-Pit inferred Resources

Inferred resources were considered as waste and not used in the economic analysis. Note that CIM Standards define inferred resources as:

> *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 drill holes. 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.*

Table 15-9 shows the inferred resources inside of the pit designs for each phase. For statement of reserves and economic analysis, inferred resources in the pit are reported as waste.

Table 15-9: In-Pit Inferred Resources Inside Base Case Pits

	In-Pit Inferred Resource		
	K Tonnes	**g-Au/t**	**K Ozs Au**
Ph_1	**992**	**0.58**	**18**
Ph_2	3,868	0.60	75
Ph_3	3,048	0.57	56
Ph_4	6,585	0.58	123
Total	**14,494**	**0.58**	**272**

NOTE:
(1) Base Case inferred resources are reported using a cutoff grade of 0.40 g-Au/t
(2) Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

15.7 Heap Leach Reserve Estimate

Heap leach reserves are provided in **Table 15-8**. In addition to the ore mined from the Batman open pit, the mine plan contemplates processing the 13.4 Mt of ore from the existing heap leach pad through the mill at the end of the mine life.

The bottle roll and column leach test work undertaken at the ALS Metallurgy Laboratory in Australia has been reviewed (ALS, 2013). The testwork indicated the following:

- Cyanidation leach tests on "as is" material on the heap will extract ± 30% of the gold.
- CIP cyanidation tests at a grind size of P_{80} of 90 microns will extract on average 72% of gold (range: 64.14% to 80.37%) in 24 hours of leach time. The average lime and cyanide consumptions were 1.75 kg/t and 0.78 kg/t, respectively.

The limited testwork indicates that it is economically feasible to process and recover gold from the heap leach material. Hence, the 13.4 Mt of heap leach ore meets the criteria necessary to be called "reserves" for the Mt Todd Gold Project and should be included in the reserve tabulation based on the following:

- The heap leach material is already mined;
- The contained gold is readily recoverable using the planned flowsheet; and
- The heap leach material can be economically processed in the plant which will be built to process fresh ore.

These reserves should be considered as probable, since limited drilling and assaying was undertaken to estimate the gold content of the heap leach residues.

16.0 MINING METHODS

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

16.1 Methods

The Project has been planned as an open-pit truck and shovel operation. The truck and shovel method provides reasonable cost benefits and selectivity for this type of deposit. Only open-pit mining methods are considered for mining at Mt Todd.

The mining method and approach are based on the geotechnical and hydrological data and studies described in **Sections 15.1.2 – Slope Parameters**, **Section 15.2.2 – Pit Design Slopes**, and **Section 0– Regional Groundwater Model and Mine Dewatering**.

16.2 Site Landforms and Impoundments

For reference, a description of the site landforms and impoundments, as well as their naming conventions and abbreviations is included as **Table 16-1**.

Table 16-1: Description of Landforms and Impoundments

Landform/Impoundment	Abbreviated Name
Tailings Storage Facility 1	TSF 1
Tailings Storage Facility 2	TSF 2
Raw Water Dam	RWD
Low Grade Ore Stockpile	LGOS
Low Grade Ore Stockpile Retention Pond	LGRP
Heap Leach Pad	HLP
Batman Pit	RP3
Process Plant Retention Pond	PRP
Waste Rock Dump	WRD
Waste Rock Dump Retention Pond	RP1
Process Water Pond	PWP
Water Treatment Plant	WTP
Process Plant	PP

16.3 Waste Material Definition

Some of the waste material at Mt Todd contains sulfide minerals, which can result in acid generation. Tetra Tech provided MDA with classification criteria for waste material so that the resulting production schedule can include the segregation of waste types for proper handling. Waste was classified into three classes based on total sulfur content as follows:

- Non-PAG Total Sulfur <= 0.25%
- Uncertain Total Sulfur > 0.25% and <= 0.40%
- PAG Total Sulfur > 0.40%

Material classified as uncertain or potentially acid generating (PAG) material was scheduled so that it could be placed inside of the ultimate waste dump. Non-potentially acid generating material (Non-PAG) was scheduled to be used to encapsulate the uncertain and PAG material, and for reclamation cover and construction material for TSF 1 and TSF 2. Due to the scheduled timing of mining of Non-PAG material, some of this material will be stockpiled on the outer portions of the waste dump and re-handled as required.

16.4 Mine-Waste Facilities

Total contained waste tonnage is 523 million tonnes for the Base Case.

For both cases, Non-PAG mine waste will be used for construction and final reclamation cover on the mine site. The construction material will be used at the tailings storage facilities ("TSF 1" and "TSF 2"). Other Non-PAG material will be used for reclamation purposes covering tailings and other facilities at the end of the mine life. Sorter tailings will be generated from the process plant sorter and either used for ongoing construction uses or hauled to a temporary stockpile near the sorter. This material is considered Non-PAG and will be re-handled at the end of the mine life as part of the reclamation material.

Tetra Tech provided the amount of material that would be required to be mined for construction and reclamation. These totals are shown in **Table 16-2** for the Base Case.

Table 16-2: Base Case Construction and Reclamation Requirements

	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
CONSTRUCTION MATERIAL REQUIREMENTS																	
Total TSF 1	K m^3	675	625	-	-	210	-	219	222	-	-	235	235	367	-	-	2,787
	K Tonnes	1,350	1,250	-	-	420	-	438	443	-	-	470	470	734	-	-	5,575
Total TSF 2	K m^3	-	-	3,381	2,902	1,878	1,348	1,919	1,387	3,296	-	2,935	2,904	218	-	-	22,169
	K Tonnes	-	-	6,763	5,804	3,757	2,697	3,837	2,774	6,592	-	5,869	5,808	437	-	-	44,337
Total - All TSFs	**K m^3**	**675**	**625**	**3,381**	**2,902**	**2,088**	**1,348**	**2,137**	**1,609**	**3,296**	**-**	**3,169**	**3,139**	**586**	**-**	**-**	**24,956**
	K Tonnes	**1,350**	**1,250**	**6,763**	**5,804**	**4,176**	**2,697**	**4,275**	**3,218**	**6,592**	**-**	**6,339**	**6,278**	**1,171**	**-**	**-**	**49,912**
RECLAMATION MATERIAL REQUIREMENTS																	
Sorter Reject to TSF 1																10,771	10,771
Sorter Reject to TSF 2																9,997	9,997
Total Sorter Reject Re-handle		-	-	-	-	-	-	-	-	-	-	-	-	-	-	20,769	20,769
TSF 1_Closure		-	-	-	-	-	-	-	-	-	-	-	-	-	135	3,750	3,885
TSF 2_Closure		-	-	-	-	-	-	-	-	-	-	-	-	-	664	3,506	4,170
Total Non-PAG		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**799**	**48,793**	**49,593**

The mine waste facility has been designed to permanently contain the remaining waste material associated with reserves in the pits for both cases. This facility is an extension of the existing waste dump at site with the ultimate dump fully encapsulating the current dump. The ultimate design incorporates an angle of repose slope of 1.5 vertical to 1.0 horizontal, with catch benches of 8.0 meters every 30 meters in height. During the construction of the ultimate dump, PAG and uncertain waste materials will be dumped in the interior of each lift of the waste dump. Non-PAG material will be dumped to the outer edge of each lift. It is anticipated that at least a 10 meter rind of Non-PAG material will surround uncertain and PAG type waste material.

For closure of the waste rock facility reference **Section 20.0 – Environmental Studies, Permitting, and Social or Community Impact.**

A 40% swell factor and an average specific gravity of 2.67 (bank) have been assumed for volume calculations. The Base Case dump design has a total capacity to contain 485 million tonnes, but it is over designed and only about 440 million tonnes of the capacity will be used. Prior to filling into the final footprint of the waste dump, the design should be optimized to minimize the footprint and minimize reclamation requirements.

16.5 Mine-Production Schedule

Proven and probable reserves and the associated waste material were used to schedule mine production. Inferred resources inside of the pit were considered as waste. The final production schedule uses the number of trucks and shovels necessary to produce the ore required to be fed into the process plant and maintain stripping requirements for each case.

Production scheduling was done using MineSched (version 9.1). This was summarized in Excel spreadsheets where additional waste re-handling was added to the schedule. **Table 16-3** shows the mine production schedule, including re-handle from stockpiles, waste material re-handle, and sorter stockpile material movement requirements for the Base Case. For the purpose of production scheduling, low-grade, medium-grade, and high-grade ore was designated. The medium-grade and high-grade cutoffs used 0.55 and 0.85 g-Au/t, respectively.

Ore from the mine is to be sent from the pit directly to the crusher, or to a mill ore stockpile. During pre-stripping, high-grade, medium-grade, and low-grade ore is stockpiled in the stockpile area northeast of the waste dump facility. Low-grade ore is processed as part of the commissioning of the mill. This assumes a ramp up to full production of 25%, 50%, 75%, and 87.5% of full production throughput through the first 4 months prior to start of full production. High-grade and medium-grade ore is processed in the mill when mill capacity becomes fully available.

For the purpose of scheduling, three ore stockpiles are assumed:

- High-grade ore stockpile (> 0.85 g-Aug/t;
- Medium-grade stockpile (0.55 to 0.85 g-Au/t); and
- Low-grade stockpile (0.40 to 0.55 g-Au/t).

The high-grade and medium-grade stockpiles are to be built within the low-grade stockpiling areas but will be exhausted during the first year of processing when mill capacity becomes available. During the life of mine, the low-grade stockpile is to be used as needed to feed the mill to full capacity. For this reason, the stockpile grows and shrinks through the life of mine. The maximum stockpile balance through the life of mine is estimated to be 21.0 million tonnes for the Base Case. Ultimate stockpile designs have been created north-east of the processing facility.

Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. **Table 16-4** shows the yearly ore stockpile balances.

Ore sent to the mill is shown in **Table 16-5** and is a combination of ore shipped directly from the mine, and ore that is reclaimed from stockpiles. Ore sent to the mill is summarized based on the level of oxidation. The recovered ounces shown are based on the recoveries used for pit optimizations.

Table 16-3: Annual Mine Production Schedule – Base Case

			Pre-Prod	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	Total
Total Mined	*StkPl	K Tonnes	2,859	7,302	5,283	6,699	6,354	12,102	235	-	-	1,000	10,903	8,172	-	-	-	60,908
		g Au/t	0.77	0.94	0.47	0.52	0.47	0.63	0.48	-	-	0.53	0.60	0.65	-	-	-	0.63
		K Ozs Au	71	220	80	113	96	244	4	-	-	17	211	172	-	-	-	1,227
	Crusher	K Tonnes	-	8,836	10,330	17,795	9,232	17,750	8,749	7,178	13,482	17,750	17,750	17,799	127	-	-	146,779
		g Au/t	-	1.26	0.86	1.04	0.85	1.10	0.87	0.64	0.63	0.70	0.93	1.18	1.05	-	-	0.93
		K Ozs Au	-	358	286	592	252	629	246	149	274	397	528	674	4	-	-	4,389
	Total Ore Mined	**K Tonnes**	2,859	16,138	15,613	24,495	15,586	29,852	8,984	7,178	13,482	18,750	28,653	25,970	127	-	-	207,687
		g Au/t	0.77	1.11	0.73	0.90	0.69	0.91	0.86	0.64	0.63	0.69	0.80	1.01	1.05	-	-	0.84
		K Ozs Au	71	578	366	705	348	872	249	149	274	414	739	846	4	-	-	5,616
	Mineralized Waste	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		g Au/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		K Ozs Au	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	NonPag_Wst	K Tonnes	1,720	150	20,484	8,711	20,829	22,261	58,227	37,390	18,570	6,948	1,496	24	-	-	-	196,810
	Pag_Wst	K Tonnes	5,433	8,897	18,469	17,473	36,445	27,894	17,272	18,276	24,290	25,310	21,567	3,887	-	-	-	225,212
	Un_Wst	K Tonnes	1,650	1,452	8,583	6,696	19,257	7,930	11,512	12,552	13,738	10,678	6,684	237	-	-	-	100,968
	Total Waste Mined	**K Tonnes**	**8,802**	**10,498**	**47,536**	**32,880**	**76,531**	**58,085**	**87,011**	**68,218**	**56,598**	**42,935**	**29,747**	**4,148**	**-**	**-**	**-**	**522,990**
	Total Tonnes Mined	**K Tonnes**	**11,661**	**26,636**	**63,149**	**57,375**	**92,117**	**87,937**	**95,995**	**75,396**	**70,080**	**61,685**	**58,400**	**30,119**	**127**	**-**	**-**	**730,677**
	Strip Ratio	W:O	3.08	0.65	3.04	1.34	4.91	1.95	9.69	9.50	4.20	2.29	1.04	0.16	-			2.52
Re-Handle Material	HG_StkPl	K Tonnes	-	1,419	3,212	3	265	-	1,626	-	-	-	-	-	2,288	-	-	8,813
		g Au/t	-	1.40	1.25	1.39	1.16	-	1.13	-	-	-	-	-	1.06	-	-	1.20
		K Ozs Au	-	64	129	0	10	-	59	-	-	-	-	-	78	-	-	340
	MG_StkPl	K Tonnes	-	2,041	39	-	744	-	4,111	-	-	-	-	-	8,497	-	-	15,432
		g Au/t	-	0.69	0.62	-	0.67	-	0.67	-	-	-	-	-	0.66	-	-	0.67
		K Ozs Au	-	45	1	-	16	-	88	-	-	-	-	-	181	-	-	331
	LG_StkPl	K Tonnes	-	165	4,168	-	7,509	-	3,264	10,620	1,647	-	-	-	6,838	2,451	-	36,663
		g Au/t	-	0.53	0.50	-	0.50	-	0.51	0.44	0.43	-	-	-	0.49	0.42	-	0.47
		K Ozs Au	-	3	67	-	121	-	53	149	23	-	-	-	107	33	-	556
	Leach Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	13,354	-	13,354
		g Au/t	-	-	-	-	-	-	-	-	-	-	-	-	-	0.54	-	0.54
		K Ozs Au	-	-	-	-	-	-	-	-	-	-	-	-	-	232	-	232
	Total Re-Handle	**K Tonnes**	**-**	**3,625**	**7,420**	**3**	**8,518**	**-**	**9,001**	**10,620**	**1,647**	**-**	**-**	**-**	**17,623**	**15,805**	**-**	**74,262**
		g Au/t	**-**	**0.96**	**0.82**	**1.39**	**0.54**	**-**	**0.69**	**0.44**	**0.43**	**-**	**-**	**-**	**0.65**	**0.52**	**-**	**0.61**
		K Ozs Au	**-**	**112**	**197**	**0**	**147**	**-**	**200**	**149**	**23**	**-**	**-**	**-**	**366**	**265**	**-**	**1,459**
	Waste Re-handle	K Tonnes	-	1,555	603	3,848	435	877	-	-	-	-	4,842	6,253	1,171	1,480	7,256	28,324
	Sorter Rejects	K Tonnes	-	1,246	1,775	1,780	1,775	1,775	1,775	1,780	1,513	1,775	1,775	1,780	1,775	245	-	20,769
	Sorter Reject Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	20,769	20,769

Table 16-4: Annual Stockpile Balance – Base Case

			Pre-Prod	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
Hg_StkPl	Added	K Tonnes	858	3,773	-	268	-	1,626	-	-	-	53	1,106	1,129	-	-
		g Au/t	1.13	1.33	-	1.17	-	1.13	-	-	-	1.08	1.07	1.06	-	-
		K Ozs Au	31	161	-	10	-	59	-	-	-	2	38	38	-	-
	Removed	K Tonnes	-	1,419	3,212	3	265	-	1,626	-	-	-	-	-	2,288	-
		g Au/t	-	1.40	1.25	1.39	1.16	-	1.13	-	-	-	-	-	1.06	-
		K Ozs Au	-	64	129	0	10	-	59	-	-	-	-	-	78	-
	Balance	K Tonnes	858	3,212	-	265	-	1,626	-	-	-	53	1,159	2,288	-	-
		g Au/t	1.13	1.25	-	1.16	-	1.13	-	-	-	1.08	1.07	1.06	-	-
		K Ozs Au	31	129	-	10	-	59	-	-	-	2	40	78	-	-
Mg_StkPl	Added	K Tonnes	1,262	818	-	744	-	4,111	-	-	-	225	4,167	4,105	-	-
		g Au/t	0.69	0.68	-	0.67	-	0.67	-	-	-	0.67	0.66	0.66	-	-
		K Ozs Au	28	18	-	16	-	88	-	-	-	5	88	88	-	-
	Removed	K Tonnes	-	2,041	39	-	744	-	4,111	-	-	-	-	-	8,497	-
		g Au/t	-	0.69	0.62	-	0.67	-	0.67	-	-	-	-	-	0.66	-
		K Ozs Au	-	45	1	-	16	-	88	-	-	-	-	-	181	-
	Balance	K Tonnes	1,262	39	-	744	-	4,111	-	-	-	225	4,392	8,497	-	-
		g Au/t	0.69	0.62	-	0.67	-	0.67	-	-	-	0.67	0.66	0.66	-	-
		K Ozs Au	28	1	-	16	-	88	-	-	-	5	93	181	-	-
Lg_StkPl	Added	K Tonnes	738	2,711	5,283	5,688	6,354	6,365	235	-	-	722	5,630	2,938	-	-
		g Au/t	0.48	0.47	0.47	0.48	0.47	0.47	0.48	-	-	0.45	0.47	0.48	-	-
		K Ozs Au	11	41	80	87	96	97	4	-	-	10	85	45	-	-
	Removed	K Tonnes	-	165	4,168	-	7,509	-	3,264	10,620	1,647	-	-	-	6,838	2,451
		g Au/t	-	0.53	0.50	-	0.50	-	0.51	0.44	0.43	-	-	-	0.49	0.42
		K Ozs Au	-	3	67	-	121	-	53	149	23	-	-	-	107	33
	Balance	K Tonnes	738	3,284	4,399	10,086	8,931	15,297	12,267	1,647	-	722	6,351	9,290	2,451	-
		g Au/t	0.48	0.47	0.44	0.46	0.43	0.45	0.44	0.43	-	0.45	0.47	0.47	0.42	-
		K Ozs Au	11	50	63	150	124	221	172	23	-	10	95	141	33	-
All StkPl	Balance	K Tonnes	2,859	6,536	4,399	11,095	8,931	21,033	12,267	1,647	-	1,000	11,902	20,074	2,451	-
		g Au/t	0.77	0.85	0.44	0.49	0.43	0.54	0.44	0.43	-	0.53	0.60	0.62	0.42	-
		K Ozs Au	71	179	63	175	124	368	172	23	-	17	228	400	33	-

Table 16-5: Annual Ore Delivery to the Mill Crusher – Base Case

		Pre-Prod	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	Total
Sulfide Ore	K Tonnes	-	12,461	17,631	17,384	17,281	16,974	17,185	17,050	15,015	17,750	17,750	17,799	17,623	2,451	204,354
	g Au/t	-	1.17	0.85	1.04	0.70	1.12	0.79	0.52	0.61	0.70	0.93	1.18	0.65	0.42	0.84
	K Ozs Au	-	469	480	582	390	612	435	287	295	397	528	674	366	33	5,551
	Recovery	0%	93%	92%	92%	91%	92%	92%	90%	91%	91%	92%	93%	91%	88%	92%
	K Ozs Au Rec	-	434	440	537	356	565	399	258	267	362	485	624	333	30	5,089
Mixed Ore	K Tonnes	-	-	61	358	385	473	359	441	68	-	-	-	127	-	2,272
	g Au/t	-	-	0.58	0.75	0.55	0.68	0.59	0.44	0.42	-	-	-	1.05	-	0.62
	K Ozs Au	-	-	1	9	7	10	7	6	1	-	-	-	4	-	45
	Recovery	0%	0%	91%	91%	90%	91%	91%	89%	88%	0%	0%	0%	92%	0%	91%
	K Ozs Au Rec	-	-	1	8	6	9	6	5	1	-	-	-	4	-	41
Oxidized Ore	K Tonnes	-	-	58	57	84	303	207	308	45	-	-	-	-	-	1,061
	g Au/t	-	-	0.63	1.02	0.53	0.66	0.61	0.44	0.42	-	-	-	-	-	0.58
	K Ozs Au	-	-	1	2	1	6	4	4	1	-	-	-	-	-	20
	Recovery	0%	0%	91%	92%	90%	91%	91%	89%	88%	0%	0%	0%	0%	0%	90%
	K Ozs Au Rec	-	-	1	2	1	6	4	4	1	-	-	-	-	-	18
Total	K Tonnes	-	12,461	17,750	17,799	17,750	17,750	17,750	17,799	15,129	17,750	17,750	17,799	17,750	2,451	207,687
	g Au/t	-	1.17	0.85	1.04	0.70	1.10	0.78	0.52	0.61	0.70	0.93	1.18	0.65	0.42	0.84
	K Ozs Au	-	469	482	593	399	629	446	298	297	397	528	674	371	33	5,616
	Recovery	0%	93%	92%	92%	91%	92%	92%	90%	91%	91%	92%	93%	91%	88%	92%
	K Ozs Au Rec	-	434	442	546	363	580	408	267	269	362	485	624	337	30	5,148

16.6 Equipment Selection and Productivities

Mt Todd has been planned as an open-pit mine using large haul trucks, hydraulic shovels, and front-end loading equipment. Primary mine production is to be achieved using 31-cubic meter hydraulic shovels along with 226-tonne haul trucks, though final equipment selection may differ.

Secondary mine production is to be achieved using 18-cubic meter loaders along with the 226-tonne trucks. Loaders will be used mostly to mine ore from the pit to the crusher, and for reclamation of ore from stockpiles. Some waste production from the loader is anticipated as well.

Table 16-6 shows the maximum shovel productivity estimate based on scheduled time, availability, and truck and material parameters. This maximum productivity would require that trucks are always available and the shovels are always digging; however, that is not always the case.

In-pit and ex-pit centerlines were drawn for each of the pits and destinations, including the waste dump, crusher, and ore stockpile. As the dump is very large, it was divided into 20 smaller volumes to account for haulage requirements during the life of mine. Truck speeds for each profile were calculated based on published rim-pull curve data. Maximum speed limits were also applied to ensure that safe operating conditions were adhered to and that productivities were achievable.

Bench haulage routes were also drawn for each bench to ensure proper travel on the benches and that truck requirements are properly accounted for. Bench travel speed limits were applied to the profiles for both loaded and empty trucks.

Mine production schedules were run using MineSched (version 9.1) mine scheduling software. The profiles and truck parameters were supplied to MineSched to calculate the productive truck hours required. An efficiency of 83% was used to derive operating hours from the productive hours. This accounts for inefficiencies in the operations that are found between the loading units and the dumping locations. This is similar to a 50-minute working hour.

Incremental truck hours were added to waste haulage to account for waste material hauled to TSF 1 and TSF 2 for construction purposes. Haulage requirements for sorter tailings were estimated within cost sheets using a constant cycle time. The material would be loaded into a truck from a silo and the silo bin is sized to use the mine fleet. It was determined that a single truck would be able to take care of the haulage needs for the sorter.

Loading-unit hours were estimated using 83% efficiency and the production rate for loading equipment. The schedule was constrained using tonnage on a period basis to balance the use of loading and haulage equipment.

Availability was estimated dependent on the age of the piece of equipment. Availabilities start out at 90% and decrement 1% per year until they reach 85%, and then they are kept constant. Availabilities, efficiencies, operating hours, and load and haul equipment requirements are shown in **Table 16-7** for the Base Case.

Table 16-6: Maximum Loader Productivity Estimate

Description	Unit	All Rock	
MATERIAL PROPERTIES			
Material SG (BCM)	t/cm (Wet)	2.70	
Material SG (Loose)	t/cm (Wet)	1.93	
Material SG (BCM Dry)	t/cm (Dry)	2.50	
Material SG (LCM Dry)	t/cm (Dry)	1.79	
Swell Factor		1.4	
DAILY SCHEDULE			
Shifts per Day Hours per Shift	shift/day hr/shift	2 12	
Theoretical Hours per Day	hrs/day	24	
Shift Startup / Shutdown Lunch Breaks Operational Standby	hrs/shift hrs/shift hrs/shift hrs/shift	0.5 0.5 0.25 0.25	
Total Standby / Shift Total Standby / Day	hrs/shift hrs/day	1.50 3.00	
Available Work Hours	hrs/day	21.00	
Schedule Efficiency	%	87.5	
		31 cm Hyd 226 T Trks	**18 cm FEL 226 T Trks**
LOADING PARAMETERS			
Shovel Mech. Avail.	%	85%	85%
Operating Efficiency	%	83%	83%
Bucket Capacity	m3	31	18
Bucket Fill Factor	%	95%	95%
Avg. Cycle Time	Sec	34	50
TRUCK PARAMETERS			
Truck Mech. Avail.	%	85%	85%
Operating Efficiency	%	83%	83%
Volume Capacity	m3	176	176
Tonnage Capacity	lt (Wet)	227	227
Truck Spot Time	Sec	24	24
SHOVEL PRODUCTIVITY			
Effective Bucket Capacity	Cyd	29.45	17.10
Tonnes per Pass – Wet	lst (Wet)	56.8	33.0
Tonnes per Pass – Dry	lst (Dry)	52.6	30.5
Theoretical Passes – Vol	passes	5.98	10.29
Theoretical Passes – Wt	passes	4.00	6.88
Actual Passes Used	passes	4.0	7.0
Truck Tonnage – Wet	wmt/load	226	226

		31 cm Hyd 226 T Trks	**18 cm FEL 226 T Trks**
Truck Tonnage – Dry	dmt/load	210	210
Truck Capacity Utilized – Vol	%	67%	67%
Truck Capacity Utilized – Wt	%	100%	100%
Load Time	min	2.67	6.23
Theoretical Productivity	dst/hr	4,729	2,023
Tonnes per Operating Hour	dst/hr	3,930	1,680
Tonnes Per Day	**dst/day**	**70,200**	**30,000**
Potential – 355 days/year	**t/year**	**24,921,000**	**10,650,000**

Table 16-7: Annual Load and Haul Equipment Requirements – Base Case

		Pre-Prod	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	Total
HAULAGE REQUIREMENTS																	
Productive Hours	Hrs	13,186	45,149	109,615	106,344	174,037	178,598	189,570	194,550	212,805	205,229	210,421	112,304	15,234	14,382	30,033	1,811,462
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	1743%
Operating Hours	Hrs	15,887	54,397	132,067	128,125	209,683	215,179	228,398	234,397	256,391	247,264	253,520	135,306	18,354	17,328	36,184	2,182,485
Number of Trucks	#	8	11	22	27	33	35	37	38	41	41	41	22	6	6	6	376
Truck Availability	%	90%	90%	89%	89%	88%	88%	87%	86%	86%	85%	85%	85%	86%	85%	85%	
Available Operating Hours	Hrs	19,188	71,686	136,420	168,364	214,038	224,555	233,608	238,718	255,731	254,494	254,276	136,910	37,334	37,115	37,115	2,319,902
Use of Available Hours	%	86%	76%	97%	76%	98%	96%	98%	98%	100%	97%	100%	99%	49%	47%	97%	94%
Tonnes per Operating Hour	t/Hr	731	585	539	478	482	413	460	367	280	249	249	269	1,031	998	775	391
HYDRAULIC SHOVEL USAGE																	
Number of Shovels	#	2	2	3	4	4	4	4	4	3	3	3	2	1	1	1	3.3
Availability	%	90.0%	89.4%	88.9%	88.2%	87.6%	86.6%	85.9%	85.4%	85.2%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	86.4%
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83.0%
Available Operating Hrs	Op Hrs	4,834	13,009	18,860	20,836	25,509	25,217	25,005	24,930	18,601	18,558	18,558	12,407	6,186	6,186	6,186	244,880
Tonnes Mined	K Tonnes	11,661	25,732	60,623	51,413	88,432	82,918	88,315	69,364	65,174	57,367	54,312	28,914	15,975	14,225	19,617	734,044
Operating Hours	Op Hrs	2,911	6,555	15,445	13,098	22,529	21,124	22,500	17,671	16,604	14,615	13,837	7,366	4,070	3,624	4,998	186,948
Use of Available Operating Hours	%	60%	50%	82%	63%	88%	84%	90%	71%	89%	79%	75%	59%	66%	59%	81%	76%
FRONT END LOADERS																	
Number of Loaders	#	-	1	2	2	2	2	2	2	2	2	2	2	1	1	1	1.8
Availability	%	0%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	85%	86%
Operating Efficiency	%	0.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%
Available Operating Hrs	Op Hrs	-	6,550	12,954	12,844	12,663	12,463	12,372	12,407	12,372	12,372	12,372	12,407	6,186	6,186	6,186	150,680
Tonnes Mined	K Tonnes	-	6,085	10,548	9,812	12,638	5,896	16,681	16,652	6,552	4,318	8,930	7,458	2,946	3,061	8,407	119,988
Operating Hours	Op Hrs	-	3,624	6,282	5,843	7,526	3,511	9,934	9,916	3,902	2,571	5,318	4,441	1,754	1,823	5,007	71,454
Use of Available Operating Hours	%	0%	55%	48%	45%	59%	28%	80%	80%	32%	21%	43%	36%	28%	29%	81%	47%

16.7 Mine Personnel

Mine personnel estimates include both operating and mine-staff personnel. Operating personnel are estimated as the number of people required to operate trucks, loading equipment, and support equipment to achieve the production schedule. Mine staff is based on the people required for supervision and support of mine production. The mine staff organizational chart is shown in **Figure 16-1**. The estimated number of mine personnel required to execute the mine plan is shown in **Table 16-8** for the Base Case.

Salaries for each position were estimated based on information received from Vista. Salaries include an allowance for benefits at a factor of 27% of the base salary for each position. Note that the mine personnel do not include contractors. Vista anticipates using a Maintenance and Repair Contract ("MARC") to maintain the mining fleet into the third year of operation. After that time, Vista will operate all maintenance crews. For the purpose of costing, the MARC costs were reduced to take into account savings by lowering contractor's overhead. Maintenance foremen were added to personnel along with another planner starting in year 3 as part of the maintenance responsibility takeover. However, since the maintenance cost used includes labor, the mechanics are not reflected in the total count for personnel. This would add approximately 80 mechanics, servicemen, and welders to the Base Case.

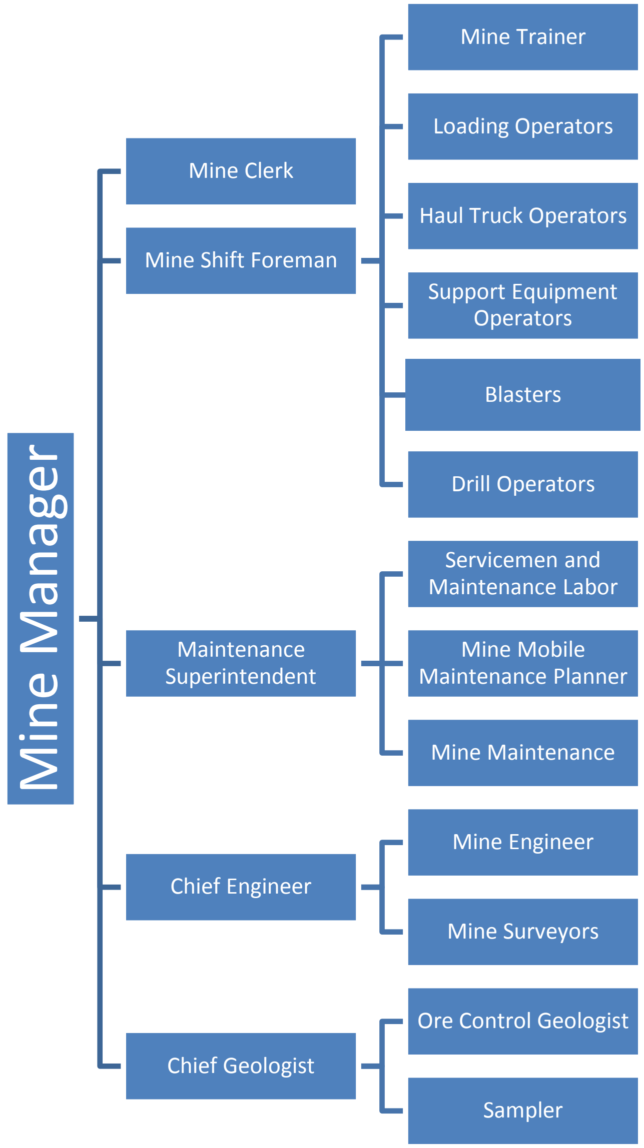


Figure 16-1: Mine Organizational Chart

Table 16-8: Mine Personnel Requirements – Base Case

	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
MINE OVERHEAD															
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Shift Foremen	8	9	9	10	12	12	12	12	12	12	12	9	6	6	6
Mine Trainer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Blaster	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-
Blaster's Helper	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-
MINE PRODUCTION															
Loading Operators	5	7	13	11	17	15	18	18	12	12	12	9	5	5	6
Haul Truck Operators	20	33	63	78	100	106	111	114	123	123	123	66	18	18	18
Drill Operators	13	15	30	31	42	45	42	33	33	31	33	20	1	-	-
Support Equipment Operators	14	14	18	20	20	20	24	24	24	24	21	18	12	12	12
Total Mine Operating	**67**	**85**	**140**	**157**	**198**	**205**	**214**	**208**	**211**	**209**	**208**	**129**	**49**	**44**	**45**
MINE MAINTENANCE															
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Foremen	-	-	-	3	3	3	3	3	3	3	3	3	3	1	1
Light Vehicle Mechanics	2	-	2	2	2	2	2	2	2	2	2	2	2	1	1
Tiremen	2	-	2	2	2	2	2	2	2	2	2	2	2	2	2
Shop Laborers	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Planner	1	1	1	2	2	2	2	2	2	2	2	2	2	1	1
Service, Fuel, & Lube	6	6	6	6	6	6	6	6	6	6	6	6	6	3	3
*Maintenance Labor				80	80	80	80	80	80	80	80	80	80	40	40
Total Mine Maintenance	**14**	**10**	**14**	**98**	**98**	**98**	**98**	**98**	**98**	**98**	**98**	**98**	**98**	**51**	**51**

	Pre-Prod	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
ENGINEERING															
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Surveyors	1	2	2	2	2	2	2	2	2	2	2	2	1	2	1
Surveyor Helper	2	2	2	2	2	2	2	2	2	2	2	2	1	2	1
Mine Engineer	2	3	3	3	3	3	3	3	3	3	3	3	1	1	1
Total Engineering	**6**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**4**	**6**	**4**
MINE GEOLOGY															
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Ore Control Geologist	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-
Sampler	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-
Total Geology	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**-**	**-**
TOTAL MINE OPERATIONS WORKFORCE															
Mine Operations	67	85	140	157	198	205	214	208	211	209	208	129	49	44	45
Mine Maintenance	14	10	14	18	18	18	18	18	18	18	18	18	18	11	11
Engineering	6	8	8	8	8	8	8	8	8	8	8	8	4	6	4
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	-
Total	**92**	**108**	**167**	**268**	**309**	**316**	**325**	**319**	**322**	**320**	**319**	**240**	**156**	**106**	**100**

** During year 3 the MARC would be removed and additional maintenance labor would be required to maintain the fleet.*

17.0 RECOVERY METHODS

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

The key criteria used in the process design of the Process Plant have been largely derived from metallurgical testwork and, where appropriate, have been provided by Vista, or based on TTP experience and industry norms. The design criteria and flowsheet development are discussed in this section.

17.1 Process Design Criteria

Detailed process design criteria have been developed for the Project. The nominal headline design criteria are listed as follows:

Table 17-1: Headline Design Criteria

	Unit	50,000 tpd
Annual Ore Feed Rate	Mt/a	17.75
Operating Days per Year	d/a	355
Daily Ore Feed Rate	t/d	50,000
Crushing Rate (6,637 hours per year availability)	tph	2,674
HPGR Rate (7,838 hours per year)	tph	2,264
Ore Sorting Rate (7,838 hours per year)	tph	408
Milling Rate (7,838 hours per year)	tph	2,055
Gold Head Grade	g/t	0.82
Copper Head Grade	%	0.055
Cyanide Soluble Copper	%	0.0024
Ore Specific Gravity	t/m^3	2.76
Primary Grind P_{80} to Secondary Grind	µm	250
Grind P_{80} to Leach	µm	40
Gold Recovery	%	91.9
Gold Production (average)	oz/d	1,165
Gold Production (average)	oz/a	413,400

The testwork results collated from the 2011 and 2012 testing campaigns and additional metallurgical and process test work conducted in 2016/2017/2018/2019, together with the process design criteria, were used to develop the process flow sheet and mass balance.

17.2 Flow Sheet Development

A schematic diagram of the process flowsheet is presented in **Figure 17-1**.

17.2.1 Crushing Modeling

Impact crush work index (CWi) tests were performed on eighty individual samples from the 2011 drill cores. The CWi values ranged from 3.2 kilowatt hours per tonne (kWh/t) to 26.5 kWh/t. For design purposes, a CWi of 20 kWh/t was selected, 75% of the maximum.

Unconfined compressive strength (UCS) was measured on 16 samples. The values ranged from 14 megapascals (MPa) (med strong) to 183 MPa (very strong). Eighty percent of the results were in the strong to very strong designation of ore hardness.

The run of mine ore from the pit is expect to have a maximum particle size of 1000 mm and F_{80} to the primary crusher of 400 mm. Two stages of crushing, primary and secondary are required to reduce particle size to a P_{80} of 31.5 mm, required as feed to the HPGR tertiary crushers. A single gyratory crusher is sized for the primary duty reducing ore size to a nominal P_{80} of 130 mm. Two secondary cone crushers operating in parallel and in closed circuit with two sizing screens cutting at 40 mm, are used to produce the feed to the HPGRs at product size P_{80} of 31.5 mm.

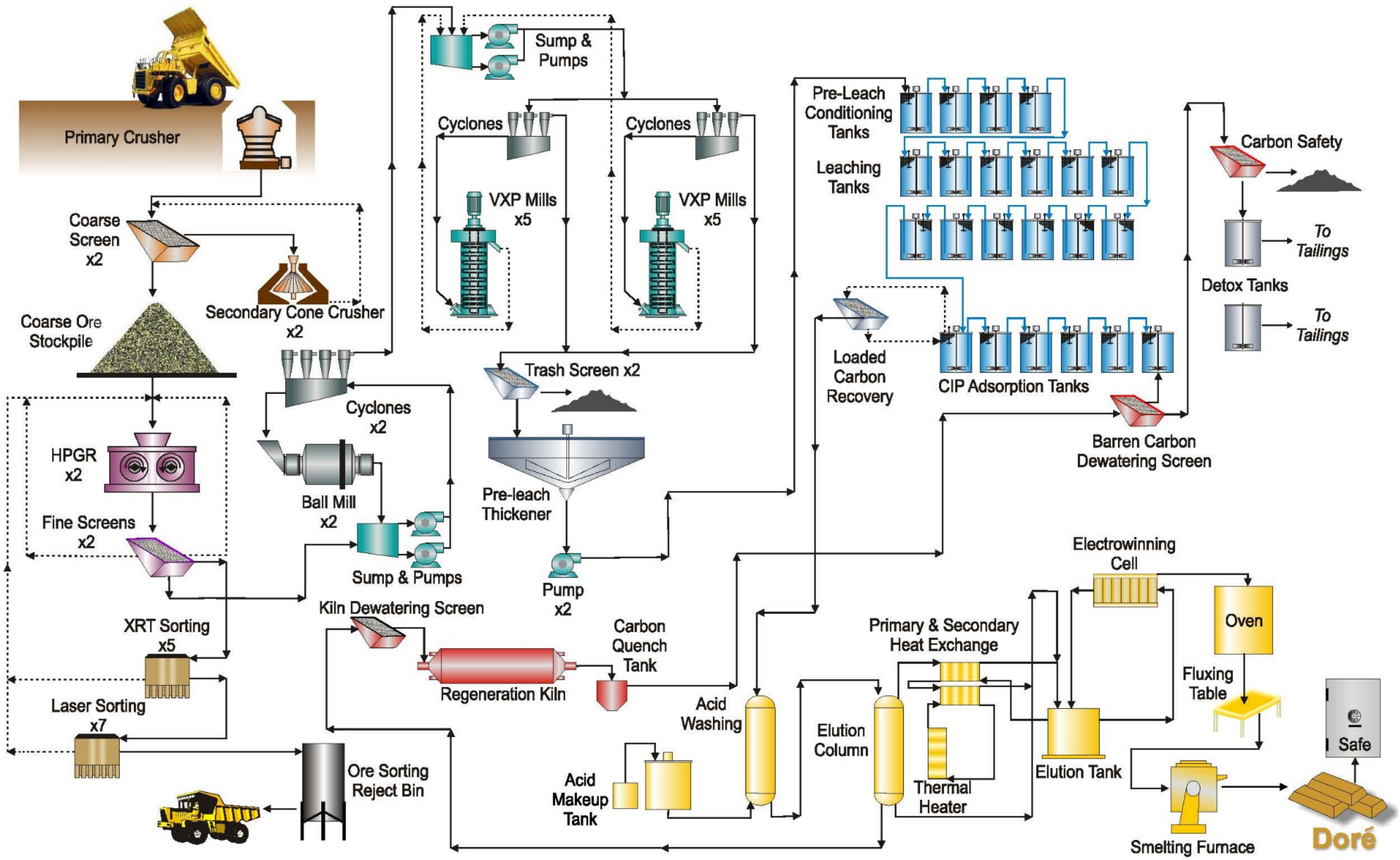


Figure 17-1: Simplified Process Flow Diagram

17.2.2 Primary Crusher

The primary crusher power was calculated using both the FLSmidth gyratory calculation model and the Metso Bruno model. Using the CWi of 20 kWh/t and a fall through percentage of zero to simulate peak conditions, both models provided peak primary crushing power requirements of a nominal 574 kW and 576 kW respectively to reduce a feed F_{80} of 400 mm to a product P_{80} of 130 mm.

17.2.3 Secondary Crushers

Secondary crushing with closed circuit screening was modelled by FLSmidth. Two Raptor 1300 cone crushers operating in parallel are used to reduce the primary crusher product to a final product P_{80} of 31.5 mm, for the 50,000 tpd Base Case.

17.2.4 HPGR

HPGR power requirements to reduce the HPGR feed to a final product P_{80} of 3.25 mm was shown by the Polysius testwork to be 1.9 kWh per tonne of feed to the HPGR. The feed to the HPGR is the sum of new feed plus the recirculating load screen oversize material, less Ore Sorting reject. The total feed to the HPGR is two times the fresh feed rate. HPGR testwork supported vendor recommendations for two HPGR Polycom PM8-24/17, each equipped with 2 x 2,650 kW drives.

17.2.5 Ore Sorting

The coarse fraction (plus 16mm) from the HPGR will be sent to the ore sorting equipment for separating the gold-bearing sulfide minerals and quartz veining from non-gold bearing waste material.

17.2.6 Grinding Modeling

A variety of internal models were used to provide the initial baseline ball mill power requirements and vendors were approached for proposals. The most price competitive and technically acceptable submission was then selected for further interaction, with the vendor calculations compared against internal calculations. The circuit incorporates two dual pinion drive ball mills.

17.2.7 Thickener / Leach / CIP Design

THICKENER

Based on thickener sizing parameters received from RDI Minerals, that were based on additional 2019 rheology and settling test work undertaken by Pocock Industrial for a final grind size of 40um, a 67 m Pre-Leach thickener for the 50,000 tpd case and a 55 m Pre-leach thickener for the 33,000 tpd was recommended.

The test work also reported an Underflow solids of approximately 53.88% solids was achievable for both of the above sizes for each respective duty.

LEACH AND ADSORPTION

The optimum leach / adsorption density as determined by SPX testwork was 55% solids for the previous grind size of 90 um. This was changed to 45% solids as the current grind size of 40µm would result in an excessive viscosity in 55% slurry.

The leach and adsorption circuits were modelled. A six-stage adsorption is required to minimize solution losses. Dissolved gold in residue solution will be ≤0.010 ppm.

At the planned gold head grade, the system will produce a loaded carbon head grade of approximately 1,250 g/t, and carbon movement requirements will be in the order of 30 tpd for the Base Case.

17.3 Description of Process Areas

17.3.1 Area 3100 – Crushing Circuit Availabilities

The crushing circuit availabilities coupled with the ore crusher work index are the two predominant factors in sizing crusher circuits. Rather than assuming a standard availability of between 70% and 75%, a review of the previous primary crusher operations at Mt Todd was conducted. When removing the downtime periods when the crushing system was not required, the average availability for the remaining duration was 59%.

Additionally, TTP reviewed results from a two-year study and dynamic simulation of a large-scale crusher operation in the tropics, which indicated the downtime was apportioned as follows:

- Dump hopper empty 19.2% (mining not keeping up)
- Cannot discharge 15.6% (downstream equipment interruptions)
- Operating Breakdown 0.6% (crusher specific)
- Mechanical breakdown 1.2% (crusher specific)
- Electrical breakdown 2.3% (crusher specific)
- Planned maintenance 2.5% (crusher specific)

The combination of this data coupled with the historical Mt Todd crusher downtime led to an initial crusher circuit availability of 60% being selected, with first pass crushing equipment initially being selected on this basis.

Subsequently it was agreed with the mining design consultant MDA that the costs of an extra loader and build of an emergency stockpile on the ROM pad be included and to remove the downtime attributable to mining lack of supply in its entirety.

This resulted in an availability of 75.8%, 6,637 operating hours per year.

17.3.1.1 Crushing Circuit Design

The crushing circuit was chosen based on reliability and similarity to existing mining operations. It consists of a single Primary Crusher in an open loop configuration and two Secondary Crushers in parallel in a closed loop configuration with sized output conveyed to a buffer stockpile, providing three days live capacity. The primary and the secondary crushers discharge onto a common conveyor that feeds the Coarse Ore Screens. This configuration allows reduced conveyor footprint and maximum plant productivity.

The Coarse Ore Screens will be fed by vibrating feeders, which regulate the flow from the feed bins. This arrangement maximizes the efficiency of the screens by ensuring full coverage of screen decks at controlled bed depth. Crusher area dust is controlled by dust collection at the screens and dust suppression in all other dust generating areas.

17.3.2 Area 3200 – Coarse Ore Stockpile, Reclaim, HPGR and Ore Sorting

A plant availability factor of 89.5% of 365 days/year has been used for the HPGRs and subsequent downstream processes, which is 7,838 operating hours per year. HPGR availability in large hard rock applications ranges from 89% to 92%, with some operations reporting periods of 95% availability when roll change has not been required (Boddington). It is considered appropriate to use a conservative availability factor of 89.5% of the annual 8,760 hours for Mt Todd ore due to its ore hardness.

The coarse ore stockpile will have approximately three days of total capacity between the secondary crushers and the HPGRs, with 23% of that total capacity representing the live volume. Ore will be removed from beneath the Coarse Ore Stockpile by two Apron Feeders.

Two HPGRs will operate in parallel to process for the Base Case. The HPGRs are protected from tramp metal by installation of metal detectors on feed conveyors.

A common HPGR Product Conveyor will receive the discharge from the HPGRs and convey the material to the Fines Screens Feed Bins. The HPGR fines screens are double decked, cutting at nominal 4.5 mm to produce an underflow product at P_{80} of 3.2 mm and 16mm to produce a mid and an oversize. The screens operate as wet screens with high pressure spray water applied to the decks to assist with screen efficiency. The screen mid material (+4.5mm-16mm), ~3-5% moisture, will be conveyed back to the HPGR feed bins and the screen oversize (+16mm) material will be conveyed to Ore Sorting.

Ore Sorting receives a nominal 408 t/h for the Base Case. Ore Sorting comprises two stages, XRT and Laser sorting. The two stages together reject 210 t/h for the Base Case, representing approximately 10% of plant feed.

The above reject performance and nominal gold loss was derived from bulk ore sorting test work completed at the Tomra sorting facility. Gold lost to Ore Sorting reject is minor at a nominal 1.3% of gold entering plant.

17.3.3 Area 3300 – Grinding and Classification

Two Ball mills will be used for the Base Case. The parallel Ball Mill circuits are in a conventional configuration. Fresh feed from the fines screens underflow will gravitate to the mill discharge hopper and will be pumped together with the mill discharge slurry to the Cyclones. The cyclone underflow will gravitate to the ball mill feed. The overflow will gravitate to the Secondary Grind feed sump. The secondary grinding cyclone overflow will be pumped to the pre-leach thickener and the underflow will be sent to the VXP mills for secondary grinding.

An automated ball charging system will be provided to deliver approximately 15 tonnes of balls per day to each mill.

17.3.4 Area 3400 – Pre-Leach Thickening, Leach Conditioning, Leach and CIP

In order to achieve the required 45% solids feed to the leach and CIP tanks, a Pre-Leach Thickener will be used.

Two Leach Conditioning stages will be incorporated ahead of Leach. These tanks will be sized to deliver a total residence time of 4 hours. In these stages, the ore is treated with lime which inhibits reaction of cyanide with pyrites and pyrrhotites by way of forming a lime coating around these gangue components.

The leach and adsorption tanks will be sized to deliver a total residence time of 24 hours for Leach and 6 hours for adsorption, as determined by the test work. For both cases, leach and adsorption will consist of eighteen mechanically agitated tanks in total. There will be twelve leach tanks and six adsorption tanks.

In order to maximize the gold adsorption kinetics, lead nitrate will be added, oxygen will be provided by sparging compressed air into the leach tanks.

Each Leach and CIP Tank can be bypassed for maintenance purposes. Carbon will be regularly pumped upstream from downstream CIP Tanks in a conventional counter-current configuration. The adsorption tanks will be equipped with Kemix interstage carbon screens. The pumping screens will be used to generate the overflow head required for downstream slurry advance.

Carbon Safety Screens will catch any fugitive carbon from the tails slurry. Usable carbon will be returned to the circuit, undersize carbon will report directly out of the circuit via detoxification and tails.

17.3.5 Area 3500 – Desorption, Goldroom and Carbon Regeneration

Loaded carbon will be acid washed in an Acid Wash Column, then stripped of copper and gold in an Elution Column. Cold cyanide wash will be used to strip adsorbed copper prior to hot caustic cyanide wash to strip gold. Acid wash effluent and copper wash effluent will be pumped to the detox tanks. The elution and electrowinning process will be the Anglo American Research Laboratories configuration. Eluant will be pumped through the column, heated to 120 deg°C, and collected as loaded eluate in one of two eluate tanks. The desorption circuit will be batch and will take up to 8 hours. The columns are sized to ensure that two elution batches can be performed in a day. After the elution is completed and the carbon is stripped of its gold to about 10 g/t Au, the eluate will be processed through the electrowinning circuit for deposition of gold onto cathodes. The electrowinning circuit will be batch and take up to 8 hours, or until the gold in solution reduces to less than 10 ppm.

The Goldroom consisting of Electrowinning, Drying and Smelting facilities will be supplied as a vendor package. Stripped carbon will be regenerated using an indirect heated horizontal rotary kiln.

17.3.6 Area 3600 – Detoxification and Tailings

Two Detoxification Tanks in series will be used to minimize short-circuiting and sized to ensure the required residence time of one hour is achieved.

The second Detox Tank will cascade overflow to a Tailings Pump Hopper from where the tailings will be pumped to the Tailings Storage Facility. Future booster pumps will be required once the second tailings facility is operational. A duty/standby configuration of pumps will be used to ensure continuous operation.

17.3.7 Area 3700 – Reagents

Sodium Meta Bi-Sulfite (SMBS) will be delivered to site as a 95% pure solid powder in sea containers. A container tipper and solids handling equipment will transfer the powder from the storage containers to the mixing tank. SMBS will be mixed to 20% w/v in solution and dosed to the detoxification tanks via duty/stand-by dosing pumps. Dust extraction equipment is present at all transfer points of the solids handling and the area where solids handling takes place will be well ventilated. The SMBS solution will have storage for 3 days of nominal usage.

The Sodium Cyanide for Leaching and Elution will be delivered as briquettes in a bulk tanker. For the Base Case, the sodium cyanide will be consumed at a rate of approximately 39 tpd. The solids will be dissolved in the tanker and cyanide solution will be transferred into a mixing tank to ensure full dissolution. The cyanide solution will then be transferred into four storage tanks allowing seven days nominal capacity. There will be a secured and covered facility to store cyanide as emergency storage.

The Hydrochloric Acid (HCL) for the Acid Wash Column will be delivered as a 33% HCl solution and will have storage for 20 days of nominal usage.

Lime will be delivered as 92% active quick lime powder in road tankers. The lime will be pneumatically transferred to storage silos approximately 200 tonnes capacity. Lime will be slaked on a daily basis. Milk of lime will be distributed from a lime surge tank to Leach.

Sodium Hydroxide (NaOH) will be delivered as a powder in bulk bags and mixed to produce a 50% NaOH solution. Sodium hydroxide is only consumed periodically and therefore does not require an additional storage tank beyond the mixing tank. A nominal 20-day dry solids storage capacity was included into the design.

The lead nitrate for the leach circuit will be delivered as a powder in bulk bags and mixed to produce a 20% solution. The lead nitrate solution will have storage for seven days of nominal usage.

17.3.8 Area 3800 – Process Plant Services

Approximately 800 normal meters cubed per hour (Nm^3/h) of medium pressure process air will be used to service the air requirements for leach and adsorption for the Base Case. Detoxification will be serviced by medium pressure air blowers at a consumption rate of approximately 5,200 Nm^3/h for the Base Case. High pressure compressors are used to provide plant and instrument air.

Raw water will be supplied via the Raw Water Dam and will service the process water, fire water and gland seal water requirements. Raw water will also service the water treatment plant for potable water required at the mining facilities, process plant, powerhouse and camp. For the Base Case, the nominal raw water consumption will be ~800 m^3/h and will occasionally peak at 2,200 m^3/h during the dry season.

Process plant water will be predominantly made-up of tailings decant return water and raw water. Process water will be used for dilution and density control in the grinding circuit.

17.4 Process Water

The water reticulation system for the process plant will consist of the following:

- Raw water supply;
- Potable water supply;
- Fire water supply;
- Gland service water supply; and
- Process water supply.

Raw water will be delivered from the raw water dam (RWD) to the 9,600 m^3 process plant raw water tank. This water will be used as make-up water for the process water supply, emergency firefighting supply, gland seal, dust suppression, plant clean-up hosing stations, powerhouse, mining facilities and water for the reagents make-up.

The fire water supply will be drawn from the reserve in the raw water tank providing water to the plant site fire water distribution system.

Gland service water for the main plant site will be drawn from the raw water tank. It will be used to supply gland service water for slurry pumps in the plant.

The process water system will include a 9,600 m^3 storage tank. Process water will be supplied to the plant via centrifugal pumps, one operating and one stand-by unit. This water supply will be used for process stream dilution and for use as spray water for the screens. The pre-leach thickener, tailings dam decant water and raw water all report to the process water tank.

17.4.1 Process Compressed Air

The plant and instrument air supply systems for the process plant will consist of high pressure compressed air units in the following locations:

- Primary Crushing (duty only);
- Reclaim Tunnel (duty only);
- HPGRs (duty only);
- Grinding and Classification (duty/stand-by); and
- Leach and CIP (duty/stand-by).

Twin-screw compressors at each location will supply plant air and instrument air to the buildings in which they are located. The air discharging from each compressor will be fed to a plant air receiver and distributed throughout the building. An off-take from the discharge of the plant air receiver will be dedicated to instrument air which will pass through a refrigerant dryer with pre and post filters to an instrument air receiver. This air will be used for instrument air purposes with the required air quality achieved. The remainder of the air generated by the compressors will be used for general plant air duties. The dry areas of the plant will only have a single duty compressor due to the limited requirement of plant and instrument air whereas the wet plant areas will have a duty/standby arrangement.

A dedicated low pressure compressed air system in a duty/stand-by arrangement will be located in the CIP area of the plant for process air in the leach and CIP tanks. The CIP process compressors will deliver air at the required pressure and flow for injection into the leach and CIP tanks.

Similarly, a dedicated low-pressure blower air system in a duty/stand-by arrangement will be located in the cyanide detoxification area of the plant for process air in the cyanide detoxification tanks.

17.5 Plant Mobile Equipment

The plant mobile equipment will be as follows:

Table 17-2: Mobile Equipment for Process Plant

Light Vehicles	**Quantity**
Landcruiser wagon	2
Dual cab Utes	21
Tray top Ute	9
Troop carrier (ambulance)	1
Bus/troop carrier (15-seat)	1
Coach	3
Subtotal	**27**
Process Plant Mobile Equipment	**Quantity**
Loader – Cat 966G	Allowed for in mining
Tool Carrier – Cat IT28	1
Bob Cat – Mustang Case	1
Crane – 15-t Franna	1
Hiab Truck – 7-t	1
Service Truck – 2-t	1
2-t Forklift – allowance	2
25-t Container Forklift	1
80-t Crane	1
Mill Relining Machine	**1**
Subtotal	**10**

18.0 PROJECT INFRASTRUCTURE

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

18.1 Facility 2000 – Mine

The following section provides a description of the Mine Support Facilities and Mine Support Services that have been developed to support the mining activities.

18.1.1 Area 2300 – Mine Support Facilities

Area 2300 Mine Support Facilities consists of the buildings and services for the maintenance and repair of the mine vehicle fleet including Heavy Vehicles (HV). The area is located along the haul road adjacent to the proposed stockpile, between the Pit and the existing Tailings Storage Facility.

18.1.1.1 Sub-Area 2305 – Support Facilities – HV Workshop/Warehouse

The Workshop Facility will consist of six Dome Shelter structures mounted on sea containers with concrete floors. The sea containers come equipped as Site Offices, Store Services, Store Consumables, Equipment Repair and Lube Storage and Dispensing facilities for the maintenance and servicing of HVs that are used for mining operations.

The Workshop will be approximately 85.6 m by 24.4 m and sized to service Caterpillar 793F mining trucks.

The Warehouse Facility will consist of one Dome Shelter structure mounted on sea containers with a concrete floor. The Warehouse Facility will be approximately 21.7 m by 24.4 m in size. The sea containers come equipped as Site Offices, Rigging Container, Equipment Repair Workshop and Stores Consumable Container for the storage of parts, components, spares and the like, used by the HV Workshop for vehicle repair.

The HV Workshops and Warehouse Facilities will be complete with all services including power, lighting, communications, lubes, compressed air, water, specialist equipment and other services necessary for the maintenance of the mine vehicle fleet.

