VISTA GOLD

NI 43-101 Technical Report

Mt Todd Gold Project | 50,000 tpd Feasibility Study

Northern Territory, Australia

Effective Date: March 12, 2024
Issue Date: April 16, 2024

Project No. 117-8348002

TETRA TECH

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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 FS; 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 FS, mineral resource and reserve estimates, and exploration and assay results; the terms and conditions of the Company's agreements with contractors and Vista's 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 March 2024 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

This report has been prepared in accordance with the requirements and standards of Canadian National Instrument 43-101 (NI 43-101) and uses terms in accordance with applicable Canadian mining standards as defined under NI 43-101. These requirements, standards and definitions differ from those set forth in the United States Securities and Exchange Commission's Regulation S-K subsection 1300 (S-K 1300). While the requirements, standards and terms under NI 43-101 and S-K 1300 are substantially similar, they are not identical and the contents of this report may differ materially from those set forth in the reports prepared in accordance with S-K 1300 requirements, including reserve and resource estimates. U.S. Investors are cautioned not to assume that the contents of this report and the reserve and resource estimates contained herein are the same as those contained in a technical report summary prepared in accordance with S-K 1300.

NOTE

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

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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
2020 PFS	NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study – Northern Territory, Australia, Effective Date September 10, 2019; Issued October 7, 2019; Amended September 22, 2020
A	ampere
a	annum (year)
AA	Atomic adsorption
ABA	acid base accounting
AD	annual deduction
ADWG	Australian Drinking Water Guidelines
AGR	Australian Gold Reagents Pty. Ltd.
AHD	Australian Height Datum
ALS	Australian Laboratory Services
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
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
DC	Dry Commissioning
DDH	Diamond drillhole core
DH	drillhole
DITT	Department of Industry, Tourism and Trade
dmt	dry metric ton
DO	Dissolved oxygen
DoR	Department of Resources
DRDPIFR	Department of Regional Development, Primary Industry, Fisheries and Resources
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

F80	80% feed passing size
FIS	Free In Store
FLS	FLSmidth
FS Case	50,000 tpd Case
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)
GPR	Gross Proceeds Royalty
GR	gross realization
GW	gigawatt
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
HAZOP	Hazard and Operability
HCL	Hydrochloric Acid
HHV	Higher Heating Value
HLP	heap leach pad
HNO3	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-MS	Inductively Coupled Plasma-Mass Spectrometry
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
LGRP	Low grade ore stockpile retention pond
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
ML	Mineral License
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
NAF	non-acid forming
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
PAF	potentially acid forming
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
PP	Process Plant
ppb	parts per billion
ppm	parts per million
Project	Mt Todd Gold Project
PRP	Process Plant Retention Pond
PSR	Procurement Status Report
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.
RESPEC	Mine Development Associates (MDA)
RKD	RKD (Company Name)
RL	Sample name

RO	runoff pond
RoM	Run of Mine
RP	retention pond
RP1	Waste rock dump retention pond
RP3	Batman Pit
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 meta bi-sulfite
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
SRM	Standard reference materials
st	short ton (2,000 lb)
st/d	short tons per day
st/y	short tons per year
S.U.	Standard unit
SWi	Standard work index
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 Feasibility Study
TEM	technical economic model
Tetra Tech	Tetra Tech, Inc.
TKI	Thyssen-Krupp Industries
tpd	tonnes per day
tph	tonnes per hour
ts/hm^3	ton-sec/hour-cubic meter
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
XRT	x-ray transmission
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
1 tonne	=	1,000 kg

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	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. SUMMARY

1.1 Introduction

Vista Gold Corp. (Vista) retained Tetra Tech, along with RESPEC (formerly Mine Development Associates (MDA)), Resource Development Inc. (RDi), Pro Solv Consulting, LLC (Pro Solv), and Tetra Tech Proteus (TTP) to prepare this feasibility study (FS) for its Mt Todd Gold Project (the Project) in Northern Territory (NT), Australia. The FS (Technical Report) evaluates a development scenario of a 50,000 tonne per day (tpd) processing facility.

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 historical 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 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.

This information is intended to assist stakeholders and other readers of this Technical Report in their understanding of the Mt Todd Gold Project and in forming judgements regarding the quality of the data collected, reported, and used in the Technical Report.

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, EL 32004, and ELA 32005 comprising approximately 158,131 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 1-1.

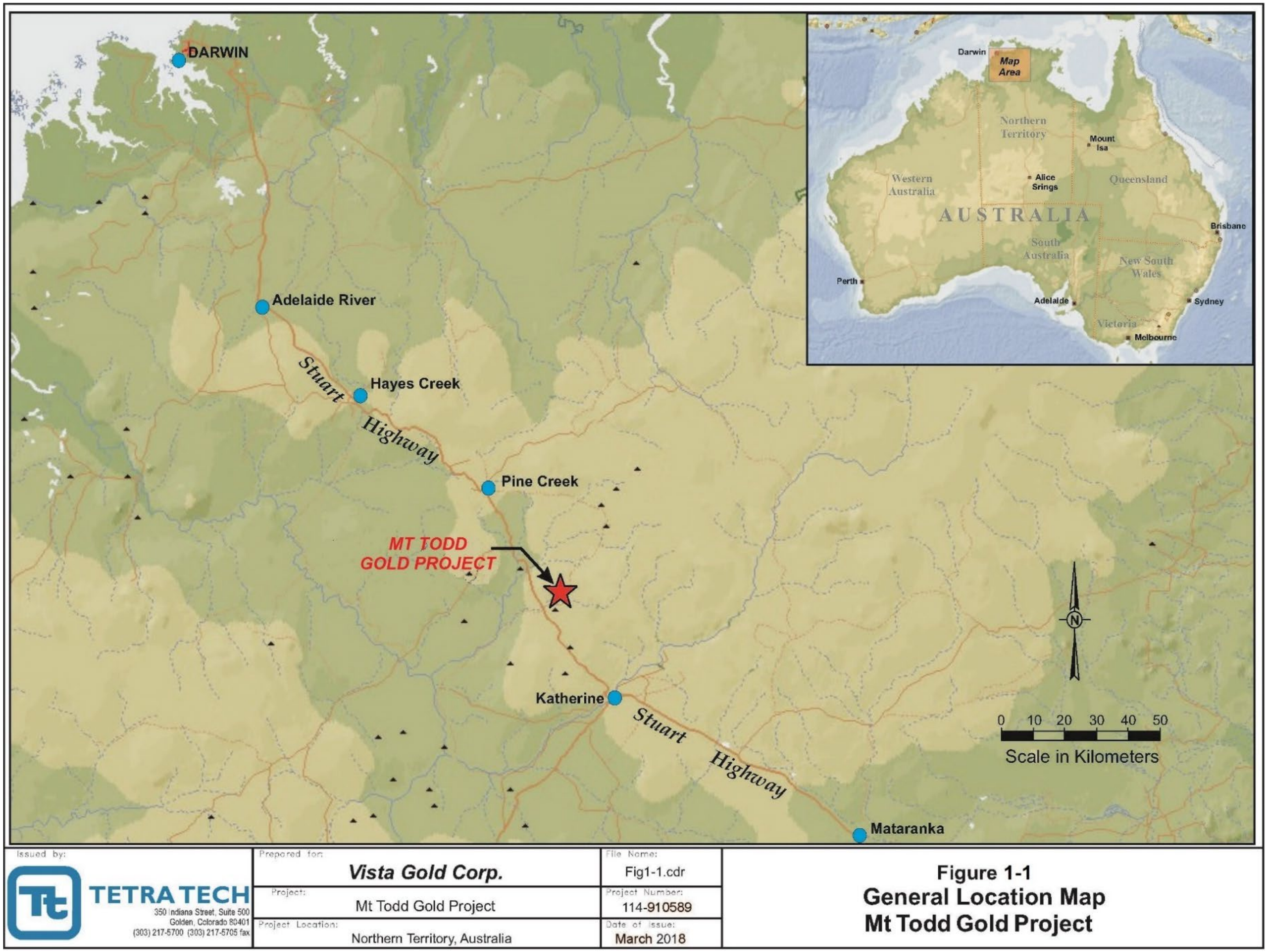
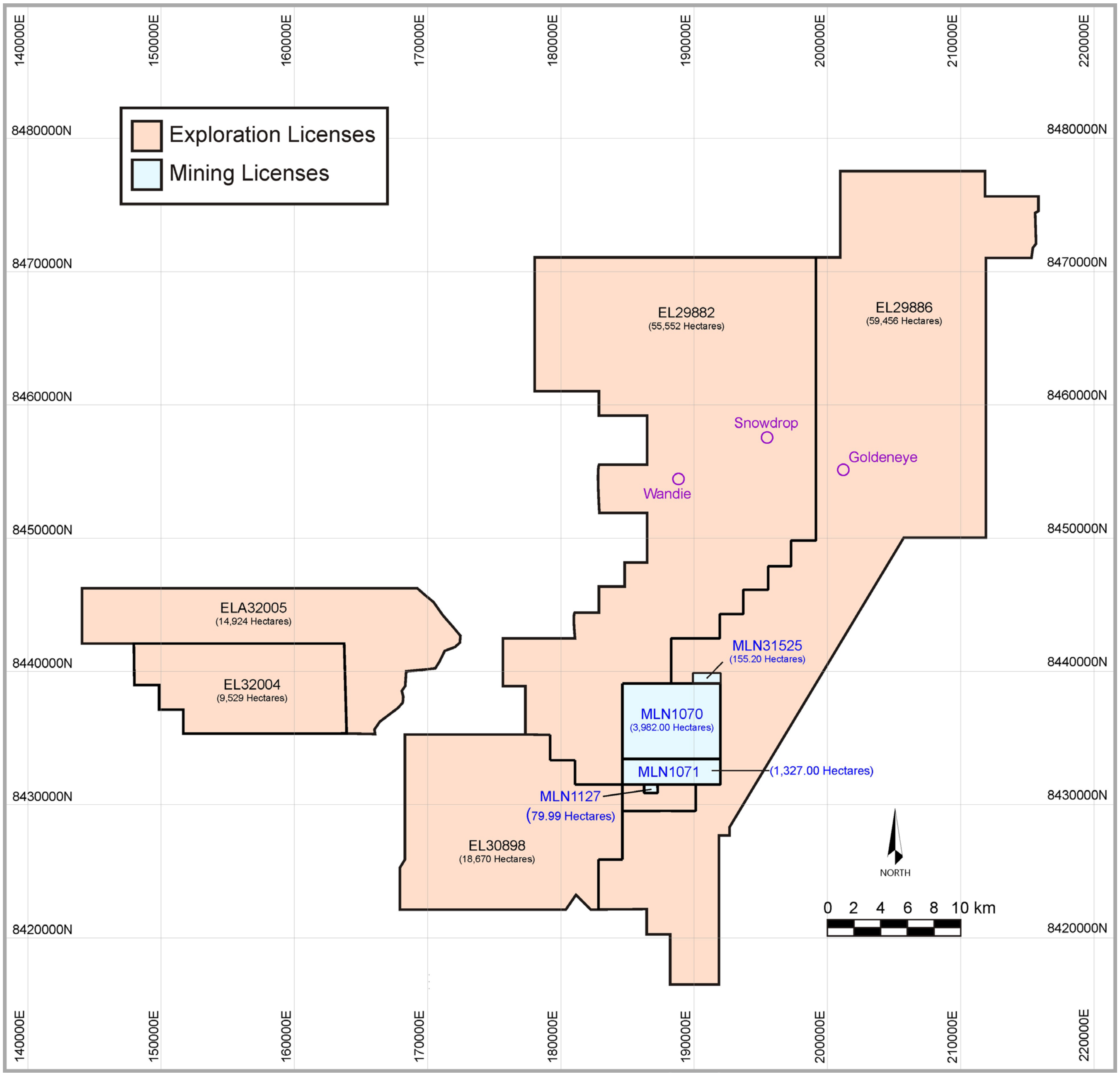


Figure 1-1: General Project Location Map

Table 1-1: Description of Landforms and Impoundments

Landform/Impoundment	Abbreviated Name
Tailings Storage Facility 1	TSF1
Tailings Storage Facility 2	TSF2
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 January 2022

Figure 1-2: Concessions

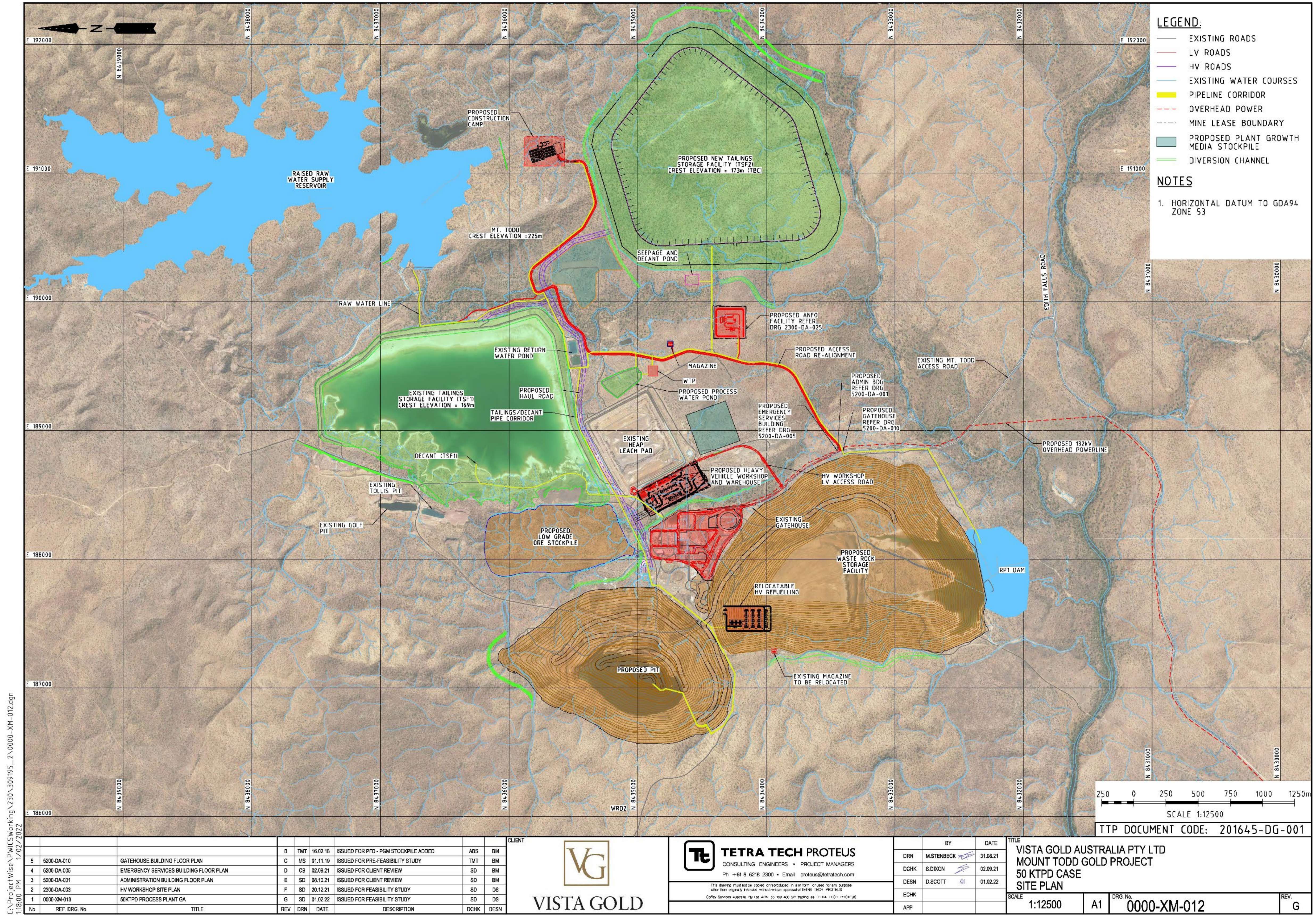


Figure 1-3: General Arrangement

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 Brothers, Golf-Tollis, 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:

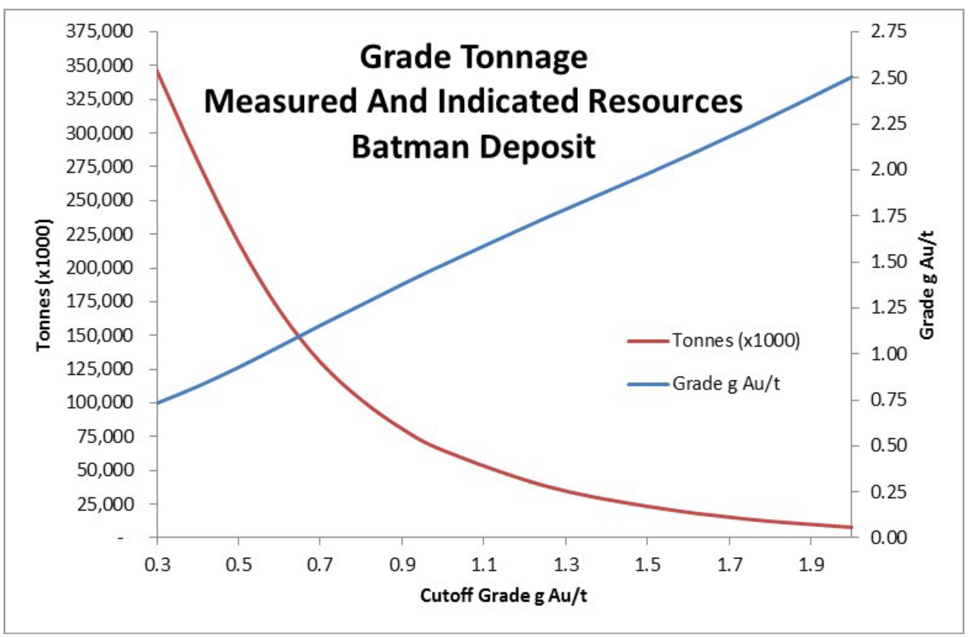
1) Batman deposit;
2) Existing heap leach pad; and
3) Quigleys deposit.

The resource estimate for the Batman deposit is updated from the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, Amended & Restated; July 7, 2014 prepared by Tetra Tech. This report includes an estimate of gold contained in a historical heap leach pad adjacent to the Batman deposit. Additionally, this report contains the resource estimation of the Batman and Quigleys deposits. The updated Project resource estimates are shown in Table 1-2, and grade tonnage curve for the measured and indicated resource for the Batman deposit is presented in Figure 1-4.

Table 1-2: Statement of Mineral Resources Estimates

	BATMAN DEPOSIT			HEAP LEACH PAD			QUIGLEYS DEPOSIT		
	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	-	-	-	594	1.15	22
Indicated (I)	200,112	0.80	5,169	13,354	0.54	232	7,301	1.11	260
Measured & Indicated	**277,837**	**0.82**	**7,360**	**13,354**	**0.54**	**232**	**7,895**	**1.11**	**282**
Inferred (F)	61,323	0.72	1,421	-	-	-	3,981	1.46	187

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, G&A/Year 8,201 K US4, Au Recovery, Sulfide 85%, Transition 80%, Oxide 80%, 0.2g Au/t minimum for resource shell.
4) Quigleys: Resources constrained within a US$1,300/oz gold Whittle™ pit shell. Pit parameters: Mining cost US$1.90/tonne, Processing Cost US$9.779/tonne processed, Royalty 1% GPR, Gold Recovery Sulfide, 82.0% and Ox/Trans 78.0%, water treatment US$0.09/tonne, Tailings US$0.985/tonne
5) Differences in the table due to rounding are not considered material. Differences between Batman and Quigleys mining and metallurgical parameters are due to their individual geologic and engineering characteristics.
6) Rex Bryan of Tetra Tech is the QP responsible for the Statement of Mineral Resources for the Batman, Heap Leach Pad and Quigleys deposits.
7) Thomas Dyer of RESPEC is the QP responsible for developing the resource Whittle™ pit shell for the Batman Deposit.
8) The effective date of the Heap Leach, Batman and Quigleys resource estimate is December 31, 2023.
9) Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.



Source: Tetra Tech, December 2021

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

The QP [Thomas L. Dyer, P.E.] has used measured and indicated resources provided by Tetra Tech to estimate mineral reserves. Pit optimization was done using Geovia's Whittle™ software to define pit limits with input for economic and slope parameters.

Optimization used only measured and indicated resources for processing. All inferred resources were 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.

Table 1-3: Statement of Mineral Reserve Estimate

	Batman Deposit			Heap Leach Pad			Total P&P		
	K Tonnes	g Au/t	K Ozs Au	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g/t)	Contained Ounces (000s)
Proven	81,277	0.84	2,192	-	-	-	81,277	0.84	2,192
Probable	185,744	0.76	4,555	13,354	0.54	232	199,098	0.75	4,787
Proven & Probable	267,021	0.79	6,747	13,354	0.54	232	280,375	0.77	6,979

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 0.35 g Au/t cutoff grade.
3) Deepak Malhotra is the QP responsible for reporting the heap-leach pad reserves.
4) Because all 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.
6) The effective date of the mineral reserve estimates is December 31, 2023

1.6.1 Heap Leach Reserve Estimate

Existing heap leach pad (HLP) reserves are provided in Table 1-3, 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 will undertake additional metallurgical test work at the targeted P_{80} 40 microns size at a later date, since this material will be processed in years 15 and 16 at the end of project life. However, for purposes of classifying the heap leach material as a reserve, the previous recovery values were used. The heap leach reserve constitutes approximately 3% of the total reserves quoted.

1.7 Mining Methods

The Project is designed to be a conventional, owner-operated, large open-pit mine 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 1-4.

Table 1-4: Initial Economic Parameters

Parameter	Value Used
Gold Recovery	Grade dependent constant tail equation
Payable Gold	99.9%
Reference Mining Cost	US$1.99 per tonne mined
Incremental Mining Cost	US$0.012 tonne per bench
Overall Mining Cost	US$2.35 per tonne mined
Processing Cost	US$11.20 per tonne processed
General & Administrative	$1.50 per tonne processed
Royalty[1]	1% GPR

The mining costs used were varied by bench. An incremental cost of US$0.012 was added for each six-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 below the 145-meter elevation. 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 of US$1.99 was determined using first principles from previous studies. The overall mining cost (reference plus incremental) is US$2.35 per tonne.

Processing, tailings construction, tailings reclamation, and water treatment costs were provided by Vista and their consultants. Calculated cutoff grade based on the economic parameters and a $1,750 gold price is 0.27 g Au/t. At Vista's request, the QP [Thomas L. Dyer, P.E.] used a minimum cutoff grade of 0.35 g Au/t. This was done to maintain higher grades with respect to material allowed to be processed. The elevated cutoff grade of 0.35 g Au/t is appropriate for the Project.

Several iterations of pit optimizations were reviewed to determine the final pit limits. A US$1,500/oz-Au pit shell was used to guide the ultimate pit design. Table 1-5 shows the Whittle™ optimization results. Note that the ultimate pit used for pit design is highlighted in green.

Table 1-5: Whittle™ Pit Optimization Results – Using 0.35 g Au/t Cutoff

Pit	Gold Price USD/oz Au	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio
		K Tonnes	g Au/t	K Ozs Au			
1	$ 300	173	1.97	11	97	269	0.56
5	$ 400	4,746	1.73	263	4,671	9,417	0.98
9	$ 500	9,246	1.55	461	9,975	19,222	1.08
13	$ 600	15,039	1.37	663	16,431	31,469	1.09
17	$ 700	25,305	1.21	986	31,484	56,789	1.24
21	$ 800	75,890	1.05	2,566	163,312	239,202	2.15

[1] Prior royalty used for initial Lerch-Grossman cone runs. Final designs use actual royalty data.

Pit	Gold Price USD/oz Au	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio
		K Tonnes	g Au/t	K Ozs Au			
22	$ 825	81,985	1.04	2,731	175,576	257,561	2.14
23	$ 850	88,612	1.02	2,907	189,939	278,551	2.14
24	$ 875	93,241	1.01	3,018	197,209	290,450	2.12
25	$ 900	100,529	0.99	3,209	215,507	316,036	2.14
29	$ 1,000	126,157	0.93	3,780	261,842	387,999	2.08
33	$ 1,100	160,130	0.88	4,548	351,498	511,628	2.20
37	$ 1,200	200,710	0.83	5,362	447,259	647,969	2.23
41	$ 1,300	234,102	0.80	5,992	530,199	764,301	2.26
45	$ 1,400	257,028	0.79	6,516	647,721	904,748	2.52
49	$ 1,500	267,434	0.79	6,757	711,493	978,927	2.66
52	$ 1,600	275,395	0.79	6,952	776,204	1,051,600	2.82
55	$ 1,700	278,304	0.78	7,014	795,653	1,073,956	2.86
57	$ 1,750	281,900	0.78	7,105	831,474	1,113,374	2.95
59	$ 1,800	282,278	0.78	7,112	833,339	1,115,617	2.95
63	$ 1,900	286,188	0.78	7,208	875,572	1,161,760	3.06
66	$ 2,000	288,273	0.78	7,256	897,322	1,185,596	3.11
69	$ 2,100	289,809	0.78	7,291	914,481	1,204,290	3.16
72	$ 2,200	290,102	0.78	7,296	916,368	1,206,470	3.16
75	$ 2,300	293,182	0.78	7,370	958,631	1,251,814	3.27
79	$ 2,400	294,574	0.78	7,397	973,287	1,267,862	3.30
82	$ 2,500	296,705	0.78	7,443	1,003,010	1,299,714	3.38

Pit 49 was used for design purposes.

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.

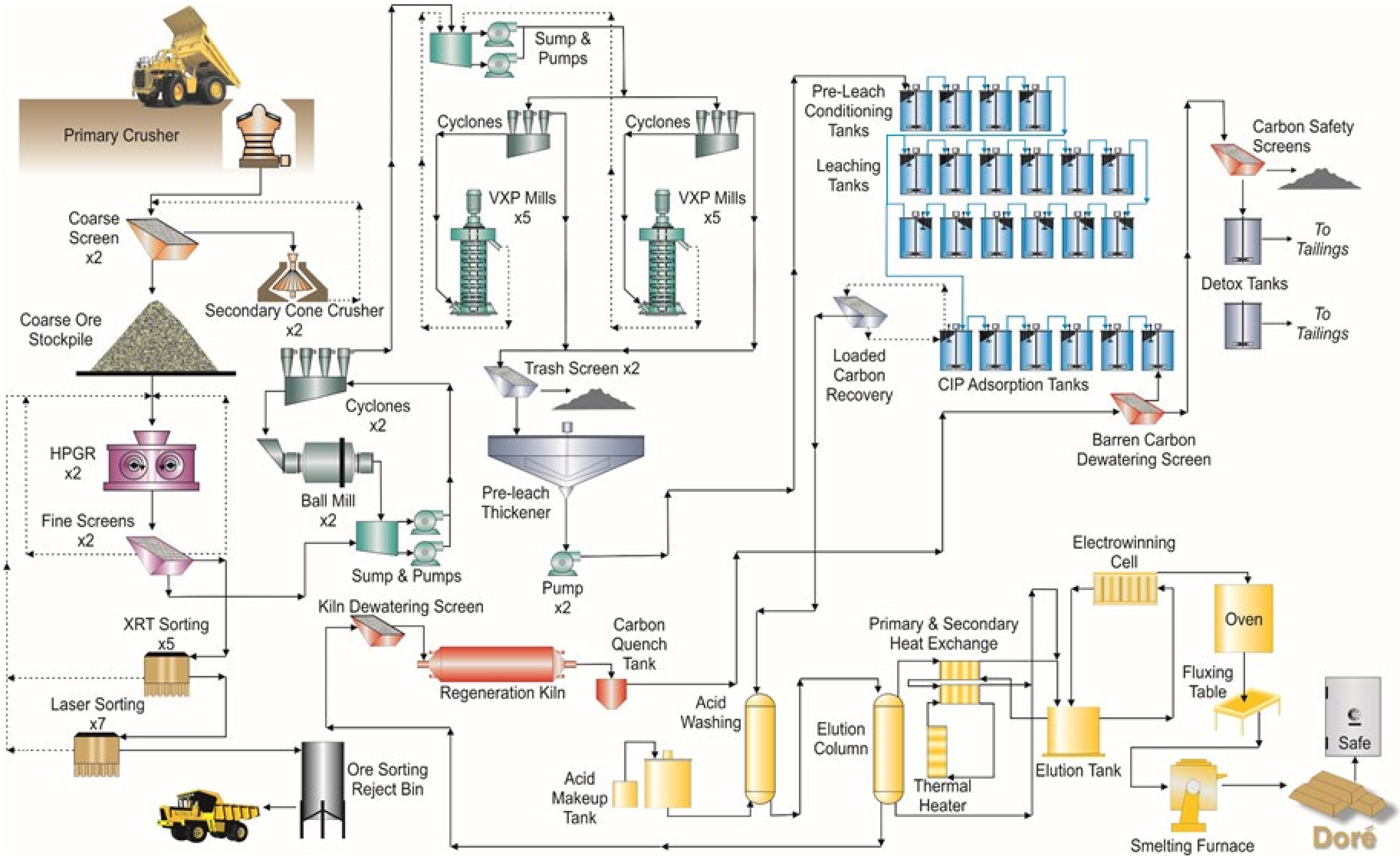


Figure 1-5: Mt Todd Flowsheet

1.9 Mineral Processing

Detailed design criteria have been developed for the process plant. The nominal headline design criteria are listed in Table 1-6 below.

Table 1-6: Headline Design Criteria

	Unit	Value Used
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	1,211
Gold Production (average)	oz/a	430,050

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 historical 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 (supplied by a third-party supplier by contract);
- Pit Dewatering;
- Mine Services;

- Communications;
- Gatehouse;
- Emergency Services Building;
- Administration Building;
- Process Plant Office;
- Process Plant Workshop;
- Process Plant Control Rooms;
- Sample Preparation and Laboratory; 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 strong, with prices sustained in the range of $1,900–$2,050 per ounce. The gold price used in this Technical Report is US$1,800/oz. Detailed information used for the determination of the minable reserves can be found in Section 15.1—Pit Optimization of this Feasibility Study.

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 FS-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;
  - Third Party Power Generation (build, own, operate)
  - 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 Aboriginal Community. Areas of aboriginal significance have been designated, and the mine plan has avoided development in these restricted works areas. Vista modernized its agreement with the JAAC in November 2020 and works closely with the JAAC and its representatives in many social and economic matters.

1.12.3 Approvals, Permits and Licenses

The Project will require approvals, permits and licenses for various components of the Project. Table 1-7 includes a list of approvals, permits, and licenses required for the Project and their current status.

Table 1-7: 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 Plan Approval from NT Department of Primary Industry and Resources	Approval April 2021 based on a 50kt/day operation. An amendment will need to be submitted for the minor changes as a product of the transition from PFS to this FS.	Approved Jun. 2021	NA
Heritage Act permit to destroy or damage archeological sites and scatters/ Aboriginal Areas Protection Authority Clearances	Authority Certificate Number C2021/028 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 and exploration activities.	Aboriginal Areas Protection Authority dated Jun. 07 2021	NA
Aboriginal Areas Protection Authority Certificate	The use of, or work on, certain areas can proceed without a risk of damage to, or interference with, the sacred sites identified at Mt Todd. Covers the mining licenses and the 1,581 km^2 of exploration licenses contiguous with the mining leases.	Jun. 7, 2021	NA
Surface Water Extraction License	Provides the right to annually harvest 3.48 gigaliters of surface run-off to use for mine operations.	Jun. 1, 2021	Jun. 1, 2031

Approval/Permit/License	Current Status	Approval/ Permit License Date	Expiration Date
Approval to reopen and operate the existing Mt Todd Gold Mine	Approved in accordance with Part 9 of the Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act) by the Australian Department of the Environment and Energy – EPBC Ref: 2011/5967	Jan. 19, 2018	NA
Permit to Interfere with a Waterway Diversions – Approval from Department of Environment, Parks and Water Security	Assessment done as part of the MMP assessment in 2021, including a site visit. Approval IWW:VDG-001 Diversions	Approved Feb. 03, 2022	N/A
Permit to Interfere with a Waterway RWD – Approval from Department of Environment, Parks and Water Security	Assessment done as part of the MMP assessment in 2021, including a site visit. Approval IWW:VDG-002 Dam	Approved Feb. 27, 2022	N/A
Dangerous Goods Act (1988) permit for blasting activities	On hold until FID	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
Water Extraction License Approval from Department of Environment, Parks and Water Security	Approved via License No: 8141014 issued for 3,480 ML/year to be harvested via the Raw Water Dam	Jun. 01 2021	Jun. 01 2031
Waste Discharge License (under Section 74 of the Water Act 1992) for management of water discharge from the site	WDL 178-8 licensing discharge of treated water into the Edith River from the Mt Todd mine site, granted with conditions	Nov. 30 2020	Revoked at our request in 2021 as not required until operational
Waste water treatment system permits under Public Health Act 1987 and Regulations	Required for the waste water treatment system for the construction and operations accommodation village. Permit application not yet in progress pending FID.	NA	NA
Approval to Disturb Site of Conservation Significance (SOCS)	Batman pit expansion will disturb SOCS as breeding/foraging habitat for the Gouldian finch. Plan has been approved via EPBC 2011/5967 An extension will be applied for late 2022.	Jan. 19, 2018	Jan. 2023

1.13 Capital and Cost Estimates

1.13.1 Capital Cost Estimates

LoM capital cost requirements are estimated at US$1,746 million as summarized in Table 1-8. Initial capital of US$1,030 million is required to commence operations. At the end of operations, the Project will receive an estimated US$43 million credit for asset sales and salvage.

Table 1-8: 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	6.2%	$94,127	$5,738	$99,865	$583,957	$36,152	$620,109	$678,084	$41,890	$719,974
3000	Process Plant	11.0%	$560,796	$61,691	$622,487	$33,498	$3,539	$37,036	$594,294	$65,229	$659,523
4000	Project Services	9.6%	$56,893	$6,625	$63,518	$86,468	$7,082	$93,551	$143,361	$13,707	$157,069
5000	Project Infrastructure	10.5%	$49,389	$5,203	$54,592	$7,502	$773	$8,275	$56,891	$5,976	$62,867
6000	Permanent Accommodation	10.0%	$422	$42	$464	$0	$0	$0	$422	$42	$464
7000	Site Establishment & Early Works	12.6%	$26,553	$3,334	$29,886	$0	$0	$0	$26,553	$3,334	$29,886
8000	Management, Engineering, EPCM Svcs	12.0%	$111,185	$13,384	$124,569	$0	$0	$0	$111,185	$13,384	$124,569
9000	Pre-Production Costs	10.0%	$31,071	$3,098	$34,169	$0	$0	$0	$31,071	$3,098	$34,169
10000	Asset Sale	0.0%	$0	$0	$0	($42,756)	$0	($42,756)	($42,756)	$0	($42,756)
	Capital Cost	**9.2%**	**$930,436**	**$99,114**	**$1,029,550**	**$668,670**	**$47,546**	**$716,216**	**$1,599,106**	**$146,660**	**$1,745,766**

1.13.2 Operating Cost Estimates

LoM operating costs requirements are estimated to be US$19.33/t-milled as summarized in Table 1-9.

Table 1-9: Estimated LoM Operating Costs (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.10	0.03
Mine Maintenance	0.16	0.05
Engineering	0.06	0.02
Geology	0.04	0.01
Drilling	1.12	0.33
Blasting	1.20	0.36
Loading	0.81	0.24
Hauling	3.69	1.10
Mine Support	0.50	0.15
Mine Dewatering	0.01	0.01
Open Pit Mine	**7.68**	2.30
CIP PROCESS PLANT		
Labor	0.90	-
3100 - Crush/Screen/Stockpile	0.48	-
3200 - Reclaim & HPGR	0.78	-
3300 - Classification & Grinding	4.17	-
3400 - Pre-Leach,Thick/Aeration/CIP	0.21	-
3500 - Desorption, Gold Room	0.03	-
3600 - Detox & Tailings Pumping	0.09	-
3700 - Reagents	3.47	-
3800 - Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.02	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**10.30**	-
Project Services	0.30	-
G&A	1.05	-
Operating Costs	**19.33**	**-**

1.14 Financial Analysis

Estimated economic results are summarized in Table 1-10. The analysis suggests the following conclusions, assuming a 100% equity project, a gold price of US$1,800/oz and a US$0.69:AUD1.00 exchange rate:

- Mine Life: .... 16 years;
- Pre-Tax NPV5%: .... US$2,149.4 million, IRR: 29.4%;
- After-tax NPV5%: .... US$1,131.4 million, IRR: 20.4%;
- Payback (After-tax): .... 4.0 years;
- NT Royalty Paid: .... US$765 million;
- Australian Income Taxes Paid: .... US$927 million; and
- Cash costs (including Royalties): .... US$913/oz-Au.

Table 1-10: Estimated Technical-Economic Results (US$000s)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
GOLD SALES			
Gold Produced (koz)	6,313	-	-
Gold Price (US$/oz)	1,800	-	-
Gold Sales	**11,364,288**	**40.53**	**1,800.00**
REFINING & ROYALTIES			
Refinery Costs	(23,206)	(0.08)	(3.68)
Royalties	(324,038)	(1.16)	(51.32)
Gross Income from Mining	**11,017,044**	**39.29**	**1,745.00**
OPERATING COSTS			
Open Pit Mine	(2,153,191)	(7.68)	(341.05)
CIP Process Plant	(2,889,166)	(10.30)	(457.62)
Project Services	(84,130)	(0.30)	(13.33)
G&A	(293,212)	(1.05)	(46.44)
Operating Costs	**(5,419,700)**	**(19.33)**	**(858.43)**
Cash Cost of Goods Sold (COGS)	**(5,442,905)**	**(20.57)**	**(862.11)**
Operating Margin	**5,597,345**	**19.96**	**886.57**
CAPITAL COSTS			
Mining	719,974		
Process Plant	659,523		
Project Services	157,069		
Project Infrastructure	62,867		
Permanent Accommodation	464		
Site Establishment & Early Works	29,886		
Management, Engineering, EPCM Services	124,569		
Pre-Production Costs	34,169		
Asset Sale	(42,756)		

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Capital Costs	**1,745,766**		
Pre-Tax Cash Flow	3,851,579		
$NPV_{5\%}$	2,149,401		
IRR (%)	29.4%		
After-tax Cash Flow	2,160,177		
$NPV_{5\%}$	1,131,432		
IRR (%)	20.4%		
After-tax Payback (years)	4.0		

1.15 Conclusions and Recommendations

All of the required test work is completed for this FS and no additional work is necessary for this level of study. This FS presents a project that is ready for submission for financial and other support necessary for initiation.

2. INTRODUCTION

Vista Gold Corp. and its subsidiaries (collectively, "Vista" or the "Company) operate in the gold mining industry. The Company's flagship asset is its 100% owned Mt Todd Gold Project (Mt Todd) in the Northern Territory (NT) Australia. The Company recently received authorization for the last major environmental permit and completed this PS, as amended, for Mt Todd, which confirms the project's robust economics at today's gold price and within the current cost environment. 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 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 retained Tetra Tech, along with RESPEC (formerly Mine Development Associates (MDA)), Resource Development Inc. (RDi), Pro Solv Consulting, LLC (Pro Solv), and Tetra Tech Proteus (TTP) to prepare this FS for its Mt Todd Gold Project (the Project) in Northern Territory (NT), Australia. The FS (Technical Report) evaluates a development scenario of a 50,000 tonne per day (tpd) processing facility.

The 50,000 tpd operation includes:

- Estimated proven and probable reserves of 6.98 Moz of gold (267 Mt at 0.77 g Au/t) at a cut-off grade of 0.35 g Au/t;
- Average annual production of 395 koz of gold per year over the mine life, including average annual production of 479 koz of gold per year during the first seven years of operations;
- LoM average cash costs of US$913 per ounce, including average cash costs of US$845 per ounce during the first seven years of operations;
- A 16-year operating life;
- After-tax NPV5% of US$1,131 million and internal rate of return (IRR) of 20.4% at US$1,800 per ounce gold prices and US$0.69:AUD1.00 exchange rate, and
- Initial capital requirements of US$1,030 million.

2.2 Terms of Reference and Purpose of the Report

This Feasibility Study was prepared as an 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.

This 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 QP, 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.

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—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 Q4 2021 U.S. dollars (US$) unless otherwise stated.

2.5 Detailed Personal Inspections

1) Dr. Rex Bryan visited and inspected the property from September 12–14, 2011 and February 6–8, 2013. Dr. Bryan last visited and inspected the property June 28–29, 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) Thomas Dyer visited and inspected the subject property during March 2011. Thomas Dyer last visited and inspected the property June 28–29, 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.
3) Chris Johns visited and inspected the property June 28–29, 2017. Mr. Johns inspected the existing Tailings Storage Facility 1 (TSF 1) and the proposed site for Tailings Storage Facility 2 (TSF 2).
4) Zvonimir Ponos last visited and inspected the property June 28–29, 2017. Mr. Ponos inspected the existing site infrastructure and process facility.
5) Vicki J. Scharnhorst visited and inspected the property June 28–29, 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. Due to Covid restrictions in 2020 and 2021, the above QPs were unable to travel to the site for this update to the Technical Report.

The QPs consider the 2017 site visits current personal inspections on the basis that the work completed on the property since that time has been reviewed and the QPs are of the opinion that the limited work carried out on the property since 2017 is not material. The QPs are satisfied that no unauthorized access or other work has been conducted on the property based on the site security including site access via a paved road through a locked security gate combined with the fact that the site is continuously manned by company personnel. Further, the Jawoyn Association Aboriginal Corporation (JAAC) rangers regularly patrol the area around the site. With regard to specific conditions at the site, the hardness and average grade of the Batman deposit rock make the potential for theft or high-grading by unauthorized persons very low. Finally, the QPs also review publicly available information on the Company and its activities including the audited financial statements of the Company, which the QPs are satisfied do not point to any additional work being conducted on the property.

3. 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 QP [Vicki J. Scharnhorst, P.E.] relied upon the following experts to prepare portions of Section 20—Environmental Studies, Permitting, and Social or Community Impact:

- Environmental Impact Statement for the Project prepared by GHD (June 2013) and the Flora and Fauna Management Plan (GHD, November 2018) were used to describe the existing environmental studies (Section 0—Environmental Studies)

The QP [Maurie Marks, P.Eng.] relied upon Vista and its management to prepare the owner costs, closure and reclamation security bond, and the applicable taxes and royalties used in the economic analysis and listed in Section 22—Economic Analysis and in different parts throughout the report.

4. PROPERTY DESCRIPTION AND 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.

4.1 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, EL 32004, and ELA 32005, comprising approximately 158,131 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.

4.2 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, 2017, and again in May 2023, extending it to December 31, 2029, with a 3-year option thereafter.

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, makes a definitive investment decision, and commences construction.

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. In November 2020 Vista and the JAAC modernized the 2006 JAAC agreement. The parties agreed to replace the 10% participating interest right previously granted to the JAAC with a sliding-scale gross process production royalty that can vary between 1/8% and 2% depending on gold price and foreign exchange rates. This production royalty is in addition to the 1% gross proceeds royalty previously granted to the JAAC.

Vista Australia entered into a royalty agreement ("Royalty Agreement") with Wheaton Precious Metals (Cayman) Co., an affiliate of Wheaton Precious Metals Corp. ("Wheaton") in relation to Mt Todd. Pursuant to the terms of the Royalty Agreement, Wheaton is entitled to receive 1% of the gross revenue from Mt Todd (the "Royalty") if the defined completion objectives for the Project are achieved by April 1, 2028. Beginning April 1, 2028, if the completion objectives are not achieved, the Royalty shall increase annually at a rate of up to 0.13% to a maximum Royalty rate of 2%. Any annual increases beginning April 1, 2028 shall be reduced on a pro rata basis to the extent that Mt Todd has initiated operations but has yet to achieve a completion test at an average daily processing rate of 15,000 tonnes per day. The Royalty rate, the annual increase percentage, and maximum Royalty rate can each be reduced by one-third upon the occurrence of one of the following events: (i) a change of control of Vista Gold Australia occurs prior to April 1, 2028 and Vista Australia provides timely notice and payment to Wheaton of certain amounts; or (ii) payment to Wheaton of the applicable Royalty associated with Vista Australia delivering 3.47 million gold ounces to a third party. The Royalty is payable on production from the Mt Todd mining and exploration licenses.

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 any presently identified mineral resources or mineral reserves at Mt Todd.

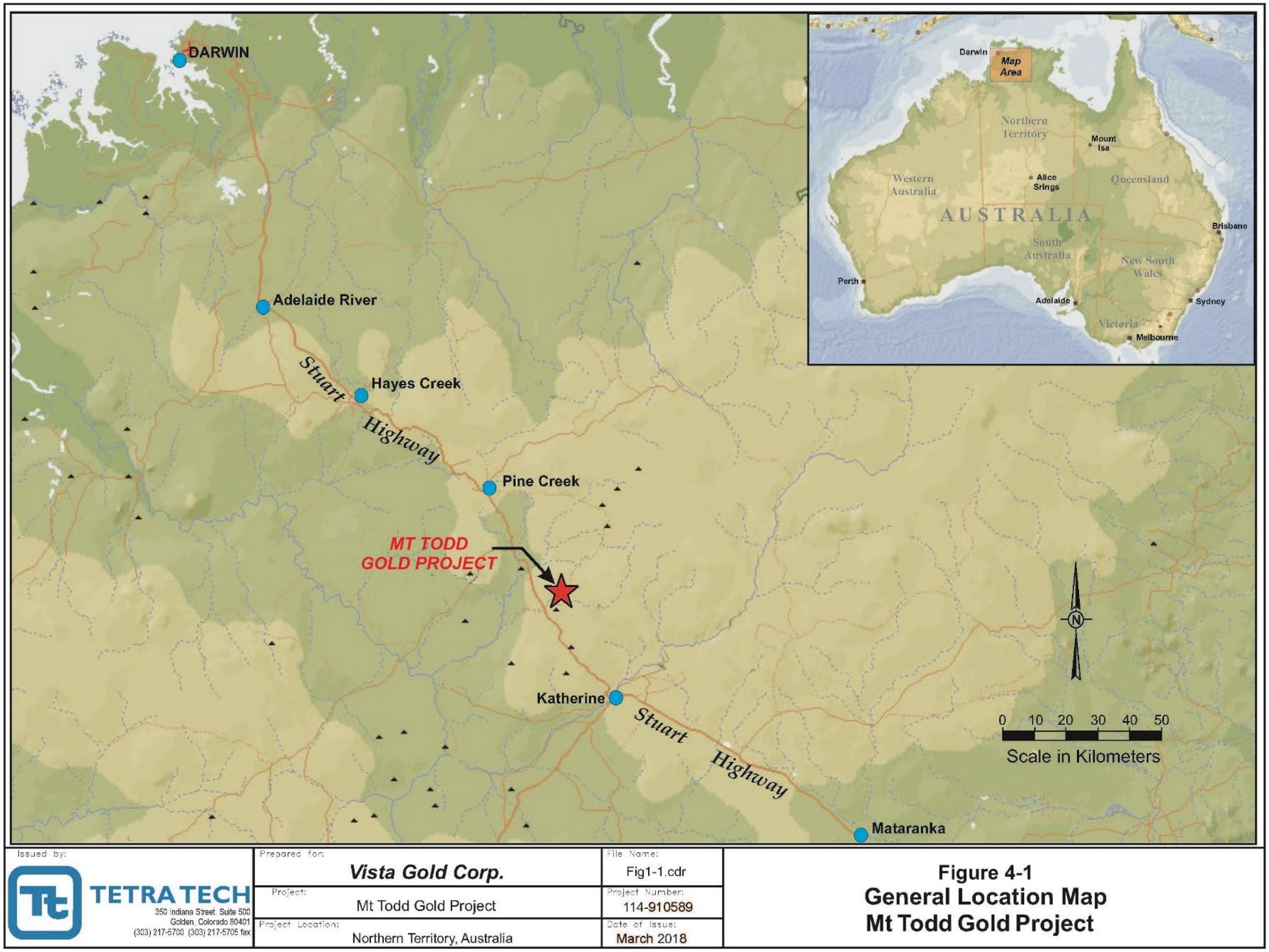
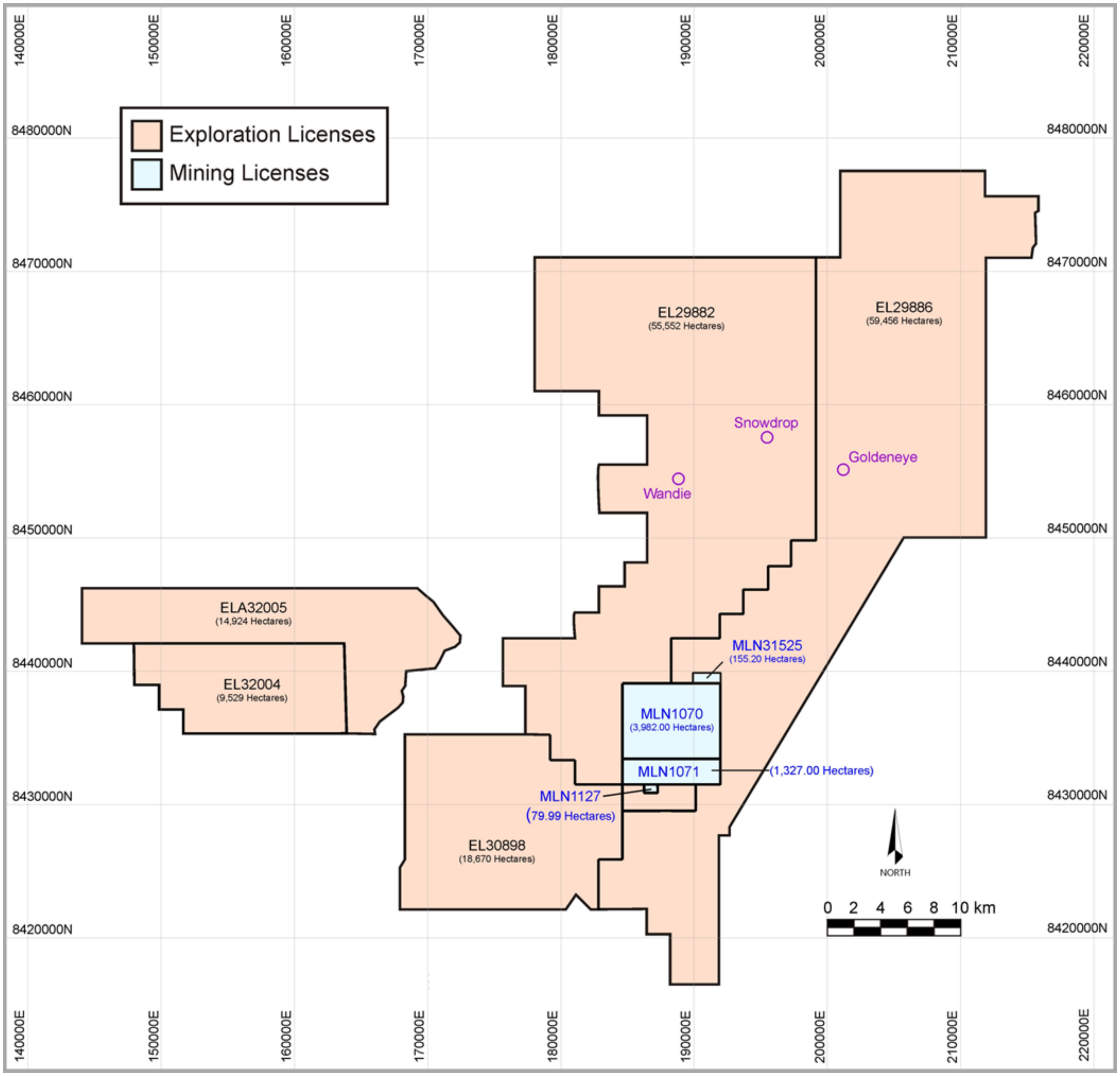


Figure 4-1: General Project Location Map



NOTE: Prepared by Vista Gold Corp.; updated January 2022

Figure 4-2: Concessions

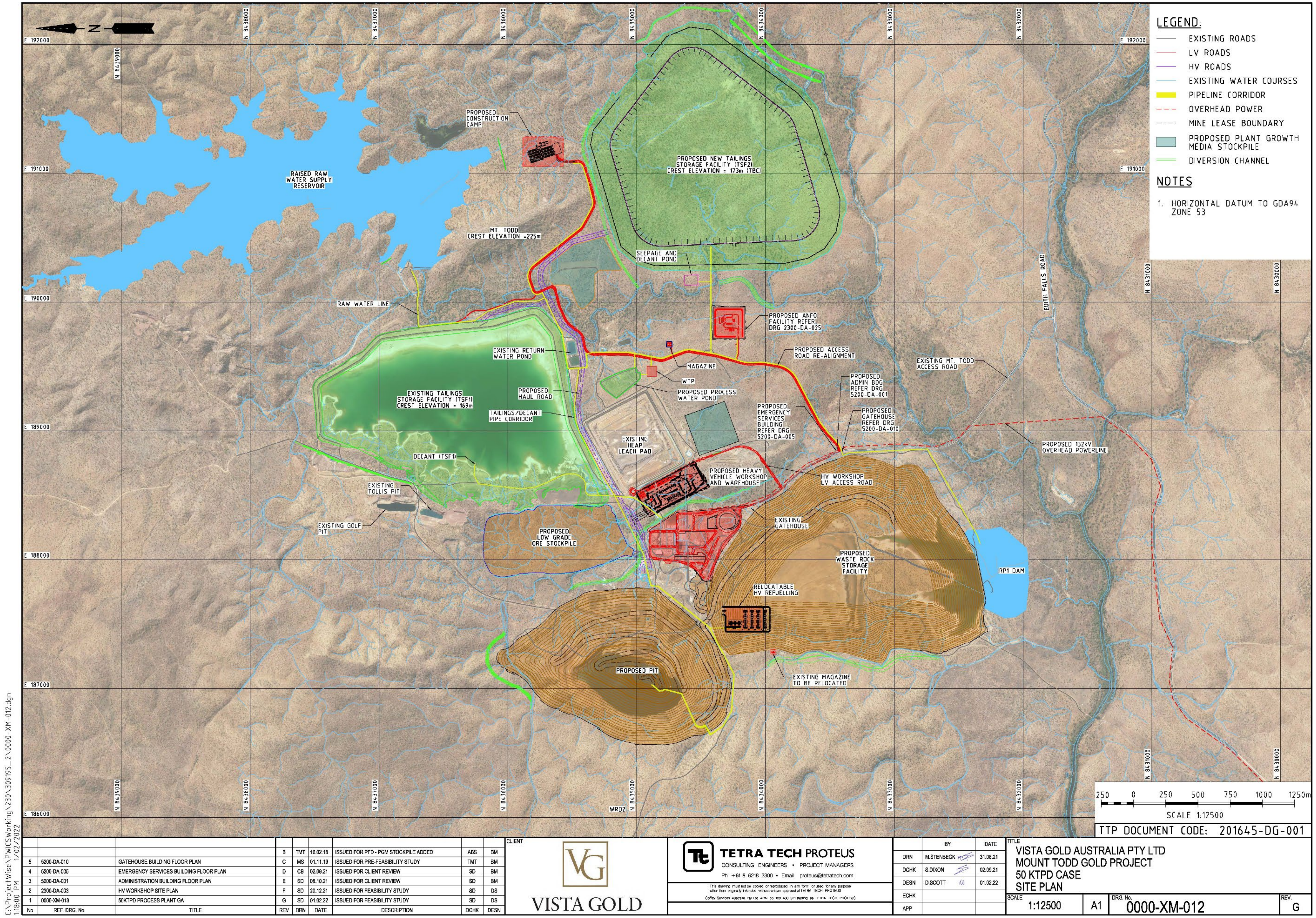


Figure 4-3: General Arrangement

4.3 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 an 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 received approval of the Mt Todd Project MMP in June 2021; an amended MMP will be submitted subsequent to feasibility design. With the approval of the MMP, Vista is now in possession of all major permits required to start development.

5. 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 Northern Territory 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 historical 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. 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.

Preliminary studies 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.

Historical production is shown in Table 6-1.