The Dome Shelters will be constructed of steel frame and tensile fabric with a fabric life expectancy of 10 years.

A mobile crane will be used externally to the Dome Shelters for the lifting and removal of vehicle parts.

18.1.1.2 Sub-Area 2310 Support Facilities – Bulk Fuel Storage

The Bulk Fuel Storage will consist of the relocated 600 kL tank complete with six new bowsers for dispensing into the HV fleet and one new 110 kL self-bunded diesel fuel tank complete with one bowser for dispensing into the LV's.

The new 110 kL tank will be utilized for refueling the Mine fleet when the 600 kL tank is not in service and will be located adjacent to the HV Workshop.

18.1.1.3 Sub-Area 2315 – Support Facilities – HV Washdown

The HV Washdown Facility will primarily be used for washing down the body and undercarriage of heavy vehicles prior to entering the HV Workshop. The facility will consist of a single bay with raised platforms with stair access to four manually operated high pressure water cannons. The run-off water will be connected to the oily water separator and will include drive-in concrete sumps and pits for waste water storage and recovery. The entire facility's footprint is 18 m by 21.8 m and is sized to service Caterpillar 793F mining trucks.

18.1.1.4 Sub-Area 2320 – Support Facilities – Crib / Ablutions / Lockers

The Crib / Ablutions / Lockers Facilities will be a transportable building used by mining personnel and is located adjacent to the HV Workshop. The building will include the necessary system furniture. The Crib Area will also double as a Pre Start Area.

The building will be sized to serve 75 people per shift and cover approximately 19.8 m by 14.4 m.

18.1.1.5 Sub-Area 2325 – Support Facilities – HV Tire Change

The Tire Change Facility will consist of one Dome Shelter mounted on sea containers with a concrete floor. The sea containers come equipped as Tire Change Workshop and Store Consumables for the maintenance and changing of HV tires.

The Tire Change Facility will be approximately 26.9 m by 18.1 m and sized to service Caterpillar 793F mining trucks.

The Tire Change Facility will be complete with services including power, lighting, communications, compressed air, water, specialist equipment and other services necessary for the changing of tires.

The Dome Shelter will be constructed of steel frame and tensile fabric with a fabric life expectancy of 10 years.

18.1.1.6 Sub-Area 2335 – Support Facilities – Lube Storage

The Lube Storage Facility will consist of a bunded concrete slab for the storage of Intermediate Bulk Containers (IBCs) containing oils and lubricants for the servicing of HVs. The Lube Storage Facility will be located in-between the HV Workshop and the Fuel Storage Facility. Full IBCs will replace containerized IBCs within the Workshops. Lube will be distributed manually. Used oil will be collected in a designated area for approved recycle/disposal.

18.1.1.7 Sub-Area 2340 – Support Facilities – ANFO / Magazine Facility

The Ammonium Nitrate Fuel Oil (ANFO) facility is capable of distribution of 10,000 tpa. It is a secure compound for the Ammonium Nitrate (AN), Ammonium Nitrate Emulsion (ANE) and diesel fuel.

The facility includes an area for AN storage, concrete hardstand for AN transfer to a Mobile Process Unit (MPU) and containment pond for spill material.

The ANE is tank stored on concrete plinths with air compressor and pumps for in-loading and out-loading of emulsion.

The diesel is stored in a 110 kL self-bunded tank and includes a spill containment unit.

Magazine storage will consist of two secured modified shipping containers for the storage of detonators, accessories and explosives. The magazines are located adjacent to the ANFO Facility and are surrounded by earth bunding and secure fencing.

The MPU will be used to transport, mix and deliver ANFO to the mine.

A transportable building will be provided to include Office / Crib / Ablution facilities at the Site for driver and delivery personnel.

The ANFO facility footprint is approximately 84.7 m by 128.5 m, excluding the diesel tank.

18.1.1.8 Sub-Area 2345 – Support Facilities – Mining Offices

The Mining Offices will be a transportable building used by mining personnel and is located adjacent to the HV Workshop. The building will include a kitchen, ablutions, cellular and open planned offices, meeting rooms, training spaces and necessary system furniture.

The footprint of the Mining Offices is approximately 24 m by 16.5 m and will be sized to account for 25 people.

18.1.1.9 Sub-Area 2355 – Support Facilities – Core Shed

The Core Storage Facility will consist of one Dome Shelter mounted on sea containers with a sealed asphalt floor for the storage of core samples at the mine support area.

The sea containers will be equipped with racking for additional storage.

The Core Storage facility will be complete with power and lighting and located north-east of the mining offices.

The Dome Shelter will be constructed of steel frame and tensile fabric with a fabric life expectancy of 10 years.

The Core Storage Facility has a footprint of approximately 48.8 m by 47.3 m.

18.1.2 Area 2400 – Mine Support Services

Mine Support Services consists of the services for the Mine Support Facilities.

18.1.2.1 Sub-Area 2410 – Support Services – Potable Water

Potable water will be provided to the Mine Support Facilities from the Process Plant Area via pipework in common services trenching.

18.1.2.2 Sub-Area 2420 – Support Services – Raw Water

Raw water will be provided to the HV Washdown storage tank at the Mine Support Facilities via a connection from the raw water pipework running along the existing haul road to the Process Plant Area.

18.1.2.3 Sub-Area 2430 – Support Services – Fire Water

The Fire Water Main will be provided to the Mine Support Facilities and camps from the Process Plant Area via pipework in common services trenching. Fire hydrants will be provided at required locations.

18.1.2.4 Sub-Area 2440 – Support Services – Air

Compressed air will be provided at the HV Workshop and HV Tire Change facilities via suitably sized standalone air compressors and receivers.

18.1.2.5 Sub-Area 2450 – Support Services – Power

Power will be provided to the Mines Support Facilities via a connection from the 33 kV overhead power line running past the site into a kiosk substation. From the kiosk, 400V/230V power will be reticulated to all required buildings and services in common services trenches.

18.1.2.6 Sub-Area 2450 – Support Services – Communications

Communications will be provided to the Mine Support Facilities from the Process Plant Area via a fiber optic cable in the overhead power lines and will terminate into a server room within the Mine Offices. Cat 6 cables will be reticulated to required building and services.

18.2 Facility 4000 – Project Services

This section details the supply and distribution of services outside the process plant.

18.2.1 Area 4100 – Water Supply

Area 4100 covers the water supply to the process plant and between facilities.

18.2.1.1 Sub-Area 4110 – Water Treatment Plant (WTP)

A Water Treatment Plant will be fed with a combination of decant return, runoff pond water and pit dewatering discharge at a nominal rate of 500 m^3/hr.

18.2.1.2 Sub-Area 4120 – Raw Water

The raw water requirement for the Base Case will be approximately 30,000 m^3/day, fluctuating due to current operations and weather. The existing line from the Raw Water Dam will be supplemented with an additional 250 mm poly line approximately 4 km in length in order to handle the increased raw water requirements of the higher throughput. This would run parallel to the existing 400 mm poly line.

Raw water will be supplied to the mine support facilities via a one km supply line to a storage tank in that facility. Raw water will be supplied to the power plant via a two km supply line and to the construction camp via a 5 km supply line.

Supply of water to the construction camp via tanker was investigated and it was deemed that a supply pipeline was the most cost-efficient method for transferring water to the construction camp.

18.2.1.3 Sub-Area 4130 – Potable Water

Potable water will be produced by a Potable Water Treatment Plant within the processing facility, and will be distributed to the process plant, construction camp, residual operating camp, mining, administration offices and laboratory facilities. For the Base Case there will be nominally 100 m^3 of potable water consumed per day.

18.2.2 Area 4200 – Power Supply

18.2.2.1 Sub-Area 4210 – Power Generation

Refer to **Section 0 – Electric Power Plant** for the discussion on power supply design.

18.2.2.2 Sub-Area 4230 – High Voltage Electrical Distribution

A Main 33kV Switchroom will be the main point of connection for incoming power from the Power Station, as well as fiber optic communications between Telstra and the plant. The Switchroom includes the main plant 33 kV switchgear feeders, metering, and an allowance for process plant power quality equipment.

33 kV power distribution is via the Main 33 kV Switchroom, which will feed 33 kV buried cables supplying the Process Plant 33 kV Substations, as well as the site wide overhead power line.

18.2.2.3 Sub-Area 4231 – Power Distribution

It is not desirable to install overhead power lines close to the Process Plant, where it may cause a hazard to over-height vehicular traffic such as cranes. Therefore, in order to keep the overhead power lines away from these areas, 33 kV power to and from the Process Plant will be connected by buried cables. The buried cables will be connected to the Main 33 kV Switchroom.

18.2.2.4 Sub-Area 4232 – Overhead Power Lines

33 kV power will be provided from the Power Station via a single feeder. New 33 kV overhead power lines will be required to connect the Power Station to the Process Plant, which are approximately 1.2 km apart. These will be installed along a similar route as the main access road.

The 33 kV power line will also need to be distributed around site to the following facilities:

- ANFO Facility
- Heap Leach Pad (existing)
- Construction Camp/Residual Accommodation Camp
- Waste Water Treatment Plant (WWTP)
- Pit Dewatering
- Mine Services
- Site Radio Communication Tower (depending on final location)
- Gatehouse
- Future Tailings Storage/Decant

The total length of overhead power line required to reach the Process Plant and the above locations from the Power Station is 7.1 km.

The overhead power line will incorporate a fiber optic cable into the earth conductor. Overhead power lines will be suitably rated for a high dust and lightning strike region.

The Accommodation Camp is assumed to be within 2 km of either the existing 22 kV power lines or the new site 33 kV power lines.

18.2.3 Area 4300 – Communications

18.2.3.1 Sub-Area 4310 – Fiber Optic

Two fiber optic cable ring mains will be installed around the Process Plant to form redundant topology. These cables will generally be installed on cable ladders within the plant, although sections of the cable will be buried where cable ladder access is not available. The second cable is to provide redundancy within the Process Plant in case of damage to the first cable and will follow a separate route where this is practical.

The plant fiber optic cables will contain up to 72 cores and will incorporate separate networks for data communications including those for the Plant Process Control System, the site IT system, a site Voice over Internet Protocol (VoIP) phone system, site Closed Circuit TV (CCTV) and security network, and fire detection system.

Outside of the Process Plant, the fiber optic cables will be incorporated into the earth conductor of the overhead power lines. Optical Ground Wire (OPGW) is a dual functioning cable. It is designed to replace traditional earth wires on overhead power lines with the added benefit of containing optical fiber cores that can be used for communications purposes. These will connect communications equipment from locations such as the Power Station, Water Treatment Plant, Gatehouse and ANFO Facility to the plant communications network.

A communications hut will be provided (by others) outside the gatehouse. A fiber optic cable will be installed underground between this communications hut and the site overhead power line network at the first overhead power line pole from the Power Station.

As it utilizes the OPGW, the fiber optic cable between the Telstra Hut and Process Plant will not have a second redundant cable, although some redundancy will be provided by using additional fiber cores in the OPGW. Redundancy requirements will be investigated and implemented during the detail design stage.

18.2.3.2 Sub-Area 4311 – Phones

Telephone communications will be via digital VoIP technology. This allows telephone calls to be made over an Internet Protocol (IP) network rather than through a separate copper network. Calls can traverse the company's Information Technology (IT) network or an external portal.

18.2.3.3 Sub-Area 4312 – Radios

Refer to **Section 18.3.5 – Area 5800 – Communications**.

18.2.3.4 Sub-Area 4313 – Telemetry

A Radio Telemetry System will be used to communicate to remote locations that require data exchange between the Process Plant and the remote location. Radio Telemetry will be provided to communicate with the decant water return pump station, ANFO Facility and pit dewatering pump station.

The system will incorporate a Master Telemetry Station, located in a switchroom of the Process Plant, and a number of remote Telemetry Stations, located in remote equipment switchboards.

The Master Telemetry Station will communicate with the Plant Process Control System via the preferred communications network and will communicate with the remote locations via radio. Suitable antennas will be installed at each location.

Control of the remote equipment will be made by the Plant Process Control System, with sufficient data exchange to ensure correct operation of the remote equipment.

18.2.4 Area 4400 – Tailings Dam

A total of 202 Mt of process tailings will be stored in two separate tailings storage facilities (TSFs) over a design operating life of 13 years at a nominal ore processing rate of 50,000 tpd. The starter embankments for the existing TSF 1 were constructed during active mining operations between 1996 and 2000. A total of approximately 9 Mt of ore was processed during this period (MWH, 2006). Approximately 87 Mt of additional tailings will be stored in the existing TSF 1 through staged raises of the existing facility constructed using a combination of centerline and upstream construction techniques. TSF 2 will be constructed east of the Process Plant and raised in stages using upstream construction techniques. A total of approximately 114 Mt of tailings will be deposited in TSF 2. The embankments for TSF 1 and TSF 2 will be constructed using non-acid generating waste rock from the open pit operations.

Table 18-1: 50 ktpd TSF 1 and TSF 2 Parameters

TSF 1 Design Parameter	Value
TSF 1 EXPANSION	
Design Tailings Storage Capacity	87.4 million tonnes
Average Tailings Dry Density	1.5 t/m^3
Design Life	12 years
TSF 2	
Design Tailings Storage Capacity	114.7 Mt
Average Tailings Dry Density	1.5 t/m^3
Design Life	13 years

The design storage capabilities for TSF 1 and TSF 2 were based on an assumed average in-place dry density of 1.5 t/m^3. Conventional thickened slurry tailings will be pumped to the TSFs at a nominal rate of 50,000 tpd. Tailings will be deposited within the TSF using subaerial deposition techniques through multiple spigot points along the perimeter embankment crest of the TSF.

The existing TSF 1 is a side-hill type conventional slurry tailings storage with perimeter embankments constructed using mine waste and select borrow materials. The existing TSF 1 embankment is referred to as the Stage 1 embankment. The existing facility incorporates an extensive underdrainage system and decant towers with gravity drainage pipes that penetrate the perimeter embankment and connect to an external water collection pond. The existing embankment will be initially raised by the centerline method using mine waste and select borrow material. This approach provides for a robust platform for future raising construction. Subsequent embankment raises will be constructed using mine waste and upstream methods. The TSF 1 raises will be constructed in an alternating sequence with construction of TSF 2 starter and raises. This alternating sequence was adopted to provide adequate time for tailings consolidation and strength gain to permit upstream raising construction. The installation of wick drains in the foundation of each tailings raise is planned to improve the tailings consolidation rate, reduce risks associated with upstream embankment raising construction, and improve water recovery from the deposited tailings.

The TSF 2 starter embankment will be constructed using mine waste and select borrow material after the TSF 1 Stage 2 raise is completed and operational. The TSF 2 embankment will be raised by upstream methods and using mine waste. TSF 2 raises will be constructed in an alternating sequence with construction of TSF 1 raises. Similar to TSF 1, this alternating sequence was adopted to provide adequate time for tailings consolidation and strength gain to permit upstream raising construction. The installation of wick drains in the foundation of each tailings raise is planned to improve the tailings consolidation rate, reduce risks associated with upstream embankment raising construction, and improve water recovery from the deposited tailings.

18.2.5 Area 4500 – Waste Disposal

Sewage waste disposal will be via a Waste Water Treatment Plant (WWTP) installed at the Process Plant Area. The Mine Support and Process Plant buildings will be connected to the WWTP via the sewer pipework reticulation system.

18.2.6 Area 4600 – Plant Mobile Equipment

The plant mobile equipment to be purchased for the Base Case process plant will be as follows:

Table 18-2: Mobile Equipment for Process Plant

Light Vehicles	Quantity
Landcruiser wagon	2
Dual cab Ute	11
Tray top Ute	9
Troop carrier (ambulance)	1
Bus/troop carrier (15 seater)	1
Coach	3
Subtotal	**27**
Loader – Cat 966G	Allowed for in mining
Tool Carrier – Cat IT28	1
Bob Cat – Mustang Case	1
Crane – 15t Franna	1
Hiab Truck – 7t	1
Service Truck – 2t	1
2t Forklift – allowance	2
25t Container Forklift	1
80t Crane	1
Subtotal	**9**

18.3 Facility 5000 – Project Infrastructure

This section provides a description of the Project infrastructure required for the construction and operation of the process plant.

18.3.1 Area 5100 – Site Preparation

Bulk earthworks for the Process Plant will be designed to minimize the import of fill material. Where fill material is required to be imported, material from the existing RoM Pad ramp and from the existing stockpile located adjacent to the Tollis and Golf Pits will be utilized.

The site will be prepared such that there is a mono slope fall from the proposed boundary of the pit toward the existing drainage channel on the east side of the proposed process plant. To minimize the extent of stormwater run-off across the plant site, cut-off drainage channels will be installed to divert stormwater run-off around the plant. This will also minimize underground drainage and depth of open channels required on the plant site. A settling pond will be located north of the stockpile and is designed to minimize solids overflowing into the drainage channel.

Stormwater channels will be designed to collect water alongside the unsealed plant roads and direct them beneath the roads via corrugated steel culverts to prevent scouring of plant roads. All stormwater run-off will be directed toward the existing drainage channel on the east side of the proposed process plant. Rip-rap protection to earthwork embankments adjacent to the existing drainage channel on the east side of the proposed process plant will also be installed for flood protection.

18.3.2 Area 5200 – Support Buildings

The Support Buildings consist of the building infrastructure for the Process Plant. The support building sizes and number of operations personnel has been developed for the Base Case.

18.3.2.1 Sub-Area 5210 – Administration Offices

The Administration Offices will be complexed with multiple transportable buildings and used by plant management and administration personnel and is located at the northern end of the Process Site. The building will include necessary system furniture and provide cellular and open planned offices along with conference and meeting spaces.

The footprint of the Administration Offices is approximately 14.4 m by 29.7 m and will be sized to accommodate 30 people.

18.3.2.2 Sub-Area 5211 – Process Plant Offices

The Process Plant Offices will be complexed with multiple transportable buildings located within the existing Flotation building. The buildings will include the necessary system furniture and provide cellular and open planned offices.

The Process Plant Offices will be sized to accommodate 17 people per shift.

18.3.2.3 Sub-Area 5220 – Workshop / Warehouse

The Workshop / Warehouse will be incorporated into the existing Flotation Building along with the Process Plant Offices, Main Control Room, Crib and Ablutions and the Light Vehicle Workshop. The Offices / Ablutions / Crib facilities will be transportable building located within the annex of the building.

The existing Flotation building will require modifications to steelwork and replacement of the concrete floors. The building will be complete with services including overhead traveling crane, power, lighting, communications, compressed air, water, specialist equipment and other services necessary for the maintenance of process plant equipment and the LV fleet.

The Workshop / Warehouse will be sized to accommodate 25 people per shift.

18.3.2.4 Sub-Area 5230 – Reagent Store

The Reagent Store will consist of one Dome Shelter mounted on sea containers with a concrete floor. The reagent store will be sized approximately 16.7 m by 24.4 m, which includes four sea containers that will act as additional space for the storage of reagents.

The Reagent Store will be complete with all services including power and lighting.

The Dome Shelter will be constructed of steel frame and tensile fabric with a fabric life expectancy of 10 years.

The Reagent Yard will cover an area of 1,750 m^2 and contain a secured fenced hardstand area for the storage of sea containers used at the Reagent Store.

18.3.2.5 Sub-Area 5240 – Crib / Ablutions

The Crib / Ablutions facilities will be complexed with transportable buildings located within the existing Flotation Building. The buildings will include the necessary system furniture fixtures and fittings and will be suitable for operations and periodic shutdown personnel.

18.3.2.6 Sub-Area 5250 – Emergency Services

The Emergency Services Facilities will be a transportable building used by the First Aid and Fire and Emergency Services personnel. It will be located adjacent to the Administration Offices in the Process Plant and will be sized 14.4 m by 9.9 m. This area will include an undercover area for an ambulance bay and an area for additional services.

18.3.2.7 Sub-Area 5255 – Helipad

An allowance has been made for a bitumen helipad to be located close to the Process Plant. The helipad landing zone will be in a fenced-off enclosure and contain a wind sock. The helipad location is not confirmed at this stage.

18.3.2.8 Sub-Area 5260 – Sample Preparation and Laboratory

The Sample Preparation and Laboratory facility will be a structural steel shed with insulated metal clad walls and roof and concrete floor for the receipt and storage of samples and a transportable building containing the preparation areas, laboratory and offices for processing samples. The Sample Preparation and Laboratory building and equipment has been sized to process 450 samples/day. Sampling will be taken from various points throughout the process plant. Samples will be assayed for composition and gold loading.

18.3.2.9 Sub-Area 5270 – Gatehouse / Security

The Gatehouse / Security Facilities will be a single transportable building used by security personnel for recording movement to and from the Site and drug and alcohol testing of contractors and employees. The facility will include a boom gate, pedestrian turnstile and swipe card access. The Gatehouse will be located along the access road to the Process Plant.

18.3.2.10 Sub-Area 5280 – Control Building – Crushing

The Crushing Control Room will be a single transportable building located at the Primary Crusher. The buildings will include the necessary system furniture for one operator.

18.3.2.11 Sub-Area 5281 – Control Building – Main Control Building

The Main Control Room will be a single transportable building located within the Flotation Building. The building will be sized 11 m by 3 m and include the necessary system furniture for one supervisor and three operators.

18.3.2.12 Sub-Area 5282 – Control Building – CIP

The CIP Control Room will be a single transportable building located on top of the Leach Tanks and subdivided into a Control Room and a Titration Room. The building will be sized 9.6 m by 3 m and include the necessary system furniture for one supervisor and two operators.

18.3.2.13 Area 5300 – Access Roads, Parking and Laydown

The existing Plant Access Road is suitable for the Base Case. Miscellaneous road repairs will be to the existing Plant Access Road.

The existing Process Plant Retention Pond corrugated steel culvert crossing the drainage channel on the east side of the proposed Process Plant is suffering from corrosion. These corrugated steel culverts will be replaced.

18.3.3 Area 5400 – Heavy Lift Cranage

Heavy lift cranage covers the cranage that will be needed on site during the construction period for the heavy lifts on site, approximated as follows:

Table 18-3: Heavy Lift Cranage Requirements

Crane	Duration (Hours Per Year)
600 t	270
450 t	470
200 t	540
180 t	540
100 t	810
80 t	3090
50 t	1610

18.3.4 Area 5600 – Bulk Transport

Bulk transport in and out of site will be weighed on a weighbridge near the gatehouse. The weighbridge will be located on a dedicated off take from the main road. The site weighbridge will be capable of weighing a triple trailer tanker or truck.

18.3.5 Area 5800 – Communications

18.3.5.1 Sub-Area 5810 – Site-wide Radio Communications

The site will require radio communication for both individual division usage and also across all site personnel for emergencies. Some divisional usage will be localized, but coverage across the site will generally be required.

To cover all radio communications requirements across the site, there will be a suitably located, approximately 50 m tall, communications tower complete with appropriate antenna arrays and ancillary equipment. A communication hut will be located at the base of the tower. This hut will house the repeaters, servers, communications equipment and back-up batteries to provide a robust radio communications system. A maximum of eight individual radio channels will be provided.

Depending on the final location, the communications hut will either be connected to the overhead power line network or, in the case where this is not practicable, a solar powered power supply will be provided. The communication hut back-up battery life will last for a minimum of 10 hours on loss of incoming power.

The radio system will include the following radio quantities for individual personnel and vehicle usage:

- 320 hand held radios and spare batteries
- 50 mobile (vehicle) radios complete with battery charger, remote speaker/microphone and antennas
- 10 base station radios complete with battery charger, remote speaker/microphone and antennas
- 50 multi-bay chargers for portable radios.

18.4 Facility 6000 – Permanent Accommodation

Permanent accommodation for plant operating staff will be in the town of Katherine at the discretion of operators. A portion of the camp will remain after the construction period for temporary accommodation for staff, fly-in maintenance teams and shutdown personnel. Refer to **Section 18.5.1 – Area 7300 – Construction Camp** for the permanent camp details.

18.4.1 Area 6100 – Personnel Transport

A bus transit area consisting of three bus shelters will be constructed in the town of Katherine for transport of operators to and from site. This is to ensure staff will not be driving from the Mt Todd mine site to Katherine after 12-hour shifts.

18.5 Facility 7000 – Site Establishment and Early Works

The site establishment will occur prior to the operation of the Construction Camp with the hire / purchase of EPCM Contractor and Client Offices / Crib / Ablutions for the duration of the project. The facilities will be located at the Process Plant Area.

The early works will require a 'Fly Camp' for bulk earthworks and services Contractors. This accommodation has been allowed for at the town of Katherine for 40 people for three months to complete the early work at the Construction Camp Facilities and Process Plant Area.

18.5.1 Area 7300 – Construction Camp

The Construction Camp will be sized for 390 construction workers for the Base Case based on the manning histogram developed for the Project. The Construction Camp will be located approximately 2 km from the access road. The final location of the camp will be determined during the Feasibility Study.

The Construction Camp will be hired for the 24-month construction duration with the exception of 60 rooms which will be purchased from the outset. Bulk earthworks and all services including power, communications, water and sewerage will be completed prior to the arrival of the hire buildings.

The accommodation village will consist of the following building and services:

- 390 rooms certified in accordance with the Building Code of Australia
- First Aid
- Laundry Buildings
- Male / Female Ablutions
- Dry Mess including Kitchen / Dining / Crib Facilities
- Wet Mess
- Ice Rooms
- Administration Building
- Covered Outdoor Area
- Gymnasium Building
- Power Supply and Distribution
- Communications Nodes and Distribution
- Potable Water and Reticulation
- Fire Services
- Organic Materials Waste Dump
- Waste Water Treatment Plant
- LV Parking Area and Bus Drop Off / Pick Up
- Unsealed Access Road

18.6 Facility 8000 – Management, Engineering, EPCM Services

Facility 8000 will cover the indirect costs associated with the management of the project from detailed design through to handover to operations. Included within this section will be the EPCM team, external consultants, commissioning team, owner's team and any costs for licenses, fees, legal costs and insurances.

18.6.1 Area 8100 – EPCM Services

This area includes the costs for engaging the services of one or more contractors to perform the engineering, procurement and construction management for the project. The costs in this area have been derived by way of a bottom-up estimate.

18.6.2 Area 8200 – External Consultants/Testing

This area is a Prime Cost (PC) Sum that allows for the engagement of any environmental, Human Resources/Industry Relations or Health, Safety, Environment and Community (HSEC) consultants that might be required through the execution phase of the project.

18.6.3 Area 8300 – Commissioning

Area 8300 is concerned with the costs for the management and engineering associated with commissioning and was derived, for the Process Plant, as 3% of the total mechanical equipment supply costs.

18.6.4 Area 8400 – Owners Engineering/Management

Area 8400 contains costs associated with the owner's team located either on site and or in the project office.

18.6.5 Area 8800 – License, Fees and Legal Costs

This area contains a PC Sum for the costs of licenses, fees and legal costs that would need to be expended throughout the execution phase of the project. Additional costs to this area may need to be incorporated by Vista based on information that is not yet known.

18.6.6 Area 8900 – Project Insurances

Project insurances are a PC Sum included to allow Vista to take out any insurances that are deemed necessary to ensure project success. The amount of funds to be included in this area will be dependent on Vista's criteria for an acceptable risk profile and, as such, is subject to interpretation by Vista.

18.7 Facility 9000 – Preproduction Costs

Facility 9000 will cover the indirect costs associated with direct labor during commissioning, the purchase of spare equipment and replacement of equipment damaged during commissioning. Areas 9600 to 9900 are sums of money associated with working capital, corporate reserves, escalation and exchange rate fluctuation, contingency and management reserve.

18.7.1 Area 9100 – Preproduction Labor

Preproduction labor covers the costs that are not part of Construction Contracts, not part of Commissioning, not part of post-handover operating ramp up costs but are for costs that may arise prior to operations taking over the Project in an operating context. This area is proposed for minor plant modifications and additions deemed necessary to achieve Project handover status.

18.7.2 Area 9200 – Commissioning Expenses

Commissioning expenses is intended to cover the power, materials, labor and spare parts that are associated with making plant modifications, additions and operations during the commissioning period.

18.7.3 Area 9300 – Capital Spares

Capital spares are all spares which are non-consumables. These are large items that are not expected to be used; however, these items must be kept in spare for the project due to long lead times, high cost and process importance. These items include, but are not limited to, spare mill motors, HPGR motors, HPGR rolls, intertank screens, conveyor drives, pulleys and belts.

18.7.4 Area 9400 – Stores and Inventories

Stores and Inventories allows for a first fill of the primary warehouse for smaller items that are replaced frequently, including but not limited to valves, flanges, pipe fittings, pulleys and 'V' belts.

18.7.5 Area 9600 – Working Capital and Finance

Working Capital and Finance will be an allowance for a sum of money to be left for use after the plant is operational before the revenue stream is stable. Costs for this have not been included by Proteus, as this provision has been included by Vista in the Technical Economic Model.

Project working capital provides for estimated normal timing delays associated with receipts and disbursements of cash, with such amounts being fully recovered by the end of the project life. An additional non-recovered working capital amount provides for final owner's closeout expenditures.

A corporate reserve will be required to support, if necessary, Project operations after the plant is operational but before revenues are sufficient to generate positive and stable cash flows. No corporate reserve was included in the estimate as this provision will be made by Vista.

18.7.6 Area 9700 – Escalation and Foreign Currency Exchange

Escalation and Foreign Currency Exchange allowances will be necessary to cover potential inflation and fluctuation of foreign currencies from the date of this study until actual transaction dates. Such allowances have not been included in the estimate as provision for this will be made by Vista.

18.7.7 Area 9800 – Contingency Provision

The contingency provision covers those items within the scope that are known to exist but have not yet be defined. Contingencies are estimated on a line item by line item bases in the TEM.

18.7.8 Area 9900 – Management Reserve Provision

The management reserve provision is a measure of the accuracy of this cost estimate and is a portion of additional money that would not be available to the project manager but will be held in reserve by Vista to cover unforeseeable and uncontrollable events including, but not limited to: strikes, unusual weather conditions, premium payments arising from accelerated construction programs to recover lost time. A reserve for such potential costs has not been included in the estimate as provision for this will be made by Vista.

18.8 Electric Power Plant

The mine's electrical power demands are estimated to be approximately 70 MW for the Base Case based upon the load list provided by TTP dated November 27, 2017. Electrical demand will be met through the installation of seven Jenbacher J920 reciprocating gas engines to meet the power demand for the Base Case. Water consumption in this power plant arrangement is very small and primarily for makeup of the closed loop engine cooling system and general housekeeping washdown. It is estimated that intermittent water use will be up to 11.5m^3/hr (50 gpm).

Two potential locations are considered for the location of the power station. *Option 1* is locating the power station near the entrance to the mine and includes connection to the existing natural gas pipeline spur with the shortest powerline connection to the mine. This pipeline spur requires a tolling fee of AUD0.60 per gigajoule (GJ) of gas bringing the wholesale price to AUD7.00 per GJ.

Near the main gas transmission pipeline is an alternate *Option 2* location for the power station that will avoid the tolling fee for the spur pipeline but requires the upgrade of the 10 km electrical powerlines with new conductors and as well as new towers designed to carry the additional weight of conductors rated for a higher megavolt-ampere (MVA) class. The *Option 2* location will also require additional infrastructure for non-potable service water and fire protection, which would entail a non-potable service/fire water storage tank, firewater pumps, and non-potable service water pumps.

Figure 18-1: Power Station Location

The plant required net output is the mine's required electrical demand plus the power stations auxiliary loads and is based on the preferred generation technology.

Table 18-4: Power Station Location Budgetary Comparison

Mine Output	Base Case
Generator Manufacturer	GE-Jenbacher
Model	J920 (7)
Plant Required Net Output	69.6 MW
Estimated Fuel Demand (GJ/yr)	5,253,349

For the purpose of this Preliminary Feasibility Study, the *Option 1* location will be carried forward.

18.8.1 Generation Option Selection

POWER Engineers (POWER) performed an evaluation of generation options based upon providing reliable power for a steady load demand with minimum onsite personnel requirements and low life cycle operating costs. The evaluation has concluded that natural gas-fueled reciprocating engines provide the most economical and reliable means of power generation at the mine site. POWER recommends that the Base Case electrical demand be met with seven reciprocating engines with backup power being provided by the electric utility grid. Gas reciprocating engines will provide excellent reliability, increased operational flexibility and redundancy required by the Project. If startup loads exceed the capacity of the initial engines, the utility grid connection is available for supplemental power.

The technology configuration for this study uses the General Electric Jenbacher J920 engine but there are a number of commercially available gas engine models in the 10MW range with operating plants in-country to provide a technically sound field to select a reliable equipment supplier with competitive pricing and local technical support.

18.8.2 Mt Todd Electrical

18.8.2.1 Conceptual Design

A conceptual electrical one-line diagram, **Figure 18-2**, has been created to show the electrical distribution system from the 33kV utility interconnect down to the 400V power distribution bus. The equipment ratings are preliminary and based on generator ratings provided by Jenbacher and budgetary quotes for the balance of plant equipment.

The equipment ratings shown are for cost estimating purposes only. The actual equipment ratings will be determined using detailed load flow and short circuit studies during detailed design.

18.8.2.2 Plant Arrangement

The auxiliary electrical equipment is included on the mechanical general arrangement drawing (**Figure 18-3**). All the equipment physical sizes are based upon similar equipment from reference projects.

18.8.2.3 Step Load Capability

Reciprocating engine generators are designed for fast startup times, fast ramp rates and flexibility. For best performance when starting and fast ramp times, the engine must be pre-heated or operating at low loads with the cooling water at 55°C (by onboard electric heater) to avoid engine damage. A single 10 megawatt (MW) engine can accelerate from start to 100% load in 5 minutes with ramp up as fast as 100 kilowatts per second (kW/sec). If a large motor at the mine is brought online, the starting load can be spread across all engines in the power station for a ramp rate as high as 700 kW/sec for the Base Case with the connection to the electric utility grid available for supplementary starting support.

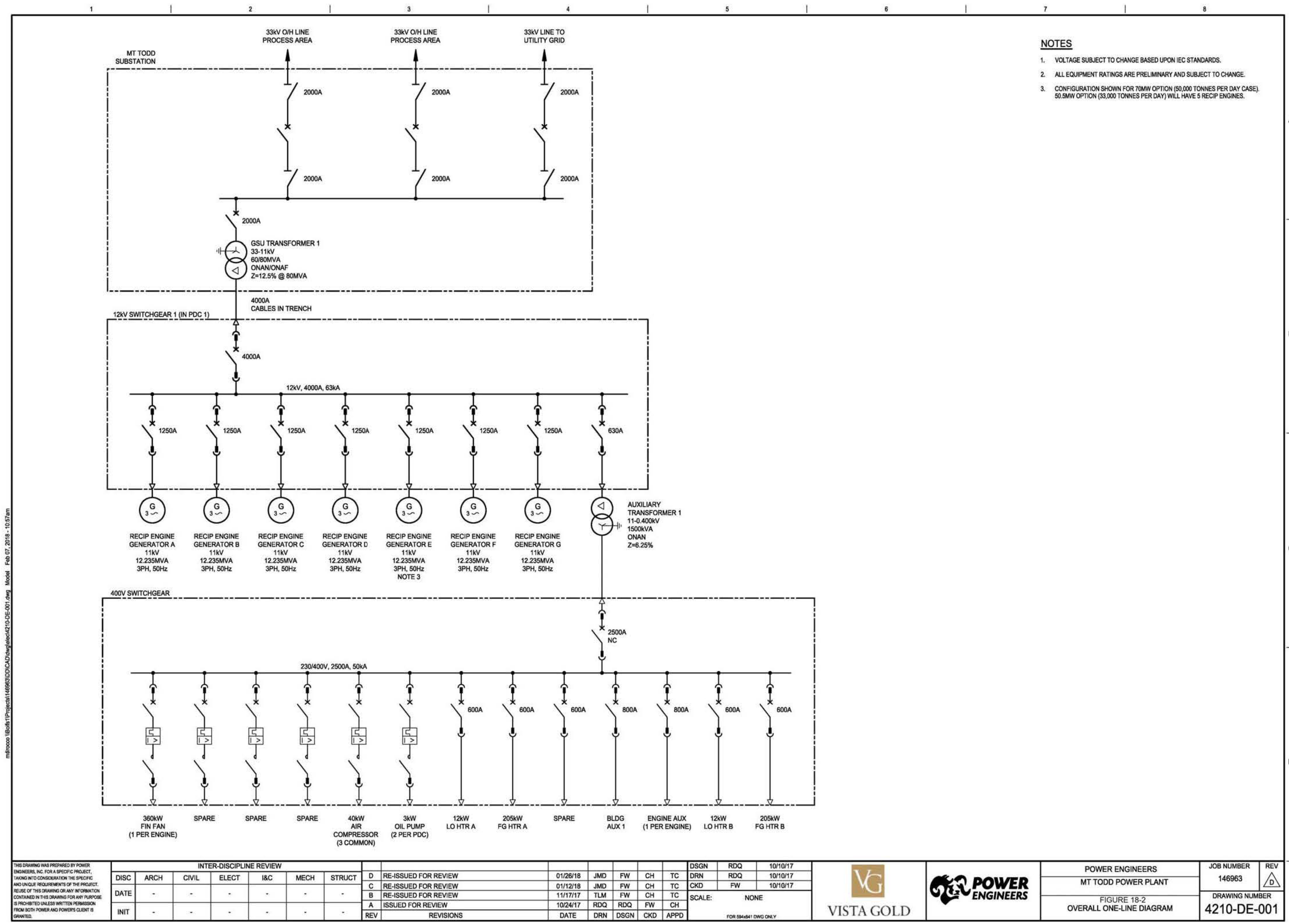


Figure 18-2: Conceptual Electrical Line Diagram

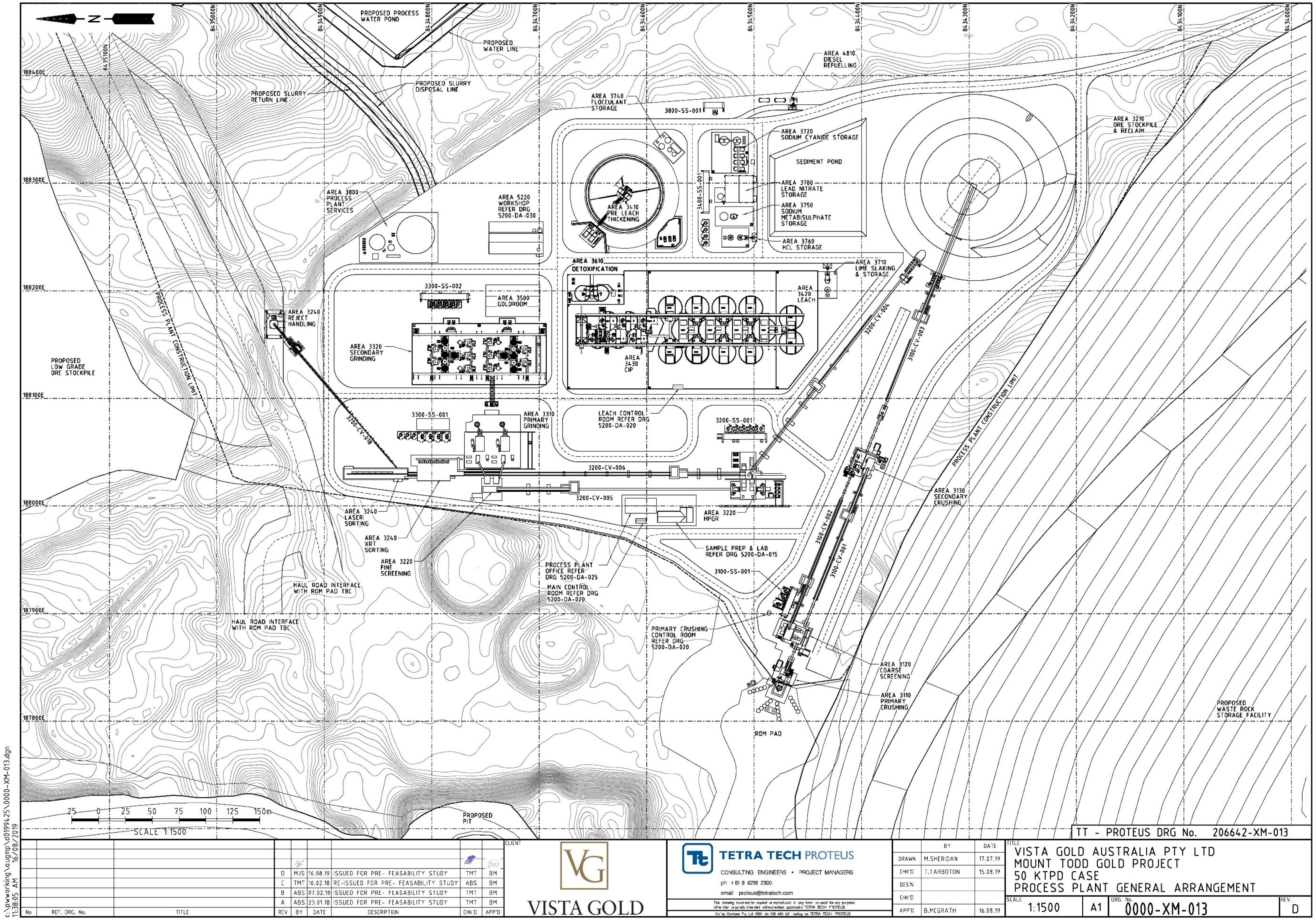


Figure 18-3: General Plant Arrangement

19.0 MARKET STUDIES AND CONTRACTS

19.1 Markets

Gold metal markets are mature, with many reputable refiners and brokers located throughout the world. The advantage of gold, like other precious metals, is that virtually all production can be sold in the market. As such, market studies, and entry strategies are not required.

Metallurgical process studies confirm that the Project will produce doré of a specification comparable with existing operating mines.

Demand is presently high with prices showing remarkable increases during recent times. The 36-month average London PM gold price fix through August 31, 2019 was US$1,279/oz.

19.2 Contracts

Currently, there are no contracts in place for development and operations. However, Vista has obtained budgetary quotes, as is common for PFS level studies, for future materials and service needs. The following contracts are expected to be in place upon project commencement:

- Secure doré transportation to refinery;
- Doré refining;
- Supplier and service contracts including;
  - EPCM;
  - Equipment supply;
  - D&C;
  - Diesel and fuel oil;
  - Natural gas for the power plant;
  - Process reagents;
  - Equipment preventive maintenance and repair (MARC) services;
  - Site security services; and
  - Camp management, catering and support services.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

This section discusses the environmental permitting and social impact aspects of the Project. The EIS was submitted in June 2013. The Northern Territory Environmental Protection Authority (NTEPA), as the responsible government authority to advise on the environmental impact of development proposals, provided its final assessment of the Project in September 2014.

In January 2018, the “authorization of a controlled activity” was received for the Project as required under the Australian Environmental Protection and Biodiversity Conservation Act of 1999 (EBPC) as it relates to the Gouldian Finch, and as such has received approval from the Australian Commonwealth Department of Environment and Energy.

In November 2018 the “Mine Management Plan” (MMP) was submitted to the Northern Territory Government Department of Primary Industry and Resources (DPIR). This is the last approval required before works can occur. The approval of the MMP will result in the “Mining Authority” being issued.

20.1 Environmental Studies

A number of environmental studies have been conducted at the Project in support of development of EISs and as required for environmental and operational permits. Studies conducted have investigated soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology, and water quality.

The Mt Todd Project Environmental Impact Statement (EIS) submitted June 28, 2013 to the Northern Territory Environment Protection Authority (NTEPA), approved in September 2014, provides an understanding of the existing environmental conditions and an assessment of the environmental impact of the Project.

Key issues of concern regarding the Project impacts that were addressed in the EIS include:

- Acid and metalliferous drainage (AMD) seepage and runoff from the waste rock dump, ore stockpiles and tailings storage facilities potentially contaminating surface and ground waters continuing long after the mine has ceased operation;
- Potential contamination of surface water from AMD causing adverse impacts on downstream water quality, aquatic environment and downstream users;
- Management and treatment of a large quantity of acidic and metal laden water currently existing on the site;
- The proposed WRD covers an approximate area of 217ha with an estimated height of 160m. Final design of the WRD must ensure the structure is safe, stable, not prone to significant erosion, minimizes AMD seepage and runoff and meets stakeholder expectations as a final land use structure;
- Biodiversity impacts, including matters of environmental significance, associated with disturbance footprint of mining activities and infrastructure requirements;
- The challenges of successful mine closure and rehabilitation; and

- Potential social, economic, transport and heritage impacts.

The Project is located in the Pine Creek Bioregion and part of the Yinberrie Hills Site of Conservation Significance (SOCS30). Each of these potential impacts were assessed and mitigation or management measures were outlined in the EIS.

20.1.1 Flora and Vegetation

Eight vegetation types covering 5,462.56ha were mapped in the Mineral Leases. *Eucalyptus tectifica, E. latifolia, E. tintinnans, E. spp. Woodland; E. phoenicea, Corymbia latifolia* low woodland – woodland (scattered *E. tintinnans*); and *C. dichromophloia, E. tintinnans, Erythrophleum chlorostachys* Woodland covers 80% of the site. The Project is not expected to significantly impact vegetation in the area.

Eight-hundred and forty species of flora are known to occur within 10km of the leases. The 2011/12 surveys identified 226 taxa, of which 67 were not recorded from previous surveys. The total number of species known from the area is 959. The only threatened plant species recorded from the area is the bladderwort, *Utricularia singeriana*. This species is listed as Vulnerable under the Territory Parks and Wildlife Conservation (TPWC) Act 2000. The closest known record is 6 km west of the Mineral Leases. The Project is not expected to have an impact on any threatened flora.

20.1.2 Nationally Threatened Fauna

Threatened fauna species are those that are listed as threatened (or a related category) under the Commonwealth EPBC Act and/or Northern Territory's TPWC Act.

Eighteen threatened fauna species that do or could occur within the mine site include:

- Six mammals;
- Eight birds;
- Three reptiles; and
- One fish.

Six of the eighteen threatened species have recorded in the mine site during field assessments.

20.1.3 Migratory and / or Marine Species

Fourteen EPBC Act listed migratory bird species potentially occur within 10km of the project area. Ten have been recorded from the leases. Seven EPBC listed marine species potentially occur with 10km of the project area. This includes six bird species and one reptile species. The freshwater crocodile was recorded in the leases. None of the listed marine species is likely to have a high risk of impact from the proposed development.

20.1.4 National Heritage Places

The Yinberrie Hills is a Site of Conservation Significance and was placed on the Interim Register of the National Estate for its natural values. However in 2007 the Register of the National Estate was declared no longer a statutory list.

Surveys located 20 archaeological sites. The most significant was Mt Todd 26 – an extensive greywacke quarry, extraction and reduction site, one of the largest recorded in the Northern Territory. The remainder were lithic scatters or quarry and reduction sites with low to medium heritage significance.

With respect to Jawoyn Resource Knowledge, 62 animal, 63 plant and one fungal taxa were identified and the associated Jawoyn knowledge recorded. Amongst the Jawoyn, the mine site is not considered a notably productive environment. Plants and animals encountered and discussed during the ecological knowledge consultation are widespread and not unique to the mine site. Vista employs Jawoyn Rangers for reviewing and potentially clearing any heritage sites prior to disturbance.

20.2 Waste and Tailings Disposal, Site Monitoring and Water Management

20.2.1 Waste Rock Disposal

Waste rock will be disposed of in a WRD constructed as an expansion of the existing WRD. All waste rock will be analyzed to identify the rock as potentially acid generating (PAG) or non-PAG material before being hauled to the WRD. Non-PAG material will be stockpiled for use in reclamation covers or placed in the WRD. Construction of the WRD is described in **Section 16.0 – Mining Methods**. Reclamation and closure of the WRD is described in **Section 20.5 – Mine Reclamation and Closure**.

20.2.2 Tailings Disposal

Tailings will be disposed of in two tailings storage facilities, TSF 1 and TSF 2. TSF 1, an existing tailings storage facility, will be expanded with eight additional raises to the embankment and construction of two new saddle dams at the west end of the impoundment. A second tailings storage facility, TSF 2, is to be constructed after re-commissioning of TSF 1. The engineered containment system for the TSF 2 impoundment includes a 60-mil linear low-density polyethylene (LLDPE) textured (double sided) liner and a tailings overdrainage collection network to mitigate the risk of seepage. Tailings decant water and water collected in the TSF seepage interception network will be treated in the water treatment plant or used for the process plant. Construction of the tailings storage facilities is described in **Section 18.2 – Facility 4000 – Project Services**.

Reclamation and closure of the TSFs is described in **Section 20.5 – Mine Reclamation and Closure**.

20.2.3 Site Monitoring

Currently, surface water monitoring is conducted at various locations at the site. A comprehensive site monitoring plan has been incorporated into the MMP.

20.2.4 Water Management

The primary existing environmental issue at the site is water management resulting from the project shutdown without implementation of closure or reclamation activities. The pit and existing water RPs (excluding the raw water pond) contain acidic water with elevated concentrations of regulated constituents. This water has been managed through evaporation, pumping to the Batman Pit for containment, micronized lime treatment of the pit lake, and controlled discharge of treated water to the Edith River in accordance with the approved WDL. Historically, wet season rainfall resulted in short-term uncontrolled overflow from retention ponds to the Edith River due to the high amount of precipitation received in short periods of time coupled with insufficient pumping capabilities. Current water management strategies employed by Vista appear to be successful at preventing recurrence of historic uncontrolled discharges and are minimizing impacts on the Edith River downstream of the Project Site.

Prior to, during, and following resumed mining operations, water management at the site involves distinct water management components including in-pit treatment, seepage management, treatment of acid rock drainage and metal laden leachates (ARD/ML), and surface water management. Each of these components is discussed in the subsections below:

20.2.4.1 In-situ Pit Treatment

In-situ treatment of the Batman Pit (RP3) was conducted by use of limestone and quicklime. Treatment has been undertaken to produce water to be discharge at rates protective of water quality in the Edith River in a suitable timeframe to meet project requirements. The treatment methodology included raising the pH of the water within the pit lake to greater than pH 8.0 using limestone and quicklime in succession to capitalize on the capabilities of the low-cost limestone and minimize the quantity of quicklime required to attain a pH sufficient to precipitate additional metals. Raising the pH to greater than 8.0 will result in the precipitation of key metals of concern including iron, aluminum, chromium, copper, lead, nickel, cadmium, cobalt and zinc. On an ongoing basis, quicklime is used to buffer the pH as required on an annual basis.

20.2.4.2 Seepage Management

A thorough assessment of the infiltration and seepage conditions of the WRD, HLP, TSF 1, ore stockpiles, and other site facilities has not been well characterized at the current time but will be foundational to developing the site water management plan. The infiltration and seepage assessment will be included in the comprehensive site environmental system model (hydrogeologic, geologic, seepage, and geochemical conceptual models) to understand the solute-transport processes at the site and possible impacts to the aquifer from mine operation. Numeric modeling will be used for the infiltration and seepage assessment.

20.2.4.3 Ongoing ARD/ML Water Treatment

Water treatment for the project will involve active water treatment for ARD/ML. Active water treatment will occur prior to operations, as part of rehabilitation of the site necessary to restart mining, during mining operations, and for a period following cessation of operations. Passive water treatment will be conducted at the site following closure in addition to use of the active water treatment plant as required.

Active water treatment at the site has been described in **Section 24.0 – Other Relevant Data and Information**.

Passive water treatment will be conducted in four separate passive treatment systems which include (in total) one biochemical reactor (BCR), four aerobic polishing wetlands (APW) and three aeration/settling ponds (AP). The goals of the passive/semi-passive water treatment at Mt Todd are to:

- Eliminate or drastically curtail the costs and continual inputs (e.g. reagents, power, staff) required to operate and maintain the active WTP;
- Eliminate sludge disposal operations and maintenance associated with active water treatment;
- Collect, contain, and treat ARD/ML prior to effluent release year-round; and
- Ensure that treated ARD/ML complies with the WDL numeric water quality standards.

The passive water treatment technology recommended for treatment of WRD seepage, which is predicted to be net-acidic ARD/ML, is primarily metal-sulfide and metal-hydroxide precipitation via sulfate-reduction and the concomitant rise in solution alkalinity. The passive water treatment technology recommended for treatment of seepage from the TSFs, which is predicted to be net-alkaline ML, is aeration (oxidation) in aeration/settling ponds (Aps) to allow metals to precipitate and settle. Effluent from the APs will be further aerated and treated prior to release to the environment in aerobic polishing wetlands (APWs) where the concentration of dissolved metals should be further reduced through complexation to plant-derived organic substrate, and potentially, accumulation in plant tissue.

The treatment capacity of the four separate passive water treatment systems range from 10 to 50 m^3/hour, which should be adequate to treat the anticipated rate of seepage from the WRD and TSFs following closure. The quantity of seepage from the WRD and TSFs following closure was estimated by simply multiplying the predicted infiltration of daily precipitation through the proposed WRD and TSF closure covers by the ultimate two-dimensional surface area of each facility. Using stochastic precipitation developed in the water balance model from site and Katherine gage data statistics, 1000 simulations (realizations) of daily precipitation were calculated in GoldSim at the following probabilities: 0%, 1%, 5%, 15%, 25%, 35%, 45%, 50%, 55%, 65%, 75%, 85%, 95%, 99%, and 100%. The mean of these precipitation probabilities was then calculated to represent daily precipitation. To estimate the daily seepage rate from each facility the calculated mean daily precipitation was multiplied by the ultimate facility surface area and the estimated rate of infiltration through the closure cover.

Estimating flows and water quality 20 years in the future is wrought with uncertainty. These and other uncertainties inherent to passive water treatment are magnified by changes in mine plans and changes in closure plans and designs, which occur during normal operations, as well as unpredictable circumstances such as changes in climatic conditions, unforeseen material characteristics, etc. Therefore, the estimates and recommendations provided at this time should be considered preliminary and design parameters such as: hydraulic retention time; biochemical oxygen demand removal rate; metals and metal-precipitates removal and settling rate; and reactive substrate type, quantities, depletion rate and permeability overtime must be checked and updated or entirely modified as the project progresses and more information becomes available.

20.2.4.4 Surface Water Management

Surface water at the site is well-documented and its management has been the object of study by both Vista and the NT Government in recent years. Surface water management is described further in **Section 24.4 – Surface Water Hydrology**.

20.3 Permitting and Authorizations

On January 1, 2007, Vista became the operator of the Project Site and accepted the obligation to operate, care for, and maintain the assets of the NT Government on the site. Vista developed an Environmental Management Plan (EMP) for the care and maintenance of the Mt Todd mine site in accordance with the provisions of the Mineral Leases 1070, 1071, 1127 and 31525 granted under the Mining Act. The EMP identified the environmental risks found at the Project Site at its then present state of operations and defined the actions for Vista to take to control, minimize, mitigate, and/or prevent environmental impacts originating at the Project Site. As part of the agreement, the NT Government acknowledged its commitment to rehabilitate the site and that Vista has no obligations for pre-existing conditions until it submits and receives approval of an MMP for resumption of mining operations.

The Project requires approvals, permits and licenses for various components of the Project. **Table 20-1** includes a list of approvals, permits, and licenses required for the project and their current status.

Table 20-1: Mt Todd Permit Status

Approval/ Permit/ License	Current Status	Approval/ Permit License Date	Expiration Date
Environmental Impact Statement	The NT Environmental Protection Authority provided its final assessment of the Project in June 2014.	Approved Sep. 2014	NA
Mining Management Act (or Plan) Approval from NT Department of Mines & Energy	Mine operating permit request has been submitted. The MMP submitted in November 2018 IS for 50kt/day operations.	Prior to commencing mine operations	NA
Heritage Act permit to destroy or damage archeological sites and scatters/ Aboriginal Areas Protection Authority Clearances	Authority Certificate Number 2011/15538 issued. This certificate defined restricted works areas and granted select clearances to allow for initial investigations. Additional clearances will be required for further investigations as well as prior to disturbance associated with mine development.	Aboriginal Areas Protection Authority dated Jul. 31, 2012	NA
Dangerous Goods Act (1988) permit for blasting activities	Waiting on final mine plan	NA	NA
Extractive Permit (under DME Guidelines) for development of borrow pits outside of approved mining areas	Would be required for PGM or LPM borrow areas. Permit application not yet in progress pending final selection of borrow areas	NA	NA
Waste Discharge License (under Section 74 of the Water Act 1992) for management of water discharge from the site	WDL 178-6 licensing discharge of waste water into the Edith River from the Mt Todd mine site, granted with conditions	Nov. 26, 2018	Nov. 30, 2020
Waste water treatment system permits under Public Health Act 1987 and Regulations	May be required for the waste water treatment system for the construction and operations accommodation village. Permit application not yet in progress pending design and siting of accommodation village.	NA	NA
Approval to Disturb Site of Conservation Significance (SOCS)	Batman pit expansion will disturb SOCS as breeding / foraging habitat for the Gouldian finch, pending determination on EIS.	Jan. 22, 2018	NA

In addition, permits that are required to commence construction works will be obtained prior to any construction activity.

20.4 Social or Community Requirements

The Jawoyn people have strong involvement in the planning for the future of the Project. Vista has a good relationship with the Jawoyn. Areas of aboriginal significance have been designated, and the mine plan has avoided development in these restricted works areas.

Those parts of the JAAC agreement that are within the public domain are presented in this report; the remaining part of the agreement, which is confidential, is not presented in this Technical Report.

20.5 Mine Reclamation and Closure

A reclamation plan for the Project was developed in support of the Technical Report for renewed mining operations. This reclamation plan evaluates the reclamation activities that will be conducted for the landforms planned as part of mining commencement. Reclamation plans and strategies for each major facility at Mt Todd are briefly summarized in **Table 20-2**.

Table 20-2: Reclamation Approach

Task	Facility							
	Batman Pit	WRD	HLP	TSF 1&2 Impounded Surface	TSF 1&2 Dams (Embankments)	Process Plant and Pad	LGOS 2	Mine Roads
Surface of Facility at Cessation of Production Composed of Non-PAG Material		X			X			
Final Overall Slopes > 3H:1V*	X	X						
Final Overall Slopes < 3H:1V*			X	X	X	X	X	X
Benches Created During Construction	X	X			X			
Install minimum 1.0 m-Thick non-PAG Material		X		X				
Install 0.8 m-Thick Store and Release Cover				X	X			X
Install 0.2 m-Thick Plant Growth Medium (PGM) Cover			X	X	X	X	X	X
Revegetate with Native Seed Mix			X	X	X	X	X	X
Install geosynthetic liner (with under and overlayer of fines)		X						
Install Erosion and Sediment Controls		X		X	X	X	X	X
Construct Access Restriction Bund	X							
Additional Remedial Measures (as necessary)	X	X	X	X	X	X	X	X

** > and < indicates slopes are steeper and less steep, respectively.*
"X" denotes where the task or characteristic is applicable to the landform

Costs associated with reclamation and closure are provided in **Section 21.1.5.** In accordance with regulatory requirements, a reclamation bond will be required for the site. Calculation of bond amounts will be conducted with the NT Security Calculation excel-based worksheet periodically throughout the mine life in accordance with regulatory requirements. Costs associated with reclamation bonding have been included in the technical economic model.

20.5.1 Batman Pit

Based on a preliminary regional groundwater flow model that included enlargement of the Batman pit and post-mining recovery of the groundwater system (outlined in **Section 0– Regional Groundwater Model and Mine Dewatering**), a terminal-sink pit lake may result during the post-closure phase, making active dewatering and treatment of pit water unnecessary following closure. All water inflow to the pit lake, including precipitation, storm-water runoff and groundwater, will leave the pit lake only via evaporation. No surface water or groundwater drainage from the pit lake is expected to occur.

An access restriction berm (also termed “bund”) will be constructed around the perimeter of the Batman pit to impede human access and reduce the inflow of surface water to the pit. The safety berm will be offset 30 m from the pit perimeter per the requirements outlined in the guidelines “Safety Bund Walls around Abandoned Open Pit Mines” from the Department of Industry and Resources in Western Australia.

20.5.2 Waste Rock Dump

The existing WRD will be slightly enlarged based on plans for the resumption of mining. The WRD will be constructed at an angle of repose slope of 1.5 vertical to 1.0 horizontal, with catch benches of 8.0 meters every 30 meters in height. Each lift will be constructed with 8 m wide benches at 30 m vertical intervals on the face of the WRD.

As described in **Section 16.0 – Mining Methods**, the WRD will be constructed with an encapsulating non-PAG material outer shell on each lift. Concurrent installation of a low permeability geosynthetic liner (i.e., LLDPE or GCL) following attainment of final grades will serve to reduce infiltration of precipitation into the WRD core. This liner system will include a 0.3 m thick bedding layer of fine material to serve as liner bedding, followed by placement of the liner material, and capped with a 0.3-m thick protecting layer of fine material placed over the liner. The liner will span approximately 52 m on top of each lift, covering the 8 m bench, and running to just below the subsequent lift. The liner will be installed at five percent slopes toward the outside of the WRD, and will be constructed with a 0.5-m tall berm with 1:1 side slopes at the interior edge of the liner. A minimum 1-m thick layer of non-PAG waste rock will cover all surfaces of the WRD to aid in erosion control.

Prior to WRD grading, a seepage collection system will be constructed along the down-gradient toe of the WRD and subsequently covered with waste rock from grading activities. ARD/ML collected by the WRD seepage collection system will initially be pumped to the New WTP for treatment prior to release until it is feasible to treat this and other ARD/ML on-site using a passive treatment system.

20.5.3 Tailings Disposal Facility

The TSF embankment and impoundment surfaces will be reclaimed at closure by installing and revegetating a 1-m thick store and release cover. The 1-m thick store and release cover will consist of a 0.8-m thick layer of blended non-PAG waste rock (40%) and low-permeability material (60%), overlain by a 0.2-m thick layer of plant growth medium (PGM). Following placement, the cover surface will be roughened and revegetated with native species. The store and release cover will serve to effectively reduce percolation of precipitation into waste rock, PAG, and/or metalliferous materials.

The majority of the impounded surface of the TSF at closure will be primarily composed of thixotropic tailings (thick like a solid but flows like a liquid when a sideways force is applied) which will maintain a high degree of saturation for many years unless actively dewatered and consolidated, covered with material, or chemically treated to increase their strength. A crowned cover constructed using non-PAG and PAG waste rock and sorter reject material will result in a final tailing surface that drains and does not impound water. This crowned cover is assumed to adequately bridge the thixotropic tailings and allow for equipment to place the 1-m thick store and release cover.

To the degree possible, store and release covers will be installed concurrently during construction when portions of facilities reach final grade. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion and scour of active reclamation areas.

20.5.4 Processing Plant and Pad Area

A new process plant will be built for renewed mining. Once ore processing ceases, the process plant will be decommissioned, decontaminated, demolished and any reusable equipment and materials will be salvaged and resold. Material that cannot be treated in-situ will be excavated and disposed of in the WRD, TSF, or an off-site facility that is certified to accept and dispose of contaminated soil. Concrete foundations, building walls, and other inert demolition waste will be broken up and either:

- Placed in the WRD;
- Buried in-place; and/or
- Backfilled against cut banks and highwalls throughout the process plant and pad area, as well as other areas that will be reclaimed at Mt Todd.

Surface and large shallow pipes will be removed and pipes at depth will be plugged with concrete or other suitable materials.

The process plant area will be graded to blend into the surrounding topography and drain towards Batman Creek. The process plant area and pad will be covered with a 0.2-m thick layer of plant growth medium (PGM) and revegetated. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion.

The WTP and equalization pond (EQP) will be left in place, up-graded if necessary, and used to treat acid rock drainage and metal-laden leachates (ARD/ML) during the closure and post-closure phases. These facilities will be closed when it is feasible to treat ARD/ML in passive treatment systems.

20.5.5 Heap Leach Pad and Pond

The HLP and Pond will be left in place and reprocessed following processing of ore and low-grade ore. Following reprocessing of the heap material, the pad and pond footprint will be reclaimed by cutting and removing the liner for consolidation in TSF 2. It is anticipated that the integrity of the heap liner will have been compromised and removal of 0.5-m thick of impacted soils below the liner will be necessary. These materials would be removed and consolidated in TSF 2. The area will then be regraded to prevent ponding of water and will be covered with a 0.2-m thick layer of PGM and revegetated.

20.5.6 Low Grade Ore Stockpile

The existing LGOS1 will be eliminated during the expansion of the Batman Pit and it is assumed that no reclamation is required for the closure of this facility.

The LGOS2 will be located near the pit and the process plant area. Closure of LGOS2 will include removal of residual ore from the stockpile areas, regrading, covering the material with a 0.2-m thick layer of PGM and revegetating the area. In addition, storm-water drainage, erosion, and sediment controls will be constructed to minimize erosion. It is assumed that RP2 will be closed during the closure phase and that the LGOS will no longer be a source of ARD/ML following closure.

Any potential ARD generated during operations reports to the process water pond, and therefore the WTP.

20.5.7 Mine Roads

Mine access roads will remain in place to provide post-closure access to the area. All haul roads will be closed by grading into surrounding topography, ripping subgrade materials, placing 0.2 m of PGM (when applicable), and revegetating the areas.

20.5.8 Water Storage Ponds

Prior to construction of the active WTP, a process water pond (PWP) will be constructed for mixing of ARD/ML from various on-site sources prior to treatment and to temporarily store ARD/ML in case of system upset. All proposed and existing ponds at Mt Todd will be maintained for the collection of seepage, storm water and ARD/ML until long-term quality of water collected by the WRD seepage collection system meets applicable standards, flows to the collection system cease, or an alternative passive water treatment system is installed.

The return water, polishing and overdrain ponds for the TSFs shall remain post-closure and be incorporated into the passive water treatment system. These and potentially other ponds may be used post-closure as backup water storage in case treatment upset occurs.

To decommission and close ponds, residual standing water will be pumped to the PWP for processing by the WTP, and sediments and foundation materials will be tested to determine their chemical characteristics with acidic, PAG and metalliferous materials treated *in-situ* or buried in place. Following sediment testing and removal, pond liners will be cut and folded in place. Pond berms will be pushed into the pond void to cover the liners and until the area no longer impounds water. The top 0.6 m of graded material is assumed to have physical and chemical properties to support plant growth. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion and channel scour, and the areas will be revegetated.

20.5.9 Low Permeability Borrow Area

A low permeability borrow area will be developed to provide low permeability material for use in project feature construction and for use in reclamation. As portions of the low permeability borrow area are taken out of service and are no longer used to generate material, they will be reclaimed by ripping and amending the remaining soils with organic matter, constructing channels to route drainage within the borrow area footprint and revegetating the area. Some portions of the low permeability borrow area may also be used as stock water ponds.

20.5.10 Closure Cost Estimate

Costs for reclaiming major facilities at the Project were estimated using closure material quantities based on the Base Case ultimate designs and following the closure plans discussed above. Closure costs are accrued and contained in the financial model.

21.0 CAPITAL AND OPERATING COSTS

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

For the purposes of understanding how the mine will operate, **Table 21-2** details the Project based on the principal operating time periods.

Table 21-1: Operating Periods

Principal Assumptions	Unit	Parameter
Construction Period	Years	2
Commissioning & Ramp-Up	Years	0.5
Mine Life	Years	13
Closure Period	Years	4
Operating Days	Days / Year	355

Estimated capital and operating costs are summarized in this section and are prepared by Vista's engineers and consultants as follows:

- Open Pit Mining: MDA;
- Process Plant: Tetra Tech Proteus;
- Tailings Dam: Tetra Tech;
- Infrastructure: Tetra Tech Proteus;
- Raw Water Dam & Water Treatment: Tetra Tech;
- Reclamation: Tetra Tech; and
- Owner's Costs: Vista.

Costs are presented in Q3 2019 US dollars and are based on an US$0.70:AUD1.00 exchange rate, unless otherwise noted.

Section 21.0 presents costs as provided to JDS Energy & Mining for incorporation into the Technical Economic Model (TEM). These costs are based on their source data and in some cases use different foreign exchange rates or unit rates for fuels, etc. The cash flow results presented in Section 22 are all tied to the same foreign exchange and unit costs rates. These costs are summarized using the listed foreign exchange rate provided in **Section 22.0 – Economic Analysis**.

21.1 Capital Cost

LoM capital cost requirements are estimated at US$1,222 million as summarized in **Table 21-2**. Initial capital of US$826 million is estimated to be required to commence operations. At the end of operations, the Project will receive a US$140 million credit for remaining asset sales and salvage (reference **Table 22-13**).

Table 21-2: Estimated Capital Cost Summary (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
2000	Mining	7.3%	$121,239	$5,720	**$126,958**	$406,347	$32,677	**$439,024**	$527,586	$38,396	**$565,982**
3000	Process Plant	13.9%	$366,693	$51,073	**$417,766**	$17,027	$2,222	**$19,249**	$383,720	$53,295	**$437,016**
4000	Project Services	10.0%	$109,204	$12,681	**$121,885**	$72,448	$5,455	**$77,903**	$181,651	$18,136	**$199,787**
5000	Project Infrastructure	13.2%	$26,160	$3,463	**$29,623**	$0	$0	**$0**	$26,160	$3,463	**$29,623**
6000	Permanent Accommodation	10.0%	$60	$6	**$66**	$0	$0	**$0**	$60	$6	**$66**
7000	Site Establishment & Early Works	11.4%	$17,537	$1,995	**$19,532**	$0	$0	**$0**	$17,537	$1,995	**$19,532**
8000	Management, Engineering, EPCM Svcs	11.8%	$82,058	$9,721	**$91,779**	$0	$0	**$0**	$82,058	$9,721	**$91,779**
9000	Pre-Production Costs	12.3%	$16,121	$1,982	**$18,102**	$0	$0	**$0**	$16,121	$1,982	**$18,102**
10000	Asset Sale	0.0%	$0	$0	**$0**	($139,631)	$0	**($139,631)**	($139,631)	$0	**($139,631)**
	Capital Cost	**11.6%**	**$739,072**	**$86,641**	**$825,712**	**$356,191**	**$40,354**	**$396,545**	**$1,095,263**	**$126,994**	**$1,222,257**

21.1.1 Mining (MDA)

Table 21-3 shows the estimated mine capital requirements for the Base Case by year. The initial mine capital is estimated to be US$115 million, with a LoM capital of US$414 million. This includes capitalized operating costs of US$68 million for construction, US$20 million for pre-stripping, and US$31 million for reclamation. Note that the treatment of the capitalized mining in the final cash-flow model differed slightly, but the difference is insignificant ($4,000 less).

Table 21-3: Estimated Mine Annual Capital Costs (US$000s) – Base Case

	Pre-Prod	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	Total
PRIMARY MINING EQUIPMENT																
Atlas Copco PV235	$ 14,811	$ 2,468	$ 9,874	$ 12,342	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 39,495
165mm Rotory Blast Hole Drill	$ 1,242	$ 1,242	$ 0	$ 1,242	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 3,725
28m^3 Hyd. Shovel (PC 5000)	$ 17,308	$ 8,654	$ 0	$ 8,654	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 34,616
19m^3 Front End Loader (994)	$ 0	$ 9,146	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 9,146
250t Haul Truck	$ 43,382	$ 4,338	$ 56,397	$ 34,706	$ 8,676	$ 8,676	$ 4,338	$ 4,338	$ 13,015	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 177,868
Total Primary Equipment	**$ 76,743**	**$ 25,848**	**$ 66,271**	**$ 56,994**	**$ 8,676**	**$ 8,676**	**$ 4,338**	**$ 4,338**	**$ 13,015**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 264,850**
SUPPORT EQUIPMENT																
630 Kw Dozer (D11)	$ 1,912	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 1,912
300 Kw Dozer (D9)	$ 967	$ 0	$ 967	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 967	$ 967	$ 0	$ 0	$ 0	$ 0	$ 3,867
7.3 m Motor Grader (24M)	$ 2,561	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 2,561
4.9 m Motor Grader (16H)	$ 997	$ 0	$ 997	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 997	$ 997	$ 0	$ 0	$ 0	$ 0	$ 3,988
Water Truck - 70,000 Liter	$ 4,217	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 4,217
RTD Dozer (834H)	$ 1,150	$ 0	$ 1,150	$ 0	$ 0	$ 0	$ 0	$ 0	$ 1,150	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 3,451
Rock Breaker - Impact Hammer (691 Kg m)	$ 43	$ 0	$ 0	$ 0	$ 0	$ 0	$ 43	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 86
Backhoe/Loader (1.5 cu m-446D)	$ 281	$ 0	$ 0	$ 0	$ 0	$ 0	$ 281	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 561
Pit Pumps (5299 lpm)	$ 55	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 55
36 ton Crane	$ 365	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 365
2 cm excavator (Cat 392)	$ 358	$ 0	$ 0	$ 0	$ 0	$ 0	$ 358	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 716
Low Boy	$ 994	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 994
Flatbed	$ 56	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 56
Manlift	$ -	$ 0	$ 0	$ 0	$ 21	$ 21	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 42
Total Support Equipment	**$ 13,954**	**$ 1,980**	**$ 0**	**$ 559**	**$ 21**	**$ 21**	**$ 681**	**$ 2,736**	**$ 1,150**	**$ 1,964**	**$ 1,964**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 22,868**
BLASTING																
Skid Loader	$ 57	$ 0	$ 0	$ 0	$ 57	$ 0	$ 0	$ 57	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 171
Total Blasting	**$ 57**	**$ 0**	**$ 0**	**$ 0**	**$ 57**	**$ 0**	**$ 0**	**$ 57**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 171**
MINE MAINTENANCE																
Lube/Fuel Truck	$ 602	$ 0	$ 0	$ 0	$ 0	$ 0	$ 301	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 903
Mechanics Truck	$ 187	$ 0	$ 0	$ 187	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 374
Tire Truck	$ 137	$ 0	$ 0	$ 137	$ 0	$ 0	$ 137	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 411
Total Mine Maintenance	**$ 926**	**$ 0**	**$ 0**	**$ 247**	**$ 0**	**$ 0**	**$ 438**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ 1,688**

	Pre-Prod	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	Total
OTHER MINE CAPITAL																
Light Plant	$ 66	$ 33	$ 33	$ 0	$ 0	$ 66	$ 33	$ 16	$ 16	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 263
Mobile Radios	$ 55	$ 7	$ 21	$ 38	$ 4	$ 3	$ 21	$ 7	$ 5	$ 2	$ 2	$ 0	$ 0	$ 0	$ 0	$ 165
Shop Equipment	$ 491	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 491
Engineering & Office Equipment	$ 200	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 200
Water Storage (Dust Suppression)	$ 98	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 98
Base Radio & GPS Stations	$ 105	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 105
Unspecified Miscellaneous Equipment	$ 150	$ 0	$ 0	$ 2,000	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 2,150
Access Roads - Haul Roads - Site Prep	$ 175	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 175
Light Vehicles	$ 726	$ 50	$ 50	$ 813	$ 50	$ 50	$ 603	$ 210	$ 50	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 2603
Total Other Mine Capital	**$ 2,066**	**$ 90**	**$ 104**	**$ 2,851**	**$ 54**	**$ 119**	**$ 657**	**$ 233**	**$ 72**	**$ 2**	**$ 2**	**$ 0**	$ 0	$ 0	$ 0	**$ 6,250**
CAPITALIZED MINE OPERATING COSTS																
Pre-Stripping Mining Cost	$ 19,489	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 19,489
Tailings Construction Costs	$ 1,631	$ 2,060	$ 6,612	$ 7,062	$4,293	$ 3,187	$ 4,703	$ 4,250	$ 10,776	$ 0	$ 11,357	$ 11,258	$ 1,240	$ 0	$ 0	$ 68,430
Reclamation (Occurs in Years 13 and 14)														$ 7,729	$ 22,843	$ 30,575
Total Capitalized Mining Costs	**$ 21,137**	**$ 2,060**	**$ 6,608**	**$ 7,063**	**$4,294**	**$ 3,188**	**$ 4,702**	**$ 4,249**	**$ 10,779**	**$ 0**	**$ 11,359**	**$ 11,257**	**$ 1,241**	**$ 7,730**	**$ 22,823**	**$ 118,494**
CAPITAL SUMMARY																
Primary Mining Equipment	$ 76,743	$ 25,848	$ 66,271	$ 56,944	$ 8,676	$ 8,676	$ 4,338	$ 4,338	$ 13,015	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 264,850
Support Equipment	$ 13,954	$ 0	$ 3,114	$ 0	$ 21	$ 21	$ 681	$ 0	$ 1,150	$ 1,964	$ 1,964	$ 0	$ 0	$ 0	$ 0	$ 22,868
Blasting	$ 57	$ 0	$ 0	$ 0	$ 57	$ 0	$ 0	$ 57	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 171
Mine Maintenance	$ 926	$ 0	$ 0	$ 324	$ 0	$ 0	$ 438	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 0	$ 1,688
Other Mine Capital	$ 2,066	$ 90	$ 104	$ 2,851	$ 54	$ 119	$ 657	$ 233	$ 72	$ 2	$ 2	$ 0	$ 0	$ 0	$ 0	$ 6,250
Capitalized Mine Operating Costs	$ 21,137	$ 2,060	$ 6,008	$ 7,063	$4,294	$ 3,188	$ 4,702	$ 4,249	$ 10,779	$ 0	$ 11,359	$ 11,257	$ 1,241	$ 7,730	$ 22,823	$ 118,494
Total - All Mining Capital	**$ 114,882**	**$ 27,999**	**$ 76,097**	**$ 67,181**	**$ 13,103**	**$ 12,005**	**$ 10,817**	**$ 8,878**	**$ 25,016**	**$ 1,966**	**$ 13,324**	**$ 11,257**	**$ 1,241**	$ 7,730	$ 22,823	**$ 414,322**

21.1.1.1 Drilling and Blasting

Primary drilling equipment capital is based on equipment quotations for a total of 16 Atlas Copco Pit Viper 235 blast-hole drills required through the life of mine. Eight of the drills will be purchased at the start of mining in Year -1, an additional three drills purchased in Year 1, and then four additional drills will be purchased in Year 3 at a cost of US$ 2,468,400 each (including shipping and assembly). The cost of the drills was provided by EMG LLC.

In addition to the production drills, smaller 45K pull-down drills will be used for pre-split drilling. These will use 165mm bits and will cost approximately US$ 1,241,800 each. One drill is purchased in Year -1, one more in year 1, and a replacement drill has been planned for Year 6.

Quotes for explosives trucks, powder magazines, and bulk ANFO storage has been obtained by TTP. These capital costs are included in the infrastructure costs. Additional, capital expense for a skid loader is provided to be used by the blasting crew for stemming holes. The skid loader would be purchased at an estimated cost of US$ 57,000 during Year -1 and then two additional units would be purchased in Year 4 and Year 7.

21.1.1.2 Loading

Capital costs for loading equipment have been quoted by EMG LLC and include four Komatsu PC5000 hydraulic shovels and two Caterpillar 994 Loaders. Two of the hydraulic shovels would be purchased during Year -1, with a third being purchased during Year 1. The fourth shovel is purchased in Year 3. The estimated cost for each shovel is US$ 8,653,900, which includes freight and assembly.

The cost of the 18-cubic meter loaders is based on a quote for a Caterpillar 994 loader, with the first one being purchased at the start of production in Year 1, and the second purchased in Year 2, at a cost of US$ 4,573,100 each.

21.1.1.3 Haulage

The 226-tonne haulage truck costs are based on CAT 793F trucks and were quoted by EMG LLC. Nine trucks are purchased during Year -1, with another 8 trucks purchased in Year 1. Trucks are purchased as they are required through the mine life. The trucks are staged in to allow ramp up of production through each year as they are needed to meet production requirements. The total number of trucks required by year is shown as follows:

Year	Number of Trucks Added	Trucks in Use
-1	8	8
1	3	11
2	11	22
3	5	27
4	6	33
5	2	35
6	2	37
7	1	38
8	3	41
9	0	41
10	0	41
11	0	22
12	0	6
13	0	6
14	0	6

Throughout the mine life, a total of 41 trucks are purchased. The number of operating trucks is reduced toward the end of the mine life as haulage requirements are decreased. The cost of each truck is estimated at US$ 4,338,200, including freight and assembly.

21.1.1.4 Mine Support

Capital estimates for mine support equipment include freight and erection. The initial support equipment to be purchased in Year -1 is as follows:

- One Caterpillar D11 track dozer (US$ 1,911,600 each quoted by EMG LLC);
- Four Caterpillar D9 track dozers (US$ 966,700 each quoted by EMG LLC);
- One Caterpillar 24M motor grader (US$ 2,560,600 quoted by EMG LLC);
- Four Caterpillar 16H motor graders (US$ 996,900 quoted by EMG LLC);
- Two Caterpillar 777 trucks with 70K liter water tanks (US$ 2,108,700 quoted by EMG LLC);
- Three Caterpillar 834H rubber tire dozers (US$ 1,150,300 quoted by EMG LLC);
- Two Caterpillar 392DL excavators (US$ 357,900 quoted by EMG LLC);
- One low-boy trailer complete with a used 60t haul truck to tow it (US$ 993,600);
- One flatbed truck (US$ 55,700);
- Two pit pumps (US$ 27,500 each);
- Two rock breakers to be attached to the 392DL excavator as needed (US$ 42,800); and
- 16 light plants (US$ 16,400).

Replacements are purchased for most units in Year 6.

21.1.1.5 Maintenance

Capital for mine maintenance equipment includes three fuel/lube trucks (US$ 301,000 each), two mechanic's truck (US$ 187,000 each), and three tire trucks (US$ 137,000 each). Note that requirements for mechanic's trucks are reduced through year 3 due to the assumption of MARC for maintenance. This single mechanic's truck is intended for support of a small number of owner-operated equipment. At year 3, an additional mechanics truck is put into service.

An additional US$ 491,000 has been included for shop equipment / tooling. Shop facilities were estimated by TTP and included in facility capital.

21.1.1.6 Mine Facilities

Mine facility capital has been estimated by TTP and is included in facility capital.

21.1.1.7 Light Vehicles

Initial capital for light vehicles is estimated to be US$ 540,000 while sustaining light vehicle capital is US$ 1,047,200. Initial and sustaining light vehicle capital is shown in **Table 21-4**.

Table 21-4: Estimated Mine Light Vehicle Capital (US$)

	Type	Initial Capital (US$)			Sustaining Capital (US$)		
		Quantity	Unit Cost	Ext. Cost	Quantity	Unit Cost	Ext. Cost
MINE DEPARTMENT							
Mine Superintendent	3/4 ton 4wd Pickup	1	$ 44,100	$44,100	2	$ 44,100	$88,200
Shift Foreman	4wd Pickup	2	$ 36,400	$72,800	9	$ 36,400	$327,600
Trainer	4wd Pickup	1	$ 32,200	$32,200	2	$ 32,200	$64,400
Blasting	4wd Pickup	1	$ 36,400	$36,400	2	$ 36,400	$72,800
Blasting	1 ton 4wd Pickup	1	$ 36,400	$36,400	2	$ 36,400	$72,800
Crew Vans	3/4 ton Passenger Van	2	$ 50,400	$100,800	11	$ 50,400	$554,400
ENGINEERING							
Chief Engineer	4wd Pickup	1	$ 36,400	$36,400	2	$ 36,400	$72,800
Short Range Planning	4wd Pickup	1	$ 32,200	$32,200	2	$ 32,200	$64,400
Survey	4wd Pickup	1	$ 36,400	$36,400	2	$ 36,400	$72,800
GEOLOGY							
Chief Geologist	4wd Pickup	1	$ 36,400	$36,400	2	$ 36,400	$72,800
Ore Control	4wd Pickup	1	$ 32,200	$32,200	2	$ 32,200	$64,400
Samplers	4wd Pickup	1	$ 32,200	$32,200	2	$ 32,200	$64,400
MINE MAINTENANCE							
Maintenance Superintendent	4wd Pickup	2	$ 36,400	$72,800	4	$ 36,400	$145,600
Mechanics / Labor	4wd Pickup	2	$ 32,200	$64,400	4	$ 32,200	$128,800
Total		**18**		**$665,700**	**48**		**$1,866,200**

21.1.1.8 Other Mine Capital

Other miscellaneous capital includes mobile radios for mobile equipment (US$ 1,000 per unit), engineering and office equipment ($ 200,000 US), water storage for dust suppression (US$ 97,900), GPS stations and surveying equipment (US$ 105,000), and other unspecified miscellaneous equipment (US$ 150,000). At the end of year three, Mt Todd personnel will take over the maintenance of equipment. Accordingly, US$ 2,000,000 as unspecified equipment has been added in year three for additional maintenance equipment.

21.1.2 CIP Process and Infrastructure (TTP)

Please note that this Section describes costs in Australian Dollars (AUD).

TTP's capital cost estimates (CCEs) are based on an Enhanced Factored Cost Estimate (EFCE) methodology, which features higher confidence levels around Contingency Provision and Management Reserve. The capital estimates are supported by the design work carried out throughout the study including process documentation, schematics, general arrangement drawings, 3D models and calculations. Note all currencies are in Australian Dollars. All capital estimated by TTP is summarized in **Table 21-5**.

Table 21-5: Estimated Capital Cost Summary (AUD000s)

Capital Cost	Initial Capital (AUD000s)
Facility 1000 – Geology	0.000
Facility 2000 – Mine Infrastructure	15,689
Facility 3000 – Process Plant	587,690
Facility 4000 – Project Services	16,654
Facility 5000 – Project Infrastructure	39,048
Facility 6000 – Permanent Accommodation	0,074
Facility 7000 – Site Establishment & Early Works	27,903
Facility 8000 – Management, Engineering, EPCM Services	94,353
Facility 9000 – Preproduction Costs	19,312
Subtotal	**617,316**
Direct	574,468
Indirect	125,731
Subtotal	**700,199**
Contingency Provision (14.6%)	100,544
TOTAL	**800,744**

The total capital cost, base cost plus contingency provision, represents the expected cost for the project, with approximately a 55% confidence level of completion within cost. This estimate has an accuracy range of approximately -0 to 15% based on the expected cost. At the upper limit of the accuracy range, there is an 85% confidence level of completion within cost.

Typically, the EPCM Project Manager would initially receive Owner's approval for expenditure up to the expected cost (i.e., this is the initial project budget).

However, funding arrangements would also need to be in place for expenditure up to the 85% confidence level. This additional funding is commonly referred to as Management Reserve. The selection of Management Reserve quantity will rest with Vista, and will be determined by Vista's attitude to risk.

21.1.2.1 Exclusions

The TTP scope of work is a significant part of the overall Project scope, although other parties have compiled capital costs for other areas on behalf of Vista. The potential impacts of possible price or labor rate fluctuations or currency exchange rate fluctuations are the role of a qualified actuary and should be covered by Vista in its standard business practices.

21.1.2.2 Capital Cost Estimating Methodology

A PFS design has been developed for a 50,000 tpd plant as the basis for an EFCE. The EFCE approach uses a combination of bottom-up calculations and factoring methods for each area in the estimate. The methods used to estimate capital in the CCE are summarized in the following sections.

The EFCE for the process plant features the methodology shown in the table below.

Table 21-6: CCE Methodology for Facility 3000 – Process Plant

Bulk Commodity	Base Case
Mechanical Equipment	A detailed mechanical equipment list, with supply and installation pricing based on budget quotations and internal body of knowledge
Concrete	MTOs based on 3D model and unit rates.
Structural Steel	MTOs based on 3D model and unit rates.
Platework	MTOs from previous projects a unit rates
Tankage	MTOs based on preliminary design calculations and unit rates
Piping	Percentage factor of the supplied mechanical equipment supply price, assessed on an area by area basis.
Electrical	Percentage factor based on total mechanical equipment supply price
Instrumentation and Control	Costs factored.

Subsequently estimate factors, by area, were back calculated for each bulk commodity as a percentage of the mechanical equipment supply cost estimate. In turn, the resultant estimate factors were critiqued against published data and industry experience.

21.1.2.3 Other Area Capital Cost Estimates

The EFCE for all areas outside of the process plant features the methodology shown in **Table 21-7**.

Table 21-7: Methodology for Other Areas of the Capital Cost Estimate

Area / Sub Area	Base Case
FACILITY 2000 – MINE	
Area 2300 – Mine Support Facilities	Drawings developed for the buildings and priced largely on a building per square meter basis
Area 2400 – Mine Support Services	Drawings developed for the buildings and priced largely on a building per square meter basis
FACILITY 4000 – PROJECT SERVICES	
Area 4100 – Water Supply	Sub-area 4110 – Water Supply WTP estimated by Tetra Tech Golden Office Sub-Area 4120 – Raw Water Distribution was estimated by mechanical equipment costs and factoring of bulk commodities
Area 4200 – Power Supply	Based on length of power distribution cables and trenching, and overhead power lines using rates developed from previous projects. Power – Generation (7.3.1) is by Power Engineers
Area 4300 – Communications	Based on MTOs for the fiber optic cables, phones and telemetry using budget pricing and rates developed from previous projects.
Area 4200 – Power Supply	Based on length of power distribution and overhead power lines and rates developed from previous projects
Area 4300 – Communications	Based on MTOs and provisional sums for the fiber optics, phones and telemetry
Area 4400 – Tailings Dam	Estimated by Tetra Tech Golden Office
Area 4500 – Waste Disposal	Based on provisional sums for sewerage services from previous project designs
Area 4600 – Plant Mobile Equipment	Vendor pricing of the proposed fleet for plant operation
FACILITY 5000 – PROJECT INFRASTRUCTURE	
Area 5100 – Site Preparation	Based on MTOs from preliminary drawings and rates developed from first principles
Area 5200 – Support Buildings	Drawings developed for the buildings and priced largely on a building per square meter basis
Area 5300 – Access Roads, Parking and Laydown	Provisional sums based on miscellaneous road and culvert repairs
Area 5400 – Heavy Lift Cranage	Based on the proposed fleet for plant construction and rates from previous project experience
Area 5600 – Bulk Transport	Based on the mechanical equipment cost for the weigh bridge and MTO for concrete
Area 5800 – Communications	Based on budget pricing and quantities provided by Vista
FACILITY 6000 – PERMANENT ACCOMMODATION	
Area 6100 – Personnel Transport	Based on unit rates for bus shelters with an allowance for the small amount of concrete required
FACILITY 7000 – SITE ESTABLISHMENT AND EARLY WORKS	
Area 7300 – Construction Camp	Based on MTOs for the access roads and site works and vendor quotes for the camp and operation

Area / Sub Area	Base Case
FACILITY 8000 – MANAGEMENT, ENGINEERING, EPCM SERVICES	
Area 8100 – EPCM Services	Factored up from 33,000 tpd Case based on the Total Direct Costs
Area 8200 – External Consultants/Testing	Provisional Sums based on previous project experience
Area 8300 – Commissioning	Process Plant commissioning costs based on 3% of the total mechanical equipment costs. Provisional Sums allowed for Mine, Project Services and Infrastructure commissioning.
Area 8400 – Owners Engineering/Management	Based on 2% of the project direct costs.
Area 8800 – License, Fees and Legal Costs	Based on 0.5% of the project direct costs.
Area 8900 – Project Insurances	Based on 0.5% of the project direct costs.
FACILITY 9000 – PRE-PRODUCTION COSTS	
Area 9100 – Preproduction Labor	Based on 0.25% of the project direct costs.
Area 9200 – Commissioning Expenses	Based on 0.5% of the project direct costs.
Area 9300 – Capital Spares	Based on 5% of the mine and process plant mechanical equipment costs.
Area 9400 – Stores and Inventories	Based on 1% of the mine and process plant mechanical equipment costs
Area 9800 – Contingency Provision	Priced based on a weighted average of the contingency of each facility
Area 9900 – Management Reserve Provision	A weighted average Management Reserve of 20% was allowed for, the selection of Management Reserve quantity will rest with Vista, and will be determined by Vista's attitude to risk.

21.1.2.4 Construction Labor Rates

A detailed calculation of composite, direct man-hour site rates has been carried out using TTP standard templates. The calculation is based upon current ordinary time wages for various classes of labor including direct supervision, to which the following factor may apply; site allowance, tool allowance, leave provisions, taxes and insurances, overtime, etc. This develops a gang rate that is combined with costs of incumbent support equipment (such as light vehicles, light mobile cranes, small tools, consumables, first-aid facilities and accommodation) and management support to arrive at an all-purpose site gang rate.

The construction labor rates developed for the CCE include the following construction contractors:

- Concrete;
- Structural, Mechanical and Piping (SMP); and
- Electrical and Instrumentation (E&I).

BASE LABOR RATES

The base labor rate includes the direct labor allocated for the installation of equipment and bulk commodities. Base pay rates were derived from award rates for similarly sized projects currently underway in the North West of Western Australia and in the Northern Territory. These are considered to be the benchmark for the area, including Mt Todd. Allowances were made for overtime loadings above a 36-hour week including time and a half for the initial 12 hours overtime, followed by double time for the final 17 hours overtime, to provide for a 65-hour working week. The rates were averaged over a standard mix of trades, to produce a composite rate per man per hour.

The base labor rates were developed to include items listed below:

- All direct payments including the site allowances and special project allowances for straight time and overtime worked for personnel;
- Overtime at penalty rates;
- Provision for holiday leave and loadings thereon;
- Provision for sick leave;
- Provision for cost of travel time to site and return travel on job completion;
- Provision for additional manpower turnover, bereavement leave and miscellaneous paid non-work days;
- Payroll tax;
- Workers compensation insurance;
- Superannuation considerations; and
- Industry redundancy payments.

A rest and recreation (R&R) loading was also added to the composite rate, together with a 15% contractors allowance for overheads and margin to produce the base labor rate for each contractor type.

CONTRACTOR INDIRECT RATES

The contractor indirect rate is a combination of costs associated with indirect contractor personnel, contractor vehicles, contractor overheads and construction plant equipment. An estimate of construction contract duration and installation hours was based on the EPCM schedule and bulk quantity development.

The contractor indirect rates were developed to include the items listed below:

- Project Management personnel;
- Construction Supervision personnel;
- Site Quality Assurance and Control personnel;
- Site Health, Safety, Environmental personnel;
- Other indirect labor (stores officer, surveyor, etc.);
- Contractor vehicles for the Project Management team;
- Office accommodation;
- Workshop and stores facilities;
- Staff travel including airfares ;
- Office overheads; and
- Vehicle consumables.

Provisions for the accommodation and messing are also not included in the indirect contractor rates. This is allowed for in the construction camp cost estimate to supply and operate the camp.

Although they are considered indirect costs, construction plant equipment rates are estimated separately to include the following:

- Construction plant equipment mobilization / demobilization;
- Construction plant management support; and
- Construction plant and equipment.

The provision for task specific heavy lift cranes >50 tonnes were not included in the indirect contractor rates build-up; instead it was allowed for in a separable line item in the chart of accounts (CoA).

CONSTRUCTION GANG RATES

The overall site construction gang rates were developed by summing the base labor rate, contractor indirect rates and construction plant rates to provide an overall site construction gang rate for Concrete, SMP and E&I contractors as shown in **Table 21-8** below.

Table 21-8: Estimated Construction Gang Rate Development (AUD)

Contractor	Base Labor Rate (AUD/hr)	Contractor Indirect Rate (AUD/hr)	Construction Plant Rate (AUD/hr)	Construction Gang Rate (AUD/hr)
Concrete	$ 89.69	$ 23.98	$ 24.36	$ 138.03
SMP	$ 98.75	$ 45.31	$ 20.84	$ 164.90
E&I	$ 104.03	$ 56.27	$ 24.07	$ 184.36

21.1.2.5 Mechanical Equipment

The supply costs comprise the direct mechanical equipment cost plus the cost for freight to site. Installation costs are estimated based on an evaluation of installation hours multiplied by the SMP contractor gang rate. These estimating methods are discussed in the following sections.

EQUIPMENT COSTS

The basis for estimating the mechanical equipment supply costs was largely based on budgetary pricing from vendors. The vendors were provided with preliminary specifications and/or data sheets for major equipment items. The budget quotations received from vendors are expected to have an accuracy equal to ±10%.

All other minor equipment items were priced from a TTP's database of costs from recent similar sized projects. The basis of the supply cost estimate for each mechanical equipment line item is documented in the process plant CCE.

FREIGHT COSTS

Several methods were used to determine and validate the allowance for delivery costs of mechanical equipment to site. These methods included:

- Quotes provided by the manufacturer;
- Estimates based on the weight and volume of the load;
- Estimates based on published and in-house guides for similar installations; and
- Estimates based on a validated percentage of the mechanical equipment cost (determined to be 9% of the supply price).

INSTALLATION HOURS

Several methods were used to determine and validate the installation hour allowance for mechanical equipment. These methods included:

- Quotes provided by the manufacturer;
- Estimates based on the weight of the equipment; and
- Estimates based on published and in-house guides for similar installations.

The installation hour estimates for large process equipment (>3000 man hours/ equipment) including the crushers, HPGRs, ball mills, VXP mills and thickeners were reviewed in detail against historical records and published guidelines.

21.1.2.6 Quantity Development and Unit Rates

The basis for the development of supply and installation costs of bulk commodities is discussed in the following section. Bulk commodities include civil, concrete, structural steel, etc. which will be used in the construction of the process plant. These costs were largely derived based on an estimate of material take-off (MTO) quantities which were multiplied by a unit rate for each type of material. The unit rates were calculated using TTP standard methods including obtaining current market rates from contractors, historical data and reference books (Rawlinsons) and comprise of allowances for supply of the raw material, fabrication, freight and erection.

CIVIL

Preliminary bulk earthwork quantities were estimated using civil 3D modelling software 12D Model. The 12D Model accurately calculates earthworks volumes utilizing the existing topography and proposed design levels. Structural excavation and backfill required for concrete structures are included in the concrete quantities. Trenching requirements for underground utilities distribution were determined from service plans. Stormwater drainage quantities were determined from the civil site plan with vee-drains alongside plant roads directing surface run-off beneath roads via corrugated steel culverts. All quantities were categorized by standard type of work classification.

Unit rates for this work classification were developed from the in-house rates database. This rates database is constantly maintained so as to be current and has proven to be sufficiently accurate over several recent projects. The availability of water and local earthworks materials was taken into account in to the development of unit rates.

CONCRETE

Concrete quantities for foundations and ground slabs for all equipment and structures in the process plant were calculated by 3D models. Concrete quantities were categorized by standard classes of concrete including spread/pad footings, strip footings, raft footings, ring beams, ground slabs, walls, sumps and pits, etc.

Unit pricing was obtained from industry sources by standard classification, each having an assessment of formwork, props, bracing reinforcing, embedment's, joints in slabs plus a miscellaneous allowance for curing, formwork hardware and other sundries. Concrete supply was costed at a rate deemed to include plant control testing, some admixtures and out of hours pouring. A wastage factor was included in the rates. A Contractor's mark-up was also applied to all materials. Direct labor unit man-hours were sought from industry sources and checked against historical data and various published references.

STRUCTURAL STEEL

Steel quantities were categorized by standard classes of steel including light, medium, heavy and very heavy. There are also provisions made for grating, handrailing and stair treads.

Unit rates for the supply and installation of structural steel were calculated using TTP Standard templates. Supply of steel was based on rates quoted from Thai fabricators from a similar project. The supply rate includes provisions for steel supply, shop drawings, shop fabrication, painting and freight to site. Estimates for the installation costs of structural steel are based on estimates of erection hours and the SMP gang rate.

PLATEWORK

Quantities of steel required for custom designed platework was calculated using the 3D models. The cost items for platework includes plate thicknesses of <10mm, 12-20mm and floor plate of 6mm and also allowances for Bisalloy or rubber lining where applicable.

Unit rates for platework were provided by Thai fabricators as is described for structural steel.

TANKAGE

Quantities of steel required for custom designed tankage were quantified by structural engineers using TTP standard spread sheets to determine the required tank shell and base thickness in accordance with the provisions of API 650. This includes allowances for the mass of steel for shell plates, top rings, and base plates.

Unit rates for supply and installation of tankage were based on Thai fabricated steel using the same methodology as is described for structural steel. There are two classes of tankage allowed for in the CCE including shop fabricated and site erected tanks (assumed to be greater than 7m in diameter).

PIPING

The estimate for the supply and installation of process piping was factored based on a percentage of the supplied mechanical equipment price, assessed on an area by area basis. These percentages were based on in-house and industry typical piping allowances for similar gold plants. These factors were validated with values reported in published guidelines.

ELECTRICAL

The estimate for the supply and installation of electrical components for the process plant was factored based on a percentage of the total Mechanical Equipment supply and installation cost. These percentages were based on in-house and industry typical electrical allowances for similar gold plants.

INSTRUMENTAL AND CONTROL

The estimate for the supply and installation of the Instrumentation and Process Control System for the Alternate Case process plant was estimated based on preliminary P&IDs and equipment lists based on a highly automated gold plant with all field instruments marshalled to remote Input / Output (I/O) cabinets. The estimate of Instrumentation and Control costs for the Base Case was up-scaled based on an expected 30% increase in equipment, I/O, programming and instrumentation.

21.1.2.7 Indirect Costs

CONSTRUCTION CAMP

An estimate of the construction facilities was developed from previous project experience for the various scopes of work. This includes a breakdown of costs for contractor preliminaries, transportable building (supply and install) and establishing the infrastructure, power supply, communications and water supply. It also includes allowances for removal of the infrastructure following completion. Provisions were made for the operation of the camp based on a man-day rate. The man-day rate applied is based on budget enquiries for appropriate contractors.

EPCM SERVICES

Engineering, drafting and documentation functions are task and deliverable related. Hence, their estimates were based on task and deliverable identification with time estimates based in industry experience. Procurement activities were estimated from hours related to purchasing, expediting, inspection and transport functions derived by time involvements, and then checked against industry experience. Management, administrative and project engineering functions are mostly time-related and were assessed by title, rate and man-months of key personnel and other staff proposed.

To the extent possible, site office items were detailed and estimated on an item-by-item basis. Management, supervisory and administrative staffing were estimated on an hours basis.

EXTERNAL CONSULTANTS AND TESTING

Cost allowances for Environmental, Human Resources and Industrial Relations, and Health and Safety consultants are based on industry experience of required manning and market contract values.

OTHER INDIRECT COSTS

The following costs were calculated based on industry validated percentages of the total direct costs of the Project:

- Owners engineering / management;
- License, fees and legal costs;
- Project insurances; and
- Pre-production labor.

The following costs were calculated based on industry validated percentages of the mechanical equipment supply cost for the Project:

- Commissioning Expenses;
- Capital Spares; and
- Stores and Inventories.

21.1.2.8 Contingency Provision

The contingency provision is an allowance added to an estimate to provide for costs which cannot be estimated due to inadequate information, but which are known to be implicit in the scope.

The contingency provision represents costs which are expected to be incurred to complete the project and must be regarded as part of the total funds placed under the direct control of the project manager.

The contingency provision includes an allowance for:

- Unidentified items not included in the quantity calculations or equipment lists, due to lack of knowledge, but implicit in the scope.
- Small changes, arising from detailed design, which normally occur during the course of the project, as knowledge becomes firmer.
- Design Omissions.

Changes in concept, scope or production rates which depart from those on which the estimate has been based require a new estimate. These changes are not allowed for in the contingency.

21.1.3 Power Plant (POWER Engineers)

Please note that this Section describes costs in Australian Dollars (AUD).

Estimated capital costs compiled for this study cover direct and indirect costs of power station construction including equipment quoted from suppliers, material quantities estimated from the preliminary design, and installation labor and supervision with local material and labor costs applied to those estimates. Power plant estimated capital costs are shown in **Table 21-9** in Australian dollars.

Table 21-9: Estimated Power Station Installed Capital Cost Summary (AUD)

	69.6 MW (Seven Reciprocating Engines) (AUD)
Reciprocating Engines	$32,705,556
BOP Equipment	$10,448,551
Mechanical, Civil, & Electrical Direct Costs	$48,650,267
Engineering Fees	$6,944,444
Contractor’s Fees	$6,002,083
Taxes, and Other Indirect Costs	$10,938,749
Total Installed Costs AUD	**$115,689,650**
Capital Cost per Net Installed Capacity AUD/kW	$1,624

21.1.4 Mine Dewatering (Tetra Tech)

Mine dewatering capital costs are based on direct vendor quotes or Tetra Tech in-house estimates and include 5% indirect costs.

Table 21-10: Estimated Mine Dewatering Capital Cost Summary (US$000s)

WBS No.	Description	Initial Capital (US$000s)	Sustaining Capital (US$000s)	Total Capital (US$000s)
2501	PPD Dewatering	$532	$0	**532**
2502	Self-Priming Pump on Pontoon	$0	$0	**0**
2503	Pump	$50	$299	**349**
2504	Piping in Pit	$74	$114	**188**
2505	Piping from Pit to PWP	$287	$0	**287**
2506	Electrical	$27	$41	**68**
2507	Indirects	$16	$19	**35**
	2500 Mine Dewatering/Drainage	$987	$473	**1,459**

21.1.5 Reclamation and Closure (Tetra Tech)

Costs for reclaiming major facilities at the Project were estimated using closure material quantities based on ultimate designs and following the closure plans discussed above. Capital costs for reclamation are estimated at US$138 million for LoM.

Table 21-11: Estimated Reclamation Capital Cost Summary (US$000s)

WBS No.	Description	Initial Capital (US$000s)	Sustaining Capital (US$000s)	Total Capital (US$000s)
2901	Heap Leach Pad	$0	$1,009	**1,009**
2902	Low Grade Ore Stockpile	$103	$645	**747**
2903	TSF 1	$0	$28,875	**28,875**
2904	TSF 2	$0	$32,991	**32,991**
2905	WRD (GCL Cover)	$0	$36,163	**36,163**
2906	Process Plant Area	$0	$8,555	**8,555**
2907	Soil Stockpiles	$0	$226	**226**
2908	Mine Roads	$0	$397	**397**
2909	Batman Pit	$0	$1,131	**1,131**
2910	Passive Treatment Systems	$0	$3,722	**3,722**
2911	Indirect Costs	$21	$24,393	**24,415**
	2900 Mine Closure	$124	$138,108	**138,232**

21.1.6 Water Treatment Plant (Tetra Tech)

Water treatment plant capital costs are based on direct vendor quotes or Tetra Tech in-house estimates, initial capital costs are estimated at US$14.7 million and no sustaining capital improvements are expected.

Table 21-12: Estimated Water Treatment Plant Capital Cost Summary (US$000s)

WBS No.	Description	Initial Capital (US$000s)	Sustaining Capital (US$000s)	Total Capital (US$000s)
4111	Earthwork	$493	$0	**$493**
4112	Concrete	$227	$0	**$227**
4113	Building	$1,411	$0	**$1,411**
4114	Equipment	$7,111	$0	**$7,111**
4115	Mechanical	$1,472	$0	**$1,472**
4116	Electrical and Instrumentation	$2,016	$0	**$2,016**
4117	Engineering Procurement	$192	$0	**$192**
4118	Construction Management	$403	$0	**$403**
	4110 Water Treatment Plant	**$14,746**	**$0**	**$14,746**

21.1.7 Raw Water Dam (Tetra Tech)

Raw water dam capital costs are based on direct vendor quotes or Tetra Tech in-house estimates, initial capital costs are estimated at US$2.5 million and no sustaining capital improvements are expected.

Table 21-13: Estimated Raw Water Dam Capital Cost Summary (US$000s)

WBS No.	Description	Initial Capital (US$000s)	Sustaining Capital (US$000s)	Total Capital (US$000s)
4121	Site & Foundation Prep	$682	$0	**$682**
4122	Embankment Construction	$886	$0	**$886**
4123	Relocate Outlet Downstream	$10	$0	**$10**
4124	Spillway Construction	$271	$0	**$271**
4125	Other Construction Costs	$296	$0	**$296**
4126	Pump Operation Cost	$0	$0	**$0**
4127	Engineering Procurement	$225	$0	**$225**
4128	Construction Management	$92	$0	**$92**
4129	Temporary Construction Facilities	$47	$0	**$47**
	4120 Raw Water Dam	**$2,509**	**$0**	**$2,509**

21.1.8 Tailings Storage Facilities (Tetra Tech)

Tailings storage facility capital costs are based on direct vendor quotes or Tetra Tech in-house estimates. Initial capital costs are estimated at US$6.6 million with sustaining capital of US$85 million.

Table 21-14: Estimated Tailings Storage Facility Capital Cost Summary (US$000s)

WBS No.	Description	Initial Capital (US$000s)	Sustaining Capital (US$000s)	Total Capital (US$000s)
4410	TSF 1			
1	Site & Foundation Preparation	$491	$1,894	**$2,386**
2	Embankment Construction	$226	$4,509	**$4,735**
3	Downstream Embankment Toe Drain	$913	$0	**$913**
4	Tailings Delivery & Return Pipelines	$1,917	$186	**$2,103**
5	Return Water Ponds	$161	$0	**$161**
6	Diversion Channels	$0	$0	**$0**
7	Equipment Purchase	$1,312	$0	**$1,312**
8	Mobilization	$761	$898	**$1,660**
9	EPCM	$837	$988	**$1,826**
10	Instrumentation	$0	$175	**$175**
	4410 TSF 1	**$6,618**	**$8,651**	**$15,268**
4420	TSF 2			
1	Site & Foundation Preparation	$0	$12,754	**$12,754**
2	Underdrain Construction	$0	$1,218	**$1,218**
3	Downstream Toe Drain	$0	$588	**$588**
4	Embankment Construction	$0	$7,039	**$7,039**
5	Impoundment Liner	$0	$21,449	**$21,449**
6	Overdrain & Reclaim Sump/Pond Construction	$0	$1,689	**$1,689**
7	Tailings Delivery & Return Pipelines	$0	$6,200	**$6,200**
8	Surface Water Management	$0	$1,245	**$1,245**
9	Equipment Purchase	$0	$1,312	**$1,312**
10	Mobilization	$0	$7,371	**$7,371**
11	EPCM	$0	$8,108	**$8,108**
12	Instrumentation	$0	$280	**$280**
	4420 TSF 2	$0	$69,252	**$69,252**
	4400 Tailings Dam	**$6,618**	**$77,903**	**$84,521**

21.2 Operating Costs

LoM operating costs requirements are estimated to be US$15.18/t-milled as summarized in **Table 21-15**.

Table 21-15: Estimated LoM Operating Costs (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.10	0.03
Mine Maintenance	0.11	0.03
Engineering	0.05	0.01
Geology	0.03	0.01
Drilling	0.77	0.23
Blasting	1.17	0.35
Loading	0.60	0.18
Hauling	2.74	0.83
Mine Support	0.43	0.13
Mine Dewatering	0.01	0.004
Open Pit Mine	**6.02**	**1.82**
CIP PROCESS PLANT		
Labor	0.79	-
3100-Crush/Screen/Stockpile	0.18	-
3200-Reclaim & HPGR	0.44	-
3300-Classification & Grinding	3.14	-
3400-Pre-Leach,Thick/Aeration/CIP	0.13	-
3500-Desorption, Gold Room	0.02	-
3600-Detox & Tailings Pumping	0.06	-
3700-Reagents	2.98	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.01	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**7.88**	**-**
Project Services	$0.16	-
G&A	$1.11	-
Operating Costs	**$15.18**	**-**

21.2.1 Mining (MDA)

Annual mine operating costs have been estimated based on personnel requirements and equipment hourly costs. **Table 21-16** summarizes annual mine operating costs before the allocation of capitalized mining (capitalized mining is included in **Section 21.1.1 – Mining (MDA)**). Costs are provided based on functionality (drilling, blasting, loading, hauling, support, mine general services, mine maintenance, engineering, and geology).

The following subsections describe the operating cost estimate by functionality. Note that these costs are described before the allocation of costs to capital for pre-stripping, mining of waste for construction purposes, and mining and waste re-handle of material to be used for reclamation. The total average mining cost (open pit to primary crusher only) for the Base Case is estimated to be US$ 1.97/t mined (based on a net operating cost of US$ 1,436 million and 731 million tonnes). Operating costs shown in the economic model reflect operating costs after capitalization.

21.2.1.1 Drilling Costs

The average life-of-mine drilling cost is estimated to be US$ 0.24/t mined after allocation of drilling costs for pre-stripping and tailings construction. This includes maintenance allocations based on MARC cost assumptions.

21.2.1.2 Blasting Costs

The average life-of-mine blasting cost is estimated to be US$ 0.36/t mined.

21.2.1.3 Loading Costs

The average life-of-mine loading cost is estimated to be US$ 0.20/t mined. The cost per tonne moved includes the re-handle of ore and waste from stockpiles at the end of the mine life. Maintenance costs assume the use of MARC costs provided by EMG LLC.

21.2.1.4 Haulage Costs

The average life-of-mine haulage cost is estimated to be US$ 0.92/t mined. The cost per tonne moved includes re-handling of stockpiled ore and waste at the end of the mine life. Maintenance costs assume the use of MARC costs provided by EMG LLC.

21.2.1.5 Mine Support Costs

Mine-support costs include the operation of all of the mine-support equipment. The average life-of-mine support cost is estimated to be US$ 0.15/t mined. The cost per tonne moved includes support during re-handling of stockpiled ore and waste at the end of the mine life. Maintenance costs assume the use of MARC costs provided by EMG LLC.

Support costs also include the costs to reinforce the eastern high wall in the ultimate pit with bolts and mesh as recommended by Call & Nicholas.

21.2.1.6 Mine Maintenance Costs

Most maintenance will be done under a MARC cost structure for pre-production and the first two years of production. Beyond this it was assumed that Vista would take over all maintenance tasks. The vendor with the contract will be expected to supply mechanics and maintenance parts for major equipment repair. Costs associated with the contract have been included in the equipment hourly cost. Prior to the beginning of Year 3, the contractor will provide MARC services, Vista will employ one maintenance planner.

After the beginning of Year 3, the MARC costs for the parts and labor were still used for maintenance cost estimates of mining equipment, but the anticipated overhead and profit of the contractor would be removed. For this reason, during Year 3 and beyond, the MARC costs were multiplied by 85%. MDA has assumed that this will require hiring of a maintenance foreman and an additional maintenance planner.

Owner mine-maintenance costs have been included to cover items not covered by the MARC costs, as well as supervision. This includes salaries for a Maintenance Superintendent and Maintenance Planner to track costs associated with the contract. Tiremen will be hired by the owner to maintain all equipment tires, and servicemen will be hired to keep equipment fueled and lubricated. An allocation for shop laborers has been included for light maintenance of facilities.

The average life-of-mine mine-maintenance cost is estimated to be US$ 0.04/t mined. This does not include the specific parts and labor allocations to individual equipment, as those costs are allocated to the equipment and the cost center for which the equipment is used.

21.2.1.7 Mine General Services, Engineering and Geology Costs

Mine General costs include salaries for a Mine Manager, Mine Clerk, Shift Foremen, and trainers. Mine general costs also include an allocation for various supplies and office costs.

Engineering and geology services are provided to maintain surveying, mine planning, and ore control for the operations. The average life-of-mine general services, Engineering, and Geology costs are estimated to be US$ 0.06/t mined.