Table 6-1: Heap Leach – Historical Actual Production

Category	Historical 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 historical production numbers that pre-date Vista's ownership. The QPs and issuer consider historical 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 several 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. Several 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 (historical reported production, presented for context). 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) – B P18-70, (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.

2006-2023	
	Vista Gold Corp. completed drilling campaigns, produced environmental, economic, geotechnical, regulatory, and required studies. Vista undertook remediation of Batman Pit water. A series of NI 43-101 reports were produced over the period with increasing detail.

6.2 Historical Drilling

The following discussion centers on the historical 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 opinion of the QP [Rex Clair Bryan, Ph.D., SME RM] that the drillhole databases used as the bases of this report contain all of the available data. The QP is unaware of any drillhole data that have been excluded from this report.

6.2.1 Batman Deposit

There are 730 historical 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 to 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 most of the 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, "*the 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 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

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

The QP [Rex Clair Bryan, Ph.D., SME RM] for this section has reviewed the Snowden (1990) report which completed a statistical study of the Quigleys drillhole database to bias test it. The report included a comparison of historical and recent data by Snowden which suggested that a bias might exist. Further study by Snowden 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 QP [Rex Clair Bryan, Ph.D., SME RM] has reviewed and concurs with this information. The March 14, 2008 report entitled "Mt Todd Gold Project, Gold Resource Update, Northern Territory, Australia, NI 43-101 Technical Report" prepared by John W. Rozelle contains additional information regarding the Snowden findings summarized above.

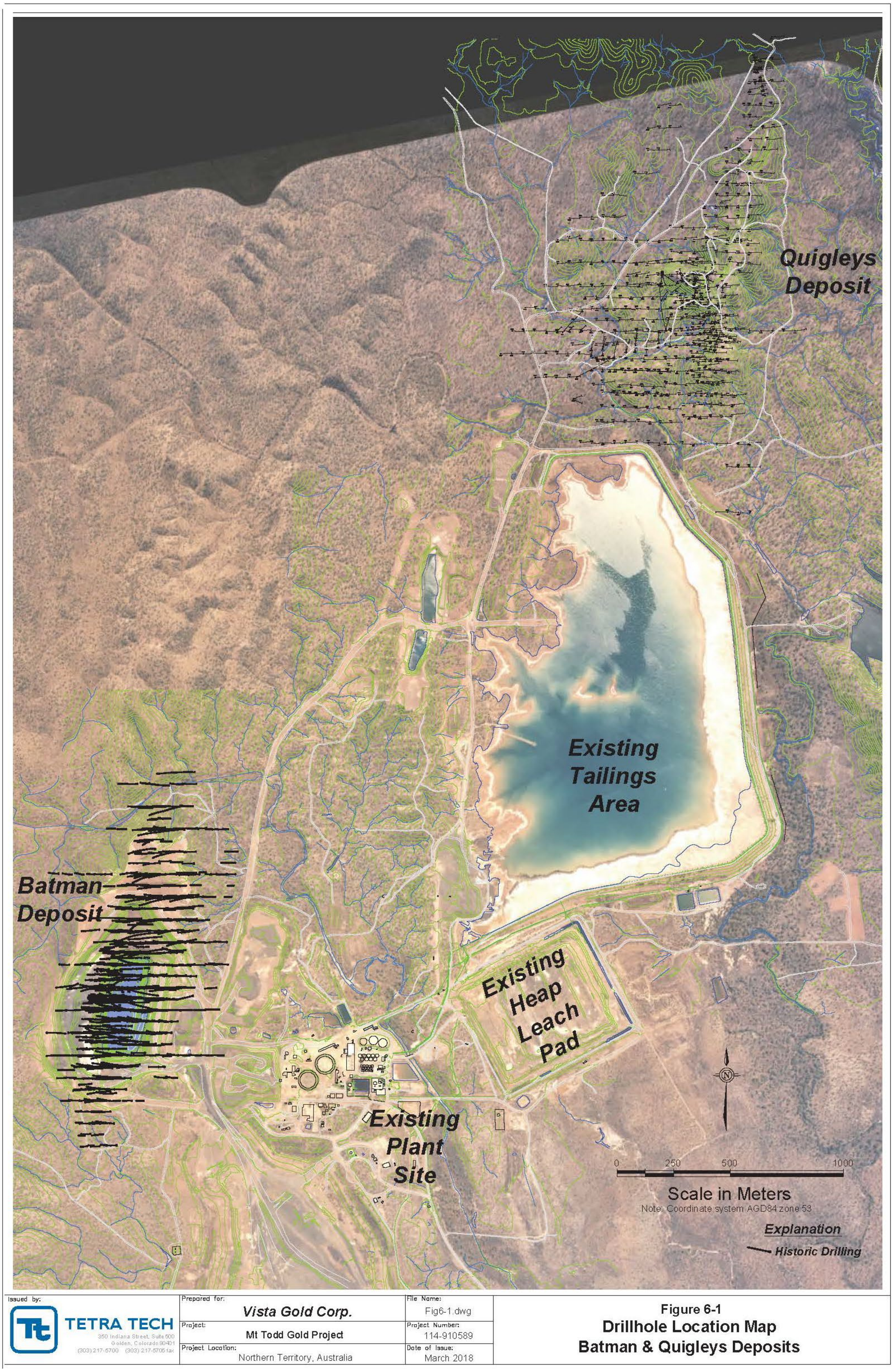


Figure 6-1: Drillhole Location Map – Batman and Quigleys Deposits

6.3 Historical 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, the QP [Rex Clair Bryan, Ph.D., SME RM] did not witness any drilling and sampling personally. The QP has 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 Historical 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, several 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 several 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, most 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 Historical 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 historical 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.

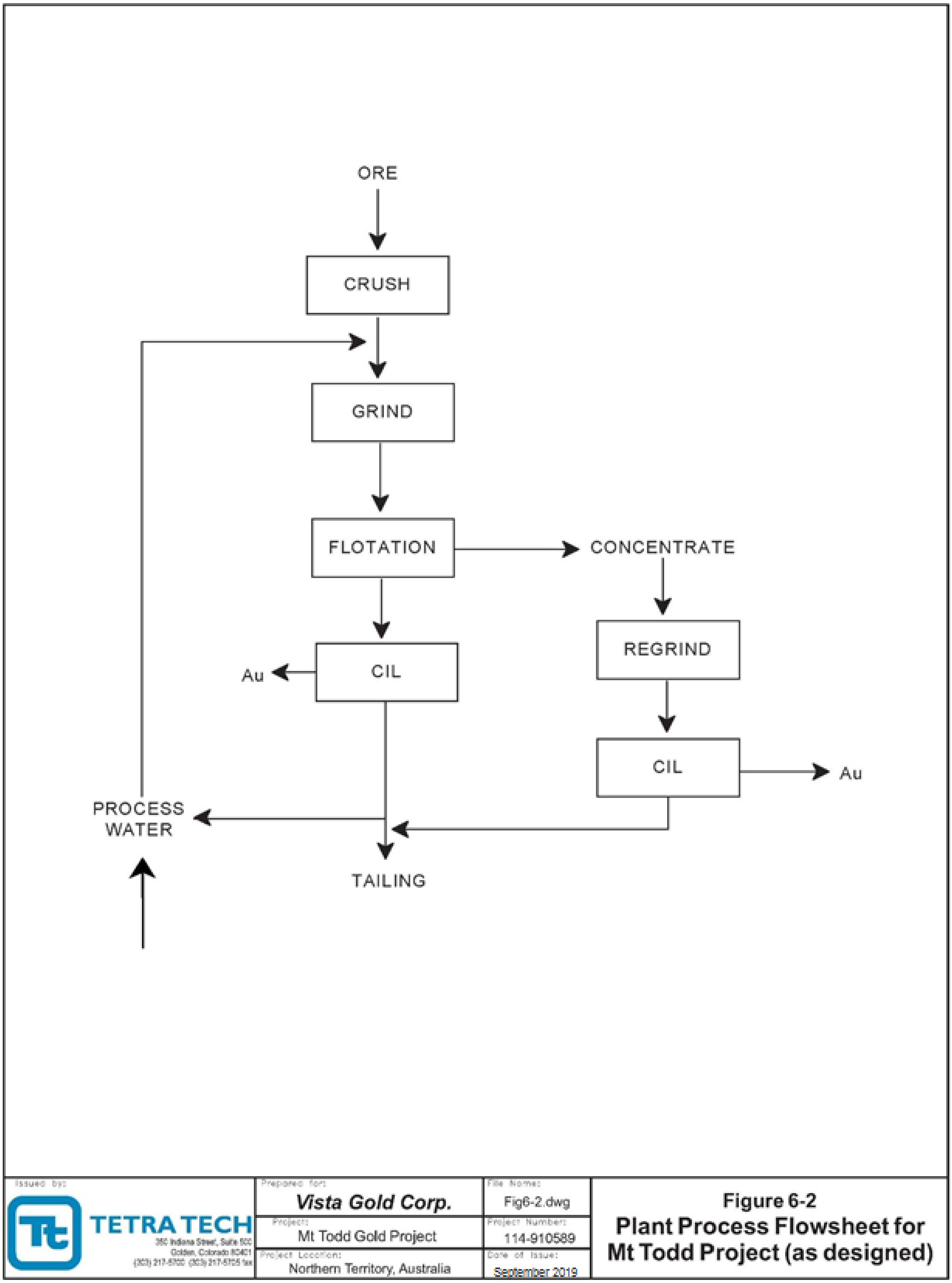


Figure 6-2: Plant Process Flowsheet for Project as Designed

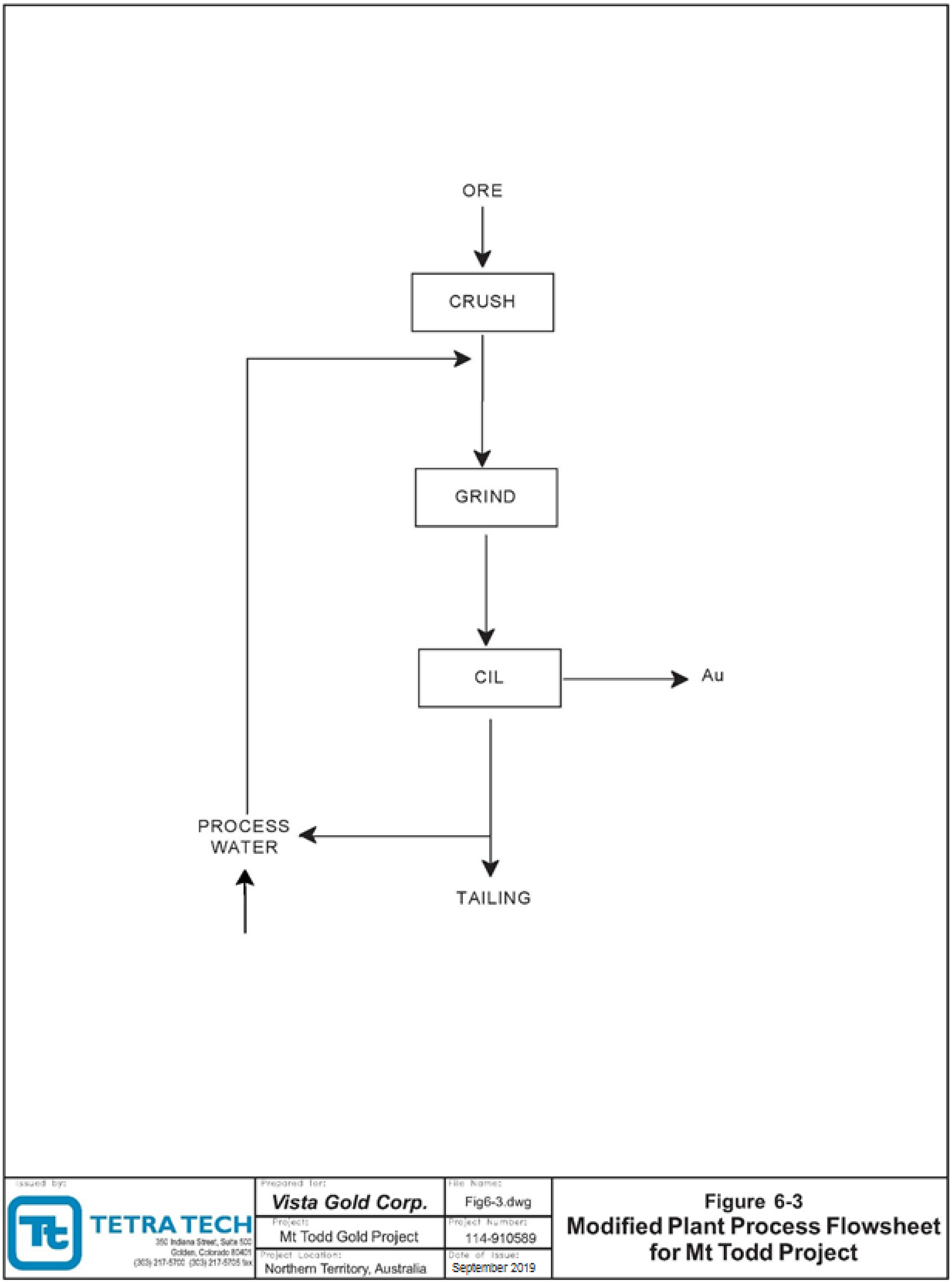


Figure 6-3: Modified Plant Process Flowsheet for Project

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 cyanide in process water had been detoxified, the problems would not have occurred. This was not done because of the cost associated with a cyanide detoxification circuit.

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 Vista's 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. 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

Unit Code	Lithology	Description
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.

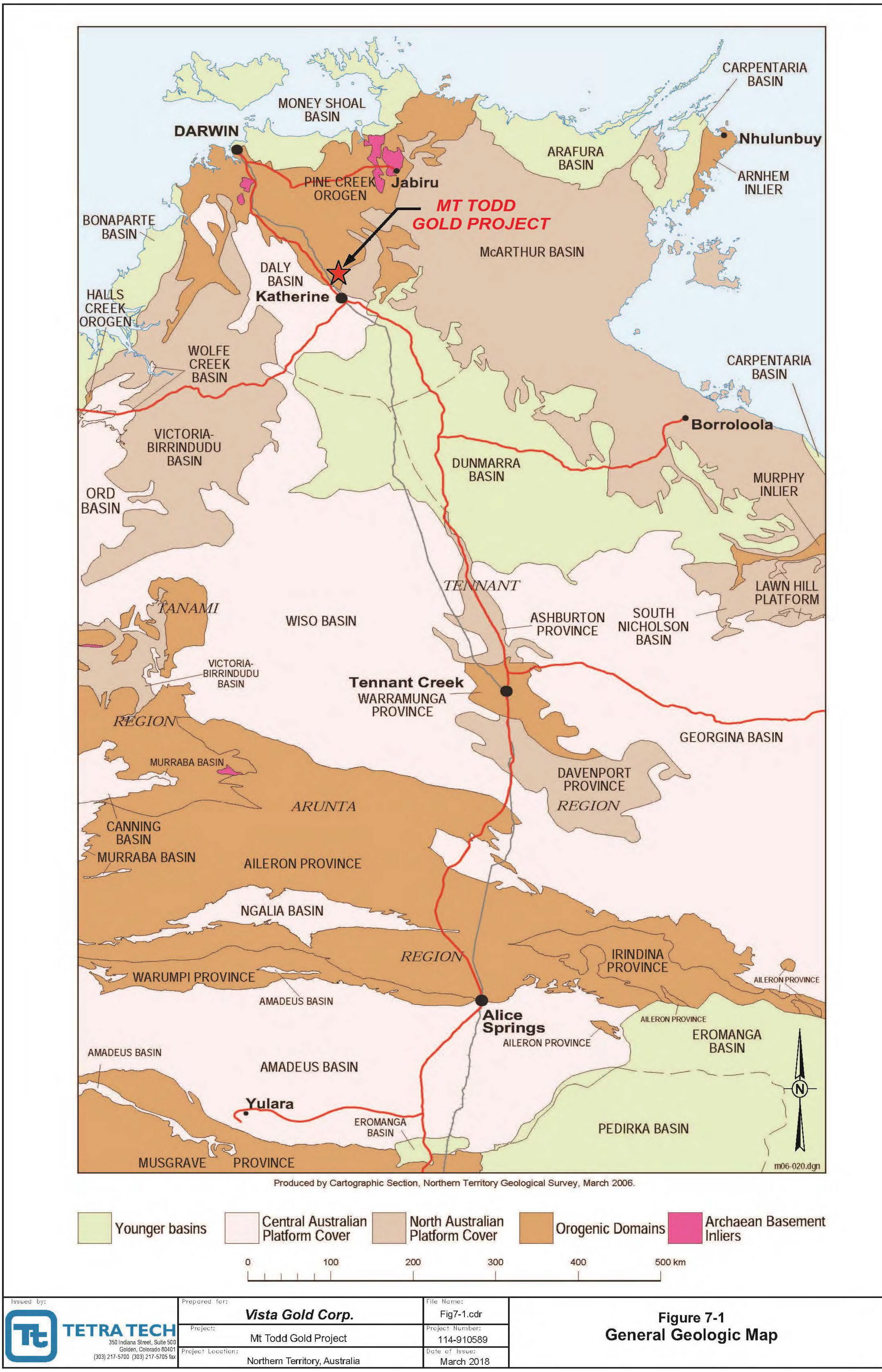


Figure 7-1: General Geologic Map

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. The QP [Rex Clair Bryan, Ph.D., SME RM] has reviewed and concurs with this information.

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 based on 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. The QP agrees with the conclusion of the Snowden report 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 currently.

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.

8. 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). 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—Exploration and Section 14—Mineral Resource Estimates, respectively.

9. 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 historical work in the area and site visits. To date, two of the geophysical targets (Golden Eye and Snowdrop) have been drilled 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	Soil Samples Collected	Rock Chip 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	313	170
2020	278	9
2021	0	11
2022	60	556
Total Samples	**15,050**	**2,037**

Within the same ELs, Vista Gold obtained 654 soil samples and 222 rock-chip samples in an exploration program between March 2, 2018 and October 7, 2019. Table 9-2 lists the type, sample count and general location. Table 9-3 presents information on known exploration prospects.

Table 9-2: Exploration Sampling Between 2018 and 2019 by Target Area

Type	Start Date	End Date	Location	Count
Soil	07/14/2018	07/28/2018	Wandie Creek NW infill	231
Soil	07/27/2018	07/29/2018	SW of Crest of the Wave	109
Soil	01/01/2019	01/03/2019	Batman North	77
Soil	02/10/2019	10/05/2019	Blue Sage	237
Total Soil	**07/14/2018**	**10/05/2019**	**All Soil Areas**	**654**
Rock Chip	03/02/2019	10/07/2019	Multiple Tenements	222
Total Chip	**03/02/2019**	**10/07/2019**	**All Rock Chip**	**222**

Table 9-3: 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

Year	Drill Hole	LOCATION		Zone	GDA94 COORDS		TASKS COMPLETED		
		Prospect	Lease No		Easting	Northing	RL	Depth	Rehab Status
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	
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 70 ppb and one of 50 ppb, 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 mining 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 are 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, like 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.

9.5 Sample Preparation Methods and Quality Control (QC) Measures

Soil samples were planned on a regular grid and a sample sheet is generated, GPS is used to locate sample positions and a pelican pick is used to clear debris and any topsoil from the sample location, hole is dug to the B horizon and 7 to 10 kg of soil is collected and coarse sieved to remove stones etc., a fine mesh is then employed, and the entire sample recovered post sieving is bagged. Soil sampling is usually undertaken in the dry season, however if wet samples are obtained, they are dried in the logging shed prior to sieving. Sample bags are calico and purchased pre-numbered, these are then place 5 each in green plastic bags for transportation to the Assay lab. As the site is closed to public access, no special security measures are undertaken. A sample submission sheet is sent to the lab, detailing required methodology, and number of samples. There is no identifying data relating to sample location on the bags submitted or the paperwork beyond bag numbers. It is the author's opinion that the sample preparation methods and quality control measures employed before dispatch of samples to an analytical or testing laboratory ensured the validity and integrity of samples taken.

9.6 Relevant Information Regarding Sample Preparation, Assaying, and Analytical Procedures

Repeat samples and standards are employed in soil sampling programs, with blind repeats being the most effective, as standards are easily distinguishable from raw samples by the lab. The lab conducts its own QA/QC of which it provides the data to Vista Gold. All sample preparation and analytical work is performed at North Australia Assay laboratories, in Pine Creek MLN, 792 Eleanor Rd, Pine Creek NT 0847. The laboratory is owned and managed by Ray Wooldridge (MRACI, FAusIMM) who has 40+ years' experience in mineral Chemistry. Anomalous samples are re-assayed at the lab with up to 5 repeats being performed if repeatability is poor. The soil samples are retained onsite bagged and placed in bulk container bins and forklifted onto a site vehicle for transport to the lab, the samples are removed and run as a batch at the lab. Low-level assay work is conducted exclusively to minimize the chance of contamination.

Relevant QA/QC standards were applied to the soil sampling that is utilized as a tool to determine the geographical extent and magnitude of possible mineralization. Typically, a 100m x 100m grid is sampled over a broad target, with 20m x 20m infill spacing being used as follow-up, or to better define the extent of any anomalism identified. Duplicate field samples are undertaken, and highly anomalous field samples are investigated by the geologist and may be repeat sampled. The soils database has been designed to allow the date, batch number and associated repeats to be queried direct from database. This is an enhancement to the previous methodology of using an excel spreadsheet, which lends itself to copy/paste errors and makes analysis and reporting of QA/QC on the soils difficult. It is recommended by the author that soils, rock-chip and drill core assaying performed in the future to be subject to a monthly review with standardized reporting forms for QA/QC. This will ensure that any problems are identified rapidly as opposed to during the project analysis phase. Security onsite and at the lab is currently adequate but it is recommended that lockable sample transport boxes be employed in the same manner as drill core. The QP [Rex Clair Bryan, Ph.D., SME RM] is of the opinion that the preparation, analytical, and security procedures followed for the samples are sufficient and reliable for the purpose of exploring for potential drilling targets. The QP is of the opinion that these samples are representative, and no factors were identified that would result in sample bias.

10. DRILLING

10.1 Drilling

See also the historical drilling section (Section 6.2) for additional information on the 730 drillholes from various drilling campaigns before Vista from 1988 to 2007. Section 10.1.1 summarizes Batman drilling from 1988 to 2017. Section 10.1.2 focuses on the Vista drilling at Batman from 2012 to 2021. Since 2017, the only drilling completed at Mt Todd has been metallurgical sample collection drilling and not exploration drilling.

The QP [Rex Clair Bryan, Ph.D., SME RM] has reviewed the methodology and results of drilling and 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.

10.1.1 Summary of Batman Drilling 1988-2017

Table 10-1 shows a summary of Batman drilling from 1988 to 2017. Note that a large percentage of the historical drilling was by reverse circulation (RC) of less than 100 meters in depth. That RC drilling was used for ore grade control during the mining operations of Pegasus and General Gold Resources. Vista's drilling discovered a larger Batman resource by probing deeper with diamond drilling averaging 550 meters in depth.

Table 10-1: Batman Deposit Drilling History

Date	Reference	Holes (#)	Percussion (m)	Diamond (m)	RC (m)
1988	Truelove	17	1,475		
1989	Kenny, Wegmann, Fuccenecco	133	6,263	8,562	3,065
1990	Wegmann, Fuccenecco, Gibbs	122		5,060	8,072
1991	Billiton	149	501	202	3,090
1992	Zapopan	18		1,375	1,320
1993	Zapopan	16			2,814
1994-1997	Pegasus Gold	170			22,534
1998-2000	General Gold Resources	105		7,436	26,365
2007	Vista	25		9,883	
2008	Vista	16		8,938	
2010	Vista	12		6,864	
2011	Vista	7		4,480	
2012	Vista	27		17,439	
2015	Vista	5		3,185	
2016-2017	Vista	4		1,635	
2020-2022	Vista	26		8,887	
1988-2022	**Batman Total**	**852**	**8,239**	**83,946**	**67,260**

10.1.2 Vista Drilling 2012-2022

Between the fourth quarter of 2012 and the end of the first quarter of 2022, the Vista exploration program at the Batman deposit consisted of 52 diamond core drillholes containing 23,354.08 m that targeted both infill definitional drilling and step-out drilling. Table 10-2 lists 8 metallurgical diamond holes and 44 exploratory diamond holes drilled after 2015 that were not used in the resource estimation. These holes were used to help validate the current resource model. Data is not available as yet for a final hole VB21-014 drilled in 2021.

Table 10-2: 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
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
VB20-001**	187603.0	8435654.0	148.0	270.0	-58.0	362.8	Diamond
VB20-002**	187287.0	8435936.0	143.0	270.0	-58.0	280.0	Diamond
VB20-003**	187272.0	8435933.0	140.0	266.0	-54.0	299.8	Diamond
VB20-004**	187251.0	8435933.0	144.0	269.9	-50.0	148.0	Diamond
VB20-005**	187263.0	8435898.0	151.0	269.9	-61.0	197.9	Diamond
VB21-001**	187290.0	8345899.0	152.0	269.9	-61.0	234.5	Diamond
VB21-002**	187662.0	8436402.0	164.0	275.0	-40.0	458.6	Diamond
VB21-003**	187322.0	8435849	158.8	271.9	-62.0	285.7	Diamond

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB21-004**	187942.0	8436407.0	148.0	87.9	-50.0	410.8	Diamond
VB21-005**	187586.0	8436404.0	154.0	270.0	-50.0	445.7	Diamond
VB21-006**	187629.0	8435852.0	132.0	92.9	-50.0	347.7	Diamond
VB21-007**	187618.0	8436518.0	148.0	272.9	-50.0	299.9	Diamond
VB21-008**	187758.0	8436406.0	137.0	276.0	-48.0	477.3	Diamond
VB21-009**	188222.0	8436800.0	143.0	89.9	-50.0	437.5	Diamond
VB21-010**	188071.0	8436413.0	153.0	86.0	-50.0	417.4	Diamond
VB21-011**	187728.0	8436500.0	148.0	265.0	-50.0	398.8	Diamond
VB21-012**	188435.0	8436405.0	155.0	260.9	-50.0	901.2	Diamond
VB21-013**	187423.0	8436409.0	169.0	86.4	-53.0	311.9	Diamond
VB21-014**	187385.0	8436200.0	164.0	88.0	-50.0	368.75	Diamond
VB21-015**	187352.0	8436200.0	164.0	264.87	-55.0	341.69	Diamond
VB21-016**	188936.0	8437334.0	142.0	150.87	-68.0	449.53	Diamond
VB22-001**	187386.0	8436295.0	178.5	86.87	-56.0	203.76	Diamond
VB22-002**	188671.0	8437377.0	151.0	267.87	-55.0	116.67	Diamond
VB22-003**	187614.0	8436703.0	173.0	84.87	-56.0	281.80	Diamond
VB22-004**	187520.0	8436600.0	181.0	85.87	-55.0	224.71	Diamond
VB22-005**	187467.0	8436499.0	184.0	84.87	-56.0	220.94	Diamond

NOTE:
* Metallurgical drillholes are not used in the resource estimation.
** Exploratory drillholes extending north of the designed Batman pit—not used in the resource estimation
*** Data not available yet

Table 10-3 lists the complete set of drillholes used in the resource estimation. 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. Five sectional views are shown in Figure 10-2 through Figure 10-6. Drill hole traces are colored by drilling campaigns. Note the density of drilling and the scale of the maps obscures the names of the individual drill holes. Selected sectional views and other relevant results are also shown in Section 14—Mineral Resource Estimates.

Most of the drilling has been angled to be approximately perpendicular to the mineralized core. This orientation more accurately transects the true thickness of the mineralization. The Batman mineralization forms a set of stacked plates that strike to the north and plunges steeply to the east. These mineralized zones have been defined by wireframes which are used to constrain the higher grades for resource estimation shown in Figure 14-2. Early drilling sampled the deposit near the surface allowing for shorter drillhole depths. Exploring the deeper portions of the deposit has required drill collars to be offset to the east with longer drillhole lengths to reach the mineralized zone. Recent Vista drilling has targeted the deeper portions of the Batman deposit requiring the drillhole depths shown in Table 10-2. The positioning of the Vista drillhole collars has been constrained to be outside of the flooded historical mine pit. Most latter drilling has been oriented so as to transect the higher-grade mineralized zone.

While there are random high-grade intercepts outside of the core, the majority of higher-grade mineralization resides in the core.

Table 10-3: Batman Drillhole Details

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
01-MBP-001	187346.1	8435081.0	146.1	270.61	77.00	95.0
01-MBP-002	187326.1	8435081.0	145.7	270.61	77.00	42.0
01-MBP-003	187303.0	8435082.0	144.4	270.61	77.00	24.0
04-NSL-01	187552.7	8435508.0	140.0	270.61	61.69	191.8
04-NSL-02	187517.5	8435507.0	142.4	270.61	60.00	102.0
04-NSL-03	187491.6	8435504.0	144.2	270.61	60.00	84.0
BD001	187040.6	8435002.0	167.9	270.00	60.00	269.6
BD002	187011.6	8435005.0	176.8	270.00	60.00	270.0
BD003	186986.8	8435007.0	184.5	270.00	60.00	270.0
BD004	186950.7	8435009.0	193.8	270.00	60.00	120.0
BD005	186951.2	8435009.0	194.0	277.00	46.00	121.0
BD006	187074.2	8435002.0	159.5	278.50	60.50	380.7
BD007	187115.4	8435202.0	170.2	272.50	61.00	381.0
BD008	187084.1	8435201.0	180.4	272.00	61.00	320.8
BD009	187052.0	8435205.0	191.4	270.00	60.50	120.0
BD010	187024.7	8435207.0	196.2	270.00	60.00	120.0
BD011	187000.2	8435206.0	197.7	274.00	59.00	120.0
BD012	187159.7	8435298.0	183.5	272.00	61.00	305.5
BD013	187132.0	8435300.0	174.8	269.00	60.50	120.0
BD014	187103.2	8435299.0	186.5	270.00	60.00	120.0
BD015	187074.4	8435298.0	194.7	274.00	61.00	119.0
BD016	187078.9	8435101.0	160.8	270.00	60.00	270.8
BD017	187050.0	8435101.0	169.2	270.00	60.00	120.0
BD018	187017.1	8435098.0	182.7	270.00	60.00	120.0
BD019	186966.8	8435102.0	197.7	270.00	60.00	120.0
BD020	186924.9	8435097.0	191.1	269.00	60.00	120.0
BD021	187038.4	8434901.0	158.7	269.00	60.00	270.0
BD022	187008.3	8434901.0	160.4	274.00	60.00	120.0
BD023	186980.2	8434902.0	167.8	270.00	55.00	120.0
BD024	186950.5	8434904.0	181.3	273.00	54.00	120.0
BD025	186890.2	8434900.0	194.1	270.00	60.00	120.0
BD026	187106.9	8435101.0	161.9	273.00	61.00	120.0
BD027	187174.3	8435201.0	169.5	271.00	61.50	544.0
BD028	187004.4	8434803.0	165.6	270.00	45.50	140.0
BD029	186941.9	8434800.0	185.1	270.00	45.00	120.0
BD030	187050.0	8435102.0	169.3	270.00	60.00	100.0
BD031	186926.9	8435157.0	172.9	270.00	55.00	50.0
BD032	186961.0	8435159.0	188.1	270.00	55.00	69.6
BD033	186987.3	8435158.0	191.1	270.00	55.00	100.0
BD034	187016.5	8435158.0	188.1	270.00	55.00	120.0
BD035	187042.7	8435158.0	181.0	270.00	55.00	130.0
BD036	187072.7	8435158.0	175.2	270.00	55.00	153.0
BD037	187107.8	8435156.0	167.8	273.00	55.00	150.0
BD038	187136.5	8435156.0	161.7	272.00	56.50	151.0
BD039	186953.9	8435060.0	199.0	270.00	55.00	90.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BD040	186984.3	8435049.0	191.9	270.00	55.00	110.0
BD041	187020.3	8435052.0	179.6	270.00	55.00	120.0
BD042	187047.8	8435054.0	169.3	270.00	55.00	131.0
BD043	187082.2	8435053.0	160.8	270.00	55.00	150.0
BD044	187106.6	8435053.0	156.0	268.50	55.00	150.3
BD045	187134.5	8435059.0	156.4	269.00	55.00	150.0
BD046	187166.5	8435052.0	157.7	270.00	54.50	150.0
BD047	186915.0	8434960.0	195.6	270.00	55.00	71.0
BD048	186950.6	8434959.0	189.8	274.50	55.00	110.0
BD049	186978.8	8434959.0	180.7	274.00	55.50	120.0
BD050	187006.2	8434959.0	170.6	275.00	54.50	126.0
BD051	187043.2	8434963.0	160.6	275.00	56.50	137.1
BD052	187064.5	8434960.0	157.9	269.00	55.00	139.2
BD053	187103.3	8434960.0	153.5	272.50	56.00	141.0
BD054	186891.2	8434869.0	198.8	270.00	55.00	50.0
BD055	186921.2	8434868.0	192.7	272.00	56.50	100.0
BD056	186953.7	8434865.0	180.3	270.00	55.00	120.0
BD057	186982.2	8434865.0	172.1	269.00	55.50	139.5
BD058	187013.1	8434865.0	163.9	270.00	56.00	140.0
BD059	187041.4	8434864.0	158.2	270.00	57.00	257.6
BD060	186917.2	8434798.0	194.9	269.50	47.00	69.6
BD061	186966.6	8434801.0	176.4	270.00	45.00	110.0
BD062	187019.6	8435260.0	207.3	272.00	55.00	80.0
BD063	187041.9	8435261.0	201.6	270.00	55.00	120.0
BD064	187072.8	8435258.0	188.5	268.00	54.00	119.0
BD065	187105.6	8435256.0	174.2	270.00	55.00	130.0
BD066	187131.0	8435260.0	168.8	270.00	55.00	140.0
BD067	186975.4	8435208.0	193.1	270.00	60.00	70.0
BD068	186983.6	8435095.0	194.1	270.00	60.00	110.0
BD069	187068.5	8434902.0	155.7	272.00	61.50	130.0
BD070	186989.9	8435097.0	193.3	4.00	90.00	120.7
BD071	186992.0	8435098.0	193.1	52.00	50.00	120.0
BD072	187074.2	8435002.0	159.5	269.50	62.50	120.0
BD073	187115.4	8435202.0	170.2	270.00	60.00	120.0
BD074	186955.0	8435000.0	188.1	270.00	60.00	120.0
BD075	187019.5	8434901.0	163.2	269.00	62.00	120.0
BD076	187138.5	8435098.0	159.8	270.00	61.50	503.0
BD077	187108.5	8434903.0	152.7	272.00	61.00	467.3
BD078	187178.6	8435002.0	156.7	270.00	60.00	393.0
BD079	187158.3	8434902.0	150.6	272.00	61.00	375.4
BD080	187118.3	8435002.0	153.7	271.00	60.00	308.5
BD081	187238.3	8435002.0	152.7	270.00	60.00	449.4
BD082	187190.0	8435098.0	160.5	269.00	61.50	299.7
BD083	187234.4	8435200.0	180.8	274.00	60.00	319.6
BD084	187303.7	8435900.0	155.2	260.00	60.00	359.4
BD085	187089.9	8434701.0	153.0	267.50	62.00	400.0
BD086	187099.6	8434801.0	153.9	273.00	62.00	366.7

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BD087	187169.0	8434861.0	149.5	268.00	60.50	400.0
BD088	187184.7	8435380.0	207.8	273.00	61.00	330.0
BD089	187159.9	8435499.0	221.1	270.00	60.50	303.0
BD090	187182.2	8435603.0	213.6	268.00	62.00	301.0
BD091	187371.8	8436053.0	143.2	271.00	60.00	348.6
BD092	187064.9	8435001.0	161.9	269.50	62.00	61.1
BD093	187055.8	8435037.0	168.9	270.00	47.00	187.0
BD094	187040.8	8435037.0	173.7	269.00	46.00	140.0
BD095	187028.1	8435037.0	174.3	269.00	45.00	120.0
BD096	187084.1	8435136.0	168.0	269.00	51.00	194.0
BD097	187055.2	8435137.0	171.4	272.00	50.00	171.6
BD098	187026.2	8435137.0	182.5	266.00	51.00	156.5
BD099	186993.8	8435136.0	191.7	269.00	51.00	85.0
BD100	187064.9	8435002.0	161.8	273.00	61.00	300.1
BD101	187008.0	8434900.0	162.4	270.00	60.00	120.0
BD102	187043.1	8434962.0	159.6	270.00	60.00	116.8
BD103	187074.2	8435001.0	159.4	270.00	60.00	180.0
BD104	187021.6	8435051.0	176.9	270.00	60.00	81.3
BD105	187050.0	8435101.0	168.3	270.00	60.00	121.0
BD106	187016.4	8435157.0	187.5	270.00	60.00	101.3
BD107	187219.3	8435050.0	158.9	267.50	61.00	500.0
BD108	187219.5	8434846.0	148.9	267.50	60.00	500.0
BD109	187240.3	8434951.0	150.2	270.50	60.00	499.9
BD110	187120.1	8434754.0	154.4	267.50	60.00	392.5
BD111	187248.2	8435160.0	176.9	264.50	62.00	500.0
BD112	187304.7	8435300.0	172.2	267.50	55.50	478.8
BD113	187249.3	8435271.0	186.9	269.50	60.00	501.8
BD114	187225.9	8435325.0	198.6	267.00	60.00	350.1
BD115	187311.6	8435497.0	174.3	269.50	61.00	520.6
BD116	187306.1	8435402.0	159.6	270.50	60.00	501.3
BD117	187044.3	8434705.0	156.5	270.50	60.00	249.9
BD118	187044.3	8434801.0	159.2	269.50	57.00	260.2
BD119	187232.2	8434751.0	147.8	269.50	58.00	115.0
BD120	187153.0	8434812.0	150.8	269.50	59.50	113.0
BD121	187200.2	8434852.0	148.4	269.50	63.00	120.0
BD122	187008.2	8434745.0	164.9	269.50	58.00	218.8
BD123	187003.9	8434902.0	162.5	269.50	57.00	219.6
BD124	187094.5	8435045.0	159.4	269.50	57.50	314.8
BD125	187118.6	8435151.0	164.8	269.50	57.00	296.7
BD126	187097.7	8435301.0	188.9	269.50	63.50	312.8
BD127	187134.4	8435251.0	169.5	269.50	57.00	350.5
BD128	187195.2	8435402.0	198.8	269.50	64.00	401.0
BD129	187069.9	8434751.0	154.7	270.00	57.00	278.0
BD130	187098.5	8434851.0	153.9	270.00	55.50	302.3
BD131	187120.1	8434951.0	152.5	270.00	58.00	362.0
BD132	187192.3	8435151.0	166.2	269.50	59.00	380.2
BD133	187110.7	8435252.0	172.7	270.00	58.50	271.4

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BD134	187221.7	8435344.0	195.1	270.00	60.00	410.0
BD135	187118.4	8434951.0	152.6	270.00	58.00	78.0
BD136	187167.7	8435035.0	157.5	271.00	59.00	407.2
BD137	187065.5	8434652.0	151.9	270.00	58.50	300.0
BD138	187286.5	8435001.0	151.7	267.50	60.00	400.0
BD139	187369.8	8435101.0	157.8	270.00	61.30	444.0
BD140	187287.8	8435091.0	161.9	269.50	60.00	239.8
BD141	187334.4	8435154.0	163.2	270.00	60.00	333.0
BD142	187321.6	8435200.0	159.4	270.00	60.00	270.0
BD143	187361.8	8435303.0	159.6	270.00	59.00	248.8
BD144	187332.0	8435300.0	166.3	270.00	60.00	309.8
BD145	187381.8	8435404.0	151.7	270.00	61.00	269.3
BD146	187377.1	8435609.0	152.4	270.00	60.00	180.0
BD147	187376.6	8435702.0	159.5	270.00	60.00	227.8
BD148	186999.2	8434642.0	163.0	272.50	61.00	206.8
BD149	186983.5	8434601.0	162.9	270.00	60.00	198.4
BD150	186984.2	8434502.0	153.9	269.50	60.00	129.0
BD151	186998.8	8434553.0	155.2	269.50	61.00	170.0
BD152	187296.0	8435049.0	153.5	280.00	59.00	300.0
BD153	187371.1	8435204.0	150.0	268.50	60.00	219.5
BD154	187320.2	8435355.0	158.8	273.00	59.00	255.6
BD155	187015.4	8434500.0	150.7	274.50	60.00	159.5
BD156	187039.5	8435352.0	183.3	272.50	61.00	138.5
BD157	187088.4	8435351.0	178.8	277.50	60.00	206.0
BD158	187139.1	8435351.0	178.8	272.50	60.00	280.2
BD159	187089.8	8435456.0	187.1	268.50	61.00	195.5
BD160	187139.3	8435452.0	187.2	270.00	60.00	171.0
BD161	187189.5	8435451.0	187.0	269.50	61.00	193.8
BD162	187119.1	8435552.0	186.6	270.00	60.00	147.4
BD163	187167.2	8435553.0	186.7	272.50	61.00	219.0
BD164	187203.7	8435551.0	186.7	272.50	60.00	150.3
BD165	187253.4	8435552.0	183.5	268.50	60.00	303.0
BD166	187168.2	8435651.0	187.0	272.50	60.00	144.4
BD167	187218.4	8435652.0	186.9	269.50	60.00	169.4
BD168	186909.1	8435405.0	177.4	82.00	50.00	340.0
BD169	187018.0	8435114.0	155.0	266.50	50.50	145.0
BD170	186719.3	8434799.0	182.9	84.50	48.00	450.0
BD171	187039.8	8434951.0	151.5	270.00	60.00	21.9
BD172	187040.7	8434951.0	151.5	270.00	60.00	168.2
BD173	187259.3	8435202.0	171.6	270.00	60.50	370.0
BD174	187169.4	8435251.0	167.2	275.00	61.00	190.0
BD175	187094.0	8435202.0	163.0	265.50	61.50	182.0
BD176	187061.5	8434902.0	154.8	270.50	60.00	171.5
BD177	187174.6	8435351.0	167.3	269.00	60.00	188.0
BD178	187180.1	8434951.0	152.1	277.00	60.00	212.0
BD179	187249.0	8435052.0	158.0	273.50	59.00	208.0
BD180	186974.1	8435002.0	130.7	272.00	58.00	151.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BD181	187109.2	8435102.0	159.7	270.00	60.00	181.0
BD182	187242.2	8435502.0	178.9	270.00	60.00	405.0
BD183	187291.4	8435601.0	179.2	270.00	60.00	493.3
BD184	187197.1	8435713.0	186.8	278.50	61.00	263.8
BD185	187298.8	8435702.0	178.8	267.50	60.50	398.8
BD186	187326.8	8435800.0	167.8	283.50	58.00	401.6
BD187	187423.2	8435805.0	149.8	269.50	61.00	497.9
BD188	187242.5	8435900.0	151.5	269.50	61.00	353.9
BD189	187364.1	8435903.0	159.0	270.00	45.00	549.7
BD190	187207.3	8435502.0	175.2	270.00	60.00	212.7
BD191	187258.0	8435502.0	175.1	270.00	60.00	206.5
BD192	187080.3	8435508.0	163.3	270.00	60.00	100.0
BD193	187103.9	8435506.0	162.9	270.00	60.00	150.1
BD194	187149.4	8435502.0	162.6	270.00	65.00	209.9
BD195	187289.5	8435507.0	179.2	270.00	60.00	199.9
BD196	187333.1	8435501.0	167.8	270.00	60.00	199.9
BD197	187364.5	8435500.0	156.0	270.00	60.00	201.6
BD198	187229.1	8435402.0	175.3	270.00	60.00	302.8
BD199	187205.8	8435452.0	175.6	270.00	60.00	251.7
BD200	187094.1	8435451.0	147.2	270.00	60.00	150.5
BD201	187128.3	8435451.0	147.6	270.00	65.00	200.0
BD202	187407.6	8435462.0	167.2	270.00	60.00	200.0
BD203	187254.5	8435452.0	175.6	270.00	60.00	205.9
BD204	187244.2	8435602.0	175.0	270.00	65.00	214.8
BD205	187219.0	8435602.0	174.8	270.00	65.00	199.6
BD206	187116.5	8435610.0	178.7	270.00	60.00	148.0
BD207	187217.3	8435546.0	170.7	270.00	60.00	212.9
BD208	187188.5	8435602.0	175.1	267.50	60.00	204.5
BD209	187243.3	8434899.0	147.0	269.00	58.00	148.0
BD210	187194.1	8435002.0	140.5	266.00	64.00	1.0
BD211	187209.1	8435002.0	140.8	257.00	63.00	0.0
BD212	187200.3	8435001.0	140.6	258.00	65.00	465.7
BD213	187248.0	8434901.0	146.8	259.50	58.00	360.7
BD214	187331.3	8435266.0	162.8	265.50	64.00	300.5
BD215	187240.5	8435350.0	162.9	270.50	69.00	261.2
BD216	187039.2	8435251.0	123.3	269.50	57.00	131.0
BD217	187208.8	8435065.0	140.8	279.50	67.00	326.8
BD218	187133.6	8434801.0	135.4	263.50	68.00	239.4
BD219	186994.0	8434950.0	123.2	261.50	70.00	210.5
BD220	187166.0	8435259.0	122.6	270.50	66.00	299.7
BD221	187109.8	8434852.0	134.8	259.50	69.00	260.6
BD222	187264.7	8434851.0	146.4	267.50	70.00	120.0
BD223	187364.7	8435065.0	145.8	256.50	60.00	148.0
BD224	187131.7	8434952.0	134.7	261.50	69.00	334.7
BD225	187295.5	8435452.0	161.9	267.50	63.00	140.0
BD226	187389.0	8435356.0	149.4	263.50	61.00	200.0
BP001	187074.4	8435004.0	159.4	270.00	62.00	78.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP002	187040.2	8435004.0	167.9	270.00	60.00	81.0
BP003	187011.1	8435007.0	176.8	270.00	60.00	126.0
BP004	186986.2	8435007.0	184.5	270.00	60.00	76.0
BP005	186949.9	8435011.0	194.0	269.50	63.00	81.0
BP006	187114.8	8435203.0	170.2	270.00	60.00	81.0
BP007	187082.4	8435202.0	180.4	269.00	60.00	81.0
BP008	187052.5	8435206.0	191.4	268.00	62.00	82.0
BP009	187023.9	8435208.0	196.2	270.50	61.50	81.0
BP010	186999.3	8435206.0	197.6	270.00	60.00	81.0
BP011	186956.7	8435097.0	199.4	270.00	60.00	81.0
BP012	187211.5	8435604.0	210.0	269.50	60.50	81.0
BP013	187182.2	8435603.0	213.6	268.00	62.00	81.0
BP014	187161.6	8435605.0	215.6	269.50	63.50	141.0
BP015	186984.4	8435097.0	193.5	270.00	60.00	81.0
BP016	186922.9	8434906.0	191.5	294.00	61.50	81.0
BP017	187044.1	8435296.0	204.8	271.00	60.00	81.0
BP018	187000.3	8434801.0	166.1	269.00	61.50	124.0
BP019	186939.0	8434800.0	185.6	270.00	60.00	120.0
BP020	186879.2	8434799.0	202.7	272.00	62.00	120.0
BP021	186899.6	8434699.0	189.5	270.00	60.00	120.0
BP022	186955.0	8434699.0	188.1	273.00	61.50	120.0
BP023	187019.5	8434700.0	163.2	272.50	61.50	120.0
BP024	186872.0	8434601.0	169.0	271.00	60.00	120.0
BP025	186935.3	8434602.0	170.7	275.00	61.50	120.0
BP026	186993.7	8434601.0	163.9	278.00	61.50	120.0
BP027	187174.3	8435201.0	169.5	271.00	61.50	120.0
BP028	187234.4	8435200.0	180.8	274.00	60.00	120.0
BP029	187290.9	8435199.0	171.7	274.00	61.00	120.0
BP030	187354.1	8435199.0	151.3	270.00	60.00	120.0
BP031	187415.8	8435201.0	147.5	270.00	60.00	120.0
BP032	187368.9	8435404.0	153.9	268.00	61.50	120.0
BP033	187311.0	8435400.0	158.7	270.00	60.00	120.0
BP034	187417.9	8435002.0	147.1	270.00	60.00	100.0
BP035	187479.2	8435004.0	144.2	270.00	60.00	120.0
BP036	187253.7	8435398.0	170.5	270.00	60.00	120.0
BP038	187129.4	8435397.0	213.7	274.00	61.00	120.0
BP039	187118.3	8435002.0	153.7	271.00	60.00	120.0
BP040	187068.8	8435396.0	207.7	268.00	59.00	120.0
BP041	187178.6	8435002.0	156.7	270.00	60.00	120.0
BP042	187184.7	8435380.0	207.8	273.00	61.00	120.0
BP043	187238.3	8435002.0	152.7	270.00	60.00	120.0
BP044	187372.4	8435600.0	153.9	271.50	63.50	120.0
BP045	187298.1	8435002.0	150.0	270.00	60.00	120.0
BP046	187311.5	8435603.0	174.2	270.00	60.00	120.0
BP047	187357.6	8435002.0	149.5	270.00	60.00	105.0
BP048	187322.7	8435802.0	167.8	269.00	61.50	120.0
BP049	187434.6	8435400.0	145.7	269.00	60.00	120.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP050	187257.9	8435799.0	150.6	270.00	60.00	120.0
BP051	187387.4	8435802.0	155.1	270.00	60.00	120.0
BP052	187203.0	8435801.0	175.7	271.00	62.00	120.0
BP053	187379.2	8436001.0	136.7	266.00	58.50	120.0
BP054	187151.1	8435799.0	176.4	270.00	60.00	120.0
BP055	187320.1	8435999.0	136.8	271.50	59.00	120.0
BP056	187335.3	8435700.0	169.8	270.00	60.00	120.0
BP057	187257.1	8435994.0	137.0	270.00	60.00	120.0
BP058	187283.2	8435701.0	185.1	270.00	60.00	120.0
BP059	187194.3	8435999.0	144.1	272.00	60.00	120.0
BP060	187222.0	8435696.0	188.0	269.00	60.00	120.0
BP061	187364.1	8435903.0	159.0	270.00	60.00	120.0
BP062	187159.5	8435693.0	208.7	268.00	60.00	120.0
BP063	187303.7	8435900.0	155.2	260.00	60.00	117.0
BP064	187258.3	8435501.0	183.6	270.00	60.00	120.0
BP065	187184.3	8435896.0	154.5	266.50	60.50	120.0
BP066	187195.6	8435507.0	210.0	274.00	62.00	120.0
BP067	187240.8	8435807.0	159.6	269.50	61.00	120.0
BP068	187130.3	8435500.0	223.5	278.50	61.00	118.0
BP069	187215.9	8435603.0	209.3	270.00	65.50	120.0
BP070	187308.1	8435001.0	149.6	4.00	90.00	60.0
BP071	187258.5	8435301.0	186.3	275.00	60.00	120.0
BP072	187307.0	8435300.0	171.8	271.00	51.50	120.0
BP073	187365.0	8435302.0	159.3	274.50	53.00	120.0
BP074	187414.5	8435301.0	151.9	270.00	60.00	120.0
BP075	187340.0	8435400.0	152.7	270.00	60.00	100.0
BP076	187407.4	8435398.0	146.7	271.50	61.00	130.0
BP077	187312.8	8435500.0	174.2	269.50	61.50	120.0
BP078	187363.2	8435507.0	156.7	275.00	61.00	120.0
BP079	187409.2	8435503.0	150.2	272.00	61.00	120.0
BP080	187461.4	8435501.0	148.2	271.00	60.00	120.0
BP081	187189.8	8435695.0	199.4	269.50	59.00	100.0
BP082	187169.5	8435746.0	195.8	248.00	60.00	60.0
BP083	186925.7	8434999.0	196.5	272.50	59.00	75.0
BP084	186922.7	8434698.0	192.0	270.00	60.50	90.0
BP085	187178.2	8435801.0	179.8	270.00	60.00	60.0
BP086	187228.3	8435801.0	164.0	265.00	61.00	120.0
BP087	187286.5	8435799.0	160.4	270.00	60.00	140.0
BP088	187199.2	8435844.0	171.9	270.00	60.00	70.0
BP089	187230.9	8435847.0	160.7	268.00	61.50	110.0
BP090	187263.4	8435848.0	149.2	270.00	62.00	120.0
BP091	187169.2	8435156.0	163.3	253.00	61.00	120.0
BP092	187197.8	8435054.0	159.9	269.00	61.00	120.0
BP093	187134.5	8434960.0	152.4	275.00	61.00	120.0
BP094	187074.3	8434864.0	155.1	270.00	60.00	120.0
BP095	186988.9	8434699.0	171.6	269.50	60.00	120.0
BP096	187169.5	8435256.0	178.7	273.00	61.00	120.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP097	186948.8	8435208.0	185.7	271.50	63.00	50.0
BP098	187138.5	8435098.0	159.8	270.00	61.50	130.0
BP099	186989.9	8435262.0	202.3	273.00	65.50	100.0
BP100	187108.5	8434903.0	152.7	272.00	61.00	120.0
BP101	187026.9	8434403.0	146.7	269.00	61.50	50.0
BP102	187001.5	8434403.0	147.7	270.00	60.50	50.0
BP103	186982.6	8434404.0	147.9	274.00	60.00	50.0
BP104	186926.0	8434403.0	151.7	271.50	60.50	50.0
BP105	187335.1	8435399.0	153.0	94.00	60.00	50.0
BP111	186989.6	8435096.0	193.6	4.00	90.00	103.0
BP112	186986.4	8435081.0	193.2	4.00	90.00	101.0
BP113	186985.3	8435066.0	193.0	4.00	90.00	103.0
BP114	186978.8	8435036.0	192.5	4.00	90.00	102.0
BP115	186959.2	8434997.0	191.7	4.00	90.00	94.0
BP116	186946.0	8435001.0	194.6	4.00	90.00	83.0
BP117	186961.2	8435038.0	196.2	4.00	90.00	100.0
BP118	186968.8	8435052.0	196.9	4.00	90.00	104.0
BP119	186974.9	8435067.0	197.0	4.00	90.00	103.0
BP120	186978.2	8435100.0	195.6	4.00	90.00	87.0
BP121	186947.0	8435069.0	198.1	4.00	90.00	112.0
BP122	186945.3	8435054.0	196.8	94.00	90.00	111.0
BP123	186944.0	8435039.0	195.6	94.00	90.00	110.0
BP124	186906.3	8435002.0	191.0	4.00	90.00	100.0
BP125	186923.8	8435040.0	190.8	4.00	90.00	102.0
BP126	186927.9	8435054.0	191.2	55.00	89.00	106.0
BP127	186929.4	8435069.0	191.8	55.00	89.50	104.0
BP128	186936.9	8435102.0	194.0	4.00	90.00	85.0
BP129	186915.8	8435070.0	186.2	4.00	90.00	85.0
BP130	186916.1	8435055.0	185.4	4.00	90.00	50.0
BP131	186909.9	8435041.0	186.0	4.00	90.00	79.0
BP132	186892.7	8435004.0	186.8	4.00	90.00	80.0
BP133	186940.0	8435172.0	176.0	4.00	90.00	75.0
BP134	186998.9	8435067.0	187.9	4.00	90.00	102.0
BP135	186997.3	8435047.0	187.8	4.00	90.00	102.0
BP136	186994.2	8435037.0	188.0	4.00	90.00	102.0
BP137	187007.7	8435038.0	182.5	4.00	90.00	96.0
BP138	187014.0	8435066.0	182.1	4.00	90.00	96.0
BP139	187022.2	8435101.0	182.4	4.00	90.00	96.0
BP140	187034.5	8435101.0	177.1	4.00	90.00	89.0
BP141	187026.7	8435064.0	178.3	4.00	90.00	92.0
BP142	187019.5	8435039.0	178.6	4.00	90.00	93.0
BP143	187022.6	8435011.0	174.3	4.00	90.00	88.0
BP144	187031.3	8435038.0	175.1	4.00	90.00	89.0
BP145	187033.5	8435049.0	175.2	4.00	90.00	89.0
BP146	187046.6	8435064.0	170.3	4.00	90.00	84.0
BP147	187053.3	8435102.0	169.3	4.00	90.00	83.0
BP148	187053.1	8435082.0	168.0	4.00	90.00	82.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP149	187054.1	8435069.0	168.8	4.00	90.00	83.0
BP150	187044.2	8435048.0	170.9	4.00	90.00	85.0
BP151	187043.7	8435038.0	170.2	4.00	90.00	84.0
BP152	187040.9	8435024.0	170.2	4.00	90.00	84.0
BP153	187079.5	8435083.0	161.0	4.00	90.00	76.0
BP154	187075.7	8435102.0	161.6	4.00	90.00	76.0
BP155	187071.1	8435137.0	165.3	4.00	90.00	79.0
BP156	187057.8	8435137.0	171.4	4.00	90.00	84.0
BP157	187038.7	8435136.0	178.1	4.00	90.00	92.0
BP158	187026.1	8435135.0	182.5	4.00	90.00	94.0
BP159	187010.2	8435134.0	186.8	4.00	90.00	101.0
BP160	186994.5	8435134.0	191.7	4.00	90.00	106.0
BP161	186979.2	8435134.0	193.3	4.00	90.00	70.0
BP162	186965.0	8435135.0	189.0	4.00	90.00	103.0
BP163	186949.1	8435133.0	185.9	4.00	90.00	100.0
BP164	186931.1	8435133.0	182.2	4.00	90.00	96.0
BP165	186914.9	8435133.0	179.1	4.00	90.00	93.0
BP166	187045.5	8435173.0	185.3	4.00	90.00	84.0
BP167	187029.7	8435173.0	185.9	4.00	90.00	91.0
BP168	187018.1	8435173.0	186.8	4.00	90.00	95.0
BP169	187000.5	8435174.0	192.8	4.00	90.00	97.0
BP170	186986.0	8435173.0	193.5	4.00	90.00	93.0
BP171	186970.0	8435175.0	190.0	4.00	90.00	85.0
BP172	186956.4	8435174.0	186.2	4.00	90.00	83.0
BP175	187158.3	8434902.0	150.6	272.00	61.00	120.0
BP176	187190.0	8435098.0	160.5	269.00	61.50	91.0
BP177	187161.0	8435397.0	216.0	268.00	60.00	60.0
BP178	187098.8	8435396.0	213.4	268.00	60.00	60.0
BP179	187038.4	8435396.0	201.7	269.00	59.50	60.0
BP180	187287.0	8435501.0	180.0	270.00	60.50	60.0
BP181	187225.9	8435507.0	194.9	271.00	60.00	60.0
BP182	187159.9	8435499.0	221.1	270.00	60.50	60.0
BP183	187101.1	8435499.0	222.4	269.50	59.50	60.0
BP184	187276.3	8435603.0	191.0	271.00	60.00	60.0
BP185	187244.2	8435602.0	202.8	270.00	60.00	60.0
BP186	187132.5	8435605.0	213.1	269.00	60.00	60.0
BP187	187101.1	8435606.0	200.6	275.00	60.00	60.0
BP188	186964.0	8435022.0	193.8	290.50	49.50	30.0
BP189	186961.4	8435023.0	194.0	290.00	50.00	30.0
BP190	186958.6	8435024.0	194.2	290.00	50.00	30.0
BP191	186955.7	8435024.0	194.3	290.00	50.00	30.0
BP192	186952.6	8435025.0	194.5	290.50	50.00	30.0
BP193	186969.2	8435035.0	194.6	291.00	50.00	30.0
BP194	186966.7	8435035.0	194.8	290.00	50.00	30.0
BP195	186963.9	8435037.0	194.9	289.50	50.00	30.0
BP196	186960.4	8435038.0	195.1	290.00	50.00	30.0
BP197	186957.6	8435039.0	195.3	291.00	49.00	30.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP198	186974.6	8435050.0	195.8	286.00	49.00	30.0
BP199	186972.0	8435050.0	196.1	287.00	50.00	30.0
BP200	186969.3	8435051.0	196.4	290.00	50.00	30.0
BP201	186966.2	8435052.0	196.6	290.00	50.00	30.0
BP202	186963.3	8435053.0	196.6	290.00	50.00	30.0
BP203	186979.8	8435064.0	195.8	290.00	50.00	30.0
BP204	186977.1	8435065.0	196.1	290.00	50.00	30.0
BP205	186974.1	8435066.0	196.5	290.00	50.00	30.0
BP206	186971.3	8435067.0	196.9	290.00	50.00	30.0
BP207	186968.7	8435068.0	197.1	291.50	51.00	30.0
BP208	187089.9	8434701.0	153.0	267.50	62.00	90.0
BP209	187110.1	8434862.0	153.2	276.50	60.50	150.0
BP210	187169.0	8434861.0	149.5	268.00	60.50	150.0
BP211	187049.1	8434801.0	158.4	269.50	62.00	100.0
BP212	187099.6	8434801.0	153.9	273.00	62.00	100.0
BP213	187148.9	8434801.0	150.2	271.00	61.00	100.0
BP214	187198.1	8434801.0	147.7	271.00	60.00	100.0
BP215	187249.0	8434801.0	146.3	271.00	61.00	100.0
BP216	187299.5	8434801.0	145.0	275.00	61.50	100.0
BP217	187348.1	8434801.0	144.0	270.00	60.50	100.0
BP218	187398.9	8434801.0	141.6	270.00	61.50	100.0
BP219	187449.8	8434801.0	140.3	271.00	59.50	100.0
BP220	187498.5	8434801.0	140.3	271.00	60.50	100.0
BP221	187118.5	8434861.0	152.4	288.50	50.00	20.0
BP222	187075.2	8435002.0	159.3	267.50	61.00	90.0
BP223	187041.1	8435004.0	167.8	271.00	60.00	100.0
BP224	187022.4	8435011.0	174.4	268.00	60.00	110.0
BP225	187011.0	8435006.0	176.7	269.00	61.00	110.0
BP226	186986.7	8435007.0	182.9	263.50	60.50	120.0
BP227	186959.6	8434996.0	191.1	266.00	61.00	120.0
BP228	186951.6	8435009.0	193.7	266.00	60.00	120.0
BP229	187081.2	8435054.0	160.8	269.00	56.00	110.0
BP230	187048.6	8435053.0	169.8	265.00	56.00	110.0
BP231	187032.2	8435049.0	175.2	264.50	61.50	110.0
BP232	187018.7	8435051.0	178.7	269.50	55.50	120.0
BP233	186975.2	8435050.0	195.8	266.50	59.00	120.0
BP234	186951.6	8435061.0	197.1	267.50	54.50	90.0
BP235	186945.1	8435055.0	196.9	264.00	60.00	80.0
BP236	186926.6	8435054.0	191.3	262.00	60.50	50.0
BP237	187052.9	8435068.0	168.4	266.00	61.50	100.0
BP238	187046.6	8435064.0	169.3	262.00	60.50	105.0
BP239	187025.8	8435063.0	177.8	270.00	61.50	115.0
BP240	187012.7	8435067.0	182.1	268.00	61.00	120.0
BP241	186973.9	8435067.0	196.4	268.00	60.50	120.0
BP242	186959.4	8435067.0	197.2	268.00	60.50	120.0
BP243	186946.5	8435068.0	197.6	267.00	59.50	80.0
BP244	186928.4	8435070.0	192.4	262.00	60.00	50.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP245	187371.8	8436053.0	143.2	271.00	60.00	104.8
BP248	187519.6	8435601.0	141.8	268.00	60.00	50.0
BP249	187469.6	8435602.0	142.3	270.00	60.00	60.0
BP250	187420.2	8435611.0	145.5	271.00	60.00	54.0
BP254	187519.1	8435802.0	141.7	270.00	59.00	50.0
BP255	187473.0	8435801.0	144.7	266.00	60.00	45.0
BP256	187445.1	8435802.0	147.0	270.00	60.00	24.0
BP257	187500.1	8435801.0	141.9	270.00	61.00	50.0
BP261	187393.0	8435602.0	149.4	267.00	61.00	50.0
BP262	187444.4	8435602.0	144.3	273.00	61.00	50.0
BP263	187493.9	8435602.0	141.6	268.00	61.00	50.0
BP266	187445.7	8435801.0	146.6	273.00	60.00	50.0
BP267	187008.0	8434900.0	162.4	270.00	60.00	61.8
BP268	187043.1	8434962.0	159.6	270.00	60.00	85.0
BP269	187074.2	8435001.0	159.4	270.00	60.00	58.0
BP270	187021.6	8435051.0	176.9	270.00	60.00	70.3
BP271	187050.0	8435101.0	168.3	270.00	60.00	70.0
BP272	187016.4	8435157.0	187.5	270.00	60.00	50.0
BP273	186919.1	8435135.0	180.1	268.00	50.00	45.0
BP274	186942.7	8435136.0	185.1	265.00	52.00	45.0
BP275	186975.1	8435135.0	192.7	269.00	49.00	45.0
BP276	187108.1	8435136.0	167.0	268.00	51.00	40.0
BP277	187133.0	8435137.0	162.0	268.00	52.00	36.0
BP278	186873.1	8434751.0	203.0	266.00	49.00	50.0
BP279	186892.8	8434751.0	201.3	268.00	51.00	50.0
BP280	186920.8	8434750.0	192.6	267.00	51.00	50.0
BP281	186944.5	8434752.0	183.2	267.00	50.00	50.0
BP282	186971.0	8434752.0	174.6	270.00	49.00	45.0
BP283	186994.3	8434751.0	168.7	263.00	50.00	45.0
BP284	187018.0	8434751.0	161.8	269.00	50.00	45.0
BP285	187040.0	8434751.0	157.3	269.00	50.00	45.0
BP286	186895.2	8434832.0	201.8	266.00	51.00	65.0
BP287	186921.2	8434830.0	195.9	268.00	50.00	65.0
BP288	186944.3	8434832.0	187.8	269.00	50.00	55.0
BP289	186967.9	8434831.0	180.7	268.00	49.00	45.0
BP290	186993.0	8434831.0	171.9	268.00	49.00	45.0
BP291	187018.8	8434831.0	166.1	266.00	51.00	45.0
BP292	187044.3	8434831.0	160.1	268.00	52.00	35.0
BP293	187068.0	8434831.0	157.4	268.00	52.00	25.0
BP294	186892.4	8434884.0	196.8	270.00	50.00	65.0
BP295	186916.7	8434881.0	192.6	269.00	51.00	60.0
BP296	186944.6	8434882.0	181.1	267.00	49.00	55.0
BP297	186970.1	8434880.0	170.4	265.00	43.00	50.0
BP298	186994.5	8434882.0	162.5	268.00	51.00	40.0
BP299	187018.2	8434881.0	159.0	268.00	50.00	40.0
BP300	187043.3	8434881.0	156.8	272.00	51.00	35.0
BP301	187068.1	8434881.0	155.8	273.00	50.00	30.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP302	187093.4	8434882.0	153.7	270.00	51.00	25.0
BP303	186900.6	8434931.0	195.0	269.00	51.00	60.0
BP304	186917.7	8434931.0	193.6	268.00	51.00	60.0
BP305	186946.0	8434931.0	186.7	271.00	48.00	60.0
BP306	186969.2	8434931.0	180.0	265.00	50.00	50.0
BP307	186994.8	8434934.0	172.4	271.00	51.00	45.0
BP308	187020.5	8434931.0	166.1	271.00	51.00	35.0
BP309	187045.8	8434933.0	160.4	269.00	51.00	35.0
BP310	187067.6	8434931.0	156.4	270.00	51.00	35.0
BP311	187094.1	8434931.0	154.2	270.00	51.00	25.0
BP312	186917.8	8435032.0	190.6	269.00	50.00	60.0
BP313	186943.6	8435036.0	195.4	269.00	50.00	65.0
BP314	186968.1	8435036.0	194.6	272.00	50.00	65.0
BP315	186992.8	8435035.0	188.1	271.00	50.00	65.0
BP316	187019.4	8435036.0	176.5	266.00	51.00	60.0
BP317	187072.3	8435036.0	162.6	270.00	51.00	35.0
BP318	187103.4	8435036.0	156.1	272.00	51.00	25.0
BP319	186953.2	8435181.0	185.9	268.00	55.00	40.0
BP320	186968.5	8435182.0	189.8	276.00	55.00	45.0
BP321	186992.8	8435182.0	193.9	265.00	53.00	45.0
BP322	187024.9	8435174.0	186.1	271.00	49.00	40.0
BP323	187042.8	8435179.0	185.4	270.00	50.00	40.0
BP324	187067.2	8435182.0	181.2	271.00	49.00	35.0
BP325	187091.6	8435182.0	175.4	271.00	50.00	35.0
BP326	187114.9	8435184.0	168.8	271.00	50.00	25.0
BP327	187139.2	8435184.0	163.1	272.00	51.00	25.0
BP328	187169.9	8435186.0	167.7	275.00	51.00	30.0
BP329	186970.9	8435212.0	194.5	267.00	53.00	50.0
BP330	186994.7	8435228.0	199.9	267.00	51.00	60.0
BP331	187021.0	8435232.0	202.6	268.00	52.00	60.0
BP332	187043.0	8435233.0	198.8	279.00	51.00	60.0
BP333	187070.2	8435232.0	188.3	271.00	50.00	45.0
BP334	187096.2	8435232.0	178.5	269.00	50.00	35.0
BP335	187118.9	8435232.0	170.9	272.00	52.00	25.0
BP336	187137.0	8435233.0	166.2	270.00	52.00	25.0
BP337	187170.3	8435232.0	174.5	273.00	51.00	30.0
BP338	186990.6	8435271.0	202.5	273.00	58.00	60.0
BP339	187015.1	8435280.0	209.6	270.00	50.00	60.0
BP340	187040.7	8435291.0	205.4	274.00	48.00	60.0
BP341	187074.0	8435281.0	190.0	271.00	50.00	50.0
BP342	187119.1	8435281.0	173.6	292.00	51.00	30.0
BP343	187140.5	8435282.0	178.9	266.00	50.00	25.0
BP344	187168.0	8435282.0	183.3	267.00	51.00	35.0
BP345	186889.0	8435004.0	186.0	265.00	50.00	70.0
BP346	186850.0	8435001.0	175.6	268.00	51.00	70.0
BP347	186804.2	8435001.0	167.8	267.00	49.00	70.0
BP348	186856.7	8435118.0	168.2	248.00	57.00	68.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP349	186917.0	8434552.0	163.5	270.50	60.50	60.0
BP350	186898.7	8434553.0	163.8	268.50	60.00	84.0
BP351	186904.6	8434602.0	165.2	270.50	62.00	86.0
BP352	186864.6	8434648.0	177.1	265.50	61.00	60.0
BP353	186870.8	8434555.0	165.1	268.50	60.00	50.0
BP354	186873.8	8434504.0	161.2	269.50	61.00	56.0
BP355	186906.4	8434500.0	154.9	269.50	60.00	84.0
BP356	186954.2	8434500.0	157.9	267.50	60.00	66.0
BP357	186966.3	8434453.0	153.3	266.50	60.00	72.0
BP358	186886.5	8434447.0	158.8	269.50	61.00	58.0
BP359	186896.4	8434399.0	154.8	264.50	60.00	60.0
BP360	186950.9	8434399.0	148.2	268.50	60.00	78.0
BP361	186930.2	8434501.0	157.9	268.50	60.00	50.0
BP362	186949.8	8434654.0	181.5	269.50	60.00	90.0
BP363	186927.9	8434651.0	182.2	266.50	60.00	54.0
BP364	186974.1	8434653.0	175.0	267.50	60.00	50.0
BP365	186968.3	8434603.0	167.5	266.50	60.00	84.0
BP366	186950.8	8434553.0	155.9	268.50	60.00	60.0
BP367	186943.8	8434450.0	152.3	264.50	60.00	54.0
BP368	186911.0	8434454.0	152.9	268.50	61.00	84.0
BP369	186972.1	8434551.0	155.3	266.00	60.00	42.0
BP370	186968.0	8434551.0	155.4	269.50	61.00	90.0
BP371	187238.6	8434750.0	147.6	269.50	63.00	58.0
BP372	187259.1	8435051.0	156.6	269.50	60.00	100.0
BP373	187239.2	8435101.0	165.4	269.50	59.50	100.0
BP374	187287.8	8435091.0	161.9	269.50	60.00	102.0
BP375	187208.2	8435151.0	168.1	269.50	60.00	101.0
BP376	187291.6	8435152.0	175.9	269.50	59.00	114.0
BP377	187261.0	8435201.0	182.8	269.50	59.00	100.0
BP378	187319.6	8435202.0	159.4	269.50	60.00	100.0
BP379	187220.6	8435252.0	191.5	269.50	60.00	102.0
BP380	187293.6	8435256.0	169.8	269.50	60.00	113.0
BP381	187218.0	8435302.0	197.3	269.50	59.50	100.0
BP382	187283.2	8435301.0	177.7	269.50	60.00	100.0
BP383	187211.0	8435352.0	198.8	269.50	59.00	100.0
BP384	187269.1	8435352.0	177.1	269.50	59.00	100.0
BP385	187320.2	8435355.0	158.8	269.50	61.00	100.0
BP386	187280.7	8435402.0	162.2	269.50	59.00	100.0
BP387	187381.8	8435404.0	151.7	269.50	61.00	102.0
BP388	187247.2	8435451.0	182.6	269.50	60.00	100.0
BP389	187283.3	8435444.0	170.5	269.50	61.00	100.0
BP390	187345.3	8435456.0	162.9	269.50	59.00	100.0
BP391	187242.4	8435552.0	198.5	269.50	59.00	100.0
BP392	187283.5	8435554.0	190.8	269.50	58.50	100.0
BP393	187333.5	8435552.0	167.9	269.50	61.00	100.0
BP394	187345.6	8435600.0	163.9	269.50	59.00	100.0
BP395	187266.4	8435660.0	184.8	269.50	59.00	100.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP396	187316.6	8435655.0	165.0	269.50	60.00	100.0
BP397	187368.9	8435649.0	153.8	269.50	60.00	101.0
BP398	187376.6	8435702.0	159.5	269.50	60.00	100.0
BP399	187255.5	8435701.0	185.8	269.50	60.00	100.0
BP400	187305.8	8435704.0	180.5	269.50	60.50	101.0
BP401	187170.5	8435751.0	195.1	269.50	60.00	83.0
BP402	187231.7	8435755.0	165.7	269.50	53.00	130.0
BP403	187270.1	8435747.0	169.8	269.50	61.00	100.0
BP404	187190.5	8435805.0	178.7	269.50	60.00	100.0
BP405	187216.3	8435903.0	154.7	269.50	60.00	113.0
BP406	187242.5	8435900.0	151.5	269.50	61.00	100.0
BP407	187296.0	8435049.0	153.5	269.50	60.00	100.0
BP408	187169.6	8435104.0	158.6	269.50	60.00	97.0
BP409	187211.5	8435108.0	163.7	269.50	59.00	108.0
BP410	187265.9	8435101.0	164.6	269.50	60.00	102.0
BP411	187323.5	8435103.0	168.9	269.50	60.00	64.0
BP412	187328.2	8435152.0	165.4	269.50	59.20	120.0
BP413	187142.8	8435202.0	164.5	269.50	60.00	102.0
BP414	187203.3	8435206.0	175.4	269.50	60.00	108.0
BP415	187265.8	8435255.0	180.3	269.50	60.00	117.0
BP416	187204.3	8435303.0	198.5	269.50	59.00	100.0
BP417	187332.8	8435302.0	165.9	269.50	59.00	100.0
BP418	187391.9	8435302.0	156.4	269.50	60.00	102.0
BP419	187318.7	8435258.0	164.8	269.50	61.00	130.0
BP420	187195.2	8435352.0	198.8	269.50	59.00	100.0
BP421	187291.8	8435352.0	166.9	269.50	60.00	100.0
BP422	187350.6	8435352.0	156.8	269.50	60.00	78.0
BP423	187253.3	8435404.0	170.2	269.50	60.00	100.0
BP424	187318.2	8435449.0	156.8	269.50	60.50	100.0
BP425	187366.3	8435451.0	160.4	269.50	60.00	100.0
BP426	187334.9	8435505.0	166.6	269.50	60.00	100.0
BP427	187308.9	8435550.0	180.9	269.50	60.50	100.0
BP428	187397.1	8435554.0	156.1	269.50	59.00	100.0
BP429	187361.8	8435303.0	159.5	270.00	59.00	120.0
BP430	187388.6	8435511.0	153.0	269.50	60.50	100.0
BP431	187372.8	8435607.0	153.8	269.50	60.00	106.0
BP432	186976.2	8435602.0	189.5	269.50	60.50	100.0
BP433	187005.1	8435602.0	186.1	269.50	60.00	101.0
BP434	187069.6	8434652.0	151.7	270.00	60.00	99.0
BP435	187016.8	8435550.0	193.6	269.50	61.00	100.0
BP443	187069.3	8435552.0	186.6	269.50	60.00	80.0
BP445	187118.8	8435651.0	186.9	270.50	60.00	80.0
BP446	187133.8	8435702.0	186.7	269.50	61.00	70.0
BP447	187144.5	8435752.0	186.6	267.50	60.50	82.0
BP448	187186.0	8435751.0	186.6	263.50	60.00	130.0
BP454	187071.2	8436002.0	172.9	272.50	60.00	80.0
BP455	187118.0	8436002.0	160.3	267.50	60.00	80.0

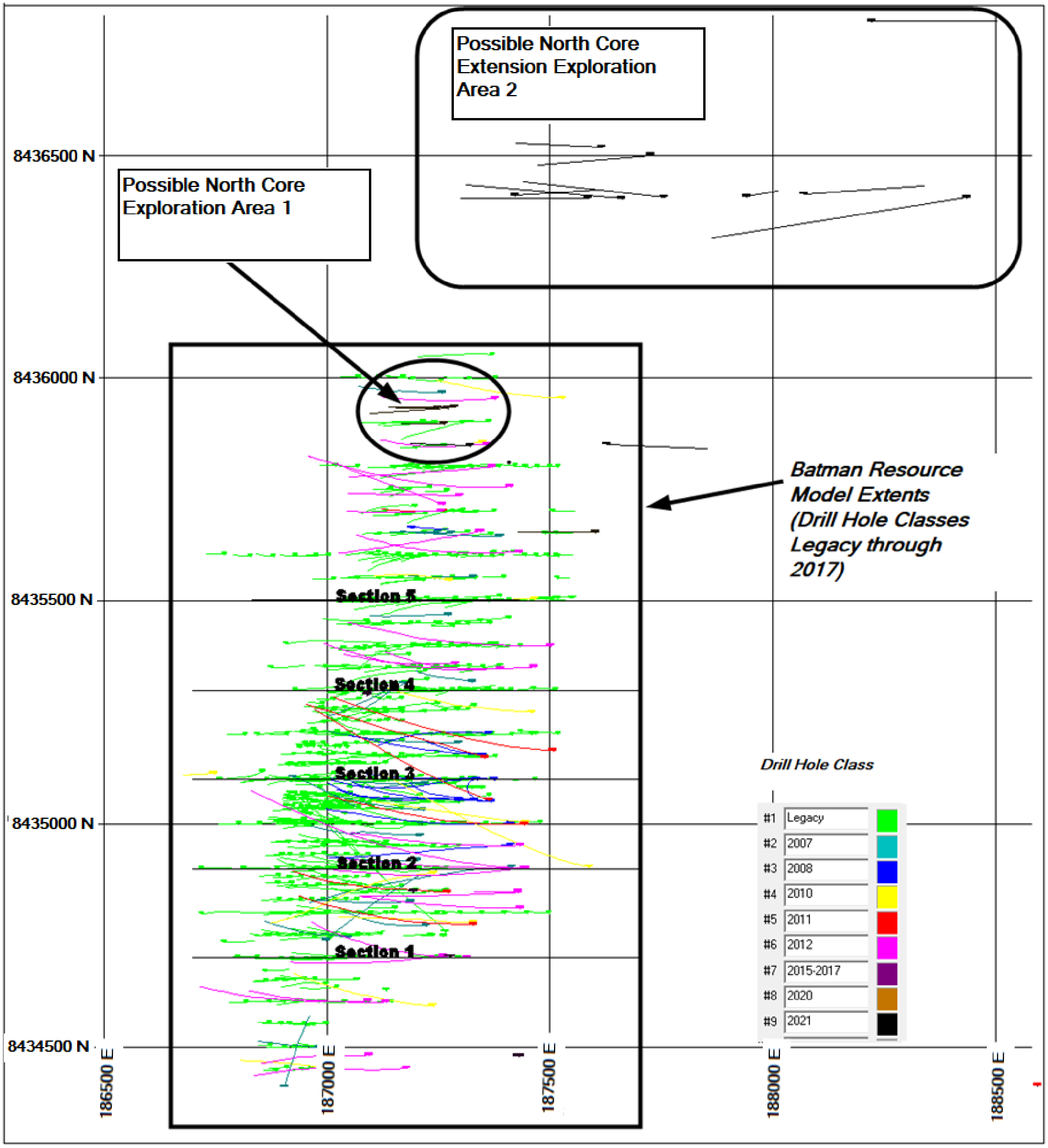
HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP456	187171.9	8436003.0	141.4	270.50	60.00	76.0
BP458	186817.2	8435201.0	158.1	268.50	60.00	80.0
BP459	187050.9	8435204.0	165.8	269.50	60.00	80.0
BP460	186911.7	8435201.0	163.0	270.50	60.00	80.0
BP461	186829.3	8435601.0	151.3	271.50	60.00	80.0
BP463	186930.4	8435602.0	177.2	268.50	61.00	80.0
BP464	186881.2	8435601.0	162.3	269.50	60.00	80.0
BP465	186770.4	8435605.0	146.4	269.50	60.00	80.0
BP466	187019.9	8435804.0	140.3	270.00	59.00	69.0
BP467	187069.9	8435802.0	150.1	268.50	60.00	80.0
BP468	187145.0	8434601.0	148.1	270.00	60.00	63.0
BP469	187094.2	8434601.0	148.9	270.50	60.00	80.0
BP470	187040.9	8434601.0	156.0	267.50	59.00	80.0
BP471	186820.5	8434601.0	158.8	267.50	60.00	80.0
BP472	186801.1	8434702.0	179.4	265.50	60.00	80.0
BP473	186845.1	8434703.0	178.9	269.50	60.00	80.0
BP477	186890.5	8434651.0	172.6	266.50	61.00	110.0
BP478	186901.9	8434651.0	171.9	266.00	75.00	120.0
BP479	186928.6	8434751.0	139.0	284.50	60.00	150.0
BP480	187023.8	8434635.0	159.3	262.00	60.00	160.0
BP481	186997.3	8434751.0	139.2	265.00	75.00	110.0
BP482	186998.5	8434751.0	139.1	89.00	74.50	60.0
BP483	187269.7	8434951.0	149.3	265.00	59.50	150.0
BP484	187296.7	8434951.0	149.6	266.50	60.00	150.0
BP485	186959.0	8434751.0	139.0	267.50	60.00	120.0
BP486	187209.2	8435001.0	155.0	268.00	60.00	120.0
BP487	186974.2	8435102.0	138.6	265.50	57.00	170.0
BP488	187149.9	8435002.0	155.7	266.00	60.50	120.0
BP489	187261.6	8435001.0	152.7	267.00	59.00	120.0
BP490	187354.5	8435151.0	153.0	270.00	60.00	190.0
BP491	186997.2	8435352.0	179.8	85.00	60.00	100.0
BP492	187229.1	8435402.0	175.3	266.00	60.50	90.0
BP493	187246.6	8435352.0	173.0	265.00	60.00	90.0
BP494	186929.5	8435301.0	178.4	87.00	59.50	170.0
BP495	187379.2	8435152.0	150.6	266.50	59.00	190.0
BP496	186898.4	8434652.0	172.3	85.00	61.00	100.0
BP497	186865.8	8434751.0	171.0	268.00	75.00	150.0
BP498	186874.2	8434752.0	171.0	78.00	75.00	80.0
BP499	186862.5	8434671.0	178.8	85.00	70.00	150.0
BP500	187073.5	8434801.0	154.8	274.00	62.00	120.0
BP501	187138.4	8434851.0	151.9	265.00	60.00	120.0
BP502	187133.1	8434901.0	151.4	264.00	60.00	120.0
BP503	187180.1	8434901.0	150.0	264.50	59.00	190.0
BP504	187207.0	8434901.0	148.9	264.50	59.50	190.0
BP505	187158.4	8434951.0	152.5	266.50	59.50	150.0
BP506	187213.3	8434951.0	151.3	268.00	60.00	150.0
BP507	187063.6	8434701.0	153.4	266.00	60.00	130.0

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
BP508	187093.8	8434751.0	154.1	266.50	61.00	130.0
BP511	187117.0	8435402.0	146.8	264.50	63.50	150.0
BP512	187303.9	8435102.0	143.3	269.50	59.50	180.0
BP513	187332.3	8435102.0	142.9	260.00	60.00	170.0
BP514	186814.0	8434901.0	171.1	86.50	66.00	150.0
BP515	186719.3	8434901.0	170.3	90.00	60.00	150.0
BP516	186760.3	8435101.0	162.9	88.00	59.00	150.0
BP517	186844.8	8435201.0	162.1	84.50	60.00	150.0
BP518	186875.3	8435351.0	168.2	89.50	59.00	150.0
BP519	187300.7	8435302.0	167.4	85.00	59.00	140.0
BP520	186769.6	8434736.0	186.9	89.00	59.00	130.0
BP521	187070.7	8434755.0	155.0	85.50	60.00	120.0
BP522	186866.3	8435101.0	162.5	84.50	74.00	120.0
BP523	187093.4	8435352.0	142.5	266.00	63.00	110.0
BP524	187083.4	8435392.0	142.9	264.50	61.00	110.0
DP001	187459.3	8435202.0	145.8	0.00	90.00	31.0
DP002	187435.8	8435207.0	147.3	89.50	60.00	50.0
DP029	187506.6	8435702.0	142.2	90.00	60.00	100.0
DP034	187515.0	8435652.0	142.2	90.00	61.50	75.0
DP038	187515.2	8435602.0	141.7	90.00	59.50	70.0
DP041	187505.2	8435502.0	144.2	90.00	61.00	59.0
DP053	187490.0	8435652.0	143.3	90.00	61.00	80.0
MHT-001	187133.5	8434628.0	147.5	0.00	90.00	0.0
MHT-003	187266.2	8434759.0	144.9	319.20	25.00	201.0
MHT-004	187261.9	8434749.0	159.0	0.00	90.00	0.0
QP089	187464.7	8435303.0	144.8	270.00	60.00	100.0
QP090	187513.8	8435302.0	140.5	270.00	60.00	100.0
QP092	187467.6	8435100.0	146.8	270.00	60.00	100.0
QP093	187468.4	8435203.0	145.0	270.00	60.00	100.0
QP094	187518.7	8435203.0	143.9	270.00	60.00	100.0
QP096	187295.4	8434701.0	146.6	270.00	60.00	100.0
QP097	187247.3	8434703.0	146.6	270.00	60.00	100.0
QP131	187269.5	8434902.0	146.8	270.00	60.00	100.0
QP132	187269.3	8434852.0	146.0	270.00	60.00	82.0
QP133	187265.8	8434799.0	145.6	270.00	60.00	68.0
TP156	187517.9	8435552.0	143.0	90.00	59.00	52.0
VB07-001	187210.4	8434975.0	122.3	270.00	60.80	486.8
VB07-002	187223.1	8434773.0	142.7	270.00	66.00	492.0
VB07-003	187220.9	8435092.0	110.0	270.00	67.50	93.9
VB07-004	187136.3	8435299.0	114.5	248.00	62.00	328.9
VB07-005	187220.9	8435092.0	110.0	270.00	67.00	363.9
VB07-006	187173.9	8435316.0	115.0	248.00	73.00	440.8
VB07-007	187220.9	8435092.0	110.0	270.00	55.00	374.1
VB07-008	187174.7	8435316.0	115.1	0.00	90.00	498.7
VB07-009	187215.5	8435032.0	116.4	270.00	60.00	416.1
VB07-010	187203.4	8434919.0	128.0	270.00	63.00	463.0
VB07-011	187249.6	8435655.0	170.2	270.00	65.00	249.7

HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
VB07-012	187330.0	8435555.0	162.0	270.00	65.00	398.8
VB07-013	187189.5	8434826.0	136.9	270.00	60.00	452.4
VB07-014	187326.8	8435320.0	161.3	281.50	70.00	567.4
VB07-015	186925.8	8435106.0	118.5	113.00	57.00	284.0
VB07-016	187279.5	8435653.0	169.9	270.00	65.00	303.0
VB07-017	187272.3	8435469.0	162.0	270.00	60.00	378.3
VB07-018	187295.8	8435183.0	137.7	270.00	60.00	570.5
VB07-019	186994.6	8434399.0	146.5	270.00	60.00	237.2
VB07-020	186982.5	8434499.0	153.1	270.00	60.00	261.3
VB07-021	187049.9	8434745.0	120.0	278.00	63.00	426.8
VB07-022	186998.9	8434739.0	120.0	45.00	65.00	533.6
VB07-023	187258.2	8435967.0	140.8	272.00	65.00	473.1
VB07-024	187389.1	8435646.0	151.2	270.00	61.00	362.7
VB07-025	186898.3	8434383.0	153.2	15.00	65.00	426.3
VB08-026	187416.9	8434904.0	144.9	267.20	49.20	700.5
VB08-027	187413.7	8434953.0	146.0	266.60	51.70	661.3
VB08-028	187412.9	8435002.0	146.4	268.10	52.90	647.8
VB08-029	187296.9	8435053.0	146.0	266.30	59.10	0.0
VB08-030	187296.8	8435055.0	146.3	275.10	59.60	599.1
VB08-031	187367.3	8435051.0	146.3	273.00	60.60	640.6
VB08-032	187331.8	8435054.0	146.4	273.00	58.20	632.7
VB08-033	187367.9	8435051.0	146.3	278.20	72.70	0.0
VB08-034	187369.0	8435051.0	146.3	274.70	73.20	750.0
VB08-035	187337.3	8435100.0	141.8	268.60	59.80	678.0
VB08-036	187349.2	8435155.0	143.3	274.10	60.00	657.1
VB08-037	187365.5	8435204.0	153.2	272.50	60.50	655.1
VB08-038	187349.6	8435155.0	143.3	278.30	76.30	730.7
VB08-039	187376.3	8435100.0	147.3	272.40	59.50	615.3
VB08-040	187377.0	8435100.0	147.3	274.70	73.70	669.1
VB08-041	187190.6	8435665.0	171.3	88.60	75.40	300.4
VB10-001	187528.9	8435955.0	138.2	274.00	62.31	550.8
VB10-002	187468.4	8435505.0	147.4	269.76	55.56	287.4
VB10-003	186748.0	8435114.0	163.5	267.00	80.08	525.7
VB10-004	187589.0	8434904.0	141.7	280.75	59.40	864.4
VB10-005	187019.6	8434457.0	148.8	268.39	61.38	410.4
VB10-006	187460.3	8435251.0	145.3	273.83	62.26	721.7
VB10-007	187330.1	8434779.0	144.4	270.00	66.00	704.5
VB10-008	187446.0	8435004.0	145.0	274.91	60.83	735.5
VB10-009	187239.0	8434593.0	148.7	270.00	60.00	669.5
VB10-010	187349.9	8435855.0	167.8	271.00	67.00	48.0
VB10-011	187275.2	8435547.0	162.4	279.41	68.45	630.5
VB10-012	187240.0	8434890.0	150.0	263.35	54.96	725.9
VB11-001	187364.6	8435054.0	146.4	290.63	50.00	596.1
VB11-002	187354.7	8435149.0	143.4	288.63	50.00	572.9
VB11-003	187271.4	8434847.0	149.2	273.63	55.00	535.0
VB11-012	187443.5	8434999.0	145.0	269.23	58.60	806.8
VB11-013	187261.3	8435702.0	169.7	271.63	67.20	388.3

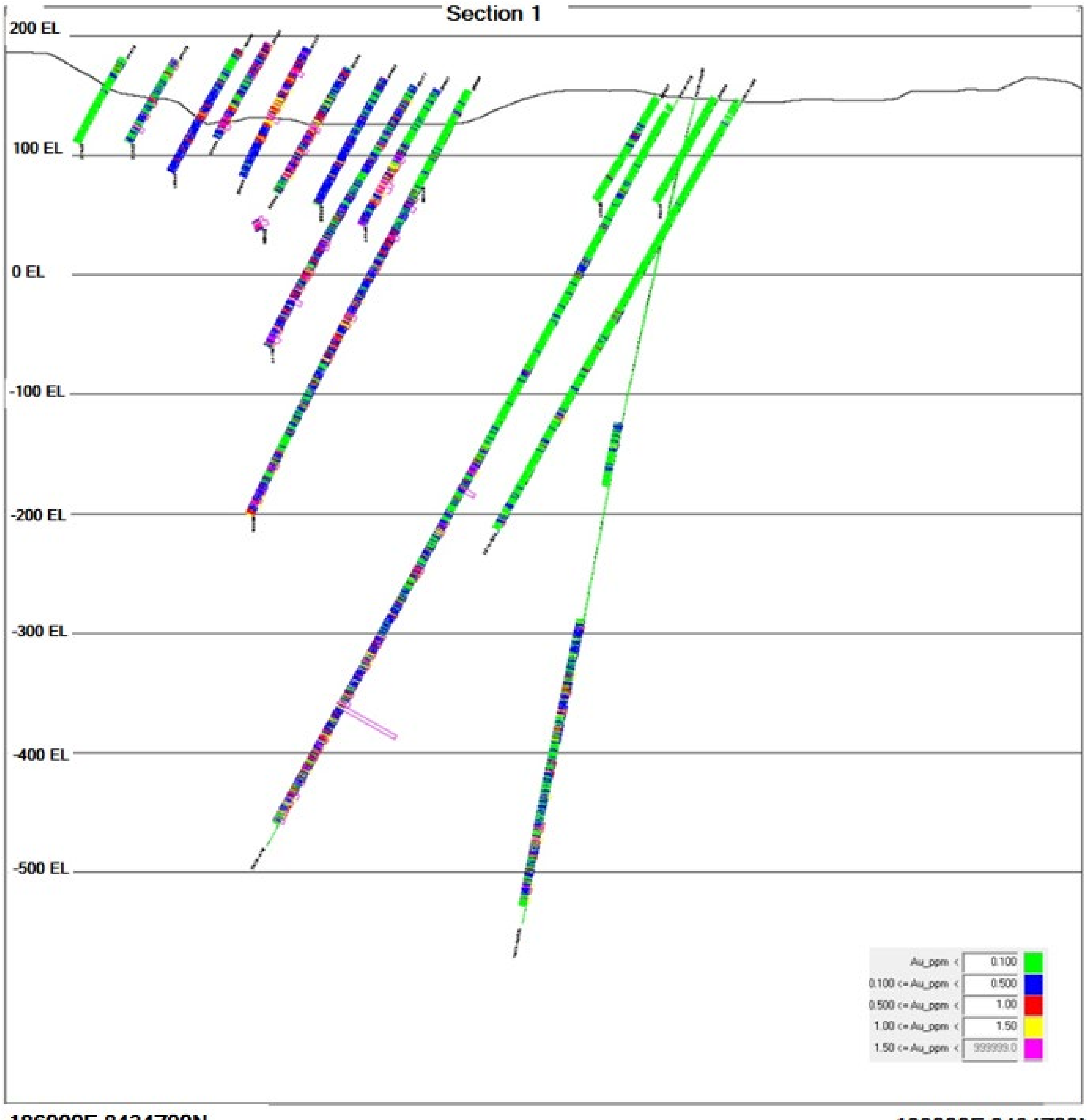
HOLE-ID	EASTING	NORTHING	ELEV.	AZIMUTH	DIP***	DEPTH
VB11-014	187329.2	8434775.0	144.2	271.13	58.90	704.9
VB11-015	187508.3	8435165.0	142.6	271.63	61.00	875.9
VB12-001	187434.8	8434812.0	144.7	268.63	65.00	744.2
VB12-002	187446.6	8434901.0	144.4	268.63	58.00	750.3
VB12-003	187133.3	8434601.0	147.3	268.63	60.00	625.0
VB12-004	187257.2	8435702.0	169.7	268.63	55.00	383.4
VB12-005	187433.6	8434952.0	145.7	268.63	60.00	759.1
VB12-006	187299.4	8435737.0	169.4	269.63	55.00	475.6
VB12-007	187501.5	8435400.0	143.3	268.63	60.00	887.8
VB12-008	187289.7	8435360.0	161.9	268.63	65.00	645.9
VB12-009	187315.5	8434701.0	144.5	268.63	60.00	717.2
VB12-010	187432.9	8435610.0	145.1	266.63	60.00	751.0
VB12-011	187093.5	8434604.0	146.4	266.63	57.00	629.8
VB12-012	187466.2	8435352.0	146.0	266.63	63.00	793.7
VB12-013	187445.2	8434901.0	144.4	270.13	55.00	883.1
VB12-014	187412.0	8435757.0	149.5	266.63	60.00	754.0
VB12-015	187446.7	8434902.0	144.4	263.63	56.00	745.8
VB12-016	187262.7	8434704.0	147.3	265.63	60.00	713.5
VB12-017	187391.2	8435349.0	150.8	265.63	63.00	833.3
VB12-018	187429.9	8434849.0	144.7	265.63	58.00	177.0
VB12-019	187429.5	8434847.0	144.8	265.63	60.00	731.8
VB12-020	187359.6	8435852.0	167.3	265.63	65.00	611.9
VB12-021	187378.8	8435954.0	149.9	266.70	65.20	602.9
VB12-022	187179.3	8434453.0	153.3	265.64	56.59	647.9
VB12-023	187371.0	8435801.0	161.3	264.45	60.03	650.9
VB12-024	187094.7	8434482.0	149.8	268.63	60.00	460.1
VB12-025	187344.7	8435656.0	158.6	260.63	60.00	650.6
VB12-026	187066.8	8434393.0	144.8	268.63	60.00	378.9
VB12-027	187259.7	8435717.0	169.8	290.63	55.00	434.8
VB15-001	187431.0	8434480.0	147.0	268.30	75.81	455.5
VB15-001W1	187431.0	8434480.0	147.0	268.30	75.81	831.8
VB15-001W2	187431.0	8434480.0	147.0	268.30	75.81	746.0
VB15-002	187277.0	8434703.0	147.3	266.07	76.19	446.3
VB15-002W1	187277.0	8434703.0	147.3	266.07	76.19	705.0

NOTE:
* Positive dip represents downward measurements in this table



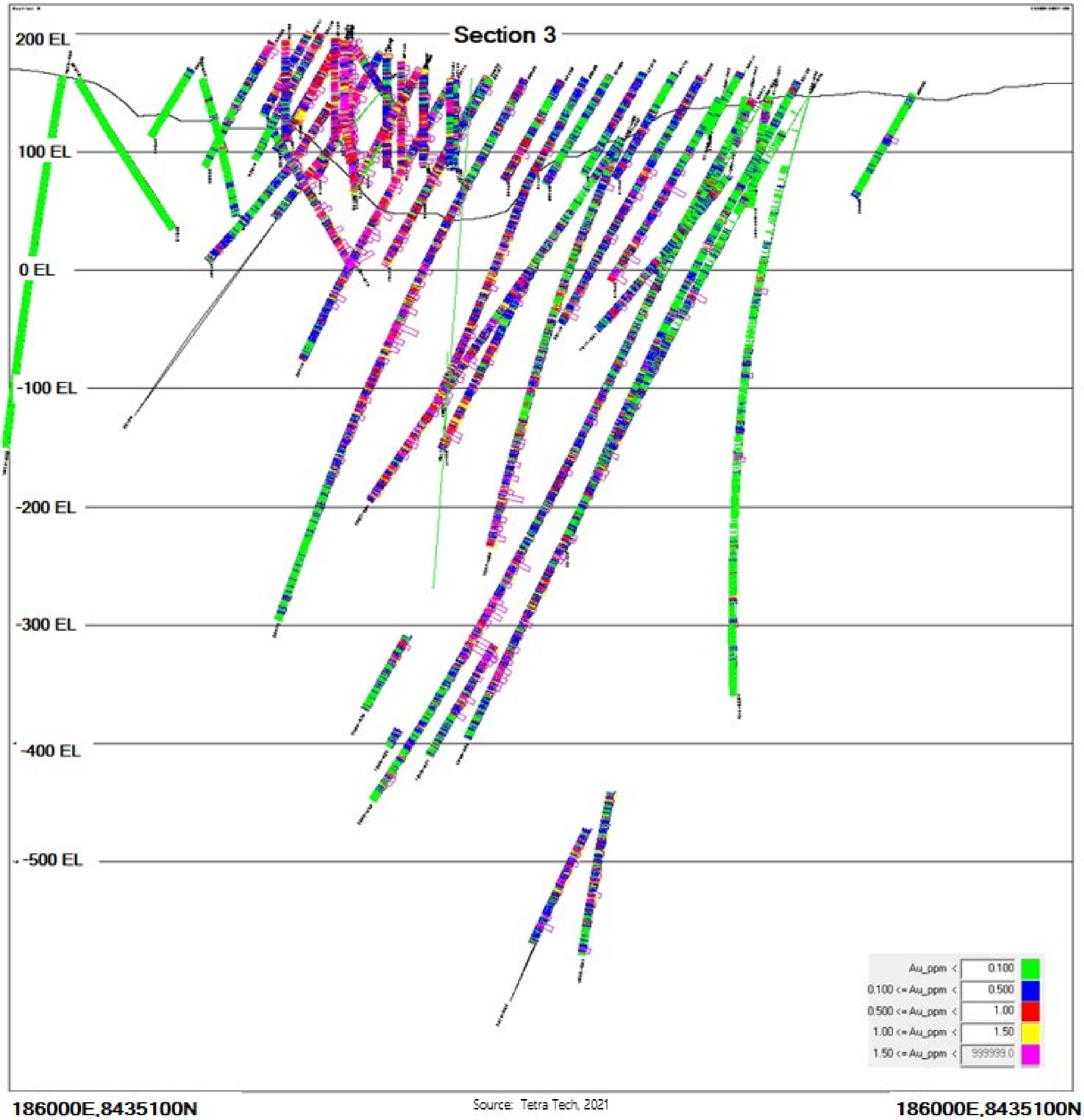
Source: Tetra Tech, 2021

Figure 10-1: Drillhole Location Map Batman Deposit to VB17-003



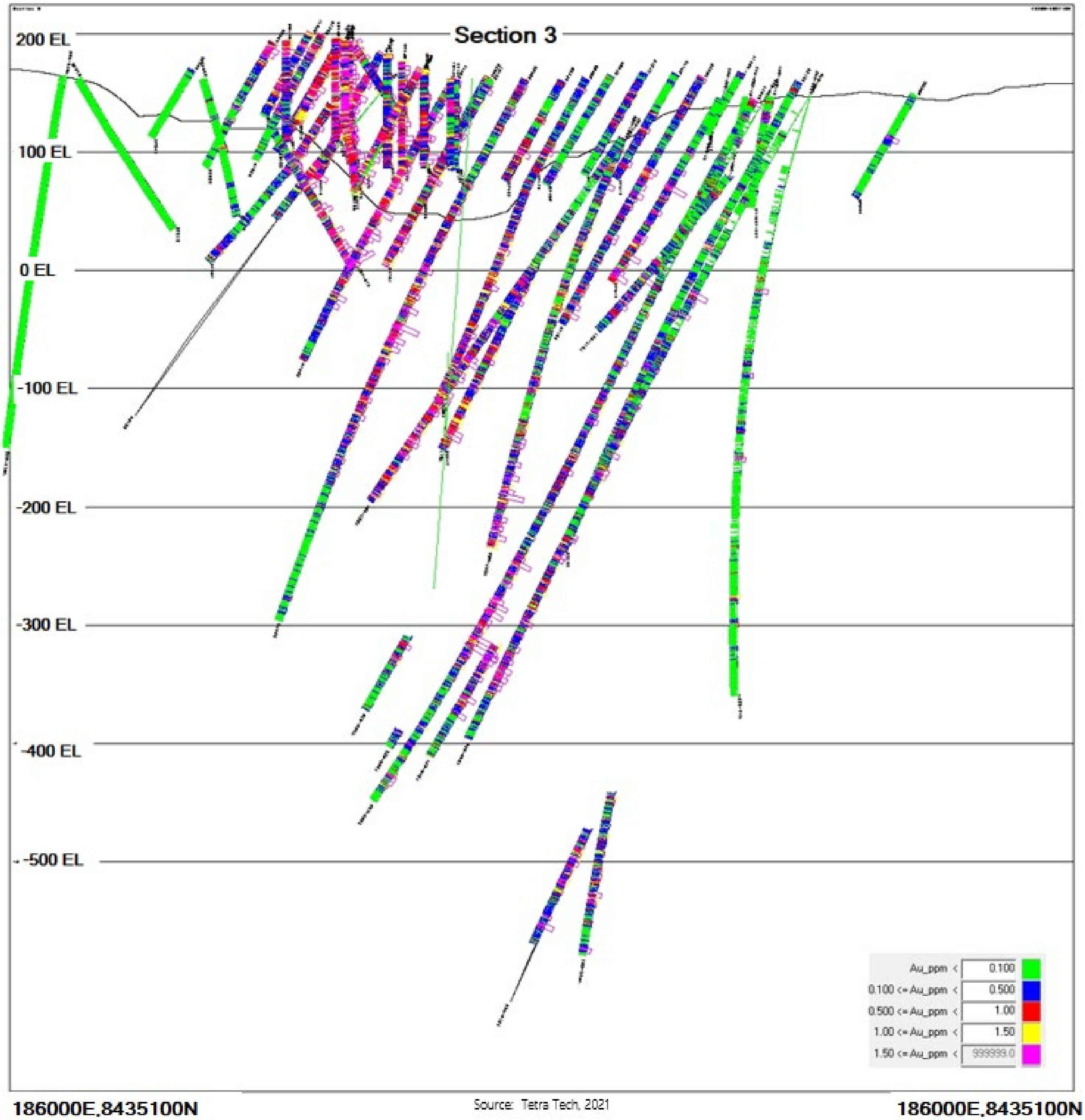
Source: Tetra Tech, 2021

Figure 10-2: Batman Cross-section 1 (see Figure 10-1 for location)



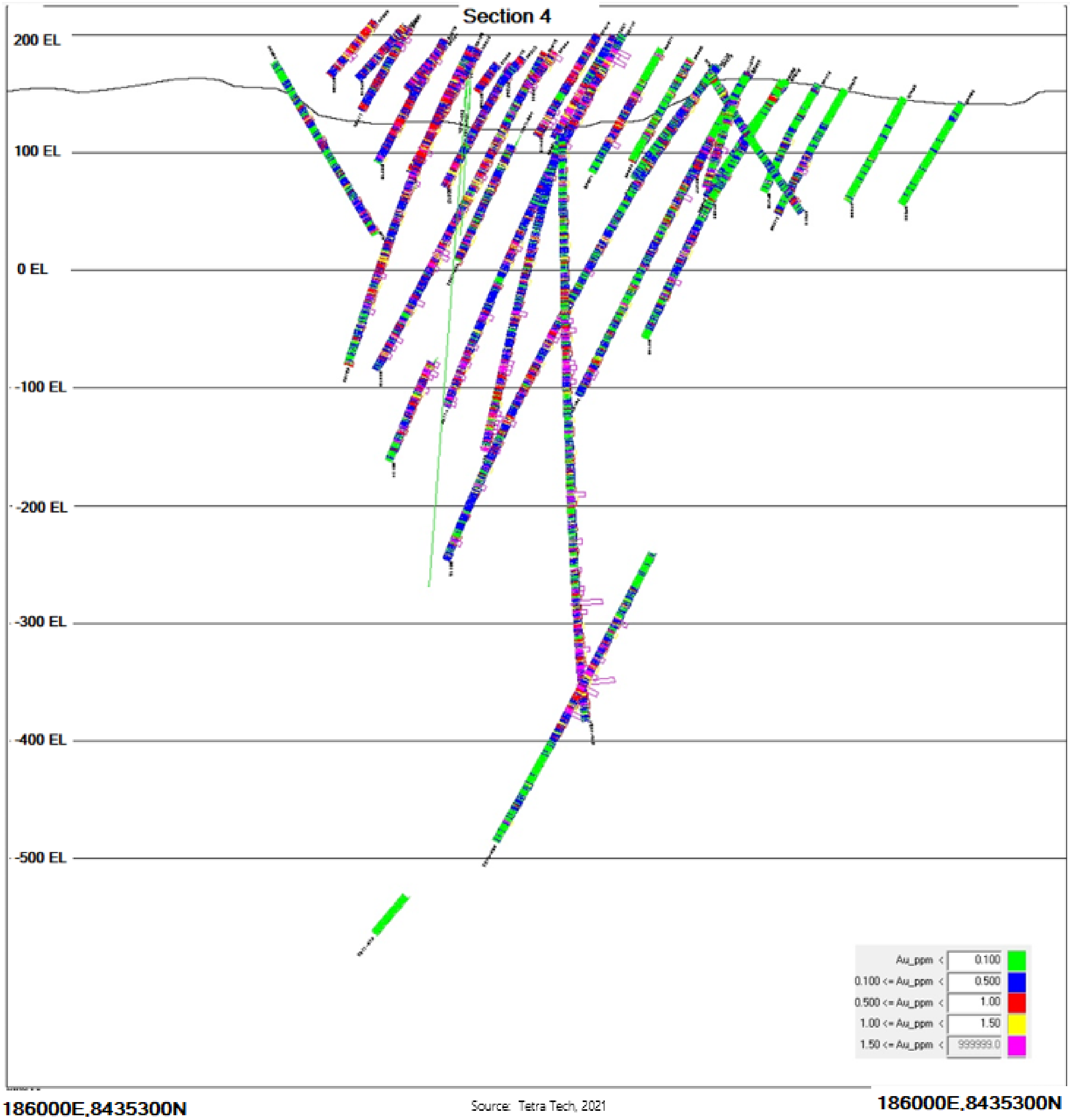
Source: Tetra Tech, 2021

Figure 10-3: Batman Cross-section 2



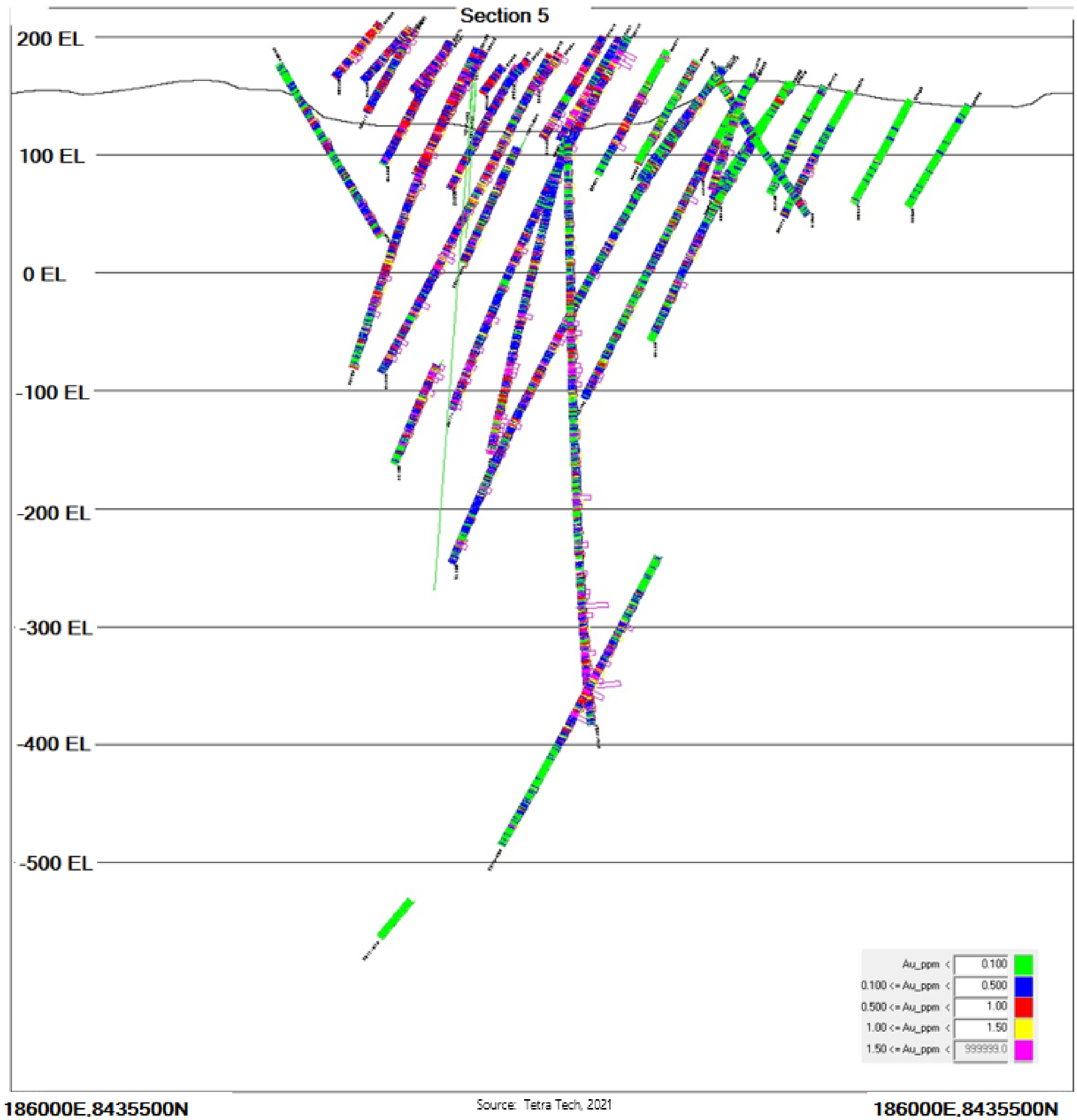
Source: Tetra Tech, 2021

Figure 10-4: Batman Cross-section 3



Source: Tetra Tech, 2021

Figure 10-5: Batman Cross-section 4



Source: Tetra Tech, 2021

Figure 10-6: Batman Cross-section 5

10.2 Summary of Quigleys Drilling 1975-2011

Table 10-4 shows the Quigleys deposit drilling history. Quigleys was mined from 1982 to 1987 during which the largest amount of drilling was percussion type used for ore grade control. Table 10-5 lists the drillholes used for the current estimate the Quigleys deposit.