Table 21-16: Estimated Annual Mine Operating Costs (US$)

	Units	Pre-Prod	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	Total
MINED TONNES																	
Ore to Mill	k tonnes	-	8,836	10,330	17,795	9,232	17,750	8,749	7,178	13,482	17,750	17,750	17,799	127	-	-	146,779
Ore to Stkpl	k tonnes	2,859	7,302	5,283	6,699	6,354	12,102	235	-	-	1,000	10,903	8,172	-	-	-	60,908
Total Ore Mined	k tonnes	2,859	16,138	15,613	24,495	15,586	29,852	8,984	7,178	13,482	18,750	28,653	25,970	127	-	-	207,687
Re-handle Ore	k tonnes	-	3,625	7,420	3	8,518	-	9,001	10,620	1,647	-	-	-	17,623	15,805	-	74,262
Re-handle Waste	k tonnes	-	1,555	603	3,848	435	877	-	-	-	-	4,842	6,253	1,171	1,480	7,256	28,324
Re-handle Sorter Rejects	k tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	20,769	20,769
Waste to Dumps	k tonnes	8,802	10,498	47,536	32,880	76,531	58,085	87,011	68,218	56,598	42,935	29,747	4,148	-	-	-	522,990
Total Tonnes Mined	k tonnes	11,661	26,636	63,149	57,375	92,117	87,937	95,995	75,396	70,080	61,685	58,400	30,119	127	-	-	730,677
Total Tonnes Moved	k tonnes	11,661	31,816	71,172	61,226	101,070	88,814	104,996	86,016	71,727	61,685	63,242	36,372	18,921	17,285	28,025	854,032
Strip Ratio	w:o	3.08	0.65	3.04	1.34	4.91	1.95	9.69	9.50	4.20	2.29	1.04	0.16	-			2.52
Mined Waste to Construction	k tonnes	675	150	6,381	1,956	3,741	1,820	4,275	3,218	6,592	-	1,496	24	-	-	-	30,328
Mined Material for Pre-Production	k tonnes	122	-	-	-	-	-	-	-	-	-	-	-	-	-	-	122
Mined Waste for Closure	k tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	1,480	7,256	8,740
Net Tonnage Mined	**k tonnes**	**10,864**	**28,041**	**57,371**	**59,266**	**88,811**	**86,994**	**91,720**	**72,178**	**63,488**	**61,685**	**61,746**	**36,348**	**1,298**	**-**	**-**	**719,811**
MINING COSTS																	
Mine General Service	K USD	$ 689	$ 1,611	$ 1,618	$ 1,705	$ 1,966	$ 1,966	$ 1,966	$ 1,971	$ 1,966	$ 1,966	$ 1,966	$ 1,615	$ 1,257	$ 1,225	$ 919	$ 24,402
Mine Maintenance	K USD	$ 796	$ 1,592	$ 1,592	$ 2,138	$ 2,198	$ 2,198	$ 2,198	$ 2,204	$ 2,198	$ 2,198	$ 2,198	$ 2,204	$ 2,198	$ 1,380	$ 1,093	$ 28,381
Engineering	K USD	$ 348	$ 912	$ 912	$ 915	$ 912	$ 912	$ 912	$ 915	$ 912	$ 912	$ 912	$ 915	$ 502	$ 678	$ 376	$ 11,946
Geology	K USD	$ 316	$ 628	$ 628	$ 629	$ 628	$ 628	$ 628	$ 629	$ 628	$ 628	$ 628	$ 629	$ 628	$ 195	$ 38	$ 8,087
Drilling	K USD	$ 2,805	$ 8,094	$ 15,344	$ 14,784	$ 19,926	$ 21,416	$ 19,657	$ 15,465	$ 15,475	$ 14,793	$ 15,814	$ 9,858	$ 139	$ 0	$ 0	$ 173,570
Blasting	K USD	$ 4,294	$ 10,852	$ 22,473	$ 21,674	$ 31,737	$ 32,085	$ 32,198	$ 25,395	$ 24,437	$ 22,344	$ 22,451	$ 13,119	$ 460	$ 0	$ 0	$ 263,518
Loading	K USD	$ 2,127	$ 6,058	$ 13,229	$ 10,612	$ 17,288	$ 15,051	$ 18,054	$ 15,206	$ 12,187	$ 10,624	$ 10,968	$ 6,585	$ 3,383	$ 3,140	$ 4,882	$ 149,394
Hauling	K USD	$ 5,183	$ 17,928	$ 41,984	$ 40,377	$ 63,515	$ 65,485	$ 69,394	$ 71,253	$ 77,754	$ 75,383	$ 77,008	$ 41,148	$ 6,400	$ 6,133	$ 10,623	$ 669,572
Mine Support	K USD	$ 2,642	$ 5,062	$ 6,756	$ 7,044	$ 7,264	$ 8,046	$ 11,867	$ 11,113	$ 10,951	$ 10,357	$ 9,333	$ 6,204	$ 3,846	$ 3,811	$ 2,836	$ 107,131
Total Mine Cost	**K USD**	**$ 19,201**	**$ 52,736**	**$ 104,540**	**$ 99,889**	**$ 145,431**	**$ 147,816**	**$ 156,852**	**$ 144,128**	**$ 146,535**	**$ 139,241**	**$ 141,292**	**$ 82,270**	**$ 18,826**	**$ 16,576**	**$ 20,748**	**$ 1,436,096**
MINE COST PER TONNE MINED																	
Mine General Service	$ /t	$ 0.06	$ 0.06	$ 0.03	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 9.88	$ 0	$ 0	$ 0.03
Mine Maintenance	$ /t	$ 0.07	$ 0.06	$ 0.03	$ 0.04	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.04	$ 0.04	$ 0.07	$ 17.29	$ 0	$ 0	$ 0.04
Engineering	$ /t	$ 0.03	$ 0.03	$ 0.01	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.03	$ 3.95	$ 0	$ 0	$ 0.02
Geology	$ /t	$ 0.03	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 4.94	$ 0	$ 0	$ 0.01
Drilling	$ /t	$ 0.24	$ 0.30	$ 0.24	$ 0.26	$ 0.22	$ 0.24	$ 0.20	$ 0.21	$ 0.22	$ 0.24	$ 0.27	$ 0.33	$ 1.09	$ 0	$ 0	$ 0.24
Blasting	$ /t	$ 0.37	$ 0.41	$ 0.36	$ 0.38	$ 0.34	$ 0.36	$ 0.34	$ 0.34	$ 0.35	$ 0.36	$ 0.38	$ 0.44	$ 3.62	$ 0	$ 0	$ 0.36
Loading	$ /t	$ 0.18	$ 0.23	$ 0.21	$ 0.18	$ 0.19	$ 0.17	$ 0.19	$ 0.20	$ 0.17	$ 0.17	$ 0.19	$ 0.22	$ 26.61	$ 0	$ 0	$ 0.20
Hauling	$ /t	$ 0.44	$ 0.67	$ 0.66	$ 0.70	$ 0.69	$ 0.74	$ 0.72	$ 0.95	$ 1.11	$ 1.22	$ 1.32	$ 1.37	$ 50.34	$ 0	$ 0	$ 0.92
Mine Support	$ /t	$ 0.23	$ 0.19	$ 0.11	$ 0.12	$ 0.08	$ 0.09	$ 0.12	$ 0.15	$ 0.16	$ 0.17	$ 0.16	$ 0.21	$ 30.25	$ 0	$ 0	$ 0.15
Total Mine Cost	**$ /t**	**$ 1.65**	**$ 1.98**	**$ 1.66**	**$ 1.74**	**$ 1.58**	**$ 1.68**	**$ 1.63**	**$ 1.91**	**$ 2.09**	**$ 2.26**	**$ 2.42**	**$ 2.73**	**$ 147.97**	**$ 0**	**$ 0**	**$ 1.97**

21.2.2 Mine Dewatering (Tetra Tech)

Operating costs are related to power consumption of pumps; labor is excluded, as supervision of the dewatering system is planned for existing mine or environmental staff and will not require dedicated personnel. Mine dewatering estimated operating costs average US$0.012/t-milled.

21.2.3 CIP Process and G&A (TTP)

Please note that this Section describes costs in Australian Dollars (AUD).

Overall the approach taken for the PFS operating cost estimate establishment was to perform the estimates at a FS level of detail, leading to a higher than usual level of detail presented for the Technical Report. This approach was deliberately adopted to minimize rework during the FS stage, with additional information expected to be limited to the use of improved accuracy quotes for the FS cost estimate.

Final plant operating cost estimates issued for the Technical Report were AUD217 million per year, giving a cost of AUD 12.24/t treated as shown in **Table 21-17.**

21.2.3.1 Cost Distribution

The distribution of operating costs was not unexpected for large scale gold operations, with the five main operating cost expenditures in descending order being:

- Reagents and Consumables;
- Power;
- Labor;
- Maintenance; and
- G&A.

Items of expenditure higher than normally expected for gold mining operations related specifically to:

- Ore hardness, and included consumables (mill media) and power consumption; and
- High volume / low grade ore treatment schedule and related predominantly to reagents.

Table 21-17: Estimated Plant Operating Costs (@ Steady State) (AUD)

Cost Center	Operating Cost			
	AUD/ a	AUD/ t	AUD/oz	%
LABOR				
Total	**28,640,000**	**1.61**	**63.83**	**13.2%**
TRANSPORT & ACCOMMODATION				
Total	**1,810,000**	**0.10**	**4.03**	**0.8%**
POWER				
Processing Plant	48,170,000	2.71		
Miscellaneous	570,000	0.03		
Total	**48,740,000**	**2.75**	**108.63**	**22.4%**

Cost Center		Operating Cost			
		AUD/ a	AUD/ t	AUD/oz	%
FUEL					
Vehicles		420,000	0.02		
Plant Gas		710,000	0.04		
Total		**1,130,000**	**0.06**	**2.52**	**0.5%**
MAINTENANCE					
Fixed Plant		11,370,000	0.64		
Mobile Equipment		150,000	0.01		
Total		**11,520,000**	**0.65**	**25.67**	**5.3%**
REAGENTS & CONSUMABLES					
Reagent		75,810,000	4.27		
Consumables		45,860,000	2.58		
Total		**121,670,000**	**6.85**	**271.16**	**56.0%**
EQUIPMENT HIRE					
Total		**0**	**0.00**	**0.00**	**0.0%**
PRODUCT TRANSPORT					
Total		**0**	**0.00**	**0.00**	**0.0%**
CONTRACT – GENERAL EXPENSES					
GENERAL CONSUMABLES		260,000	0.01		
CONTRACT EXPENSES		910,000	0.05		
GENERAL EXPENSES		2,530,000	0.14		
MINING CONTRACT		0	0.00		
Total		**3,700,000**	**0.21**	**8.25**	**1.7%**
TOTAL	**AUD**	**217,210,000**	**12.24**	**484.09**	**100%**

21.2.3.2 Labor

Estimated labor costs were developed by a build-up of base labor rates, on-costs and required work force numbers.

Workforce numbers were developed using a bottom-up approach by assessing requirements in each area, and in consultation with Vista personnel, adjusting for areas specific to Mt Todd requirements.

Labor rates were initially taken as the TTP standard rates (actual operating mine data from 2010), but were subsequently adjusted up by 7% in consultation with Vista. A review was conducted by recruitment consultant Michael Page, which indicated labor rates for 4 out of the 154 categories presented required an upwards adjustment.

Labor rates have since been revised, based on recently completed projects (2017 & 2018/19) and industry consultation.

The whole site labor force was presented in the TTP operating cost analysis to ensure that there was some consistency in labor rates across the board, however mining and mining related labor costs were not included in the TTP operating cost estimate as these costs were ultimately in the domain of the mining consultant MDA's operating cost schedule.

Final process plant and general and administrative (G&A) labor cost estimates issued for the Technical Report were AUD28.64 million per year.

21.2.3.3 Transport and Accommodation

ACCOMMODATION COST DEVELOPMENT

Taking on board the Vista model for labor force accommodation of a workforce self-funded housing scheme based in Katherine and Pine Creek, the requirements for on-going use of any camp post the construction period was estimated as follows:

- Accommodation allowance to cover personnel recruitment, assuming a 20% turnover of the entire workforce annually, and assuming these personnel would consist of a four unit family requiring accommodation in the camp for an average of 2 months before sourcing their own accommodation. This provided an estimated requirement for 54 rooms in the camp per annum.
- Accommodation for contractors flying to site, largest of which would predominantly consist of the mill reline crew. Assuming a nominal sum of 10 other contractors throughout the year, and assuming these could be staggered to require accommodation for periods other than during mill relines, gave an estimated requirement for an additional 18 rooms.
- Accommodation for miscellaneous visitors, etc. where accommodation for whatever reason could not be mutually exclusive with mill relines provided a nominal requirement for 7 rooms.
- For the total on-going accommodation estimate of 69 rooms per annum, a requirement for 70 rooms was anticipated.

An allowance of AUD62.99 per man per day was made for a continuation of the partial construction camp.

TRANSPORT COST DEVELOPMENT

Using the numbers developed for the accommodation requirement, flights to Darwin from Perth were estimated at 225 return flights per annum. Allowing a 42% / 17% / 42% split between Low, Shoulder and High seasons respectively, and assuming all flights were at fully flexible fares provided the basis for annual flight expenditures.

TRANSPORT AND ACCOMMODATION COSTS

Final transport and accommodation cost estimates issued for the Technical Report were AUD1.810 million per year.

21.2.3.4 Power Requirements

Power usage was developed by a combination of methods, namely:

- Significant power consuming items had power consumptions calculated from base formulae and models, and included the following items:
  - Primary crusher;
  - Secondary crushers;
  - Ball mills;
  - Secondary Mills; and
  - HPGR Units.
- For smaller or steady state power consumers the power consumed was calculated as a factor of installed power, with the factor varying on known vendor motor oversizing propensities.
- Nominal allowances were made for some areas where actual installed power was estimated based on usual loads for such duties, and included items such as the air conditioners, lighting and small power, etc.

The total estimated power consumption is approximately 649 GWh/year.

21.2.3.5 Fuel

Fuel consumption estimates were developed for each item of process plant mobile equipment, by estimating annual operating hours and using vendor documented or estimated fuel consumptions for each equipment item.

Other plant items usually consuming fuel, namely power generation, product drying, borefield, etc. were all zero for the Mt Todd proposed operating plant.

21.2.3.6 Maintenance

Maintenance costs were developed by applying factors to FIS equipment costs for each of the two OPEX cases. The TTP maintenance cost estimating methodology is consistent with that of the Australasian Institute of Mining and Metallurgy (*Cost Estimation Handbook for the Australian Mining Industry*, AUSIMM, 1993). TTP factors have been developed over a period of time and fall within the AUSIMM guidelines.

Large wear items (crusher wear liners, ball mill lifters / liners) were identified and listed separately in the consumables section.

An additional allowance of 1.5% was applied across the site equipment to allow for sustaining capital expenditure. Maintenance cost estimates issued for the Technical Report were AUD11.520 million per year.

21.2.3.7 Reagents

Reagent costs were estimated by applying the ALS determined consumption rates with a quoted cost of delivered reagents to site.

Instances where consumption rates were altered from the original ALS testwork or previous assumptions included:

- Consumption of carbon was increased from 15 g/t to 20g/t based on TTP industry experience.

- Flocculant consumption was changed to 40 g/t for the Pre-Leach thickener as advised by RDi Minerals, based on recent test work.
- Sodium Cyanide changed to 876 g/t (leach feed) and Quick Lime increased to 2,800 g/t (leach feed) as advised by Vista and RDi Minerals, based on recent test work and the removal of the tailing's thickener.

Reagent prices were obtained from quotes from relevant suppliers. For the Technical Report only one vendor quote for the majority of reagents was available, with multiple additional quotes still pending.

Multiple suppliers were engaged for the highest expenditure reagent (sodium cyanide), with an Australian supplier chosen as the most cost-effective supplier. Further price sourcing from overseas suppliers was on-going at the time of writing.

Transport costs of reagents to site were sourced from reagent suppliers, in addition to an independent quote from a transport agency. The cheapest of the quotes for delivery from Darwin to Katherine was chosen as the cost to be used in the Technical Report, in this case it was from Seatram.

Reagent cost estimates issued for the Technical Report were AUD75.810 million per year.

21.2.3.8 Consumables

Consumable costs were estimated by calculating or estimate consumable consumption rates coupled with quotes or estimates for unit prices.

Consumption of mill balls was estimated by the selected mill vendor and based on the ore abrasion index, and since this item was one of the largest expenditures in the consumable category three quotes were received, with the most cost effective being Shandong Humain (China)

Where possible, transport costs were sourced from suppliers, however if they were not provided costs were sourced from other quotes. The quote from Shandong Huamin only included shipping to Darwin. Transport costs from Darwin to Katherine were sourced from the Molycorp quote.

In some instances where vendor advice was not received in a timely fashion, consumable quotes were scaled from previous studies. Consumable cost estimates issued for the Technical Report were AUD45.860million per year.

21.2.3.9 Equipment Hire

The Vista requirement to minimize upfront capital costs was used as the basis to initially assume all process plant mobile equipment, all process plant light vehicles and general site vehicles (ambulance, bus, coaches, etc.) would be hired or leased rather than purchased outright.

The overall cost effectiveness of the lease decision was further analyzed with the ultimate decision to purchase the vehicles outright. Consequently, the equipment hire operating costs reverted to zero, with the purchase costs then added to the capital costs. With all plant vehicles then treated as fully owned, an allowance was added for vehicle maintenance.

21.2.3.10 Contract/General Expenses

TTP standard factors were used for general expenses and general consumables, some items of which are a standard allowance and others which are linked to site personnel numbers (clothing, medical supplies, etc.).

General expenses and consumables allowed for included:

- General Consumables; Office and General Supplies, Tools and Equipment, Communications Maintenance Materials, Sampling and Analysis Consumables
- Contract Expenses; Environmental Monitoring Costs, Contracting Electrical Expenses
- General Expenses; Emergency Supply, Personnel Recruitment, Legal/Compliance, Office Communications, Safety Supplies

TTP's standard allowances were included for contract expenses, with the adjustments specific for Mt Todd including:

- Additional allowances for environmental monitoring costs as advised by Vista
- Additional contract electrical costs to allow for the complexity of interaction and maintaining dual source High Voltage power supplies

General / Contract Expenses in addition to General Consumables cost estimates issued for the Technical Report were $3.700 million per year.

21.2.4 Power Plant (POWER Engineers)

Please note that this Section describes costs in Australian Dollars (AUD).

21.2.4.1 Fuel Costs

Fuel cost analysis is based on baseload operation of the power station calculated using generating equipment Higher Heating Value (HHV) heat rates operating at the manufacturer standard conditions (37°C and 38% relative humidity).

Table 21-18 lists the fuel gas requirements for the Jenbacher J920 reciprocating engines.

Table 21-18: Estimated Fuel Cost Summary (AUD)

Description	Base Case GE Jenbacher J920 (Seven) (AUD)
Gross Output (kW)	72,709
Net Output (kW)	71,255
Plant Required Average Output (kW)	69,664
Auxiliary Loads (kW)	1,454
Availability	99.9%
HHV Net Heat Rate (kJ/kWh)	8,961
Thermal Efficiency	44.47%
Annual Fuel Gas Consumption (GJ/yr)	5,253,349
Annual Fuel Cost for Base Load (AUD/yr)	$36,773,445
Pipeline toll fees (AUD/yr)	$3,152,010

More than 80% of the costs associated with the power station operation can be attributed to fuel costs with the remainder being scheduled maintenance and replacement parts. Properly maintained reciprocating gas engines can operate up to 80,000 hours (approximately 9 years) between major engine block overhauls with annual inspections and general maintenance. Annual maintenance and inspections consist of functional checks and parts replacement of the lubricating oil and filters, inlet air filters, combustion system (spark plugs), and cooling system inspection. Minor maintenance occurs at the 20,000- and 40,000-hour intervals. Minor maintenance consists of inspection and possible replacement of bearings, cylinder heads/liners, pistons, couplings, and turbocharger components. It is anticipated that engine maintenance will be sequenced such that only a single engine is down at any time for maintenance to avoid plant disruption.

Operating costs include a minimal staff dedicated to the power station and will be based in the Administration Building. Personnel staff will consist of three swing shifts, at two operators per 12-hour shift, with a mechanic and instrumentation/electrical technician on one shift per day. One of the day shift operators can also serve as the control room manager. Labor rates are provided by Vista and include 27% salary on-costs. Plant personnel costs are shown in **Table 21-19**.

Table 21-19: Estimated Personnel Costs, Power Plant (AUD)

	Annual Salary (AUD)	Number on Staff
Power & Water Superintendent	$193,040	1
Power Station Operator	$115,570	5
Instrumentation/Electrical Technician	$154,940	1
Mechanic	$154,940	1
Total	**$1,080,770**	**8**

Table 21-20 lists the annual operating costs for the GE Jenbacher J920 gas engines to generate a nominal net output of 69.6MW, as required by the Base Case for the 20-year life of the mining project. This estimate includes operating overhead personnel to meet the mine's base power demand. Engine maintenance costs are estimated to be AUD0.0129 AUD/kWh, which are typical costs for reciprocating engine maintenance with a full maintenance contract (NREL, GRI, 2013). In addition, the engine maintenance costs include the cost associated with purchasing power from the utility grid during engine maintenance downtime. The power plant was estimated on a 20-year life to account for pre-production, production, and closure operations, which is approximately 19 years, and then rounded to 20 years of life to estimate the value for determination of sale at the planned end of mine life.

Table 21-20: Estimated Gas Turbine Maintenance Cost Schedule – 70MW (AUD)

Year	Engine Fuel Costs (AUD)	Engine Maintenance Costs (AUD)	On-Site Personnel (AUD)
1	$36,773,445	$13,584,342	$1,080,770
2	$36,773,445	$13,584,342	$1,080,770
3	$36,773,445	$13,584,342	$1,080,770
4	$36,773,445	$13,584,342	$1,080,770
5	$36,773,445	$13,584,342	$1,080,770
6	$36,773,445	$13,584,342	$1,080,770
7	$36,773,445	$13,584,342	$1,080,770

Year	Engine Fuel Costs (AUD)	Engine Maintenance Costs (AUD)	On-Site Personnel (AUD)
8	$36,773,445	$13,584,342	$1,080,770
9	$36,773,445	$13,584,342	$1,080,770
10	$36,773,445	$13,584,342	$1,080,770
11	$36,773,445	$13,584,342	$1,080,770
12	$36,773,445	$13,584,342	$1,080,770
13	$36,773,445	$13,584,342	$1,080,770
14	$36,773,445	$13,584,342	$1,080,770
15	$36,773,445	$13,584,342	$1,080,770
16	$36,773,445	$13,584,342	$1,080,770
17	$36,773,445	$13,584,342	$1,080,770
18	$36,773,445	$13,584,342	$1,080,770
19	$36,773,445	$13,584,342	$1,080,770
20	$36,773,445	$13,584,342	$1,080,770
Subtotal	***$735,468,900***	***$271,686,840***	***$21,615,400***
		TOTAL	**$1,028,771,140**

21.2.5 Water Treatment Plant (Tetra Tech)

Water treatment plant operating costs averaging US$0.09/t-milled.

21.2.6 Tailings Storage Facilities (Tetra Tech)

Tailings operating costs are estimated to average US$0.07/t-milled over the LoM. Tailings operating costs include shaping and compaction of the mine waste in the tailings embankments that hauled as a mining cost. Pumping and power costs for tailings facility operation are included in the Process Plant costing.

21.2.7 General & Administrative

G&A is estimated to be an average of US$1.11/t-milled over the LoM.

22.0 ECONOMIC ANALYSIS

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

Project economics for the Base Case are based on inputs developed by MDA, TTP, POWER Engineers and Tetra Tech. Economic results presented in the report suggest the following conclusions, assuming a 100% equity project:

- Mine Life: 13 years;
- Pre-Tax NPV5%: US$1,440 million, IRR: 30.4%;
- After-tax NPV5%: US$823 million, IRR: 23.4%;
- Payback (After-tax): 2.9 years;
- NT Royalty Paid: US$473 million;
- Australian Income Taxes Paid: US$553 million; and
- Cash costs (including JAAC Royalty): US$645.14/oz-Au.

After-tax net present value (NPV) is US$823 million, discounted at 5%. The after-tax internal rate of return (IRR) for the Project is 23.4%.

Costs and economic results are presented in Q3 2019 U.S. dollars unless otherwise stated. No escalation has been applied to capital or operating costs. The 5% discount rate used is a gold industry norm.

Technical economic tables and figures presented in this volume 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 Principal Assumptions

Parameters used in the analysis are shown in **Table 22-1**. These parameters are based upon current market conditions, vendor quotes, design criteria developed by Vista and their consultants, and benchmarks against similar existing projects.

Table 22-1: TEM Principal Assumptions

Principal Assumptions	Unit	Parameter
Construction Period	Years	2
Commissioning & Ramp-Up	Years	0.5
Mine Life	Years	13
Closure Period	Years	4
Operating Days	Days / Year	355
Gold Price	US$	$1,350
JAAC Royalty	%	1%
Exchange Rate	AUD:US$	0.7:1
Diesel Fuel	AUD/L	$0.850
Natural Gas	AUD/GJ	$7.00

Principal Assumptions	Unit	Parameter
Electric Power – From Grid	AUD/kWh	$0.300
Electric Power – From Plant	AUD/kWh	$0.076

The Project will commence at a production rate of 50,000 tpd. Fresh ore production will originate from the open pit mine and will be treated using conventional CIP technology. Once ore is exhausted from the pit, the reserves in the existing heap leach pad will then be processed.

Projected revenues from the sale of gold doré are based upon a market price of US$1,350/oz-Au. Vista has used indicative pricing from the Perth Mint for the sale of its product. It is too early to enter into definitive agreement with refiners as of the date of this Technical Report. However, refinery assumptions used in the technical economic model (TEM) are indicative of current refiner rates.

Refining costs are summarized in **Table 22-2** resulting in an all-in refining cost of US$3.22/Au-oz over the LoM.

Table 22-2: Estimated Refining Costs (US$)

Cost Component	Units	Cost (US$)
Refining Fee	$/oz	0.75
Gold Retention	% of gold sales	0.10%
Purchase Discount-Gold	$/oz	0.50
Assay Fee	$/oz	95.00
Environmental Fee	$/oz	50.00
Freight & Insurance	$/oz	0.20

The Project is subject to a 20% net value-based mineral royalty imposed by the Norther Territory Government and the Commonwealth corporate income tax based on 30% of taxable income. The NT Royalty is among deductions permitted in determining taxable income.

22.2 LoM Production

Ore will be mined using open pit mining methods. Production over the LoM is summarized in **Table 22-3**.

Table 22-3: LoM Ore Production

Production	kt	g/t	Contained Au (koz)
Waste	522,990	-	-
Ore	207,687	0.84	5,616
Heap Leach	13,354	0.54	232
Total Production*	**221,041**	**0.82**	**5,848**

**Total production excludes waste tonnes.*

The Project has been planned as an open-pit truck and shovel operation. Open pit ore totals 208 Mt grading 0.84 g/t and contains 5.6 Moz of gold. Open pit production will have a 2.5:1 strip ratio over the 13-year LoM. Upon completion of conventional mining, the existing heap leach pad will be processed.

Ore is planned to be processed in a large comminution circuit consisting of a gyratory crusher, two cone crushers, two HPGR crushers, and two primary ball mills followed by 10 FLS VXP mills for secondary grinding as discussed in **Section 17.0 – Recovery Methods**. Vista plans to recover gold in a conventional carbon-in-pulp ("CIP") recovery circuit. Process recovery was determined based on ore types. Three ore types, sulfide, mixed, and oxide were identified for the open pit and will have recoveries of 92.9%, 91.8%, and 91.5%, respectively. The heap leach pad will have a recovery of 90.7%. An additional 1% for net solution loss is applied to all the deposits and heap leach which results in a LoM average recovery of 91.9%.

22.3 Capital Costs

LoM capital cost requirements are estimated at US$1,222 million as summarized in **Table 22-4**. Initial capital of US$826 million is estimated to be required to commence operations. Sustaining capital of US$397 million is required over the LoM and accounts for capitalized stripping in the open pit, mine equipment additions and replacements, and tailings dam raises.

Table 22-4: Estimated LoM Capital Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
2000	Mining	7.3%	$121,239	$5,720	**$126,958**	$406,347	$32,677	**$439,024**	$527,586	$38,396	**$565,982**
3000	Process Plant	13.9%	$366,693	$51,073	**$417,766**	$17,027	$2,222	**$19,249**	$383,720	$53,295	**$437,016**
4000	Project Services	10.0%	$109,204	$12,681	**$121,885**	$72,448	$5,455	**$77,903**	$181,651	$18,136	**$199,787**
5000	Project Infrastructure	13.2%	$26,160	$3,463	**$29,623**	$0	$0	**$0**	$26,160	$3,463	**$29,623**
6000	Permanent Accommodation	10.0%	$60	$6	**$66**	$0	$0	**$0**	$60	$6	**$66**
7000	Site Establishment & Early Works	11.4%	$17,537	$1,995	**$19,532**	$0	$0	**$0**	$17,537	$1,995	**$19,532**
8000	Management, Engineering, EPCM Svcs	11.8%	$82,058	$9,721	**$91,779**	$0	$0	**$0**	$82,058	$9,721	**$91,779**
9000	Pre-Production Costs	12.3%	$16,121	$1,982	**$18,102**	$0	$0	**$0**	$16,121	$1,982	**$18,102**
10000	Asset Sale	0.0%	$0	$0	**$0**	($139,631)	$0	**($139,631)**	($139,631)	$0	**($139,631)**
	Capital Cost	**11.6%**	**$739,072**	**$86,641**	**$825,712**	**$356,191**	**$40,354**	**$396,545**	**$1,095,263**	**$126,994**	**$1,222,257**

22.3.1 2000 Mining

LoM capital cost requirements are estimated to value US$566 million with an initial cost of US$127 million as seen in **Table 22-25**.

Table 22-5: Estimated Mining Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
2000	MINING										
2100	Capitalized Mine Operating	10.0%	$19,201	$1,920	**$21,121**	$88,522	$8,852	**$97,374**	$107,722	$10,772	**$118,494**
2200	Mine Production Equipment	2.9%	$91,116	$2,629	**$93,745**	$196,415	$5,667	**$202,083**	$287,531	$8,296	**$295,827**
2300	Mine Support Facilities	9.0%	$8,762	$787	**$9,548**	$906	$81	**$988**	$9,668	$868	**$10,536**
2400	Mine Support Services	20.0%	$1,195	$239	**$1,434**	$0	$0	**$0**	$1,195	$239	**$1,434**
2500	Mine Dewatering/Drainage	15.0%	$858	$129	**$987**	$411	$62	**$473**	$1,269	$190	**$1,459**
2900	Mine Closure	15.0%	$108	$16	**$124**	$120,094	$18,014	**$138,108**	$120,201	$18,030	**$138,232**
	Mining	**7.3%**	**$121,239**	**$5,720**	**$126,958**	**$406,347**	**$32,677**	**$439,024**	**$527,586**	**$38,396**	**$565,982**

22.3.2 3000 Process Plant

Estimated CIP process plant capital costs are shown in **Table 22-6**. Initial capital totaling US$418 million is estimated to be required for the CIP process plant; a total capital of US$437 million is required.

Table 22-6: Estimated CIP Process Plant Capital Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
3000	PROCESS PLANT										
3100	Crushing & Screening	13.7%	$44,878	$6,141	**$51,019**	$2,168	$297	**$2,464**	$47,046	$6,438	**$53,483**
3200	Coarse Ore Stockpile, Reclaim, HPGR	13.9%	$84,151	$11,683	**$95,834**	$4,644	$645	**$5,288**	$88,794	$12,328	**$101,122**
3300	Classification & Grinding	12.3%	$96,862	$11,905	**$108,767**	$7,208	$886	**$8,094**	$104,069	$12,791	**$116,860**
3400	Pre-leach Thickening, Leach & CIP	12.7%	$59,623	$7,555	**$67,179**	$1,499	$190	**$1,689**	$61,122	$7,745	**$68,868**
3500	Desorption & Goldroom	13.4%	$8,067	$1,077	**$9,144**	$891	$119	**$1,010**	$8,958	$1,196	**$10,154**
3600	Detoxification & Tailings	15.3%	$7,765	$1,185	**$8,950**	$335	$51	**$386**	$8,100	$1,236	**$9,336**
3700	Reagents	10.0%	$10,317	$1,035	**$11,352**	$209	$21	**$230**	$10,526	$1,056	**$11,583**
3800	Process Plant Services	19.1%	$55,031	$10,491	**$65,521**	$74	$14	**$88**	$55,105	$10,505	**$65,609**
	Process Plant	**13.9%**	**$366,693**	**$51,073**	**$417,766**	**$17,027**	**$2,222**	**$19,249**	**$383,720**	**$53,295**	**$437,016**

22.3.3 4000 Project Services

Project services are estimated to have a LoM capital value US$200 million, with an initial capital value of US$122 million.

Table 22-7: Estimated Project Services Capital Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
4000	PROJECT SERVICES										
4100	Water Distribution & Water Treatment Plant	14.0%	$17,783	$2,486	**$20,269**	$0	$0	**$0**	$17,783	$2,486	**$20,269**
4200	Power Supply	11.6%	$81,679	$9,457	**$91,137**	$0	$0	**$0**	$81,679	$9,457	**$91,137**
4300	Communications	15.9%	$644	$102	**$746**	$0	$0	**$0**	$644	$102	**$746**
4400	Tailings Dams 1 & 2	7.5%	$6,154	$463	**$6,618**	$72,448	$5,455	**$77,903**	$78,602	$5,918	**$84,521**
4500	Waste Disposal	15.0%	$250	$37	**$287**	$0	$0	**$0**	$250	$37	**$287**
4600	Plant Mobile Equipment	5.0%	$2,693	$135	**$2,828**	$0	$0	**$0**	$2,693	$135	**$2,828**
4800	Fuel Storage & Distribution (Plant)	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
4900	Project Services - Closure	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
	Project Services	**10.0%**	**$109,204**	**$12,681**	**$121,885**	**$72,448**	**$5,455**	**$77,903**	**$181,651**	**$18,136**	**$199,787**

22.3.4 5000 Project Infrastructure

The total project infrastructure is estimated to value US$30 million, which consists only of initial costs, no sustaining capital is expected. A detailed outline of costs is shown in **Table 22-8**.

Table 22-8: Estimated Project Infrastructure Capital Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
5000	PROJECT INFRASTRUCTURE										
5100	Site Preparation	13.8%	$17,997	$2,483	**$20,480**	$0	$0	**$0**	$17,997	$2,483	**$20,480**
5200	Support Buildings	10.1%	$4,256	$431	**$4,687**	$0	$0	**$0**	$4,256	$431	**$4,687**
5300	Access Roads, Parking & Laydown	10.0%	$740	$74	**$814**	$0	$0	**$0**	$740	$74	**$814**
5400	Heavy Lift Cranage	15.0%	$2,062	$309	**$2,371**	$0	$0	**$0**	$2,062	$309	**$2,371**
5500	TBA	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
5600	Bulk Transport	15.0%	$396	$59	**$455**	$0	$0	**$0**	$396	$59	**$455**
5700	Power Transmission	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
5800	Communications	15.0%	$710	$106	**$816**	$0	$0	**$0**	$710	$106	**$816**
	Project Infrastructure	**13.2%**	**$26,160**	**$3,463**	**$29,623**	**$0**	**$0**	**$0**	**$26,160**	**$3,463**	**$29,623**

22.3.5 6000 Permanent Accommodation

Total capital for Permanent Accommodations values at US$66 thousand as shown in **Table 22-9**.

Table 22-9: Estimated Permanent Accommodation Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
6000	PERMANENT ACCOMMODATION										
6100	Permanent Accommodation	10.0%	$60	$6	**$66**	$0	$0	**$0**	$60	$6	**$66**
	Permanent Accommodation	**10.0%**	**$60**	**$6**	**$66**	**$0**	**$0**	**$0**	**$60**	**$6**	**$66**

22.3.6 7000 Site Establishment & Early Works

Site Establishment and early works capital costs are estimated to total US$20 million over the LoM as shown in **Table 22-10**. These costs occur in pre-production.

Table 22-10: Estimated Site Establishment & Early Works (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
7000	SITE ESTABLISHMENT & EARLY WORKS										
7300	Construction Camp	11.4%	$17,537	$1,995	**$19,532**	$0	$0	**$0**	$17,537	$1,995	**$19,532**
7400	Dewatering	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
7500	Demolition & Removal	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
	Site Establishment & Early Works	**11.4%**	**$17,537**	**$1,995**	**$19,532**	**$0**	**$0**	**$0**	**$17,537**	**$1,995**	**$19,532**

22.3.7 8000 Management, Engineering, EPCM Services

Management, engineering, and EPCM services are estimated to value US$92 million. These costs are shown in **Table 22-11**.

Table 22-11: Estimated Management, Engineering, EPCM Services (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
8000	MANAGEMENT, ENGINEERING, EPCM SVCS										
8100	EPCM Services	9.7%	$42,177	$4,076	**$46,253**	$0	$0	**$0**	$42,177	$4,076	**$46,253**
8200	External Consulting & Testing	20.0%	$840	$168	**$1,008**	$0	$0	**$0**	$840	$168	**$1,008**
8300	Commissioning	16.4%	$4,472	$734	**$5,206**	$0	$0	**$0**	$4,472	$734	**$5,206**
8400	Owner's Engineering & Management	12.9%	$30,547	$3,939	**$34,486**	$0	$0	**$0**	$30,547	$3,939	**$34,486**
8800	License, fees & Legal Services	20.0%	$2,011	$402	**$2,413**	$0	$0	**$0**	$2,011	$402	**$2,413**
8900	Project Insurance	20.0%	$2,011	$402	**$2,413**	$0	$0	**$0**	$2,011	$402	**$2,413**
	Management, Engineering, EPCM Svcs	**11.8%**	**$82,058**	**$9,721**	**$91,779**	**$0**	**$0**	**$0**	**$82,058**	**$9,721**	**$91,779**

22.3.8 9000 Pre-Production Costs

Pre-production capital values US$18 million as shown in **Table 22-12.** This cost will occur pre-preproduction.

Table 22-12: Estimated Pre-Production Costs (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
9000	PRE-PRODUCTION COSTS										
9100	PPD Labor	20.0%	$1,005	$201	**$1,206**	$0	$0	**$0**	$1,005	$201	**$1,206**
9200	Commissioning Expenses	25.0%	$2,011	$503	**$2,513**	$0	$0	**$0**	$2,011	$503	**$2,513**
9300	Capital Spares	15.0%	$7,101	$1,065	**$8,166**	$0	$0	**$0**	$7,101	$1,065	**$8,166**
9400	Stores & Inventory	15.0%	$1,420	$213	**$1,633**	$0	$0	**$0**	$1,420	$213	**$1,633**
9500	PPD Capitalized Operating	0.0%	$4,584	$0	**$4,584**	$0	$0	**$0**	$4,584	$0	**$4,584**
9600	Escalation & Foreign Currency Exchange	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
	Pre-Production Costs	**12.3%**	**$16,121**	**$1,982**	**$18,102**	**$0**	**$0**	**$0**	**$16,121**	**$1,982**	**$18,102**

22.3.9 1000 Asset Sale

Table 22-13 depicts a total capital value of US$140 million.

Table 22-13: Estimated Asset Sale (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
10000	ASSET SALE										
10100	Mine	0.0%	$0	$0	**$0**	($52,926)	$0	**($52,926)**	($52,926)	$0	**($52,926)**
10200	Process Plant	0.0%	$0	$0	**$0**	($18,352)	$0	**($18,352)**	($18,352)	$0	**($18,352)**
10300	Power Plant (Sold to 3rd Party)	0.0%	$0	$0	**$0**	($68,352)	$0	**($68,352)**	($68,352)	$0	**($68,352)**
	Asset Sale	**0.0%**	**$0**	**$0**	**$0**	**($139,631)**	**$0**	**($139,631)**	**($139,631)**	**$0**	**($139,631)**

22.4 Operating Costs

Estimated LoM operating costs are summarized in **Table 22-14**. The operating costs will average US$15.18/t-milled over the LoM.

Table 22-14: Estimated LoM Operating Costs (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.10	0.03
Mine Maintenance	0.11	0.03
Engineering	0.05	0.01
Geology	0.03	0.01
Drilling	0.77	0.23
Blasting	1.17	0.35
Loading	0.60	0.18
Hauling	2.74	0.83
Mine Support	0.43	0.13
Mine Dewatering	0.01	0.004
Open Pit Mine	**6.02**	**1.82**
CIP PROCESS PLANT		
Labor	0.79	-
3100-Crush/Screen/Stockpile	0.18	-
3200-Reclaim & HPGR	0.44	-
3300-Classification & Grinding	3.14	-
3400-Pre-Leach,Thick/Aeration/CIP	0.13	-
3500-Desorption, Gold Room	0.02	-
3600-Detox & Tailings Pumping	0.06	-
3700-Reagents	2.98	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.01	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**7.88**	**-**
Project Services	$0.16	-
G&A	$1.11	-
Operating Costs	**$15.18**	**-**

22.4.1 Open Pit Mining

Mining costs (including open pit mining, rehandle, and heap leach pad, but excluding capitalized preproduction mining costs) are shown in **Table 22-15**. Costs will average US$1.82/t-mined (US$6.02/t-milled) over the LoM. Hauling is the highest cost item, US$0.83/t-mined (US$2.74/t-milled). Hauling costs include transport of select mine waste to the TSF for embankment construction. Note also that unit costs per tonne milled include 13.4 Mt of heap leach ore which is not mined.

Table 22-15: Estimated Open Pit Operating Costs (US$)

Description	US$/t-mined	US$/t-milled	Total (US$000s)
Mine General Service	$0.03	$0.10	**$21,208**
Mine Maintenance	$0.03	$0.11	**$24,605**
Engineering	$0.01	$0.05	**$10,410**
Geology	$0.01	$0.03	**$7,181**
Drilling	$0.23	$0.77	**$170,765**
Blasting	$0.35	$1.17	**$259,224**
Loading	$0.18	$0.60	**$133,424**
Hauling	$0.83	$2.74	**$606,582**
Mine Support	$0.13	$0.43	**$94,880**
Subtotal	**$1.82**	**$6.01**	**$1,328,278**
Mine Dewatering	$0.004	$0.012	$2,698
Total Open Pit Mining	**$1.82**	**$6.02**	**$1,330,976**

22.4.2 CIP Process Plant

CIP process plant operating costs averaging US$7.88/t-milled are shown in **Table 22-16**.

Table 22-16: Estimated CIP Process Plant Operating Costs (US$)

Description	US$/t-milled	Total (US$000s)
Labor	$0.79	$174,985
3100 – Crush/Screen/Stockpile	$0.18	$40,426
3200 – Reclaim & HPGR	$0.44	$96,902
3300 – Classification & Grinding	$3.14	$693,910
3400 – Pre-Leach, Thick/Aeration/CIP	$0.13	$28,863
3500 – Desorption, Gold Room	$0.02	$4,087
3600 – Detox & Tailings Pumping	$0.06	$12,856
3700 – Reagents	$2.98	$659,075
3800 – Plant Services	$0.04	$8,463
Mining, Infrastructure, & Misc	$0.06	$12,494
Generable Consumables	$0.01	$2,327
Plant Mobile Equipment	$0.01	$1,793
Plant Gas consumption	$0.03	$6,339
Total CIP Process Plant	**$7.88**	**$1,742,519**

22.4.3 Water Treatment Plant

Water treatment plant operating costs averaging US$0.09/t-milled are shown in **Table 22-17**.

Table 22-17: Estimated Water Treatment Plant Operating Costs (US$)

Description	US$/t-milled	Total (US$000s)
CHEMICALS		
Caustic	$0.000	$0
Chlorine	$0.000	$0
Citric Acid	$0.000	$0
Ferric Chloride	$0.015	$3,340
Ferrous Sulfate	$0.000	$0
Lime	$0.031	$6,919
Sodium Hydrosulfate	$0.002	$342
Sulfuric Acid	$0.005	$1,073
POWER		
Electricity	$0.010	$2,285

Description	US$/t-milled	Total (US$000s)
LABOR		
Operator	$0.006	$1,302
Maintenance	$0.016	$3,455
Total Water Treatment Plant	**$0.085**	**$18,715**

22.4.4 Tailings

Tailings will average US$0.07/t-milled over the LoM as shown in **Table 22-18**. Tailings operating costs include shaping and compaction of the mine waste in the tailings embankments that hauled as a mining cost. Pumping and power costs for tailings facility operation are included in the Process Plant costing.

Table 22-18: Estimated Tailings Operating Costs (US$)

Description	US$/t-milled	Total (US$000s)
Labor	$0.032	$7,137
Equipment	$0.041	$9,155
Total Tailings	**$0.074**	**$16,291**

22.4.5 General & Administrative

G&A will average US$1.11/t-milled over the LoM as shown in **Table 22-19**.

Table 22-19: Estimated G&A Operating Costs (US$)

Description	US$/t-milled	Total (US$000s)
Labor, G&A	$0.456	$100,725
Expenses	$0.142	$31,331
Transport & Accommodation	$0.074	$16,450
Fleet Vehicles	$0.015	$3,403
Corporate Overhead	$0.427	$94,375
Total G&A	**$1.114**	**$246,285**

22.4.6 JAAC Royalty

JAAC Royalty costs averaging US$0.32/t-milled are shown in **Table 22-20**.

Table 22-20: Estimated JAAC Royalty Costs (US$)

	US$/t-milled	Total (US$000s)
JAAC Royalty	$0.324	$71,615
Total JAAC Royalty	**$0.324**	**$71,615**

22.4.7 Refining Costs

Refining costs averaging US$0.08/t-milled are shown in **Table 22-21**.

Table 22-21: Estimated Refining Costs (US$)

	US$/t-milled	Total (US$000s)
Refining Fee	$0.018	$3,979
Golden Retention	$0.032	$7,161
Purchase Discount-Gold	$0.012	$2,652
Assay Fee	$0.003	$761
Environmental Fee	$0.007	$1,456
Freight & Insurance	$0.005	$1,066
Total Refinery Costs	**$0.077**	**$17,075**

22.4.8 Operating Cost Inputs

Inputs used to estimate operating costs are summarized in this section.

22.4.8.1 Labor

The labor breakdown shown in **Table 22-22** represents the personnel contingent at steady state operations. Labor rates are fully burdened, are presented in Australian Dollars, and are based upon recent Australian labor rate surveys provided by Vista. Additionally, matrix showing salaries at levels by position is provided in **Table 22-23**.

Table 22-22: Estimated Labor Rates & Costs (AUD)

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Resident Manager	$335,000	27.0%	$90,450	1	DP	1	$425,450
Mining Manager	$244,000	27.0%	$65,880	1	DP	1	$309,880
Processing Manager	$244,000	27.0%	$65,880	1	DP	1	$309,880
Admin Manager*	$203,000	27.0%	$54,810	0	DP	0	$0
OHS Manager*	$178,000	27.0%	$48,060	0	DP	0	$0
NPI Manager	$244,000	27.0%	$65,880	1	DP	1	$309,880
Subtotal				**4**		**4**	**$1,355,090**
HR Director	$183,000	27.0%	$49,410	1	DP	1	$232,410
Recruiting Officer	$107,000	27.0%	$28,890	2	DP	2	$271,780
Administration Secretary	$86,000	27.0%	$23,220	1	DP	1	$109,220
Administrative Assistant	$81,000	27.0%	$21,870	1	SW	3	$308,610
Receptionist	$66,000	27.0%	$17,820	1	DP	1	$83,820
Indigenous Liaison Officer	$96,000	27.0%	$25,920	1	DP	1	$121,920
Security Officer	$81,000	27.0%	$21,870	2	SW	6	$617,220
Community Liaison Officer	$96,000	27.0%	$25,920	1	DP	1	$121,920
Head of Security	$112,000	27.0%	$30,240	1	DP	1	$142,240
External Affairs Director	$178,000	27.0%	$48,060	1	DP	1	$226,060
Support Services Director	$178,000	27.0%	$48,060	1	DP	1	$226,060
Subtotal				**13**		**19**	**$2,461,260**
Financial Controller	$183,000	27.0%	$49,410	1	DP	1	$232,410
Senior Accountant	$137,000	27.0%	$36,990	1	DP	1	$173,990

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Accountant	$112,000	27.0%	$30,240	1	DP	1	$142,240
Accounting Clerk	$76,000	27.0%	$20,520	1	DW	2	$193,040
Payroll Clerk	$76,000	27.0%	$20,520	1	DP	1	$96,520
Subtotal				**5**		**6**	**$838,200**
IT Supervisor	$112,000	27.0%	$30,240	1	DP	1	$142,240
IT Technician	$91,000	27.0%	$24,570	1	DP	1	$115,570
Database Administrator	$91,000	27.0%	$24,570	1	DP	1	$115,570
Subtotal				**3**		**3**	**$373,380**
Metallurgical Superintendent	$178,000	27.0%	$48,060	1	DP	1	$226,060
Chief Metallurgist	$178,000	27.0%	$48,060	1	DP	1	$226,060
Plant / Production Metallurgist	$157,000	27.0%	$42,390	2	DP	2	$398,780
Process Control Engineer	$137,000	27.0%	$36,990	1	DP	1	$173,990
Metallurgical Clerk	$91,000	27.0%	$24,570	1	SW	3	$346,710
Gold Room Supervisor	$112,000	27.0%	$30,240	1	DP	1	$142,240
Refiner	$96,000	27.0%	$25,920	1	DW	2	$243,840
Gold Room Technician	$91,000	27.0%	$24,570	1	SW	3	$346,710
Subtotal				**9**		**14**	**$2,104,390**
Production Superintendent	$183,000	27.0%	$49,410	1	DP	1	$232,410
General Foreman	$152,000	27.0%	$41,040	1	DP	1	$193,040
Shift Foreman	$127,000	27.0%	$34,290	1	SW	3	$483,870
Plant Lead Operator	$112,000	27.0%	$30,240	1	SW	3	$426,720
Shift Operator - Crushing	$102,000	27.0%	$27,540	1	SW	3	$388,620
Shift Operator - HPGR	$102,000	27.0%	$27,540	1	SW	3	$388,620
Shift Operator - Mills	$102,000	27.0%	$27,540	1	SW	3	$388,620
Shift Operator - Leach	$102,000	27.0%	$27,540	1	SW	3	$388,620
Shift Operator - Elution	$102,000	27.0%	$27,540	1	SW	3	$388,620

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Shift Operator - Detox / Tailings	$102,000	27.0%	$27,540	1	SW	3	$388,620
Shift Operator - Reagents	$102,000	27.0%	$27,540	2	SW	6	$777,240
Shift Operator - CCR	$102,000	27.0%	$27,540	1	SW	3	$388,620
Shift Operator - Tailings Dam	$91,000	27.0%	$24,570	2	SW	6	$693,420
Shift Operator - Day Gang	$91,000	27.0%	$24,570	4	SW	12	$1,386,840
Subtotal				**19**		**53**	**$6,913,880**
Maintenance Superintendent	$178,000	27.0%	$48,060	1	DP	1	$226,060
Maintenance General Foreman	$152,000	27.0%	$41,040	1	DP	1	$193,040
Maintenance Planner	$157,000	27.0%	$42,390	1	DP	1	$199,390
Mechanical Fitter	$122,000	27.0%	$32,940	3	SW	9	$1,394,460
Crane Operator	$102,000	27.0%	$27,540	1	DW	2	$259,080
Boilermaker / Welder	$127,000	27.0%	$34,290	2	DW	4	$645,160
Pipe Fitters	$127,000	27.0%	$34,290	1	DW	2	$322,580
Greasers	$91,000	27.0%	$24,570	1	SW	3	$346,710
Trades Assistants	$86,000	27.0%	$23,220	1	SW	3	$327,660
Electrical General Foreman	$147,000	27.0%	$39,690	1	DP	1	$186,690
HV Electrical Supervisor	$122,000	27.0%	$32,940	1	DW	2	$309,880
Electrician	$122,000	27.0%	$32,940	3	SW	9	$1,394,460
Instrument Technician	$122,000	27.0%	$32,940	1	SW	3	$464,820
Apprentices	$61,000	27.0%	$16,470	2	SW	6	$464,820
Subtotal				**20**		**47**	**$6,734,810**
Laboratory Supervisor	$127,000	27.0%	$34,290	1	SW	3	$483,870
Chemist	$122,000	27.0%	$32,940	1	SW	3	$464,820
Lab Technician	$102,000	27.0%	$27,540	2	SW	6	$777,240
Sample Prep Technician	$76,000	27.0%	$20,520	3	SW	9	$868,680
Subtotal				**7**		**21**	**$2,594,610**

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Engineering Superintendent	$183,000	27.0%	$49,410	1	DP	1	$232,410
Chief Mining Engineer	$162,000	27.0%	$43,740	1	DP	1	$205,740
Senior Mining Engineer	$142,000	27.0%	$38,340	0	DP	0	$0
Mining Engineer	$132,000	27.0%	$35,640	1	DW	2	$335,280
Senior Mine Planning Engineer	$132,000	27.0%	$35,640	1	DP	1	$167,640
Mine Clerk	$102,000	27.0%	$27,540	1	DP	1	$129,540
Subtotal				**5**		**6**	**$1,070,610**
Operations Superintendent	$183,000	27.0%	$49,410	1	DP	1	$232,410
Mine General Foreman	$152,000	27.0%	$41,040	1	DP	1	$193,040
Drill and Blast Foreman	$127,000	27.0%	$34,290	1	DW	2	$322,580
Drill and Blast Technician	$91,000	27.0%	$24,570	1	DW	2	$231,140
Blasting Assistant	$86,000	27.0%	$23,220	1	DW	2	$218,440
Loading Operator	$117,000	27.0%	$31,590	3	SW	9	$1,337,310
Haul Truck Operator	$102,000	27.0%	$27,540	16	SW	48	$6,217,920
Drill Operators	$117,000	27.0%	$31,590	8	SW	24	$3,566,160
Mechanics	$122,000	27.0%	$32,940	0	SW	0	$0
Welders	$127,000	27.0%	$34,290	0	SW	0	$0
Servicemen	$81,000	27.0%	$21,870	0	SW	0	$0
Aux Equipment Operators	$107,000	27.0%	$28,890	12	SW	36	$4,892,040
Mine Shift Foreman	$127,000	27.0%	$34,290	1	SW	3	$483,870
Subtotal				**45**		**128**	**$17,694,910**
Maintenance Superintendent	$178,000	27.0%	$48,060	1	DP	1	$226,060
Maintenance General Foreman	$152,000	27.0%	$41,040	1	DP	1	$193,040
Light Vehicle Mechanic	$122,000	27.0%	$32,940	2	DW	4	$619,760
Tireman	$91,000	27.0%	$24,570	1	DW	2	$231,140
Shop Laborer	$86,000	27.0%	$23,220	2	SW	6	$655,320

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Service, Fuel & Lube	$81,000	27.0%	$21,870	3	SW	9	$925,830
Maintenance Planner	$127,000	27.0%	$34,290	1	DP	1	$161,290
Subtotal				**11**		**24**	**$3,012,440**
Chief Surveyor	$152,000	27.0%	$41,040	1	DP	1	$193,040
Mine Surveyor	$117,000	27.0%	$31,590	1	DW	2	$297,180
Surveying Helper	$81,000	27.0%	$21,870	1	DW	2	$205,740
Subtotal				**3**		**5**	**$695,960**
GEOLOGY							
Geology Superintendent	$178,000	27.0%	$48,060	1	DP	1	$226,060
Grade Control Geologist	$147,000	27.0%	$39,690	1	DW	2	$373,380
Exploration Geologist	$112,000	27.0%	$30,240	1	DP	1	$142,240
Resource Geologist	$157,000	27.0%	$42,390	1	DP	1	$199,390
Pit Geology Technician	$91,000	27.0%	$24,570	3	DP	3	$346,710
Geology Field Technician	$81,000	27.0%	$21,870	2	DP	2	$205,740
Subtotal				**9**		**10**	**$1,493,520**
Purchasing Director	$152,000	27.0%	$41,040	1	DP	1	$193,040
Business Development Officer	$96,000	27.0%	$25,920	1	DP	1	$121,920
Logistics Officer	$132,000	27.0%	$35,640	1	DP	1	$167,640
Purchasing Officer	$96,000	27.0%	$25,920	1	DP	1	$121,920
Contracts Officer	$112,000	27.0%	$30,240	1	DP	1	$142,240
Store Person	$86,000	27.0%	$23,220	1	SW	3	$327,660
Subtotal				**6**		**8**	**$1,074,420**
OHS Superintendent	$178,000	27.0%	$48,060	1	DP	1	$226,060
Safety Officer	$117,000	27.0%	$31,590	1	SW	3	$445,770
Paramedic / Nurse	$117,000	27.0%	$31,590	1	SW	3	$445,770
Environmental Superintendent	$152,000	27.0%	$41,040	1	DP	1	$193,040

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Environmental Officer - Monitoring	$107,000	27.0%	$28,890	1	DP	1	$135,890
Environmental Officer - Compliance	$107,000	27.0%	$28,890	1	DP	1	$135,890
Subtotal				**6**		**10**	**$1,582,420**
Training Coordinator	$152,000	27.0%	$41,040	1	DP	1	$193,040
Training Officer - Plant	$122,000	27.0%	$32,940	2	DW	4	$619,760
Training Officer - Mining	$122,000	27.0%	$32,940	2	DW	4	$619,760
Subtotal				**5**		**9**	**$1,432,560**
Camp Manager	$96,000	27.0%	$25,920	0	DP	0	$0
Camp Admin	$71,000	27.0%	$19,170	0	DW	0	$0
Cook Staff	$86,000	27.0%	$23,220	0	SW	0	$0
Cleaning Staff	$86,000	27.0%	$23,220	0	DW	0	$0
Camp Maintenance	$102,000	27.0%	$27,540	0	DP	0	$0
Bus Drivers	$81,000	27.0%	$21,870	4	SW	12	$1,234,440
Subtotal				**4**		**12**	**$1,234,440**
Power Station Operator	$91,000	27.0%	$24,570	1	SW	3	$346,710
Electrician	$122,000	27.0%	$32,940	1	SW	3	$464,820
Mechanic	$122,000	27.0%	$32,940	1	SW	3	$464,820
Subtotal				**3**		**9**	**$1,276,350**
Power & Water Superintendent	$152,000	27.0%	$41,040	1	DP	1	$193,040
Water Plant Operator	$91,000	27.0%	$24,570	1	SW	3	$346,710
Water Plant Mechanic	$122,000	27.0%	$32,940	1	SW	3	$464,820
Subtotal				**3**		**7**	**$1,004,570**
Dozer Operator	$107,000	27.0%	$28,890	2	DW	4	$543,560
Loader Operator	$107,000	27.0%	$28,890	0.8	DP	0.8	$108,712
Haul Truck Operator	$102,000	27.0%	$27,540	0.3	DP	0.3	$38,862
Subtotal				**3**		**5**	**$691,134**

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Dozer Operator	$107,000	27.0%	$28,890	1.3	DP	1.3	$176,657
Loader Operator	$107,000	27.0%	$28,890	0.17	DP	0.17	$23,101
Haul Truck Operator	$102,000	27.0%	$27,540	0.51	DP	0.51	$66,065
Crane Operator	$112,000	27.0%	$30,240	0.14	DP	0.14	$19,914
Subtotal				**2**		**2**	**$285,737**
Project Superintendent	$203,000	27.0%	$54,810	1	DP	1	$257,810
Project Engineer	$167,000	27.0%	$45,090	1	DW	2	$424,180
Civil Engineer	$132,000	27.0%	$35,640	0	DW	0	$0
Geotechnical Engineer	$152,000	27.0%	$41,040	0	DW	0	$0
CAD Draftsman	$89,000	27.0%	$24,030	0	DW	0	$0
Piping Engineer	$107,000	27.0%	$28,890	0	DW	0	$0
Document Controller	$76,000	27.0%	$20,520	0	DW	0	$0
Construction Supervisor	$132,000	27.0%	$35,640	0	DW	0	$0
Subtotal				**2**		**3**	**$681,990**
TOTAL ONSITE PERSONNEL						**405**	**$56,606,681**

**Vista has identified these as possible needs, but they are not currently in the total manpower calculations.*

Table 22-23: Position & Salary Matrix (AUD)

Salary (AUD)	Mine Technical	Mine Operations	Mine Maintenance	Plant Technical	Plant Operations	Plant Maintenance	NPI	Administration
$335,000								Resident Manager
$244,000		Mining Manager			Processing Manager		NPI Manager	
$203,000								Admin Manager*
$203,000							Projects Supt.	
$183,000	Engineering Supt.	Operations Supt.			Production Supt			Financial Controller
								HR Director
$178,000	Geology Supt.		Maintenance Supt	Metallurgy Supt		Maintenance Supt.		OHS Supt.*
								External Affairs Director
								Support Services Director
$167,000							Project Engineer	
$162,000	Chief Mining Engineer							
$157,000	Resource Geologist			Chief Metallurgist		Maintenance Planner		
$152,000	Chief Surveyor	Mine General Foreman		Metallurgist	General Foreman	Mechanical General Foreman	Power & Water Supt	Environmental Supt
								Purchasing Director
								Training Coordinator
$147,000	Ore Control Geologist							
$152,000			Maintenance General Foreman			Electrical General Foreman		

Salary (AUD)	Mine Technical	Mine Operations	Mine Maintenance	Plant Technical	Plant Operations	Plant Maintenance	NPI	Administration
$137,000								Sr. Accountant
$132,000	Sr. Mine Planning Engineer							Logistics Officer
	Mining Engineer							
$127,000		Mine Shift Foreman	Maintenance Planner	Laboratory Supervisor	Plant Shift Foreman	Welder/Pipefitter		
$127,000			Light Vehicle Mechanic	Chemist		High Voltage Electrician	Power Station Electrician	Training Officer - Mine Equip
		Drill & Blast Foreman				Electrician	Power Station Mechanic	Training Officer - Fixed Plant
						Instrumentation Tech	Water Plant Mechanic	
						Mechanical Fitter		
$117,000	Mine Surveyor	Drill Operator						Safety Officer
		Shovel Operator						Paramedics
$112,000	Exploration Geologist				Plant Lead Operator			Contracts Officer
								Accountant
					Gold Room Supervisor			IT Supervisor
$107,000		Aux Equipment Operator						Env Officer - Monitoring
								Env. Officer - Compliance
$102,000	Mine Clerk	Haul Truck Operator			Crushing/ Sorting Operator	Crane Operator		Recruiting Officer
					Grinding/Leach Operator			Head of Security

Salary (AUD)	Mine Technical	Mine Operations	Mine Maintenance	Plant Technical	Plant Operations	Plant Maintenance	NPI	Administration
$96,000					Refiner			Purchasing Officer
								Business Dev. Officer
								Community Liaison Officer
								Indigenous Liaison Officer
$91,000	Pit Geology Technician	Drill & Blast Technician	Tireman	Metallurgy Clerk	Grinding/Leach Technician	Greaser	Power Station Operator	IT Technician
				Lab Technician	Tailings Technician		Water Plant Operator	Database Administrator
					Gold Room Technician			
					Plant Day Gang			
$86,000		Blasting Assistant	Maintenance Shop Labor			Trades Assistant		Store Person
								Administrative Secretary
$81,000	Surveyor Helper		Fuel & Lube Technician					Administrative Assistant
	Geology Field Technician							Security Officer
								Bus Driver
$76,000				Sample Prep Technician				Accounting Clerk
								Payroll Clerk
$66,000								Receptionist

**Vista has identified these as possible needs, but they are not currently in the total manpower calculations.*

22.4.8.2 Reagents

Reagent consumption rates and costs are shown in **Table 22-24**. Consumption rates are based upon metallurgical testwork and prices are based on vendor quotes, including a delivery to site. Unit costs of reagents are provided in AUD.