Figure 10-7 is an isometric view of the Quigleys drilling database with the position of the A-A′ section at 8,438,200 north shown. Figure 10-8 shows the A-A′ cross-sectional view of the deposit looking north; with the mineralized zone dipping to the west with the orientation of the drillholes dipping to the east. Relevant intervals of mineralization are contained within blanket-like zones which are modeled with 3-D wireframes for resource estimation. These zones are shown in Figure 14-14. The mineralized zones have been defined by wireframes which are used to constrain the higher grades for the resource estimation. Most of the drilling has been angled to be approximately perpendicular to the mineralized core. This orientation more accurately transects the true thickness of the mineralization. While there are random high-grade intercepts outside of the core, the majority of higher-grade mineralization resides within the defined zones. In 2011, Vista explored the potential for a deeper deposit with three diamond drillholes, each over 350 meters in depth.

Table 10-4: Quigleys Deposit Drilling History

Date	Reference	Holes (#)	Percussion (m)	Diamond (m)	RC (m)
1975	Australian Ores and Minerals/Esso	2		200	
1981	Arafura Mining Corp/CRA	14		676.5	
1982-1987	Pacific Gold Mines NL (Small Scale Mining)	603	41,429	9710	4,013
1989	Pacific Gold Mines	9	501	202	
2011	Vista	3		1,090	
1988-2017	**Quigleys Total**	**631**	**41,930**	**11,878**	**4,013**

Table 10-5: Quigleys Drillhole Details

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
DDH1	173	189529.1	8438240	186	108	-60
DDH2	105.81	189621.7	8438288	170	0	-90
MT01	82.54	189693.1	8438406	175	102	-60
MT02	54.04	189648.4	8438198	197.7	0	-90
MT02A	17.78	189648	8438196	197.8	0	-90
MT03	73.74	189623.3	8438159	194.5	0	-90
MT03A	24.25	189624.3	8438159	194.5	110	-60
MT04	35.8	189736.3	8438397	178	0	-90
MT05	36.9	189734	8438305	205.8	111	-60
MT06	53.73	189654.6	8438334	176.5	0	-90
MT07	23.7	189793.6	8438649	169.6	0	-90
MT08	23.44	189854.5	8438754	167	0	-90
MT09	49	189748.5	8438647	158	0	-90
MT10	49.15	189860	8438851	157	98	-60
MT11	50.66	189808	8438775	153	113	-60
MT12	34.85	189752.4	8438547	168	0	-90
MT13	37.75	189630.5	8438283	161	126	-50
MT14	29.21	189635.7	8438042	148.9	86	-45
MT15	24.5	189673.3	8438045	163.4	0	-90
MT16	35.5	189636.1	8438045	149.2	0	-60
MT17	41	189618.5	8438196	197.4	90	-70
MT18	49.6	189618.3	8438196	197.4	0	-90
MT19	14.5	189647.9	8438201	197.5	90	-60
MT20	33	189645	8438201	197.4	270	-70
MT21	35.3	189706.8	8438347	199	0	-90
MT22	50	189706.8	8438347	199	270	-70
MT24	31.5	189755.4	8438553	168.9	90	-60
MT25	20.5	189756.2	8438523	169.6	0	-90
MT26	17.5	189764.7	8438503	169.3	0	-90
MT27	17	189795.7	8438654	169.9	90	-60
MT28	40	189795.7	8438654	169.9	270	-70
MT29	36.2	189751.5	8438654	158.8	90	-60
MT30	22.5	189793.1	8438663	179.4	0	-90
MT31	14.5	189789.8	8438605	168.2	0	-90
MT32	20.5	189790.4	8438629	168.6	0	-90
MT33	20.6	189864.6	8438755	169	0	-90
MT34	17.5	189870.1	8438774	160.8	0	-90
MT35	19.6	189874.4	8438799	159.5	0	-90
MT36	23.3	189877	8438823	160	0	-90
MT37	40.2	189882.4	8438847	161	0	-90
MT38	13	189893.8	8438848	163.5	0	-90
MT39	19.5	189732.7	8438347	199.1	0	-90
MT40	27.7	189732.7	8438347	199.1	90	-60
MT41	33	189732.7	8438347	199	270	-60
MT42	15	189756.8	8438347	198.8	0	-90

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
PRP017	30	189458.6	8437854	158	99	-60
PRP018	30	189447.8	8437774	159.6	0	-90
PRP019	29.5	189454.9	8437830	162.5	98	-59
QD001	196.5	188989.8	8437999	143.7	42	-47
QD002	320.5	189053.1	8437801	134	60	-66
QD003	108.1	189448	8438016	146.1	13	-50
QD004	295.8	189447.1	8438013	146	341	-50
QD005	90	189577.9	8438230	195.92	92	-61
QD006	100	189519.7	8438232	185.59	0	-90
QD007	119.5	189535.5	8438308	163.4	110	-70
QD008	130	189422.4	8438202	163.53	90	-90
QD009	139	189431.6	8438258	166.4	126	-69
QD010	140	189432.1	8438147	146.67	90	-90
QD011	250	189470.7	8438070	160.92	94	-60
QD012	251.9	189493.5	8437942	142.8	90	-60
QD013	251.56	189491.6	8437887	150.8	94	-60
QD014	250	189491.6	8437985	143.77	92	-60
QD015	251.81	189515.5	8438306	163.1	92	-59
QD016	251.7	189570.9	8438410	155.08	88	-61
QD017	180.5	189417.8	8438007	145.12	88	-60
QD019	185.81	189410	8438307	153.13	96	-60
QD021	260.5	189216.7	8438308	135.19	94	-61
QD022	251.7	189221.6	8438003	133.24	92	-60
QD024	150	189414.5	8438206	167.02	90	-60
QD025	228	189222.5	8438203	134.64	90	-60.2
QD026	249.5	189186.8	8438104	133.21	90	-60
QD027	249	189317.7	8438307	141.79	88	-60
QD028	180.5	189316.6	8438106	158.88	96	-60
QD029	245.5	189318.3	8438007	150.84	94	-60
QD030	216.5	189276.6	8438212	139.11	92	-60
QD031	249.4	189314.2	8437965	139.7	98	-60
QD035	123.3	189359	8437906	131.65	0	-60
QD036	111.6	189310.8	8437919	135.8	0	-60
QD037	114.6	189270.6	8437895	134.13	0	-60
QD038	111.61	189167.5	8437905	132.62	0	-60
QNE001	30	189881.3	8439055	149	88	-60
QNE002	30	189876.5	8439074	147.3	88	-60
QNE003	25	189872.8	8439093	146.4	86	-60
QNE004	30	189863.7	8439115	145.4	88	-60
QNE005	25	189864.3	8439132	144.9	88	-60
QNE006	30	189851.4	8439155	143.7	90	-60
QNE007	40	189840.2	8439172	141.7	87	-60
QNE008	30	189853.6	8439171	143.2	90	-60
QNE009	40	189835.3	8439200	140.5	83	-60
QNE010	29.5	189843	8439199	140.3	83	-60
QNE011	30	189836	8439218	140.7	93	-60
QP001	50	188967.3	8438006	145.5	47	-60

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP002	50	189020.3	8438009	143.7	47	-60
QP003	50	189074.3	8438012	139.3	47	-60
QP004	74	189572.5	8438262	182.2	90	-60
QP005	62	189602.5	8438260	182.3	90	-60
QP006	100	189525.4	8438233	185.4	90	-60
QP007	80	189578.1	8438231	195.8	90	-60
QP008	74	189572.4	8438201	189.4	90	-60
QP009	73	189582.2	8438182	186.7	0	-90
QP010	70	189574	8438153	180.4	90	-90
QP011	78	189548.2	8438131	175.8	90	-60
QP012	60	189599.8	8438131	180.3	90	-60
QP013	64	189594.3	8438081	155.1	90	-60
QP014	48	189638.6	8438081	158.7	90	-60
QP015	71	189598.2	8438046	147.4	90	-60
QP016	40	189647.8	8438045	154.6	90	-60
QP017	70	189617.2	8438006	146	90	-60
QP018	50	189642.7	8438007	150.1	90	-60
QP019	56	189669.7	8438007	160	90	-60
QP020	60	189691.6	8438006	167.6	90	-60
QP021	62	189633.4	8437982	155.9	90	-60
QP022	62	189658	8437981	162.6	90	-60
QP023	50	189644.4	8437933	163.7	90	-60
QP024	55	189663.5	8437931	173.3	90	-60
QP025	80	189571.6	8438202	189.5	0	-65
QP026	74	189568.7	8438259	182.6	180	-65
QP027	60	189628.2	8438255	181.4	90	-60
QP028	56	189552.8	8438328	155.9	90	-60
QP029	60	189585	8438331	152.1	90	-60
QP030	60	189676.8	8438052	165.57	70	-60
QP031	60	189677.5	8438129	183.81	90	-60
QP033	60	189625.8	8438240	181.28	130	-65
QP035	60	189729.1	8438331	197.58	0	-90
QP036	39	189577.9	8438230	195.92	91	-61
QP037	75	189510.9	8438170	157.5	90	-90
QP038	54.41	189519.7	8438232	185.59	0	-90
QP039	69	189535.5	8438308	163.4	110	-70
QP041	60	189422.4	8438202	163.53	90	-90
QP042	68	189431.6	8438258	166.4	126	-69
QP043	74	189817.2	8439072	143.71	90	-60
QP045	96	189764.8	8438725	160.13	118	-65
QP046	66	189717.8	8438658	152.9	90	-60
QP047	60	189762.2	8438657	164.43	90	-60
QP048	50	189781.7	8438657	164.43	86	-60
QP049	50	189816.7	8438652	170.1	270	-60
QP050	60	189693.9	8438580	153.71	90	-60
QP051	80	189629.7	8438406	174.4	90	-60
QP052	60	189717.7	8438397	179.27	90	-60

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP053	50	189749.7	8438407	177.85	90	-60
QP054	50	189778.6	8438411	181.02	90	-60
QP055	60	189629.4	8438245	181.34	180	-60
QP056	60	189627.3	8438212	174.22	0	-60
QP057	50	189629.9	8438183	173.22	0	-60
QP058	50	189637.6	8438153	173.61	0	-60
QP059	60	189632	8438254	181.47	180	-60
QP060	88	189431.8	8438254	166.39	170	-60
QP061	46	189431.5	8438245	167.68	0	-90
QP062	94	189426.4	8438224	165.48	0	-60
QP063	51	189510.6	8438170	157.73	90	-60
QP064	60	189494.4	8438173	155.19	0	-90
QP065	94	189544.3	8438257	182.38	90	-60
QP066	90	189516.2	8438237	185.4	0	-60
QP067	40	189497.1	8438259	184.68	180	-60
QP068	80	189572.4	8438231	195.59	0	-90
QP069	100	189571.5	8438223	196.14	0	-60
QP070	60	189571.3	8438200	189.24	0	-90
QP071	64	189572.9	8438155	180.72	0	-60
QP072	94	189567.8	8438146	178.47	0	-60
QP073	64	189567.9	8438145	178.26	0	-90
QP074	77	189546.6	8438131	176.3	0	-90
QP075	88	189515.1	8438131	170.4	0	-90
QP076	90	189517.3	8438119	171.68	0	-60
QP077	97	189432.9	8438151	147.29	0	-90
QP078	112	189515.8	8438108	171.87	0	-90
QP079	70	189539.6	8438221	186.21	0	-90
QP080	112	189512.4	8438254	184.6	0	-90
QP081	97	189538.4	8438257	182.81	0	-90
QP082	48	189651.9	8438281	181.7	0	-90
QP083	54	189655.1	8438307	179.7	162	-89
QP084	60	189697.9	8438183	200.23	354	-60
QP085	60	189697.9	8438220	199.99	180	-60
QP086	34	189649	8438267	181.9	85	-50
QP134	100	189718.9	8437206	134.53	270	-59
QP135	102	189767.5	8437203	133.4	276	-60
QP136	102	189818.1	8437202	130.64	268	-60
QP137	102	189813.8	8437407	135.3	96	-60
QP138	102	189717	8437406	144.67	90	-63
QP139	102	189667.6	8437507	133.53	88	-60.5
QP140	102	189621.4	8437506	132.16	92	-60
QP141	100	189691.3	8437626	149.28	90	-61
QP142	100	189645	8437627	147.6	90	-59.5
QP143	100	189597.9	8437626	143.44	88	-61
QP144	100	189767.5	8437405	141.43	92	-59.9
QP145	100	188870	8437801	136.24	92	-60.4
QP146	100	188917.3	8437804	135.96	90	-59.8

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP147	100	188962	8437800	135.65	90	-60
QP148	100	189021.2	8437804	133.88	90	-61
QP149	100	189127	8437767	132.06	90	-61
QP150	100	188816.5	8438007	139.51	92	-60
QP151	100	188867.3	8438006	139.87	92	-60.5
QP152	100	188966.1	8438004	145.37	90	-61
QP153	100	188919.9	8438004	139.12	90	-60.2
QP154	100	189222.5	8438203	134.64	90	-60.2
QP155	100	189417.5	8438406	144.28	90	-60.8
QP156	100	189317.3	8438407	139.62	90	-60
QP170	100	189690.4	8437677	157.11	92	-60
QP171	97	189647.8	8437678	146.37	90	-60
QP172	80	189721.4	8437721	170.85	92	-65
QP173	80	189695	8437721	166.73	90	-60
QP174	80	189669.4	8437724	159.03	92	-60
QP175	80	189649.5	8437728	151.5	90	-59.5
QP176	80	189698.5	8437778	181	92	-60
QP177	80	189587.1	8437776	158.77	92	-60.5
QP178	80	189723.5	8437827	191.67	90	-60
QP179	80	189694.4	8437829	193.81	104	-60
QP180	80	189667	8437829	190.58	90	-59
QP181	80	189642	8437832	184.35	92	-59
QP182	80	189618	8437835	176.95	92	-61
QP183	80	189595.4	8437834	168.85	92	-61
QP184	100	189596.6	8437881	161.4	92	-60
QP185	80	189699.2	8437930	181.86	94	-60
QP186	80	189623.6	8437936	155.12	88	-60
QP187	60	189641	8438081	158.25	92	-60
QP188	80	189721.5	8437876	193.02	94	-60
QP189	80	189630	8437726	146.24	90	-61
QP190	80	189603.3	8437727	146.79	90	-61
QP191	100	189594.2	8437683	140.86	94	-60
QP192	100	189568.8	8437505	135.54	88	-60
QP193	100	189166.6	8437780	131.68	92	-60
QP194	100	189276.6	8438212	139.11	92	-60
QP195	100	189328.8	8438209	154.87	90	-60
QP196	100	189370.4	8438211	169.99	92	-60
QP197	100	189012.9	8438204	148.84	92	-60
QP198	97	189165.9	8438207	141.52	94	-59.5
QP199	100	189119.4	8438204	147.53	94	-60
QP205	100	189068.2	8438203	155.01	94	-60.5
QP206	100	189170.4	8438407	154.7	94	-60
QP207	100	189070.4	8438407	151.3	96	-58
QP208	100	189117.9	8438404	151.74	96	-59
QP209	100	189628.9	8437990	153.47	94	-58.5
QP210	100	189599.4	8437989	146.41	94	-60
QP211	100	189561.1	8437989	142.66	92	-59.5

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP212	100	189568.5	8437964	143.69	90	-60
QP213	100	189620.5	8437960	154.68	90	-60
QP214	100	189593.5	8437963	147.43	96	-59.5
QP215	100	189594.8	8437939	146.31	94	-60
QP216	100	189617.1	8437915	150.72	94	-59
QP217	100	189575.1	8437943	144.52	94	-60
QP218	100	189558.8	8437909	148.98	94	-59.5
QP219	100	189567.7	8437888	151.8	92	-60
QP237	100	189595.9	8437804	167.79	94	-58.5
QP238	100	189568.1	8437803	157.68	94	-61
QP239	100	189560.2	8437826	155.45	98	-59.5
QP240	100	189590.9	8437856	161.63	90	-59.5
QP241	100	189569	8437871	149.53	90	-60
QP242	100	189585.4	8437916	149.26	90	-58.5
QP253	100	189069.2	8438312	163.59	92	-60
QP254	100	189116	8438308	152.37	88	-57
QP255	100	189662.6	8438037	159.66	92	-59
QP256	106	189665.7	8438074	164.97	94	-59.5
QP257	100	189710.4	8438015	166.56	92	-59
QP258	100	189663.7	8438112	176.49	92	-59
QP259	100	189619.4	8438114	173.34	90	-57
QP260	100	189569	8438153	176.65	92	-59
QP261	100	189569.6	8438123	171.28	94	-59.5
QP262	112	189516.6	8438105	170.03	94	-58.5
QP263	100	189513.8	8438142	167.97	100	-58.5
QP264	100	189468.5	8438103	161.54	92	-59
QP265	100	189470.7	8438070	160.92	94	-60
QP266	100	189518.1	8438066	156.18	88	-60
QP267	100	189568.2	8438069	148.06	90	-60
QP268	100	189613.2	8438074	152.05	88	-60
QP269	100	189643.5	8437961	161.94	90	-60
QP270	100	189367.1	8438006	153.55	92	-60
QP271	106	189318.3	8438007	150.84	94	-60
QP272	68	189466.7	8438007	146.8	92	-60
QP273	106	189515.4	8438007	144.93	92	-60
QP274	100	189491.6	8437985	143.77	92	-60
QP275	100	189541.7	8437988	140.91	94	-60
QP276	100	189542.9	8438012	141.18	94	-60
QP277	100	189565.4	8438015	142.86	94	-61
QP278	106	189613	8438038	147.22	90	-60
QP279	106	189592.5	8438016	145.31	92	-60
QP280	100	189566.1	8438035	143.22	90	-60
QP281	100	189542	8437943	142.35	94	-60
QP282	100	189493.5	8437942	142.8	90	-60
QP283	100	189441.5	8437941	137.32	90	-61
QP284	112	189491.6	8437887	150.8	94	-60
QP285	100	189443.5	8437885	148.82	90	-60

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP286	100	189298.1	8437882	135.97	90	-60
QP287	100	189194.8	8437883	137.89	92	-59
QP288	58	189146.7	8437881	132.5	90	-59
QP289	100	188917.4	8437916	138.68	88	-62
QP290	105	188862.9	8437907	137.21	92	-61
QP291	100	188817.9	8437905	137.03	92	-61
QP292	100	188765.8	8437905	140.29	90	-61
QP293	100	188717	8437906	140.03	88	-61
QP294	106	188767.9	8438006	139.49	94	-60
QP295	100	188722.3	8438006	141	94	-60
QP296	100	189247.4	8437883	134.08	94	-60
QP297	100	188825.7	8438110	136.21	90	-60
QP298	100	188914.8	8438203	135.35	94	-59
QP299	100	189121	8438509	144.05	90	-60
QP300	100	189542.4	8437827	148.9	88	-60
QP301	100	189537	8437773	142.6	90	-60
QP302	100	189549.6	8437730	138.7	88	-59
QP303	100	189578.2	8437728	142.39	88	-60
QP304	106	189619.4	8437680	140.59	92	-58
QP305	100	189570.9	8437678	138.31	88	-60
QP306	100	189525.8	8437684	134.13	88	-60
QP307	100	189549.5	8437625	134.88	88	-59.5
QP308	100	189815.1	8437511	139.99	88	-60
QP309	100	189764.9	8437504	141.5	90	-58
QP310	100	189713.4	8437508	140.09	86	-60
QP311	100	189738.7	8437636	158.74	86	-59
QP312	100	189739.3	8437680	166.39	90	-58
QP313	100	189747.6	8437721	166.88	88	-59
QP314	100	189623.8	8437855	176.33	86	-59
QP315	100	189541.5	8437888	145.81	90	-60
QP316	112	189488.9	8438138	166.5	86	-59
QP317	100	189537.4	8438144	170.04	90	-61
QP318	100	189538.1	8438115	171.87	94	-60
QP319	100	189595.6	8438120	173.9	94	-59
QP320	100	189636.8	8438114	174.2	88	-54
QP321	100	189686.9	8438112	178.54	88	-58
QP322	100	189688.9	8438074	173.11	90	-58
QP323	100	189732	8438015	161.07	88	-55
QP324	100	189756.7	8438012	153.96	92	-59
QP325	100	189538	8438071	156.04	90	-59
QP326	100	189603.2	8438154	190.35	90	-59
QP327	100	189585.8	8438196	192.38	92	-59
QP328	100	189642.7	8438149	174.87	90	-59
QP329	106	189630.8	8438197	174.68	86	-58
QP330	100	189069.1	8438508	142.15	92	-60
QP331	100	189021	8438505	141.37	90	-60
QP332	100	188968.9	8438408	140.06	90	-60

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP333	104	189166.3	8438103	133.24	92	-60
QP334	100	189119.8	8438106	135.09	90	-59
QP335	100	189067.4	8438101	135.41	90	-60
QP336	100	189027.2	8438096	136.3	94	-60
QP337	100	189013.5	8437904	137.96	90	-60
QP338	100	188966.3	8437905	138.77	92	-61
QP339	100	189061.8	8437904	136.46	90	-60
QP340	100	189116.2	8438006	137.09	90	-61
QP341	100	189068.9	8438008	139.53	88	-61
QP342	100	189026	8438013	143.43	94	-61
QP343	100	189165.9	8437999	134.94	94	-61
QP344	100	189139.4	8437880	132.59	90	-60
QP345	100	189216.7	8438308	135.19	94	-61
QP346	100	189266.9	8438307	137.51	94	-60
QP347	100	189471.2	8438413	144.96	90	-61
QP348	100	189367	8438403	140.76	90	-61
QP349	100	189271	8438409	137.4	94	-60
QP350	100	189191.7	8438153	134.24	94	-60
QP351	100	189186.8	8438104	133.21	90	-60
QP352	100	189140.9	8438105	134.38	88	-60
QP353	100	189142	8438153	135.45	88	-60
QP354	106	189185.9	8438053	133.44	92	-60
QP355	100	189139.7	8438054	134.67	92	-61
QP356	100	189221.6	8438003	133.24	92	-60
QP357	108	189137.9	8437951	133.64	90	-60
QP358	100	189184.4	8437953	132.82	84	-61
QP359	100	189120.8	8437896	132.4	90	-60
QP360	100	189673.3	8437879	184.83	90	-60
QP361	100	189711.2	8437953	177.87	90	-59
QP362	100	189690.9	8437958	177.44	92	-58.5
QP363	100	189661.8	8437911	174.1	90	-59
QP364	100	189669.7	8437955	173.28	88	-58
QP365	100	189713.5	8438072	169.54	94	-60
QP366	100	189669.1	8437982	166.94	90	-60
QP367	100	189653.1	8438017	153.53	94	-60
QP368	100	189574	8437625	138.91	100	-59
QP369	98	189524.7	8437627	134.1	94	-60
QP370	99	189446.7	8437773	158.78	90	-61
QP371	100	189487.8	8437768	148.71	90	-61
QP372	100	189504.4	8437725	140.78	92	-61
QP373	100	189495.9	8437826	149.66	90	-59
QP374	100	189515.5	8438306	163.1	92	-59
QP375	100	189566.4	8438307	163.41	94	-61
QP376	100	189605.9	8438307	156.15	92	-61
QP377	100	189471.1	8438315	152.85	96	-60
QP378	98	189410	8438307	153.13	96	-60
QP379	100	189364.4	8438303	151.68	88	-61

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP380	100.11	189317.7	8438307	141.79	88	-60
QP381	98	189167.8	8438308	140.75	90	-61
QP382	100	189020.7	8438308	164.19	92	-61
QP383	98	188968.4	8438304	148.84	94	-61
QP384	100	188969	8438206	142.98	92	-61
QP385	100	189018.7	8438416	144.94	92	-61
QP386	100	189316.6	8438106	158.88	96	-60
QP387	100	189269.9	8438106	154.74	94	-60
QP388	100	189414.5	8438206	167.02	90	-60
QP389	100	189411.8	8438254	174	92	-60
QP390	100	189640.5	8438255	181.07	90	-57
QP391	100	189691.3	8438252	182.25	90	-58
QP392	100	189640.5	8437780	170.87	86	-57
QP393	100	189472.5	8438255	183.12	86	-57
QP394	106	189519.8	8438257	182.33	90	-58
QP395	100	189540.6	8438207	182.81	94	-59
QP396	100	189497.3	8438224	183.78	88	-60
QP397	106	189591.8	8438261	182.34	90	-58
QP398	100	189661.1	8438300	178.96	94	-59
QP399	100	189675.4	8438197	197.72	90	-59
QP400	100	189669.2	8438416	175.52	90	-58
QP401	100	189765.3	8438304	208.96	90	-59
QP402	100	189765.7	8438351	199.1	92	-58
QP403	100	189728.2	8438314	196.01	102	-59
QP404	100	189714.7	8438353	200.3	92	-59
QP405	100	189765	8438407	181.81	92	-59
QP406	100	189766.2	8438452	170.63	90	-58
QP407	100	189720.9	8438398	178.63	92	-60
QP408	100	189719.3	8438455	156.24	96	-59
QP420	100	189365.4	8438251	165.78	84	-51
QP421	100	189369.1	8438102	158.5	94	-60
QP422	100	189420.1	8438103	143.77	88	-61
QP423	104	189217.5	8438105	143.06	88	-60
QP424	99	189222.3	8438408	141.7	94	-60
QP425	100	189417.8	8438007	145.12	88	-60
QP426	101	189350.7	8437879	140.69	88	-61
QP427	100	189669.2	8438510	151.05	92	-61
QP428	101	189621.3	8438513	152.93	92	-60
QP429	100	189571.9	8438355	149.93	88	-60
QP430	100	189570.9	8438410	155.08	88	-61
QP431	100	189512.8	8438357	149.58	90	-60
QP432	100	189268.8	8437997	140.04	94	-60
QP433	103	189232.7	8437956	132.27	90	-60
QP434	100	189469.8	8438358	148.27	98	-60
QP435	100	189611.9	8438374	165.57	92	-60.5
QP436	100	189668	8438355	185.9	92	-60
QP437	100	189627.3	8438415	169.93	96	-60

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP438	100	189469.1	8438464	142.8	92	-61
QP439	100	189619.9	8438462	162.12	88	-60
QP440	100	189670.5	8438458	162.02	91	-60.5
QP441	100	189717.5	8438505	161.56	94	-61
QP442	100	189764.6	8438508	159.96	94	-61
QP443	104	189765.9	8438559	169.01	90	-60
QP444	100	189819.2	8438807	154.21	90	-60
QP445	100	189821	8438857	151.71	90	-60
QP446	100	189763.7	8438858	153.2	90	-60
QP447	100	189069.6	8437513	139.26	92	-61
QP448	100	189119.4	8437511	138.99	94	-61.5
QP449	104	189169.1	8437510	137.85	84	-61.5
QP450	100	189168.9	8437885	132.73	90	-60
QP451	100	189219.1	8437884	136.28	90	-60
QP452	108	189167.5	8437834	132.22	90	-60
QP453	100	188871.5	8438105	135.2	94	-61
QP454	100	188965.1	8438097	134.2	88	-61
QP455	100	189520.1	8438462	145.54	92	-61
QP456	100	189533.3	8438418	150.3	88	-60
QP457	100	189569.6	8438462	153.3	88	-60
QP458	100	189617.9	8438557	147.31	88	-62
QP459	100	189669.9	8438557	148.28	90	-60
QP460	104	189716	8438559	159.23	90	-60
QP461	104	189815.9	8438708	164.1	90	-60
QP462	100	189715.7	8438749	160.05	92	-60
QP463	100	189325.6	8437607	132.6	94	-60.5
QP464	100	189267	8437604	133.65	92	-61
QP465	100	189214.3	8437606	134.45	92	-61
QP466	100	189163.6	8437604	138.44	92	-59
QP467	100	189140.6	8437606	139.5	94	-61
QP468	100	189401.4	8437881	146.41	92	-60.5
QP469	100	189219	8437833	138.45	90	-60
QP470	100	189266	8437830	147.58	84	-61
QP471	100	189314.1	8437830	150.54	92	-59
QP472	100	189415.8	8437828	160.13	94	-60
QP473	100	189370.6	8437828	151.24	94	-61
QP474	100	189410.5	8437726	141.98	94	-60
QP475	100	189360.2	8437729	139.21	92	-59.5
QP476	100	189313.9	8437728	146.89	92	-60
QP477	100	189262.1	8437733	136.04	98	-59
QP478	100	189217.6	8437733	132.78	94	-60
QP479	100	189166.2	8437736	131.83	92	-58
QP510	100	189219.5	8437511	136.08	94	-61
QP511	100	189319.7	8437511	133.65	94	-61
QP512	100	189269.2	8437512	134.61	92	-61.5
QP513	100	189370.4	8437513	132.93	94	-61.5
QP514	100	189417.9	8437507	132.41	90	-60

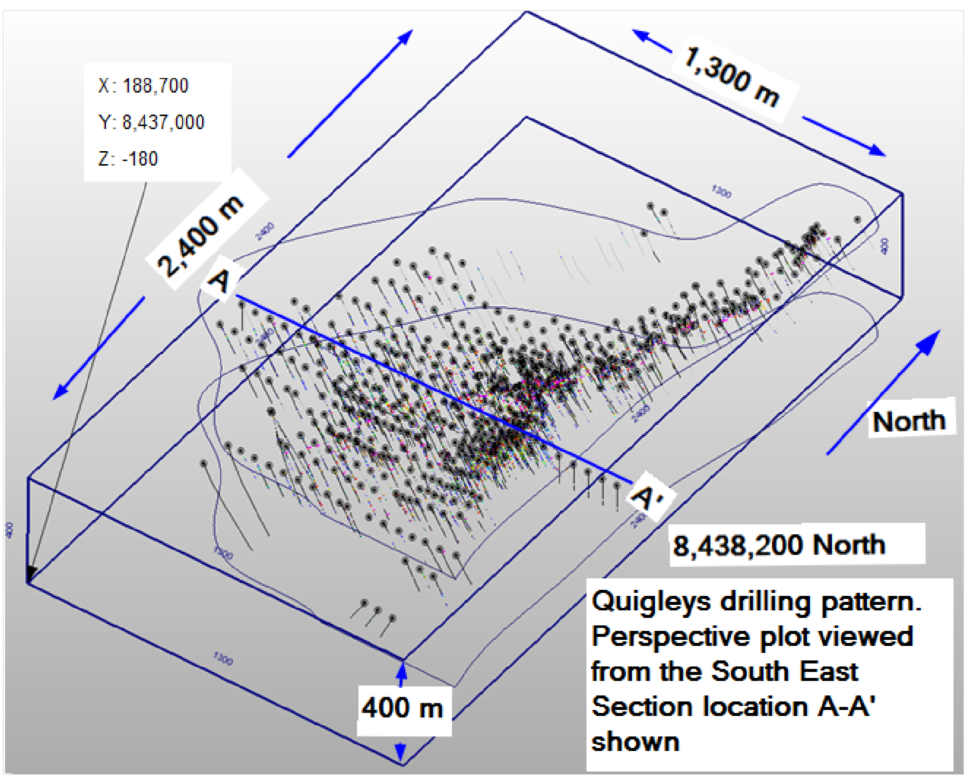
HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP515	100	189443.5	8437505	131.79	0	-90
QP516	100	189518.2	8437507	130.25	96	-61.5
QP517	100	189469.1	8437602	138.29	92	-61.5
QP518	81	189377.5	8437603	131.18	90	-60
QP519	100	189421.9	8437600	129.68	92	-60
QP520	105	189671.4	8438657	148.43	92	-60
QP521	100	189719.1	8438658	151.56	92	-57
QP522	100	189716.8	8438705	151.86	98	-59
QP523	100	189765.7	8438710	160.44	94	-61
QP524	100	189670.8	8438606	150.09	96	-59
QP525	100	189718.3	8438607	155.59	96	-62
QP526	100	189767.7	8438605	166.26	92	-59
QP527	100	189817.6	8438606	175.99	94	-59
QP528	100	189818.3	8438657	169	90	-59
QP529	100	189118.8	8437837	132.43	90	-61
QP530	100	189869	8438706	168.05	91	-59
QP531	100	189818.9	8438752	151.83	88	-61
QP532	100	189773.2	8438761	152.88	90	-61
QP533	101	189769.8	8438805	149.38	92	-61
QP534	100	189713.6	8438806	152.33	92	-60
QP535	100	189873.8	8438902	147.52	90	-59
QP536	100	189882	8438809	142.23	104	-59
QP537	100	189893.2	8438852	141.52	94	-58
QP538	100	189823.3	8438907	146.43	98	-59
QP539	100	189921	8438909	146.14	94	-60
QP540	100	189910.6	8439009	153.4	92	-58
QP541	100	189866.7	8439003	149.15	98	-58
QP542	100	189813.9	8439004	144.61	92	-60
QP543	100	189813.6	8439108	142.22	90	-60
QP544	100	189314.2	8437965	139.7	98	-60
QP545	100	189363.1	8437957	142.81	96	-60
QP546	100	189864.7	8439108	145.4	92	-58
QP547	100	189914.3	8439108	145.07	94	-60
QP548	100	189915.5	8439208	139.34	92	-60
QP549	100	189866	8439206	140.59	86	-59
QP550	100	187121.5	8437020	173.24	0	-90
QP551	100	187067.9	8437024	177.06	0	-90
QP552	100	187167.7	8437224	175.01	0	-90
QP553	100	187116.5	8437224	167.15	0	-90
QP554	100	189215.8	8438608	164.4	0	-90
QP555	100	189167	8438607	150.8	90	-61
QP556	100	189170.2	8438512	149.9	92	-60
QP557	100	189219.3	8438509	146.18	92	-60
QP558	100	189367.1	8438805	154.97	91	-60
QP559	100	189413.3	8438807	158.06	92	-60
QP560	100	189310.3	8438701	156.12	94	-60
QP561	100	189360.3	8438701	154.97	92	-59.5

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP562	100	189366.5	8438904	148.38	90	-60
QP563	100	189417.6	8438904	150.63	94	-60.5
QP564	100	189465.2	8438906	148.65	98	-60
QP565	100	189913.4	8439306	142.35	88	-59
QP566	100	189463.8	8439002	153.88	90	-59
QP567	100	189415.5	8439003	151.37	86	-60
QP568	100	189163.3	8437672	131.49	92	-60
QP569	92	189212.8	8437670	130.5	94	-60
QP570	100	189264.8	8437668	131.46	90	-58
QP571	100	189312.4	8437666	131.16	92	-59
QP572	100	189363.8	8437664	132.78	94	-60
QP573	100	189412.9	8437662	136.21	92	-60
QP574	100	189462	8437661	140.38	92	-60
QP575	100	189391.8	8437760	139.34	88	-60
QP576	100	189339.8	8437762	140.56	94	-60
QP577	100	189294	8437765	143.64	90	-60
QP578	100	189241.4	8437767	147.3	90	-60
QP579	115	189347	8438054	165.74	92	-59
QP580	115	189297.4	8438052	154.07	92	-59
QP581	110	189241.8	8438056	145	90	-59
QP582	110	189392.1	8438054	152.82	88	-60
QP583	110	189454.9	8438055	154.77	90	-60
QP584	110	189344.8	8438155	152.69	90	-59
QP585	115	189389.4	8438155	157.5	88	-59
QP586	110	189440.4	8438156	147.58	90	-61
QP588	100	189244.4	8438154	150.45	94	-60
QP589	100	189037.7	8437705	133.37	70	-69
QP590	100	189043.9	8437699	133.17	114	-70
QP591	100	189034.8	8437944	138.14	94	-60
QP592	100	189036.6	8438049	132.75	94	-57
QP593	100	189090.2	8438155	141.37	84	-60
QP594	100	189277.3	8438255	139.83	88	-59
QP595	100	189318	8438256	151.87	88	-59
QP596	100	189821.3	8438561	180.8	86	-58
QP597	100	189811.4	8438517	182.51	86	-58
QP598	100	189814.8	8438454	159.82	88	-57
QP599	100	189809.3	8438394	183.85	90	-60
QP600	100	189814.1	8438352	195.2	90	-59
QP601	100	189814.9	8438302	199	94	-60
QP602	100	189773.4	8438265	197.4	94	-60
QP603	106	189713.6	8438207	200.22	90	-59
QP604	100	189707.3	8438163	195.82	90	-57
QP605	100	189470	8438173	150.17	50	-60
QP606	100	189465	8437944	140.62	0	-59
QP607	100	189415.6	8437957	139.79	0	-60
QP608	100	189362.7	8437931	138.6	2	-58
QP609	100	189312.8	8437940	137.84	4	-59

HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
QP610	100	189270.3	8437918	138.65	0	-60
QP611	100	189218.3	8437894	138.48	6	-59
QP612	100	189118.2	8437902	132.61	6	-57
QP613	60	189413.4	8437939	139	0	-60
QP614	59	189359	8437906	131.65	0	-60
QP615	60	189310.8	8437919	135.8	0	-60
QP616	60	189270.6	8437895	134.13	0	-60
QP617	60	189167.5	8437905	132.62	0	-60
QP618	100	189817.8	8438953	145.3	92	-59
QP619	100	189791.7	8438952	144.5	92	-60
QP620	100	189791.6	8439003	144.42	90	-60
QP621	100	189840.9	8439098	143.51	90	-59
QP622	100	189814.5	8439052	143.1	90	-59
QP623	100	189839.6	8439109	142.97	90	-59
QP624	100	189865.4	8439162	144.22	92	-58
QP625	100	189839.4	8439159	141.71	90	-60
QP626	100	189838.8	8439207	138.86	91	-58
QP627	100	189117.9	8437928	132.96	1	-59.5
QP628	100	189167.7	8437928	131.69	2	-59
QP629	100	189216.3	8437951	132.22	2	-59
QP630	100	189823.1	8438003	151.5	90	-60
QP631	100	189873.1	8438003	149	88	-59
QP632	100	189923.1	8438003	147.5	88	-60
QP633	100	189973.1	8438003	145	92	-61
QP634	100	190023.1	8438003	146.5	84	-60
QSP001	40.5	189616.1	8438077	152.9	70	-60
QSP002	20.5	189632.3	8438081	157.5	0	-90
QSP003	35.4	189620.3	8438048	149.2	0	-90
QSP004	30.5	189647.3	8438010	150.5	74	-60
QSP005	21	189651.1	8438048	157.1	0	-90
QSP006	40	189567.4	8438141	178.1	60	-60
QSP007	26	189632.2	8438129	200	0	-90
QSP009	50	189602.8	8438163	193.5	210	-80
QSP010	36	189608.8	8438153	200	0	-80
RGP001	50	189200	8437834	134.83	92	-55
RGP003	50	189254.2	8437833	144.8	92	-55
RGP004	50	189283	8437832	147.94	92	-55
RGP005	55	189311.2	8437831	149.81	92	-55
RGP006	55	189337.3	8437832	149.42	92	-55
RGP007	50	189362.4	8437831	150.86	92	-55
RGP008	50	189397.1	8437828	155.19	92	-55
RGP009	50	189424.9	8437826	162.04	92	-55
RGP010	60	189452.9	8437829	162.89	92	-55
RGP011	50	189213	8437742	132.67	92	-55
RGP012	61	189232.1	8437731	133.4	92	-55
RGP013	53	189260.8	8437730	136.33	92	-55
RGP014	55	189287.7	8437729	144.8	92	-55

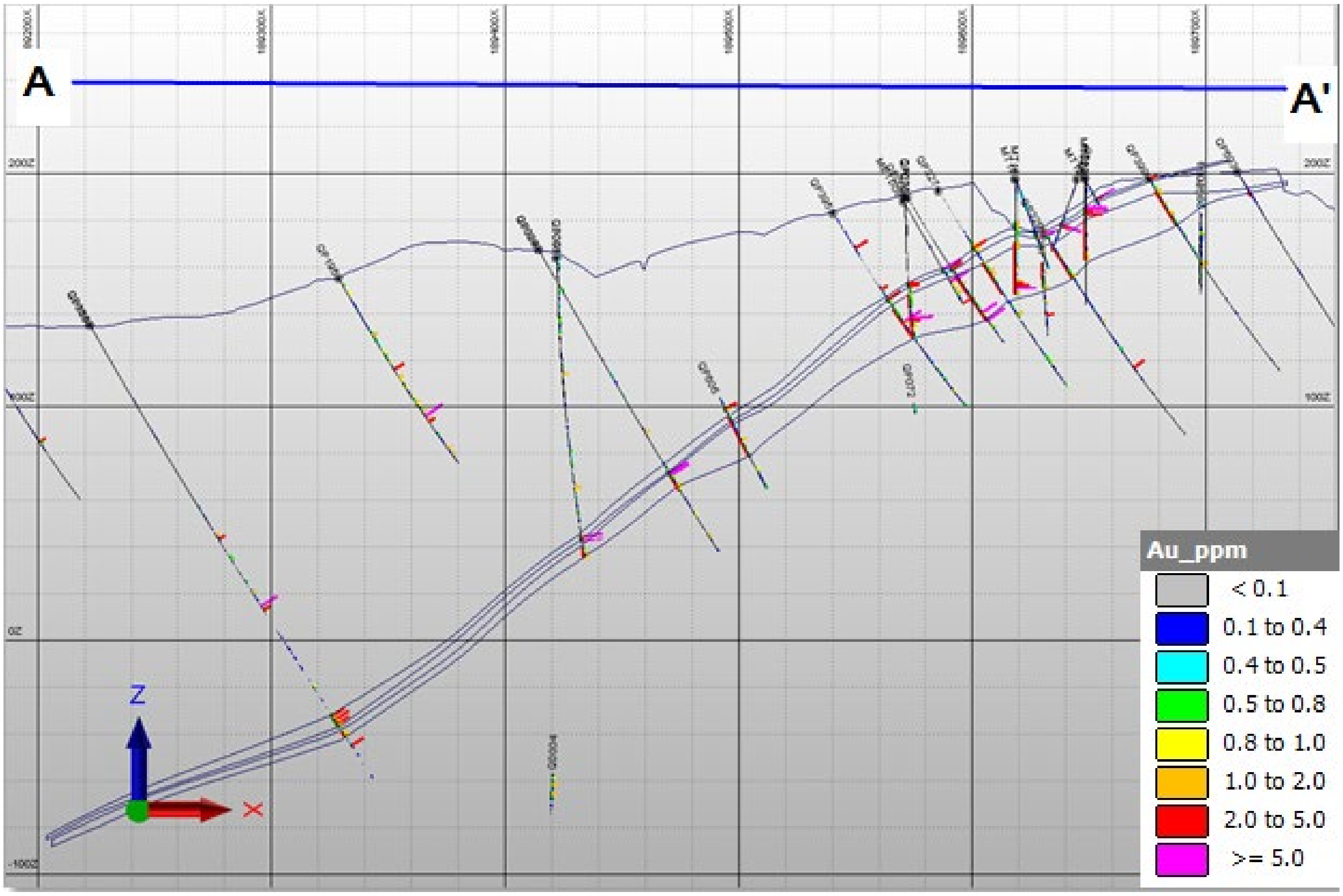
HOLE-ID	DEPTH	EASTING	NORTHING	ELEVATION	AZIMUTH	* DIP
RGP015	50	189317.4	8437729	147.06	90	-55
RGP016	55	189344.5	8437729	141.47	90	-55
RGP017	50	189369.9	8437729	137.05	90	-55
RGP018	49	189399.7	8437728	138.18	92	-55
RGP019	55	189427.6	8437727	146.3	92	-55
RGP020	60	189452.4	8437728	151.54	92	-55
RGP021	50	189618.3	8437880	165.09	92	-55
RGP022	80	189643.4	8437880	173.14	92	-55
RGP023	50	189670.5	8437880	183.23	92	-55
RGP024	50	189688	8437880	188.8	92	-55
RGP025	50	189609.2	8437774	162.83	92	-55
RGP026	50	189631.7	8437774	167.63	92	-55
RGP027	50	189654.1	8437776	169.88	92	-55
RGP028	60	189679.1	8437977	170.48	92	-55
RGP029	50	189700.1	8437983	170.97	92	-55
RGP030	50	189731.6	8437982	161.33	92	-55
RGP031	50	189751.7	8437962	162.57	92	-55
RGP032	50	189676.3	8437776	174.41	92	-55
VQ11-001	366	189125.7	8438153	150.52	90	-60
VQ11-002	368	189221.6	8438003	133.24	92.5	-60
VQ11-003	356	189331.8	8438153	149.72	93.53	-60

* Negative dip is downward



Source: Tetra Tech, August 2020

Figure 10-7: Isometric View of the Quigleys Drilling Pattern



Source: Tetra Tech, August 2020

Figure 10-8: Quigleys Cross-section A-A' (8,438,200 North +/- 10m window)

10.3 Drilling Procedures

The drilling procedures followed by the various companies were reviewed by Vista and were found to meet industry standard practices. The drilling procedures employed by Vista follow these general guidelines:

1) A geologic model is utilized to select a drilling target in both map location and depth.
2) The position of the drill collar is selected and surveyed with the initial azimuth and dip of the drill selected.
3) The method of drilling chosen (i.e., percussion, RC, or diamond) determines the type and quality of geologic information that can be recorded. A mix of methods is commonly used. Both percussion and RC produce ground up fragments of rock which tend to obscure detailed geologic description. Diamond drilling is used to obtain intact core samples which retain the geologic structure along with the spatial relationship of where mineralization occurs. Because percussion and RC drilling methods are cheaper than diamond drilling, many of the historical drillholes listed in Table 10-1 use a combination of RC at the top of the hole and diamond drilling as it passes through the mineralized target.
4) The location of the drillhole at depth is monitored by recording changes in azimuth and dip at depth.
5) Rock material obtained is described by a geologist during drilling. Material is collected for further geological description and assay analysis.
6) Core samples obtained during drilling are described and recorded on site by a geologist. Diamond core samples are placed in a special container for transport for a more detailed geologic description called "logging"

along the complete drillhole. The results along with photographs of this logging exercise are entered into the drillhole database.

7) The core is split with one half sent to an outside laboratory to be assayed (see Section 12 for a generalized description of the laboratory protocols) and the remaining core placed in a secured repository.

8) Assay results are entered into the drillhole database. All of the remaining material from the laboratory, referred to as "pulp", is also placed in a secured repository.

The geologic model is then updated with the results of the new drilling. Vista typically utilizes only the diamond drilling method.

10.4 Sampling

The sampling method and approach for all drillholes completed after 2012 are as follows:

- The drill core, upon removal from the core barrel, is placed into plastic core boxes.
- The plastic core boxes are transported to the sample preparation building.
- 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 test work.
- 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 this work was done under the supervision of a Vista geologist.

Please see Section 6—History of this Technical Report for information on historical drilling sampling.

The QP [Rex Clair Bryan, Ph.D., SME RM] is not 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.

10.5 Summary and Interpretation of Relevant Results

The results of drilling at Quigleys and Batman has been used to determine and update the gold resource estimates for the Batman and Quigleys deposits as described in this Technical Report. Vista's drilling discovered a larger Batman resource by probing deeper with diamond drilling averaging 550 meters in depth. Certain results for Batman are shown in five sectional views in Figure 10-2 through Figure 10-6. While there are random high-grade intercepts outside of the core, the majority of higher-grade mineralization at Batman resides in the core. Relevant intervals of mineralization at Quigleys are contained within blanket-like zones which are modeled with 3-D wireframes for resource estimation. These zones are shown in Figure 14-14. While there are random high-grade intercepts outside of the core, the majority of higher-grade mineralization resides within the defined zones. See also Section 6.2 for additional information on historical drilling and Section 14 for additional information on the drilling results including geologic modeling of the deposit based on the drilling and additional sectional views of the drillhole data and results.

These results may change when there is new drilling, new assays, a new interpretation of geology and economic or mining parameters. For Batman and the existing heap leach material, all of these conditions have remained static since the 2018 Technical Report. The resource estimate for the Quigleys deposit was upgraded to reflect a higher gold price. The following lists the relevant factors that have stayed the same or changed from the 2018 Technical Report:

- There has been no new drilling or assays incorporated into the block model or resource estimates. The last drilling at the Quigleys deposit was in 2011.
- The drilling at both the Batman deposit and the existing heap leach material for the purpose of resource definition was in 2017. Later drilling has been used for further metallurgical work and to explore for possible additional resources within the Batman deposit.
- There have been no corrections or additions to the assay data base.
- The geological models, 3D wire frames and surveyed locations of drilling at the Batman and Quigleys deposits are unchanged.
- There has been no change in the geostatistical interpretation of the continuity of mineralization and the assignment of resource classifications. The orientation of the mineralized zones of Batman and Quigleys differ; Batman is steeply dipping, while Quigleys is shallower.
- There has also been no change in the surveyed shape and estimated tonnage of the Tailings Pile.
- There has been no change in the assumed mining method, costs or metallurgical parameters that are used to classify estimates for all three deposits.
- The assumed price of gold has been updated for the Quigleys deposit from $1,200 to $1,300 per ounce. This change has altered the potential mineable material within a Whittle™ pit.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following sections describe the sample preparation, analyses, and security undertaken by Vista.

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 analytical and testing laboratories used by Vista for lab preparation, analyses, and check assays are listed in Table 11-1. The laboratories are separate commercial entities from Vista. Each laboratory meets all the required standards established by their industry associations and regulatory agencies. Except for Genalysis, all have presented their certification of being an accredited laboratory meeting all the international standards for testing and calibration.

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

Vista is completely independent of any analytical testing entity presented in this Technical Report. The QP [Rex Clair Bryan, Ph.D., SME RM] has determined that there is no apparent conflict of interest between Vista and its analytical laboratories.

Each of the Laboratories listed follow their own quality controls based on international standards. For example, ALS uses accredited methods specified by ISO/IEC 17025 in North America and Australia. The standards specify a recipe and set of quality control steps that the laboratory should follow:

1) How the sample should be coded to obscure its relationship to the drilling geometry.
2) How the received sample should be prepared.
3) What analytical steps be taken?
4) Given the required detection level and material analyzed, what instruments should be employed.
5) What internal quality controls should be done such as: periodic assaying of duplicate samples, the insertion of certified calibration samples; utilizing blanks; and including a required number of randomized samples

Vista as a gold project requires assays to be done with the industry standard fire assay. To get these fire assay results a generalized discussion of the steps are:

1) Core samples from drillholes are split at Mt Todd into two with one archived and the other sent to analytical laboratory. A more detailed discussion of how samples are prepared at Mt Todd is in Section 11.1.
2) At the lab the sample is pulverized into a powder, with a subsample taken for fire assay.
3) This subsample is then mixed with a fluxing agent. The remaining pulverized material is called a pulp archive, which can be used for within and between laboratory validations.
4) The chosen sample is then heated in a furnace where it melts and separates into a “button” which contains the gold. There are several methods to extract the gold from the button.
5) The most common method is by forming the button with lead as a collector. The lead oxidizes and is absorbed into a cupel leaving a gold bead.
6) Due to the relatively low concentration of gold at Mt Todd the lab must choose an analytical method able to detect at least 5ppb gold. The methods are generally by atomic absorption (AA) or inductively coupled plasma-mass spectrometry (ICP-MS).
7) The bead is dissolved in aqua regia or dissolved in hydrochloric acid and then analyzed by the selected instrument.
8) The resultant assay values are reported by an assay certificate which is electronically or physically sent to the staff at Mt Todd. The assay results are entered with the drilling database.

The QP [Rex Clair Bryan, Ph.D., SME RM] has reviewed results of the quality control procedures employed by Vista and has determined that they meet industry standards. For example, a comprehensive check of the quality of 12,365 assays in the database was undertaken by an outside auditor (Mine Development Associates, 2011). Records were selected from among those that relate to mineralization that is still in situ. These were divided into three subsets, to be checked by three individual checkers. An additional 1,812 records were spot-checked in greater detail by a fourth individual. After the checking was done, from the original 12,365 records, 95% were selected that had gold value in the database and a gold assay in a source document such as an assay certificate. Of the assay pairs, 8,549 were "historical" in the sense of dating prior to Vista's acquisition of the project and 3,262 assay pairs originate with Vista's work. For context, Mt Todd assay table as of August of 2011 contained 118,550 records, 26,579 of them originating from Vista's work.

Eight significant outliers were found with gold values in the database that differed from the source documents. Those eight were double-checked and were found to be real cases of the database containing data that differ from the source documents. Table 11-2 shows that most of the differences between the gold values in the database and those gleaned from the source documents are very small, although around economic cutoff grades the differences may well represent large percentages. More than 99% of the differences fall in the range -0.1 ppm Au to +0.1 ppm Au which is below the 0.4 ppm cutoff grade. However, a Mann-Whitney Test suggests that the differences between the two populations are not statistically different.

Table 11-2: Comparison of Assay Values between the Database and Source Documents (MDA, 2011)

Center of Cell Range in ppm Au (+/- 0.1 ppm Au)	Frequency	Percent	Cumulative Percent
-1.2	0	0.00	0.00
-1	0	0.00	0.00
-0.8	1	0.01	0.01
-0.6	0	0.00	0.01
-0.4	0	0.00	0.01
-0.2	3	0.04	0.05
0	8,539	99.88	99.93
0.2	5	0.06	0.99
0.4	0	0.00	99.99
0.6	0	0.00	99.99
0.8	0	0.00	99.99
1	0	0.00	99.99
1.2	1	0.01	100.00

Differences with no rounding or truncation of data

Table 11-3 and Table 11-4 show the comparison of the gold grade assays within the database and source documents. One of the three data sets checked contained 3,262 assays from drilling campaigns by Vista in 2007 and 2008. Checks of the Vista data against original sources were done by one individual, using essentially the same procedures as had been used for checking the historical assays. A summary table of the findings is presented, as Table 11-3. Of the 20 differences noted in Table 11-3, 4 differences are significant.

- A gold value of 0.005 ppm Au in the database compared to the correct gold value of 0.8 ppm Au.
- A gold value of 1.08 ppm Au in the database compared to the correct gold value of 0.01 ppm Au.