Table 22-24: Process Reagents (AUD)

Reagent	Consumable Rate	Unit	Unit Cost (AUD)	Unit
Quick Lime	2,800	g/t leach feed	$370	per tonne
Sodium Cyanide	876	g/t leach feed	$2,887	per tonne
Sodium Hydroxide	40	g/t ore	$1,123	per tonne
Flocculant	40	g/t leach feed	$3,239	per tonne
Sodium Metabisulphite (SMBS)	732	g/t leach feed	$602	per tonne
Hydrochloric Acid	81	g/t ore	$723	per tonne
Lead Nitrate	100	g/t ore	$3,866	per tonne
Activated Carbon	20	g/t ore	$2,909	per tonne
Borax	150	kg/t conc.	$3,286	per tonne
Silica	150	kg/t conc.	$1,866	per tonne
Soda Ash	100	kg/t conc.	$1,966	per tonne
Potassium Nitrate	30	kg/t conc.	$4,066	per tonne

22.4.8.3 Consumables

Consumable consumption rates are based upon benchmark data and vendor information given the ores processed at the site. Costs for consumables are based upon vendor quotes including delivery to site. These costs are shown in **Table 22-25**. Unit costs of consumables are provided in AUD.

Table 22-25: Process Consumables (AUD)

Consumables	Consumable Rate	Unit	Unit Cost (AUD)	Unit
CRUSHING				
Primary Crusher mantle	131	days per set	$277,242	per mantle
Primary Crusher concaves	272	days per set	$265,861	per set
Secondary Crushers Main frame Liners	481	days per unit	$44,115	per unit
Secondary Crushers Bowl Liners	61	days per unit	$37,193	per unit
Secondary Crusher Mantle	61	days per unit	$29,997	per unit
MILLING				
Mill Balls 65mm	0.06	kg / kWh	$1,373	per tonne
Mill Liners	1.0	sets per annum / mill	$1,005,714	per set
Secondary Grinding Media	0.35	kg / kWh	$6.23	per kg

Consumables		Consumable Rate	Unit	Unit Cost (AUD)	Unit
HPGR					
Cheek plates		7,838	h/set	$242,000	per set
Tires		13,000	h/set	$1,544,871	per set
LIME SLAKER					
Mill Balls	50 mm Lime Slaking Mill	0.5	kg/t lime	$1,380	per tonne

22.4.8.4 Diesel Consumption

The primary consumer of diesel is mining, which totals 494 million liters of diesel. The total project consumption of diesel is 503 million liters.

22.4.8.5 Plant Power Consumption

The primary consumer of power is the process facility, which totals 8,086,773 MWh power. The total project consumption of power is 8,137,558 MWh.

22.5 Economic Results

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 ±25%.

Economic results are summarized in **Table 22-26**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,350:

- Mine Life: 13 years;
- Pre-Tax NPV5%: US$1,440 million, IRR: 30.4%;
- After-tax NPV5%: US$823 million, IRR: 23.4%;
- Payback (After-tax): 2.9 years;
- NT Royalty Paid: US$473 million;
- Australian Income Taxes Paid: US$553 million; and
- Cash costs (including JAAC Royalty): US$645.14/oz-Au.

Table 22-26: Technical-Economic Results (US$000s)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Gold Sales			
Gold Produced (koz)	5,305	-	-
Gold Price (US$/oz)	1,350	-	-
Gold Sales	**7,161,494**	**32.40**	**1,300**
Refining & Royalties			
Refinery Costs	(17,075)	(0.077)	(3.22)
JAAC Royalty	(71,615)	(0.324)	(13.50)
Gross Income from Mining	**7,072,805**	**31.998**	**1,333**

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Operating Costs			
Open Pit Mine	(1,330,976)	(6.02)	(251)
CIP Process Plant	(1,742,519)	(7.88)	(328)
Project Services	(35,007)	(0.16)	(6.60)
G&A	(246,285)	(1.11)	(46.43)
Operating Costs	**(3,354,787)**	**(15.18)**	**(632.40)**
Power Sales Credit	21,156	0.096	3.99
Cash Cost of Goods Sold (COGS)	**(3,422,321)**	**(15.48)**	**(645.14)**
Operating Margin	**3,739,174**	**16.92**	**704.86**
Capital Costs			
Mining	565,982		
Process Plant	437,016		
Project Services	199,787		
Project Infrastructure	29,623		
Permanent Accommodation	66		
Site Establishment & Early Works	19,532		
Management, Engineering, EPCM Services	91,779		
Pre-Production Costs	18,102		
Asset Sale	(139,631)		
Capital Costs	**1,222,257**		
Pre-Tax Cash Flow	2,511,917		
$NPV_{5\%}$	1,440,469		
IRR (%)	30.4%		
After-tax Cash Flow	1,439,863		
$NPV_{5\%}$	823,125		
IRR (%)	23.4%		
After-tax Payback (years)	2.9		

Cash costs for the Project are presented in **Table 22-27**.

Table 22-27: All-In Sustaining Costs (US$/oz)

Period	Cash Cost	Sustaining	AISC
First 60 Mo. Of Prod.	USD 574.71	USD 112.93	USD 687.64
LoM	USD 645.14	USD 101.07	USD 746.21

Cash costs would typically include Non-cash remuneration for site personnel and AISC would include corporate G&A (including share-based remuneration).

For cash costs, non-cash remuneration could be defined to include share-based comp for site personnel. We have not included the model's $780k per year. This expense has no impact on cashflows that generate NPV and IRR and is only about $1.50/oz (after tax) LOM. This is not a material deviation from the World Gold Counsel's (WGCs) definition.

As to AISC, the model taxes an accrual-based tax deduction of $1.2 million per year but does not deduct such expense from cashflows, so there in no impact on NPV or IRR except to the extent this expense reduces NT royalty and Commonwealth income taxes. This would equate to about $2/oz, which is not material.

Table 22-28: Annual Cash Flow

Description	Qty	Unit Cost	units	Total or Avg	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
					PPD	<<>>Start of Production												>>Start HLP	<<>>Start Closure							
					1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
					-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Technical-Economic Results																										
Mill Feed																										
RoM Ore Milled			kt	207,687	0	0	12,461	17,750	17,799	17,750	17,750	17,750	17,799	15,129	17,750	17,750	17,799	17,750	2,451	0	0	0	0	0	0	0
HLP Material Milled			kt	13,354	0	0	0	0	0	0	0	0	0	0	0	0	0	0	13,354	0	0	0	0	0	0	0
RoM through Sorter			kt	221,041	0	0	12,461	17,750	17,799	17,750	17,750	17,750	17,799	15,129	17,750	17,750	17,799	17,750	15,805	0	0	0	0	0	0	0
RoM through CIL			kt	200,273	0	0	11,215	15,975	16,019	15,975	15,975	15,975	16,019	13,616	15,975	15,975	16,019	15,975	15,560	0	0	0	0	0	0	0
Gold																										
RoM Gold			koz	5,543	0	0	463	476	585	393	620	440	294	293	392	521	665	366	33	0	0	0	0	0	0	0
HLP Gold			koz	232	0.0	0.0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	231.80	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Gold to Mill			koz	5,775	0	0	463	476	585	393	620	440	294	293	392	521	665	366	265	0	0	0	0	0	0	0
Gold Recovery			%	91.9%	0.0%	0.0%	92.8%	92.0%	92.5%	91.5%	92.5%	91.8%	89.9%	90.8%	91.4%	92.2%	92.8%	91.2%	89.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recovered Gold			koz	5,305	0	0	430	438	541	360	574	404	254	266	358	481	618	334	237	0	0	0	0	0	0	0
Gold Sales																										
Gold Price				$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350
Gold Sales			USD000s	7,161,494	0	0	580,516	591,042	730,131	485,750	775,059	545,826	357,007	359,086	483,534	648,737	833,952	450,299	320,555	0	0	0	0	0	0	0
Cash Cost Of Goods & Services (COGS)																										
Perth Mint		USD/t mill																								
Refining Fee		($0.018)	USD000s	-3,979	0	0	-323	-328	-406	-270	-431	-303	-198	-199	-269	-360	-463	-250	-178	0	0	0	0	0	0	0
Gold Retention		($0.032)	USD000s	-7,161	0	0	-581	-591	-730	-486	-775	-546	-357	-359	-484	-649	-834	-450	-321	0	0	0	0	0	0	0
Purchase Discount/Gold		($0.012)	USD000s	-2,652	0	0	-215	-219	-270	-180	-287	-202	-132	-133	-179	-240	-309	-167	-119	0	0	0	0	0	0	0
Assay Fee		($0.003)	USD000s	-761	0	0	-49	-59	-59	-59	-59	-59	-59	-59	-59	-59	-59	-59	-59	0	0	0	0	0	0	0
Environmental Fee		($0.007)	USD000s	-1,456	0	0	-118	-120	-148	-99	-158	-111	-73	-73	-98	-132	-170	-92	-65	0	0	0	0	0	0	0
Freight & Insurance		($0.005)	USD000s	-1,066	0	0	-86	-88	-109	-72	-115	-81	-53	-53	-72	-97	-124	-67	-48	0	0	0	0	0	0	0
Refinery Costs		($0.077)	USD000s	-17,075	0	0	-1,372	-1,406	-1,723	-1,166	-1,825	-1,303	-873	-877	-1,161	-1,537	-1,959	-1,085	-790	0	0	0	0	0	0	0
Net Smelter Return		$32.322	USD000s	7,144,420	0	0	579,145	589,636	728,408	484,584	773,235	544,523	356,134	358,209	482,373	647,200	831,993	449,214	319,765	0	0	0	0	0	0	0
Jawoyn Royalty (Net Proceeds)		($0.324)	USD000s	-71,615	0	0	-5,806	-5,910	-7,301	-4,858	-7,751	-5,458	-3,570	-3,591	-4,835	-6,487	-8,340	-4,503	-3,206	0	0	0	0	0	0	0
Gross Income		($31.998)	USD000s	7,072,805	0	0	573,339	583,726	721,107	479,727	765,484	539,065	352,564	354,618	477,537	640,713	823,653	444,711	316,560	0	0	0	0	0	0	0
Operating Costs		USD/t mill																								
Open Pit Mine-Owner	1.00	$6.02	USD000s	-1,330,976	0	0	-50,958	-98,595	-93,546	-141,637	-145,060	-152,812	-140,541	-136,977	-139,542	-131,290	-72,427	-18,012	-9,578	0	0	0	0	0	0	0
CIP Process Plant	1.00	$0.00	USD000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Labor	1.00	$0.79	USD000s	-174,985	0	0	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	-13,460	0	0	0	0	0	0	0
3100-Crush/Screen/Stockpile	1.00	$0.18	USD000s	-40,426	0	0	-2,426	-3,455	-3,464	-3,455	-3,455	-3,455	-3,464	-2,945	-3,455	-3,455	-3,464	-3,455	-477	0	0	0	0	0	0	0
3200-Reclaim & HPGR	1.00	$0.44	USD000s	-96,902	0	0	-5,463	-7,781	-7,803	-7,781	-7,781	-7,781	-7,803	-6,632	-7,781	-7,781	-7,803	-7,781	-6,929	0	0	0	0	0	0	0
3300-Classification & Grinding	1.00	$3.14	USD000s	-693,910	0	0	-38,859	-55,351	-55,502	-55,351	-55,351	-55,351	-55,502	-47,177	-55,351	-55,351	-55,502	-55,351	-53,913	0	0	0	0	0	0	0
3400-Pre-Leach Thick/Aeration	1.00	$0.13	USD000s	-28,863	0	0	-1,616	-2,302	-2,309	-2,302	-2,302	-2,302	-2,309	-1,962	-2,302	-2,302	-2,309	-2,302	-2,243	0	0	0	0	0	0	0
3500-Desorption, Gold Room	1.00	$0.02	USD000s	-4,087	0	0	-331	-337	-417	-277	-442	-312	-204	-205	-276	-370	-476	-257	-183	0	0	0	0	0	0	0
3600-Detox & Tailings Pump	1.00	$0.06	USD000s	-12,856	0	0	-720	-1,025	-1,028	-1,025	-1,025	-1,025	-1,028	-874	-1,025	-1,025	-1,028	-1,025	-999	0	0	0	0	0	0	0
3700-Reagents	1.00	$2.96	USD000s	-659,075	0	0	-36,912	-52,572	-52,719	-52,570	-52,576	-52,571	-52,712	-44,806	-52,570	-52,573	-52,721	-52,570	-51,202	0	0	0	0	0	0	0
3800-Plant Services	1.00	$0.04	USD000s	-8,463	0	0	-474	-675	-677	-675	-675	-675	-677	-575	-675	-675	-677	-675	-658	0	0	0	0	0	0	0
Mining, Infrastructure & Misc	1.00	$0.06	USD000s	-12,494	0	0	-721	-1,027	-1,030	-1,027	-1,027	-1,027	-1,030	-876	-1,027	-1,027	-1,030	-1,027	-615	0	0	0	0	0	0	0
General Consumables	1.00	$0.01	USD000s	-2,327	0	0	-151	-181	-181	-181	-181	-181	-181	-181	-181	-181	-181	-181	-181	0	0	0	0	0	0	0
Plant Mobile Equipment	1.00	$0.01	USD000s	-1,793	0	0	-116	-140	-140	-140	-140	-140	-140	-140	-140	-140	-140	-140	-140	0	0	0	0	0	0	0
Plant Gas Consumption	1.00	$0.03	USD000s	-6,339	0	0	-412	-494	-494	-494	-494	-494	-494	-494	-494	-494	-494	-494	-494	0	0	0	0	0	0	0
CIP Process Plant		$7.88	USD000s	-1,742,519	0	0	-101,662	-138,802	-139,224	-138,740	-138,910	-138,776	-139,005	-120,328	-138,739	-138,836	-139,285	-138,719	-131,494	0	0	0	0	0	0	0
Project Services	1.00	$0.16	USD000s	-35,007	0	0	-2,084	-2,871	-2,721	-2,537	-2,264	-2,552	-2,387	-2,919	-1,736	-2,917	-3,180	-1,906	-1,448	-1,165	-1,166	-1,154	0	0	0	0
G&A	1.00	$1.11	USD000s	-246,285	0	0	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-19,098	-15,973	-325	-325	-325	-163	0	0	0
Operating Costs		$15.18	USD000s	-3,354,787	0	0	-173,801	-259,366	-254,590	-302,012	-305,333	-313,237	-301,031	-279,322	-299,114	-292,141	-233,990	-177,736	-158,492	-1,490	-1,491	-1,479	-163	0	0	0
Power Sales Credit	(0.08)	$0.0366	USD000s	21,156	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2,664	5,285	5,300	5,285	2,621	0	0	0
Cash COGS			USD000s	-3,350,706	0	0	-175,173	-260,771	-256,312	-303,178	-307,158	-314,540	-301,904	-280,199	-300,275	-293,678	-235,949	-178,821	-156,617	3,795	3,809	3,806	2,458	0	0	0
Cash Cost per Ounce		$566.35	$/oz	$645.90	$0.00	$0.00	$417.68	$605.92	$484.23	$852.85	$545.33	$788.24	$1,151.83	$1,063.62	$848.61	$621.43	$392.26	$546.35	$680.98	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
Operating Margin	0.0957	-	USD000s	3,739,174	0	0	399,538	324,360	466,518	177,715	460,151	225,827	51,533	75,296	178,424	348,572	589,663	266,976	160,732	3,795	3,809	3,806	2,458	0	0	0
Capital Costs																										
2000-Open Pit Mine																										
2100-Capitalized Mine Operating																										
2105-Construction			USD000s	68,430	0	1,631	2,060	6,612	7,062	4,293	3,187	4,703	4,250	10,776	0	11,357	11,258	1,240	0	0	0	0	0	0	0	0
2110-Pre-Production			USD000s	19,489	0	19,489	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2115-Reclamation			USD000s	30,575	0	0	0	0	0	0	0	0	0	0	0	0	0	0	7,729	22,843	0	0	0	3	0	0
2900-Closure			USD000s	138,232	0	124	7,415	3,197	3,197	3,197	3,197	3,197	3,197	3,197	3,197	3,328	13,288	7,549	54,467	25,017	635	199	636	0	0	0
2200-Mine Production Equipment			USD000s	296,827	0	93,745	25,939	69,489	60,118	8,809	8,817	6,114	4,629	14,237	1,966	1,966	0	0	0	0	0	0	0	0	0	0
2300-Mine Support Facilities			USD000s	10,536	6,693	2,855	0	104	104	104	104	104	104	104	104	104	52	0	0	0	0	0	0	0	0	0
2400-Mine Support Services			USD000s	1,434	1,188	246	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2500-Mine Dewatering/Drainage			USD000s	1,459	532	455	141	0	148	0	183	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4300-Communications			USD000s	746	3	744	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4600-Plant Mobile Equipment			USD000s	2,828	2,827	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Open Pit Mine (less prestrip/closure)			USD000s	312,830	11,243	98,044	26,080	69,593	60,370	8,913	9,104	6,218	4,733	14,341	2,070	2,070	52	0	0	0	0	0	0	0	0	0
3000-CIP Process Plant																										
3000-Process Plant			USD000s	437,016	82,308	335,458	0	2,026	2,026	2,026	2,026	2,026	2,026	2,026	2,026	2,026	1,013	0	0	0	0	0	0	0	0	0
4800-Fuel Storage & Distribution			USD000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CIP Process Plant			USD000s	437,016	82,308	335,458	0	2,026	2,026	2,026	2,026	2,026	2,026	2,026	2,026	2,026	1,013	0	0	0	0	0	0	0	0	0
4400-Tailings Dam																										
4410-TSF 1			USD000s	15,268	3,444	3,174	1,910	0	0	1,196	0	1,266	1,196	0	0	1,196	1,187	700	0	0	0	0	0	0	0	0
4420-TSF 2			USD000s	69,252	0	0	0	30,147	24,561	1,186	967	3,104	1,037	2,153	0	2,153	2,153	1,792	0	0	0	0	0	0	0	0
Tailings Dam			USD000s	84,521	3,444	3,174	1,910	30,147	24,561	2,382	967	4,370	2,233	2,153	0	3,349	3,340	2,492	0	0	0	0	0	0	0	0
4200-Power Supply																										
4200-Power Supply			USD000s	91,137	316	90,821	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Power Supply			USD000s	91,137	316	90,821	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4100-Water Dist. & Water Treat Plant																										
4110-Water Treatment Plant			USD000s	15,045	0	15,045	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4120-Raw Water Dam			USD000s	5,224	9	5,215	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Water Dist & Treat			USD000s	20,269	9	20,260	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner's Costs																										
4500-Waste Disposal			USD000s	287	287	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5000-Project Infrastructure			USD000s	29,623	28,026	1,597	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
6000-Permanent Accommodation			USD000s	66	0	66	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
7000-Site Establishment & Early Works			USD000s	19,532	19,532	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
8100 EPCM Services			USD000s	46,253	12,897	33,356	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
8000-Mgmt, Engineering, Owner's Costs			USD000s	45,526	19,936	25,590	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
9000-Pre-Production Costs			USD000s	18,102	6,418	11,685	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Contingency			USD000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Management Provision			USD000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner's Costs			USD000s	159,390	87,097	72,293	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CAPITAL COST		0	USD000s	-1,361,888	-184,418	-641,294	-37,465	-111,574	-97,216	-20,811	-18,481	-20,514	-16,438	-32,493	-7,292	-22,129	-28,962	-11,281	-62,196	-47,861	-635	-199	-636	-3	0	0
			USD000s		-184,418	-825,712	-863,178	-974,751	-1,071,967	-1,092,778	-1,111,259	-1,131,773	-1,148,211	-1,180,704	-1,187,996	-1,210,125	-1,239,078	-1,250,358	-1,312,555	-1,360,415	-1,361,050	-1,361,250	-1,361,886	-1,361,888	-1,361,888	-1,361,888
10000-Asset Sale																										
10000-Mine			USD000s	52,926	0	0	0	0	0	0	0	0	0	0	0	17,209	0	0	0	30,791	0	0	4,927	0	0	0
10000-Process Plant			USD000s	18,352	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	18,352	0	0	0	0	0	0
10000-Power Plant (Sold to 3rd Party)			USD000s	68,352	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	68,352	0	0	0
10000-Asset Sale			USD000s	139,631	0	0	0	0	0	0	0	0	0	0	0	17,209	0	0	0	49,143	0	0	73,279	0	0	0
CAPITAL COST WITH ASSET CREDIT		0	USD000s	-1,222,257	-184,418	-641,294	-37,465	-111,574	-97,216	-20,811	-18,481	-20,514	-16,438	-32,493	-7,292	-4,920	-28,962	-11,281	-62,196	1,282	-635	-199	72,643	-3	0	0
			USD000s		-184,418	-825,712	-863,178	-974,751	-1,071,967	-1,092,778	-1,111,259	-1,131,773	-1,148,211	-1,180,704	-1,187,996	-1,192,916	-1,221,868	-1,233,149	-1,295,345	-1,294,063	-1,294,698	-1,294,898	-1,222,255	-1,222,257	-1,222,257	-1,222,257
Working Capital																										
Project Working Capital			USD000s	0	0	-15,000	15,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner's Closeout			USD000s	-5,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-5,000	0	0	0
Change in Working Capital			USD000s	-5,000	8,436	-14,847	3,206	10,417	-199	2,489	-3,066	3,928	1,826	-1,048	-848	-2,266	-5,804	1,716	609	-3,697	-765	5	-5,092	0	-0	0
Pre-Tax Cash Flow			$000s	2,511,917	-175,983	-656,142	365,279	223,204	369,102	159,393	438,604	209,241	36,921	41,756	170,284	341,386	554,908	257,411	99,145	1,380	2,409	3,612	70,010	-3	-0	0
Cumulative		FE Override	$000s	-	-175,983	-832,124	-466,846	-243,642	125,460	284,854	723,458	932,699	969,620	1,011,375	1,181,659	1,523,045	2,077,953	2,335,364	2,434,508	2,435,889	2,438,298	2,441,910	2,511,919	2,511,917	2,511,917	2,511,917
Present Value	5%	$1,440	$000s	1,511,856	-175,983	-624,897	331,318	192,812	303,661	124,889	327,293	148,703	24,990	26,916	104,539	199,601	308,993	136,511	50,075	664	1,104	1,576	29,090	-1	-0	0
Cumulative			$000s	-	-175,983	-800,880	-469,561	-276,749	26,912	151,801	479,094	627,796	652,787	679,703	784,243	983,844	1,292,838	1,429,348	1,479,423	1,480,087	1,481,190	1,482,766	1,511,857	1,511,856	1,511,856	1,511,856
IRR			$000s	30.4%																						
Project Financials (Calculation provided by Vista)																										
Statement of Operations					-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Gold Sales			USD000s	$7,161,494	$0	$0	$580,516	$591,042	$730,131	$485,750	$775,059	$545,826	$357,007	$359,086	$483,534	$648,737	$833,952	$450,299	$320,555	$0	$0	$0	$0	0	0	0
Development costs (currently included in DD&A)			USD000s	$0																						
Cash COGS			USD000s	($3,350,706)	$0	$0	($175,173)	($260,771)	($256,312)	($303,178)	($307,158)	($314,540)	($301,904)	($280,199)	($300,275)	($293,678)	($235,949)	($178,821)	($156,617)	$3,795	$3,809	$3,806	$2,458	0	0	0
Non-cash prestripping (pro-rated on ounce production)			USD000s	($87,919)	$0	$0	($7,127)	($7,256)	($8,964)	($5,963)	($9,515)	($6,701)	($4,383)	($4,408)	($5,936)	($7,964)	($10,238)	($5,528)	($3,935)	$0	$0	$0	$0	$0	$0	$0
Non-cash COGS (DD&A)			USD000s	-1,063,728			-64,889	-77,908	-92,259	-97,840	-99,477	-99,092	-86,641	-74,812	-68,830	-60,475	-49,404	-51,456	-51,025	-51,025	-38,593	0	0	0	0	0
Non-cash COGS (ARO provision)			USD000s	-168,683	0	0	-10,121	-14,416	-14,456	-14,416	-14,416	-14,416	-14,456	-12,288	-14,416	-14,416	-14,456	-14,416	-1,991	0	0	0	0	0	0	0
Jawoyn Royalty			USD000s	-71,615	0	0	-5,806	-5,910	-7,301	-4,858	-7,751	-5,458	-3,570	-3,591	-4,835	-6,487	-8,340	-4,503	-3,206	0	0	0	0	0	0	0
Transfer pricing allocations		100,000	USD000s	-14,400			-1,200	-1,200	-1,200	-1,200	-1,200	-1,200	-1,200	-1,200	-1,200	-1,200	-1,200	-1,200								
Stock-based compensation		65,000	USD000s	-9,360			-780	-780	-780	-780	-780	-780	-780	-780	-780	-780	-780	-780								
			USD000s	0																						
Gain (loss) on asset sales			USD000s	50,716			0	0	0	0	0	0	0	0	0	17,209	0	0	0	-29,384	0	0	62,891	0	0	0
Interest expense (see Debt Roll Forward)			USD000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Income before income & mining taxes			USD000s	2,445,800	0	0	315,421	222,799	348,859	57,515	334,763	103,638	-55,927	-18,192	87,261	280,946	513,585	193,596	103,781	-76,615	-34,784	3,806	65,349	0	0	0
NT royalty (see NT Royalty)			USD000s	-473,147	0	0	-38,025	-40,005	-62,957	-1,702	-58,130	-17,528	-8,925	-8,977	-12,088	-44,884	-112,828	-48,801	-18,299	0	0	0	0	0	0	0
Income (loss) Before Income Tax			USD000s	1,972,652	0	0	277,396	182,795	285,902	55,813	276,633	86,110	-64,852	-27,169	75,173	236,061	400,757	144,794	85,482	-76,615	-34,784	3,806	65,349	0	0	0
Australian Income tax (see Tax Loss Carryf		30.0%	USD000s	-587,660	0	0	-26,225	-32,277	-69,788	-11,496	-86,562	-33,023	0	0	-17,508	-78,160	-126,233	-54,579	-37,595	0	0		-14,213	0	0	0
Net Tax benefit of restructuring	1		USD000s	34,608			11,191	8,375	7,104	4,700	3,237	0	0	0	0	0	0	0	0	0	0	0				
Net Income (loss) after tax			USD000s	$1,419,600	$0	$0	$262,362	$158,892	$223,218	$49,017	$193,308	$53,087	($64,852)	($27,169)	$57,665	$157,901	$274,524	$90,215	$47,887	($76,615)	($34,784)	$3,806	$51,137	$0	$0	$0
Effective tax rate				42.0%			16.8%	32.4%	38.1%	22.9%	43.2%	48.8%	-16.0%	-49.3%	33.9%	43.8%	46.5%	53.4%	53.9%	0.0%	0.0%					
Tax Loss Carryforwards					-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Historic costs incurred prior to this template			USD000s	-87,833	-87,833																					
Losses incurred to start of construction			USD000s	0																						
Tax losses (accounting losses)			USD000s	-203,420	0	0	0	0	0	0	0	0	-64,852	-27,169	0	0	0	0	0	-76,615	-34,784	0	0	0	0	0
Cumulative accounting income (loss)			USD000s	-	-87,833	-87,833	189,563	372,358	658,260	714,073	990,706	1,076,816	1,011,964	984,795	1,059,968	1,296,029	1,696,786	1,841,580	1,927,062	1,850,446	1,815,663	1,819,470	1,884,819	1,884,819	1,884,819	1,884,819
Timing differences (SEE SUMMARY BELOW)			USD000s	74,046			-102,146	-75,203	-53,276	-17,492	11,907	23,965	24,393	15,707	35,109	24,473	20,021	37,136	39,835	51,025	38,593	0	0	0	0	0
Cumulative Income for income tax purposes			USD000s	-			87,417	196,009	427,634	465,955	754,495	864,570	824,112	812,649	922,931	1,183,465	1,604,243	1,786,173	1,911,490	1,885,901	1,889,710	1,893,516	1,958,866	1,958,866	1,958,866	1,958,866
Taxable base			USD000s	1,958,866			87,417	107,591	232,626	38,321	288,540	110,075	0	0	58,360	260,534	420,778	181,930	125,317	0	0	0	47,375	0	0	0
			USD000s	488,934			30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%	30.0%
Taxable Period income (Loss)			USD000s	0	0	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Timing Differences:																										
Pre-strip acctg			USD000s	87,919	0	0	7,127	7,256	8,964	5,963	9,515	6,701	4,383	4,408	5,936	7,964	10,238	5,528	3,935	0	0					
pre-strip tax (period expense)			USD000s	-87,919	-	-21,121	(2,060)	(6,612)	(7,062)	(4,293)	(3,187)	(4,703)	(4,250)	(10,776)	-	(11,357)	(11,258)	(1,240)	-	-	-	-	-	-	-	-
net change			USD000s	0	0	-21,121	5,067	645	1,902	1,670	6,328	1,998	133	-6,368	5,936	-3,393	-1,020	4,288	3,935	0	0					
Dep'n - acctg			USD000s	1,063,728			64,889	77,908	92,259	97,840	99,477	99,092	86,641	74,812	68,830	60,475	49,404	51,456	51,025	51,025	38,593	0	0	0	0	0
Dep'n - tax			USD000s	-989,682			-150,982	-153,756	-147,437	-117,003	-93,898	-77,125	-62,381	-52,738	-39,657	-32,609	-28,363	-18,608	-15,126	0	0	0	0	0	0	0
net change			USD000s	74,046			-86,092	-75,848	-55,178	-19,162	5,579	21,968	24,261	22,074	29,173	27,866	21,041	32,848	35,900	51,025	38,593	0	0	0	0	0
Total Timing differences			USD000s	74,046	0	-21,121	-81,025	-75,203	-53,276	-17,492	11,907	23,965	24,393	15,707	35,109	24,473	20,021	37,136	39,835	51,025	38,593	0	0	0	0	0
Statement of Cash Flows					-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Net Income (Loss) after tax			USD000s	1,419,600	0	0	262,362	158,892	223,218	49,017	193,308	53,087	-64,852	-27,169	57,665	157,901	274,524	90,215	47,887	-76,615	-34,784	3,806	51,137	0	0	0
Add back non-cash and non-ops income/expenses:			USD000s																							
Transfer pricing allocations			USD000s	14,400	0	0	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	0	0	0	0	0	0	0	0
Stock-based compensation			USD000s	9,360	0	0	780	780	780	780	780	780	780	780	780	780	780	780	0	0	0	0	0	0	0	0
Gain/loss on asset sales				-50,716			0	0	0	0	0	0	0	0	0	-17,209	0	0	0	29,384	0	0	-62,891	0	0	0
NT Royalty - P&L Basis			USD000s	473,147	0	0	38,025	40,005	62,957	1,702	58,130	17,528	8,925	8,977	12,088	44,884	112,828	48,801	18,299	0	0	0	0	0	0	0
- Cash flow basis (no change when annualized)			USD000s	-473,147	0	0	0	0	0	-15,800	-72,228	-31,626	-23,024	-23,076	-26,187	-58,963	-126,926	-62,899	-32,397	0	0	0	0	0	0	0
Aus Income tax - P&L Basis			USD000s	553,052	0	0	15,034	23,903	62,683	6,796	83,325	33,023	0	0	17,508	78,160	126,233	54,579	37,595	0	0	0	14,213	0	0	0
- Cash flow basis (no change when annualized, ov			USD000s	-553,052	0	0	-15,034	-23,903	-62,683	-6,796	-83,325	-33,023	0	0	-17,508	-78,160	-126,233	-54,579	-37,595	0	0	0	-14,213	0	0	0
Non-cash COGS (DD&A) tax purposes = acctg			USD000s	1,063,728	0	0	64,889	77,908	92,259	97,840	99,477	99,092	86,641	74,812	68,830	60,475	49,404	51,456	51,025	51,025	38,593	0	0	0	0	0
prestripping accounting			USD000s	87,919	0	0	7,127	7,256	8,964	5,963	9,515	6,701	4,383	4,408	5,936	7,964	10,238	5,528	3,935	0	0	0				
prestripping cash flow			USD000s	-66,799			-2,060	-6,612	-7,062	-4,293	-3,187	-4,703	-4,250	-10,776	0	-11,357	-11,258	-1,240	0	0	0	0	0	0	0	0
Non-cash COGS (ARO provision)			USD000s	168,683	0	0	10,121	14,416	14,456	14,416	14,416	14,416	14,456	12,288	14,416	14,416	14,456	14,416	1,991	0	0	0	0	0	0	0
Reclamation spending			USD000s	-168,683	0		-7,415	-3,197	-3,197	-3,197	-3,197	-3,197	-3,197	-3,197	-3,197	-3,328	-13,288	-7,549	-62,196	-47,861	-635	-199	-636	-3	0	0
Changes in working capital			USD000s	-5,000	8,436	-14,847	3,206	10,417	-199	2,489	-3,066	3,928	1,826	-1,048	-848	-2,266	-5,804	1,716	609	-3,697	-765	5	-5,092	0	-0	0
Cash (used in) provided by operations			USD000s	2,472,493	8,436	-14,847	378,234	301,067	393,376	150,118	295,149	157,206	22,889	37,200	130,684	194,478	306,153	142,425	29,152	-47,762	2,409	3,612	-17,482	-3	-0	0
Initial capex (PP&E, excl stripping)			USD000s	-825,712	-184,418	-641,294	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0			
Sustaining capex			USD000s	-300,695	0	0	-27,990	-101,766	-86,957	-13,321	-12,097	-12,614	-8,991	-18,520	-4,096	-7,445	-4,405	-2,492	0	0	0	0	0		0	0
Salvage values			USD000s	139,631	0	0	0	0	0	0	0	0	0	0	0	17,209	0	0	0	49,143	0	0	73,279	0	0	0
Cash used in investing, net			USD000s	-986,776	-184,418	-641,294	-27,990	-101,766	-86,957	-13,321	-12,097	-12,614	-8,991	-18,520	-4,096	9,765	-4,405	-2,492	0	49,143	0	0	73,279	0	0	0
Operations net of Capex			USD000s	1,485,717	-175,983	-656,142	350,244	199,301	306,419	136,797	283,052	144,592	13,898	18,680	126,589	204,243	301,748	139,933	29,152	1,380	2,409	3,612	55,797	-3	-0	0
Timing of interest payments			USD000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Operations net of Capex and interest payments, bef			USD000s	1,485,717	-175,983	-656,142	350,244	199,301	306,419	136,797	283,052	144,592	13,898	18,680	126,589	204,243	301,748	139,933	29,152	1,380	2,409	3,612	55,797	-3	-0	0
Cumulative			USD000s	-	-175,983	-832,124	-481,880	-282,579	23,840	160,637	443,689	588,281	602,178	620,858	747,446	951,689	1,253,437	1,393,369	1,422,522	1,423,902	1,426,311	1,429,923	1,485,720	1,485,717	1,485,717	1,485,717
Present Value (FE override in 195 below)			USD000s																							
NPV	5%	$ 823,125	USD000s	-																						
Payback (yrs)		2.9					1.0	1.0	0.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0					
IRR after tax at: 2019-10-04		23.4%																								
LoM Wt Avg Au/oz of:		$1,350																								

22.5.1 Taxes, Royalties

Taxes, royalties, and working capital were incorporated to the economic model by Vista.

22.5.1.1 Royalties

NORTHERN TERRITORY ROYALTY

Under the NT Mineral Royalty Act (November 2014) (the "MRA"), royalties are based on the net value of production from a mine when annual gross production revenue of a production unit exceeds AUD500,000, irrespective of the nature of the land holding.

The royalty payable under the MRA is the greater of:

(a) 20 per cent of the net value, less $10 000, or

(b) the percentage of the gross production revenue applying to the royalty year as follows:

- (i) 1% for the royalty payer's first royalty year that begins on or after July 1, 2019;
- (ii) 2% for the royalty year that follows the royalty year mentioned in subparagraph (i); or
- (iii) 2.5% for each royalty year that follows the royalty year mentioned in subparagraph (ii).

The Northern Territory Government imposes a net value-based royalty, much like an income tax, on mine production (the "NT Royalty"). The MRA codifies the calculation of the Northern Territory Royalty; however, determination and collection of the NT Royalties is not fully a matter of public record. Some mines appear to be subject to legacy customized arrangements with the NT Government, which seem to offer relief with respect to the amount and/or timing of royalty payments; and other mines appear to pay no royalty. Such agreements are confidential and each is subject to a formal application process. The NT Royalty calculated for the Project cashflows is based on the MRA rules, together with reasonable assumptions about the nature of relief that appears to be available to new mines.

Net value of production for the purposes of calculating the royalty is based the formula:

$$\mathbf{NV = GR - (OC + CRD + EEE + AD)}$$

where:

NV is the net value from a production unit in a royalty year;
GR is the gross realization from the production unit in the royalty year;
OC is the operating costs of the production unit for the royalty year;
CRD is the capital recognition deduction;
EEE is the eligible exploration expenditure, if any; and
AD is any additional deduction.

22.5.1.2 Other Royalties

For rent of the surface rights from the current mining licenses, including the mining license on which the Batman deposit is located, the JAAC is entitled to an annual amount equal to 1% of the gross value of production with a minimum annual payment of AUD50,000.

There is also a royalty of 5% based on the gross value of any gold or other metals that may be commercially extracted from certain mineral concessions (the Denehurst Royalty). The Denehurst Royalty would not apply to presently identified mineralized zone at Mt Todd.

22.5.1.3 Taxes

AUSTRALIAN COMMONWEALTH INCOME TAX

The applicable corporate income tax rate in Australia is 30%.

Taxable income is based on assessable income less allowable deductions. Assessable income generally includes gross income from the sale of goods, the provision of services, dividends, interest, royalties and rent. Assessable income may also include capital gains after offsetting capital losses. Normal business expenses are deductible.

Tax losses may be utilized and carried forward indefinitely to offset against future assessable income provided a “continuity of ownership” (more than 50% of voting, dividend and capital rights) or a “same business” test is satisfied.

Thin capitalization provisions can limit the deductibility of interest and other “debt deductions” in certain cases. In general, a deduction will be partly disallowed if the company’s debt exceeds three times its equity.

Transfer pricing rules apply to international transactions/dealings between separate legal entities. Covered cross-border transactions include those involving tangible or intangible property, the provision of services and financing. There are several generally accepted transfer pricing methods available in Australia.

Consolidation allows wholly owned corporate groups to operate as a single entity for income tax purposes.

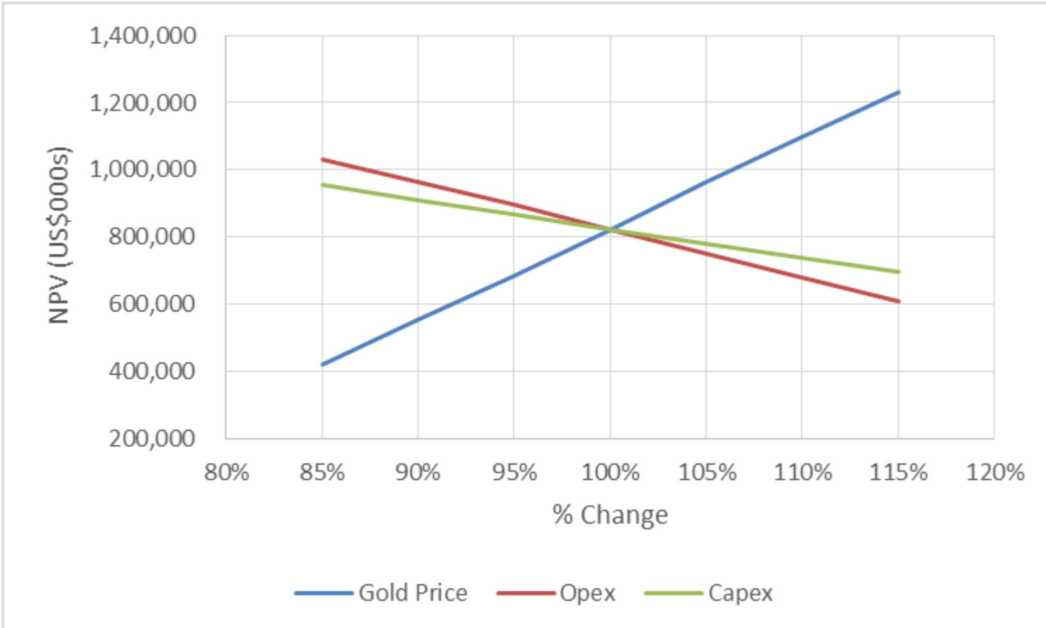
Australia operates a full imputation system for the avoidance of double taxation of dividends. Under this system, the payment of company tax is imputed to shareholders so that shareholders are relieved of their tax liability to the extent profits have been taxed at the corporate level. Dividends paid out of profits on which corporate tax has been paid are said to be “franked” and generally entitle shareholders to an offset for the corporate tax paid.

22.5.2 Sensitivity

Project sensitivities are summarized in **Table 22-29**, **Table 22-30**, and **Table 22-31**; sensitivities are shown graphically in **Figure 22-1**. As seen, the Project is most sensitive to gold production and gold price. Sensitivity on operating and capital cost is closely matched, with the Project being only slightly more sensitive to operating costs.

Table 22-29: Project Sensitivity

Parameter	85%	90%	95%	Base	105%	110%	115%
Gold Price	419,574	554,147	683,823	823,125	962,354	1,096,323	1,230,036
Opex	1,030,506	963,805	896,328	823,125	748,383	679,009	609,361
Capex	954,399	910,641	866,883	823,125	779,904	736,755	695,643



Source: JDS Energy & Mining, September 9, 2019

Figure 22-1: Project Sensitivity

Table 22-30: Base Case Sensitivity to Gold Price versus Foreign Exchange Rate (US$:AUD)

	GOLD PRICE (US$/oz-Au)									
	$1,200		$1,300		$1,350		$1,400		$1,500	
FEX Rate	IRR	$NPV_{5\%}$	IRR	$NPV_{5\%}$	IRR	$NPV_{5\%}$	IRR	$NPV_{5\%}$	IRR	$NPV_{5\%}$
0.60	21.6%	687	26.3%	895	28.4%	994	30.5%	1094	34.7%	1,296
0.65	19.2%	604	23.7%	807	25.8%	911	27.9%	1011	32.0%	1,209
0.70	16.9%	525	21.2%	718	**23.4%**	**823**	25.4%	928	29.4%	1,126
0.75	14.7%	440	18.9%	636	20.9%	734	23.1%	839	27.0%	1,043
0.80	12.6%	355	16.8%	557	18.8%	652	20.7%	750	24.7%	954

Base Case uses a foreign exchange rate of US$0.70:AUD1.00 and a gold price of $1,350/oz-Au.

Table 22-31: Base Case Sensitivity to Gold Prices versus NPV Discount Rate

	GOLD PRICE (US$/oz-Au)						
Discount Rate	$ 1,100	$ 1,200	$ 1,300	$ 1,350	$ 1,400	$ 1,500	$ 1,600
5%	325	525	718	**823**	928	1,126	1,329
8%	160	326	485	571	656	818	983
10%	78	225	366	442	516	659	804

Foreign Exchange is held constant at US$0.70:AUD1.00

23.0 ADJACENT PROPERTIES

There are no adjacent properties that are considered relevant to this Technical Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

The Base Case is identified as a 50,000 tpd operation, as presented in this section. Another option, defined as the Alternate Case (33,000 tpd) is presented in **Section 24.7 – Alternate Case**.

24.1 Process Plant Geotechnical

Bulk earthworks for the process plant are designed to minimize the import of fill material and excavation of rock. Where fill material is required to be imported, either material from the existing RoM Pad ramp; from the existing stockpile located adjacent to the Tollis and Golf Pits; or from the WRD will be utilized. The civil basis of design took into consideration the following geotechnical information:

- Geotechnical Desktop Study Mt Todd Process Plant DFS undertaken by Coffey Geotechnics in December 2012. The study reviewed previous Soil and Rock Engineering (SRE) geotechnical data from December 1992 and April 1993 for the original Mt Todd development. The study also reviewed SRE earthworks monitoring data for construction of the original Mt Todd development. Geotechnical test pit data was also reviewed (Tetra Tech, 2013). The study focused on foundations for heavy vibrating loads including the crusher and mill as well as screening structures and ancillary plant buildings. A review of potential borrow material in close proximity to the proposed plant site suitable for structural fill and pavement construction was also included.
- Foundation Recommendations report produced by Tetra Tech in April 2012. The geotechnical test pit investigation was conducted in October 2011 at the then proposed location of the process plant. A summary of the test pit investigation and preliminary recommendations for foundation design at the site were provided.
- Technical Memorandum regarding "Results of Test Pit Excavation Program and Borrow Source Investigation, Mt Todd Project, Vista Gold Corporation, Northern Territory, Australia" from Tetra Tech dated 20 December 2012. A summary of the test pit results and potential borrow sources were provided.
- Foundation Recommendations report produced by Tetra Tech in February 2013. The report reviewed previous SRE geotechnical data from December 1992 and April 1993 for the original Mt Todd development. The report also reviewed previous Foundation Recommendations report produced by Tetra Tech in April 2012 and the previous geotechnical test pit investigation conducted in December 2012 at the proposed location of the process plant. A summary of the test pit investigation and recommendations for foundation design at the site were provided.

Further geotechnical investigation is recommended during the next design phase of the project to obtain geotechnical data in the final location of foundations for heavy vibrating loads including the crusher and mill as well as screening structures and ancillary plant buildings. The investigation also is required to confirm fill material and rock excavation requirements, as well as locating borrow sources that are closer to the planned operation that may reduce these costs.

24.2 Water Management

This section describes the overall Project water management and infrastructure considerations.

24.2.1 Site-wide Water Balance

A site-wide water balance (SWWB) was developed within the GoldSim® software platform (Version 11.1) to simulate 13 years of mine production (12 active mining years and 1 additional year processing stockpiles) at the Vista Project Site for the Base Case.

The site-wide water balance was developed to simulate site conditions in order to:

- Identify water treatment plant capacity;
- Determine equalization pond sizing; and
- Quantify make up water requirements from the RWD for process make up water, dust control, and potable/elution needs.

24.2.1.1 Site-wide Water Balance Model

WATER BALANCE MODELING

The site-wide water balance model was constructed using deterministic (known with certainty) inputs, such as pond stage-storage relationships, as well as stochastic (known, but with some uncertainty) inputs, such as rainfall. Water storage within retention ponds (RPs) was modeled using the basic formula:

Change in Storage = Inputs – Outputs

Information provided to the model and the rules by which the site features interacted are summarized below.

MODEL ELEMENTS

The site features (pits, facilities and associated RPs) represented within the model are:

- Waste Rock Dump (WRD, RP1);
- Low Grade Ore Stockpile (LGOS)
- Low Grade Ore Stockpile Retention Pond (LGRP);
- Batman Pit (BP, RP3);
- Process Plant Retention Pond (PRP);
- Heap Leach Pad (HLP);
- Raw Water Dam (RWD);
- Process Water Pond (PWP);
- Water Treatment Plant (WTP);
- Process Plant (PP);
- Dust Control;
- Tailings Storage Facility 1 (TSF 1, RP7); and
- Tailings Storage Facility 2 (TSF 2).

GENERAL ASSUMPTIONS

Interaction between site features was modeled based on the following set of guidelines:

- RP1, LGRP, RP3, PRP and the HLP report to the PWP which feeds the WTP;
- The PWP receives water only if it is not at risk for overtopping. Given this logic, overtopping events are allowed to occur at the RPs;
- Inputs to ponds included precipitation, catchment runoff (where applicable), seepage (where applicable), and groundwater inflow (where applicable);
- Outputs from ponds included evaporative loss, pumping and overtopping events (uncontrolled releases);
- All RPs report to the PWP which feeds directly to WTP. The PWP was sized to contain six days of WTP capacity (72,000m^3);
- A process plant bleed stream of 229 m^3/hr is sent to the PWP to maintain chemistry of the process circuit;
- Discharges to the Edith River were not allowed from any of the RPs.
- WTP effluent was allowed to discharge to the Edith River only during the wet season;
- The HLP was run through process at the end of the Life of Mine (LoM);
- WRD water that reported to RP1 was not allowed to be used for dust control;
- Seepage losses from the ponds are not modeled and are assumed to be zero; and
- RWD is modeled as an infinite source for site water needs due to lack of information about its catchment area and stage-storage relationship.

INITIAL CONDITIONS

- RP1, LGRP, RP3, PRP and the PWP were assigned water surface elevations based on outputs from the pre-production model, which concluded at the initiation of production with initial conditions based on real site water surface elevation observations from December 2016.
- The dust suppression tank was assumed to be full at the initiation of production.

FLOW RATES

- Process makeup water requirements throughout the LoM were 1,764 m^3/hr. This value accounts for recycle from the thickener overflow within the Process circuit.
- TSF decant flows were 1,460 m^3/hr.
- RWD process makeup water flows were 304 m^3/hr.
- Dust suppression requirements varied between 220 and 1,153 m^3/day.
- WTP capacity is 500 m^3/hr.

CLIMATOLOGICAL INPUTS

The Vista Project Site-wide water balance model was designed to reflect weather conditions as accurately as possible, given the arid tropical climate (i.e., wet, monsoon conditions with intense, short-lived events and extended hot, dry periods). Features within the climatological section of the model included:

- Stochastic precipitation inputs, such that a range of likely scenarios may be determined, thus allowing the user an understanding of dry (5-percentile), typical (mean) and wet (95-percentile) climatological effects on the site. Precipitation was entered as a statistical probability, rather than assuming rainfall on a given day and assigning a rainfall depth. The mean monthly total precipitation values (total mm per month) provided to the model are shown in **Table 24-1**.

Table 24-1: Mean Monthly Precipitation

Month	Precipitation (mm)
January	292
February	259
March	193
April	38
May	6
June	2
July	1
August	1
September	7
October	35
November	111
December	239

- Synthetic data were used to extend the precipitation period of record and identify extreme rainfall events. Project Site precipitation data were collected from 1993 to present. A correlation of the site data to a nearby Katherine gage allowed the rainfall time series to be extended to that of the Katherine gage, 138 years, thus allowing determination of extreme rainfall events that may not otherwise present within a data set spanning less than 25 years.
- Linking incidental rainfall and runoff within the Edith River using the Australian Water Balance Method. Catchment parameters within the model were adjusted to ensure optimal agreement between modeled runoff and observed Edith River flows. Measured precipitation, evaporation and Edith River flow data were used to calibrate this portion of the model.
- The model used stochastic evaporation, based on a monthly time step and calculated by the Blaney-Criddle method. The Blaney-Criddle approach recommends against use of a time step smaller than one month.

MODEL RUN

A time step of one day was selected for the site-wide water balance model. Use of stochastic inputs allowed a "Monte Carlo" analysis to be run wherein the 13-year LoM was simulated across 1,000 realizations (or equally likely weather scenarios), each incorporating the uncertainty associated with meteorological conditions and collectively providing an envelope of expected outcomes at the site. All RPs were subjected to the stochastic weather events as described in the previous section and reported to the WTP.

24.2.1.2 Results

- The site-wide water balance model results show that the Batman Pit will see water storage during the wet season and later in the life of mine. This is a result of incorporating groundwater inflows into this iteration of the SWWB model. Further investigation of potential groundwater inflows will be required to validate these groundwater inflows. Optimization of process makeup water usage and onsite reuse of stormwater could reduce the amount of water reporting to the PWP, thus, allowing for faster dewatering of the Batman Pit.
- The greatest amount of make-up water required from the RWD is quantified as 11,955 m^3/day. RWD requirements were found to be the most dependent upon TSF decant volumes.
- RP1, LGRP, PRP, and the HLP show less than a 1% percent probability of having an overtopping over the LoM[1]. LGRP storage may be optimized.

24.2.2 Wet Infrastructure

Section 18.2 – Facility 4000 Project Services discusses water supply inclusive of the water treatment plant (WTP), raw water, and potable water supply. Additional information regarding regulations, design criteria and receiving water is provided herein.

24.2.2.1 Water Treatment Plant

Flow to the Process Water Pond, a combination of decant return, runoff pond water, and pit dewatering discharge, is stored and pumped to the Water Treatment Plant (WTP). The maximum design capacity of the WTP is 500 m^3/hr. The WTP has been designed by Tetra Tech and its discharge will be returned to the Edith River for disposal, pursuant to the conditions defined by Water Discharge Licence 178-06 (WDL). During the dry season, when discharge is not allowed by the WDL, the WTP effluent will be used in the process plant for process water and around the site as dust suppression.

24.2.2.2 Water Quality Standards for Waste Water Discharge

Discharges from the site are currently regulated by Waste Discharge Licence 178-06 (WDL), issued by the Northern Territory Government on November 26, 2018. The WDL is formal approval under section 74 of the Northern Territory Water Act that authorizes and regulates the release of potential contaminants to water in the Northern Territory to ensure environmental protection objectives are met. The Mt Todd Environmental Impact Statement (GHD, 2013) indicates that after the WTP is operational, the WDL will be revised to implement 95% species protection trigger values, as defined in the Australian and New Zealand Environment and Conservation Council (ANZECC) Guidelines for Fresh and Marine Water Quality (ANZECC 2000 Guidelines). This change will be reflected in a revision to WDL 178.

[1] A typical value is given. Separate model runs provide a range of overtopping events, due to the stochastic nature of the model.

The purpose of the 95% species protection trigger value (TV) is to protect water quality in the Edith River downstream of the discharge from the WTP. The WTP will discharge effluent into Batman Creek, a tributary to the Edith. Concentrations of contaminants measure in the Edith River shall not exceed the TV during discharge events. For the contaminants of concern at the Project, the TVs are presented in **Table 24-2**.

Table 24-2: Site-specific Trigger Values, Edith River Downstream of WTP Discharge

Analyte	Unit	Trigger Value	Source
pH	s.u.	6-8	ANZECC 2000 Guidelines, Table 3.3.4, Lowland River value
Dissolved Oxygen	% Saturation	85-120	*ANZECC 2000 Guidelines, Table 3.3.4, Lowland River Value*
Conductivity	*µS/cm*	*20-250*	ANZECC 2000 Guidelines, Table 3.3.5, Upland and Lowland River Value
Magnesium	mg/L	2.5	Van Dam, et al. 2010 Environ Toxicol Chem 29(2):410-421
Sulphate	mg/L	129	Elphick et al. 2011 Environ Toxicol Chem 30(1):247-253
Aluminum	µg/L	55	ANZECC 2000 Guidelines Table 3.4.1
Cadmium	µg/L	0.2	ANZECC 2000 Guidelines Table 3.4.1
Cobalt	µg/L	90	ANZECC 2000 Guidelines p. 8.3-118
Chromium (III)	µg/L	3.3	ANZECC 2000 Guidelines p. 8.3-116
Chromium (VI)	µg/L	1.0	ANZECC 2000 Guidelines Table 3.4.1
Copper	µg/L	1.4	ANZECC 2000 Guidelines Table 3.4.1
Manganese	µg/L	1900	ANZECC 2000 Guidelines Table 3.4.1
Nickel	µg/L	11	ANZECC 2000 Guidelines Table 3.4.1
Lead	µg/L	3.4	ANZECC 2000 Guidelines Table 3.4.1
Iron	µg/L	300	ANZECC 2000 Guidelines p. 8.3-123
Mercury	µg/L	0.6	ANZECC 2000 Guidelines Table 3.4.1
Zinc	µg/L	8.0	ANZECC 2000 Guidelines Table 3.4.1

The TVs for magnesium and sulphate have been held over from previous work, and are not referenced in the ANZECC 2000 Guidelines.

To determine the allowable level of water quality constituents in the discharge of the WTP, a mass balance was performed on the Edith River system. Upstream water quality values at sampling location SW2 on the Edith River, flow in the river at sampling location SW4 downstream of the WTP discharge, and the maximum WTP effluent flow were used to calculate effluent limits at the WTP that would maintain the site-specific trigger value at site SW4. The equation used to determine the effluent limits is:

$$\boldsymbol{Q_{WTP}C_{WTP} + Q_{SW2}C_{SW2} = Q_{SW4}C_{SW4}}$$

Where:

Q_{WTP} is the WTP maximum flow rate
C_{WTP} is the allowable concentration of a given analyte in the WTP effluent
Q_{SW2} is the flow in the Edith River upstream of the WTP
C_{SW2} is the background concentration of a given analyte in the Edith River upstream of the WTP
Q_{SW4} is the flow in the Edith River downstream of the WTP
C_{SW4} is the background concentration of a given analyte in the Edith River downstream of the WTP

Field data was used to determine values for Q_{SW4}, and C_{SW2}. **Table 24-3** presents Edith River flows at field monitoring location SW4 for the wet season, when the water treatment plant will discharge to the environment.

Table 24-3: Edith River Flow at SW4 (m3/h), February 2013 – September 2017

Month	Mean	Median	5th Percentile	95th Percentile	Maximum Day	Minimum Day
January	91,374	46,807	8,990	307,729	740,452	4,942
February	95,600	43,478	*5,787*	*319,207*	*865,929*	*0*
March	*43,820*	*20,662*	5,575	154,660	523,141	3,064
April	15,686	7,286	851	33,399	419,285	480
May	3,121	1,892	0	9,912	12,655	0
June	1,595	300	0	6,635	7,346	0
July	1,024	0	0	4,485	4,915	0
August	3,990	0	0	9,677	110,348	0
September	996	0	0	5,489	19,784	0
October	6,575	0	0	73,395	101,972	0
November	4,199	0	0	35,611	83,242	0
December	32,600	11,835	3,831	114,225	297,810	3,543

The wet season reliably extends between December and March, and the flow in the Edith River will provide a significant amount of dilution for the WTP effluent. Discharges to the environment will only occur between December and March.

Table 24-4 provides a summary of field data showing background water quality concentrations of constituents of concern at sampling site SW2, upstream of the WTP on the Edith River. For this assessment, it was assumed that non-detectable sampling events were equal to the minimum detection limit of the analytical method.

Table 24-4: Water Quality Data at Sampling Site SW2, Edith River Upstream of WTP Discharge, January 2015 – April 2017

Analyte	Unit	No. of Samples	Minimum Value	Maximum Value	5th Percentile Value	95th Percentile Value	Average Value
Magnesium	mg/L	81	0.5	1.0	0.5	1.0	0.7
Sulphate	mg/L	139	1	19	1	1	1.16
Aluminum	µg/L	139	30	940	69	622	247
Cadmium	µg/L	139	0.1	0.1	0.1	0.1	0.1
Cobalt	µg/L	75	0.5	1.4	0.5	1.1	0.70
Chromium	µg/L	88	1	2	1	1	1.01
Copper	µg/L	134	1	20	1	2	1.34
Manganese	µg/L	88	7	51	8	24.3	14.4
Nickel	µg/L	88	1	2	1	1	1.03
Lead	µg/L	88	1	1	1	1	1
Iron	µg/L	139	430	1600	450	1100	751
Mercury	µg/L	88	0.05	0.05	0.05	0.05	0.05
Zinc	µg/L	88	1	16	1	8.65	3.56

Using the TVs presented in **Table 24-2**, the background flow rate presented in **Table 24-3**, the background water quality in **Table 24-4**, and a WTP discharge flow rate of 500 m^3/h, the mass balance was solved for the allowable discharge concentrations at the WTP. **Table 24-5** summarizes the allowable effluent concentrations and the WTP effluent goals, which are set at 80% of the allowable concentration to allow for a factor of safety.

Table 24-5: Mt Todd WTP Effluent Goals

Analyte	Unit	C_{SW2}	TV	C_{WTP}	Effluent Goal
Magnesium	mg/L	1	2.5	12.5	10
Sulphate	mg/L	1	129	982	N/A
Aluminum	µg/L	622	55	55	44
Cadmium	µg/L	0.1	0.2	0.87	0.69
Cobalt	µg/L	0.1	90	680	544
Chromium	µg/L	1	1	1	0.8
Copper	µg/L	2	1.4	1.4	1.12
Manganese	mg/L	0.024	1.9	14.4	11.5
Nickel	µg/L	1	11	78	62.4
Lead	µg/L	1	3.4	19	15.2
Iron	mg/L	1.1	0.3	0.3	0.24

Analyte	Unit	C_{SW2}	TV	C_{WTP}	Effluent Goal
Mercury	µg/L	0.05	0.6	4	3.2
Zinc	µg/L	8.65	8	8	6.4

The background water quality concentration at SW2 for aluminum, chromium, copper, iron, and zinc may exceed the site specific TV. In these cases, the WTP will remove the constituent to the TV prior to discharge. WTP effluent will also be used in the process plant for process water and around the site as dust suppression. It is assumed that the water quality requirements for environmental discharge will be satisfactory for these other uses as well.

INFLUENT WATER QUALITY AND TREATMENT

The geochemistry report presents expected water quality at the equalization pond upstream of the water treatment plant in the wet season and dry season for each of the 13 operating years of the mine. The geochemistry model includes inputs from various sources on the mine site, and considers any potential chemical reactions between the various inputs prior to entering the WTP. At the WTP, we are interested in treating the worst-case scenario. **Table 24-6** presents the maximum value for each chemical constituent of concern, and compares it to the WTP effluent goal.

Table 24-6: Anticipated Influent Water Quality at the WTP

Analyte	Unit	WTP Influent	Effluent Goal	% Reduction Required
Magnesium	mg/L	195.3	10	94.9%
Sulphate	Mg/L	2022	N/A	-
Aluminum	µg/L	27,940	44	99.8%
Cadmium	µg/L	60	0.69	98.9%
Cobalt	µg/L	684	544	20.4%
Chromium	µg/L	1.1	0.8	27.2%
Copper	µg/L	4,600	1.12	99.9%
Manganese	mg/L	9.63	11.5	0%
Nickel	µg/L	646	62.4	90.3%
Lead	µg/L	26.8	15.2	43.3%
Iron	mg/L	0.28	0.24	14.3%
Mercury	µg/L		3.2	-
Zinc	µg/L	12,732	6.4	99.9%

The water treatment process is designed to meet the reductions as shown in **Table 24-6**.

Water to be treated at the site will be collected in the PWP. Collected wastewater will flow by gravity from the PWP to the Feed Pump Station. The pump station is adjacent to the PWP and uses concrete wet well construction. Two wet wells (for redundancy) will each house two submersible feed pumps. The Feed Pump Station pumps the collected water to the WTP building for treatment. The WTP process will consist of hydrated lime and chemical precipitation and high rate sedimentation, followed by filtration to remove remaining solids to meet effluent goals. Two identical treatment trains will provide full redundancy at the

WTP at 250 m^3/hr, with a maximum available treatment capacity at 500 m^3/hr. Expected capital costs are presented in **Table 24-7**.

All prices are given in US$ unless otherwise noted. Costs in the table include the equipment cost and an installation cost of approximately 30% of the capital cost of the equipment.

Table 24-7: Opinion of Probable Capital Costs

Parameter	Cost (US$)
Feed Pumps	59,000
High pH Sludge Pumps	33,000
Neutral pH Sludge Pumps	13,000
Treated Water Pumps	42,000
Dust Suppression Pumps	4,000
High Density Lime Clarifier Package System	2,352,000
Sodium Hydrosulfide (NaSH) Reaction Tanks 1B & 2B	138,000
NaSH Reaction Tanks 1B & 2B Mixers	262,000
NaSH Clarifiers 1B & 2B	590,000
Neutral pH Reaction Tank 1C & 2C	138,000
Neutral pH Reaction Tank 1C & 2C Mixers	197,000
Pressure Filters	702,000
Waste Tank Clarifiers 1 & 2	525,000
Treated Water Holding Tank	171,000
Ferric Chloride Feed System	73,000
Lime Feed System	996,000
Lime Metering Pumps	29,000
Polymer Feed System	29,000
Sodium Hydrosulfide Feed System	29,000
Sulfuric Acid Feed System	50,000
Earthwork	494,000
Concrete	227,000
Pre-engineered Building	1,411,000
Electrical and Instrumentation	2,016,000
Piping, Pipe Supports, and Valves	1,472,000
Engineering, Procurement, Construction	1,247,500
Contingency	1,422,000
Cyanide Probes	6,500
HCN Gas Alarms	13,000
Total	14,741,000

The opinion of probable operating costs consist of electricity, labor and chemical consumption. The estimated electrical use at the site is 2,254,000 kWh annually. The estimated labor use at the site includes one (1) supervisor/certified operator and two and a half (2.5) maintenance personnel.

Table 24-8 presents the probable annual chemical consumption for the Mt Todd WTP during average flow conditions. All prices are given in US$ unless otherwise noted.

Table 24-8: Opinion of Probable Annual Chemical Consumption

Date, Month and Season		Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sept	Oct	Nov	Dec
Chemical		Wet	Wet	Wet	Wet	Dry	Dry	Dry	Dry	Dry	Wet	Wet	Wet
Ferric Chloride	tonne	40	42	42	41	36	29	24	23	22	23	31	35
Lime	tonne	172	180	181	176	154	124	104	100	97	100	132	151
Sodium Hydrosulfide	tonne	1.7	1.8	1.8	1.8	1.5	1.2	1.0	1.0	1.0	1.0	1.3	1.5
Sulfuric Acid	m^3	12.1	12.6	12.7	12.3	10.8	8.7	7.3	7.0	6.8	7.0	9.2	10.5

24.2.2.3 Raw Water Reservoir and Pipeline

The Raw Water Dam (RWD) is the sole source for potable and elution water, as it is the only freshwater source on site. The RWD reservoir provides storage of fresh water for use at the mine and processing facility. The reservoir is on a tributary of Horseshoe Creek, located north and east of TSF 1, and retains a reservoir storage volume of approximately 4.5 million cubic m.

The RWD reservoir provides a ready supply of fresh water for several uses. The water balance indicates that process water obtained from recycled process water and TSF decant water will need to be supplemented, particularly in the dry season. The RWD reservoir can also provide water for dust control and onsite potable water supply. Dust control will be needed during the dry season on roads and exposed soil surfaces around the project site. The reservoir generally fills in the wet season (November through April) and will be used during the dry season (May through October). It can also supply wet season fresh water, if needed.

The existing dam is a 13-m high, 114-m long, zoned-embankment dam with a low-level outlet and a spillway. The outlet works are connected to the fresh water pipeline that extends to the process plant. The spillway is designed for the flood- event discharges to Horseshoe Creek.

The existing line from the RWD will need to be augmented with an additional 250 mm line to provide the proper volume of water for the higher throughput.

The Raw Water Pipeline is described in **Section 18.2.1.2 – Sub-Area 4120 – Raw Water**.

24.2.2.4 Potable Water

Potable water will be produced by a potable water treatment plant within the processing facility, and will be distributed to the process plant, mining, administration offices and laboratory facilities.

Drinking water quality guidelines that may be relevant to the Project include the Australian Drinking Water Guidelines (ADWG). These guidelines are intended to provide a framework for good management of drinking water supplies that will assure safety at point of use (NHMRC and NRMMC, 2004).

24.2.2.5 Sanitary Sewer System

The sanitary sewer system will consist of gravity lines conveying the sewerage to a single sewer lift station. The lift station will then pump the sewer to the septic system for treatment. The effluent will flow by gravity to a leach field.

24.3 Geochemistry

Tetra Tech was commissioned by Vista to conduct geochemical characterization studies and predictive modeling in support of the Project Technical Report.

Waste rock samples were selected from the three distinct rock units identified from the 18 mappable rock codes, specifically:

- Greywacke;
- Shale; and
- Mixed greywacke/shale (interbedded).

Eighty-seven waste rock samples were subjected to acid-base accounting (ABA). Nine samples, including three samples from each of the three distinct units were selected for kinetic testing using humidity cell tests. Mineralogy by quantitative x-ray diffraction (XRD) was conducted on the nine humidity cell test samples.

The greywacke waste rock sample average nitric acid (HNO_3) extractable (sulfide) sulfur content of 0.19 wt. % was comparatively low with interbedded and shale samples containing 0.51 and 0.31 wt. %, respectively. Hydrochloric acid (HCl) extractable (sulfate) sulfur was largely absent suggesting that minimal sulfide oxidation occurred prior to geochemical characterization. On average, insoluble sulfur made up approximately 30% of the sulfur distribution in the 87 samples that underwent ABA testing. The average sulfur content of the waste rock samples was $\leq$ 0.51 wt. % HNO_3 extractable sulfide sulfur; however, the potential for acid formation remains a concern due to the limited amount of neutralization potential (NP). On average, samples contained NP $\leq$ 11 kg $CaCO_3$/tonne rock. An acid base accounting (ABA) neutralization potential ratio (NPR) screening criteria < 2 suggests that a majority of the waste rock samples are either potentially acid generating or highly likely to generate acid whereas approximately 30% of the samples were highly unlikely to generate acid. The samples contained high insoluble sulfur (> 30 wt. %) which may be from sulfidic species that are resistant to HNO_3 digestion such as sphalerite (ZnS) and/or galena (PbS).

Preliminary sulfur cutoff criteria were developed based on ABA and Net Acid Generation (NAG) pH results, to assist with waste rock management and closure planning. The specific sulfur cutoff values are:

- Non-PAG waste rock is defined by total sulfur content from 0.005 wt. % through 0.25 wt. %;
- Waste rock with uncertain acid generation potential ranges from 0.25 wt. % through 0.4 wt. % total sulfur;
- The total sulfur content of PAG waste rock is > 0.4 wt. %; and
- Waste rock with > 1.5 wt. % sulfur was considered to be likely acid generating.

The cutoffs were used for geochemical modeling of the WRD seepage and pit lake wall rock runoff and can be used in combination with the total sulfur block model based on the exploration database to assist with proper routing of waste rock.

The nine waste rock samples selected for kinetic testing were subjected to humidity cell testing. Weekly leachate quality results were obtained for pH, acidity, alkalinity, electrical conductivity, and sulfate over the entire test duration. Monthly leachate composites for dissolved constituent concentrations were also obtained over the testing period. Of the nine samples subjected to kinetic testing, a shale sample with 0.43 wt. % HNO_3 extractable sulfide sulfur and low NP = 3.7 kg $CaCO_3$/tonne rock produced acidic leachate (pH < 6) from the initiation of testing. Elevated copper, lead, nickel, and zinc levels were observed in leachate from the acid generating cell. The remaining humidity cells produced circumneutral pH values, with relatively low concentrations of metals. However, it is anticipated that given ample time these cells will likely produce acidic leachate and concomitant increased metal concentrations.

Geochemical characterization of two tailings samples was also conducted including ABA, mineralogy, water leaching, and supernatant analysis. Humidity cell testing has been initiated on one of the samples. The samples contain 1.25 wt. % and 1.13 wt. % total sulfur with net acid production potential (NAPP) and NPR values that show the tailings have potential to eventually generate acid. However, the tailings supernatant and water leach testing produced alkaline pH values. Concentrations of some metals/metalloids, major ions, and cyanide in the tailings supernatant were above ANZECC water quality guidelines, whereas levels were lower in the water leachate but some metals and metalloids and cyanide remained elevated above the guidelines. After 32 weeks, kinetic testing of one of the samples shows a neutral pH with low concentration of metals. Calculations suggest that abundant sulfide sulfur still remains, suggesting the sample may produce acidic leachate given ample time.

Predictive geochemical modeling was conducted to determine the production phase water quality of the WTP Process Water Pond. The water quality estimates were used as a basis for the WTP design and further assist with LoM site water management planning.

Inputs to the Process Water Pond included precipitation and inputs from ponds/facilities from across the site including:

- RP 1 – WRD Retention Pond;
- RP 2 – Low Grade Ore Stockpile Retention Pond (LGRP);
- RP 3 – Batman Pit;
- PWP – Process Water Ponds;
- HLP – Heap Leach Pad Pond; and
- RP 7 or RP 8 – the TSF 1 or TSF 2 Ponds; and
- Precipitation.

Biannual water quality estimates suggest the Process Water Pond may potentially be acidic, with a majority of metal concentrations above the ANZECC water quality guidelines. Metal concentrations fluctuate depending on the relative input source proportions reporting to the Process Water Pond.

In order for Vista to re-commence mining activities, the water in RP3 must be lowered to a level below where mining is scheduled to occur. Treatment of RP3 by micronized lime has been conducted with success, with pH levels becoming circumneutral with a general decrease in metal concentrations that are sufficient for discharge under WDL 178-07 during the wet season.

24.4 Surface Water Hydrology

The Project Site is drained by the perennial Edith River, located approximately 1 km south of RP 1, and also drained by several ephemeral streams, namely: Batman Creek, which bisects the center of the site, and Horseshoe Creek, which is located east of the site. Both Batman and Horseshoe feed Stow Creek, which enters the Edith River at a location south of the discharge point from the Waste Rock Dump Retention Basin (RP 1).

Horseshoe Creek and Batman Creek catchments are approximately 45 and 11 km^2, respectively. The RWD was built across Horseshoe Creek immediately above the mine, forming a sub-catchment covering about 55% of the Horseshoe Creek catchment. The remainder of the Stow Creek catchment is approximately 144 km^2 and is not impacted by mining activity. Stow Creek flows for a short distance after its confluences with both Batman Creek and Horseshoe Creek, prior to joining the Edith River. The catchment area of the Edith River upstream of Stow Creek confluence is approximately 540 km^2.

Surface water at the site is well-documented and its management has been the object of study by both Vista and the NT Government in recent years. Historically, flows from the mine have exceeded the capacity of the water management system, thus allowing uncontrolled discharges to the Edith River. The effectiveness of the water management system has improved as a result of revisions to the pumping systems, installation of a stage height and telemetry station at SW4 and a flow meter on the siphon and pumping outlets from RP 1. Additionally, the NT Government completed a raise of the spillway crest and dam at RP 1 by 1.5 m.

Drainage from the Project Site enters the Edith River at two locations: discharge point for RP 1 and West Creek. The RP 1 discharge point is located 0.8 km below the Stow Creek and the Edith River confluence. West Creek joins the Edith River approximately 1.5 km below the Stow Creek and the Edith River confluence. West Creek delivers water diverted from the undisturbed, natural terrain on the western side of the WRD via the Western Diversion Drain, and overflow from the RP 1 spillway. The West Creek catchment is small and it is reported that the creek only delivers mine water to the Edith River after substantial rainfall events exceed capacity at RP 1. During the wet season (approximately November to April) uncontrolled discharges to the Edith River could occur from any or all of the following during high rainfall events: the WRD Retention Pond (RP 1), the Low Grade Ore Stockpile Retention Pond (LGRP) and the Process Plant Retention Pond (PRP). However, for a large part of the year (approximately May to October), no runoff from the mine area enters the Edith River.

24.5 Regional Groundwater Model and Mine Dewatering

The Project will enlarge the existing Batman pit significantly below the water table. After the existing pit has been emptied, the pit is expected to require additional dewatering as mining progresses. Historical data indicate that the primary driver for dewatering design will likely be runoff entering the pit from precipitation during the wet season, rather than groundwater inflow.

The following sections provide a brief summary of the applicable hydrogeologic information, historical observations, and conceptual pit inflow model. This information and surface water hydrology information provide the basis for the dewatering cost estimate. Geologic information related to the geological setting, mineralization and exploration of the project site was presented in **Sections 7.0 – Geological Setting and Mineralization**, **8.0 – Deposit Types**, **9.0 – Exploration**, and **10.0 – Drilling**; the geologic information in this section is presented from a hydrogeologic perspective as it relates to groundwater flow and pit dewatering.

24.5.1 Regional and Site Hydrogeology

In the Mt Todd area, bedrock occurs either at the surface or, in some valleys and streambeds, beneath a thin layer of alluvial sediment. The 1:250,000 regional geologic map of Katherine, NT[2] indicates that the formations in the vicinity of the BP are the Finniss River Group (Burrell Creek and Tollis Formations) and the Cullen Batholith (specifically the Yinberrie and Tennysons Leucogranites). The Finniss River Group consists of greywacke, siltstone, and shale, interspersed with minor volcanics. Bedding normally strikes at 325° and dips 40° to 60° to the southwest. The Finniss River Group strata have been folded about north-trending F1 fold axes. The folds have moderately west-dipping axial planes, with some sections overturned. The rocks exhibit varying degrees of contact metamorphism which increases with proximity to the intrusive units of the Cullen Batholith. In the vicinity of the Project, metamorphism is typically noted as silicified or hornfelsed material.

The existing Batman pit is located in the Burrell Creek Formation, approximately 2 km from the surface expression of the Cullen Batholith units. However, at the proposed final depth of the pit, the contact has been shown to be only a few hundred meters west of the pit. Thus, the materials encountered during drilling in the immediate vicinity of the pit are typically hornfelsed or silicified greywackes and siltstones with almost no primary porosity. East-west trending faults and joint sets and north-south trending quartz sulfide veining crosscut the bedding. The faults exhibit only minor movement.

While there is little primary porosity in the bedrock of the Mt Todd area, the weathering profile is extensive. In the late 1980s and early 1990s, when the existing Batman pit was under development, a number of production and monitoring bores were installed[3]. These bores are located both near the pit and up to 4 km north and south of the pit. In addition, Vista has advanced a number of boreholes both for exploration and geotechnical evaluation. The borehole logs generally indicate that the upper 3 m are unconsolidated. Below that, weathering typically extends to approximately 30 m below ground surface (m bgs), with the degree of weathering decreasing with depth.

The Mt Todd area experiences heavy rainfall during the wet season. The average rainfall is 1,129 mm/year, but more than 80% of it falls from December through March. Thus, anecdotally, sheet flow of precipitation runoff occurs as the thin crust of soil and alluvial material reaches saturation. During heavy rain events and for some time afterward numerous ephemeral streams develop in the valleys. These subsequently stop flowing during the dry season.

The conceptual model of groundwater flow is that nearly all of the precipitation becomes runoff. Of the precipitation that does infiltrate, most flows within the upper 3 meters of unconsolidated material toward the nearest valley, where it feeds the stream system. Within the valleys, flow occurs as surface water in the streams and also within the thin layer of alluvium beneath and adjacent to the streams. Within bedrock, most water is believed to flow in the weathered profile, through fractures. The regional flow of groundwater is generally from higher to lower elevations.

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[2] National Geoscience Mapping Accord, Katherine (NT), Sheet SD 53-9, Second Edition, 1994.

[3] Rockwater, 1994. Mt Todd Gold Mine, Bore Water Supply Expansion Programme Bore Completion Report.

24.5.2 Regional Numerical Groundwater Flow Model

Tetra Tech constructed a regional numerical groundwater flow model to estimate groundwater inflows to the open pit at Mt Todd and potential impacts to regional and local water resources. The model uses the finite-difference model code MODFLOW-SURFACT, which is widely accepted and commonly used for such applications. The model is regional in scale and incorporates hydraulic properties for regional and local geologic units as derived from on-site testing, precipitation-derived recharge, natural and man-made surface hydrologic features such as ephemeral and perennial streams, the RWD, TSF, WRD, and the existing Batman pit. The proposed enlargement of the Batman pit is incorporated into predictive simulations of groundwater inflows to the pit and post-mining recovery of the groundwater system. Although preliminary calibration of the regional groundwater model has been completed, additional calibration is required and the model has not yet been finalized. Thus, only preliminary estimates of groundwater inflow to the expanded Batman pit and post-mining groundwater system recovery are currently available. The model will be finalized and used to generate updated estimates of dewatering flows and dewatering effects on the groundwater system for the feasibility study.

For this Technical Report, Tetra Tech developed preliminary estimates of groundwater discharge into the pit based on preliminary model output coupled with historical observations (discussed below). Preliminary estimates from the groundwater modeling conducted to date suggest that groundwater inflows should initially be approximately 3 m^3/hr and will gradually increase as the pit is enlarged and deepened, reaching a cumulative average rate of approximately 75 to 120 m^3/hr during the latter part of Phase IV. Under expected normal conditions, a portion of the groundwater inflow would be removed by evaporation from the pit walls and floor. Pit dewatering is expected to lower groundwater levels in the vicinity of the pit. The preliminary modeling suggests that dewatering-related water level declines of 1 m or more should not extend farther than approximately 300 m from the pit.

24.5.2.1 Historical Observations

During the development of the existing Batman pit, very little dewatering was required. The following observations were made:

- In 1994, one bore (BW-30P) was installed to provide dewatering capability if needed for the pit. This bore targeted a production zone between 36 and 50 m bgs and was expected to yield up to 600 cubic m per day (Rockwater, 1994).
- Bore BW-30P may never have been used, since in 1997 a dewatering investigation indicated that the method in use was sumps and sump pumps (Dames & Moore, 1997). The geologic materials exposed in the pit were identified to have an extremely low primary permeability but slightly higher secondary permeability along fractures, bedding planes, and joints.
- In December 1999 to January 2000, a geotechnical investigation described minor seepage on bedding planes and more consistent seepage in the southwest, northwest, and northeast corners of the pit (Pells Sullivan Meynink Pty Ltd., 2000). These seepages were related closely to rainfall and were greatly diminished in the dry season. However, these seepages did not appear to raise any concern at the time with respect to water removal.

The Batman pit operations were shut down in June 2000. Vista personnel visited the site in June 2006 and reported that only 1.5 m to 2 m of water was present in the bottom of the pit, despite the pit floor being approximately 90 m to 100 m below the water table near the pit. Considering that no dewatering had been done in the intervening six years, groundwater inflow is expected to be small and, therefore, a relatively minor component of dewatering.

While the groundwater inflow component is expected to be relatively minor, precipitation during the wet season has historically been significant, especially on a short-term basis. Monthly reports prior to June 2000 indicate that on several occasions large storm events generated sufficient storm-water inflow to interrupt mine operations. One event in particular resulted in the pit floor being inaccessible for approximately a month (General Gold Operations Pty Ltd (GGO), 2000). Thus, a dewatering plan will be required to ensure that surface water runoff and precipitation inflows do not significantly hamper consistent mine operation.

24.5.3 Inflow Estimates

As noted above, groundwater inflow is expected to be a relatively minor component of dewatering. However, the large amount of precipitation and storm-water runoff has historically been a cause for concern. Therefore, for the PFS level dewatering design, the role of groundwater inflow has been assumed to be negligible, and storm-water runoff is the primary consideration for pit dewatering design. While negligible in terms of dewatering system design, groundwater inflows are more continuous than storm-water inflows and hence are significant relative to estimation of dewatering operating costs.

Thus, Tetra Tech based the conceptual dewatering plan on the 10-year recurrence interval, 72-hour and 100-year recurrence interval, 24-hour duration storm events. The precipitation values for those storm events, as obtained from the Bureau of Meteorology (BOM, 2017), are 3.47 mm/hr for 72 hours, which results in a total amount of 249.84 mm, and 10.7 mm/hr for 24 hours, which results in a total amount of 256.80 mm. The precipitation is assumed to fall uniformly over the pit and its catchment area. As the pit increases in size during mine development, the catchment area outside the pit would decrease until the pit comprises the entire drainage area in Phase II. Total volumes of storm water runoff inflow to the pit at the end of each phase of mine development are listed in **Table 24-9**. The volumes were calculated using the SCS Curve Number method (USDA, 1996).

Table 24-9: Catchment and Pit Areas, Inflow Volumes, and Dewatering Times for Mine Dewatering Design

Mine Phase	Catchment Area (m^2)	Inflow Volume (m^3)	Days to dewater pit after 10-year storm event[1]
Phase I	569,000	134,676	14
Phase II	738,787	176,479	19
Phase III	973,336.3	232,508	25
Phase IV	1,251,963	299,065	32

[1] At pumping rate of 500 m^3/hr

24.5.4 Mine Dewatering

Dewatering of the proposed Mt Todd Mine Batman Pit is anticipated to be through passive collection of water in the pit floor sump. The sump would collect surface water, pit wall run-off and precipitation, as well as groundwater inflow. **Table 24-9** shows the days to dewater the pit after a 10-year storm event for each phase of pit development at a 500 m^3/hr pumping rate. The design pumping rate does not increase throughout the LoM; instead, the time it takes to dewater the pit after a storm event increases. The dewatering system has been sized to 700 m^3/hr and reports to the PWP.

Sump water would be removed through pumping and discharge lines to the pit rim and ultimately to the PWP. Water pumped from the pit floor would first go through a pair of pumps mounted on pontoons and then through skid mounted booster pumps placed at 96-120m lifts. Lifts with booster pumps will be added in stages with increasing pit depth. Once at the surface, the water would be piped to the PWP. **Figure 24-1** shows the pit floor pump, booster pumps, and pipeline conceptual design, and **Figure 24-2** shows the conceptual layout of the dewatering system. Costs for dewatering are provided in **Section 22.0 – Economic Analysis**.

The mine dewatering system may require modification and refinement as empirical data become available during advanced exploration and initial mine construction and operation. In particular, groundwater-related mine inflow estimates should be refined based on numerical model updates incorporating observed groundwater inflow rates to the pit and observed water level changes in groundwater monitoring bores at the site.

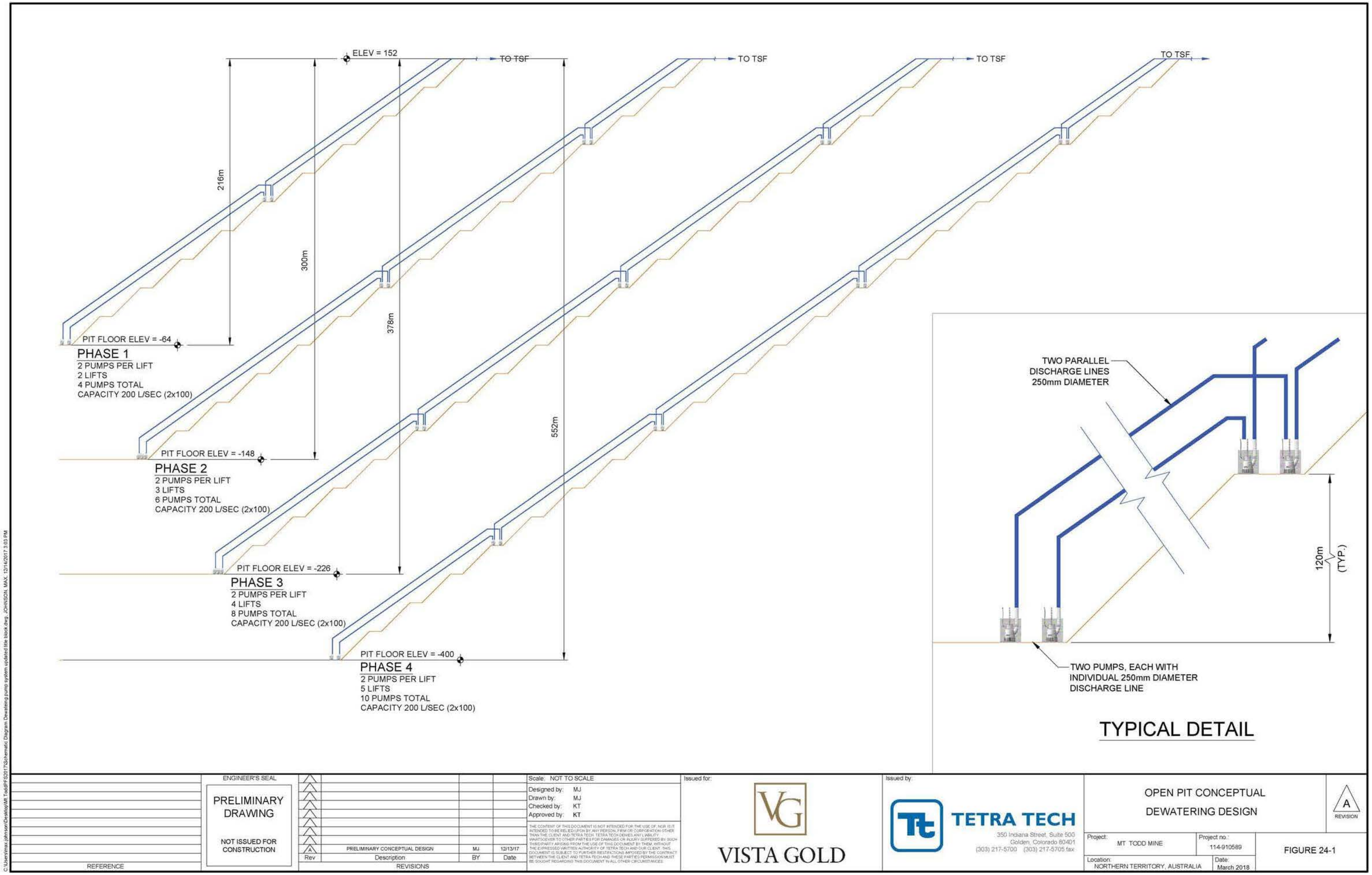


Figure 24-1: Open Pit Dewatering System Conceptual Design

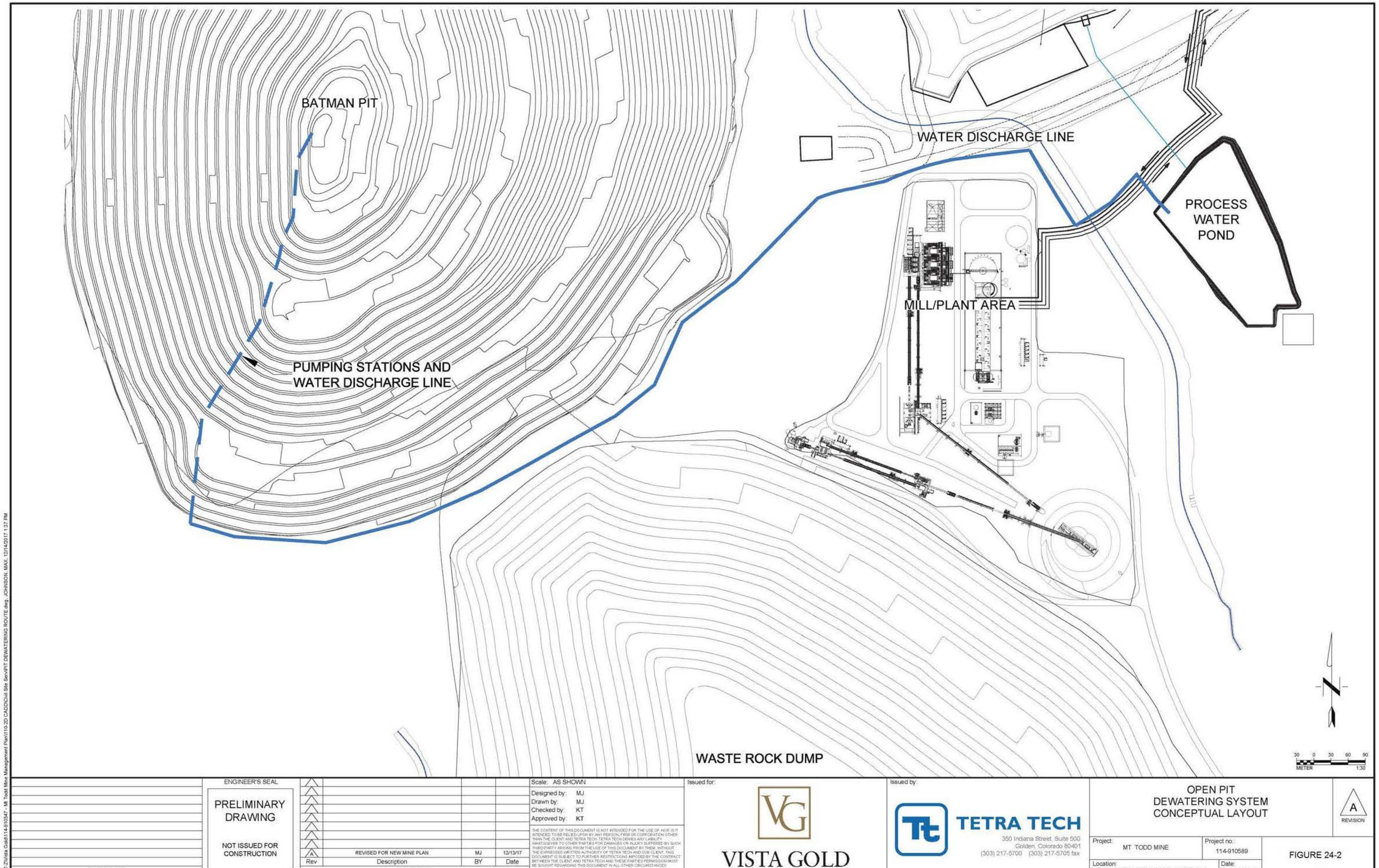


Figure 24-2: Conceptual Layout of Dewatering System

24.6 Project Implementation

24.6.1 Project Implementation Strategy

This section outlines a high level Project Development Strategy, which will be further developed during the next study phase of the Project.

The Technical Report definitions of Scope, Cost and Schedule have been established on the presumption that Vista will implement the Project utilizing the Engineering, Procurement and Construction Management (EPCM) Execution model.

Vista will appoint an EPCM Contractor (Engineer) with the prerequisite capability and experience to undertake the work.

To complement the EPCM approach, Vista will adopt Design and Construct (D&C) implementation strategies, for select areas of the Project.

Properly executed, the EPCM Execution strategy will afford Vista the following benefits:

- Lower Capital Cost Outcomes
- Project Implementation flexibility
- Fast-Track Execution opportunities
- Flexible Project Funding Strategies
- Optimal Project Quality Outcomes

24.6.2 Project Organization

24.6.2.1 EPCM Contracts

Vista's Project Manager will direct all activities including EPCM and D&C Contractors.

For the EPCM Scope, two organization charts are developed:

- EPCM Stage 1 – Design & Procure. Refer to **Figure 24-3**.
- EPCM Stage 2 – Construct & Commission. Refer to **Figure 24-4**.

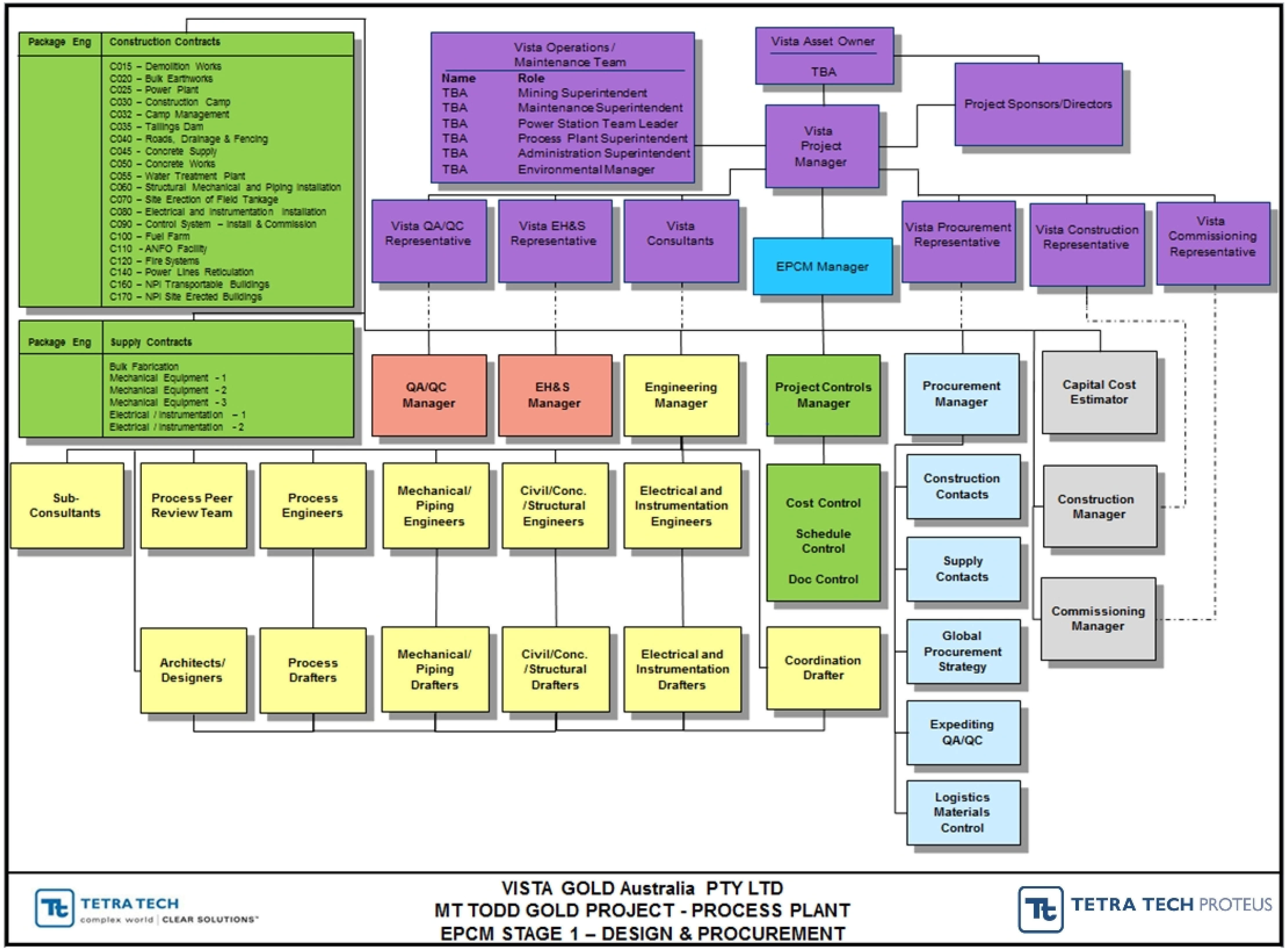


Figure 24-3: EPCM Stage 1 – Design & Procurement. Refer Diagram 1

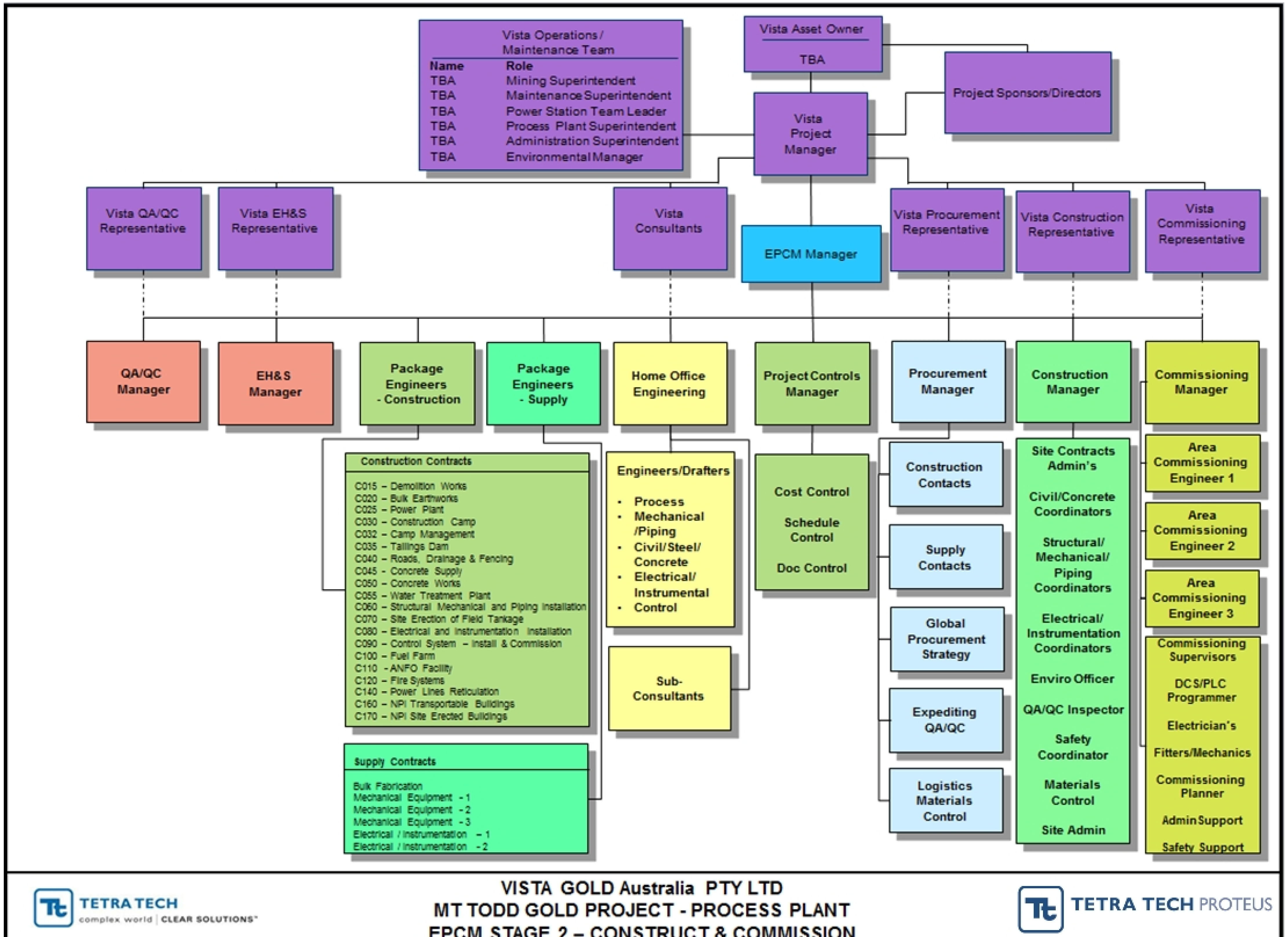


Figure 24-4: EPCM Stage 2 – Construct & Commission. Refer Diagram 2

24.6.2.2 D&C Contracts

Two D&C contracts are proposed, specifically:

- Non Process Infrastructure (NPI) – Transportable Buildings (Package C160); and
- NPI – Site Erected Buildings (Package C170).

24.6.2.3 EPCM Contract Scope of Services

Generally, the Engineer will perform the following tasks:

- Detailed process, civil, structural, mechanical and electrical design;
- Establish a document control system;
- Preparation of specification documentation;
- Calling and review of tenders for supply and installation of equipment;
- Contract evaluation, negotiations, documentation and management;
- Preparation of Purchase Orders and Contracts;
- Quality audits of major contractors and manufacturers;
- Construction management;
- Equipment and site inspections;
- Cost control, procurement, scheduling and planning, contract administration;
- Regular reporting on progress against schedule and cost against budget;
- Site testing and commissioning; and
- Preparation and review of Operation and Maintenance manuals.

24.6.3 EPCM Management

The Engineer will provide an experienced and suitably qualified Project Manager who will manage all aspects of the EPCM Contract. The EPCM Manager will be the single point of contact for the Vista Project Manager and will work closely with the Vista senior managers and other Project Managers associated with the project.

24.6.4 Engineering

The Engineer will provide an experienced and suitably qualified Engineering Manager who will manage discipline-based groups of Engineers and Draftsmen that will be responsible for coordination, direction, administration and completion of all detail design. Effort will be primarily aimed at optimizing design, uniformity and quality of design and monitoring of time spent against budget.

Where Engineering Design is undertaken, progress will be reported by the Engineering Manager through the EPCM Project Manager to the Vista Project Manager.

24.6.5 EPCM Controls

Using the Feasibility Study report as the basis for project scope and the capital cost estimate as the control budget in the first instance, the project will be managed in accordance with the Project Schedule submitted in the Study report.

Initial activities will be directed to the awarding of Construction Contracts and/or Supply Contracts for long lead time items of plant and equipment, immediately upon Vista's approval to proceed. The budget and schedule will be continually updated to reflect the current understanding of the project status.

The Engineer's Project Controls group will report to the EPCM Project Manager and will have responsibility for the following activities:

- Monitoring and reporting of contract package progress. This will be performed on a daily basis as necessary and reported weekly/fortnightly/monthly as required by means of the Procurement Status Report.
- Definitive estimate maintenance and forecasting. Records of variations to budget and other forecast estimates of cost to completion will be updated as necessary and reported by means of the Cost Control Report Summary.
- Cost control for procurement and contracting. Actual costs (invoiced and payments made), committed costs (orders placed) and estimated costs will be reported against budget using the Cost Control Report. The Trend Notice/Scope Change Notice system will be incorporated with these activities to ensure accurate forecasting.
- Coordination of construction planning and scheduling. Weekly meetings of all TTP-controlled site contractors' Project Managers will be held to coordinate changes, clashes and priorities between contractors.
- Maintenance of an overall Schedule. The Schedule will be formatted using the WBS information received from all contractors and will be updated using information obtained from the various contractors and reviewed by the Project Manager on a weekly basis as a minimum.
- Project reporting. A monthly project progress report will be issued including, but not limited to, the following information:

  Highlights for the reporting period

  - Safety, Health and Environment issues;
  - Overall project status;
  - Engineering progress;
  - Procurement and fabrication progress;
  - Construction activities;
  - Planned activities for the next reporting period;
  - Current project cost reports;
  - Outstanding issues, Variations, Technical Queries, etc.;
  - Project S-curves; and
  - Photographs depicting project progress.

24.6.6 Procurement

24.6.6.1 Procurement Strategy

The key procurement aims and objectives are to:

- Achieve the project objectives of earliest possible completion, cost-effective execution, quality workmanship and high degree of safety from suppliers.
- Adhere to the project plan, aims and schedule.
- Ensure that commercial and schedule risks are at acceptable levels.
- Provide a purchasing environment that minimizes claims and protracted disputes.

- Provide a procurement arrangement that encourages suppliers to be innovative and efficient.
- Carry out the procurement function for the project in an ethical and professional manner.

Key success factors are to:

- Meet or exceed expectations for health and safety requirements.
- Meet or better the project schedule.
- Meet or better the project budget.
- Meet project quality objectives.

24.6.6.2 Procurement Overview

The Engineer's Procurement Manager will report directly to the Engineer's EPCM Project Manager but will also liaise directly with Vista's Commercial Manager.

The Engineer's Procurement Manager will be responsible for the preparation, all approvals and proper implementation of the Project Procurement Plan.

Prior to the project receiving all necessary approvals (Vista and Statutory), award of clearly identified and specified packages containing long lead time delivery items will only be initiated by written authorization from Vista.

The Engineer's Procurement Manager will adhere to Vista's procurement policy and procedures in place at the time with regard to authorization levels for capital expenditure and the requirements to obtain competitive quotations at discreet capital expenditure levels.

All packages for supply of all project related goods and services will be prepared, tendered, assessed and awarded by the Engineer's Procurement group. All purchase orders and contracts will be prepared by the Engineer's Procurement group but issued through Vista's purchasing system.

Where goods and services are required from outside Australia, the Engineer's Procurement Manager will ensure, through liaison with Vista, that sufficient forward cover on foreign exchange transactions is in place to mitigate any risk of currency fluctuation.

24.6.6.3 Construction Packages

The Engineers Procurement Manager will be responsible for the development of a Construction Contracting Strategy.

A preliminary strategy is documented in the Contracting and Procurement Plan.

The following Construction packages are envisaged as a minimum:

Table 24-10: Construction Packages

Package No.	Package Description
C015	Demolition Works
C020	Bulk Earthworks
C025	Power Plant
C030	Construction Camp
C032	Camp Management
C035	Tailings Dam
C040	Roads, Drainage & Fencing
C045	Concrete Supply
C050	Concrete Works
C055	Water Treatment Plant
C060	Structural Mechanical and Piping Installation
C070	Site Erection of Field Tankage
C080	Electrical and Instrumentation Installation
C090	Control System - Install & Commission
C100	Fuel Farm
C110	ANFO Facility
C120	Fire Systems
C140	Power Lines Reticulation
C160	NPI Transportable Buildings
C170	NPI Site Erected Buildings
C180	Communications – Telstra Interface
C190	Communications – Temporary

24.6.6.4 Supply Packages

The Engineers Procurement Manager will be responsible for the development of an Equipment and Services Supply Contracting strategy.

The following supply packages are envisaged as a minimum:

Table 24-11: Supply Packages

Package No.	Package Description
P001	Ball Mills
P030	Secondary Grinding Mills
P002	Primary Crusher
P003	Secondary Crushers
P004	HPGRs
P005	Dry Screens
P006	Wet Screens
P007	Slurry Pumps
P008	Solution Pumps
P009	Apron Feeders
P010	Belt Feeders
P011	Cyclone Clusters
P012	Agitators
P013	Thickener
P014	Inter Tank Screens
P015	Carbon Transfer Pumps
P016	Gold Room
P017	Vibrating Feeders
P018	Container Tippers
P020	Flocculant Mixing Package
P021	Lime Slaker
P023	Potable Water Plant
P024	Mill Relining Machine
P025	Overhead Travelling Cranes
P026	Air Compressors, Driers & Receivers
P028A	Conveyor Drives
P028B	Conveyor Pulleys
P028C	Conveyor Idlers
P028D	Conveyor Belts & Splicing
P028E	Conveyor Skirts
P028F	Conveyor Scrapers & Ploughs
P029	Ore Sorting
P031	Wet Scrubber

Package No.	Package Description
P032	Isolation Gates
P033	Ventilation Fans
P034	Screw Feeders
P035	Rotary Valves
P036	Filters
P038	Hoists
P039	Ball Charging Magnets
P040	Tramp Magnets
P041	Sump Pumps
P042	Firewater System
P043	Weightometers
P045	Samplers
P046	Analyzers
P047	Rock Breaker
P048	Blowers - Detox
P049	Metal Detectors
P050	FRP Tanks
P051	Winches
P053	Manual Valves
P054	Laboratory Equipment
P055	Bag Splitters
P056	Safety Showers
P057	Pressure Relief Valves
P058	Pressure Regulators
P060	Weighbridge
P101	HV Switchgear
P102	HV Cables
P103	Transformers
P104	Motor Control Centers (MCCs)
P105	HV Variable Speed Drives
P106	Neutral / Earth Resistors
P107	Overhead Power Lines
P108	Control System - Supply
P109	Instruments
P110	Switchrooms/MCCs
P111	LV Variable Speed Drives
P112	Power Factor Correction / Harmonic Filters
P113	Control Valves
P114	CCTV

Package No.	Package Description
P115	2 way Radios
P116	Plant Fire Detection Systems
P117	RMUs/Kiosk Substations
P118	Spares
P119	Telemetry
P120	Emergency Power
P121	Security
P122	UPS
P123	WAD Cyanide Analyzers
P124	HCN Monitors
P125	Data Room
P126	Motors
P200	Fabricated Structural Steel Work
P210	Fabricated Platework

24.6.6.5 Indirect Packages

The Engineer's Procurement Manager, in collaboration with Vista, will establish and manage a series of Indirect Packages.

The Indirect Packages are envisaged as a minimum:

- EPCM Services
- Environmental Consultants
- Human Resources (HR) & Industrial Relations (IR) Consultants
- HSEC Consultants
- Commissioning
- Licenses, Fees and Legals
- Project Insurances
- Pre-Production Costs
- Capital Spare
- Stores and Inventories
- Heavy Lift Cranage

24.6.6.6 Expediting

The senior expeditor will plan and control expediting activities in consultation with procurement, establishing material status reports and ensuring suppliers comply with agreed delivery of drawings, data, materials and equipment. The post-award responsibility for the Supply Contract is vested with expediting; however, commercial responsibility stays with the purchasing officer. Expeditors will anticipate and act at the earliest possible stage to eliminate or reduce delays which may impact on the project schedule.

Manufacturing and delivery progress will be monitored and reported to the project via expediting status reports. Status reports will verify the milestones reported. Exceptions will be reported to management. These reports will detail actions being taken to resolve any issues causing concern.

The senior expeditor will utilize global support offices of a worldwide expediting third-party provider if necessary, to achieve the project schedule.

24.6.6.7 Logistics and Transport

The Engineer will be responsible to manage the consignment of equipment and materials to the Project Site in the Northern Territory. A proven international project freight forwarding group with a global network will be appointed early in the project to provide logistics support services and to aid in the preparation of the transport and logistics plan. The focus will be on the most cost effective solution for delivery to site of equipment and materials to meet the construction schedule.

The logistics specialist will develop a transport plan to be used to manage the sea, road and airfreight costs to budget. Selected land transport subcontractors will be required to display the necessary capabilities and dedicated management that will ensure equipment is suitable and operators take every precaution to meet the project safety and quality requirements.

Transport plans will be prepared for all equipment based on maximum project transport envelopes. The review of the bulk steel supply will contribute to the plan.

The plan will include, but not be limited to:

- Functional requirements of an inbound logistics system;
- Assessment of existing transport nodes and linkages (ports, roads and rail);
- Maximum load length, width, height and weight restrictions;
- Specialist heavy-lift and over-dimensional transport requirements at port, for example, liaison with statutory authorities and utilities, permitting, road closures and escorts;
- The requirement for “holding facilities” at port to manage the storage of equipment, materials and bulk steel pending transport to site;
- The movement of over-size components to site;
- Assessment of site conditions;
- Identification of alternative operational model; methodologies, constraints and risks; and
- The identification and management of shipping container and other demurrage costs.