In addition, a separate detailed audit was done on 638 assays on Vista drillhole VB08-036. This audit shows that discrepancies within the database on the global resource estimate are not material.

Based on his review, the QP has determined that the historical and Vista assays in the Mt Todd database are useable for resource modeling.

Table 11-3: Summary of Comparisons of Historical Assays (MDA, 2011)

Historical Assays	Au in PPM		Differences, Source - Database in PPM
	Database	Source	
Average	0.79	0.79	0
Std Dev	1.48	1.48	0.01
Count	1171	1171	565
Max	33.44	33.45	0.255
Min	0.005	0.005	-0.29
Median	0.3	0.3	0
Differences > 0.01 ppm Au			20
Differences < 0.01 ppm Au			4

Table 11-4: Summary of Comparisons of Vista Assays (MDA, 2011)

Vista Assays	Au in PPM		Differences, Source - Database in PPM
	Database	Source	
Average	0.79	0.78	0
Std Dev	1.89	1.89	0.02
Count	3262	3262	12
Max	55.37	55.37	0.79
Min	0.005	0.005	-1.07
Median	0.26	0.26	0
Differences > 0.01 ppm Au			3
Differences < 0.01 ppm Au			6

The QP [Rex Clair Bryan, Ph.D., SME RM] has reviewed and accepted the quality assurance protocols are employed by Vista for its drilling, sample preparation and assays. Vista requires periodic rechecking of assays both within and between laboratories. As an example, prior to the 2011 drilling campaign, most 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 pulps or rejects 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. Figure 11-1 shows the results of check assays on one in every 20 pulps or rejects that were completed by NT Environmental Laboratories. No bias in assays was found with a slope of 0.992 and a correlation of 99%. There was only one significant difference that was detected from a total of 2,948 comparisons. Figure 11-2 shows a comparison of original 78 pulp assays between the NAL and ALS laboratories. The assay values showed no bias with a slope of 0.99 and a correlation of 99%.

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

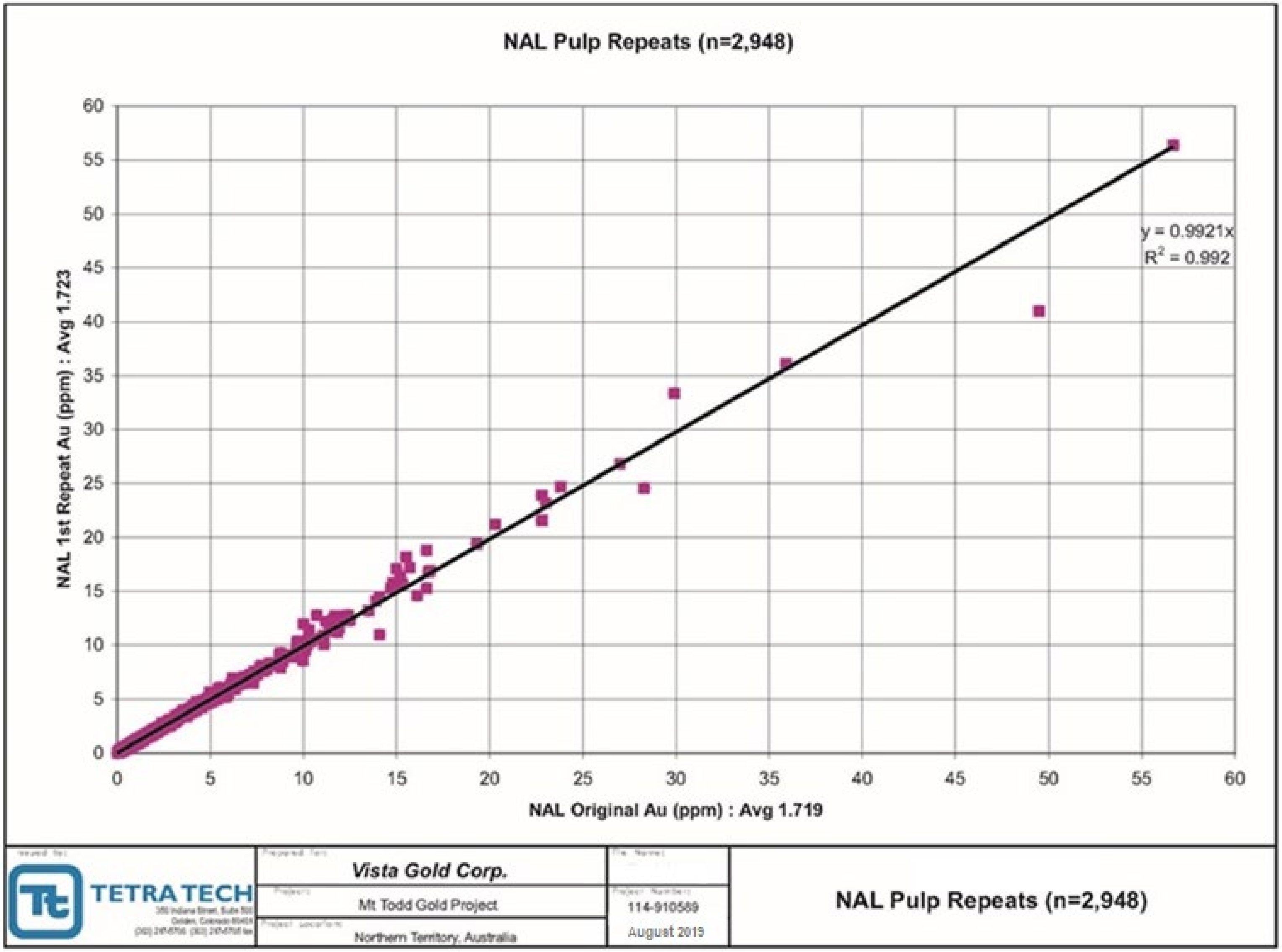


Figure 11-1: NAL Pulp Repeats

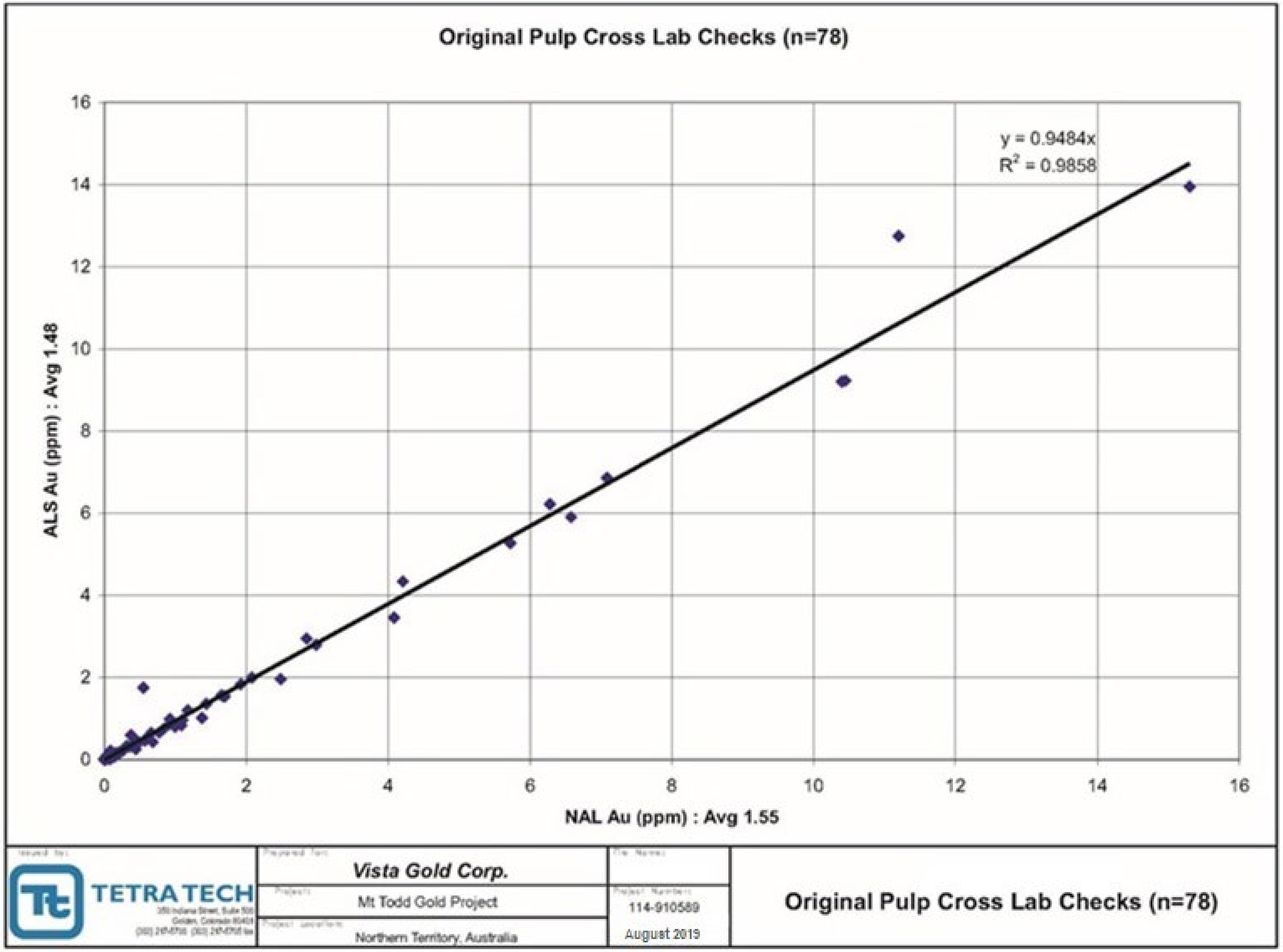


Figure 11-2: Original Pulp Cross Lab Checks

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 geology 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 [Rex Clair Bryan, Ph.D., SME RM] is satisfied with the adequacy of sample preparation 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 historical drilling and increase the resources of the Batman deposit. The QP is also satisfied that sample security measures meet industry standards. 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. DATA VERIFICATION

12.1 Drill Core and Geologic Logs

Multiple site visits were performed by the QP [Rex Clair Bryan, Ph.D., SME RM] for the resource estimation portion of this Technical Report. During those visits, the QP 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 responsible 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 verify the historical assay results and establish procedures for subsequent analytical work was completed. This program consisted of two components: re-assaying of a portion of the historical 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 fine material 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 details 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 plot, the correlation coefficient was 0.997 for the re-splits of original assays.

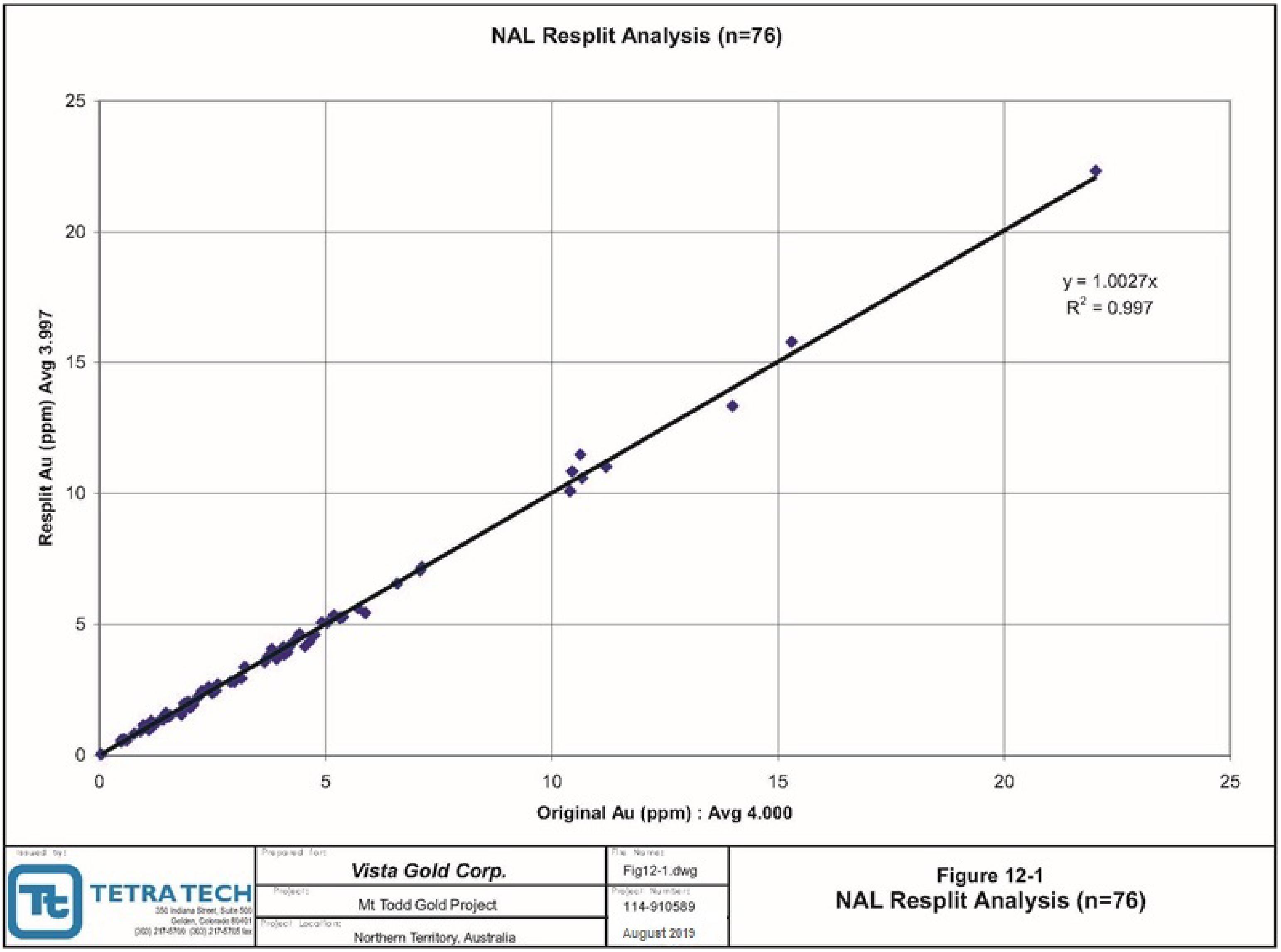


Figure 12-1: NAL Resplit Analyses

12.3.1 Latest Drilling Data Verification

For the March 2018 resource estimate, a detailed data verification procedure was undertaken by the QP [Rex Clair Bryan, Ph.D., SME RM] 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, the QP verified that the latest four metallurgical drillholes (VB17-004, VB18-001, -002, and -003) followed Vista's drilling and sampling protocols.

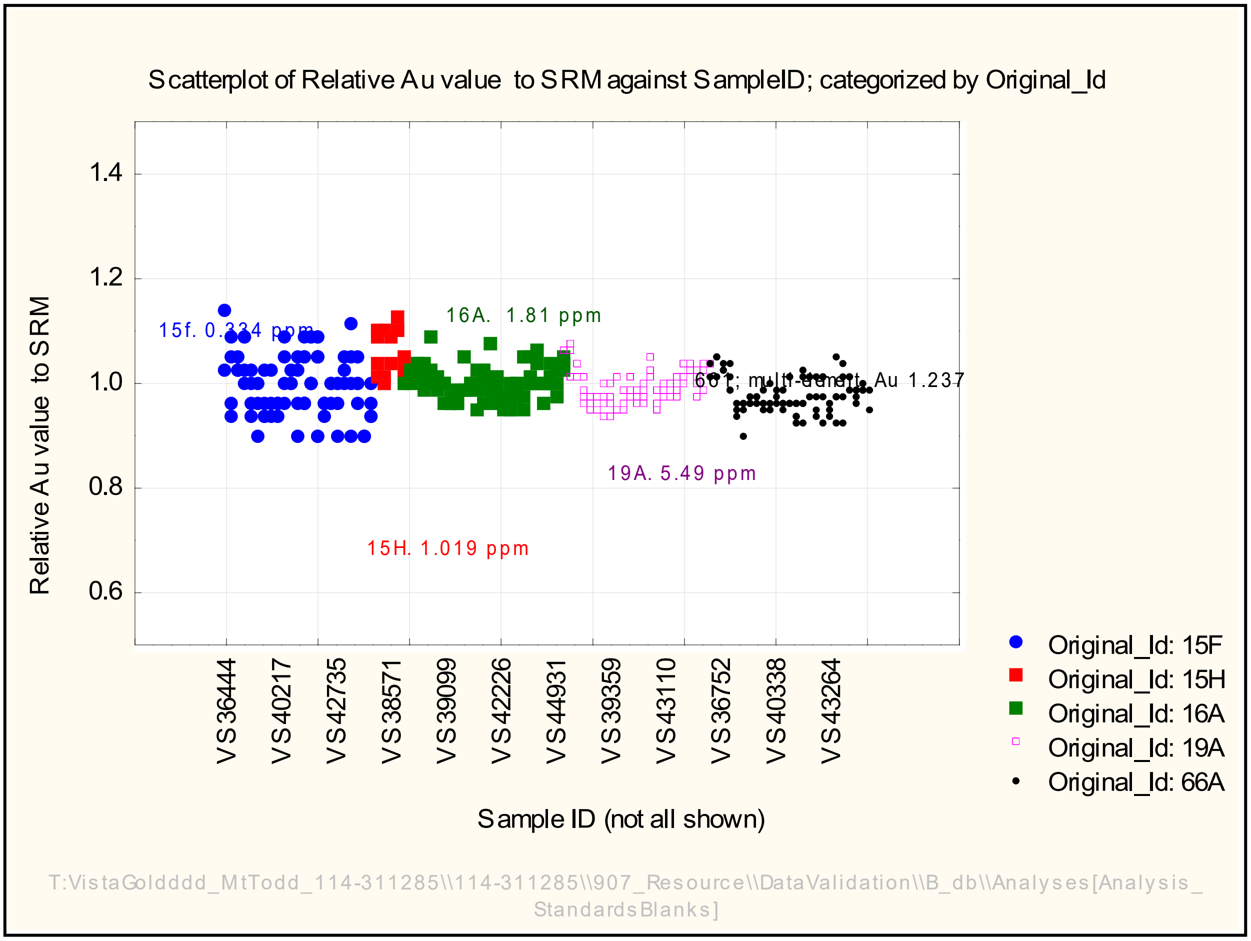
For the 13 drillholes from the 2012 exploration program, there were 7,601 intervals assayed. For the nine drillholes from the combined 2015-2017 exploration programs, there were 1,770 intervals. In addition to Au and other precious metals, most intervals had multi-element and environmental test results as well. Like 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 12-2. 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 12-2 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 ppm. The maximum value was 0.11 ppm. Again, no drift was noted in the data over time.

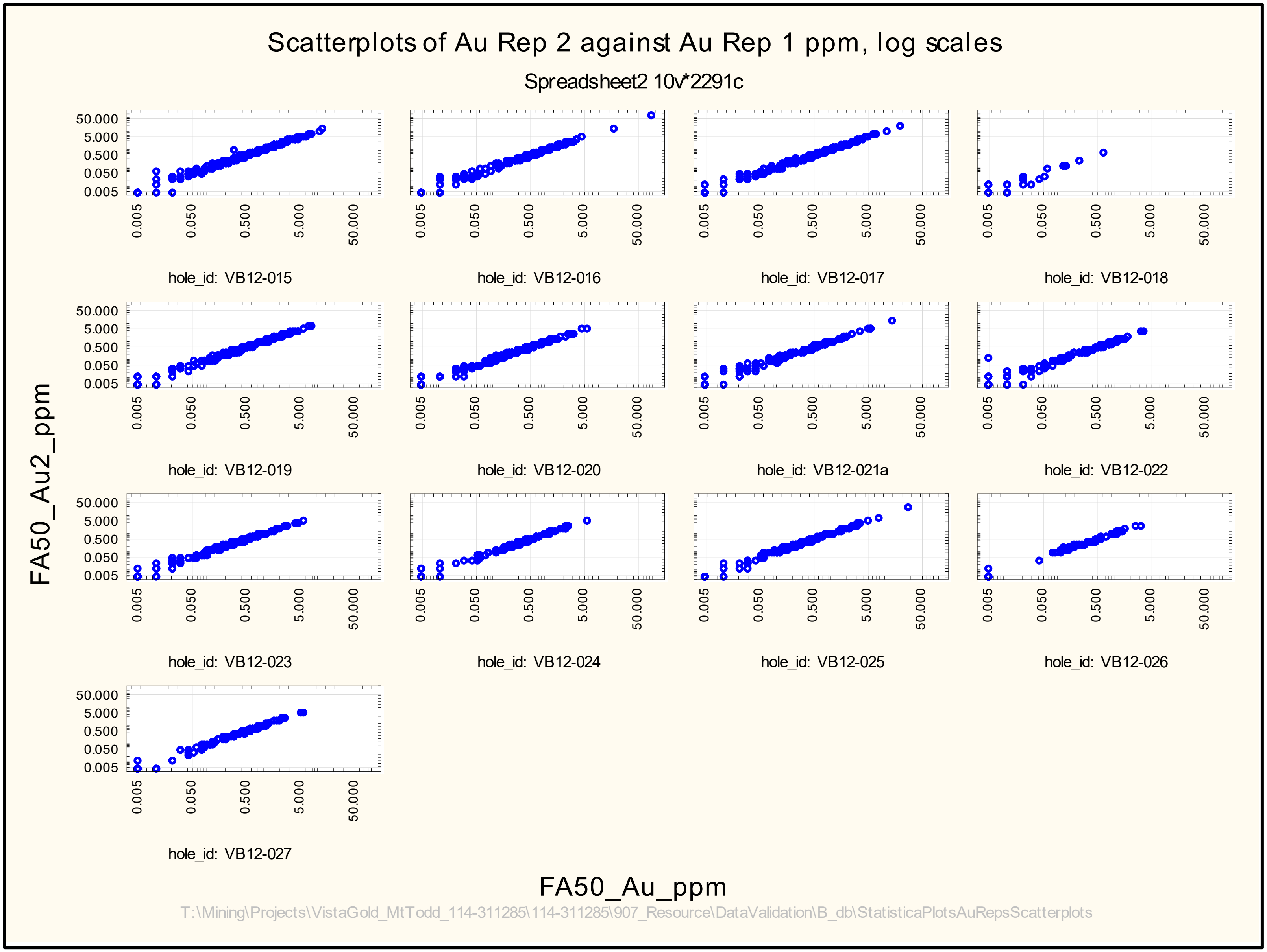
Because the current drilling campaign uses core, a regular program of field duplicates is not instituted at this time, but approximately 30% of samples have at least one replicate assay performed and an additional 3% of these have a second replicate assay. Replicates are taken from pulp when the primary 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 12-3 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 QP 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, August 2017

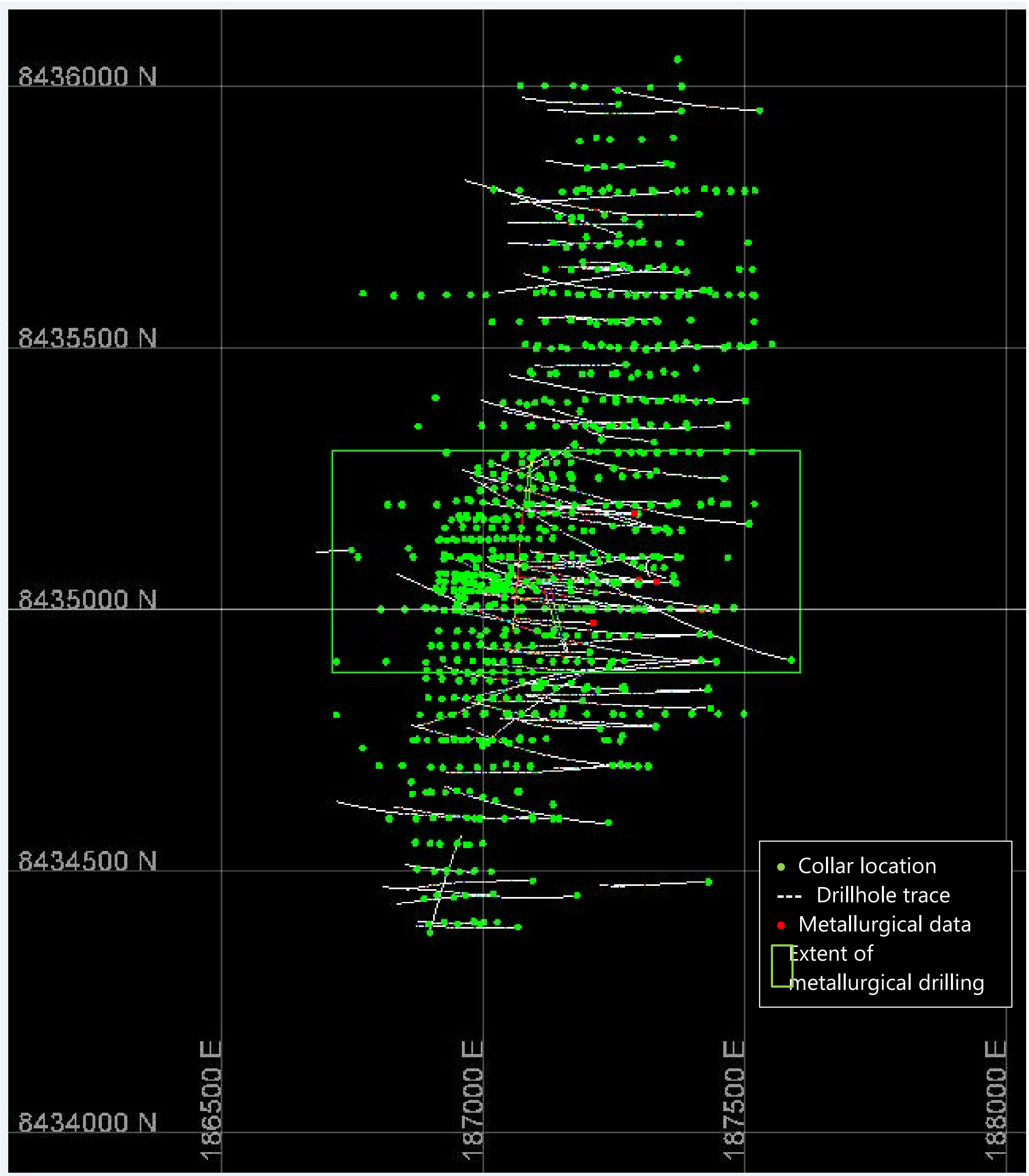
Figure 12-2: Scatterplot of Relative Au Value to Certified Standard Reference Material Value



Source: Tetra Tech, August 2017

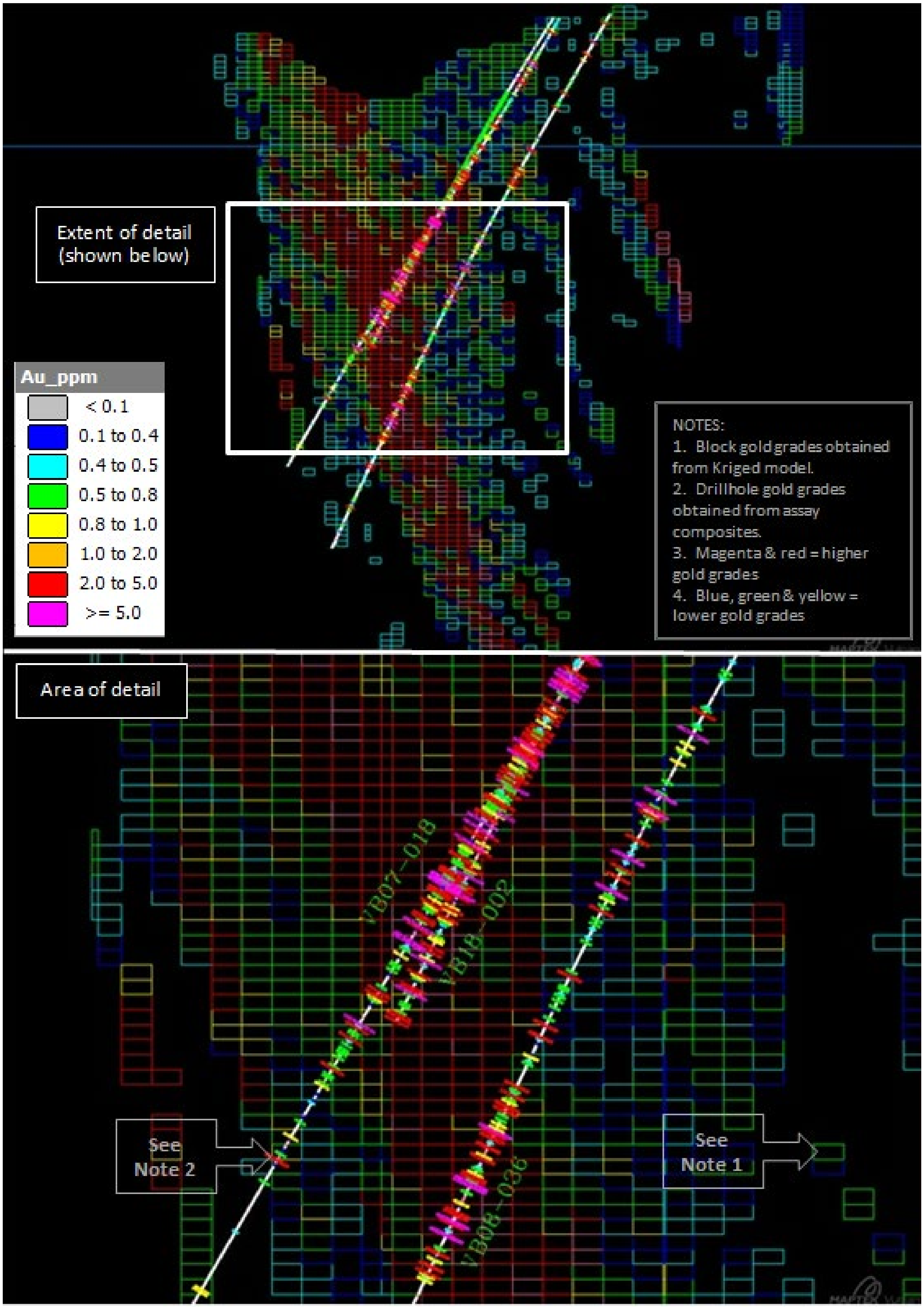
Figure 12-3: Scatterplots (Log Scale) of Replicates by Drillhole

Figure 12-4 shows the location of a metallurgical drilling campaign that produced three cross sections containing new holes that almost twin previous ones. Figure 12-5 shows one of the twinned sections. It is important to note that the twinned holes target the higher-grade ore of the Batman deposit. It is the opinion of the QP that this serendipitous twinning exercise helps confirms that the geologic modeling, drilling position, assay sampling and block modeling meet industry standards.



Source: Tetra Tech, August 2020

Figure 12-4: Location of Metallurgical Drillholes



Source: Tetra Tech, 2021

Figure 12-5: Two Views of VB07-013, VB18-002 and VB08-036 in Cross-sections

13. 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 Historical Metallurgical Test Programs

The Mt Todd deposit is a large low-grade gold deposit. The average grade of the gold mineralization is less than 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 Bond Ball Mill Work Index (BWi) measurements in the range of 23 to 30 kWh/t, and an average SWi of 25.2 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 using 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 was 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 test work 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

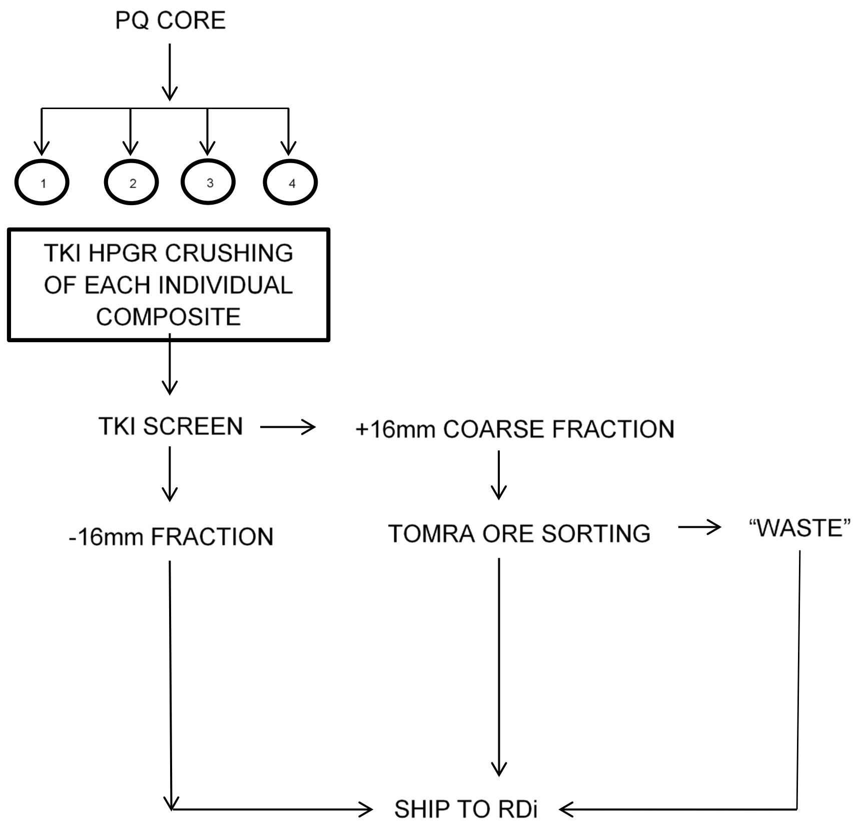
During January and February 2017 Vista completed drilling and logging of approximately 1,700 m of PQ (3.75 in diameter) core to obtain four 5-tonne bulk samples of ore representing different parts of the deposit. These composites were selected to represent both near-term and longer-term mining and are 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 0.84 g/t Au compared to 0.77 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 ¾ 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. This was subsequently evaluated in detail and is discussed in Section 13.4—2018/2019 Metallurgical Test Work.

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
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
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
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

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
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-SO2 process successfully reduced CNWAD 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 m2/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.

Table 13-16: Metallurgical Drilling Intercept Angle to Mineralized Vein

BHID	Average Intercept Angle
VB17-001	10 degrees
VB17-003	20 degrees
VB17-002	40 degrees
VB16-002	40 degrees

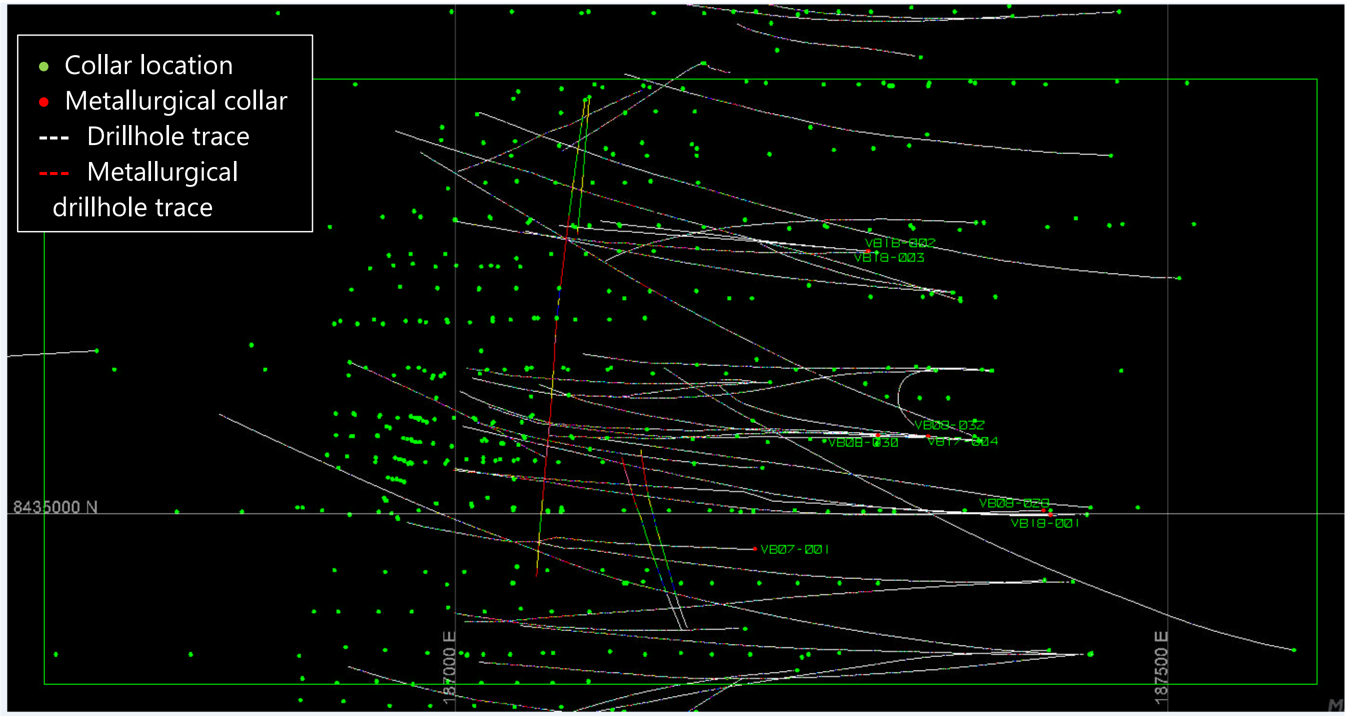
Vista initiated an additional drilling program to address two specific questions, namely, whether the geological model is correct or not and how would higher grade material perform in the proposed process flowsheet.

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.

Figure 13-2 shows the location of the metallurgical drillholes. Figure 13-3 illustrates the relationship of the resource model estimated grades nearest existing drill hole intercept grades and the grades of the metallurgical hole VB17-004 to the two proximal drillholes VB08-030 and VB08-032. The average grade of the composites and kriged blocks are shown as the drillholes transit through a high grade zone.

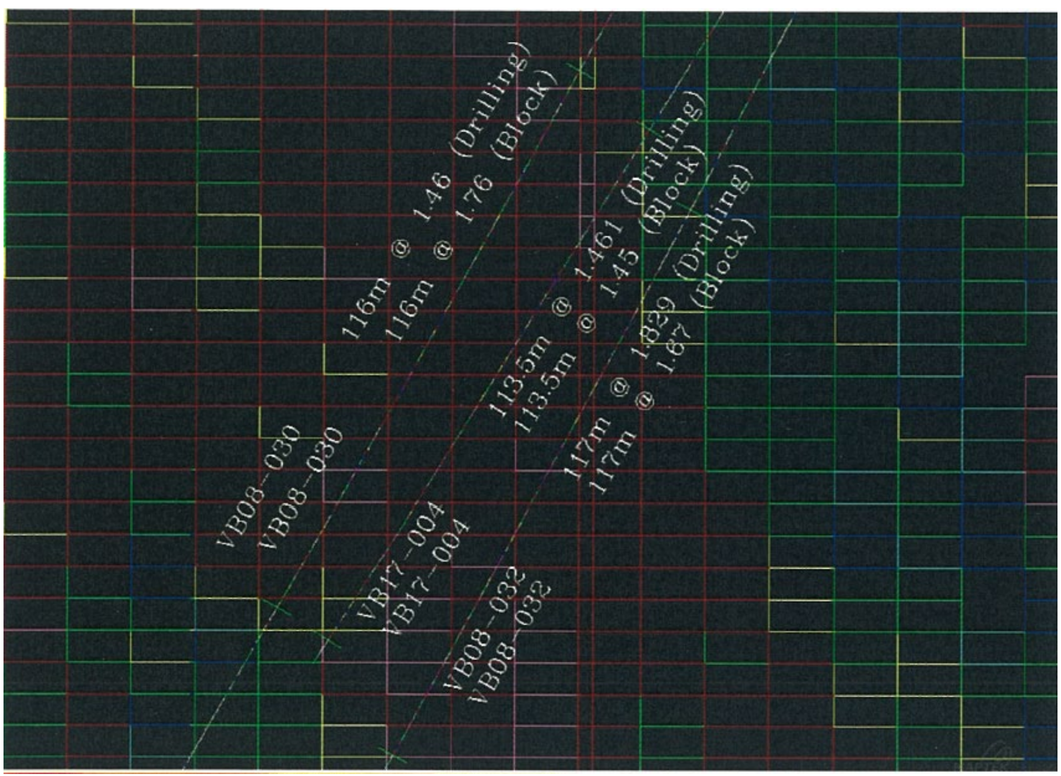
Table 13-17: Vista Drillholes and their Metallurgical Twins

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**



Source: Tetra Tech, 2020

Figure 13-2: Drillhole Trace of VB08-030, VB17-001 and VB08-012



Source: Tetra Tech, 2020

Figure 13-3: Drill hole trace of drillholes VB08-030, VB17-001 and VB08-012

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 tonnes of composite sample designated "Big Yellow" and assaying 1.7 g/t Au.
- 2.5 tonnes of composite sample designated "Big Blue" and assaying 1.4 g/t Au.
- 1.0 tonne 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 13-18. The specific throughput rate was ±300 ts/hm^3.

Table 13-18: 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 the 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 13-19. 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 13-19: 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

Under the direction of the QP [Deepak Malhotra, Ph.D., SME RM], 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, Kentucky with the objective of evaluating the STEINERT combined sensor sorter (KSS FLI XT) for separating ore and waste. The test results, summarized in Table 13-20, were similar to those obtained at Tomra test facility in 2017.

Table 13-20: 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															
Product 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 13-21, indicate the following:

- Head analyses of some of the composite samples 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 13-21: 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 13-22. 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 value selected for mill design.

Table 13-22: 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 historical 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 the 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 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 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 13-23 to Table 13-28. 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 13-23: 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**
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 13-24: 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

Test#	P_{80} Particle Size (µm)	% Recovery (Au)	Calc. Head Grade (g Au/t)	Residue Grade (g Au/t)
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 13-25: Leach Results for Feed Grade of 0.8 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
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 13-26: Leach Results for Feed Grade of 0.6 to 0.8 g/t Au

Test#	Particle Size (P_{80} µm)	% 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**

Test#	Particle Size (P_{80} µm)	% Recovery (Au)	Calc. Head Grade (g Au /t)	Residue Grade (g Au /t)
BR103	39	90.4	0.64	0.06
BRIO0	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
BRIII	15	91.0	0.61	0.06
BRII0	15	92.0	0.60	0.05
BR112	15	90.9	0.60	0.06
<53 micron average values		90.9		0.06

Table 13-27: 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 13-28: 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 fines 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 13-29. This data can be used by the process engineer to predict gold extraction in the plant.

Table 13-29: 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.

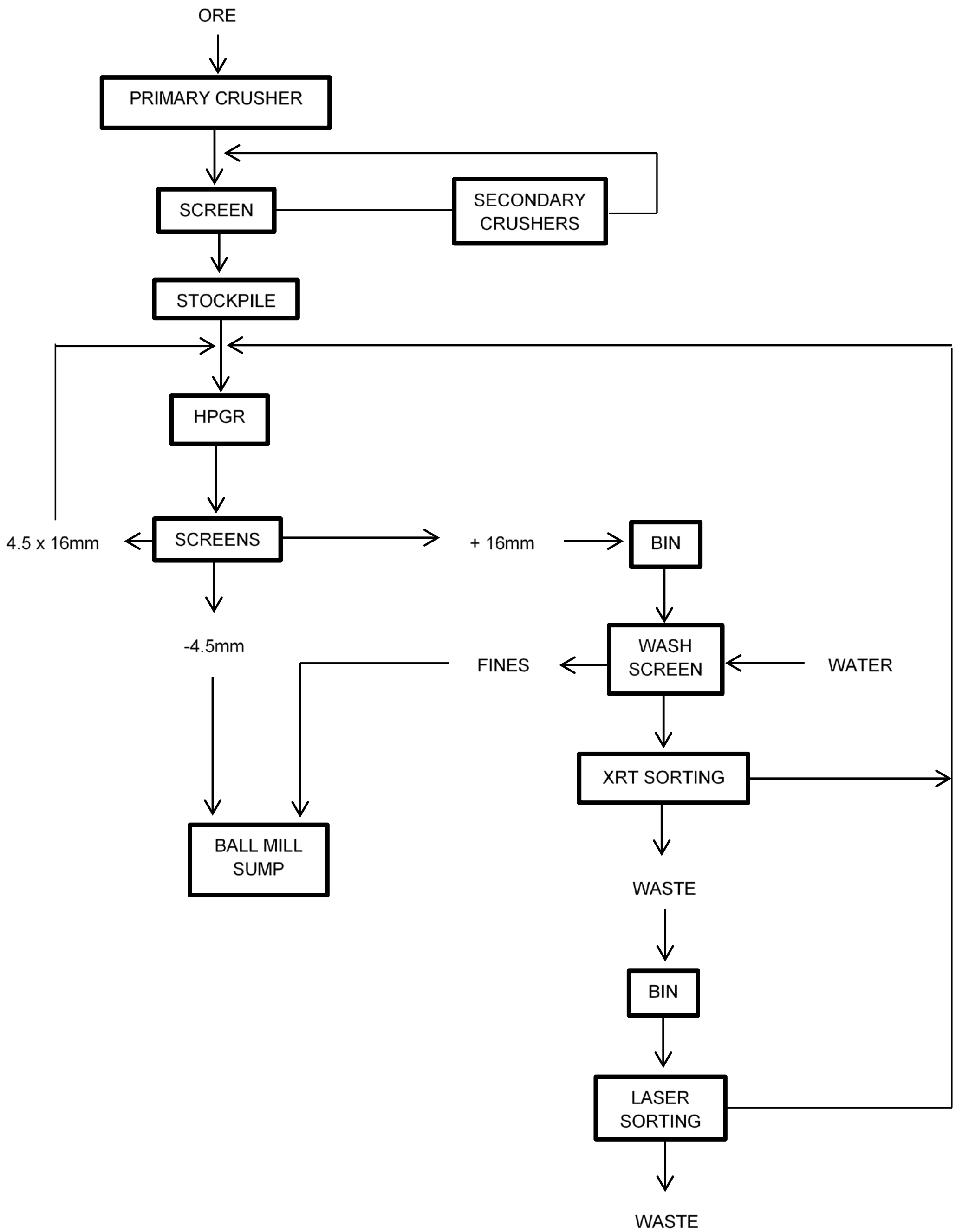
The forms of cyanide before and after destruction for the test is given in Table 13-30. The test results indicate that the air/SO_2 process will reduce the cyanide to below environmentally acceptable levels.

Table 13-30: Cyanide Destruction Test Results

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

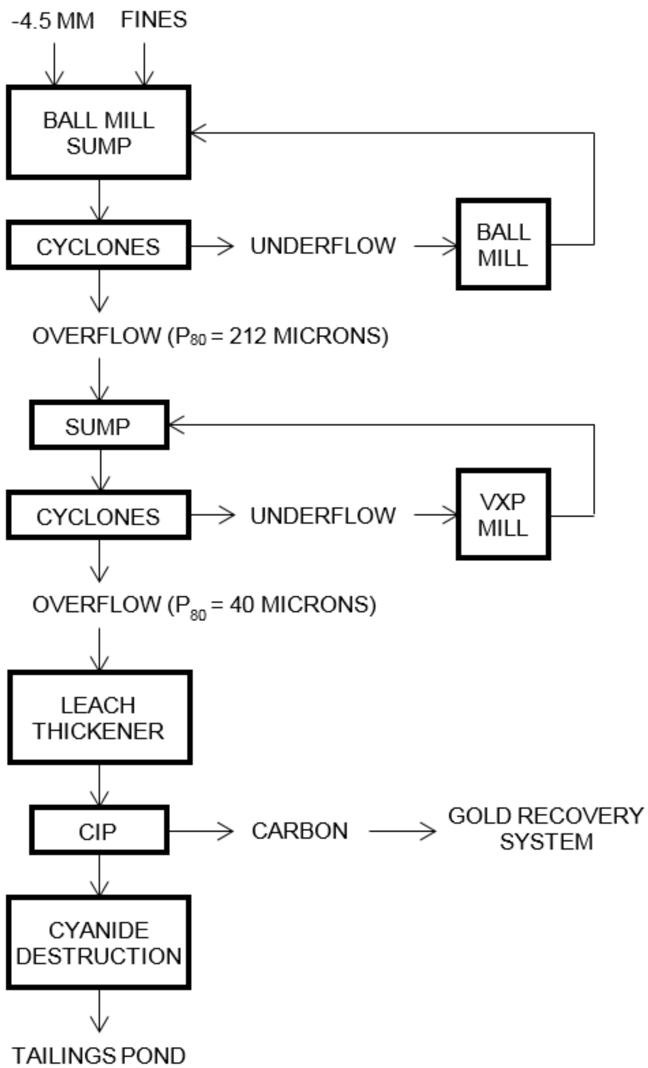
13.5 Process Flowsheet

The present FS is based on the flowsheet provided in the simplified Figure 13-4 and Figure 13-5. The process flowsheet is provided in greater detail in Figure 17-1 of this report.



Source: Resource Development Inc., September 2019

Figure 13-4: Conceptual Process Flowsheet for Mt Todd Ore (1/2)



Source: Resource Development Inc., September 2019

Figure 13-5: Conceptual Process Flowsheet for Mt Todd Ore (2/2)

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

The following sections summarize the thought process, procedures, and results of the QP's [Rex Clair Bryan, Ph.D., SME RM] independent estimate of the contained gold resources of the:

1) Batman Deposit
2) Quigleys Deposit
3) 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. The QP confirms that the Batman mineral resource presented in this report and the January 2018 PFS are the same and unchanged. No additional mineral resource drilling occurred in the Batman deposit or for the heap leach pad between the January 2018 PFS and the effective date of this report. While diamond core drilling did occur during this time, this drilling was for metallurgical samples and not applicable to mineral resource or reserves definition. The Quigleys resource has been updated to reflect a gold price of $1,300/oz. The selection of using a 0.4 g/t cutoff for gold is detailed in Section 15.1.1—Economic Parameters.

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.

> **<u>Cautionary statements regarding mineral resource estimates</u>:**
>
> 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 based on 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.

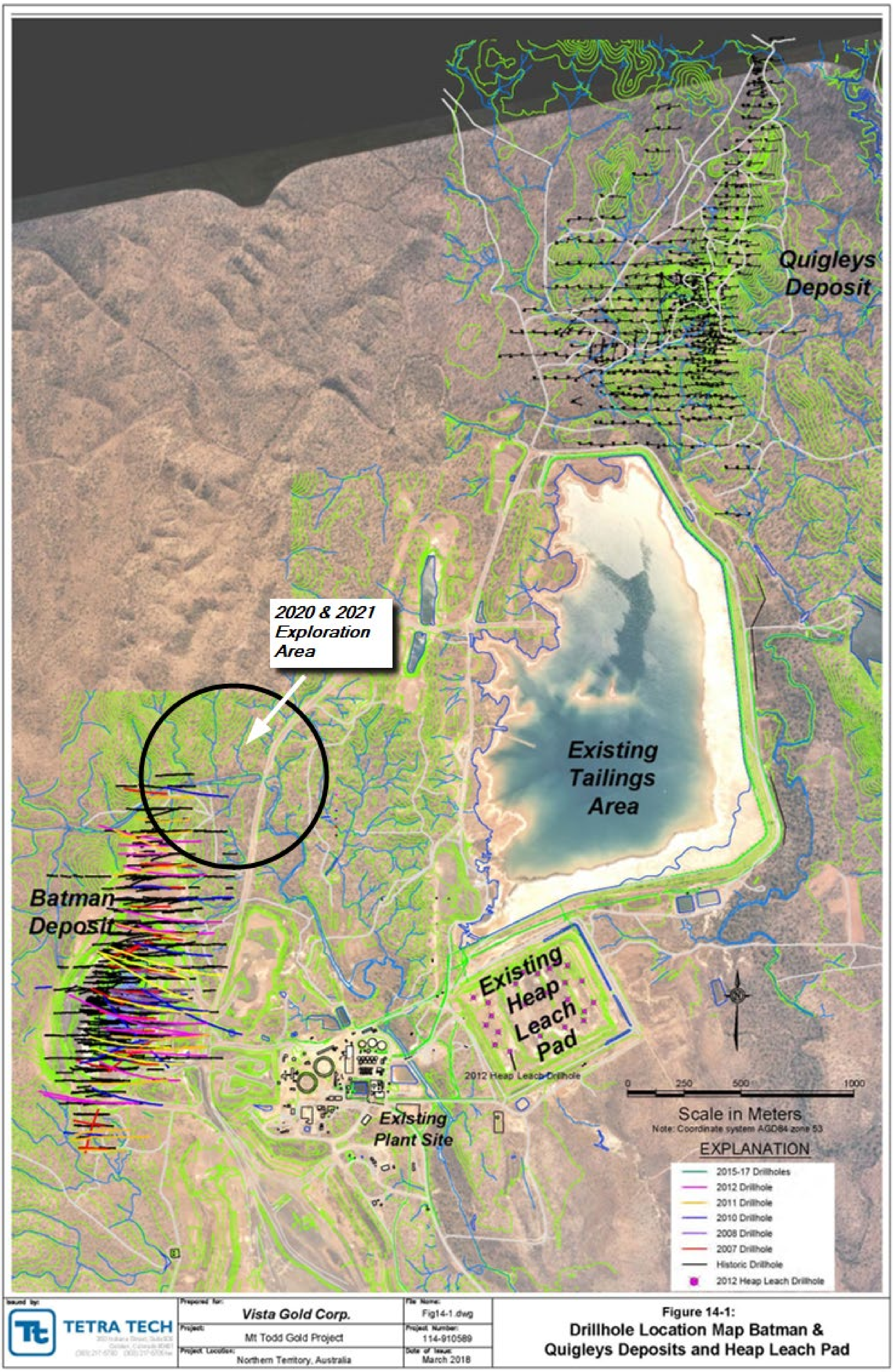


Figure 14-1: Drillhole Location Map Batman & Quigleys Deposits and Heap Leach Pad

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

	BATMAN DEPOSIT			HEAP LEACH PAD			QUIGLEYS DEPOSIT		
	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	-	-	-	594	1.15	22
Indicated (I)	200, 112	0.80	5,169	13,354	0.54	232	7,301	1.11	260
Measured & Indicated	**277,837**	**0.82**	**7,360**	**13,354**	**0.54**	**232**	**7,895**	**1.11**	**282**
Inferred (F)	61,323	0.72	1,421	-	-	-	3,981	1.46	187

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, G&A/Year 8,201 K US4, Au Recovery, Sulfide 85%, Transition 80%, Oxide 80%, 0.2g Au/t minimum for resource shell.
4) Quigleys: Resources constrained within a US$1,300/oz gold Whittle™ pit shell. Pit parameters: Mining cost US$1.90/tonne, Processing Cost US$9.779/tonne processed, Royalty 1% GPR, Gold Recovery Sulfide, 82.0% and Ox/Trans 78.0%, water treatment US$0.09/tonne, Tailings US$0.985/tonne
5) Differences in the table due to rounding are not considered material. Differences between Batman and Quigleys mining and metallurgical parameters are due to their individual geologic and engineering characteristics.
6) Rex Bryan of Tetra Tech is the QP responsible for the Statement of Mineral Resources for the Batman, Heap Leach Pad and Quigleys deposits.
7) Thomas Dyer of RESPEC is the QP responsible for developing the resource Whittle™ pit shell for the Batman Deposit.
8) The effective date of the mineral resource estimate is December 31, 2023.
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

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 probable.

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 reduced primarily inferred resources at depth. A Whittle™ pit further constrained the reported 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.

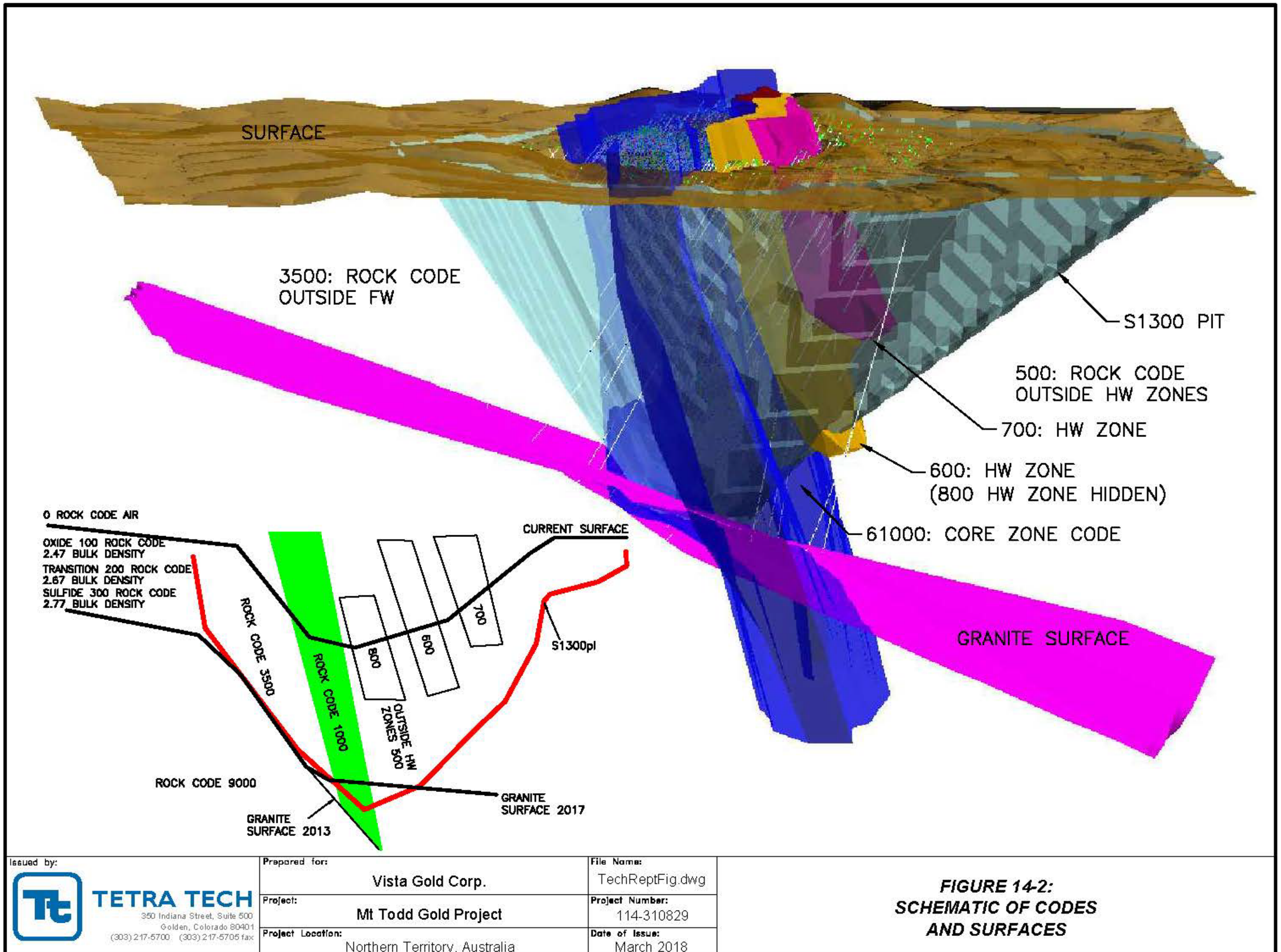


Figure 14-2: Schematic of Codes and Surface Designations (Looking North)

Figure 14-3 shows a sectional view of the drillhole data at Batman. The direction of Batman Deposit drilling is dipping at approximately 45-degrees to the west.

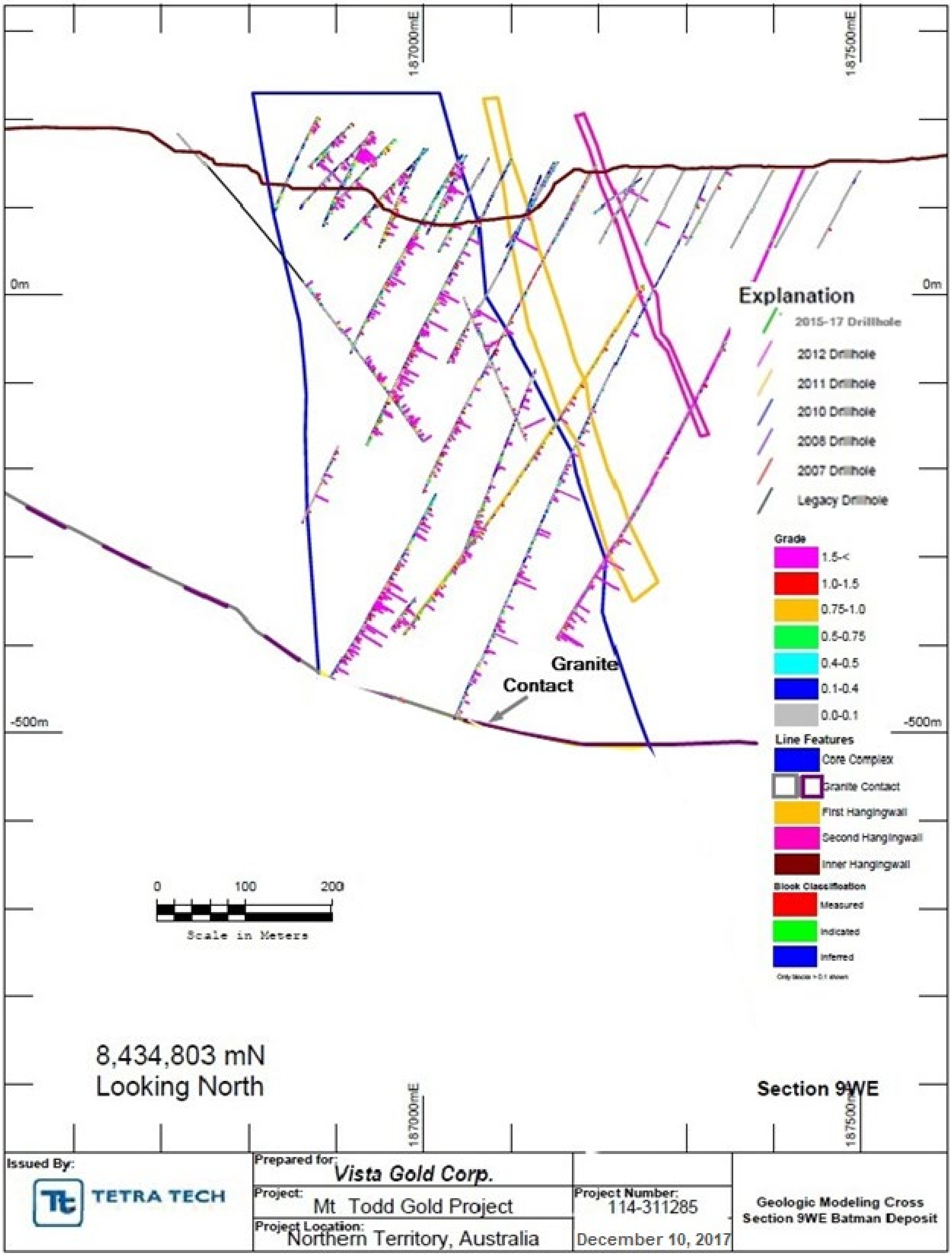


Figure 14-3: Sectional View of Drillhole Data 8,434,803 mN (Looking North)

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° with a dip of 40° to 60° to the southeast. The majority of drilling slants at a dip of approximately 65° with an azimuth of 270°.

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.

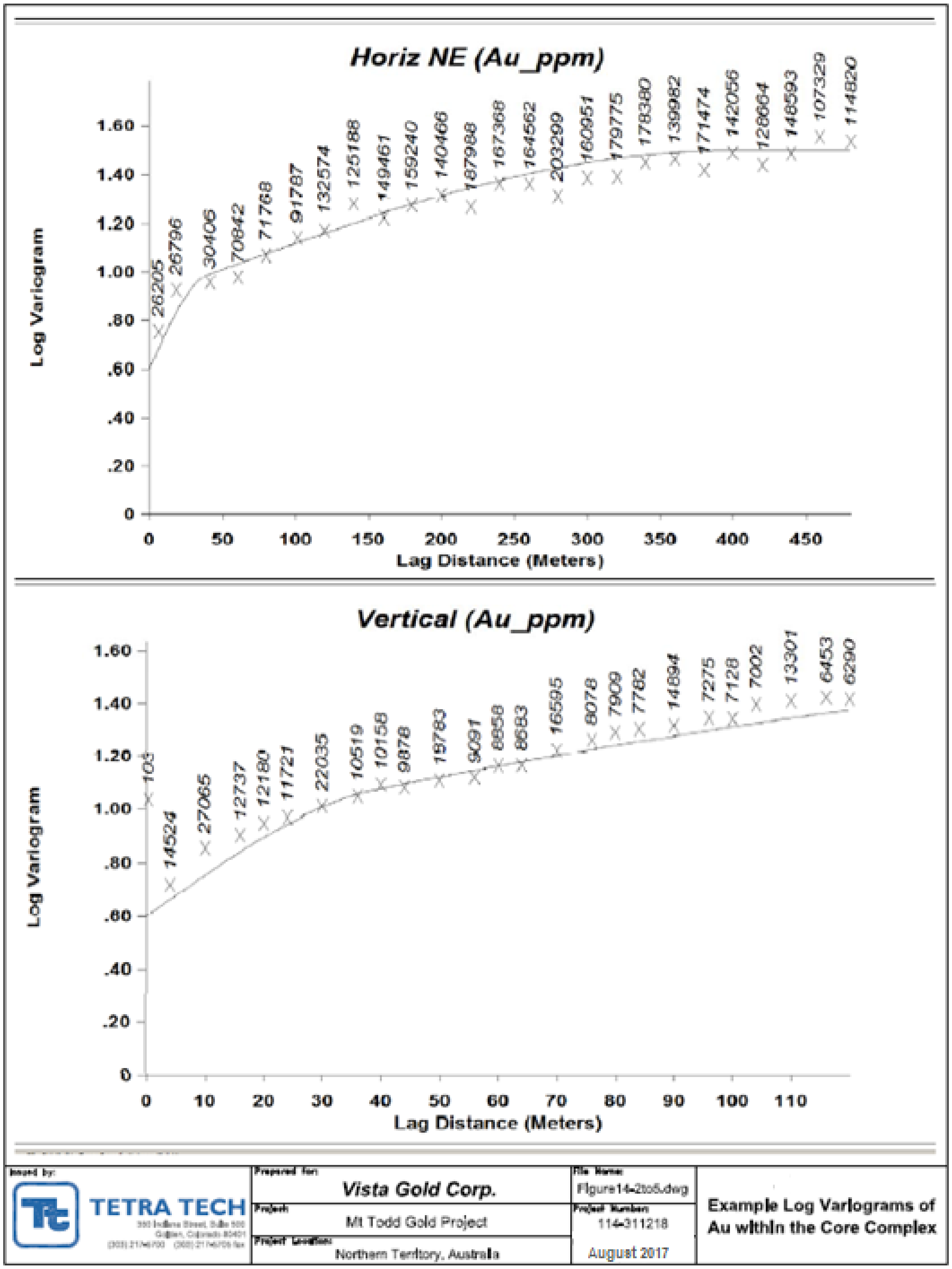
NE-SW trending faults and joint sets crosscut bedding. Only minor movement has been observed on these faults. Calcite veining is sometimes associated with these faults. These structures appear to be post mineralization.

Northerly trending quartz sulfide veins and joints striking at 0° to 20°, dipping to the east at 60°, 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 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, August 2017

Figure 14-4: 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 Primary Axis: 150m Nugget: 0.6 First Rotation (Azimuth: 110) Secondary Axis: 105m Sill 1: 0.3 Range 1: 40m Second Rotation (Dip: 80) Tertiary Axis: 60m Sill 2: 0.2 Range 2: 500m Third Rotation (Tilt: 0)

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	

Instruction (b) of the "illustration" requirements set out in Form 43-101F1 requires the inclusion of maps showing the location and sufficial outline of mineral resources and mineral reserves. Figure 14-5 through Figure 14-10 are a series of sections and plan views of the Batman mineral resource. A summary of the mineral resources is based on a 0.4 g Au/t cutoff and is shown in Table 1-2 and Table 14-1.

Figure 14-11 through Figure 14-15 show a plan view and sections of the mineral reserves in accordance to NI 43-101 guidelines. Economic Indicated Mineral Resources are within the ultimate pit calculated in Section 15 were classified as Probable Mineral Reserves. Measured Mineral resources were classified as Proven Mineral Reserves. A summary of the mining reserves is based on a 0.35 g Au /t is in Table 1-3 and Table 15-8. A more detailed discussion of the optimization of the mining pit and selection of the 0.35 g Au/g cutoff is Section 15—Mineral Reserves.

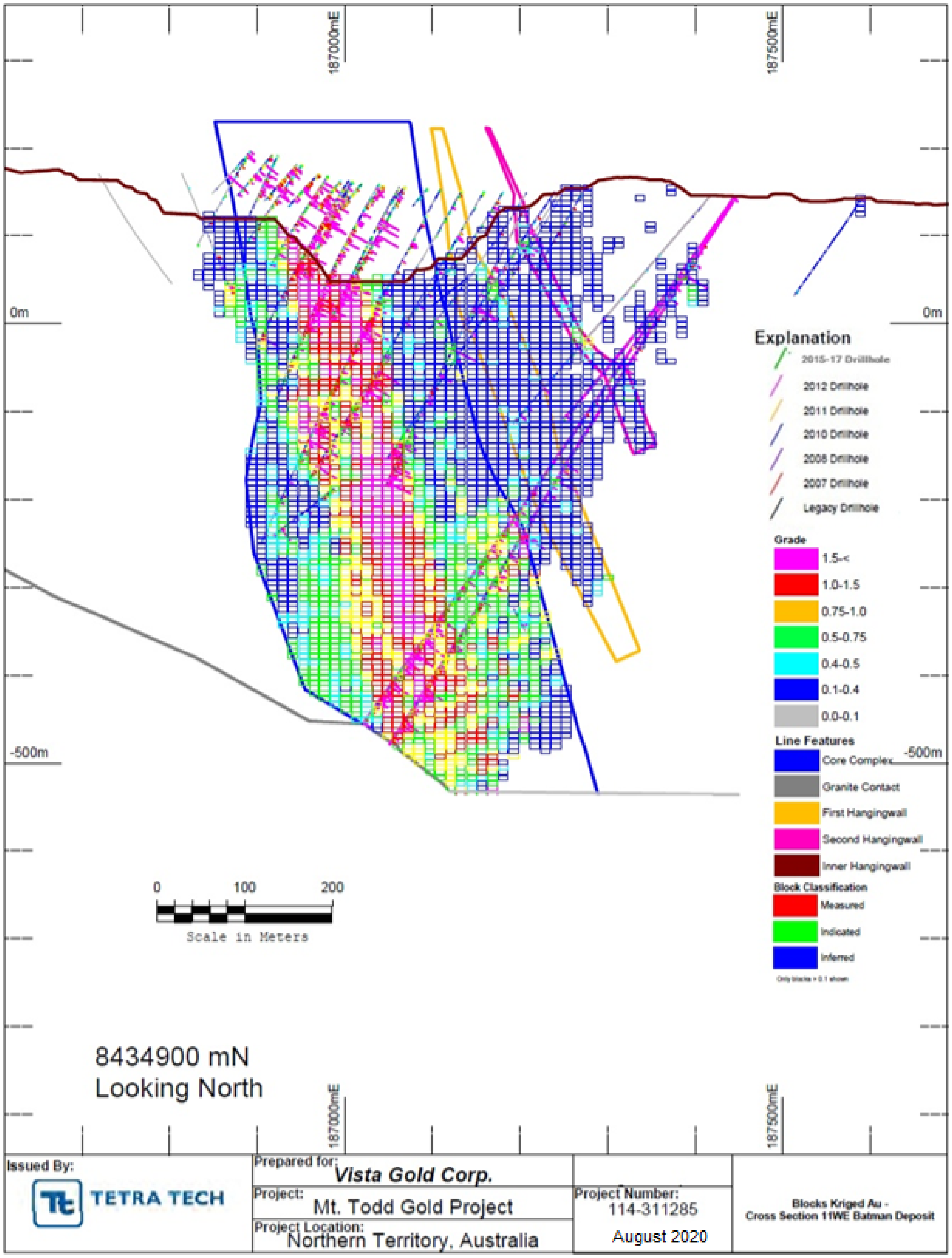


Figure 14-5: Blocks Kriged Au – Cross-section 8,434,900 mN Looking North, Batman Deposit

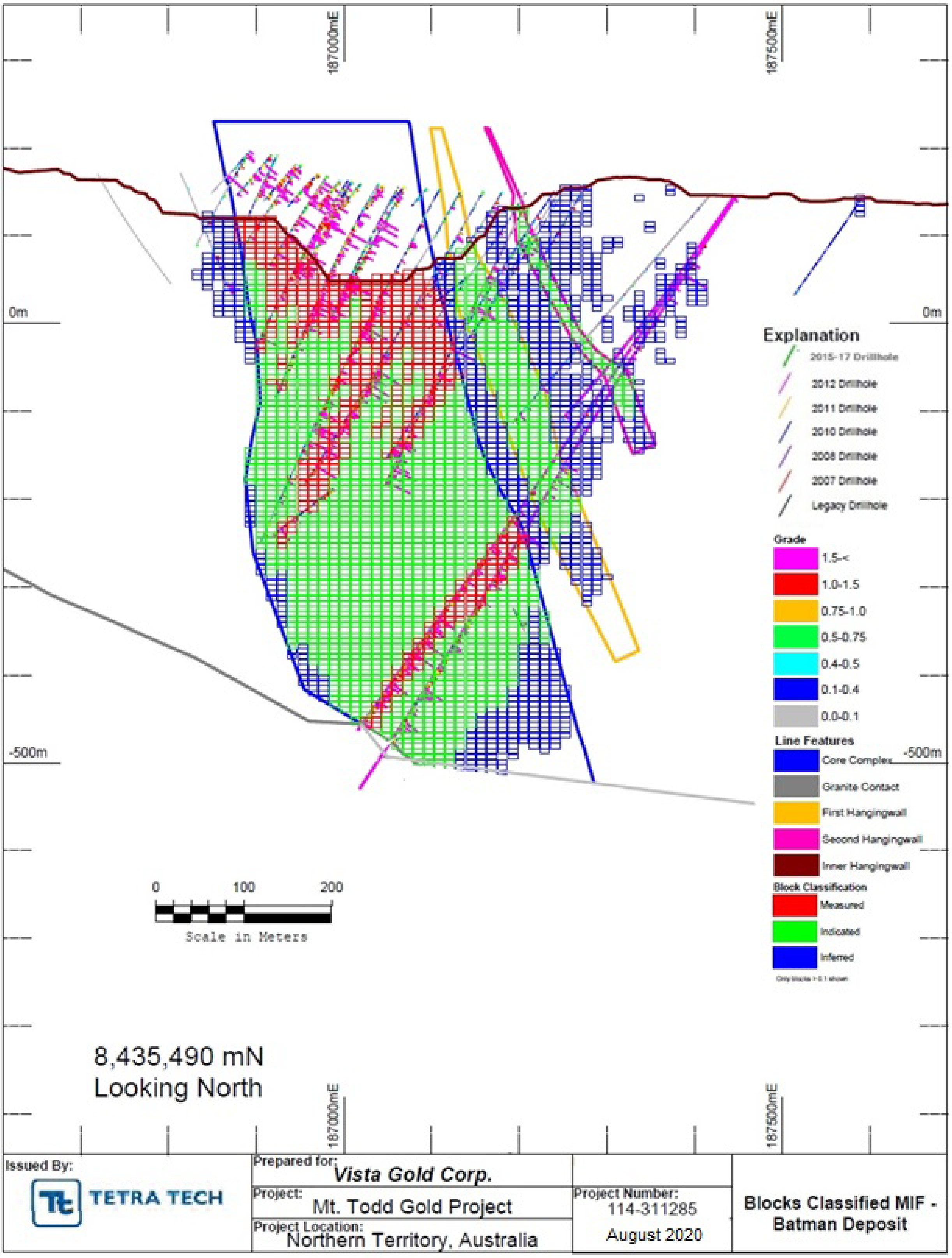


Figure 14-6: Classified Blocks Measured, Indicated, and Inferred – Cross-section 8,434,900 mN Looking North, Batman Deposit

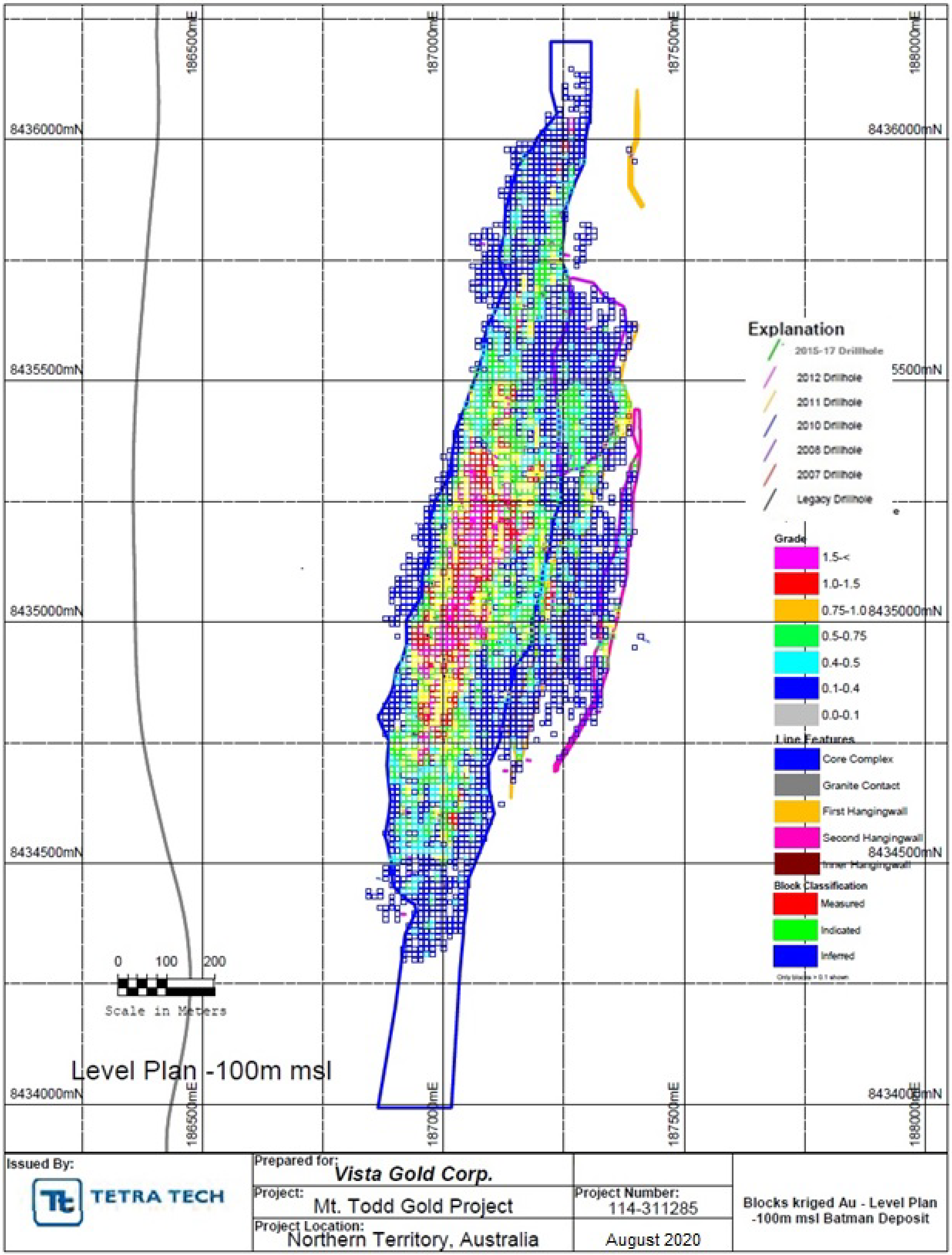


Figure 14-7: Blocks Kriged Au – Level Plan -100m msl Batman Deposit

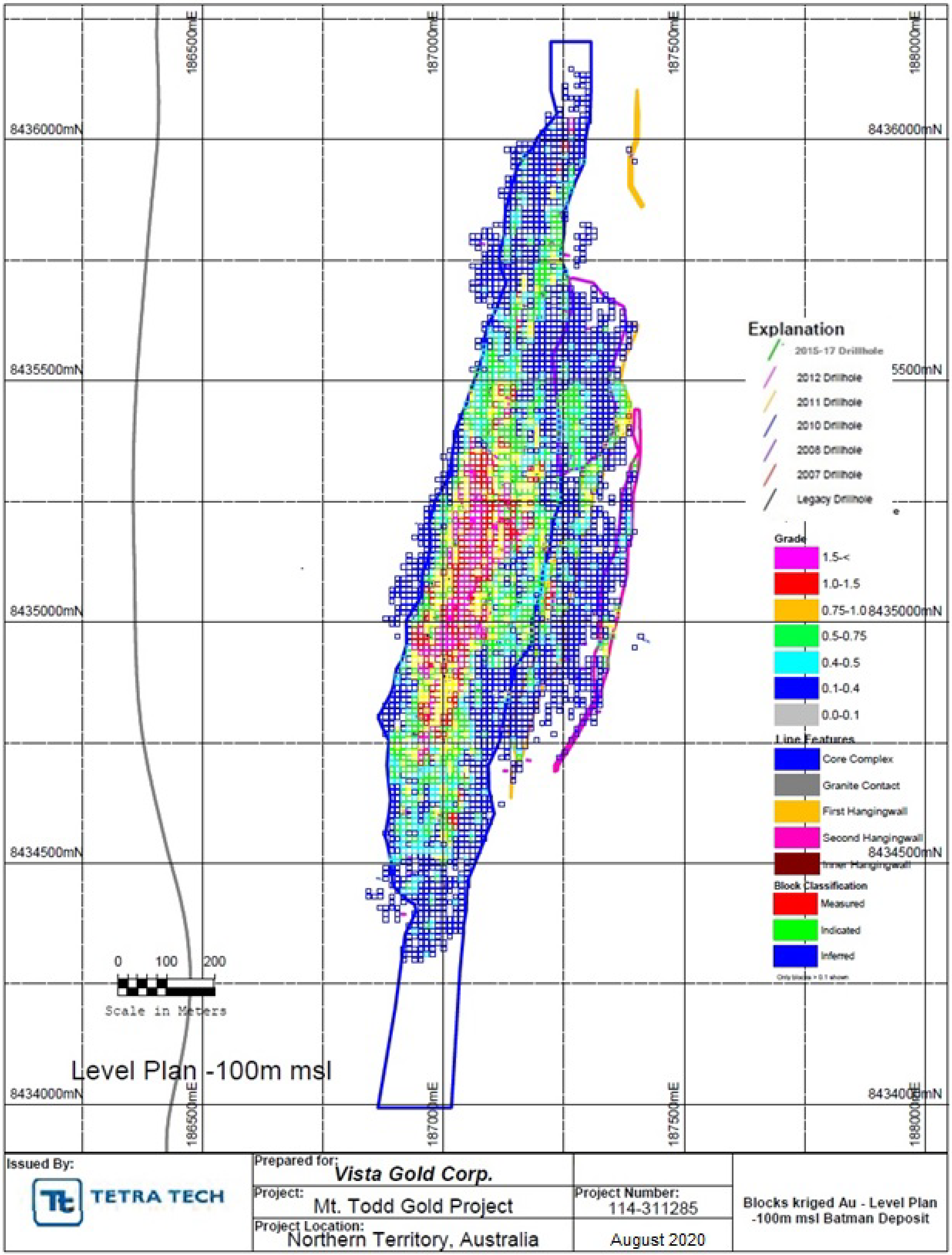


Figure 14-8: Classified Blocks Measured, Indicated, and Inferred – Level Plan -100m msl Batman Deposit

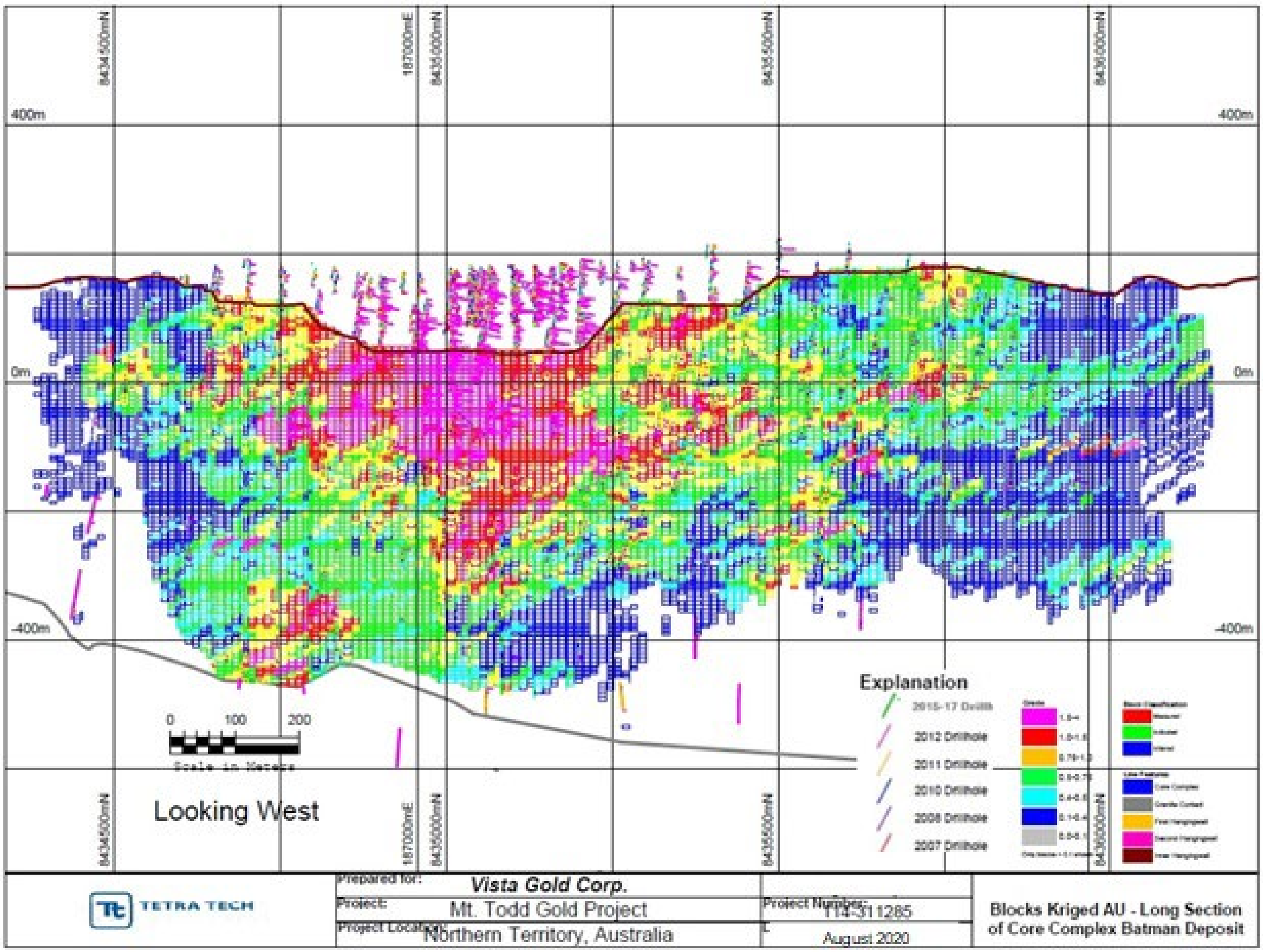


Figure 14-9: Blocks Kriged Au – Long Section of the Core Complex Looking West

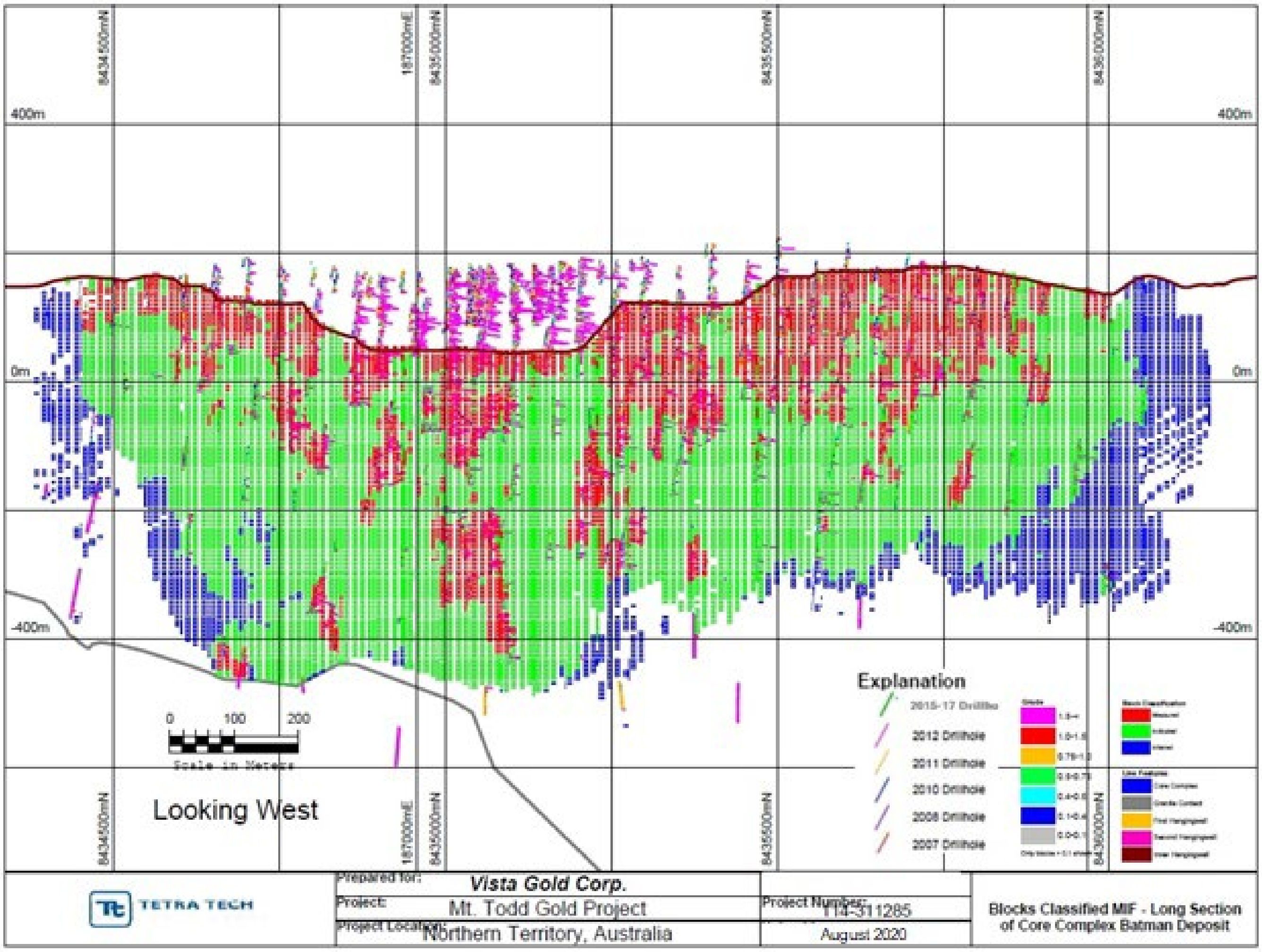
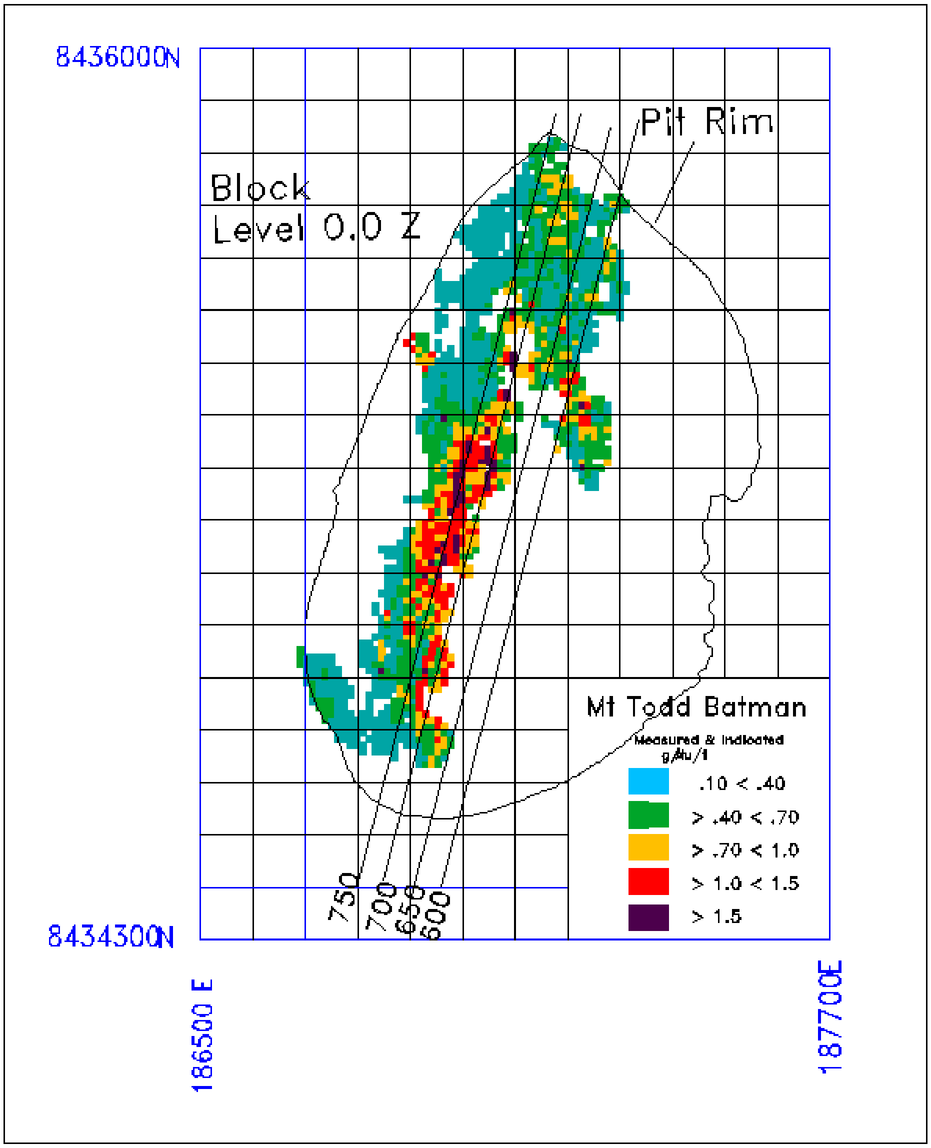
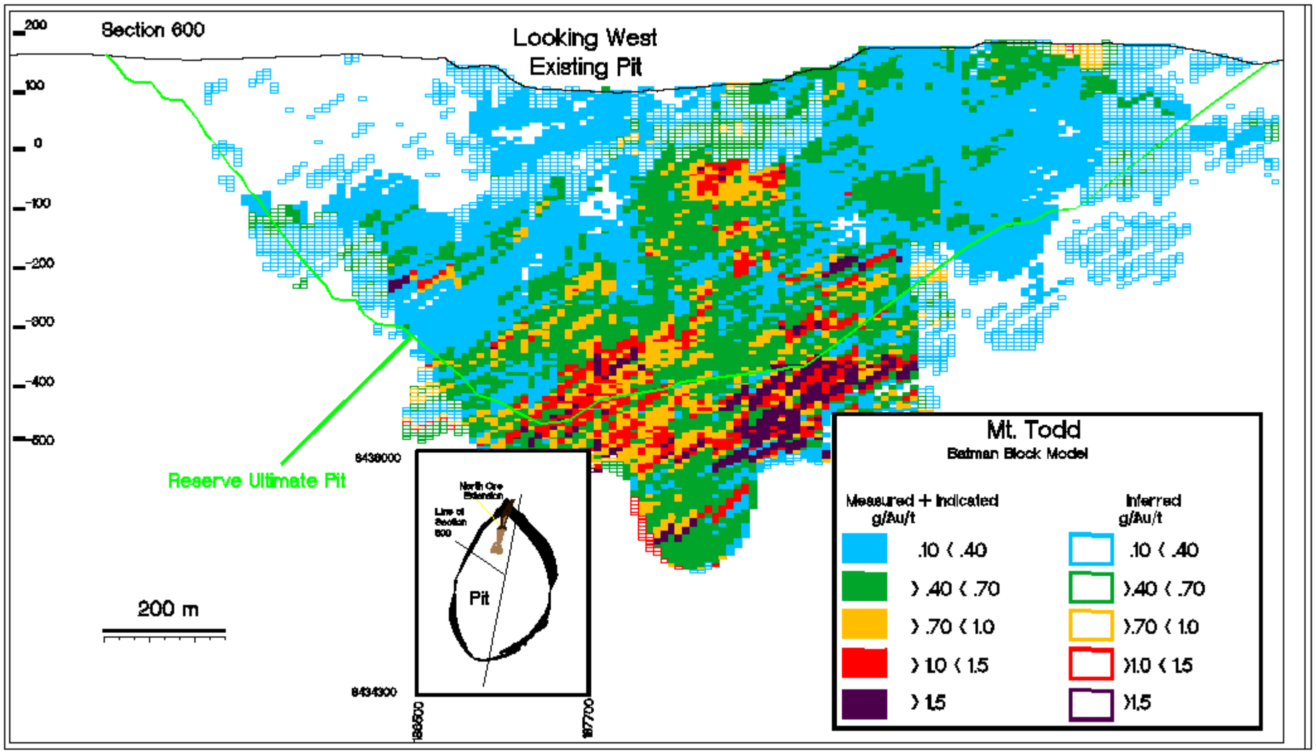


Figure 14-10: Classified Blocks Measured, Indicated, and Inferred – Long Section of the Core Complex Looking West



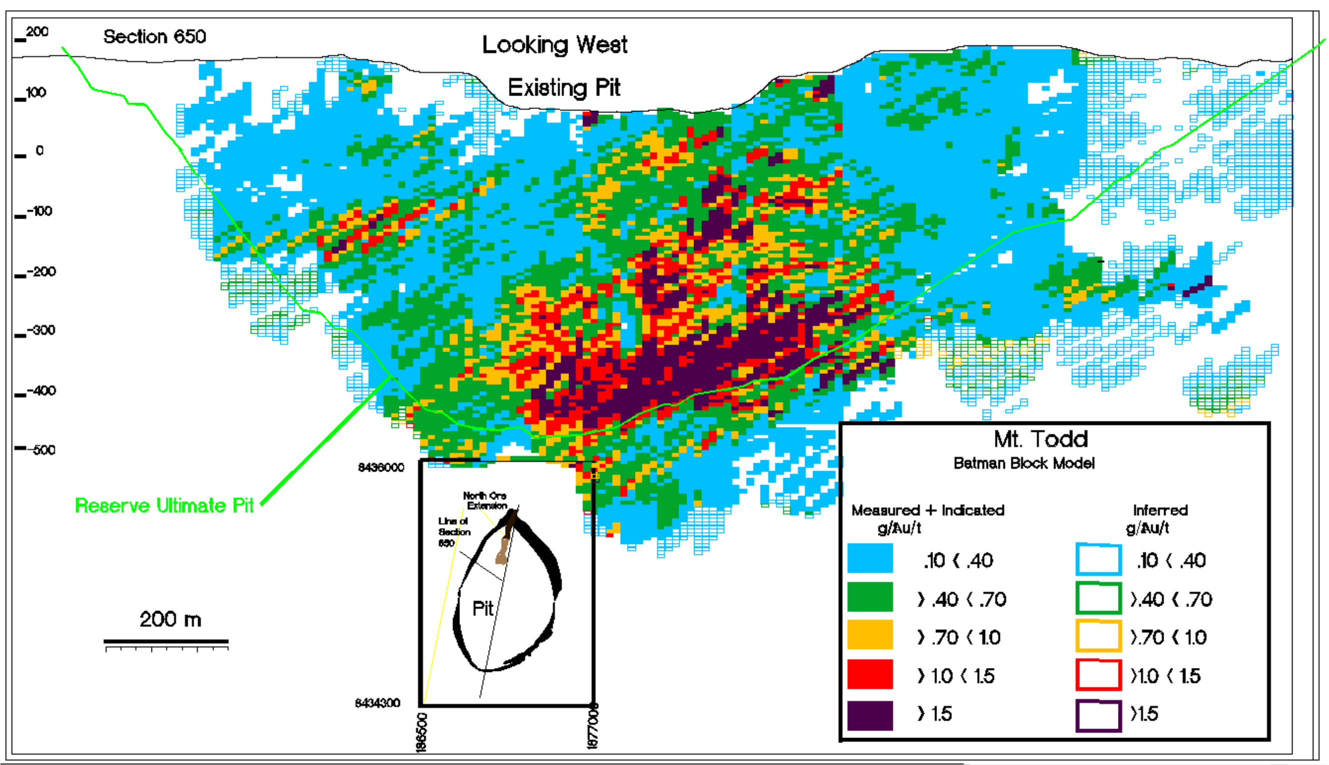
Source: Tetra Tech, 2021

Figure 14-11: Plan Map of Mineral Reserve Pit Rim and location of sections through block model



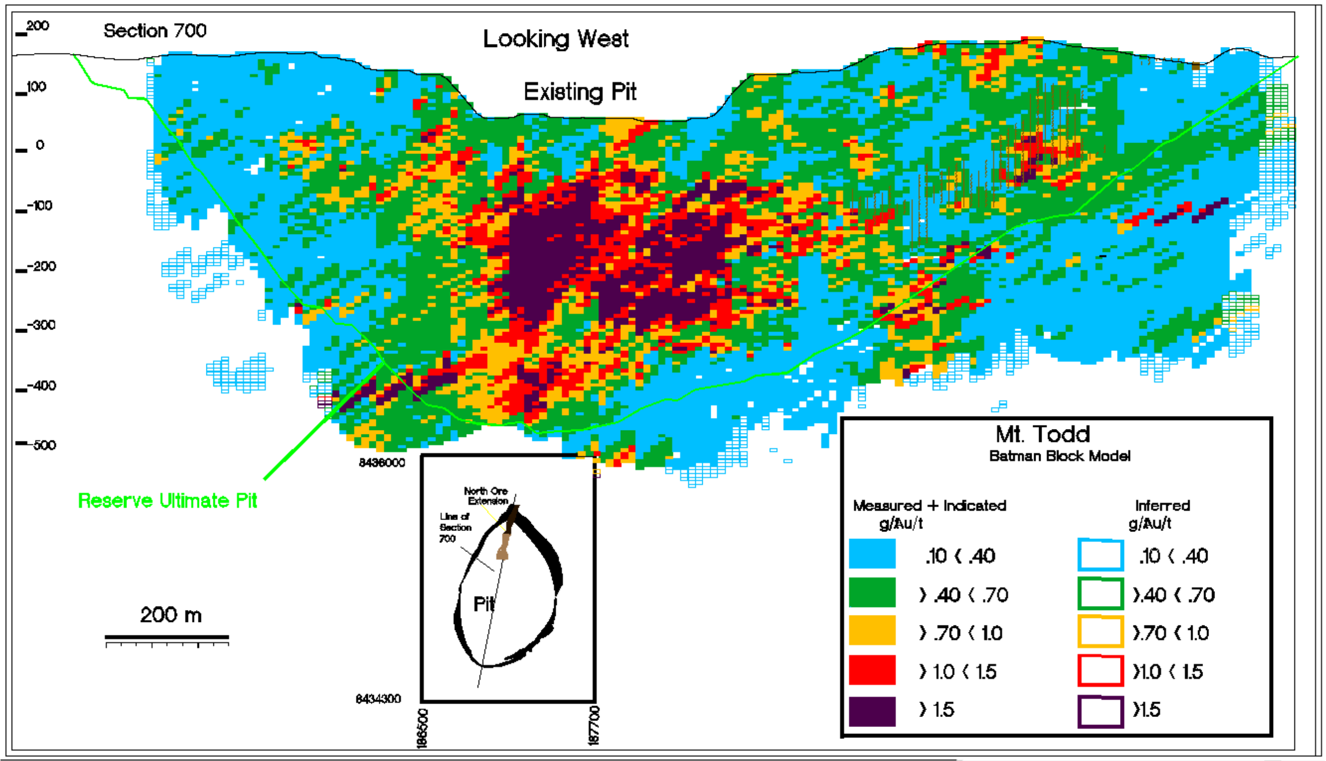
Source: Tetra Tech, 2021

Figure 14-12: Section 600 of mineral reseves above ultimate pit



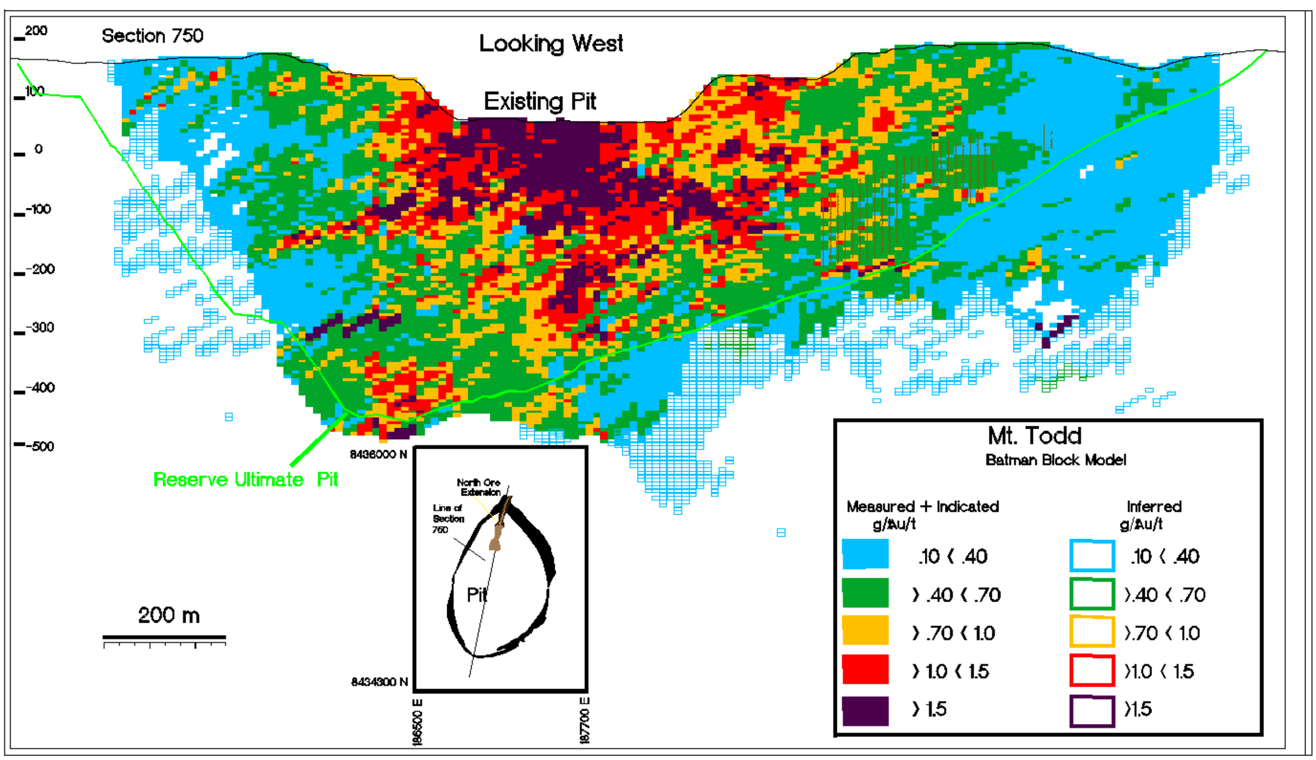
Source: Tetra Tech, 2021

Figure 14-13: Section 650 of mineral reseves above ultimate pit



Source: Tetra Tech, 2021

Figure 14-14: Section 700 of mineral reseves above ultimate pit



Source: Tetra Tech, 2021

Figure 14-15: Section 750 of mineral reseves above ultimate pit

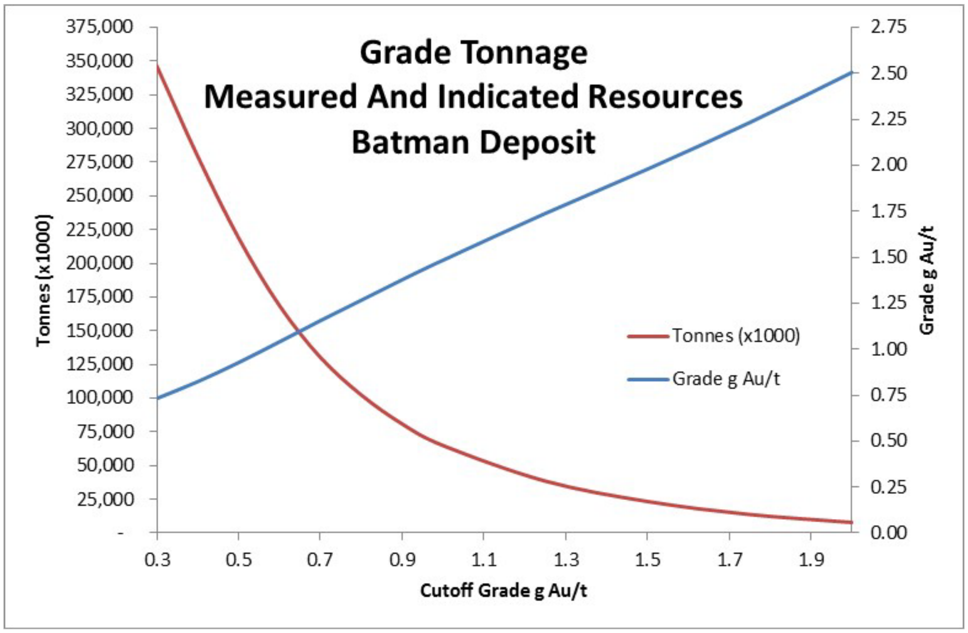
Table 14-5 lists the current Batman measured and indicated resource estimates at cutoff grades ranging from 0.3 g Au/t to 2.0 g Au/t. Table 14-6 lists the current Batman inferred resource estimates at cutoff grades ranging from 0.3 g Au/t to 2.0 g Au/t.

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**
	0.35	84,222	0.84	2,264.4
	0.3	90,719	0.80	2,337.8
INDICATED	2.00	5,413	2.55	443
	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**
	0.35	226,650	0.75	5,567
	0.30	253,187	0.71	5,765
MEASURED + INDICATED	2.00	7,887	2.50	634
	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**
	0.35	284,332	0.78	7,831
	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.
3) The effective date of the mineral resource estimate is December 31, 2023.



Source: Tetra Tech, August 2019

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

Figure 14-16: 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

NOTES:
1) Resources constrained within a US$1,300/oz gold Whittle™ Pit Shell. Tonnage, grades and totals may not total due to rounding. The reported resources at a cutoff of 0.4 g/t is highlighted. Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.
2) The effective date of the mineral resource estimate is December 31, 2023.

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.
- A reconciliation of the Batman Pit historical production by Pegasus with estimated blocks using Vista's block model parameters.