The freight forwarder (or an independent consultant) will specifically review the movement of the bulk steel supply from place of manufacture to project site.

24.6.7 Construction Management

The Engineer’s Construction Manager will establish a small on-site team prior to construction contractors mobilizing to site. The exact timing of the team’s establishment will be dependent on feedback from contractors regarding progress off site, but site establishment should not be less than four weeks in advance of contractor mobilization.

The Engineer’s Construction Manager will ensure that all construction contractors are responsible for:

- Maintaining a safe site;
- Maintaining compliance with all appropriate Statutory and Legislative requirements; and

- Maintaining compliance with all Vista site requirements in regard to Environmental Health and Safety of the construction work force and the supervising team.

All site works will be undertaken utilizing qualified construction contractors, and the Construction Manager will act in the role of Superintendent to Vista when administering the construction contracts.

The construction supervision team will comprise of suitably qualified and experienced personnel and, where possible, preference will be given to more senior professionals when selecting staff, recognizing that the construction schedule and budget are of significant importance.

24.6.8 Commissioning

The Engineer will develop a Commissioning Management Plan, in collaboration with the Vista Commissioning Representative. The Engineer's Commissioning Manager will report to the EPCM Manager, but liaise closely with Vista's Commissioning Representative. Three Commissioning Areas are contemplated:

- Primary Crusher up to Mill;
- Mill to Gold Room; and
- Non-Process Infrastructure (NPI).

Supervision of the various areas and disciplines during the discreet commissioning phases will be the responsibility of specifically appointed professional engineers assisted by key personnel from any design teams, construction teams, representatives of the various vendors and from the client's staff.

Commissioning for the Process Plant will be generally carried out in three distinct phases:

- Dry commissioning of all mechanical and electrical equipment including manual rotational checks, off load driven rotational checks, functional checks, instrument I/O checks, electrical continuity checks, etc.;
- Wet commissioning of all mechanical and electrical equipment including hydraulic pressure testing using water, coupled with flow testing using water to ensure integrity of the various pumped circuits; and
- Process commissioning of all mechanical and electrical equipment using production materials, commencing at minimum throughput requirement and gradually increasing to full design capacity prior to conducting any necessary performance testing.

Refer to **Figure 24-5** for Commissioning Phases bar chart.

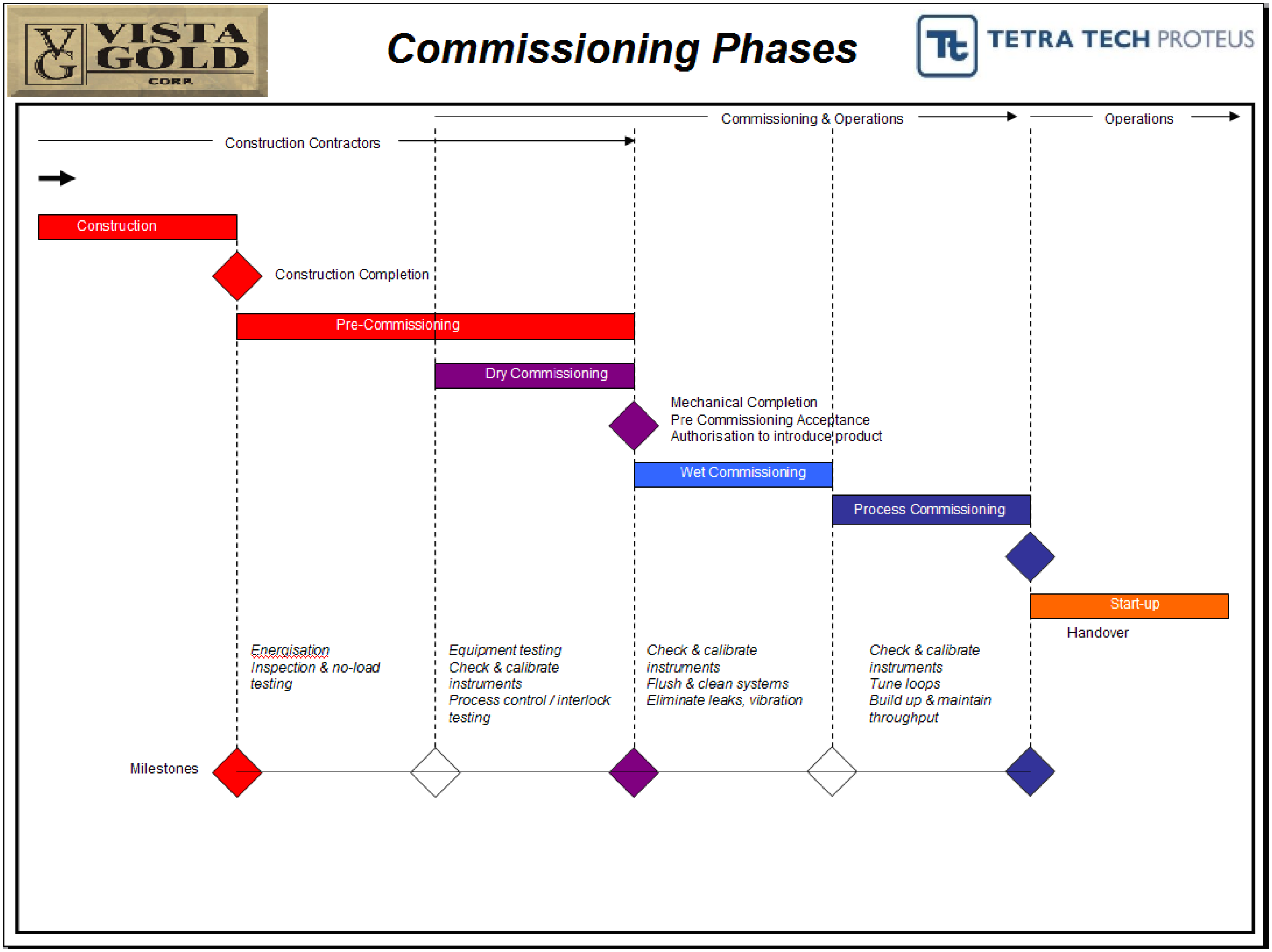


Figure 24-5: Commissioning Phases

All modifications required during commissioning will be documented in a Project Modification Register and subject to the same verification as detailed design with respect to design, fit for purpose, Environment, Health and Safety (EHS) and Hazard and Operability (HAZOP) Study requirements and drawing updates. All modifications will be carried out by either construction contractor's representatives or vendor representatives.

24.6.9 Temporary Construction Facilities

All Contractors will be responsible for the provision of their own site facilities to an appropriate standard that complies with local EHS guidelines for offices and amenities and to the approval of the Construction Manager.

All contractors will be responsible for the upkeep, cleaning and sanitation requirements of their respective facilities.

The EPCM Contractor will be responsible for the provision of suitable connection points for power, water and sewerage. The EPCM Contractor will be responsible for the provision of suitably located areas for the installation of the temporary facilities and for the provision of a suitably located receipt and lay-down area for delivered goods.

24.6.10 Industrial Relations

All contractors will be required, under the terms of their contract, to take responsibility for their own industrial relations. They must be able to demonstrate and have in place suitable policies and procedures to ensure that the handling of matters of an Industrial Relations (IR) nature cause minimum disruption to the project schedule and budget.

All contractors must be able to demonstrate compliance with the HR/IR Policy. The Plan will be incorporated into all tender documentation. This plan will also contain details of the Site Agreement on wages and conditions that will apply universally to the project.

Contractors may be required to be affiliated to an equivalent Chamber of Commerce and Industry (CCI) for the Northern Territory. The CCI being a recognized and competent employer organization that can provide adequate IR advice and advocacy service, should the contractor fail to demonstrate the adequacy of his own internal services in this area.

All contractors should make an allowance to retain the CCI to develop suitable IR strategies, policies and procedures that will ensure that, in the event of industrial action being taken by contractors, the resolution of such matters will be timely and of such a nature as to not adversely affect the project schedule and budget.

24.6.11 Health and Safety

All contractors will be required to comply with AS 4801 (Standard for Safety Management Systems) as a minimum.

All contractors must be able to demonstrate compliance with the EHS Project Management Plan. The Plan will be incorporated into all tender documentation.

The Engineer, in conjunction with Vista, will be responsible for developing a safety policy during the initial phase of the project. This policy should set out guidelines for the project safety procedures and the safety targets for the project. Particular emphasis will be placed on site attendance of project personnel and the occupation of the site by the construction team and various contractors.

The policy will address the following issues:

- The legislative responsibilities of Vista and the Engineer under the relevant Occupational Health, Safety and Welfare Act;
- The legislative responsibilities of contractors under the relevant Occupational Health, Safety and Welfare Act;
- The legislative responsibilities of employees under the relevant Occupational Health, Safety and Welfare Act;
- The establishment of safety protocols and management systems required by the Act and how they will be practically implemented to suit the needs of the project; and
- Any IR issues that need to be addressed as part of the overall safety management program.

All new employees attending site will be required to complete the necessary Vista site induction programs.

The Engineer will employ an experienced Safety Manager for the term of the project and a Site Safety Officer for the period of site occupation. The Safety Manager will be responsible for implementing the project safety policy, developing procedures in conjunction with the Engineer's Site Safety Officer and implementing the provisions of the relevant Occupational Health, Safety and Welfare Act.

The Engineer's Site Safety Officer will be responsible for enforcing all safety procedures and rules on the construction site and will organize regular communications with contractors to ensure adherence to policy, procedures and rules.

Contractors will be required to support the project safety protocols, provide individual safety management plans, perform Job Safety Analysis and ensure their employees are provided with Personal Protective Equipment to the standard defined by the overall site policy. Contractors must also provide a nominated individual at supervisory level, who has received adequate training in Occupational Health and Safety (OH&S), who will be responsible for safety procedures within the contract.

Contractors will be required to provide adequately equipped First Aid kits and have at least one formally qualified First Aid person on each shift to administer minor injuries not requiring medical attention from a Doctor. In the event of a more serious injury, Vista will make available the site First Aid facilities and personnel to all project related employees.

The Engineer's Construction Manager will ensure that adequate records are kept of all safety incidents, irrespective of whether First Aid is required. TTP's Site Safety Officer will report Lost Time Injury Frequency Rate, Disabling Injury Frequency Rate and Medically Treated Injury Frequency Rate, along with severity information, on a weekly basis as a minimum.

24.6.12 Environment

The Engineer's staff and all contractors will be made aware of the site environment conditions and constraints at the time of induction. Vista Environmental staff will be asked to audit site works on a periodic basis to identify issues of concern or non-conformance with site environmental policies and procedures.

All contractors must be able to demonstrate compliance with the EHS Project Management Plan. The Plan will be incorporated into all tender documentation.

24.6.13 Schedule

24.6.13.1 Schedule Objectives and Scope

The key objective of the PFS phase EPCM schedule is to provide a Class 3, Level 3 detail Schedule with an accuracy range of ±15%.

Class of Schedule defines the degree of completeness required for schedule development, Class 5 being a low degree of completeness, and Class 1 being a high degree of completeness. Level of Schedule defines the degree of detail for communication, reporting, and execution, Level 1 being a low degree of detail and Level 5 being a high degree of detail.

The scope included in the Schedule is that which is included in the EPCM contractor's scope, as defined in the Technical Report. Consequently, Client Activities, Mine Development, Tailings Dam, Power plant detail, or Waste Water Treatment Plant are excluded from Schedule.

24.6.13.2 Schedule Assumptions

For the Project, the specific schedule assumptions include:

- The Northern Territory wet season runs from ~1st December to ~17th April when heavy rains can impact construction activities at times, particularly civil and concrete works;
- No force majeure disruptions to scheduled work (IR or otherwise);
- Open access to all work fronts is available;
- Transportation to and from site (both air and land) is without delay; and
- The schedule has assumed that project approval will be given by Vista on or about January 1, 2021. Start up, as defined by handover after completion of commissioning is scheduled to early-mid-2023.

24.6.13.3 Critical Activities

The Critical Path of the EPCM Schedule runs through the Vista approval process and the purchase packages and contracts for Area 3300 (Classification and Grinding) as follows:

- P001 Ball Mills Scope Development and Tender Period......... 11 weeks
- P001 Ball Mills Manufacture and Delivery............................. 63 weeks
- P001 Ball Mills SMP Construction.. 24 weeks (Total)
- Area 3300 Verification and Commissioning........................... 5 weeks

The above critical activities determine a critical path of approximately 119 weeks duration after Project approval to proceed has been given.

24.6.13.4 Significant Activities

Major procurement packages with a lead time ex-works greater than 40 weeks are:

Table 24-12: Supply Packages with Significant Lead Times

Package	Lead Time
P001 – Ball Mills	55 weeks
P002 – Primary Crusher	48 weeks
P004 – HPGRs	52 weeks
P012 – Agitators	60 weeks
P024 – Mill Relining Machine	52 weeks
P029 – Ore Sorting	45 weeks
P030 – Secondary Grinding Mills	54 weeks

24.6.13.5 Commissioning Schedule

The Commissioning Schedule has been broken into five specific activities in each area / sub area:

1) Construction Verification (CV)
   a) Occurs immediately after construction completion for each area (e.g. 3100, 3200, etc.) with each area CV start date independent of the others
2) Pre commissioning
   a) Occurs once CV is finished for each sub area
   b) 100,000t of ore available before pre-commissioning commences
3) Dry Commissioning (DC)
   a) Requires equipment power up so each of the five sub stations (one for crushing / stockpile, one for HPGRs, one for milling, Secondary Grinding and one for leach / CIP / gold room / air / water services) need to be completed prior to commencing DC.
   b) Should also occur in the order of:
      - Safety Systems (fire water / safety showers, etc.)
      - Process ancillary equipment (instrument air / gland water, etc.)
      - Process equipment substation groupings
      - Check Spares receipted into site store for equipment items in the area
4) Wet Commissioning (WC)
   a) The order for WC needs to be:
      - Safety systems (fire water / safety showers, etc.)
      - Environmental systems (storm water pond pumps, sump pumps, etc.)
      - Process ancillary equipment (instrument air / gland water, etc.)
      - Process area where both the current area and downstream area dry commissioning has been completed
      - Workforce training completed

5) Process Commissioning
   a) Will occur sequentially in the order of process flows, with the proviso that each area within a process zone terminated by a large storage buffer has been completed. Large storage buffers likely to create independent process commissioning zones include:
      - Crushed ore stockpile
      - Thickener
      - Leach / CIP Tanks

24.6.13.6 Schedule Interfaces

The EPCM Schedule does not include detailed activities from contractors undertaking scopes of work outside the PFS scope such as the Tailings Dam, Waste Water Treatment Plant, Mine Development, and the Power Plant.

The Construction schedule is currently based on best estimate for the logical sequence of activities as developed by the Feasibility Study contractor. Upon award of contracts during the EP phase, construction contractors will be required to each develop and provide their schedules which will form a Class 3 Level 4 detailed schedule. This schedule will only be baselined with the approval of Vista, EPCM contractor, and the Construction contractor.

24.6.13.7 Reporting

The Engineer's Project Manager will ensure that the Schedule is updated within 3 working days of the end of each calendar month such that progress against project milestones and activities can be clearly identified. The project schedule will also show the critical path(s) at each update such that possible improvements in project completion forecast may be made.

Each month the EPCM contractor will provide the following to Vista and contractors:

- The whole schedule
- Critical Path/20 day or less Total Float view that will identify the critical path while also showing the activities with less than 20 days Total Float
- Mid-month short form status report covering expenditure and schedule compliance

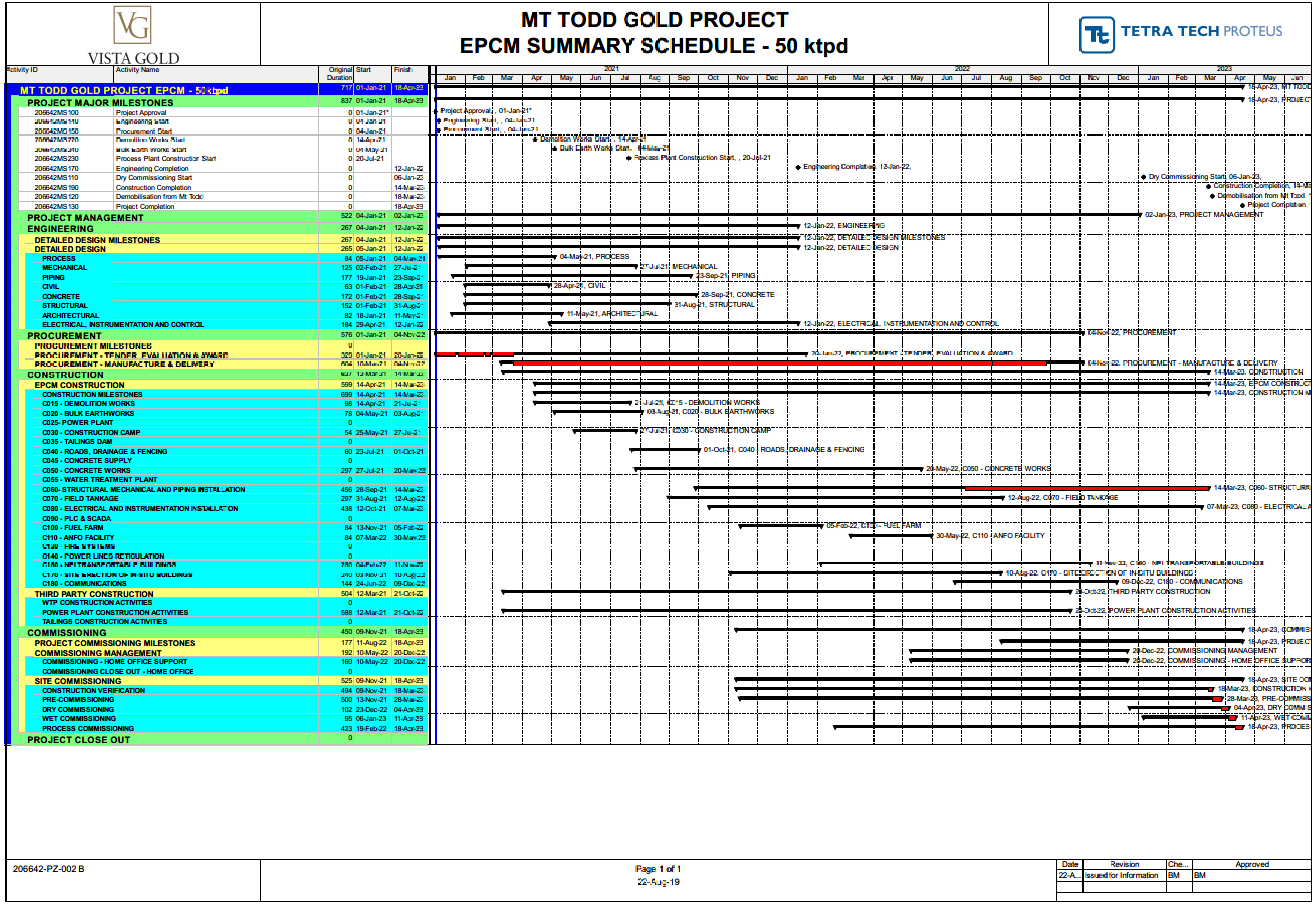


Figure 24-6: EPCM Summary Schedule

24.7 Alternate Case

Vista also prepared an Alternate Case which considers a smaller and higher-grade project. Key differences between the Base Case and the Alternate Case include:

- A 33,000 tpd milling facility with associated lower mining rates and a smaller mining fleet;
- An Alternate Case pit design based on a pit shell of US$800/oz-Au and cut-off grade of 0.40 g-Au/t; and
- A shorter LoM (11 years).

Results of the Alternate Case are presented in this Section.

24.7.1 Mineral Resource

The same Mineral Resource, as used in the Base Case is used in the Alternate Case.

24.7.2 Mining

24.7.2.1 Pit Optimization

Separate pit designs were completed for the Base and Alternate Cases. Reserves are based on the Base Case ultimate pit design as this pit is larger and encompasses all material inside of the Alternate Case ultimate pit design. Economic Parameters are shown in **Table 24-13**.

Table 24-13: Initial Economic Parameters, Alternate Case

	Alternate Case
Gold Recovery	85% Sulfide, 80% Transition, 80% Oxide
Payable Gold	99.9%
Overall Mining Cost	US$2.16 per tonne
Processing Cost	US$8.65 per tonne processed
Tailings	US$0.90 per tonne processed
General & Administrative	US$0.77 per tonne processed
Water Treatment	US$0.10 per tonne processed
JAAC Royalty	1% gross proceeds

Costs above reflect the costs used for pit optimization and do not reflect the final costs for the Alternate Case

As with the Base Case, the mining cost was varied using an additional US$0.010 per each 6 m bench below the 145 m elevation. The reference mining cost of US$1.86 was used for the Alternate Case. Processing, tailings construction, tailings reclamation, and water treatment costs were provided by Vista and based on previous studies. The total mining cost (reference plus incremental) is US$2.16 for the Alternate Case.

A minimum cutoff grade of 0.40 g-Au/t was used for the Alternate Case. This was done to maintain higher grades with respect to material allowed to be processed.

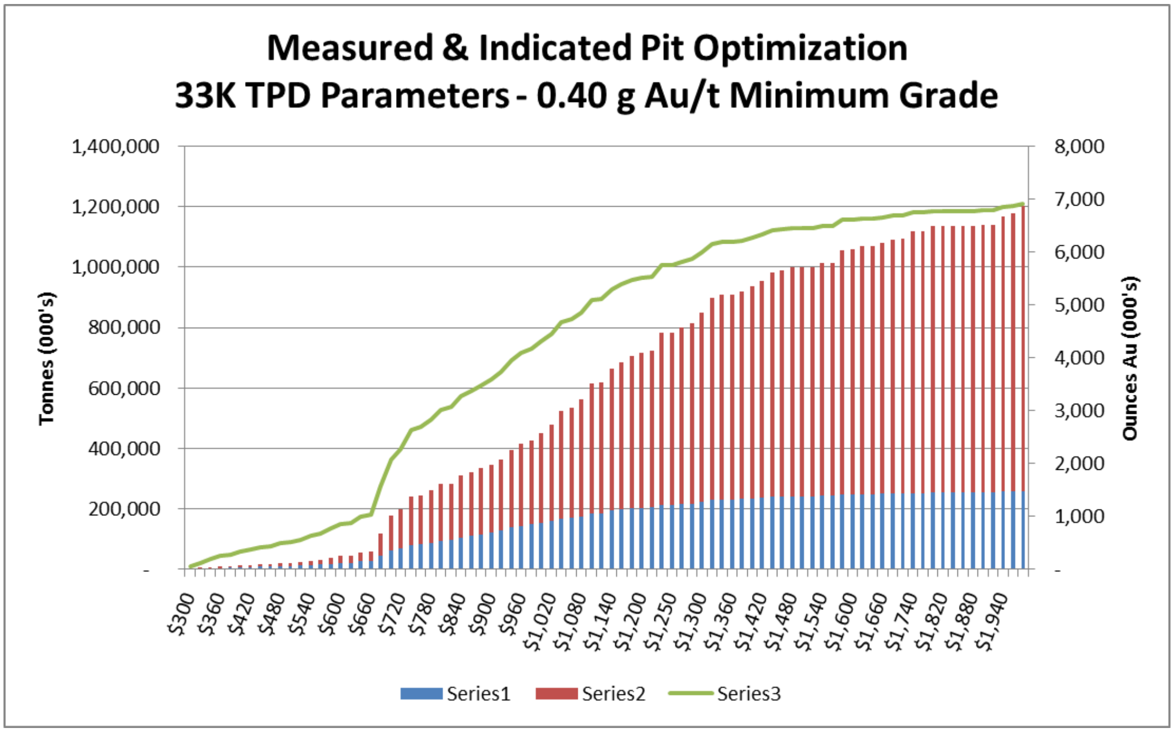
Pit optimizations were completed using prices of US$300 to US$2,000 per ounce Au in increments of US$20 per ounce in order to analyze the deposit's sensitivity to gold prices for both scenarios. **Table 24-14** includes the US$1,250 Au price, which is highlighted in light green. The $800 pit shell is highlighted as the pit shell used as an ultimate pit guide. The pit optimizations only used measured and indicated resources. Inferred materials are considered waste.

Table 24-14: Whittle™ Pit Optimization Results – Alternate Case Using 0.40 g-Au/t Cutoff

Pit	Gold Price	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio
		K Tonnes	g-Au/t	K Ozs Au			
1	$ 300	966	1.88	59	731	1,697	0.76
6	$ 400	6,467	1.63	338	5,647	12,114	0.87
11	$ 500	11,133	1.46	523	10,299	21,432	0.93
16	$ 600	21,103	1.25	850	22,828	43,931	1.08
21	$ 700	60,702	1.06	2,078	115,883	176,585	1.91
26	$ 800	95,474	0.98	3,019	185,830	281,304	1.95
31	$ 900	121,301	0.92	3,586	222,805	344,106	1.84
36	$ 1,000	154,092	0.87	4,307	295,849	449,941	1.92
41	$ 1,100	185,525	0.85	5,085	427,833	613,358	2.31
46	$ 1,200	202,606	0.84	5,504	512,606	715,212	2.53
49	$ 1,250	212,904	0.84	5,753	569,384	782,289	2.67
52	$ 1,300	223,045	0.84	5,996	626,265	849,310	2.81
57	$ 1,400	234,858	0.83	6,278	700,519	935,376	2.98
62	$ 1,500	241,417	0.83	6,454	756,553	997,970	3.13
67	$ 1,600	247,350	0.83	6,605	809,868	1,057,218	3.27
71	$ 1,700	250,425	0.83	6,684	841,140	1,091,565	3.36
75	$ 1,800	254,050	0.83	6,779	880,583	1,134,633	3.47
80	$ 1,900	254,353	0.83	6,785	883,275	1,137,628	3.47
84	$ 2,000	259,140	0.83	6,908	943,012	1,202,152	3.64

Pit 26 was used for design purposes and Pit 49 illustrates the potential floating cone using a US$1,250/oz-Au price.

Graphs of the Whittle™ results are shown in **Figure 24-7**.



NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia – May 29, 2013

Figure 24-7: Measured and Indicated Graph of Whittle™ Results – Alternate Case Using 0.40 g-Au/t Cutoff

The ultimate pit limit for the Alternate Case was based on trying to reduce the capital requirements and the general overall size of the Project. This used a US$800/oz-Au pit shell for the Alternate Case.

24.7.2.2 Pit Designs

Detailed pit design was completed, including an ultimate pit and two internal pits. The ultimate pit was designed to allow mining economic resources identified by Whittle™ pit optimization, while providing safe access for people and equipment. Internal pits or phases within the ultimate pit were designed to enhance the project by providing higher-value material to the processing plant earlier in the mine life.

The ultimate pit design along with the ultimate dump and stockpile designs and planned infrastructure is shown in **Figure 24-8**.

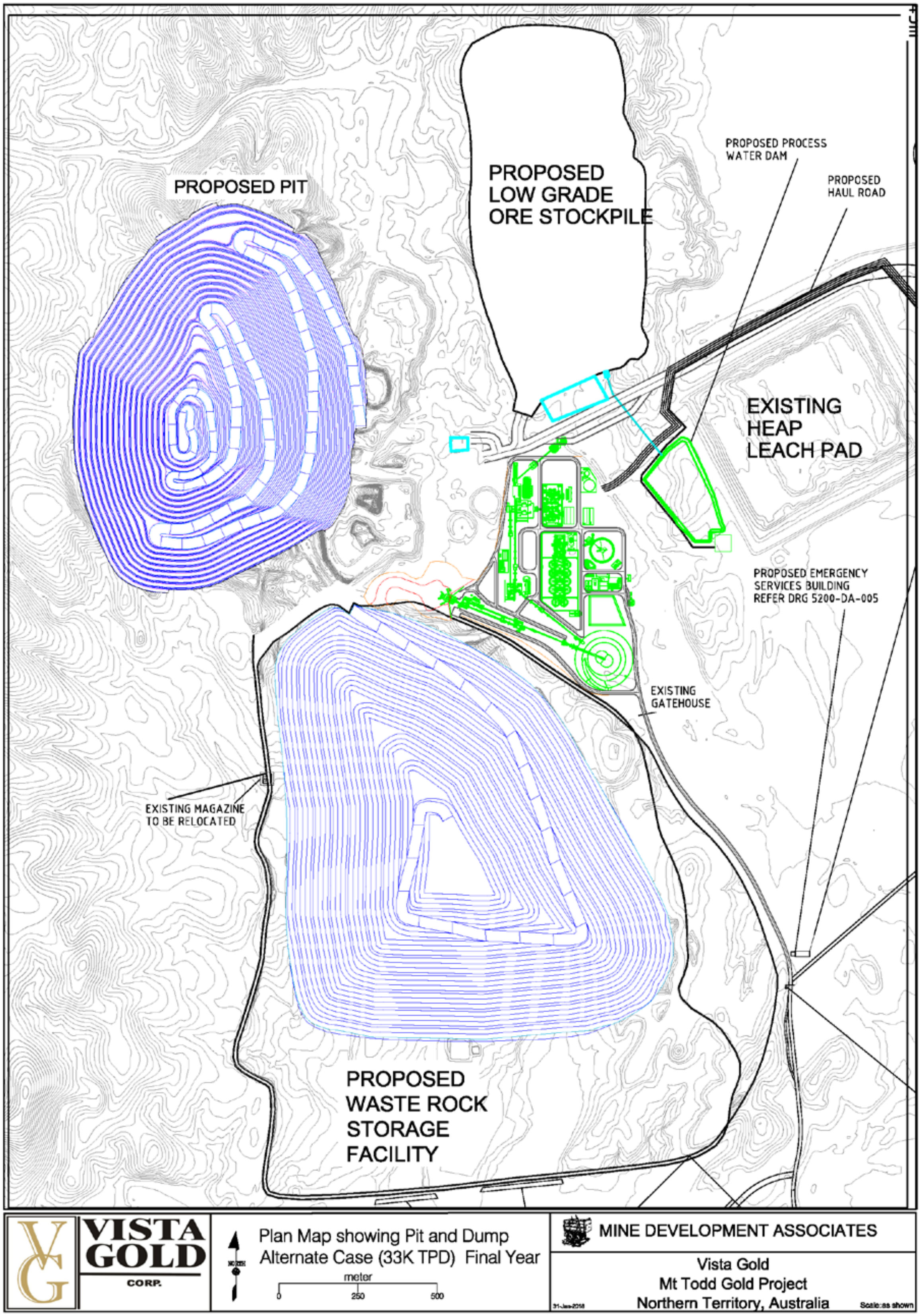


Figure 24-8: Mt Todd Ultimate Pit Design – Alternate Case (October 4, 2019)

Separate phase designs for the Alternate and Base Cases were created. For the Alternate Case, Phase 1 essentially continues the western wall down from that done by prior operators, and wraps the ramp around the pit clockwise from the south. Phase 2 expands the pit to the east, north, and south, maintaining a portion of the phase 1 west wall. The Phase 2 ramp is placed on the east wall and has a total of 5 switchbacks located in the north and south ends of the pit. For the Alternate case there is only one more phase of mining to achieve the ultimate pit. This phase mines 360 degrees around the phase 2 pit, deepening it to achieve the ultimate pit.

Figure 24-9 to **Figure 24-10** show the Alternate Case Phase I and II pit designs. The Alternate Case Phase IV design is depicted in **Figure 24-8** as the ultimate pit. Resulting reserves for each of the phases are shown in **Figure 24-9**.

Figure 24-9: Phase I Pit Design – Alternate Case (February 12, 2018)

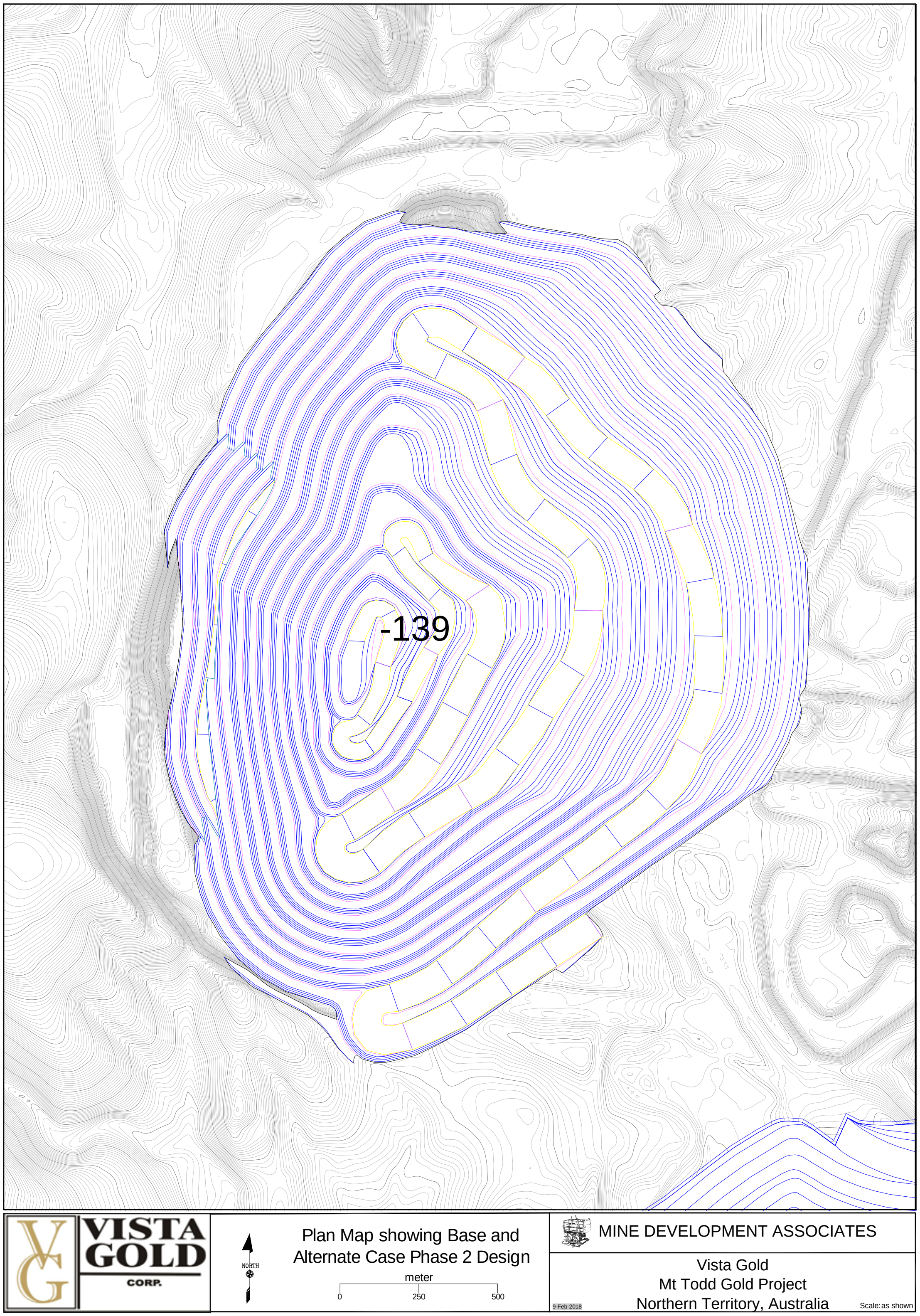


Figure 24-10: Phase II Pit Design – Alternate Case (February 12, 2018)

24.7.2.3 Cut-Off Grade

The breakeven and internal cutoff grades calculated using the economic parameters shown in **Table 24-13** are shown in **Table 24-15**. The internal cutoff grade assumes that mining is constrained to an economic pit and does not include the mining cost.

To enhance projects economics, Vista used a higher cutoff grade for reserves and scheduling than what operating costs would have predicted. Reserves are reported using a 0.40 g-Au/t cutoff grade for the Alternate Case.

Table 24-15: US$1,250 Calculated Gold Price Cutoff Grades (g-Au/t)

	Sulfide	Transition	Oxide
Breakeven	0.38	0.40	0.40
Internal	0.31	0.33	0.33
Cutoff Grade Used	0.40	0.40	0.40

For purposes of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used a 0.40g-Au/t cutoff grade. Medium-grade and high-grade material is defined using cutoffs of 0.55 and 0.85 g-Au/t.

24.7.2.4 Mineral Reserves

Mineral Reserves are shown in **Table 24-16**.

Note that these proven and probable Reserves are contained within the Base Case pit designs used to state the full official reserves. Thus, the Alternate Case Reserves are a subset of the official reserves.

Table 24-16: Alternate Case Proven and Probable Reserves by Phase

	Proven			Probable			Total P&P			Waste Total	Total Tonnes	Strip Ratio
	K Tonnes	g-Au/t	K Ozs Au	Tonnes	g-Au/t	K Ozs Au	Tonnes	g-Au/t	K Ozs Au			
Ph_1	13,551	1.09	473	6,245	1.10	221	19,796	1.09	694	19,312	39,109	0.98
Ph_2	18,980	0.80	490	19,008	0.88	538	37,988	0.84	1,028	59,167	97,155	1.56
Ph_3	21,500	0.91	626	35,375	0.86	977	56,874	0.88	1,603	114,836	171,710	2.02
Total	**54,031**	**0.91**	**1,589**	**60,628**	**0.89**	**1,736**	**114,658**	**0.90**	**3,325**	**193,316**	**307,974**	**1.69**

The 33,000 tpd reserves are reported using a cutoff grade of 0.40 g-Au/t and are a subset of the measured and indicated resources; inferred resources are considered waste.

MINE WASTE FACILITIES

Total contained waste tonnage is 193 Mt for the Alternate Case. The Alternate Case designed height is 234 meters. The south end of the Base Case dump was designed to encroach on the existing RP1 waste water storage facility. Designs are limited to the east by the process facility and to the west by another drainage basin. Designs are intended to promote any drainage to the current RP1 waste water retention pond.

MINE PRODUCTION SCHEDULE

Table 24-17 shows the mine-production schedule, including re-handle from stockpiles. For purpose of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used a cutoff grade of 0.40 g-Au/t. Medium-grade and high-grade cutoffs used were 0.55 and 0.85 g-Au/t.

Low-grade ore is processed as part of the commissioning of the mill. This assumes a ramp up to full production of 25%, 50%, 75%, and 87.5% of full production throughput through the first 4 months prior to start of full production. High-grade and medium-grade ore is processed in the mill when mill capacity becomes fully available.

As with the base case, final recoveries were estimated using a constant tail by range of grades for the processed material. The equation used to calculate the recovery based on the constant tail is:

$$\frac{Au_{grade} - Const.Tail_{grade}}{Au_{grade}}$$

The ranges for the constant tail, based on model grade input in g Au/t are:

- 0.20 to 0.40 = 0.04 g Au/t tail
- 0.40 to 0.60 = 0.05 g Au/t tail
- 0.60 to 0.80 = 0.06 g Au/t tail
- 0.80 to 1.00 = 0.08 g Au/t tail
- 1.00 to 1.50 = 0.10 g Au/t tail
- 1.50 and above = 0.13 g Au/t tail

The use of the constant tails resulted in higher final back calculated recoveries of ~92% for sulfide material, ~90% for transition material, and ~91% for oxide material.

Table 24-17: Annual Mine Production Schedule – Alternate Case

			Pre-Prod	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	Total
Total Mined	*StkPl	K Tonnes	120	2,613	4,014	3,971	10,640	3,744	2,333	3,193	3,537	9,071	47	-	-	-	43,282
		g-Au/t	0.69	0.62	0.72	0.47	0.64	0.60	0.47	0.47	0.47	0.73	0.50	-	-	-	0.61
		K Ozs Au	3	52	93	60	218	73	35	48	54	212	1	-	-	-	849
	Crusher	K Tonnes	-	4,433	9,147	7,009	11,715	5,958	2,193	6,450	10,792	11,715	1,965	-	-	-	71,376
		g-Au/t	-	0.99	1.41	0.78	1.02	1.13	0.72	0.88	0.97	1.30	1.41	-	-	-	1.08
		K Ozs Au	-	141	414	175	383	216	51	182	337	489	89	-	-	-	2,476
	Total Ore Mined	K Tonnes	120	7,046	13,160	10,980	22,355	9,703	4,526	9,643	14,329	20,786	2,012	-	-	-	114,658
		g-Au/t	0.69	0.85	1.20	0.67	0.84	0.92	0.59	0.74	0.85	1.05	1.38	-	-	-	0.90
		K Ozs Au	3	193	507	236	601	288	86	230	391	701	90	-	-	-	3,325
	NonPag_Wst	K Tonnes	873	996	7,298	13,018	2,861	17,645	12,435	4,707	311	2	-	-	-	-	60,148
	Pag_Wst	K Tonnes	794	9,835	6,107	13,553	13,524	10,363	12,168	13,481	10,922	5,259	2	-	-	-	96,008
	Un_Wst	K Tonnes	413	2,346	1,932	6,570	3,781	6,420	6,641	5,841	2,908	308	-	-	-	-	37,160
	Total Waste Mined	K Tonnes	2,081	13,177	15,337	33,141	20,166	34,428	31,244	24,029	14,141	5,569	2	-	-	-	193,316
	Total Tonnes Mined	K Tonnes	2,201	20,223	28,497	44,121	42,521	44,131	35,770	33,672	28,470	26,355	2,014	-	-	-	307,974
	Strip Ratio	W:O	17.31	1.87	1.17	3.02	0.90	3.55	6.90	2.49	0.99	0.27	0.00				1.69
Re-Handle Material	HG_StkPl	K Tonnes	-	528	957	-	-	1,978	-	-	-	-	2,178	-	-	-	5,642
		g-Au/t	-	1.08	1.30	-	-	1.19	-	-	-	-	1.16	-	-	-	1.19
		K Ozs Au	-	18	40	-	-	76	-	-	-	-	81	-	-	-	215
	MG_StkPl	K Tonnes	-	328	1,154	-	-	3,778	398	-	-	-	3,857	-	-	-	9,515
		g-Au/t	-	0.69	0.67	-	-	0.68	0.60	-	-	-	0.68	-	-	-	0.68
		K Ozs Au	-	7	25	-	-	82	8	-	-	-	85	-	-	-	207
	LG_StkPl	K Tonnes	-	1,311	458	4,671	-	-	9,125	5,297	923	-	3,715	2,626	-	-	28,126
		g-Au/t	-	0.48	0.52	0.48	-	-	0.48	0.45	0.53	-	0.49	0.42	-	-	0.47
		K Ozs Au	-	20	8	71	-	-	141	77	16	-	59	36	-	-	427
	Leach Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	9,121	4,233	-	13,354
		g-Au/t	-	-	-	-	-	-	-	-	-	-	-	0.54	0.54	-	0.54
		K Ozs Au	-	-	-	-	-	-	-	-	-	-	-	158	73	-	232
	Total Re-Handle	K Tonnes	-	2,167	2,568	4,671	-	5,757	9,522	5,297	923	-	9,750	11,747	4,233	-	56,636
		g-Au/t	-	0.66	0.88	0.48	-	0.85	0.49	0.45	0.53	-	0.72	0.51	0.54	-	0.59
		K Ozs Au	-	46	73	71	-	158	149	77	16	-	224	194	73	-	1,081
	Waste Re-handle	K Tonnes	0	1	0	-	-	-	-	-	214	5,283	-	-	315	13,675	19,490
	Sorter Rejects	K Tonnes	-	660	1,171	1,168	1,171	1,171	1,171	1,175	1,171	1,171	1,171	263	-	-	11,466
	Sorter Reject Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	11,466	11,466

Table 24-18: Annual Stockpile Balance – Alternate Case

			Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11
Hg_StkPl	Added	K Tonnes	24	524	937	-	1,485	493	-	-	-	2,178	-	-
		g-Au/t	1.06	1.08	1.31	-	1.17	1.24	-	-	-	1.16	-	-
		K Ozs Au	1	18	39	-	56	20	-	-	-	81	-	-
	Removed	K Tonnes	-	528	957	-	-	1,978	-	-	-	-	2,178	-
		g-Au/t	-	1.08	1.30	-	-	1.19	-	-	-	-	1.16	-
		K Ozs Au	-	18	40	-	-	76	-	-	-	-	81	-
	Balance	K Tonnes	24	20	-	-	1,485	-	-	-	-	2,178	-	-
		g-Au/t	1.06	0.99	-	-	1.17	-	-	-	-	1.16	-	-
		K Ozs Au	1	1	-	-	56	-	-	-	-	81	-	-
Mg_StkPl	Added	K Tonnes	55	374	1,052	-	3,573	603	-	-	-	3,857	-	-
		g-Au/t	0.69	0.67	0.67	-	0.67	0.67	-	-	-	0.68	-	-
		K Ozs Au	1	8	23	-	77	13	-	-	-	85	-	-
	Removed	K Tonnes	-	328	1,154	-	-	3,778	398	-	-	-	3,857	-
		g-Au/t	-	0.69	0.67	-	-	0.68	0.60	-	-	-	0.68	-
		K Ozs Au	-	7	25	-	-	82	8	-	-	-	85	-
	Balance	K Tonnes	55	102	-	-	3,573	398	-	-	-	3,857	-	-
		g-Au/t	0.69	0.63	-	-	0.67	0.60	-	-	-	0.68	-	-
		K Ozs Au	1	2	-	-	77	8	-	-	-	85	-	-
Lg_StkPl	Added	K Tonnes	41	1,714	2,025	3,971	5,582	2,648	2,333	3,193	3,537	3,036	47	-
		g-Au/t	0.48	0.47	0.47	0.47	0.48	0.47	0.47	0.47	0.47	0.48	0.50	-
		K Ozs Au	1	26	31	60	85	40	35	48	54	46	1	-
	Removed	K Tonnes	-	1,311	458	4,671	-	-	9,125	5,297	923	-	3,715	2,626
		g-Au/t	-	0.48	0.52	0.48	-	-	0.48	0.45	0.53	-	0.49	0.42
		K Ozs Au	-	20	8	71	-	-	141	77	16	-	59	36
	Balance	K Tonnes	41	444	2,011	1,311	6,893	9,541	2,750	646	3,259	6,295	2,626	-
		g-Au/t	0.48	0.46	0.46	0.44	0.47	0.47	0.42	0.42	0.45	0.46	0.42	-
		K Ozs Au	1	7	30	18	104	144	38	9	47	94	36	-
All StkPl	Balance	K Tonnes	120	565	2,011	1,311	11,951	9,939	2,750	646	3,259	12,330	2,626	-
		g-Au/t	0.69	0.51	0.46	0.44	0.62	0.47	0.42	0.42	0.45	0.65	0.42	-
		K Ozs Au	3	9	30	18	236	151	38	9	47	260	36	-

Table 24-19: Annual Ore Delivery to the Mill Crusher – Alternate Case

		Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Total
Sulfide Ore	K Tonnes	-	6,600	11,644	11,560	11,600	11,330	11,204	11,625	11,715	11,715	11,714	2,604	113,311
	g-Au/t	-	0.88	1.30	0.66	1.02	1.00	0.53	0.69	0.94	1.30	0.83	0.42	0.91
	K Ozs Au	-	187	485	245	380	365	191	257	353	489	313	36	3,300
	Recovery	0%	92%	93%	91%	92%	92%	90%	91%	92%	93%	92%	88%	92%
	K Ozs Au Rec	-	171	450	223	350	336	172	234	324	453	287	31	3,031
Mixed Ore	K Tonnes	-	-	30	82	68	359	420	99	-	-	1	18	1,077
	g-Au/t	-	-	0.67	0.45	0.86	0.74	0.49	0.42	-	-	0.42	0.42	0.59
	K Ozs Au	-	-	1	1	2	8	7	1	-	-	0	0	20
	Recovery	0%	0%	91%	89%	92%	91%	90%	88%	0%	0%	88%	88%	90%
	K Ozs Au Rec	-	-	1	1	2	8	6	1	-	-	0	0	18
Oxidized Ore	K Tonnes	-	-	41	38	47	26	91	23	-	-	0	4	270
	g-Au/t	-	-	0.69	0.45	1.11	0.63	0.48	0.42	-	-	0.42	0.42	0.63
	K Ozs Au	-	-	1	1	2	1	1	0	-	-	0	0	5
	Recovery	0%	0%	91%	89%	92%	91%	90%	88%	0%	0%	88%	88%	91%
	K Ozs Au Rec	-	-	1	0	2	0	1	0	-	-	0	0	5
Total Ore	K Tonnes	-	6,600	11,715	11,680	11,715	11,715	11,715	11,747	11,715	11,715	11,715	2,626	114,658
	g-Au/t	-	0.88	1.29	0.66	1.02	0.99	0.53	0.69	0.94	1.30	0.83	0.42	0.90
	K Ozs Au	-	187	487	247	383	374	199	259	353	489	313	36	3,325
	Recovery	0%	92%	93%	91%	92%	92%	90%	91%	92%	93%	92%	88%	92%
	K Ozs Au Rec	-	171	452	224	353	344	180	235	324	453	287	32	3,054

For the purpose of scheduling, three ore stockpiles are assumed: High-grade ore stockpile for high-grade ore; medium-grade stockpile for medium-grade ore; and a low-grade stockpile for low-grade ore. The high-grade and medium-grade stockpiles are to be built within the low-grade stockpiling areas but are exhausted during the first year of processing when mill capacity becomes available. During the LoM, the low-grade stockpile is used as needed to feed the mill to full capacity. For this reason the stockpile grows and shrinks through the LoM. The maximum stockpile balance through the LoM is estimated to be 13.5 Mt.

Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. **Table 24-18** shows the Alternate Case ore stockpile balances for the end of each year. Waste re-handle is shown on the bottom of **Table 24-17** to account for capping and reclamation.

Ore sent to the mill is shown in **Table 24-19**. This is a combination of ore shipped directly from the mine and ore that is reclaimed from stockpiles. These tables summarize the ore based on level of oxidation. The recovered ounces shown are based on the recoveries used for pit optimizations and are subject to change by qualified persons completing the metallurgical sections

EQUIPMENT SELECTION AND PRODUCTIVITIES

Availability, efficiencies, operating hours and load and haul equipment requirements are shown in **Table 24-17**.

Table 24-20: Annual Load and Haul Equipment Requirements – Alternate Case

	Units	Pre-Prod	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	Total
HAULAGE REQUIREMENTS																
Productive Hours	Hrs	2,036	26,551	45,585	76,850	81,755	74,126	74,086	77,812	69,228	79,316	13,311	9,135	3,753	30,087	663,633
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	1743%
Operating Hours	Hrs	2,453	31,989	54,921	92,591	98,500	89,309	89,260	93,750	83,408	95,561	16,038	11,006	4,522	36,249	799,558
Number of Trucks	#	2	6	10	16	18	18	18	18	18	18	6	6	6	6	166
Truck Availability	%	90%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	
Available Operating Hours	Hrs	3,173	36,884	61,415	97,850	114,276	112,966	111,346	111,659	111,346	111,346	37,115	37,220	18,035	37,115	1,001,746
Use of Available Hours	%	77%	87%	89%	95%	86%	79%	80%	84%	75%	86%	43%	30%	25%	98%	80%
Tonnes per Operating Hour	t/Hr	898	700	566	527	432	559	507	416	355	331	734	1,067	1,006	377	480
HYDRAULIC SHOVEL USAGE																
Number of Shovels	#	1	2	2	2	2	2	2	2	2	2	1	1	1	1	2
Availability	%	90.0%	89.8%	89.3%	88.3%	87.3%	86.3%	85.3%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	86.5%
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83.0%
Available Operating Hrs	Op Hrs	1,587	8,119	12,991	12,882	12,700	12,555	12,421	12,407	12,372	12,372	6,186	6,203	3,006	6,186	131,986
Tonnes Mined	K Tonnes	2,201	19,722	27,477	42,089	40,820	43,183	35,055	32,325	27,331	25,301	7,800	9,398	3,639	20,113	336,454
Operating Hours	Op Hrs	561	5,024	7,000	10,723	10,399	11,001	8,931	8,235	6,963	6,446	1,987	2,394	927	5,124	85,716
Use of Available Operating Hours	%	35%	62%	54%	83%	82%	88%	72%	66%	56%	52%	32%	39%	31%	83%	65%
FRONT END LOADERS																
Number of Loaders	#	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Availability	%	0%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	86%
Operating Efficiency	%	0.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%
Available Operating Hrs	Op Hrs	-	5,480	6,489	6,441	6,337	6,265	6,186	6,203	6,186	6,186	6,186	6,203	3,006	6,186	77,353
Tonnes Mined	K Tonnes	-	2,670	3,588	6,703	1,701	6,705	10,238	6,644	2,276	6,337	3,964	2,349	910	5,028	59,113
Operating Hours	Op Hrs	-	1,590	2,137	3,991	1,013	3,993	6,097	3,956	1,356	3,774	2,360	1,399	542	2,994	35,202
Use of Available Operating Hours	%	0%	29%	33%	62%	16%	64%	99%	64%	22%	61%	38%	23%	18%	48%	46%

MINE PERSONNEL

Salaries for each position were estimated based on information received from Tetra Tech and Vista. Salaries include an allowance for benefits at a rate of 27% of the base salary for each position. Note that mobile equipment labor costs are allocated to production equipment in the calculation of mining costs in later sections. In addition, a portion of the cost is allocated to construction of tailings facilities.

24.7.3 Process Facility

24.7.3.1 Design Criteria

The nominal headline design criteria are listed are shown in **Table 24-21**.

Table 24-21: Headline Design Criteria

	Unit	33,000 tpd
Annual Ore Feed Rate	Mt/a	11.72
Operating Days per Year	d/y	355
Daily Ore Feed Rate	t/d	33,000
Crushing Rate (6,637 hours per year availability)	tph	1,765
HPGR Rate (7,838 hours per year)	tph	1,495
Ore Sorting Rate (7,838 hours per year)	tph	318
Milling Rate (7,838 hours per year)	tph	1,332
Gold Head Grade	g/t	0.82
Copper Head Grade	%	0.055
Cyanide Soluble Copper	%	0.0024
Ore Specific Gravity		2.76
Primary Grind P_{80} to Secondary Grind	µm	250
Grind P_{80} to Leach	µm	40
Gold Recovery	%	91.9
Gold Production (average)	oz/d	758
Gold Production (average)	oz/a	287,822

FLOWSHEET

The Alternate Case flowsheet is the same as that of the Base Case, as shown in **Figure 17-1** with the following equipment differences:

- Cone Crushers: 2- Raptor 900 secondary crushers;
- HPGRs: 2- HPGR Polycom PM7-20/15, each equipped with 2 x 1,800 kW drives;
- Above-ground Pre-Leach Thickener: 55m diameter; and
- Leach/CIP: Carbon movement requirements will be in the order of 20 tpd.

24.7.4 Infrastructure

Project support infrastructure and services are similar to the Base Case, but with the following differences:

24.7.4.1 Area 2300 – Mine Support Facilities

- Sub-Area 2305 – Support Facilities – HV Workshop / Warehouse
  - The HV Workshop will reduce in size for the Alternate Case from six bays to four bays, resulting in a footprint of 59.3 m by 24.4 m.
  - There will be no difference in size between the Base Case and the Alternate Case for the Warehouse.
- Sub-Area 2310 – Support Facilities – Bulk Fuel Storage
  - The bulk fuel storage for the Alternate Case will require four bowsers as opposed to six.
- Sub-Area 2335 – Support Facilities – Lube Storage
  - The Lube Storage area for the Alternate Case will have a reduced footprint for the storage of lubricants.

24.7.4.2 Area 4000 – Project Services

- Sub-Area 4130 – Potable Water
  - Water supply to the project does not differ greatly between the two cases except that the water treatment plant is sized to handle a smaller throughput and several supply pumps will be decreased in size.
  - Raw water will be brought down to the raw water tank in the process plant through the existing 400 mm poly line at the raw water dam.
- Sub-Area 4231 – Power Distribution
  - For the 33,000 tpd case, the current carrying capacity of the buried cable between the overhead power lines and the Process Plant will be decreased to suit the reduction in plant load.
- Sub-Area 4232 – Overhead Power Lines
  - For the 33,000 tpd case, the current carrying capacity of the overhead power lines between the Power Station and the Process Plant will be decreased to suit the reduction in plant load.

AREA 4300 – COMMUNICATIONS

- Sub-Area 4310 – Fiber Optic
  - The Alternate Case will essentially be the same as the Base Case.
- Sub-Area 4311 – Phones
  - The Alternate Case will essentially be the same as the Base Case. The only difference may be the number of telephone handsets required due to a decrease in site personnel.
- Sub-Area 4313 – Telemetry
  - The Alternate Case will essentially be the same as the Base Case.

AREA 4500 – PLANT MOBILE EQUIPMENT

- The plant mobile equipment to be purchased for the Alternate Case will not differ from the Base Case.

24.7.4.3 Area 5000 – Project Infrastructure

AREA 5100 – SITE PREPARATION

- The site footprint for the 33,000 tpd case will be smaller than the 50,000 tpd case and as a result will have reduced earthworks, road works and drainage quantities.

SUB-AREA 5230 – REAGENT STORE

- The Alternate Case will require a reduced inventory of reagents and therefore the footprint of the Reagents Yard will be reduced to an area of 1,350 square meters (m^2).

SUB-AREA 5260 – SAMPLE PREPARATION AND LABORATORY

- The Alternate Case requires a reduced inventory of sample processing laboratory equipment to process 300 samples/day.

AREA 5400 – HEAVY LIFT CRANAGE

- The heavy lift craneage durations will be reduced for the Alternate Case based on a reduced Structural, Mechanical and Piping (SMP) contract schedule.

24.7.4.4 Area 6100 – Personnel Transport

The bus transit area for the Alternate Case will be identical to that proposed for the Base Case.

24.7.4.5 Area 7300 – Construction Camp

The Construction Camp for the Alternate Case will be sized for 350 construction workers based on the manning histogram developed for the Project. There will be a reduction in footprint and associated infrastructure, establishing access roads and site works, contractor preliminaries, transportable buildings and services.