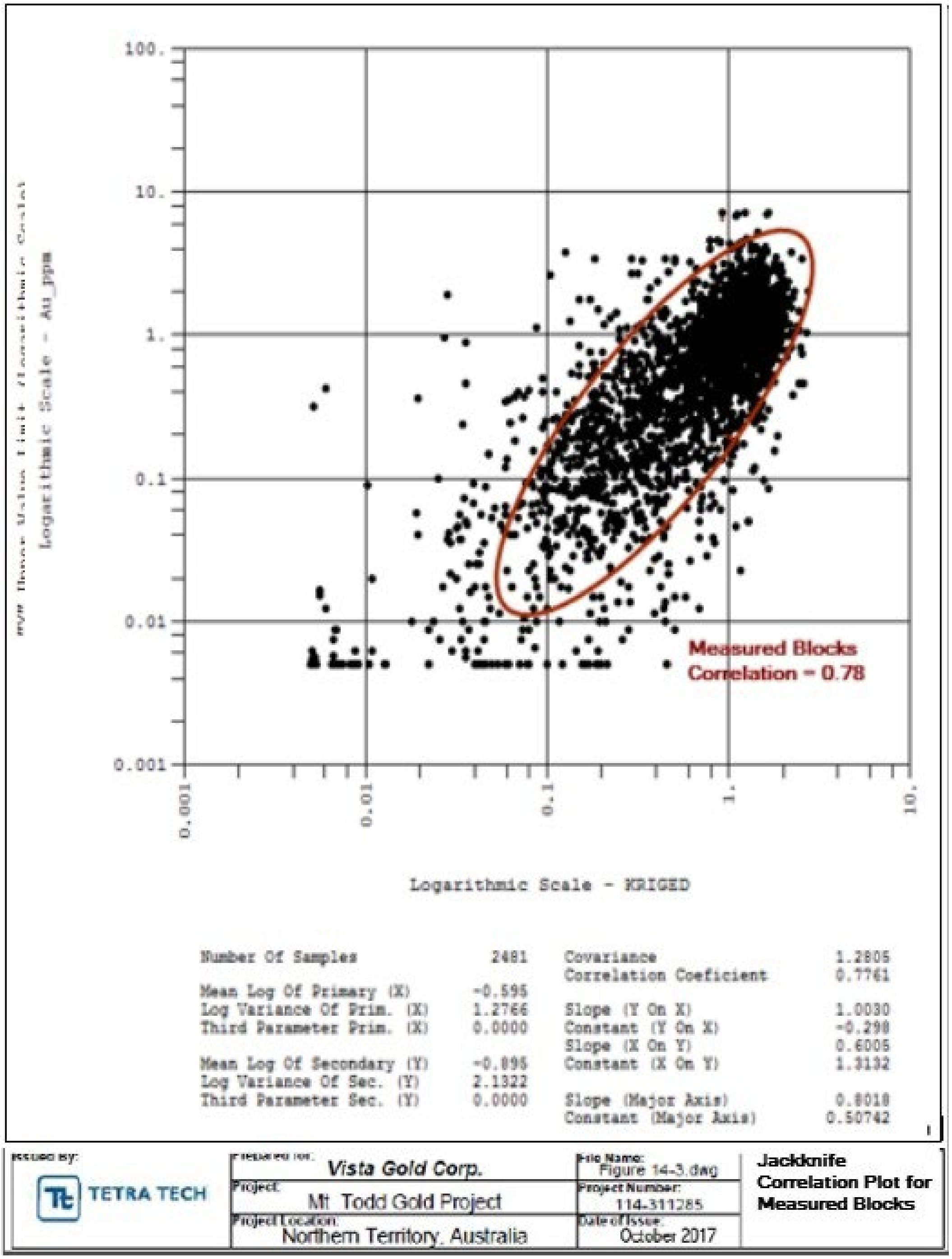
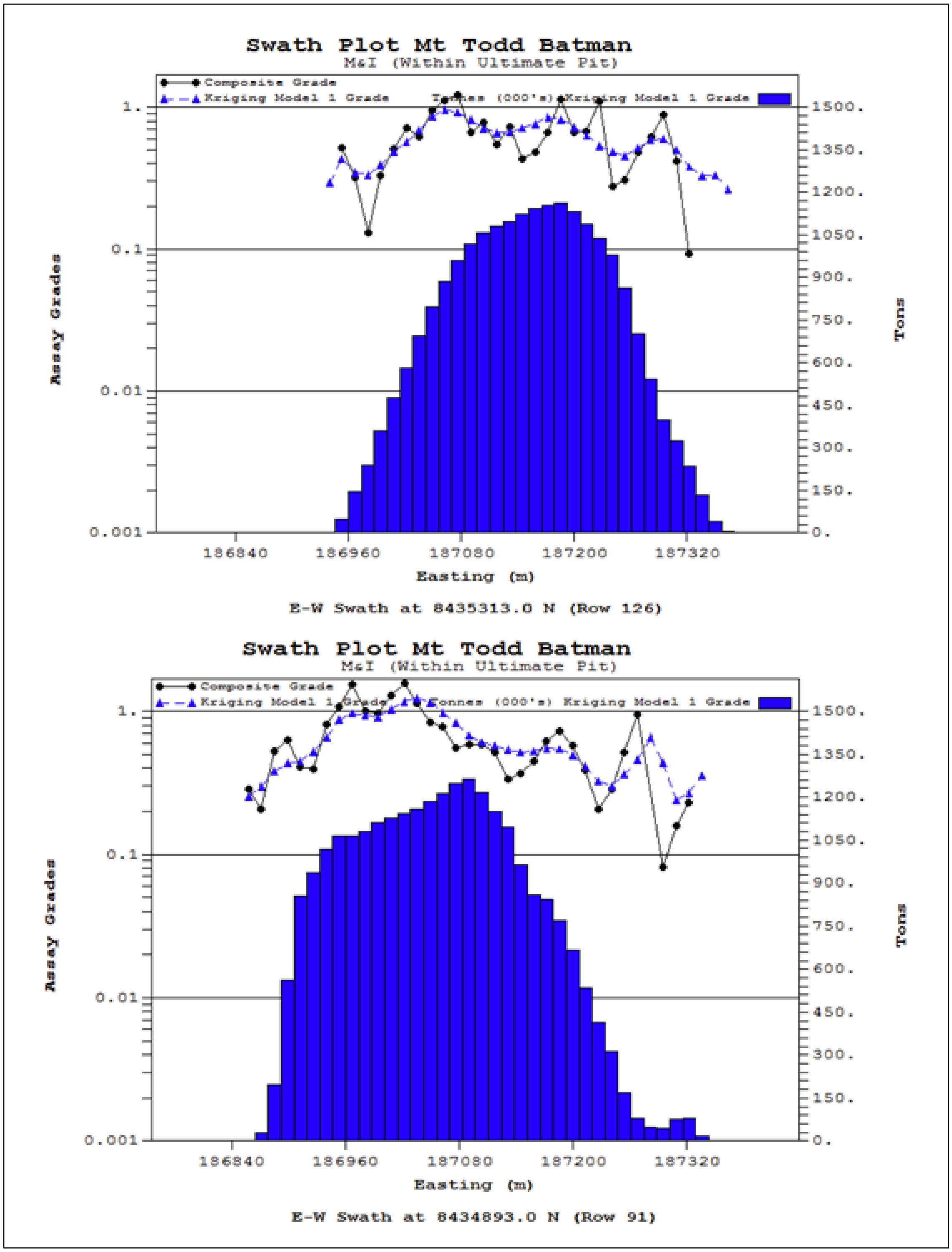


Figure 14-17: Jackknife Correlation Plot for Measured Blocks



Source: Tetra Tech, August 2019

Figure 14-18: Jackknife Correlation Plot for Inferred Blocks

The previous mining at the Batman deposit allows for a unique opportunity in checking Vista's current resource model. Pegasus Gold (Pegasus) mined 21 million tonnes of gold ore in the 1990s producing 700,000 gold ounces from the Batman Pit. This is from just the top of Batman deposit which is presently modeled with mineralization that is over ten times larger. A reconciliation study was done that compared what was mined from this historical Batman pit with what can be estimated using Vista's model. Note that reported resources by Pegasus and its consultants are non-compliant, as their values were produced prior to NI 43-101 regulations.

The reconciliation required that the Vista model be extended upward through the historically mined area. While much of Pegasus' information developed during mining is now gone, what remains are gold grade data from 730 exploratory drillholes (DH) and 100,000 ore grade control blastholes (BH). This data allowed for a check on the global resources using drillhole data (DH) that is published in the 2021 NI 43-101 Technical Report (TR) (Tetra Tech, 2021). It was produced by MicroModel™ mining software using 730 historical drillholes (DH). The results from the Vista model were then checked against using BH as well and with mine production records.

Vista's model matched the 700,000 gold ounces from 20,806,000 tonnes with an average grade of 1.049 g/t mined using production records highlighted in yellow in Table 14-7. The area highlighted in red tabulates gold resource estimates using DH data. The area in blue tabulates gold resource estimates using BH data. A match of kriging estimation using DH data requires a cutoff grade of 0.7 g/t. An alternative estimate using BH data requires a cutoff grade of 0.65 g/t. These are higher than the 0.6 g/t cutoff specified by Pegasus for its mining operation. The quality of Pegasus' BH sampling is suspect for reasons described in this report. Observation by contemporary consultants has documented poor BH sampling protocols that increased the likelihood of mixing of country rock with mineralized rock.

Table 14-7: Global Reconciliation of Batman Pit historical production with Vista model

Pegasus Mt Todd Reports and Vista Model Estimates	tons (000)	oz (000)	Mine-Model oz diff (000)	g Au/t	g (000)
Historical Reports: Grand Total in (000) Reported @ 0.6 g/t cutoff	**20,806**	**702**	**0**	**1.049**	**21,834**
Model: Grand Total in (000) estimated using DH alone @ 0.7 g/t cutoff	**20,713**	**700**	**-2**	**1.051**	**21,769**
@ 0.65 g/t cutoff	23,011	749	47	1.013	23,310
@ 0.6 g/t cutoff	25,622	801	99	0.973	24,930
@ 0.4 g/t cutoff	37,994	998	296	0.824	31,307
Model: Grand Total in (000) estimated using BH alone @ 0.70 g/t cutoff	19,002	653	-49	1.069	20,313
@ 0.65 g/t cutoff	**21,158**	**700**	**-2**	**1.029**	**21,772**
@ 0.6 g/t cutoff	23,547	748	46	0.988	23,264
@ 0.4 g/t cutoff	35,331	936	234	0.824	29,113

14.5 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 14-8 shows the specific gravity data assigned to the Quigleys area according to oxidation state.

Table 14-8: Quigleys Deposit-Specific Gravity Data

Oxide within modeled shear (t/cm)	2.60
Oxide Waste (t/cm)	2.62
QTransition 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 14-9 summarizes the Quigleys exploration database.

Table 14-9: 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

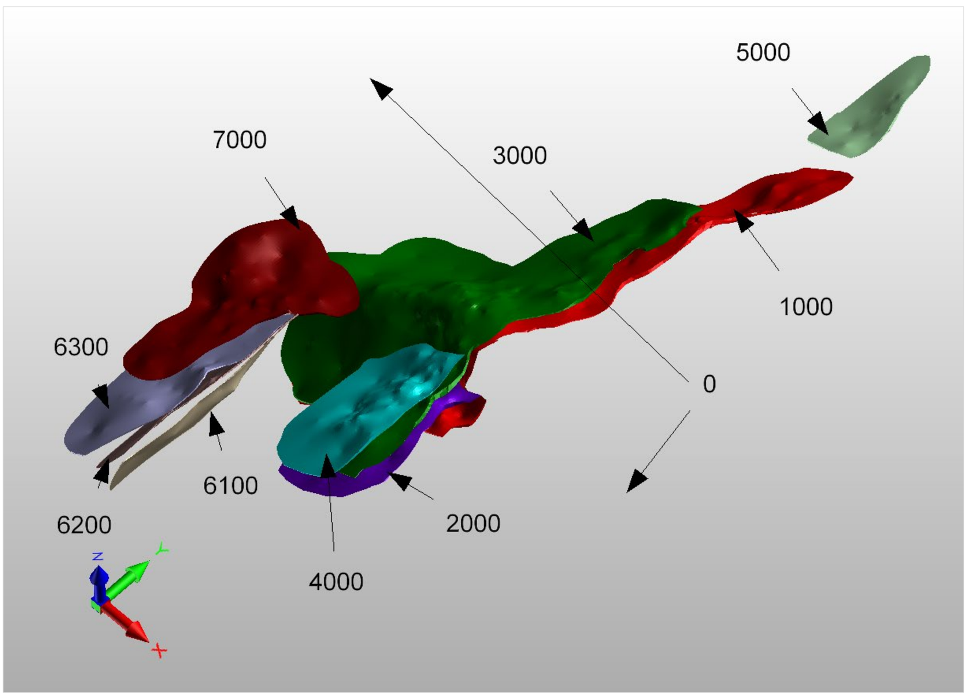
14.5.2 Quigleys Block Model Parameters

Quigleys' block model parameters are shown in Table 14-10. 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 14-10: 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.



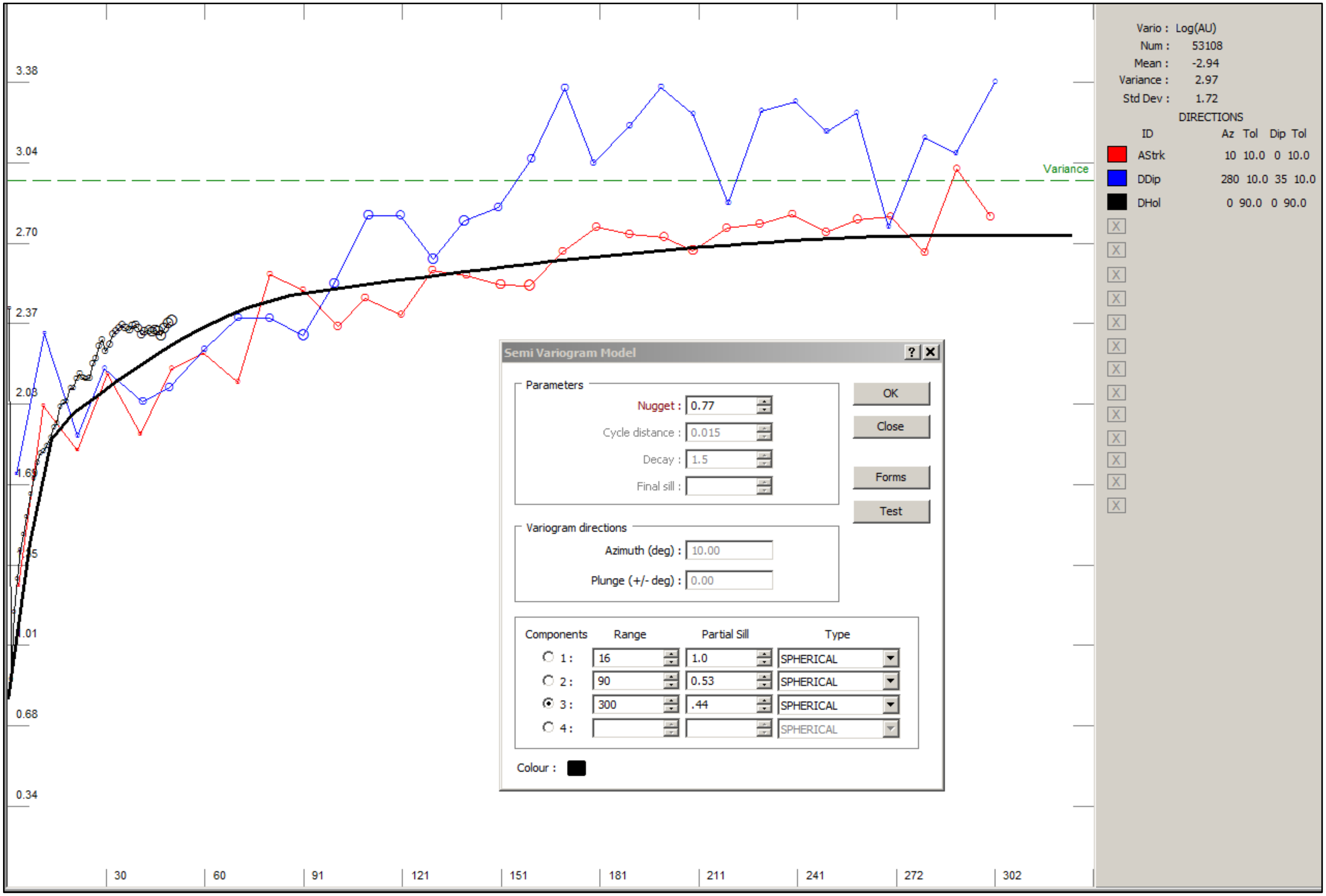
Source: Tetra Tech, August 2020

Figure 14-19: 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 historical 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 14-11 shows the search parameters selected for each domain.



Source: Tetra Tech, August 2020

Figure 14-20: Quigleys Median Indicator Variogram

Table 14-11: 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 14-12 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 block's kriging error exceeded this value.

Table 14-12: 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 were used to validate the block model and 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 14-13 lists the parameters used to generate a pit shell using Geovia's Whittle^{TM} software (version 4.7) for reporting the measured plus indicated resource in Table 14-14 and the inferred resource in Table 14-15. Note that these parameters are not the same as those specified for the Batman deposit. Quigleys has geologic, metallurgical, and mining cost characteristics that require its own parameter specifications. For example, the differing orientation of the mineralized zones between Batman and Quigleys (Batman is steeply dipping, while Quigleys is shallower) impact mining costs, as well as the location of the Quigleys deposit requiring greater haulage than for Batman. The Quigleys deposit is also anticipated to be mined with smaller equipment resulting in a loss of economies of scale. In addition, the costs are better known for Batman and can be stated with more certainty, while the costs for Quigley's are not as well-known and therefore more conservative costs were used. These parameters used to constrain a resource provide simply a reasonable potential extraction for the commodity being estimated. Mineral resources that are not mineral reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Table 14-13: Whittle^{TM} Pit Shell Parameters for the Quigleys Deposit

Item	Input
Gold Price	US$1,300 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% GPR
Sell Cost	US$3.19

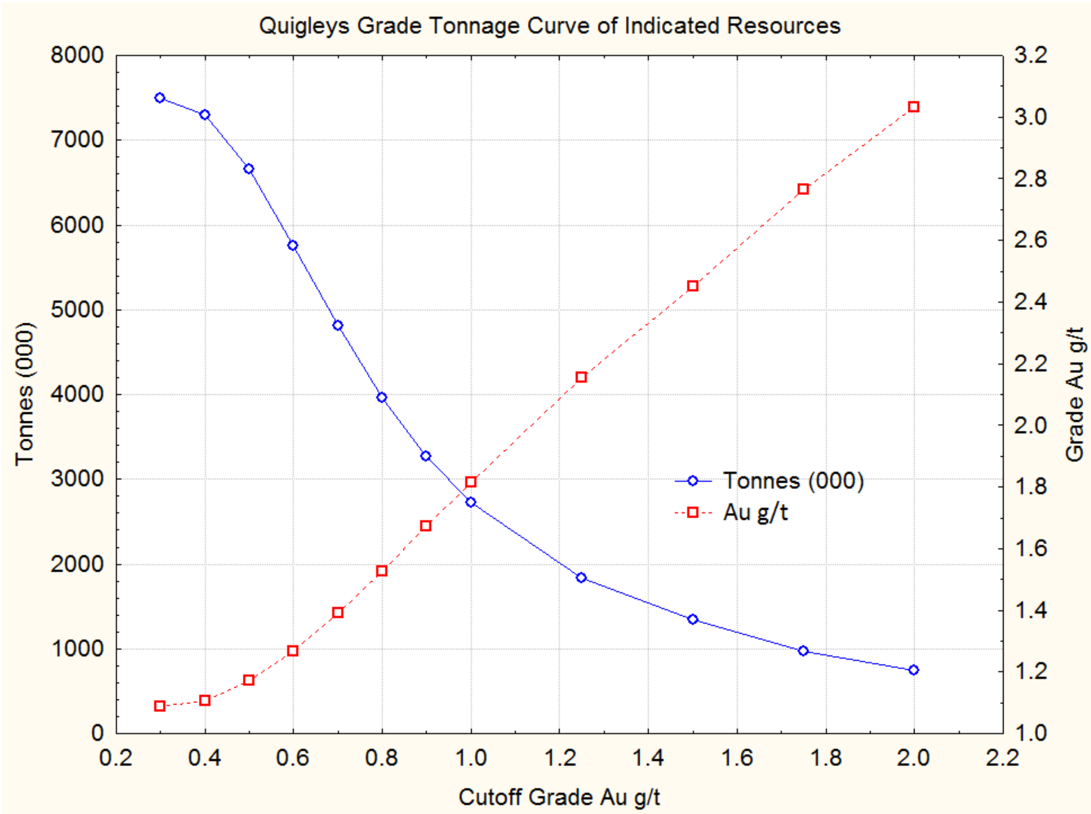
Table 14-14 lists the indicated mineral resources for the existing heap leach pad. Table 14-15 lists the inferred mineral resources for the existing heap leach pad.

Table 14-14: Estimated Resources Measured and Indicated Classes for the Quigleys Deposit, All Mineral Types

Resource Class	Cutoff	Tonnage (000)	Grade g Au/t	Ounces Au (000)
Measured & Indicated	0.3	8128	1.09	285
Measured & Indicated	0.4	7895	1.11	282
Measured & Indicated	0.5	7211	1.17	272
Measured & Indicated	0.6	6234	1.27	255
Measured & Indicated	0.7	5207	1.39	233
Measured & Indicated	0.8	4288	1.53	211
Measured & Indicated	0.9	3533	1.68	191
Measured & Indicated	1	2944	1.83	173
Measured & Indicated	1.25	1987	2.17	139
Measured & Indicated	1.5	1457	2.46	115
Measured & Indicated	1.75	1065	2.77	95
Measured & Indicated	2	826	3.03	81

NOTE:
1) Resources constrained within a US$1,300/oz gold Whittle^{TM} Pit Shell; Tonnage, grades and totals may not total due to rounding and show the reported resources at the chosen cutoff of 0.4 g/t
2) The effective date of the mineral resource estimate is December 31, 2023.

Figure 14-21 shows the grade tonnage curve of indicated resources.



Source: Tetra Tech, August 2020

Figure 14-21: Grade Tonnage Curve Estimated Resources Measured and Indicated Class, All Mineral Types for the Quigleys Deposit

Table 14-15: Estimated Resource Inferred Class for the Quigleys Deposit

Resource Class	Cutoff	Tonnage (000)	Grade Au g/t	Ounces Au (000)
Inferred	0.3	4057.10	1.44	188
Inferred	0.4	3980.65	1.46	187
Inferred	0.5	3671.75	1.55	182
Inferred	0.6	3284.60	1.66	176
Inferred	0.7	2832.35	1.82	166
Inferred	0.8	2466.84	1.98	157
Inferred	0.9	2161.41	2.14	149
Inferred	1	1982.68	2.25	144
Inferred	1.25	1532.17	2.59	127
Inferred	1.5	1195.76	2.93	113
Inferred	1.75	842.78	3.47	94
Inferred	2	642.34	3.98	82

NOTE:
1) The effective date of the mineral resource estimate is December 31, 2023.

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 Billiton-Zapopan Pegasus operations, 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 1,162 measurements in 11 drillholes using 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.

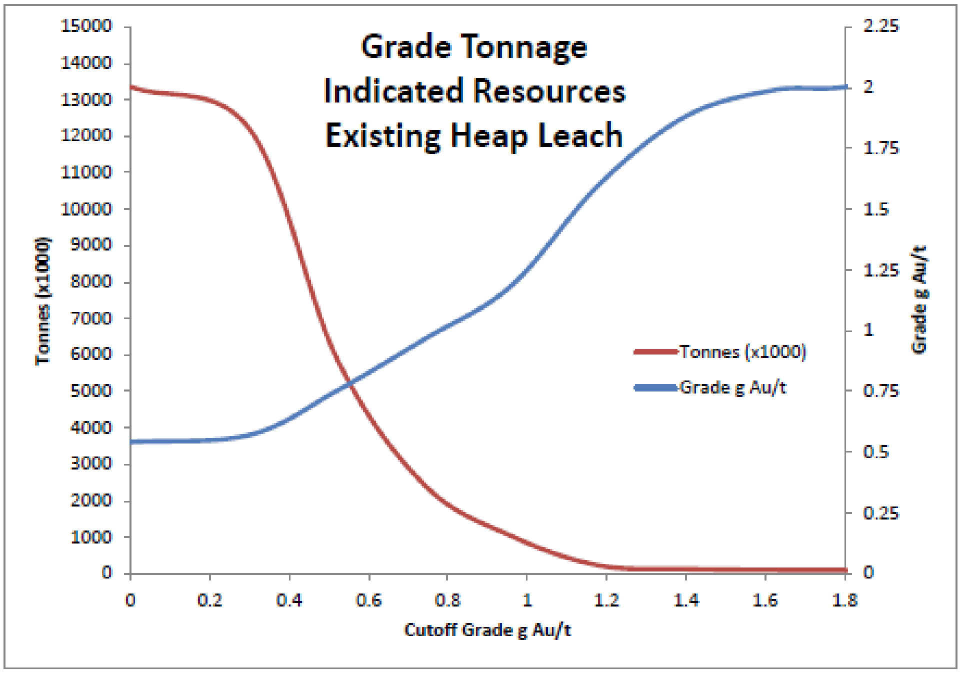
A nearest neighbor (polygon) method was employed to estimate grades within the heap leach pad since there is no apparent spatial correlation between samples. The existing heap leach pad is estimated to contain 230,000 ounces of gold within 13.4 Mt of indicated mineral resource at an average grade of 0.54 g Au/t. It is the opinion of the QP [Rex Clair Bryan, Ph.D., SME RM] 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 14-16 lists the existing heap leach resource estimate. Note the grade-tonnage plot in Figure 14-17. Most of the 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 14-16: Existing Heap Leach Indicated Gold Resource Estimate (September 2019)

	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 as all heap leach material will be re-processed.
2) Resources are reported at 0.4 g/t cutoff gold grade to be consistent with the reported Batman and Quigleys resource.
3) Resource is defined by the geometry of the existing heap leach pad.
4) 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.
5) The effective date of the mineral resource estimate is December 31, 2023.



Source: Tetra Tech, August 2020

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

Figure 14-22: Inferred Resource Grade Tonnage Curve for the Existing Heap Leach

14.7 Relevant Factors Affecting Resource Estimates

Different mining and metallurgical parameters are used for the Batman and Quigleys deposits. These parameters are used to produce a mining geometry that provides a reasonable prospect of economic viability. The different set of parameters reflects that each deposit is unique in geology and location. The Batman mining and milling costs and other technical parameters are better known and can be stated with more certainty than for Quigleys, whereas the technical parameters and costs for Quigleys are not as well-known and therefore more conservative costs were used. In addition, physical factors also impact these technical parameters. For example, haulage from the Quigleys deposit to the plant is farther than for Batman and the dumping space has not been fully determined, so additional haulage is anticipated for Quigleys. In addition, the Quigleys deposit is anticipated to be mined with smaller equipment resulting in a loss of economies of scale.

It is the opinion of the QP that 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. MINERAL RESERVES

The measured and indicated resource estimates presented in Section 14 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 the QP [Thomas L. Dyer, P.E.] used the phased pit designs to define the production schedule to be used for cash-flow analysis for the preliminary feasibility study.

Pit optimizations for this report have been up based on the resulting costs used for the cash-flow analysis. These have been done to validate the previous 2021 pit designs which are the basis of the mineral reserves, production schedule, and the resulting economic analysis. The previous pit designs have been used to report the current Mineral reserves and are shown to remain valid with a 0.35 g Au/t cutoff grade.

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 cash-flow model.

15.1 Pit Optimization

Pit optimization was done using Geovia's Whittle™ software (version 4.7.3) 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. Pit optimization of the ultimate pit limits was based on economic parameters assuming a 50,000 tpd processing of mineralized material.

Economic parameters are provided in Table 15-1. Mining cost parameters were initially based on the NI 43-101 Technical Report - Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study (2019 PFS), but later updated to reflect the final mining costs presented below.

Table 15-1: Initial Economic Parameters

Parameter	Value Used
Gold Recovery	Grade dependent constant tail equation
Payable Gold	99.9%
Reference Mining Cost	US$1.99 per tonne mined
Incremental Mining Cost	US$0.012 per tonne per bench
Overall Mining Cost	US$2.35 per tonne mined
Processing Cost	US$11.21 per tonne processed
General & Administrative	$1.50 per tonne processed
JAAC Royalty[2]	1% GPR

[2] Prior royalty used for initial Lerch-Grossman cone runs. Final designs use actual royalty data.

The mining costs used were varied by bench. An incremental cost of US$0.012 was added for each six-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 below the 145-meter elevation. 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 of US$1.99 was determined using first principles from previous studies. The overall mining cost (reference plus incremental) is US$2.35 per tonne .

Note that the processing cost of $11.21/t processed is based on the updated processing cost for this study. This is a 39% increase over the previous study. Accordingly, a 35% increase has been added to the G&A which results in a $1.50/t processed. This may be more or less than the results shown in Section 18 operating costs.

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

A base gold price of US$1,500 per ounce was determined based on the metal price required to justify the existing pit designs and has been used in scenario analysis. However, various gold prices from US$300 to US$2,500 per ounce, in increments of US$25 per ounce, were used to determine different optimized pit shells. Using the parameters shown in Table 15-1, the breakeven cutoff grade is 0.31 g Au/t using a $1,500 per ounce of gold. However, the evaluation was completed using 0.35 g Au/t as a minimum grade. This was done to decrease the mine life and optimize the grade of material fed through the plant.

Note that the $1,500 per ounce gold price results in conservatism for the pit optimization as the final gold price used for this report is $1,800 per ounce. Also, the final pit chosen to guide the ultimate pit design was created at a $1,500 per ounce gold price. Thus, the resulting reserves are robust to both the gold price used and the resulting cutoff grade and metal price used for the economic evaluation.

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 Barkley (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 eastern side were widened to allow a berm to be maintained along the road to contain any rock that would slough off the wall.

The primary change suggested by Barkley (2016) is to either place catch benches in the high wall on the eastern 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 steeper slopes with bolting and mesh. This helps to improve the overall slope and reduce the resulting stripping.

Figure 15-1 shows the slope sectors with relation to the previous 2020 PFS pit design. Each sector was modeled into a zone resulting in eight zones. For pit optimization, 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

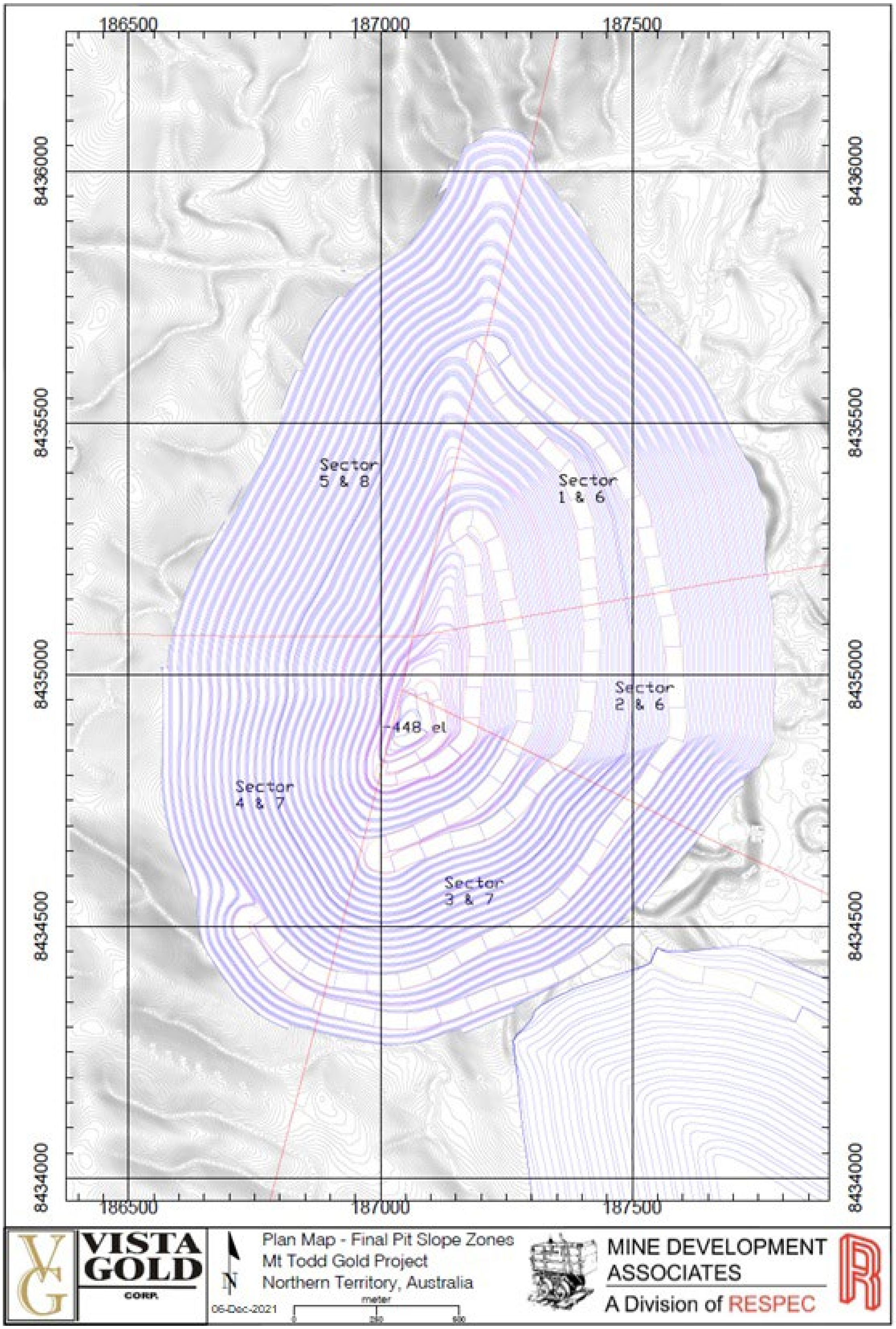


Figure 15-1: Mt Todd Geotechnical Sectors

15.1.3 Pit Optimization Results

Pit optimizations were run using Whittle™ software (version 4.7). Inputs into Whittle included the resource block model along with the economic and geometric parameters previously discussed. Ultimate pit shells were selected from the Whittle results for final design.

The selections of ultimate pits and pit phases were done as a two-step process. The first step was to optimize a set of pit shells based on varying a revenue factor. This optimization was done in Whittle using a Lerchs-Grossman algorithm. The revenue factor was multiplied by the recovered ounces and the metal prices, essentially creating a nested set of pit shells based on different metal prices. Revenue factors for the deposit were varied from 0.30 to 2.5 in increments of 0.025 with a base price of US$1,000 per ounce of gold, so the resulting pit shells represent gold prices from US$300 to US$2,500 per ounce in increments of US$25. This has the potential of generating up to 84 different pit shells that can be used for analysis. The resulting pit number one will be the first pit that optimizes, so if a pit is not viable until a given revenue factor, that will become the first pit. In addition, in some cases pit shells with increments are coincidental to other pits and reported as a single pit. Thus, the number of pits may vary for each deposit and run.

The second step of the process was to use the Pit by Pit (PbP) analysis tool in Whittle to generate a discounted operating cash flow (note that capital is not included). These were done using a constant gold price of US$1,400 per ounce. The PbP node uses a rough scheduling by pit phase for each pit shell to generate the discounted value for the pit. The program develops three different discounted values: best, worst, and specified. The best-case value uses each of the pit shells as pit phases or pushbacks. For example, when evaluating pit 20, there would be 19 pushbacks mined prior to pit 20, and the resulting schedule takes advantage of mining more valuable material up front to improve the discounted value. Evaluating pit 21 would have 20 pushbacks; pit 22 would have 21 pushbacks and so on. Note that this is not a realistic case as the incremental pushbacks would not have enough mining width between them to be able to mine appropriately, but this does help to define the maximum potential discounted operating cash flow.

The worst case does not use any pushbacks in determining the discounted value for each of the pit shells. Thus, each pit shell is evaluated as if mining a single pit from top to bottom. This does not provide the advantage of mining more valuable material first, so it generally provides a lower discounted value than that of the best case.

The specified case allows the user to specify pit shells to be used as pushbacks and then schedules the pushbacks and calculates the discounted cash flow. This is more realistic than the best case as it allows for more mining width, though the final pit design will have to ensure that appropriate mining width is available. The specified case has been used to determine the ultimate pit limits to design to, as well as to specify guidelines for designing pit phases.

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$2,500 per ounce gold with increments of US$25 per ounce. Results for US$100 per ounce increments, from US$300 to US$2,500 per ounce of gold, are shown in Table 15-3 with an additional pit shell shown for the $1,750 ounce gold metal price (pit shell 57). The highlighted price of US$1,500/oz-Au was the pit shell (pit 49) that most resembles the pit shell used for guidance in designing the ultimate pit. Pit shell 57 is highlighted as the pit shell created using a $1,750 gold price. These pit optimizations used only Measured and Indicated resources. Inferred materials were considered waste. The highlighted price of US$1,500/oz -Au was the pit shell created using a base price for pit optimization. Pit shell 34 is highlighted in green as the pit shell used to guide the ultimate pit design. The pit optimizations only used Measured and Indicated resources. Inferred materials were considered waste.

Table 15-3: Whittle™ Pit Optimization Results – using 0.35 g g Au/t Cutoff

Pit	Gold Price USD/oz Au	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio
		K Tonnes	g Au/t	K Ozs Au			
1	$ 300	173	1.97	11	97	269	0.56
5	$ 400	4,746	1.73	263	4,671	9,417	0.98
9	$ 500	9,246	1.55	461	9,975	19,222	1.08
13	$ 600	15,039	1.37	663	16,431	31,469	1.09
17	$ 700	25,305	1.21	986	31,484	56,789	1.24
21	$ 800	75,890	1.05	2,566	163,312	239,202	2.15
22	$ 825	81,985	1.04	2,731	175,576	257,561	2.14
23	$ 850	88,612	1.02	2,907	189,939	278,551	2.14
24	$ 875	93,241	1.01	3,018	197,209	290,450	2.12
25	$ 900	100,529	0.99	3,209	215,507	316,036	2.14
29	$ 1,000	126,157	0.93	3,780	261,842	387,999	2.08
33	$ 1,100	160,130	0.88	4,548	351,498	511,628	2.20
37	$ 1,200	200,710	0.83	5,362	447,259	647,969	2.23
41	$ 1,300	234,102	0.80	5,992	530,199	764,301	2.26
45	$ 1,400	257,028	0.79	6,516	647,721	904,748	2.52
49	$ 1,500	267,434	0.79	6,757	711,493	978,927	2.66
52	$ 1,600	275,395	0.79	6,952	776,204	1,051,600	2.82
55	$ 1,700	278,304	0.78	7,014	795,653	1,073,956	2.86
57	$ 1,750	281,900	0.78	7,105	831,474	1,113,374	2.95
59	$ 1,800	282,278	0.78	7,112	833,339	1,115,617	2.95
63	$ 1,900	286,188	0.78	7,208	875,572	1,161,760	3.06
66	$ 2,000	288,273	0.78	7,256	897,322	1,185,596	3.11
69	$ 2,100	289,809	0.78	7,291	914,481	1,204,290	3.16
72	$ 2,200	290,102	0.78	7,296	916,368	1,206,470	3.16
75	$ 2,300	293,182	0.78	7,370	958,631	1,251,814	3.27
79	$ 2,400	294,574	0.78	7,397	973,287	1,267,862	3.30
82	$ 2,500	296,705	0.78	7,443	1,003,010	1,299,714	3.38

Pit 34 was used for design purposes.

Table 15-4 shows the pit-by-pit results, and Figure 15-2 shows the pit-by-pit results graphically.

Table 15-4: $1,500 Au Price Pit by Pit Results

Pit	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio	Disc. Operating CF (M USD)			LOM Years
	K Tonnes	g Au/t	K Ozs Au				Best	Specified	Worst	
15	30,393	1.02	997	13,185	43,578	0.43	$819.53	$816.33	$811.28	1.71
16	34,717	1.00	1,115	18,332	53,049	0.53	$892.43	$888.16	$880.26	1.96
17	37,053	0.98	1,172	19,735	56,789	0.53	$926.78	$922.18	$913.01	2.09
18	54,086	0.92	1,606	39,615	93,701	0.73	$1,172.03	$1,160.48	$1,139.75	3.05
19	88,279	0.88	2,501	92,239	180,518	1.04	$1,626.70	$1,615.13	$1,556.68	4.97

Pit	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio	Disc. Operating CF (M USD)			LOM Years
	K Tonnes	g Au/t	K Ozs Au				Best	Specified	Worst	
20	95,634	0.87	2,678	100,716	196,350	1.05	$1,710.86	$1,699.29	$1,629.54	5.39
21	110,069	0.86	3,060	129,133	239,202	1.17	$1,881.88	$1,870.25	$1,776.30	6.20
22	115,901	0.86	3,213	141,659	257,561	1.22	$1,945.99	$1,933.04	$1,830.71	6.53
23	122,042	0.86	3,375	156,510	278,551	1.28	$2,009.03	$1,993.66	$1,882.35	6.88
24	125,200	0.86	3,459	165,250	290,450	1.32	$2,040.18	$2,024.30	$1,907.44	7.05
25	132,078	0.86	3,638	183,957	316,036	1.39	$2,105.64	$2,089.34	$1,959.42	7.44
26	132,930	0.86	3,655	184,801	317,731	1.39	$2,111.57	$2,095.27	$1,963.64	7.49
27	137,500	0.85	3,759	193,735	331,235	1.41	$2,144.86	$2,128.57	$1,986.05	7.75
28	144,391	0.84	3,919	210,163	354,554	1.46	$2,193.47	$2,176.65	$2,016.78	8.13
29	153,630	0.84	4,134	234,369	387,999	1.53	$2,254.69	$2,234.32	$2,055.52	8.66
30	160,019	0.83	4,284	252,052	412,071	1.58	$2,294.62	$2,272.44	$2,078.63	9.02
31	165,001	0.83	4,395	265,060	430,061	1.61	$2,322.76	$2,300.03	$2,095.78	9.30
32	176,712	0.82	4,681	306,987	483,699	1.74	$2,389.27	$2,363.58	$2,129.42	9.96
33	182,422	0.82	4,823	329,206	511,628	1.80	$2,420.67	$2,393.00	$2,148.21	10.28
34	185,674	0.82	4,888	336,182	521,856	1.81	$2,433.17	$2,404.16	$2,150.94	10.46
35	193,174	0.81	5,057	361,552	554,726	1.87	$2,463.98	$2,433.12	$2,157.85	10.88
36	203,236	0.81	5,288	399,223	602,460	1.96	$2,502.25	$2,470.15	$2,168.75	11.45
37	211,615	0.81	5,490	436,354	647,969	2.06	$2,532.91	$2,498.19	$2,170.78	11.92
38	220,186	0.80	5,687	472,539	692,725	2.15	$2,558.97	$2,521.12	$2,169.74	12.40
39	222,437	0.80	5,731	478,755	701,192	2.15	$2,564.32	$2,526.01	$2,168.73	12.53
40	227,994	0.80	5,859	504,610	732,604	2.21	$2,577.60	$2,538.18	$2,161.44	12.84
41	234,361	0.80	5,995	529,940	764,301	2.26	$2,590.57	$2,550.42	$2,152.63	13.20
42	237,972	0.80	6,090	553,553	791,525	2.33	$2,598.68	$2,557.75	$2,145.15	13.41
43	246,271	0.79	6,276	593,831	840,102	2.41	$2,612.24	$2,568.47	$2,127.26	13.87
44	253,639	0.79	6,442	630,427	884,066	2.49	$2,622.48	$2,576.11	$2,110.41	14.29
45	257,028	0.79	6,516	647,721	904,748	2.52	$2,625.72	$2,579.07	$2,098.71	14.48
46	259,170	0.79	6,563	658,747	917,918	2.54	$2,627.22	$2,580.39	$2,090.66	14.60
47	261,898	0.79	6,617	670,643	932,541	2.56	$2,628.14	$2,581.09	$2,080.46	14.75
48	265,472	0.79	6,714	700,646	966,118	2.64	$2,629.54	$2,582.20	$2,060.86	14.96
49	267,434	0.79	6,757	711,493	978,927	2.66	$2,629.82	$2,582.40	$2,053.86	15.07
50	268,955	0.79	6,796	724,828	993,783	2.69	$2,629.23	$2,581.65	$2,043.55	15.15
51	270,128	0.79	6,824	733,584	1,003,712	2.72	$2,628.34	$2,580.55	$2,036.45	15.22
52	275,395	0.79	6,952	776,204	1,051,600	2.82	$2,623.42	$2,573.58	$2,001.19	15.52
53	277,411	0.78	6,992	787,454	1,064,865	2.84	$2,621.19	$2,570.34	$1,989.47	15.63
54	278,180	0.78	7,012	795,162	1,073,343	2.86	$2,619.82	$2,568.43	$1,981.92	15.67
55	278,304	0.78	7,014	795,653	1,073,956	2.86	$2,619.64	$2,568.20	$1,981.43	15.68
56	278,337	0.78	7,015	795,855	1,074,192	2.86	$2,619.59	$2,568.13	$1,981.24	15.68
57	281,900	0.78	7,105	831,474	1,113,374	2.95	$2,610.84	$2,556.57	$1,947.56	15.88
58	281,917	0.78	7,105	831,515	1,113,432	2.95	$2,610.82	$2,556.54	$1,947.42	15.88
59	282,278	0.78	7,112	833,339	1,115,617	2.95	$2,610.08	$2,555.56	$1,945.42	15.90

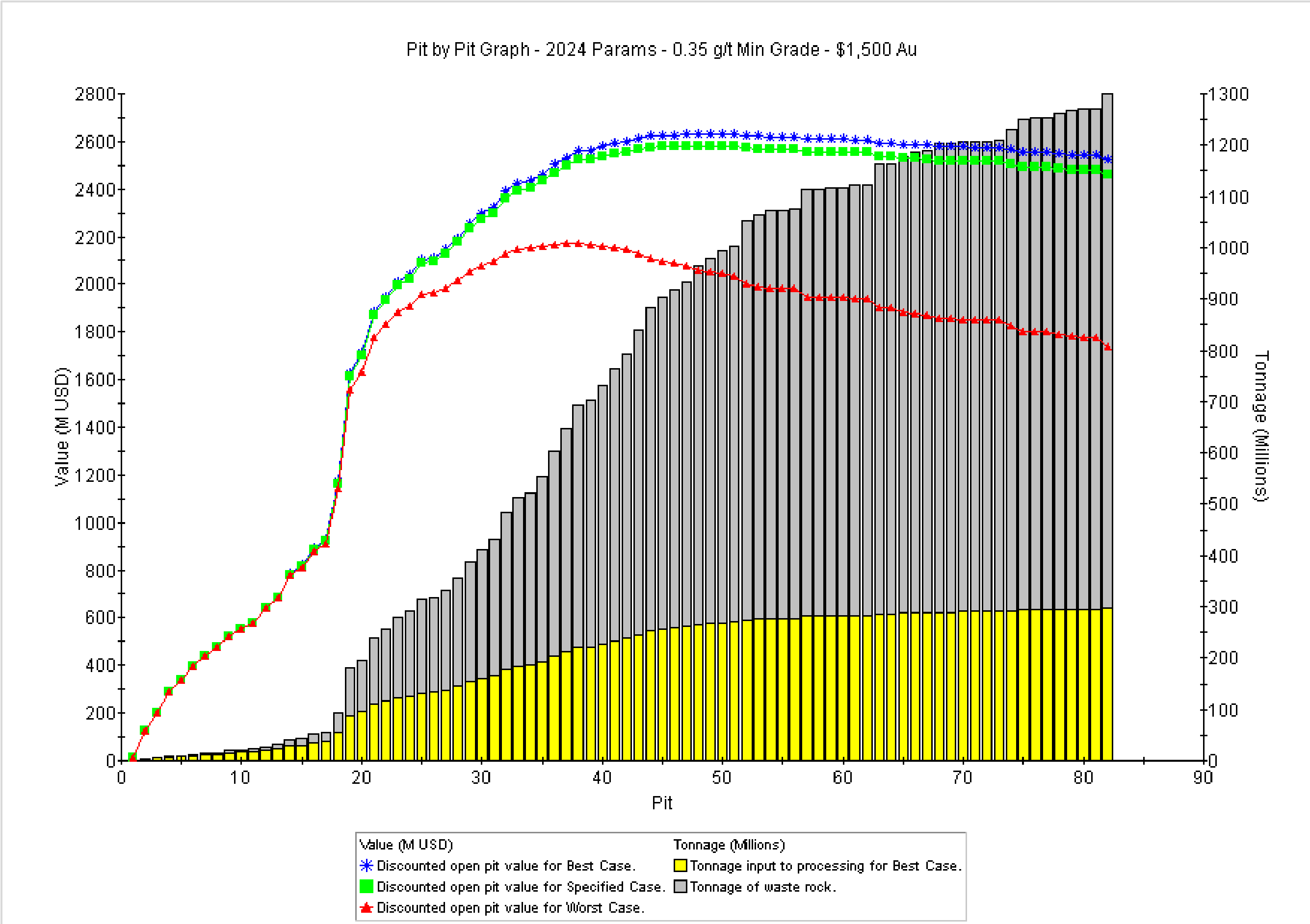


Figure 15-2: Graph of Whittle Results

Pit shell 49 is highlighted in Table 15-5 as the pit that creates the highest discounted operating cash flow for the specified case, and pit shell 49 is highlighted in Table 15-4 as the pit shell that was created using a US$1,500 gold price and was used as the basis for the ultimate pit design.

15.2 Pit Designs

Detailed pit designs were completed, including an ultimate pit and three 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 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 were created to use six-meter benches for mining. This corresponds to the resource model block heights, and RESPEC 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, 12-meter benches may be mined.

15.2.2 Pit Design Slopes

The 2014 PFS slope parameters were based on geotechnical studies by Golder Associates and Ken Rippere (Golder, September 13, 2011). These were reviewed by R. Barkley of Call & Nicholas, 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 28m wide (total width of 50m), so that a berm could be placed, and rock would collect at the base of the slope behind the berm.

Barkley (2016) commented that "for interim phases, assuming a 47- to 50-degree bedding dip the inter-ramp 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 inter-ramp slopes between the ramps".

The recommended slopes are developed around five different sectors in fresh rock and three sectors (Sectors 6,7, and 8) in weathered rock as shown in Table 15-5**.** The design parameters used are shown in Table 15-5 for the ultimate pit and Table 15-6 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 inter-ramp angles also in degrees (IRA).

Table 15-5: Pit Slope Design 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 greater.

Table 15-6: 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 utilize switchbacks to maintain the ramp system on the eastern side of the pit. This was done to better match the dip of the deposit and allows better traffic connectivity between pit phases. In areas where switchbacks are employed, a maximum centerline gradient of 8% was used.

Ramp width was determined as a function of the largest haul truck width to be used. Mine plans use 227-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. RESPEC 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 haul truck tire height plus 10%. 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 pits, 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 pits, 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 eastern 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 seven switchbacks in the ultimate pit design.

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

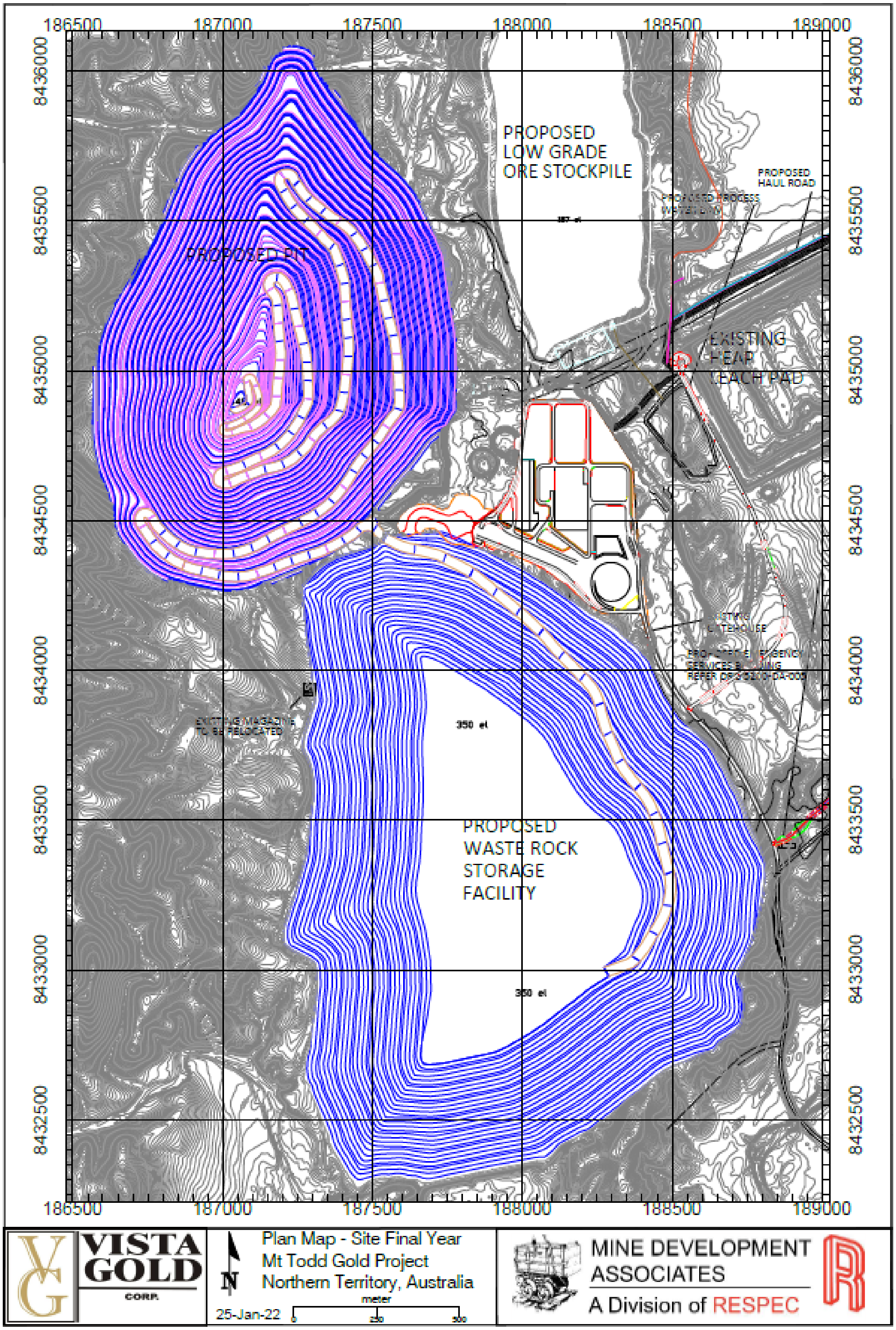


Figure 15-3: Mt Todd Ultimate Pit Design

15.2.5 Pit Phasing

Phase 1 was designed to continue mining on 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.

Figure 15-4 and Figure 15-5 show the Phase 1 and Phase 2 designs. Figure 15-6 shows Phase 3 pit design, while the ultimate pit is depicted in Figure 15-3.