24.7.5 Site-wide Water Balance Model

The Alternate Case site-wide water balance model was developed using scaled flows of the Base Case water balance. Differences between the Alternate Case and the Base Case models are:

- Total process plant makeup water for the Alternative Case is 1,164 m^3/hr versus 1,764 m^3/hr;
- Alternate Case production occurs over 11 years versus 13 years;
- Process circuit bleed stream is 151 m^3/hr for the Alternate Case and 229 m^3/hr for the Base Case;
- RWD process makeup water flows were 200 m^3/hr for the Alternate Case and 304 m^3/hr for the Base Case;

- Maximum draw from the RWD for make-up water in the Alternate Case is 7,890 m^3/day versus 11,955 m^3/day; and
- Other water requirements (TSF decant, potable, reagent mixing, gland, etc.) are consistent with lower production rates.

24.7.6 Capital Costs, Alternate Case

LoM capital cost requirements are estimated at US$874 million as summarized in **Table 21-2**. Initial capital of US$623 million is estimated to be required to commence operations. At the end of operations, the Project will receive a US$86 million credit for remaining asset sales and salvage.

Table 24-22: Estimated Alternate Case Capital Cost Summary (US$000s)

Area	Description	Cont. (%)	Initial Capital (US$000s)			Sustaining Capital (US$000s)			Total Capital (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
2000	Mining	8.3%	$58,218	$2,824	**$61,042**	$259,218	$23,424	**$282,641**	$317,436	$26,248	**$343,684**
3000	Process Plant	14.6%	$279,747	$40,780	**$320,527**	$7,886	$1,092	**$8,978**	$287,633	$41,872	**$329,505**
4000	Project Services	10.6%	$90,349	$10,859	**$101,208**	$42,256	$3,179	**$45,435**	$132,605	$14,038	**$146,643**
5000	Project Infrastructure	13.2%	$24,635	$3,246	**$27,881**	$0	$0	**$0**	$24,635	$3,246	**$27,881**
6000	Permanent Accommodation	10.0%	$60	$6	**$66**	$0	$0	**$0**	$60	$6	**$66**
7000	Site Establishment & Early Works	11.4%	$16,534	$1,889	**$18,423**	$0	$0	**$0**	$16,534	$1,889	**$18,423**
8000	Management, Engineering, EPCM Services	11.6%	$71,269	$8,279	**$79,549**	$0	$0	**$0**	$71,269	$8,279	**$79,549**
9000	Pre-Production Costs	11.4%	$13,224	$1,512	**$14,736**	$0	$0	**$0**	$13,224	$1,512	**$14,736**
10000	Asset Sale	0.0%	$0	$0	**$0**	($86,279)	$0	**($86,279)**	($86,279)	$0	**($86,279)**
	Capital Cost	**12.5%**	**$554,036**	**$69,396**	**$623,432**	**$223,080**	**$27,695**	**$250,775**	**$777,117**	**$97,091**	**$874,207**

24.7.7 Operating Costs, Alternate Case

LoM operating cost estimates are summarized in **Table 24-23**. The operating costs will average US$14.99 over the LoM.

Table 24-23: Estimated Operating Cost Summary, Alternate Case (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.12	0.05
Mine Maintenance	0.16	0.07
Engineering	0.07	0.03
Geology	0.05	0.02
Drilling	0.56	0.23
Blasting	0.83	0.35
Loading	0.47	0.20
Hauling	1.73	0.72
Mine Support	0.50	0.21
Mine Dewatering	0.02	0.007
Open Pit Mine	**4.52**	**1.88**
CIP PROCESS PLANT		
Labor	1.19	-
3100-Crush/Screen/Stockpile	0.23	-
3200-Reclaim & HPGR	0.50	-
3300-Classification & Grinding	3.21	-
3400-Pre-Leach,Thick/Aeration/CIP	0.15	-
3500-Desorption, Gold Room	0.03	-
3600-Detox & Tailings Pumping	0.07	-
3700-Reagents	3.00	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.01	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**8.51**	**-**
Project Services	$0.17	-
G&A	$1.79	-
Operating Costs	**$14.99**	**-**

24.7.8 Economic Results, Alternate Case

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 ±25%.

Economic results are summarized in **Table 24-24**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,350:

- Mine Life: 11 years;
- Pre-Tax NPV5%: US$884 million, IRR: 25.7%;
- After-tax NPV5%: US$510 million, IRR: 19.8%;
- Payback (After-tax): 3.8 years;
- NT Taxes Paid: US$285 million;
- Australian Income Taxes Paid: US$316 million; and
- Cash costs (including JAAC Royalty): US$603.79/oz-Au.

Table 24-24: Economic Results, Alternate Case (US$000s)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Gold Sales			
Gold Produced (koz)	3,232	-	-
Gold Price (US$/oz)	1,350	-	-
Gold Sales	**4,363,271**	**34.08**	**1,350**
Refining & Royalties			
Refinery Costs	(10,641)	(0.083)	(3.292)
JAAC Royalty	(43,633)	(0.341)	(13.50)
Gross Income from Mining	**4,308,997**	**(33.661)**	**1,333**
Operating Costs			
Open Pit Mine	(578,421)	(4.52)	(179)
CIP Process Plant	(1,089,355)	(8.51)	(337)
Project Services	(21,777)	(0.17)	(7)
G&A	(228,808)	(1.79)	(71)
Operating Costs	**(1,918,361)**	**(14.99)**	**(594)**
Power Sales Credit	21,156	0.17	**7**
Cash Cost of Goods Sold (COGS)	**(1,951,479)**	**(15.24)**	**(604)**
Operating Margin	**2,411,792**	**18.84**	**746**
Capital Costs			
Mining	343,684		
Process Plant	329,505		
Project Services	146,643		
Project Infrastructure	27,881		
Permanent Accommodation	66		
Site Establishment & Early Works	18,423		
Management, Engineering, EPCM Services	79,549		

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Pre-Production Costs	14,736		
Asset Sale	(86,279)		
Capital Costs	**874,207**		
Pre-Tax Cash Flow	1,532,585		
$NPV_{5\%}$	884,337		
IRR (%)	25.7%		
After-tax Cash Flow	931,075		
$NPV_{5\%}$	509,611		
IRR (%)	19.8%		
Post- Tax Payback (years)	3.8		

Cash costs for the Project are presented in **Table 24-25**.

Table 24-25: All-In Sustaining Costs (US$/oz)

Period	Cash Cost	Sustaining	AISC
First 60 Mo. Of Prod.	USD 588.10	USD 100.98	USD 689.08
LoM	USD 603.79	USD 104.28	USD 708.07

Cash costs would typically include Non-cash remuneration for site personnel and AISC would include corporate G&A (including share-based remuneration).

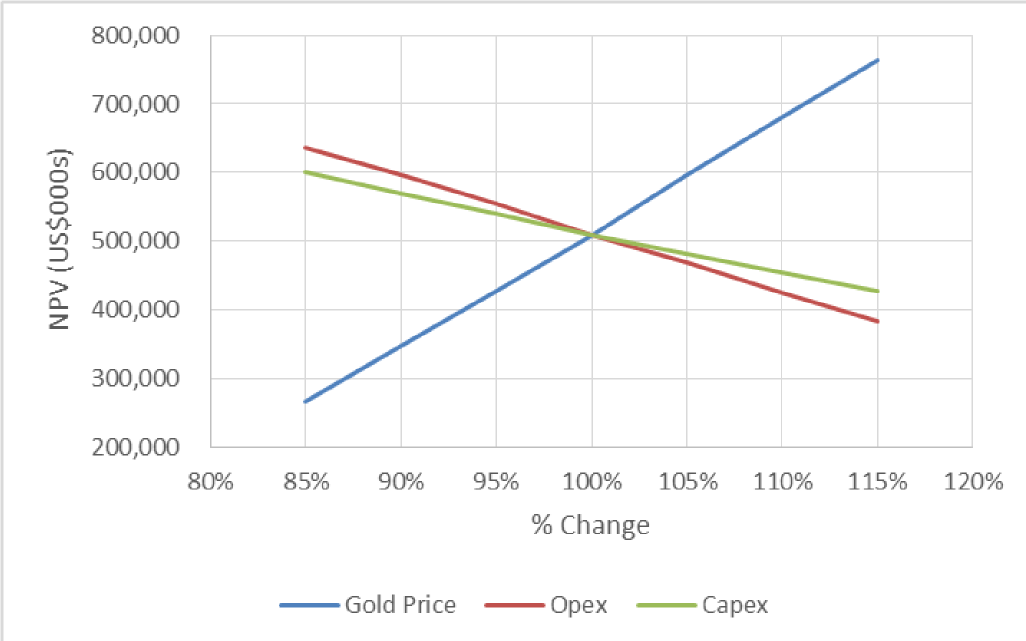
For cash costs, non-cash remuneration could be defined to include share-based comp for site personnel. We have not included the model's $780k per year. This expense has no impact on cashflows that generate NPV and IRR and is only about $1.50/oz (after tax) LOM. This is not a material deviation from the World Gold Counsel's (WGCs) definition.

As to AISC, the model taxes an accrual-based tax deduction of $1.2 million per year but does not deduct such expense from cashflows, so there in no impact on NPV or IRR except to the extent this expense reduces NT royalty and income taxes. This would equate to about $2/oz, which is not material.

Project sensitivities are summarized in **Table 24-5**, **Table 24-6**, and **Table 24-28**; sensitivities are shown graphically in **Figure 24-11**. As seen, the Project is most sensitive to gold production and gold price. Sensitivity on operating and capital cost is closely matched, with the Project being only slightly more sensitive to capital costs.

Table 24-26: Project Sensitivity

Parameter	85%	90%	95%	Base	105%	110%	115%
Gold Price	265,062	348,030	427,817	509,611	596,107	679,528	763,304
Opex	636,196	595,590	554,690	509,611	469,134	425,633	384,118
Capex	600,510	569,763	539,592	509,611	481,713	453,899	426,084



Source: JDS Energy & Mining, September 9, 2019

Figure 24-11: Project Sensitivity

Table 24-27: Alternate Case Sensitivity to Gold Price versus Foreign Exchange Rate (US$:AUD)

	GOLD PRICE (US$/oz-Au)									
	$1,200		$1,300		$1,350		$1,400		$1,500	
FEX Rate	**IRR**	**$NPV_{5\%}$**	**IRR**	**$NPV_{5\%}$**	**IRR**	**$NPV_{5\%}$**	**IRR**	**$NPV_{5\%}$**	**IRR**	**$NPV_{5\%}$**
0.60	18.8%	436	22.3%	560	24.1%	622	25.8%	684	29.1%	809
0.65	16.8%	382	20.1%	503	21.9%	567	23.6%	629	26.8%	753
0.70	14.9%	330	18.2%	451	**19.8%**	**510**	21.5%	575	24.7%	698
0.75	12.9%	275	16.3%	397	17.9%	458	19.5%	517	22.7%	643
0.80	10.8%	212	14.6%	344	16.2%	404	17.7%	463	20.7%	585

Alternate Case uses a foreign exchange rate of US$0.70:AUD1.00 and a gold price of $1,350/oz-Au.

Table 24-28: Alternate Case Sensitivity to Gold Prices versus NPV Discount Rate

	GOLD PRICE (US$/oz-Au)						
Discount Rate	**$ 1,100**	**$ 1,200**	**$ 1,300**	**$ 1,350**	**$ 1,400**	**$ 1,500**	**$ 1,600**
5%	199	330	451	510	575	698	822
8%	82	194	294	342	396	498	600
10%	22	124	212	255	303	393	483

Foreign Exchange is held constant at US$0.70:AUD1.00

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Project Risks

Significant risks and uncertainties that could reasonably affect the reliability or confidence in the Project outcome are provided in **Table 25-1**.

The Project is an advanced-staged development project that has undergone engineering and permitting for a number of years. In order to manage cost and schedule risk, Vista retained GR Engineering Services of Perth, Australia to undertake a benchmarking study to assess the appropriateness of capital and operating cost estimates, construction and ramp-up schedules, owner's costs and key components of the Project (e.g., power supply). As such, the development risks that are within the control of Vista are considered low to moderate.

Table 25-1: Project Risks

Risk	Description	Probability	Severity
Gold Price	The Project economics are sensitive to gold price. Sustained downward gold price trends could render the project uneconomic.	Low-Medium	High
Foreign Exchange	The Project capital and operating costs are sensitive to foreign exchange changes. A strengthen Australian dollar without an offsetting positive change in the gold price could render the Project uneconomic.	Low-Medium	High
Political Setting	Australia and the Northern Territory have historically been supportive of the extractive industries. Changes in legislation could have a negative impact on the project.	Low	Medium
Jawoyn	The JAAC is supportive of the Project. Changes in Vista's relationship with JAAC could have social impacts on the Project.	Low	Low-Medium
Permitting & Regulatory Approvals	The Project has received both EIS and EPBC authorizations as described in Section 20.	Low	Medium
Property Holdings	Vista has secured the Mt Todd concession holdings as described in Section 4.0. Any change could have negative impacts to the Project.	Low	Low
Infrastructure	The Project relies on the use of existing infrastructure. The condition of which is well known and is functional. Significant deficiencies would result in increased capital expense.	Low	Low
Understanding of Resource	The Project viability relies upon historical drilling as well as recent drilling to develop and assess the resource model. New drill results could adversely affect the interpretation of parts of the deposit, with impacts to resources and production estimates.	Low	Low
Power Plant Estimated Capital	The proposed power plant utilizes industry standard equipment that is currently in use in Australia. Changes in cost could affect Project economics.	Low	Low-Medium
Reagents & Consumables	The process operating costs are sensitive to global changes in reagents and consumables pricing.	Medium	Medium

Risk	Description	Probability	Severity
Fuel	The Project operating costs are sensitive to global changes in prices for diesel and natural gas.	Medium	Medium
Mobile Equipment Capital	Mobile equipment prices are an important part of the Project capital. Significant increases could impact the Project economics.	Low	Low-Medium
Process Technology	Extensive testing has been completed to identify the most suitable technology and equipment in the process are. The performance of the selected equipment could negatively impact Project economics.	Low-Medium	Medium
Climatic Events	Day to day mining operations could be significantly impacted by high precipitation events.	Low	Low-Medium
Groundwater	Day to day mining operations could be impacted by groundwater inflow.	Low	Low
Water Treatment	Heavy and sustained rains could result in water treatment in excess of capacity for short periods.	Low-Medium	Medium
Existing TSF 1	Restarting of TSF 1 operations is an integral part of the Project plan. This facility has been idle for many years, delays could impact the schedule.	Low	Low-Medium
Reclamation & Closure	There is potential for reclamation activities to extend beyond the active planned closure period, and therefore generate greater sustaining costs. Additional risk lies should the closure design not perform as intended.	Low	Medium

25.2 Geology and Resources

- The Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline which is comprised of the Burrell Creek Formation, the Tollis Formation, and the Kombolgie Formation.
- Gold mineralization in this area is constrained to a single mineralization event and the deposits are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit has characteristics of an intrusion related gold system making it the primary resource.
- The Batman deposit is defined by approximately 7.4 million ounces (Moz) of gold within 278 Mt of measured and indicated resource at an average grade of 0.82 g-Au/t and a cutoff grade of 0.4 g-Au/t as provided in **Table 14-1**.

In addition, opportunities for the Project resource may include:

- In addition to the mineral reserves at the Batman Deposit, the Company estimates measured and indicated resources of 1.7 million ounces gold (70.2 million tonnes at 0.77 g Au/t) and inferred resources of 1.4 million ounces gold (61.3 million tonnes at 0.72 g Au/t). A portion of the inferred resources are contained within the existing pit design and are currently included in the mine plan as waste material. Additional resources are predominantly at depth and lateral along strike. Potential to convert part of the mineral resources to reserves represents an opportunity to improve existing LOM economics and extend mine life.
- The Company also has known mineral resources at the Quigleys Deposit, which is close to the planned processing plant. The estimated grade of the Quigleys Deposit is higher than the

estimated average grade of the Batman Deposit and could provide a source of higher-grade feed in the mid years of the Project when higher stripping is encountered and the average grade of feed to the plant is expected to decrease. Additional drilling and metallurgical testing are required to develop mine plans and ultimately convert part of the Quigleys resource to proven or probable reserves.

- Growth through exploration represents additional opportunity to add value at Mt Todd. Both the Batman Deposit and Quigleys Deposit remain open. In addition, Vista controls over 1,100 sq. km of contiguous exploration licenses at the southeast end of the Pine Creek Mining District. Various gold targets have been identified through early-stage, grass roots exploration programs along the Cullen-Australis and Batman-Driffield structural corridors, the latter of which is the host to the Batman Deposit. To-date, Vista's exploration efforts have primarily focused on the Batman Deposit.

25.3 Mineral Reserve and Mine Planning

- Pit designs were completed based on Whittle™ pit optimizations and are appropriate for metal prices of approximately US$800 per ounce Au for the Alternate Case and US$1,000 per ounce Au for the Base Case. The Mt Todd proven and probable reserves have been defined using economics based on a gold price of US$1,250 per ounce and an elevated cutoff grade of 0.40 g-Au/t. The proven and probable reserves were used to create a production schedule for mining, and a positive cash-flow analysis has been done based on the production schedule by Tetra Tech. The reserves have reasonable economics with respect to the statement of reserves under NI 43-101 regulations.
- Mine production constraints were imposed to ensure that mining wasn't overly aggressive with respect to the equipment anticipated for use at Mt Todd. The schedule has been produced using mill targets and stockpiling strategies to enhance the project economics. The constraints and limits are reasonable to support the project economics which are used to justify the statement of reserves.
- Pit designs use six-meter benches for mining. This corresponds to the resource model block heights, and MDA believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining. In areas where the material is consistently ore or waste so that dilution is not an issue, benches may be mined in 12-m heights.

25.4 Mineral Processing

The substantial quantity and quality of metallurgical test work data developed from Mt Todd drill core samples has led to the development of a robust energy efficient comminution circuit followed by a standard gold recovery process. Key conclusions drawn from the metallurgy studies are:

- Mt Todd (Batman) ore is among the hardest and most competent ore types processed for mineral recovery. The most energy efficient comminution circuit has been determined to be the sequence of primary crushing, closed circuit secondary crushing, and closed circuit HPGR tertiary crushing followed by ball milling.
- The ore is free-milling, is not preg-robbing, and is amenable to gold extraction by conventional cyanidation processes.

- The ore has relatively high specific cyanide consumption. This is largely due to the presence of sulfides, cyanide consuming copper, and destruction of residual cyanide.
- The use of sorting has helped to decrease operational costs and remove portions of the harder rock mined.

The equipment selection criteria for the Base Case operation has received considerable interaction with specialist vendors to the point where there is a reasonably high degree of confidence in selected technology and process units at this preliminary feasibility study stage. The recommended flowsheet for FS consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using HPGRs, ball milling, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detox and disposal to conventional tailings storage facility.

25.5 Infrastructure

25.5.1 Site Preparation

Bulk earthworks are designed to minimize the import of fill materials.

25.5.2 Support Buildings

- Administration offices, gatehouse/security facilities, cribs/ablutions are planned to be transportable buildings.
- The process plant offices, workshop and warehouse are located inside the existing Flotation Building.
- Sample preparation and laboratory will have a purpose-built steel shed.

25.5.3 Access Roads Parking and Laydown

The access road is based on the repaired existing road.

25.5.4 Heavy Lifts

Heavy cranage is allowed for all lifts greater than 50 t.

25.5.5 Bulk Transport

All bulk transport will be weighed.

25.5.6 Communications

Site-wide communication is based on a 50 m tall communication tower that will support eight (8) channels.

25.6 Project Services

The economic model uses a natural gas price derived from east coast gas pricing. The Company believes that there may be a significant opportunity to achieve a lower gas price upon commitment to a long-term gas delivery contract. This belief is in part based on local expectations of significantly increased gas reserves in the Beetaloo Basin south of the Mt Todd project. The Company is also considering additional optimization of the power plant.

25.7 Environmental and Social Conclusions

25.7.1 Existing Body of Work

A number of environmental studies have been conducted at the Project Site in support of development of Environmental Impact Statements and as required for environmental and operational permits. Studies conducted have investigated soils, climate and meteorology, geology, geochemistry, biological resources, cultural and anthropological sites, socio-economics, hydrogeology, and water quality.

25.7.2 Environmental Impact Study and Approvals

The Environmental Impact Study ("EIS") was submitted in June 2013. The NT Environmental Protection Authority provided its final assessment of the Project in June 2014. Notification of approval of the EIS was given September 2014.

Vista has received all major environmental approvals to proceed with the Project.

25.7.3 Social or Community Impacts

The Jawoyn people have strong involvement in the planning of the Project. Areas of aboriginal significance have been designated, and the mine plan has avoided development in these restricted works areas.

25.8 Results of the Site-wide Water Balance Model

- The WTP rate of 500 m^3/hr and PWP sizing for 6 days of storage was determined to be appropriate for the 50,000 tpd production process water requirements.
- The greatest amount of make-up water required from the RWD was quantified as 11,955 m^3/day. RWD requirements were found to be the most dependent upon TSF decant volumes.
- The WRD retention pond (RP1) was typically observed to overtop less than 1% of the time during the 13-year simulation.[1] LGRP storage may be optimized.

[1] A typical value is given. Separate model runs provide a range of overtopping events, due to the stochastic nature of the model.

26.0 RECOMMENDATIONS

The following recommendations are presented for consideration and were developed using best engineering judgement to estimate the costs of the respective recommendations.

26.1 Feasibility Study

A FS is recommended to advance the Project to a place where any additional detailed information necessary provides support of capital and operating cost estimates which lead to a potential project development decision.

The estimated budget for the FS is approximately US$2,500,000.

26.2 Resource and Exploration

- The Batman deposit potentially extends along strike both to the north and south. Step out drilling should be used to explore this.
- Additional deep infill drilling should be used to help define potential deep mineralization not detected by earlier drillholes.
- Infill drilling within and exploration drillholes along the trend of the Quigley deposit is recommended.
- Additional drilling exploring the exploration licenses, following up on geophysical and geochemical anomalies.

The estimated budget for drilling within the mining licenses is US$500,000-1,000,000 and US$500,000-1,000,000 for initial drilling on the exploration licenses.

26.3 Mining Risks and Opportunities

26.3.1 Opportunities

Some refinement of dump designs for the chosen ultimate pit may help to reduce the overall footprint of the resulting dumps and therefore reduce closure costs.

Current blasting patterns have been tightened up to reduce oversize. With experience, the blasting patterns can be optimized to reduce both drilling and blasting costs.

26.3.2 Risks

Large stockpiles of low-grade ore are used for reasonably long periods of time. These stockpiles may oxidize during storage and the resulting recoveries from these stockpiles may be overstated due to the change of chemistry.

26.4 Environmental Studies

Additional studies will be needed to further assess environmental baseline conditions to support feasibility level design, permitting, and closure planning for the Project, including:

- Erosion analyses;
- Waste and cover material hydraulic properties characterization and analysis;
- Acid-base accounting on waste rock and tailings;
- Ongoing aquatic, benthic and wildlife studies;
- Comprehensive vegetation survey;
- Archaeological and historical assessments for all areas to be disturbed;
- Hydrogeologic investigations and site-wide hydrogeologic characterization; and
- Precipitation, stream flow, and watershed data.

It is also recommended that further investigation be conducted to identify a source of low-permeability material suitable for use in closure covers located closer to or within the project site boundaries.

The estimated budget for this work is US$350,000.

26.5 Site-wide Water Balance

Recommended model improvements include:

- The site-wide water balance model is dependent upon the TSF water balance model, which provides decant water to the process facility, and the vadose and seepage models, which characterize seepage through the various rock piles on site (WRD, LGOS, and HLP). As such, completion of these models to the greatest detail practicable affects the overall quality of the site-wide water balance model results.
- Incorporate results of future Batman Pit potential groundwater inflow investigation.
- Optimize management of Batman Pit dewatering effluent and other contact water. This water may be of sufficient quality to be used as make-up water to the process circuit offsetting the water coming from the RWD. This water would also reduce the amount of water ultimately requiring treatment.
- In this iteration of the site-wide water balance model, the entirety of the WTP effluent is being used as dust suppression around the mine site during the dry season. Further investigation of other uses of the WTP effluent should be conducted.
- Further investigation of the adequacy of RP1 storage capacity is recommended, particularly within the early stages of the LoM when a larger fraction of the catchment reports to this pond.
- Incorporate RWD stage-storage relationship and catchment area into the site-wide water balance model such that it may be modeled as a reservoir, as opposed to an infinite source.
- Inclusion of process, fire, potable and raw water tanks. At present, only the dust suppression tank is modeled. The tanks above are currently modeled as drawing water directly from the RWD, rather than demands on discrete tanks.
- Review and update dust suppression requirements.
- Incorporate evolving Batman Pit shell geometry to more accurately model that facility.

The estimated budget for this work is US$200,000.

26.6 Groundwater Hydrology and Mine Dewatering

The following work is recommended with respect to groundwater hydrology and mine dewatering:

- Additional hydrogeologic study should be completed in the vicinity of the Batman Pit to provide more detailed information on which to base calibration of the regional groundwater flow model and subsequent prediction of groundwater inflows to the pit and post-mining recovery of the groundwater system. The study should include permeability testing by packer test methods in three to five selected borings within or immediately adjacent to the pit and drilled to the proposed ultimate depth of the pit. The packer testing in those borings should be conducted over relatively short (10 m to 30 m) intervals, beginning at the water table and extending to the total depth of the borings. This testing could be combined with exploration drilling by performing the testing in exploration core holes. The study should also include measurement of depth to water in any accessible existing borings or core holes within or immediately adjacent to the pit.
- Calibration of the regional groundwater flow model should be completed with the additional data, and the calibrated model should be used to refine the estimates of groundwater inflow to the pit and predictions of the hydrogeologic effects of pit dewatering.
- The post-mining version of the groundwater flow model should be updated with the calibrated model used as its basis. Output from the post-mining model should be incorporated into any geochemical modeling of post-mining pit lake formation and geochemistry.

The estimated budget for this work is US$200,000.

26.7 Process Plant Geotechnical Investigation Recommendations

For the DFS future geotechnical work is suggested in the following areas, and is estimated at US$150,000.

26.7.1 Crushing/Screening/Grinding/HPGR/Sorting

All large vibrating structures should be founded in rock rather than on fill to reduce dynamic effects. An accurate rock level is required to confirm foundation design and accurately estimate required concrete quantities and rock excavation. The existing structure concrete slabs are located directly over the new mill location, so it is recommended that additional new test pits around all four sides be undertaken to allow interpolation.

26.7.2 Thickener/Leach/CIP

There are large tanks to be constructed in this area, with high foundation bearing pressure. Variation in rock level will impact the potential for differential settlement which needs to be considered. It is recommended at least four additional test pits to evaluate variance in rock levels in north-south and east-west directions.

26.7.3 Stockpile & Reclaim

There is no test pit data near this area (the nearest test pit is more than 200m away). Steeply sloping ground surface exists (in excess of 5 m variation in ground level across the reclaim tunnel) so there may be considerable variation in rock levels which will affect potential settlement due to stockpile surcharge and required excavation for the concrete vault and tunnel. A new borehole is recommended on the high side (determine whether this high side bench is fill material, and therefore whether rock would be encountered within the limits of an excavator for a test pit) and a new test pit on low side, to determine rock levels.

26.8 Tailings Facility Design

The following studies and investigations are recommended to support TSF design for future phases of the project:

26.8.1 TSF Construction Schedule

The TSF staging and construction schedule must be optimized during the feasibility study phase of the project to minimize mobilization costs.

26.8.2 Investigation of TSF1 Drainage Features

The condition of the existing toe drains, underdrains, and decant towers must be investigated to confirm their condition prior to re-commissioning of TSF1.

26.8.3 Geotechnical Investigation and Assessment

Additional geotechnical investigations are recommended, including drilling, sampling, and laboratory testing to characterize the geotechnical properties of the surface and subsurface materials. The geotechnical drilling program should include Standard Penetration Testing (SPT) and/or Cone Penetration Testing (CPT). CPT investigation within the existing TSF1 tailings is recommended to support assessment of the raise foundation strength and liquefaction analysis. A program of laboratory geotechnical testing of a sample of representative future tailings is recommended as part of future design work, including index testing (particle size, plasticity, specific gravity), compaction testing, and advanced testing to assess consolidation, strength and permeability after compaction, and unsaturated soil characteristics.

26.8.4 Waste Rock Testing

Additional laboratory testing of the waste rock is recommended, including, but not limited to, proctor compaction, hydraulic conductivity, and shear strength testing. No testing of the run-of-mine waste was conducted for this study, and a representative range of strength parameters would improve predictions of behavior related to slope stability.

26.8.5 Consolidation/Seepage Modeling

The seepage and stability analyses discussed in this report were based on laboratory tests conducted on the in-situ tailings. Large scale consolidation tests should be conducted on bench scale samples of the proposed process tailings to determine hydraulic conductivities as a function of effective stress. The seepage and stability analyses must be updated based on these representative material properties. Additionally, seepage and stability analyses must be conducted assuming that the underdrain system is plugged to assess its influence on the stability of future vertical expansions.

26.8.6 Water Balance

A TSF water balance analysis which includes the return water ponds must be conducted to predict the volume of recycle water available for process operations. The results of the water balance analysis will also be used to size the return water system which includes the existing decant structures and an additional barge pump.

26.8.7 TSF Consequence Classification

A formal dam classification assessment is recommended as part of future design work. Under ANCOLD guidelines, the Consequence Category is determined for a potential dam failure and environmental spill. The classification should include a failure modes assessment, a dam break and inundation study, and consideration of the potential for impacts to business, social and environment, and the potential for loss of life. The resulting Consequence Category for the tailings storages will be used to identify parameters for advancing the design such as the design storm event and the design seismic event.

26.9 Process Operating Costs

Two major items incurring operating costs are grinding media and reagents. Together these items make up 61% of the plant consumables operating costs. The FS should investigate options for reducing the consumption rate and the unit costs for these consumables.

This work is included in the estimated budget for the FS.

26.10 Geochemical Analyses

Geochemical characterization will be updated to reflect the designations of Potentially Acid Forming, Potentially Acid Forming-Low Capacity, Non-Acid Forming, Acid Consuming and Uncertain in accordance with DITR (2007) guidelines. Additionally, Tetra Tech recommends performing geochemical testing on the sorter reject material.

This work is included in the estimated budget for the FS.

26.11 Process Parameter Optimization

The on-going testwork is directed at optimization of process parameters which is expected to result in both capital and operating cost reductions as detailed in this Technical Report.

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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 and mineral reserve estimation and classification, underground 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 Vista. The Consultants are not insiders, associates, or affiliates of Vista. 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 Vista 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 and consents of which are contained herein:

Name	Title, Company	Responsible for Sections
Rex Clair Bryan, Ph.D.	Principal Geostatistician Tetra Tech, Inc.	Sections 1.4, 1.5, 1.15.2, 6, 6.1, 6.2, 6.3, 6.4, 7, 8, 9, 10, 11, 12, 14, 24.7.1, 25.2, and 26.2
Anthony Clark, P.E.	Senior Mechanical Engineer POWER Engineers, Inc.	Sections 18.8, 21.1.3, and 21.2.4
Thomas L. Dyer, P.E.	Mining Engineer Mine Development Associates	Sections 1.6, 1.7, 1.15.3, 15, 15.1, 15.2, 15.3, 15.4, 15.5, 15.6, 16, 21.1.1, 21.2.1, 24.7.2, 25.3, and 26.3
Amy L. Hudson, Ph.D., CPG, REM	Principal Hydrogeologist/ Geochemist Tetra Tech, Inc.	Sections 1.15.12, 24.3, and 26.10
Chris Johns, M.Sc., P.Eng	Geological Engineer Tetra Tech, Inc.	Sections 1.15.9, 1.15.10, 18.2.4, 21.1.8, 21.2.6, 24.1, 26.7 and 26.8
Deepak Malhotra, Ph.D.	Principal Metallurgist Pro Solv, LLC	Sections 1.6.1, 1.8, 1.9, 1.15.4, 1.15.11, 6.5, 6.6, 13, 15.7, 17, 24.7.3, 25.4, 26.9, and 26.11
Jessica I. Monasterio, P.E.	Professional Engineer JDS Energy & Mining, Inc.	Sections 1.11, 1.13, 1.14, 19, 21, 21.1, 21.2, 22, 24.7.6, 24.7.7, and 24.7.8
Zvonimir Ponos, BE, MIEAust, CPeng, NER	Senior Principal Engineer Coffey Services Australia Pty Ltd (trading as Tetra Tech Proteus)	Sections 1.10, 1.15.5, 18.1, 18.2, 18.2.1, 18.2.2, 18.2.3, 18.2.5, 18.2.6, 18.3, 18.4, 18.5, 18.6, 18.7, 21.1.2, 21.2.3, 21.2.7, 24.6, 24.7.4, 25.5, and 25.6

Name	Title, Company	Responsible for Sections
Guy Roemer, P.E.	Environmental Engineer Tetra Tech, Inc.	Sections 1.15.7, 24.2.1, 24.7.5, 25.8 and 26.5
Vicki Scharnhorst, P.E., LEED AP	Principal Tetra Tech, Inc.	Sections 1.1, 1.2, 1.3, 1.12, 1.15.1, 1.15.6, 2, 3, 4, 5, 18, 20, 21.1.5, 21.1.6, 21.1.7, 21.2.5, 23, 24, 24.2, 24.2.2, 24.4, 24.7, 25.1, 25.7, 26, 26.1, and 26.4
Keith Thompson, CPG, PG	Professional Geologist Tetra Tech, Inc.	Sections 1.15.8, 21.1.4, 21.2.2, 24.5, 26.6

28.2 Table of Responsibility

QPs are responsible for all subsections listed beneath headings unless subsections are detailed below.

Section No	Section Name	QP
1	**SUMMARY (No Intro)**	**N/A**
1.1	Introduction	Scharnhorst, Vicki
1.2	Location	Scharnhorst, Vicki
1.3	Property Description	Scharnhorst, Vicki
1.4	Geology and Mineralization	Bryan, Rex
1.5	Mineral Resource Estimate	Bryan, Rex
1.6	Mineral Reserve Estimates	Dyer, Tom
1.6.1	Heap Leach Reserve Estimate	Malhotra, Deepak
1.7	Mining Methods	Dyer, Tom
1.8	Metallurgy	Malhotra, Deepak
1.9	Mineral Processing	Malhotra, Deepak
1.10	Project Infrastructure	Ponos, Zvonimir
1.11	Market Studies and Contracts	Monasterio, Jessica
1.12	Social and Environmental Aspects	Scharnhorst, Vicki
1.13	Capital and Cost Estimates	Monasterio, Jessica
1.14	Financial Analysis	Monasterio, Jessica
1.15	Conclusions and Recommendations (No Intro)	N/A
1.15.1	Feasibility Study	Scharnhorst, Vicki
1.15.2	Geology and Resources	Bryan, Rex
1.15.3	Mineral Reserve and Mine Planning	Dyer, Tom
1.15.4	Mineral Processing	Malhotra, Deepak
1.15.5	Infrastructure	Ponos, Zvonimir
1.15.6	Environmental and Social Impacts	Scharnhorst, Vicki
1.15.7	Results of the Site-wide Water Balance Model	Roemer, Guy

Section No	Section Name	QP
1.15.8	Groundwater Hydrology and Mine Dewatering	Thompson, Keith
1.15.9	Process Plant Geotechnical Investigation	Johns, Chris
1.15.10	TSF Design	Johns, Chris
1.15.11	Process	Malhotra, Deepak
1.15.12	Geochemical Analyses	Hudson, Amy
2	**INTRODUCTION**	**Scharnhorst, Vicki**
3	**RELIANCE ON OTHER EXPERTS**	**Scharnhorst, Vicki**
4	**PROPERTY DESCRIPTION AND LOCATION**	**Scharnhorst, Vicki**
5	**ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY**	**Scharnhorst, Vicki**
6	**HISTORY**	**Bryan, Rex**
6.1	History of Previous Exploration	Bryan, Rex
6.2	Historic Drilling	Bryan, Rex
6.3	Historic Sampling Method and Approach	Bryan, Rex
6.4	Historic Sample Preparation, Analysis, and Security	Bryan, Rex
6.5	Historic Process Description	Malhotra, Deepak
6.6	Technical Problems with Historical Process Flowsheet	Malhotra, Deepak
7	**GEOLOGICAL SETTING AND MINERALIZATION**	**Bryan, Rex**
8	**DEPOSIT TYPES**	**Bryan, Rex**
9	**EXPLORATION**	**Bryan, Rex**
10	**DRILLING**	**Bryan, Rex**
11	**SAMPLE PREPARATION, ANALYSES AND SECURITY**	**Bryan, Rex**
12	**DATA VERIFICATION**	**Bryan, Rex**
13	**MINERAL PROCESSING AND METALLURGICAL TESTING**	**Malhotra, Deepak**
14	**MINERAL RESOURCE ESTIMATES**	**Bryan, Rex**
15	**MINERAL RESERVES**	**Dyer, Tom**
15.1	Pit Optimization	Dyer, Tom
15.2	Pit Designs	Dyer, Tom
15.3	Cutoff Grade	Dyer, Tom
15.4	Dilution	Dyer, Tom
15.5	Reserves and Resources	Dyer, Tom
15.6	In-Pit inferred Resources	Dyer, Tom
15.7	Heap Leach Reserve Estimate	Malhotra, Deepak
16	**MINING METHODS**	**Dyer, Tom**
17	**RECOVERY METHODS**	**Malhotra, Deepak**
18	**PROJECT INFRASTRUCTURE**	**Scharnhorst, Vicki**

Section No	Section Name	QP
18.1	Facility 2000 – Mine	Ponos, Zvonimir
18.2	Facility 4000 – Project Services	Ponos, Zvonimir
18.2.1	Area 4100 – Water Supply	Ponos, Zvonimir
18.2.2	Area 4200 – Power Supply	Ponos, Zvonimir
18.2.3	Area 4300 – Communications	Ponos, Zvonimir
18.2.4	Area 4400 – Tailings Dam	Johns, Chris
18.2.5	Area 4500 – Waste Disposal	Ponos, Zvonimir
18.2.6	Area 4600 – Plant Mobile Equipment	Ponos, Zvonimir
18.3	Facility 5000 – Project Infrastructure	Ponos, Zvonimir
18.4	Facility 6000 – Permanent Accommodation	Ponos, Zvonimir
18.5	Facility 7000 – Site Establishment and Early Works	Ponos, Zvonimir
18.6	Facility 8000 – Management, Engineering, EPCM Services	Ponos, Zvonimir
18.7	Facility 9000 – Preproduction Costs	Ponos, Zvonimir
18.8	Electric Power Plant	Clark, Anthony
19	**MARKET STUDIES AND CONTRACTS**	**Monasterio, Jessica**
20	**ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT**	**Scharnhorst, Vicki**
21	**CAPITAL AND OPERATING COSTS**	**Monasterio, Jessica**
21.1	Capital Cost	Monasterio, Jessica
21.1.1	Mining (MDA)	Dyer, Tom
21.1.2	CIP Process and Infrastructure (Proteus)	Ponos, Zvonimir
21.1.3	Power Plant (POWER Engineers)	Clark, Anthony
21.1.4	Mine Dewatering (Tetra Tech)	Thompson, Keith
21.1.5	Reclamation and Closure (Tetra Tech)	Scharnhorst, Vicki
21.1.6	Water Treatment Plant (Tetra Tech)	Scharnhorst, Vicki
21.1.7	Raw Water Dam (Tetra Tech)	Scharnhorst, Vicki
21.1.8	Tailings Storage Facilities (Tetra Tech)	Johns, Chris
21.2	Operating Costs	Monasterio, Jessica
21.2.1	Mining (MDA)	Dyer, Tom
21.2.2	Mine Dewatering (Tetra Tech)	Thompson, Keith
21.2.3	CIP Process and G&A (Proteus)	Ponos, Zvonimir
21.2.4	Power Plant (POWER Engineers)	Clark, Anthony
21.2.5	Water Treatment Plant (Tetra Tech)	Johnson, Benjamin
21.2.6	Tailings Storage Facilities (Tetra Tech)	Johns, Chris
21.2.7	General & Administrative	Ponos, Zvonimir
22	**ECONOMIC ANALYSIS**	**Monasterio, Jessica**

Section No	Section Name	QP
23	**ADJACENT PROPERTIES**	**Scharnhorst, Vicki**
24	**OTHER RELEVANT DATA AND INFORMATION**	**Scharnhorst, Vicki**
24.1	Process Plant Geotechnical	Johns, Chris
24.2	Water Management	Scharnhorst, Vicki
24.2.1	Site-wide Water Balance	Roemer, Guy
24.2.2	Wet Infrastructure	Scharnhorst, Vicki
24.2.2.1	Water Treatment Plant	Scharnhorst, Vicki
24.2.2.2	Water Quality Standards for Waste Water Discharge	Scharnhorst, Vicki
24.2.2.3	Raw Water Reservoir and Pipeline	Scharnhorst, Vicki
24.2.2.4	Potable Water	Scharnhorst, Vicki
24.2.2.5	Sanitary Sewer System	Scharnhorst, Vicki
24.3	Geochemistry	Hudson, Amy
24.4	Surface Water Hydrology	Scharnhorst, Vicki
24.5	Regional Groundwater Model and Mine Dewatering	Thompson, Keith
24.6	Project Implementation	Ponos, Zvonimir
24.7	Alternate Case	Scharnhorst, Vicki
24.7.1	Mineral Resource	Bryan, Rex
24.7.2	Mining	Dyer, Tom
24.7.3	Process Facility	Malhotra, Deepak
24.7.4	Infrastructure	Ponos, Zvonimir
24.7.5	Site-wide Water Balance Model	Roemer, Guy
24.7.6	Capital Costs, Alternate Case	Monasterio, Jessica
24.7.7	Operating Costs, Alternate Case	Monasterio, Jessica
24.7.8	Economic Results, Alternate Case	Monasterio, Jessica
25	**INTERPRETATION AND CONCLUSIONS (No Intro)**	**N/A**
25.1	Project Risks	Scharnhorst, Vicki
25.2	Geology and Resources	Bryan, Rex
25.3	Mineral Reserve and Mine Planning	Dyer, Tom
25.4	Mineral Processing	Malhotra, Deepak
25.5	Infrastructure	Ponos, Zvonimir
25.6	Project Services	Ponos, Zvonimir
25.7	Environmental and Social Conclusions	Scharnhorst, Vicki
25.8	Results of the Site-wide Water Balance Model	Roemer, Guy
26	**RECOMMENDATIONS**	**Scharnhorst, Vicki**
26.1	Feasibility Study	Scharnhorst, Vicki
26.2	Resource and Exploration	Bryan, Rex

Section No	Section Name	QP
26.3	Mining Risks and Opportunities	Dyer, Tom
26.4	Environmental Studies	Scharnhorst, Vicki
26.5	Site-wide Water Balance	Roemer, Guy
26.6	Groundwater Hydrology and Mine Dewatering	Thompson, Keith
26.7	Process Plant Geotechnical Investigation Recommendations	Johns, Chris
26.8	Tailings Facility Design	Johns, Chris
26.9	Process Operating Costs	Malhotra, Deepak
26.10	Geochemical Analyses	Hudson, Amy
26.11	Process Parameter Optimization	Malhotra, Deepak
27	**REFERENCES**	**N/A**
28	**CERTIFICATE OF QUALIFIED PERSON**	**N/A**
28.1	Qualifications of Consultants	N/A
28.2	Table of Responsibility	N/A

CERTIFICATE OF QUALIFIED PERSON

Rex Clair Bryan, Ph.D.
Principal Geostatistician
Tetra Tech, Inc.
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Telephone: (303) 217-5700
Facsimile: (303) 217-5705
Email: Rex.Bryan@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Rex Clair Bryan, Ph.D.**, do hereby certify that:

1) I am a Senior Geostatistician with Tetra Tech, Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado USA, 80401.
2) I graduated with a Ph.D. degree in 1980 from the Colorado School of Mines, Golden Colorado, USA. In addition, I graduated with a degree MSc. In Geology in 1976 from the Brown University, Providence, Rhode Island, USA. I have worked as a Geostatistician for a total of 39 years since my graduation. My relevant experience is in the areas of resources and reserve reporting. I am a Competent/Qualified Person (QP), with the Society of Mining Engineers in Colorado, USA (SME Registered Member #411340).
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have personally visited and inspected the property which is the subject of the Technical Report on June 28th and June 29th, 2017 for two days. In addition, I have visited and inspected the property September 12th, 2011 to September 14th, 2011 and February 6th, 2013 to February 8th, 2013.
5) I am responsible for Sections 1.4, 1.5, 1.15.2, 6, 6.1, 6.2, 6.3, 6.4, 7, 8, 9, 10, 11, 12, 14, 24.7.1, 25.2, and 26.2 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Rex Clair Bryan, Ph.D."
Signature of Qualified Person

Rex Clair Bryan, Ph.D.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Anthony Clark, P.E.
Area Lead Mechanical Engineer
POWER Engineers, Inc.
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Facsimile: (208) 288-6199
Email: tony.clark@powereng.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Anthony Clark, P.E.**, do hereby certify that:

1) I am a Senior Mechanical Engineer with POWER Engineers, Inc. with a business address at 2041 S. Cobalt Point Way, Meridian, Idaho, 83642, USA.
2) I graduated with a degree in Mechanical Engineering, Bachelor of Science in 2001 from the Missouri University of Science and Technology, Rolla, Missouri. I have worked as a Mechanical Engineer for a total of 18 years since my graduation. My relevant experience is in the area of electrical power plant design. I am a PE in Idaho (No. 14669) and P.Eng in Alberta (No. 208138).
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have personally visited and inspected the property which is the subject of the Technical Report on June 28th through June 29th, 2017 to inspect the existing power infrastructure at the site, natural gas pipeline, and power rights-of-way.
5) I am responsible for Sections 18.8, 21.1.3, and 21.2.4 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the Power Station Feasibility Study provided in Appendix F of NI 43-101 Technical Report dated June 2013.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Anthony Clark, P.E."
Signature of Qualified Person

Anthony Clark, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Thomas L. Dyer, P.E.
Mining Engineer
Mine Development Associates
210 South Rock Boulevard
Reno, Nevada 89502
Telephone: (775) 856-5700
Email: tdyer@mda.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Thomas L. Dyer, P.E.**, do hereby certify that:

1) I am a Senior Engineer with Mine Development Associates with a business address at 210 Rock Blvd, Reno, NV, 89502, USA.
2) I graduated with a B.S. degree in Mine Engineering in 1996 from the South Dakota School of Mines and Technology. I have worked as a Mining Engineer for a total of 23 years since my graduation. My relevant experience includes 11 years of Engineering in an operating open pit mine including underground studies. This operations experience included increasing responsibilities obtaining the position of Chief Engineer. Since that time I have worked as a Consulting Mining Engineer for numerous open pit and underground projects including Preliminary Economic Assessments, Prefeasibility, and Feasibility studies. I am a P.E. in Nevada (No. 15729) and am a Registered Member of SME (# 4029995RM) in good standing.
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have personally visited and inspected the property which is the subject of the Technical Report in April of 2017 for 2 days.
5) I am responsible for Sections 1.6, 1.7, 1.15.3, 15 (except 15.7), 16, 21.1.1, 21.2.1, 24.7.2, 25.3, and 26.3 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as a Qualified Person and author for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018 and previous reports.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP - Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared to be compliant with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Thomas L. Dyer, P.E."
Signature of Qualified Person

Thomas L. Dyer, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Amy L. Hudson, Ph.D., CPG, REM
Principal Hydrogeologist/ Geochemist
Tetra Tech, Inc.
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Amy L. Hudson, Ph.D., CPG, REM**, do hereby certify that:

1) I am a Principal Hydrogeologist/Geochemist with Tetra Tech, Inc. with a business address at 1750 Kraft Drive, Suite 1503, Blacksburg, Virginia.
2) I graduated with a degree in Geology and Environmental Science, B.S. in 1998 from the Mary Washington College, Fredericksburg, Virginia and I graduated with a degree in Environmental Science and Engineering, M.S. in 2006 from the Colorado School of Mines, Golden, Colorado. In addition, I graduated with a degree in Geoscience, Ph.D. in 2016 from the University of Massachusetts Amherst, Amherst, Massachusetts. I have worked as a Hydrogeologist/Geochemist for a total of 21 years since my graduation. My relevant experience is in the area of geochemistry, hydrogeology, and environmental science. I am a Certified Professional Geologist in Virginia (No. 002122) and a Registered Environmental Manager in the USA (No. 11854).
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have not visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 1.15.12, 24.3, and 26.10 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, Amended & Restated; July 7, 2014.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP - Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Amy L. Hudson, Ph.D., CPG, REM"
Signature of Qualified Person

Amy L. Hudson, Ph.D., CPG, REM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Chris Johns, M.Sc., P.Eng.
Geological Engineer
Tetra Tech, Inc.
1715 Dickson Avenue, Suite 150
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This certificate applies to the report entitled: “NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia” (“Technical Report”), effective date September 10, 2019, issued on October 7, 2019.

I, **Chris Johns, M.Sc., P.Eng.**, do hereby certify that:

1) I am a Senior Consultant with Tetra Tech, Inc. with a business address at 150-1715 Dickson Avenue, Kelowna, British Columbia, Canada.
2) I graduated with a degree in Geological Engineering, B.Sc., in 1994 from Queen’s University, Kingston, Ontario. In addition, I graduated with a degree in Environmental Engineering, M.Sc. in 1999 from the University of Alberta, Edmonton, AB. I have worked as a geological engineer for a total of 20 years since my graduation. My relevant experience is in the area of 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.
3) I have read the definition of “qualified person” set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” within the meaning of NI 43-101.
4) I have personally visited and inspected the property which is the subject of the Technical Report on June 28th – 29th for 2 days.
5) I am responsible for Sections 1.15.9, 1.15.10, 18.2.4, 21.1.8, 21.2.6, 24.1, 26.7 and 26.8 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have not had prior involvement with the property that is the subject of the Technical Report.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed “Chris Johns, M.Sc., P.Eng.”
Signature of Qualified Person

Chris Johns, M.Sc., P.Eng.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, Ph.D.
Principal Metallurgist
Pro Solv, LLC
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Lakewood, Colorado 80228
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Deepak Malhotra, Ph.D.**, do hereby certify that:

1) I am the President of Pro Solv, LLC, with a business address at 15450 W. Asbury Avenue, Lakewood, Colorado 80228.
2) I graduated with a degree in Metallurgical Engineering, Master of Science in 1973 from the Colorado School of Mines in Golden, Colorado. In addition, I graduated with a degree in Mineral Economics, Ph.D. in 1978 from the Colorado School of Mines in Golden, Colorado. I have worked as a metallurgist and mineral economist for a total of 46 years since my graduation. My relevant experience is in the area of metallurgy and mineral economics.
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have not visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 1.6.1, 1.8, 1.9, 1.15.4, 1.15.11, 6.5, 6.6, 13, 15.7, 17, 24.7.3, 25.4, 26.9, and 26.11 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the Process Development of the project, I was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Deepak Malhotra, Ph.D."
Signature of Qualified Person

Deepak Malhotra, Ph.D.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Jessica I. Monasterio, P.E.**, do hereby certify that:

1) I am a Professional Engineer with Tetra Tech, Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, United States.
2) I graduated with a degree in Geological Engineering, Bachelor of Science in 2006 from the Colorado School of Mines, Golden, Colorado. In addition, I graduated with a degree in Geological Engineering, Master of Engineering in 2007 from the Colorado School of Mines, Golden, Colorado. I have worked as a Geological Engineer for six years, and a Mineral Economist for six years, for a total of twelve years since my graduation. My relevant experience is in the area of geotechnical engineering and mineral economics. I am a Professional Engineer in the State of Colorado (P.E. 0045454).
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have not visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 1.11, 1.13, 1.14, 19, 21 (Introduction), 21.1 (Introduction), 21.2 (Introduction), 22, 24.7.6, 24.7.7 and 24.7.8 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP - Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Jessica I. Monasterio, P.E."
Signature of Qualified Person

Jessica I. Monasterio, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Zvonimir Ponos, BE, MIEAust, CPeng, NER
General Manager Engineering
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Zvonimir Ponos, BE, MIEAust, CPeng, NER**, do hereby certify that:

1) I am a General Manager of Engineering with Coffey Services Australia Pty Ltd (trading as Tetra Tech Proteus) with a business address at Level 1, 235 St Georges Terrace, Perth, Western Australia, 6000, Australia.
2) I graduated with a degree in Structural Engineering in 1985 from the University of Belgrade in Yugoslavia. I have worked as a Design Engineer, Engineering Manager and Project Manager for more than 30 years since my graduation. My relevant experience is in the areas of structural design, engineering management and project management of chemical, mineral processing and materials handling projects in Gold, Iron Ore, Mineral Sands, Alumina and Base Metals.
3) I am a Chartered Professional Engineer and a Member of the Institution of Engineers Australia (No. 230033). I am also Member of Concrete Institute of Australia (CIA) and Australian Steel Institute (No. 6184). I am a (lapsed) member of both Australian Institute of Project Management (AIPM No 2765) and Project Management Institute, USA (PMI No 167332). All memberships in good standing.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
5) I have personally visited and inspected the property which is the subject of the Technical Report on the 28th June 2017 for three (3) days.
6) I am responsible for Sections 1.10, 1.15.5, 18.1, 18.2, 18.2.1, 18.2.2, 18.2.3, 18.2.5, 18.2.6, 18.3, 18.4, 18.5, 18.6, 18.7, 21.2.1, 21.2.3, 21.2.7, 24.6, 24.7.4, 25.5 and 25.6 of the Technical Report.
7) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
8) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018.
9) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
10) 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.
11) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Zvonimir Ponos, BE, MIEAust, CPeng, NER"
Signature of Qualified Person

Zvonimir Ponos, BE, MIEAust, CPeng, NER
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Guy Roemer, P.E.
Environmental Engineer
Tetra Tech, Inc.
1100 South McCaslin Boulevard, Suite 150
Superior, Colorado 80027
Telephone: (303) 664-4624
Email: Guy.Roemer@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Guy Roemer, P.E.**, do hereby certify that:

1) I am an Environmental Engineer with Tetra Tech, Inc. with a business address at 1100 South McCaslin Boulevard, Suite 150, Superior, Colorado, 80027, United States of America.
2) I graduated with a degree in Nuclear Engineering, B.S. in 1995 from Texas A&M University, College Station, Texas. In addition, I graduated with a degree in Nuclear Engineering, M.S. in 1997 from the University of New Mexico, Albuquerque, New Mexico. I have worked as an Environmental Engineer for a total of 22 years since my graduation. My relevant experience is in the area of conducting site-wide and storage facility water balance models of mines using analytical and numerical models. I am a P.E. in Colorado (No. 36810).
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have not visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 1.15.7, 24.2.1, 24.7.5, 25.8 and 26.5 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Guy Roemer, P.E."
Signature of Qualified Person

Guy Roemer, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Vicki Scharnhorst, P.E., LEED AP
Principal
Tetra Tech, Inc.
1560 Broadway, Suite 1400
Denver, Colorado 80202
Telephone: (303) 825-5999
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Vicki Scharnhorst, P.E., LEED AP**, do hereby certify that:

1) I am a Principal Consultant with Tetra Tech with a business address at 1560 Broadway, Denver, CO 80202, USA.
2) I graduated with a Bachelor of Science degree in Civil Engineering in 1982 from Kansas State University, Manhattan, Kansas. In addition, I graduated with a Master of Public Administration and Policy degree in 2017 from the American University, Washington D.C. I have worked as a civil engineer for a total of 36 years since my graduation. My relevant experience includes civil engineering on large infrastructure projects inclusive of civil works, water quality programs, environmental impact studies, and permitting. I am a licensed Engineer in the states of Nevada (No. 7647), Michigan (No. 43541), Missouri (No. 27930), and Colorado (No. 41466); a water right surveyor in the State of Nevada; and a LEED Accredited Professional with the U.S. Green Building Council.
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have personally visited and inspected the property which is the subject of the Technical Report on June 28th and June 29th, 2017 for two days.
5) I am responsible for Sections 1.1, 1.2, 1.3, 1.12, 1.15.1, 1.15.6, 2, 3, 4, 5, 18.0 (Introduction), 20, 21.1.5, 21.1.6, 21.1.7, 21.2.5, 23, 24.0 (Introduction), 24.2, 24.4, 24,7, 25.1, 25.7, 26, 26.1, and 26.4 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as a qualified person who was relied upon for the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Vicki Scharnhorst, P.E., LEED AP"
Signature of Qualified Person

Vicki Scharnhorst, P.E., LEED AP
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Keith Thompson, CPG, PG
Professional Geologist
Tetra Tech, Inc.
1100 McCaslin Boulevard, Suite 150
Superior, Colorado 80027
Telephone: (303) 664-4630
Email: Keith.Thompson@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project – 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" ("Technical Report"), effective date September 10, 2019, issued on October 7, 2019.

I, **Keith Thompson, CPG, PG**, do hereby certify that:

1) I am a Senior Hydrogeologist with Tetra Tech with a business address at 1100 McCaslin Boulevard, Suite 150, Superior, Colorado 80027, USA.
2) I graduated with Bachelor of Science degree in Geology in 1975 from Youngstown State University, Youngstown, Ohio, USA. In addition, I graduated with a Master of Science degree in Geology in 1979 from the University of Wyoming, Laramie, Wyoming, USA. I have worked as a hydrogeologist for a total of 40 years since my graduation. My relevant experience is in the areas of mining hydrology and hydrogeology, environmental hydrology and hydrogeology, and groundwater flow and transport modeling. I am a Certified Professional Geologist (No. 6005) and member of the American Institute of Professional Geologists and a licensed Professional Geologist in the (USA) states of Alaska (No. 700), California (No. 5572), Idaho (No. 726), Utah (No. 5258797-2250) and Wyoming (No. 2454).
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have not visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 1.15.8, 21.1.4, 21.2.2, 24.5, 26.6 of the Technical Report.
6) I am independent of the issuer, Vista Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had prior involvement with the property that is the subject of the Technical Report. My involvement has consisted of acting as an expert who was relied upon for "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, Appendix K – Regional Hydrogeology" dated March 2, 2018.
8) I have read NI 43-101, Form 43-101F1 – Technical Report, 43-101CP – Standards of Disclosure for Mineral Projects, and the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 7th day of October 2019

Signed, Sealed "Keith Thompson, CPG, PG"
Signature of Qualified Person

Keith Thompson, CPG, PG
Print Name of Qualified Person