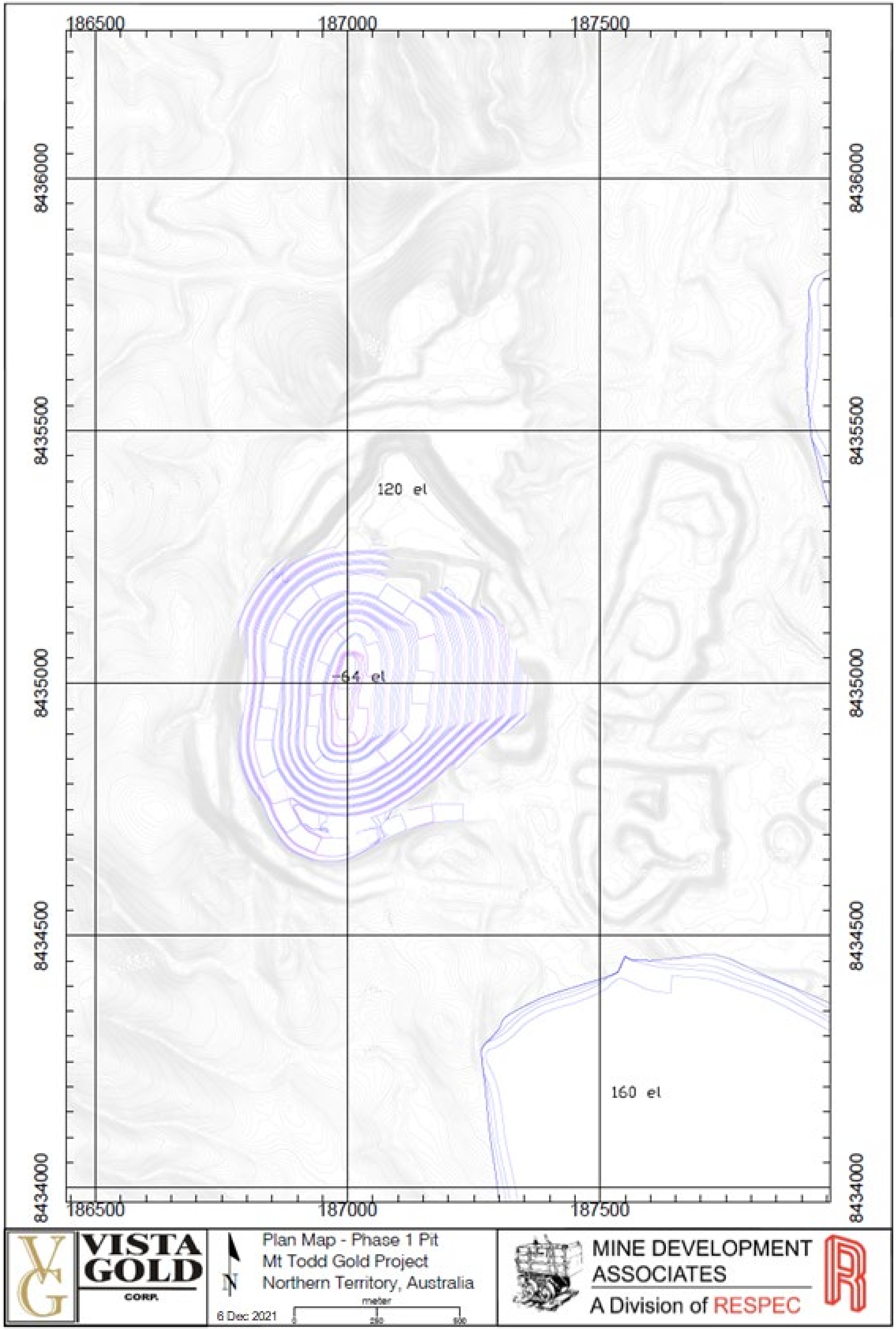


Figure 15-4: Phase 1 Design

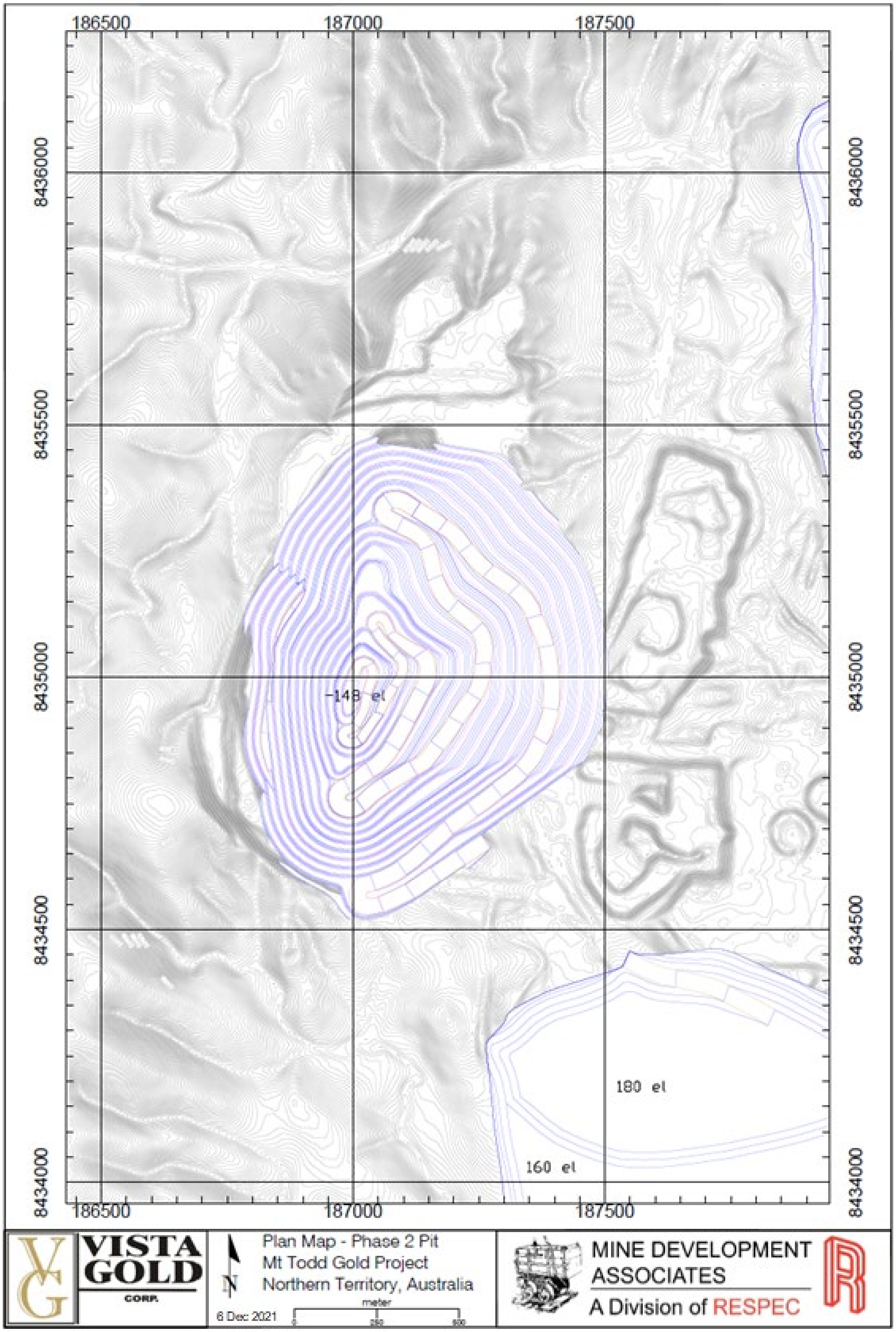


Figure 15-5: Phase 2 Design

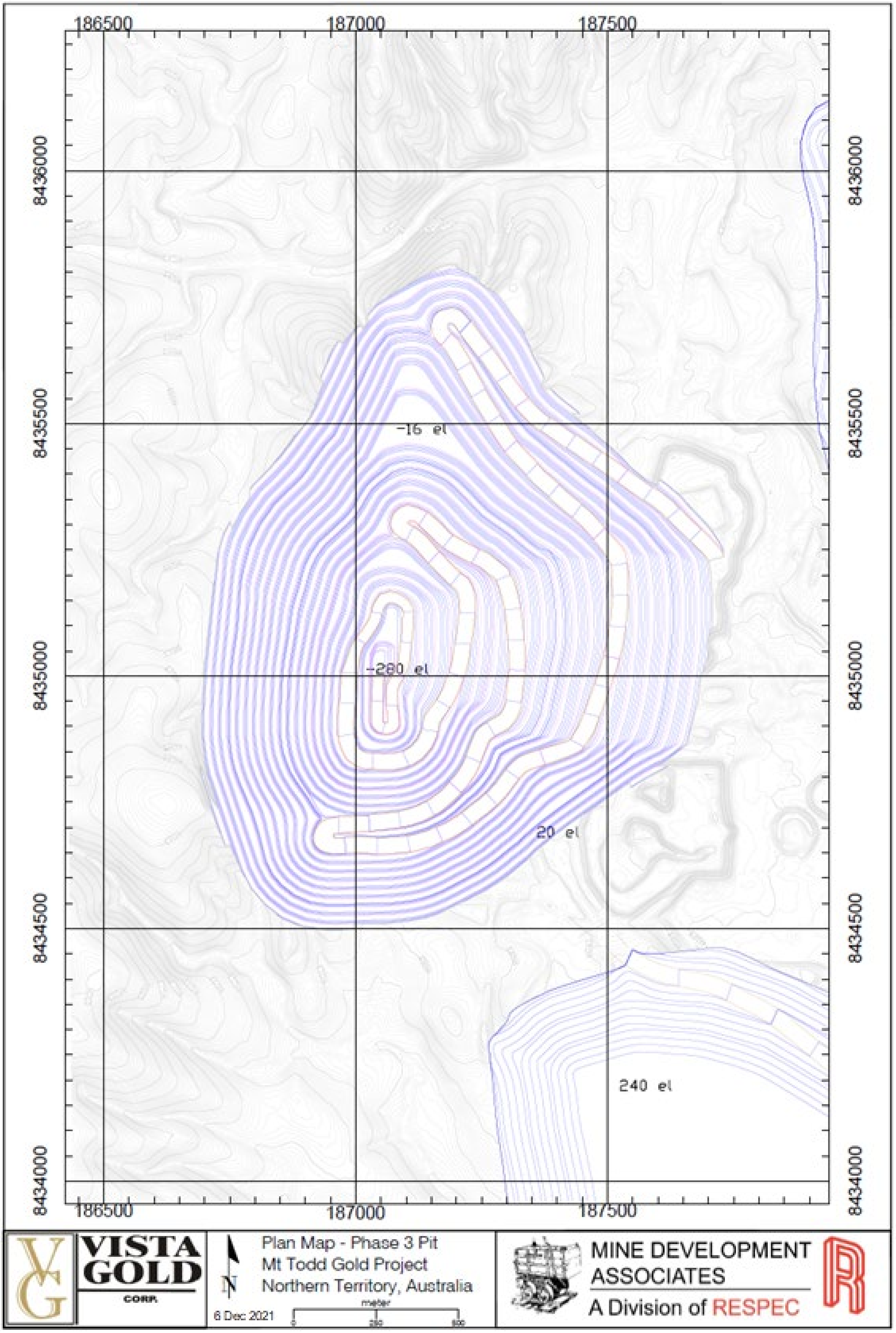


Figure 15-6: Phase 3 Design

15.3 Cutoff Grade

The breakeven cutoff grade is calculated to be 0.31 g Au/t using a US$1,500 per ounce gold price or 0.27 g Au/t using a US$1,750 gold price. To enhance project economics, Vista has decided to use an elevated cutoff grade for reserves and scheduling. Reserves are reported using 0.35 g Au/t cutoff grade.

For purposes of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used the reserve cutoff grade (0.35 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. RESPEC 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) to define the economically extractable portions of the estimated resources. RESPEC developed the reserves to be in accordance with NI 43-101, which is based on the CIM Definitions 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. In contrast, coal reserves have traditionally been reported as tonnes of "clean coal". In this coal example, reserves are reported as a "saleable product" reference point and include reductions for plant yield (recovery). 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 mineral resources in the pit designs. RESPEC 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. 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 by RDi. The heap-leach reserves are shown with the Batman reserves in Table 15-8 .

The pit phases are shown to be economically viable based on cash flows provided by Tetra Tech. RESPEC has reviewed the cash flows and believes that they are reasonable for the statement of Proven and Probable reserves.

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

	Proven			Probable			Total P&P			Waste	Total	Strip
	Tonnes (000s)	g Au/t	K Ozs Au	Tonnes (000s)	g Au/t	K Ozs Au	Tonnes (000s)	g Au/t	K Ozs Au	Tonnes (000s)	Tonnes (000s)	Ratio
Ph_1	14,501	1.04	485	6,817	1.04	227	21,318	1.04	712	17,788	39,107	0.83
Ph_2	20,428	0.77	508	20,906	0.83	561	41,334	0.80	1,069	55,822	97,156	1.35
Ph_3	31,713	0.83	842	54,080	0.79	1,369	85,793	0.80	2,211	181,730	267,524	2.12
Ph_4	14,634	0.76	358	103,941	0.72	2,397	118,575	0.72	2,755	415,990	534,565	3.51
Total	**81,277**	**0.84**	**2,192**	**185,744**	**0.76**	**4,555**	**267,021**	**0.79**	**6,747**	**671,331**	**938,352**	**2.51**

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 0.35 g Au/t cutoff grade.
3) 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			Heap Leach Pad			Total P&P		
	K Tonnes	g Au/t	K Ozs Au	Tonnes	g Au/t	K Ozs Au	K Tonnes	g Au/t	K Ozs Au
Proven	81,277	0.84	2,192	-	-	-	81,277	0.84	2,192
Probable	185,744	0.76	4,555	13,354	0.54	232	199,098	0.75	4,787
Proven & Probable	267,021	0.79	6,747	13,354	0.54	232	280,375	0.77	6,979

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 0.35 g Au/t cutoff grade.
3) Deepak Malhotra is the QP responsible for reporting the heap-leach pad reserves.
4) Because all 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 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. MINING METHODS

This section is based on 50,000 tpd operation.

16.1 Methods

The Mt Todd Batman 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.

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 Type Characterization

The WRD is intended to store waste material in perpetuity. The waste materials include potentially acid forming (PAF) and non-acid forming (NAF) materials. These materials were identified based on ICP analyses of samples from drill holes. PAF material is classified based on the percent of sulfur that is calculated from the ICP analyses. The sulfide percent was estimated into the resource block model by Tetra Tech. This sulfide estimate has been used to flag PAF material as any material that has a sulfide value greater than 0.25%. Material less than or equal to 0.25% has been flagged as NAF.

Some resource model blocks were not close enough to drilling to estimate the sulfur percent. Where blocks did not have an estimate, they were considered to be unknown. For the scheduling of waste material, unknown material was considered PAF material.

Table 16-2: Feasibility Waste Tonnages by Waste Type

Waste Type	K Tonnes	% of Total
NAF	249,986	37%
PAF	293,234	44%
Unknown	128,111	19%
Total Waste	**671,331**	

16.4 WRD Construction

RESPEC worked with other consultants to define the method of construction. The WRD will be constructed with NAF material being placed to the outside of the WRD and PAF along with unknown material would be placed on the inside of the WRD so that this material can be encapsulated in the ultimate dump.

The WRD design is shown in Figure 15-3. The WRD design is intended to push to the current water retention pond to the south and is bound by ridges to the east and west, and then by haul roads and pits to the north as shown in Figure 15-3. This waste dump is designed to cover the current waste dump on the site.

The ultimate design of the dump is intended to keep surface water from infiltrating into the dump for the long term. During operations, the water runoff requires capture and management as is currently the practice. The long-term closure of the WRD will require a cover that does not allow water to infiltrate into the facility. Accordingly, the WRD design uses geotextile material as an impermeable cover to the waste material. This cover will be included on each catch bench as well as over the entirety of the upper lift of the waste dump.

The anticipated top lift of the waste dump is currently 350m elevation but is sloped to the south by approximately 1% gradient to prevent potential ponding of surface water on the dump surface. The top lift on the southern side of the pit is 340m elevation. The ultimate height of the WRD on the north is 204m and the height on the southern end is 220m.

The WRD design is based on 10m lifts being placed with catch benches of 8m for every 30m of height. The lifts were designed using a 34˚ angle of repose. These design parameters were provided by geotechnical engineers and achieve an overall slope of 1.75 horizontal to 1 vertical.

In general, the lift height of 10m is to be used. This provides an economic short-term advantage of reducing the height that material is lifted before placement into the WRD. In addition, the compaction and stability of the WRD is improved due to haul trucks running over this material and compacting the material with the weight of the loaded haul trucks.

The outer lifts of the dumps will be constructed in the same manner using the NAF material. Where appropriate, the sorter rejects will be used to cushion the geotextiles. Oversize rock will be used to armor the outer portions of the outer shell of the waste dump. Note that in the early years, the mine will have minimal NAF material available. Initial NAF is placed to the north end of the dump, followed by placement of NAF around the southern perimeter. Once the perimeter of the dump is secured, then continual dumping of NAF around the perimeter will be done in advance of the core being dumped in place.

16.5 Operational Controls

To safeguard the environment, operational controls will be required. These operational controls should include:

- Waste Sampling Protocol;
- Geological Mapping;
- Waste Characterization; and
- Material Routing.

Waste sampling protocols will be required to ensure that different material types are identified and assigned. This requires proper sampling of material to ensure that the samples are taken in a representative manner. The basic sampling method will be to gather samples from blast hole drilling in the same manner that ore will be sampled. The samples will be analyzed using ICP analytical methods to estimate the amount of sulfide sulfur.

The sulfide sulfur in the ore is the primary source of PAF material. This can be identified to an extent through geological mapping to better determine areas that will be NAF material. In these areas the amount of sampling for ICP analyses can be reduced. A protocol for the sampling of the NAF areas should be developed that allows for more sampling during initial mining. As geological mapping of NAF areas becomes more confident, then the quantity of ICP samples in these areas can be reduced.

Waste characterization would be done similar to ore control. The sulfide sulfur values would be displayed on maps and a block model would be created to show where PAF and NAF material is located. Dig boundaries will be assigned that include the waste material types. The material type boundaries should be adjusted to provide a buffer around the PAF material to ensure PAF material is handled properly.

Material routing will be done by communication of the dig boundaries and tracking of material types being loaded into haul trucks in real time. The communication of the dig boundaries will be done in two ways. First, by marking out of dig boundaries in the field using colored pin flags and lathe with colored flagging in the field, along with distribution of dig maps showing the different material types in the field. The dig maps will be provided to supervisors and loading operators as a confirmation of the boundaries that are laid out in the field.

Second, high precision GPS units on loading equipment interconnected to a dispatch system will be used. The GPS units have become common place in mines around the world and have been proven to be of use. This provides a live map in the cab of the loading equipment so the operator can determine the type of material they are digging in and send that material accordingly. The GPS system has been integrated into a dispatch system that can detect where trucks are traveling. The dispatch system works with GPS waypoints along the routes of travel. The dispatch system will be used to detect any loads that are going to the incorrect destination. When incorrect locations are detected, a message will be sent to the operator along with the supervisor and a dispatch operator. These systems are well proven to ensure that the proper movement of material is maintained.

16.6 Mine-Waste Construction and Reclamation Requirements

NAF mine waste will be used for construction and final reclamation cover for the mine. The construction material will be used for tailings dam construction at the tailings storage facilities (TSF 1" and "TSF 2). Other NAF 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 hauled to a temporary stockpile near the sorter. This material is considered NAF. While sorter rejects are shown to be re-handled at the end of the mine life as part of the reclamation material, some of these rejects will be used along with other mined NAF waste material to construct the outer rings of the waste dump where appropriate. This allows for the encapsulation of PAF material in the center of the WRS as described in Section 16.4—WRD Construction.

Tetra Tech provided the amount of material that would be required to be mined for tailings construction and reclamation. These totals are shown in Table 16-3.

Table 16-3: 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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Total
Total TSF	K m^3	850	540	1,590	1,590	460	460	410	520	640	650	720	730	750	690	730	1,070	710	-	-	-	13,110
	K Tonnes	1,700	1,080	3,180	3,180	920	920	820	1,040	1,280	1,300	1,440	1,460	1,500	1,380	1,460	2,140	1,420	-	-	-	26,220
Reclamation Material Requirements																						
Sorter Reject to TSF 1	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12,108	12,108
Sorter Reject to TSF 2	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16,056	16,056
Total Sorter Reject Rehandle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	28,164	28,164
Tsf1_Closure	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	288	1,056	1,344
Tsf2_Closure	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	388	1,100	1,488
Total NAF	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	676	2,156	2,832

Volume calculations assume 40% swell factor and an average specific gravity of 2.67 (bank density) have been assumed for volume calculations.

16.7 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 feed the process plant and maintain stripping requirements for each case.

Production scheduling was done using MineSched (version 2021). This was summarized in Excel spreadsheets where additional waste re-handling was added to the schedule. Table 16-4 shows the mine production schedule, including re-handle from stockpiles, waste material re-handle, and sorter stockpile material movement requirements. For production scheduling, low-grade, medium-grade, and high-grade ore was designated. The low-grade cutoff was 0.35 g Au/t while the medium-grade and high-grade cutoffs were 0.55 g Au/t 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 will be stockpiled in the stockpile area northeast of the waste dump facility. Low-grade ore is to be processed as part of commissioning 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 (i.e., the commissioning and startup period) prior to start of full production. High-grade and medium-grade ore is processed in the mill when mill capacity becomes available and given priority over the processing of low-grade material.

For scheduling, three ore stockpiles are assumed:

- High-grade stockpile (>0.85 g Au/t);
- Medium-grade stockpile (0.55 to 0.85 g Au/t); and
- Low-grade stockpile (0.35 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 to feed the mill to full capacity as needed. 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 33.9 million tonnes.

Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. Table 16-5 shows the ore stockpile balances for the end of each year.

Ore sent to the mill is shown in Table 16-6. This 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 16-4: Annual Mine Production Schedule

			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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Total
Total Mined	*StkPl	K Tonnes	119	9,049	7,103	13,249	7,067	9,173	8,954	4,223	348	-	-	1,540	1,902	362	-	-	-	-	-	-	-	63,090
		g Au/t	0.60	0.68	0.46	0.53	0.44	0.45	0.48	0.48	0.48	-	-	0.47	0.48	0.49	-	-	-	-	-	-	-	0.51
		K Ozs Au	2	199	105	224	100	133	137	66	5	-	-	23	29	6	-	-	-	-	-	-	-	1,031
	Crusher	K Tonnes	-	10,037	11,185	16,606	6,914	13,958	17,823	15,141	10,156	10,823	14,194	16,136	17,774	14,553	12,568	11,489	4,574	-	-	-	-	203,931
		g Au/t	-	1.20	0.98	1.04	0.86	0.88	1.03	1.03	0.83	0.58	0.58	0.63	0.72	0.83	0.93	0.97	0.99	-	-	-	-	0.87
		K Ozs Au	-	386	351	557	191	393	591	502	272	201	266	327	410	387	377	359	146	-	-	-	-	5,716
	Total Ore Mined	K Tonnes	119	19,087	18,287	29,854	13,982	23,132	26,777	19,364	10,504	10,823	14,194	17,676	19,677	14,915	12,568	11,489	4,574	-	-	-	-	267,021
		g Au/t	0.60	0.95	0.78	0.81	0.65	0.71	0.85	0.91	0.82	0.58	0.58	0.62	0.69	0.82	0.93	0.97	0.99	-	-	-	-	0.79
		K Ozs Au	2	585	456	781	292	525	728	568	277	201	266	350	439	393	377	359	146	-	-	-	-	6,747
	NonPag_Wst	K Tonnes	1,308	4,034	17,047	16,979	38,790	17,813	29,736	39,705	36,586	24,029	13,725	7,443	2,539	206	43	2	-	-	-	-	-	249,986
	Pag_Wst	K Tonnes	885	13,678	14,524	18,956	22,635	30,902	31,672	23,303	24,082	28,543	28,848	26,161	17,492	7,329	2,888	1,132	206	-	-	-	-	293,234
	Un_Wst	K Tonnes	684	3,374	7,558	7,119	11,886	14,005	11,501	12,893	13,616	12,816	12,217	11,682	7,057	1,501	197	5	-	-	-	-	-	128,111
	Total Waste Mined	**K Tonnes**	**2,876**	**21,087**	**39,130**	**43,054**	**73,310**	**62,720**	**72,908**	**75,901**	**74,284**	**65,388**	**54,791**	**45,286**	**27,087**	**9,036**	**3,127**	**1,139**	**206**	**-**	**-**	**-**	**-**	**671,331**
	Total Tonnes Mined	**K Tonnes**	**2,995**	**40,173**	**57,417**	**72,909**	**87,292**	**85,852**	**99,685**	**95,265**	**84,788**	**76,210**	**68,985**	**62,962**	**46,764**	**23,951**	**15,695**	**12,628**	**4,780**	**-**	**-**	**-**	**-**	**938,352**
	Strip Ratio	W:O	24.25	1.10	2.14	1.44	5.24	2.71	2.72	3.92	7.07	6.04	3.86	2.56	1.38	0.61	0.25	0.10	0.05					2.51
Re-Handle Material	HG_StkPl	K Tonnes	-	352	2,201	633	-	-	-	184	184	-	-	-	-	83	83	-	-	-	-	-	-	3,721
		g Au/t	-	1.30	1.17	1.17	-	-	-	1.20	1.20	-	-	-	-	1.09	1.09	-	-	-	-	-	-	1.18
		K Ozs Au	-	15	83	24	-	-	-	7	7	-	-	-	-	3	3	-	-	-	-	-	-	141
	MG_StkPl	K Tonnes	-	830	515	512	2,698	-	-	523	523	-	-	-	-	202	202	-	-	-	-	-	-	6,004
		g Au/t	-	0.69	0.66	0.80	0.63	-	-	0.71	0.71	-	-	-	-	0.70	0.70	-	-	-	-	-	-	0.67
		K Ozs Au	-	18	11	13	55	-	-	12	12	-	-	-	-	5	5	-	-	-	-	-	-	130
	LG_StkPl	K Tonnes	-	1,115	3,848	-	8,187	3,792	-	1,902	6,912	6,952	3,556	1,614	-	2,936	4,897	4,808	2,847	-	-	-	-	53,365
		g Au/t	-	0.49	0.48	-	0.50	0.47	-	0.52	0.45	0.42	0.40	0.40	-	0.41	0.40	0.40	0.40	-	-	-	-	0.44
		K Ozs Au	-	18	59	-	132	58	-	32	101	94	46	21	-	39	64	62	36	-	-	-	-	759
	Leach Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,454	6,677	5,223	-	-	-	13,354
		g Au/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.54	0.54	0.54	-	-	-	0.54
		K Ozs Au	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	25	116	91	-	-	-	232
	Total Re-Handle	**K Tonnes**	**-**	**2,296**	**6,565**	**1,144**	**10,884**	**3,792**	**-**	**2,609**	**7,619**	**6,952**	**3,556**	**1,614**	**-**	**3,222**	**5,182**	**6,261**	**9,524**	**5,223**	**-**	**-**	**-**	**76,444**
		g Au/t	**-**	**0.69**	**0.72**	**1.00**	**0.53**	**0.47**	**-**	**0.61**	**0.49**	**0.42**	**0.40**	**0.40**	**-**	**0.44**	**0.43**	**0.43**	**0.50**	**0.54**	**-**	**-**	**-**	**0.51**
		K Ozs Au	**-**	**51**	**153**	**37**	**187**	**58**	**-**	**51**	**120**	**94**	**46**	**21**	**-**	**46**	**71**	**87**	**152**	**91**	**-**	**-**	**-**	**1,262**
	Waste Re-handle	K Tonnes	-	0	-	668	-	-	-	-	-	-	-	-	585	1,234	1,377	1,798	1,780	710	-	1,078	1,078	10,308
	Sorter Rejects	K Tonnes	-	1,233	1,775	1,775	1,780	1,775	1,782	1,775	1,777	1,777	1,775	1,775	1,777	1,777	1,775	1,630	742	-	-	-	-	26,702
	Sorter Reject Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14,082	14,082	28,164

Table 16-5: Annual Stockpile Balance

		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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20
Hg_StkPl	Added	K Tonnes	16	2,538	41	592	-	57	205	106	-	-	-	66	83	17	-	-	-	-	-	-	-
		g Au/t	0.99	1.19	1.09	1.17	-	1.13	1.23	1.19	-	-	-	1.10	1.09	1.06	-	-	-	-	-	-	-
		K Ozs Au	0	97	1	22	-	2	8	4	-	-	-	2	3	1	-	-	-	-	-	-	-
	Removed	K Tonnes	-	352	2,201	633	-	-	-	184	184	-	-	-	-	83	83	-	-	-	-	-	-
		g Au/t	-	1.30	1.17	1.17	-	-	-	1.20	1.20	-	-	-	-	1.09	1.09	-	-	-	-	-	-
		K Ozs Au	-	15	83	24	-	-	-	7	7	-	-	-	-	3	3	-	-	-	-	-	-
	Balance	K Tonnes	16	2,201	41	-	-	57	263	184	-	-	-	66	149	83	-	-	-	-	-	-	-
		g Au/t	0.99	1.17	1.09	-	-	1.13	1.21	1.20	-	-	-	1.10	1.09	1.09	-	-	-	-	-	-	-
		K Ozs Au	0	83	1	-	-	2	10	7	-	-	-	2	5	3	-	-	-	-	-	-	-
Mg_StkPl	Added	K Tonnes	46	1,299	357	2,853	-	139	585	321	-	-	-	152	202	50	-	-	-	-	-	-	-
		g Au/t	0.70	0.67	0.67	0.66	-	0.72	0.71	0.71	-	-	-	0.70	0.70	0.70	-	-	-	-	-	-	-
		K Ozs Au	1	28	8	60	-	3	13	7	-	-	-	3	5	1	-	-	-	-	-	-	-
	Removed	K Tonnes	-	830	515	512	2,698	-	-	523	523	-	-	-	-	202	202	-	-	-	-	-	-
		g Au/t	-	0.69	0.66	0.80	0.63	-	-	0.71	0.71	-	-	-	-	0.70	0.70	-	-	-	-	-	-
		K Ozs Au	-	18	11	13	55	-	-	12	12	-	-	-	-	5	5	-	-	-	-	-	-
	Balance	K Tonnes	46	515	357	2,698	-	139	724	523	-	-	-	152	354	202	-	-	-	-	-	-	-
		g Au/t	0.70	0.66	0.67	0.63	-	0.72	0.71	0.71	-	-	-	0.70	0.70	0.70	-	-	-	-	-	-	-
		K Ozs Au	1	11	8	55	-	3	16	12	-	-	-	3	8	5	-	-	-	-	-	-	-
Lg_StkPl	Added	K Tonnes	57	5,213	6,705	9,804	7,067	8,977	8,163	3,795	348	-	-	1,322	1,617	295	-	-	-	-	-	-	-
		g Au/t	0.42	0.44	0.45	0.45	0.44	0.44	0.44	0.44	0.48	-	-	0.42	0.42	0.42	-	-	-	-	-	-	-
		K Ozs Au	1	74	96	141	100	127	116	54	5	-	-	18	22	4	-	-	-	-	-	-	-
	Removed	K Tonnes	-	1,115	3,848	-	8,187	3,792	-	1,902	6,912	6,952	3,556	1,614	-	2,936	4,897	4,808	2,847	-	-	-	-
		g Au/t	-	0.49	0.48	-	0.50	0.47	-	0.52	0.45	0.42	0.40	0.40	-	0.41	0.40	0.40	0.40	-	-	-	-
		K Ozs Au	-	18	59	-	132	58	-	32	101	94	46	21	-	39	64	62	36	-	-	-	-
	Balance	K Tonnes	57	4,155	7,011	16,816	15,696	20,882	29,045	30,939	24,375	17,423	13,868	13,576	15,193	12,552	7,655	2,847	-	-	-	-	-
		g Au/t	0.42	0.43	0.42	0.44	0.40	0.41	0.42	0.41	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	-	-	-	-	-
		K Ozs Au	1	57	95	236	204	274	390	412	317	223	178	175	196	162	98	36	-	-	-	-	-
All StkPl	Balance	K Tonnes	119	6,871	7,409	19,513	15,696	21,078	30,031	31,646	24,375	17,423	13,868	13,794	15,696	12,837	7,655	2,847	-	-	-	-	-
		g Au/t	0.60	0.68	0.44	0.46	0.40	0.41	0.43	0.42	0.40	0.40	0.40	0.41	0.42	0.41	0.40	0.40	-	-	-	-	-
		K Ozs Au	2	151	104	291	204	279	417	431	317	223	178	180	210	169	98	36	-	-	-	-	-

Table 16-6: Annual Ore Delivery to the Mill Crusher

		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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Total
Sulfide Ore	K Tonnes	3,532	17,645	17,090	16,980	17,106	17,274	17,711	17,673	17,426	17,534	17,571	17,750	17,799	17,510	17,571	11,135	-	-	-	-	-	259,305
	g Au/t	0.62	1.18	0.85	0.94	0.62	0.96	1.07	0.87	0.50	0.53	0.56	0.65	0.78	0.74	0.84	0.70	-	-	-	-	-	0.79
	K Ozs Au	71	667	467	514	342	534	611	495	282	300	319	373	446	417	474	249	-	-	-	-	-	6,560
	Recovery	91%	93%	92%	92%	91%	92%	92%	92%	89%	90%	90%	91%	91%	91%	92%	91%	0%	0%	0%	0%	0%	91%
	K Ozs Au Rec	64	617	428	473	310	491	563	454	252	269	287	339	408	380	434	226	-	-	-	-	-	5,996
Mixed Ore	K Tonnes	-	52	380	688	496	170	39	67	252	144	120	-	-	161	120	3,621	-	-	-	-	-	6,310
	g Au/t	-	0.59	0.73	0.70	0.49	0.50	0.53	0.52	0.40	0.40	0.40	-	-	0.40	0.40	0.99	-	-	-	-	-	0.80
	K Ozs Au	-	1	9	15	8	3	1	1	3	2	2	-	-	2	2	115	-	-	-	-	-	163
	Recovery	0%	91%	91%	91%	90%	90%	90%	90%	88%	87%	87%	0%	0%	87%	87%	92%	0%	0%	0%	0%	0%	92%
	K Ozs Au Rec	-	1	8	14	7	2	1	1	3	2	1	-	-	2	1	106	-	-	-	-	-	149
Oxidized Ore	K Tonnes	-	53	281	82	196	306	-	11	121	71	59	-	-	79	59	86	-	-	-	-	-	1,406
	g Au/t	-	0.65	0.78	0.64	0.48	0.49	-	0.57	0.41	0.40	0.40	-	-	0.40	0.40	0.40	-	-	-	-	-	0.53
	K Ozs Au	-	1	7	2	3	5	-	0	2	1	1	-	-	1	1	1	-	-	-	-	-	24
	Recovery	0%	91%	91%	91%	90%	89%	0%	91%	88%	88%	88%	0%	0%	88%	88%	88%	0%	0%	0%	0%	0%	90%
	K Ozs Au Rec	-	1	6	2	3	4	-	0	1	1	1	-	-	1	1	1	-	-	-	-	-	22
Total	K Tonnes	3,532	17,750	17,750	17,750	17,799	17,750	17,750	17,750	17,799	17,750	17,750	17,750	17,799	17,750	17,750	14,843	-	-	-	-	-	267,021
	g Au/t	0.62	1.17	0.85	0.93	0.62	0.95	1.07	0.87	0.50	0.53	0.56	0.65	0.78	0.74	0.83	0.76	-	-	-	-	-	0.79
	K Ozs Au	71	669	483	531	353	542	611	496	287	302	321	373	446	420	476	365	-	-	-	-	-	6,747
	Recovery	91%	93%	92%	92%	91%	92%	92%	92%	89%	90%	90%	91%	91%	91%	92%	91%	0%	0%	0%	0%	0%	91%
	K Ozs Au Rec	64	619	443	488	320	498	564	455	257	271	289	339	408	383	436	333	-	-	-	-	-	6,167

16.8 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 29-cubic meter hydraulic shovels along with 227-tonne haul trucks, though final equipment selection may differ.

Secondary mine production is to be achieved using 18-cubic meter loaders along with the 227-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 16-7 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. For this purpose, an 87.5% schedule efficiency was used to model standby time for breaks, shift startup, and shift shut down.

In-pit and ex-pit centerlines were drawn for each of the pits and destinations, including the waste dump, crusher, and ore stockpile. 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 also estimated using 83% efficiency, 87.5% schedule 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 decrease 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 16-8.

Table 16-7: Maximum Loader Productivity Estimate

Description	Unit	All Rock	
Material Properties			
Material SG (BCM)	t/m^3 (Wet)	2.70	
Material SG (Loose)	t/m^3 (Wet)	1.93	
Material SG (BCM Dry)	t/m^3 (Dry)	2.50	
Material SG (LCM Dry)	t/m^3 (Dry)	1.79	
Swell Factor		1.4	

Description	Unit	All Rock	
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	

	Units	29 m3 Hyd 227 t Trucks	18 m3 FEL 227 t Trucks
Loading Parameters			
Shovel Mech. Avail.	%	85%	85%
Operating Efficiency	%	83%	83%
Bucket Capacity	m^3	31	19
Bucket Fill Factor	%	95%	95%
Avg. Cycle Time	Sec	34	50
Truck Parameters			
Truck Mech. Avail.	%	85%	85%
Operating Efficiency	%	83%	83%
Volume Capacity	m^3	176	176
Tonnage Capacity	Loose t (Wet)	227	227
Truck Spot Time	Sec	24	24
Shovel Productivity			
Effective Bucket Capacity	Cyd	29.45	18.05
Tonnes per Pass – Wet	Loose t (Wet)	56.8	34.8
Tonnes per Pass – Dry	Loose t (Dry)	52.6	32.2
Theoretical Passes – Vol	passes	5.98	9.75
Theoretical Passes – Wt	passes	4.00	6.52
Actual Passes Used	passes	4.0	7.0
Truck Tonnage – Wet	Wet tload	227	227
Truck Tonnage – Dry	Dry t/load	210	210
Truck Capacity Utilized – Vol	%	67%	67%
Truck Capacity Utilized – Wt	%	100%	100%
Load Time	min	2.67	6.23
Theoretical Productivity	Dry t/hr	4,729	2,023
Tonnes per Operating Hour	Dry t/hr	3,930	1,680
Tonnes Per Day	**Dry/day**	**70,200**	**30,000**
Potential – 355 days/year	**t/year**	**24,921,000**	**10,650,000**

Table 16-8: Annual Load and Haul Equipment Requirements

	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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Total
Haulage Requirements																						
Productive Hours	Hrs	3,471	55,636	91,693	131,192	184,760	207,792	222,567	217,501	211,545	210,384	209,723	210,125	163,169	88,029	60,948	52,196	27,055	4,910	-	14,997	2,382,691
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	42%	0%	42%	
Operating Hours	Hrs	4,182	67,032	110,473	158,062	222,602	250,352	268,153	262,050	254,874	253,474	252,679	253,162	196,590	106,059	73,431	62,887	32,596	5,916	-	18,069	2,870,712
Number of Trucks	#	4.00	12.00	24.00	26.00	38.00	42.00	42.00	42.00	42.00	42.00	42.00	42.00	32.50	21.00	18.00	13.00	5.50	1.00	-	3.00	
Truck Availability	%	90%	90%	89%	89%	89%	88%	87%	86%	86%	85%	85%	85%	85%	85%	85%	85%	85%	43%	0%	43%	
Available Operating Hours	Hrs	6,144	68,396	115,417	158,183	230,592	255,460	264,638	263,809	262,358	261,048	259,916	259,807	201,241	130,104	111,346	80,416	34,040	6,203	-	18,558	3,006,235
Use of Available Hours	%	68%	98%	96%	100%	97%	98%	101%	99%	97%	97%	97%	97%	98%	82%	66%	78%	96%	95%			95%
Tonnes per Operating Hour	t/Hr	670	630	489	479	396	356	375	372	353	303	270	240	242	315	291	380	1,003	-	-	839	367
Hydraulic Shovel Useage																						
Number of Shovels	#	1	1	2	3	4	4	4	4	4	4	4	3	3	3	3	3	1	-	-	-	-
Availability	%	0%	90%	89%	89%	88%	88%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	0%	0%	0%
Operating Efficeincy	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%	0%	0%	0%
Available Operating Hrs	Op Hrs	-	11,421	15,685	19,303	22,459	25,379	37,489	49,865	49,717	49,600	49,487	43,301	37,168	37,168	37,115	30,929	24,778	12,407	-	-	-
Tonnes Mined	K Tonnes	2,995	40,173	56,008	71,435	84,686	80,234	136,714	173,382	154,314	134,301	120,505	113,332	84,175	43,113	49,193	39,656	4,589	-	-	-	-
Operating Hours	Op Hrs	763	10,235	14,269	18,199	21,575	20,441	34,830	44,171	39,314	34,215	30,700	28,873	21,445	10,984	12,533	10,103	1,169	-	-	-	-
Use of Available Operating Hours	%	0%	90%	91%	94%	96%	81%	93%	89%	79%	69%	62%	67%	58%	30%	34%	33%	5%	0%	0%	0%	0%
Front End Loaders																						
Number of Loaders	#	-	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	2	-
Availability	%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Operating Efficeincy	%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Available Operating Hrs	Op Hrs	-	3,303	9,779	12,881	12,771	12,571	18,541	24,744	24,778	24,778	24,744	24,744	24,778	24,778	24,744	24,744	25,523	13,151	-	12,372	-
Tonnes Mined	K Tonnes	-	352	7,974	3,287	13,490	9,410	12,975	22,365	30,499	32,023	24,577	15,820	10,522	13,701	(4,684)	1,718	27,580	11,867	-	30,320	-
Operating Hours	Op Hrs	-	209	4,748	1,957	8,033	5,604	7,727	13,319	18,162	19,070	14,636	9,421	6,266	8,159	(2,790)	1,023	16,424	7,067	-	18,056	-
Use of Available Operating Hours	%	0%	6%	49%	15%	63%	45%	42%	54%	73%	77%	59%	38%	25%	33%	-11%	4%	64%	54%	0%	146%	0%

16.9 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 16-10.

Salaries for each position were estimated based on information received from Tetra Tech and Vista Gold. Salaries include an allowance for benefits at a rate of 27% of the base salary for each position. The salaries used are shown in Table 16-9 presented in US dollars. The extended cost for labor by year is shown in thousands of US dollars in Table 16-11. Note that mobile equipment labor costs are allocated to production equipment in the calculation of mining costs in later sections.

Also note that the mine personnel tables do not include contractors. Vista anticipates using a Maintenance and Repair Contract (MARC) to maintain the mining fleet during the first three years. After that time, Vista will operate all maintenance crews. For costing, the MARC costs were reduced to take into account savings by reducing overhead of the contractor. 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.

Table 16-9: Mt Todd Personnel Salaries

	US $/Year		
	Labor Rates	Benefits 27%	Total Rate
Mine Overhead			
Mine Manager	$ 198,179	$ 53,508	$ 251,687
Mine Clerk	$ 82,634	$ 22,311	$ 104,945
Mine Shift Foremen	$ 93,899	$ 25,353	$ 119,252
Mine Trainer	$ 93,246	$ 25,176	$ 118,422
Blaster	$ 93,899	$ 25,353	$ 119,252
Blaster's Helper	$ 63,586	$ 17,168	$ 80,754
Mine Production			
Loading Operators	$ 86,506	$ 23,357	$ 109,863
Mechanics	$ 90,202	$ 24,355	$ 114,557
Welders	$ 93,899	$ 25,353	$ 119,252
Servicemen	$ 59,889	$ 16,170	$ 76,058
Haul Truck Operators	$ 75,415	$ 20,362	$ 95,777
Mechanics	$ 90,202	$ 24,355	$ 114,557
Welders	$ 93,899	$ 25,353	$ 119,252
Servicemen	$ 59,889	$ 16,170	$ 76,058
Drill Operators	$ 86,506	$ 23,357	$ 109,863
Mechanics	$ 90,202	$ 24,355	$ 114,557
Welders	$ 93,899	$ 25,353	$ 119,252
Servicemen	$ 59,889	$ 16,170	$ 76,058
Support Equipment Operators	$ 79,112	$ 21,360	$ 100,472
Mechanics	$ 90,202	$ 24,355	$ 114,557

	US $/Year		
	Labor Rates	Benefits 27%	Total Rate
Welders	$ 93,899	$ 25,353	$ 119,252
Servicemen	$ 59,889	$ 16,170	$ 76,058
Mine Maintenance			
Maintenance Superintendent	$ 131,607	$ 35,534	$ 167,141
Maintenance Foremen	$ 112,384	$ 30,344	$ 142,727
Light Vehicle Mechanics	$ 90,202	$ 24,355	$ 114,557
Tiremen	$ 67,282	$ 18,166	$ 85,448
Shop Laborers	$ 63,586	$ 17,168	$ 80,754
Maintenance Planner	$ 93,899	$ 25,353	$ 119,252
Service, Fuel, & Lube	$ 59,889	$ 16,170	$ 76,058
Engineering			
Chief Engineer	$ 131,342	$ 35,435	$ 166,678
Mine Surveyors	$ 94,786	$ 25,592	$ 120,378
Surveyor Helper	$ 65,621	$ 17,718	$ 83,339
Mine Engineer	$ 106,938	$ 29,873	$ 135,812
Mine Geology			
Chief Geologist	$ 135,611	$ 36,615	$ 172,225
Ore Control Geologist	$ 126,972	$ 34,283	$ 161,225
Sampler	$ 69,965	$ 18,890	$ 88,855

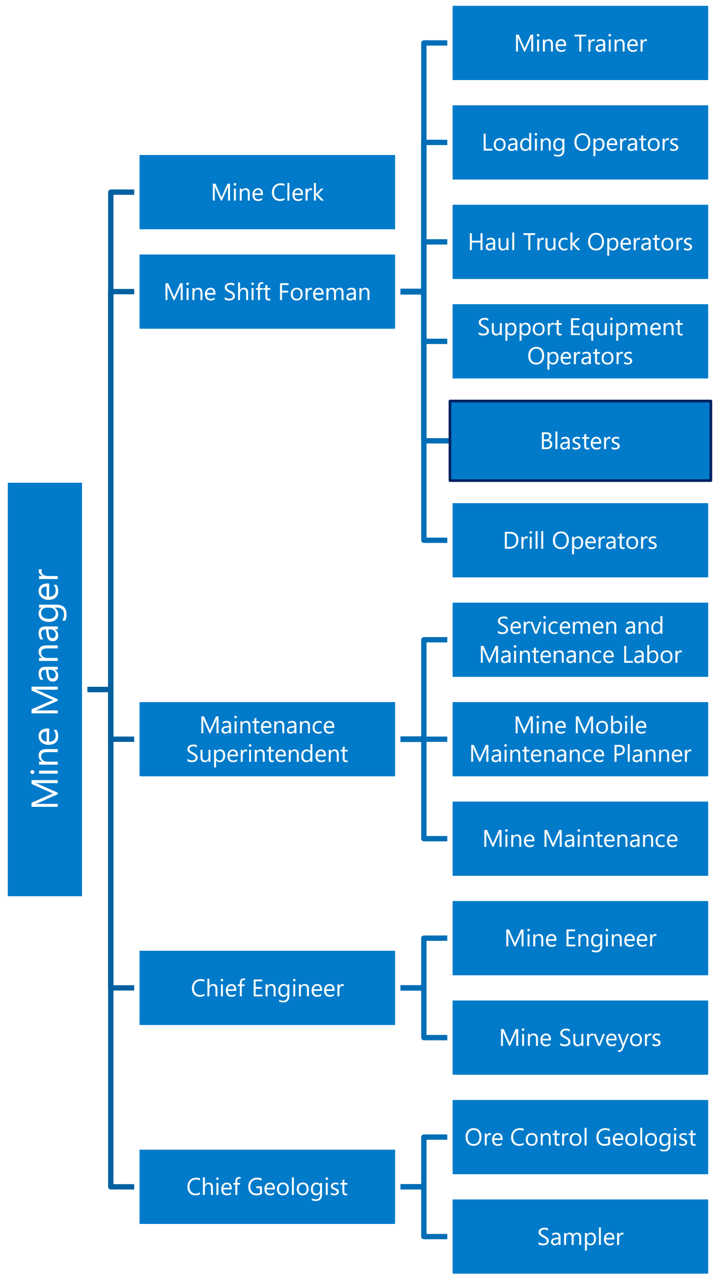


Figure 16-1: Mine Organizational Chart

Table 16-10: Mine Personnel Requirements

	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	Yr 15	Yr 16	Yr 17	Yr 18
Mine Overhead																			
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Mine Shift Foremen	6	9	9	9	12	12	12	12	12	12	12	12	9	6	6	6	6	1	1
Mine Trainer	1	1	1	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	2	2	2	2	2	-
Blaster's Helper	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Mine Production																			
Loading Operators	4	5	11	11	18	18	15	18	18	18	18	14	12	8	8	10	10	4	-
Haul Truck Operators	12	36	72	78	114	126	126	126	126	126	126	126	126	69	57	51	36	8	8
Drill Operators	7	26	42	42	43	45	51	48	41	35	35	33	33	19	11	11	7	-	-
Support Equipment Operators	15	18	21	21	21	21	24	24	24	24	24	21	18	12	12	12	12	12	-
Total Mine Operating	**51**	**101**	**162**	**168**	**215**	**229**	**235**	**235**	**228**	**222**	**222**	**213**	**205**	**121**	**101**	**97**	**78**	**30**	**9**
Mine Maintenance																			
Maintenance Superintendent	1	1	1	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	3	1	1	1	1	-
Light Vehicle Mechanics	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	-
Tiremen	2	2	2	2	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	2	2	2	-
Maintenance Planner	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Service, Fuel, & Lube	6	6	6	6	6	6	6	6	6	6	6	6	6	6	4	4	4	4	-
Total Mine Maintenance	**14**	**14**	**14**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**18**	**13**	**13**	**13**	**13**	**-**

	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	Yr 15	Yr 16	Yr 17	Yr 18
Engineering																			
Chief Engineer	1	1	1	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	2	2	2	2	2	1	-
Surveyor Helper	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Mine Engineer	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-
Total Engineering	**6**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**6**	**-**
Mine Geology																			
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Ore Control Geologist	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-
Sampler	2	2	2	2	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**	**5**	**5**	**5**	**5**	**-**	**-**
Total Mine Operations Workforce																			
Mine Operations	47	97	157	163	209	226	232	232	228	222	222	212	205	121	100	93	78	30	9
Mine Maintenance	14	14	14	18	18	18	18	18	18	18	18	18	18	18	13	13	13	13	-
Engineering	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	6	-
Geology	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	-	-
Total	**72**	**124**	**184**	**194**	**240**	**257**	**263**	**263**	**259**	**253**	**253**	**243**	**236**	**152**	**126**	**119**	**104**	**49**	**9**

Table 16-11: Mine Annual Personnel Costs ($000's USD)

	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	Yr 15	Yr 16	Yr 17	Yr 18	Total
Mine Overhead																				
Mine Manager	$85	$255	$255	$255	$256	$255	$256	$255	$256	$256	$255	$255	$256	$256	$255	$255	$128	$-	$-	$4,046
Mine Clerk	$36	$106	$106	$106	$107	$106	$107	$106	$107	$107	$106	$106	$107	$107	$106	$106	$53	$-	$-	$1,687
Mine Shift Foremen	$212	$967	$1,089	$1,089	$1,364	$1,452	$1,458	$1,452	$1,454	$1,454	$1,452	$1,270	$908	$727	$726	$726	$424	$61	$-	$18,282
Mine Trainer	$40	$120	$120	$120	$120	$120	$121	$120	$120	$120	$120	$120	$120	$120	$120	$120	$120	$60	$-	$2,024
Blaster	$81	$242	$242	$242	$243	$242	$243	$242	$242	$242	$242	$242	$242	$242	$242	$242	$121	$-	$-	$3,834
Blaster's Helper	$55	$164	$164	$164	$164	$164	$165	$164	$164	$164	$164	$164	$164	$164	$164	$164	$82	$-	$-	$2,596
Mine Production																				
Loading Operators	$-	$391	$819	$1,142	$1,717	$1,756	$1,679	$1,839	$2,009	$2,009	$1,755	$1,421	$1,088	$837	$752	$892	$781	$224	$-	$21,110
Haul Truck Operators	$268	$3,056	$5,165	$7,141	$10,446	$11,658	$12,293	$12,243	$12,260	$12,260	$12,243	$12,243	$9,483	$6,131	$5,247	$4,227	$2,139	$390	$-	$138,891
Drill Operators	$139	$2,344	$3,098	$4,178	$4,387	$4,652	$5,428	$4,960	$4,185	$3,850	$3,734	$3,622	$2,901	$1,619	$1,170	$1,003	$390	$-	$-	$51,661
Support Equipment Operators	$485	$1,403	$1,603	$1,988	$1,993	$1,988	$2,302	$2,446	$2,450	$2,450	$2,293	$1,988	$1,531	$1,225	$1,223	$1,223	$1,225	$613	$-	$30,429
Total Mine Operating	**$1,400**	**$9,048**	**$12,663**	**$16,426**	**$20,797**	**$22,394**	**$24,051**	**$23,827**	**$23,246**	**$22,911**	**$22,365**	**$21,432**	**$16,799**	**$11,427**	**$10,006**	**$8,958**	**$5,462**	**$1,347**	**$-**	**$274,561**
Mine Maintenance																				
Maintenance Superintendent	$57	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$170	$85	$-	$2,857
Maintenance Foremen	$-	$-	$-	$215	$436	$434	$436	$434	$435	$435	$434	$434	$435	$290	$145	$145	$145	$73	$-	$4,927
Light Vehicle Mechanics	$78	$232	$232	$232	$233	$232	$233	$232	$233	$233	$232	$232	$233	$175	$116	$116	$116	$58	$-	$3,451
Tiremen	$58	$173	$173	$173	$174	$173	$174	$173	$174	$174	$173	$173	$174	$174	$173	$173	$174	$87	$-	$2,921
Shop Laborers	$55	$164	$164	$164	$164	$164	$165	$164	$164	$164	$164	$164	$164	$164	$164	$164	$164	$82	$-	$2,761
Maintenance Planner	$40	$121	$121	$181	$243	$242	$243	$242	$242	$242	$242	$242	$242	$242	$242	$242	$242	$121	$-	$3,733
Service, Fuel, & Lube	$155	$463	$463	$463	$464	$463	$465	$463	$464	$464	$463	$463	$464	$386	$309	$309	$309	$155	$-	$7,183
Total Mine Maintenance	**$442**	**$1,323**	**$1,323**	**$1,599**	**$1,884**	**$1,879**	**$1,886**	**$1,879**	**$1,881**	**$1,881**	**$1,879**	**$1,879**	**$1,881**	**$1,601**	**$1,318**	**$1,318**	**$1,320**	**$661**	**$-**	**$27,834**
Engineering																				
Chief Engineer	$57	$169	$169	$169	$170	$169	$170	$169	$169	$169	$169	$169	$169	$169	$169	$169	$169	$85	$-	$2,849
Mine Surveyors	$41	$183	$244	$244	$245	$244	$245	$244	$245	$245	$244	$244	$245	$245	$244	$244	$183	$61	$-	$3,891
Surveyor Helper	$57	$169	$169	$169	$170	$169	$170	$169	$169	$169	$169	$169	$169	$169	$169	$169	$127	$42	$-	$2,764
Mine Engineer	$92	$344	$413	$413	$414	$413	$415	$413	$414	$414	$413	$413	$414	$414	$413	$413	$414	$207	$-	$6,849
Total Engineering	**$246**	**$865**	**$996**	**$996**	**$998**	**$996**	**$1,000**	**$996**	**$997**	**$997**	**$996**	**$996**	**$997**	**$997**	**$996**	**$996**	**$894**	**$396**	**$-**	**$16,353**
Mine Geology																				
Chief Geologist	$58	$175	$175	$175	$175	$175	$175	$175	$175	$175	$175	$175	$175	$175	$175	$175	$87	$-	$-	$2,769
Ore Control Geologist	$109	$327	$327	$327	$328	$327	$329	$327	$328	$328	$327	$327	$328	$328	$327	$327	$164	$-	$-	$5,185
Sampler	$60	$180	$180	$180	$181	$180	$181	$180	$181	$181	$180	$180	$181	$181	$180	$180	$90	$-	$-	$2,857
Total Geology	**$228**	**$682**	**$682**	**$682**	**$684**	**$682**	**$685**	**$682**	**$683**	**$683**	**$682**	**$682**	**$683**	**$683**	**$682**	**$682**	**$341**	**$-**	**$-**	**$10,810**
Total Mine Operations Workforce																				
Mine Operations	$1,400	$9,048	$12,663	$16,426	$20,797	$22,394	$24,051	$23,827	$23,246	$22,911	$22,365	$21,432	$16,799	$11,427	$10,006	$8,958	$5,462	$1,347	$-	$274,561
Mine Maintenance	$442	$1,323	$1,323	$1,599	$1,884	$1,879	$1,886	$1,879	$1,881	$1,881	$1,879	$1,879	$1,881	$1,601	$1,318	$1,318	$1,320	$661	$-	$27,834
Engineering	$246	$865	$996	$996	$998	$996	$1,000	$996	$997	$997	$996	$996	$997	$997	$996	$996	$894	$396	$-	$16,353
Geology	$228	$682	$682	$682	$684	$682	$685	$682	$683	$683	$682	$682	$683	$683	$682	$682	$341	$-	$-	$10,810
Total	**$2,316**	**$11,919**	**$15,664**	**$19,703**	**$24,364**	**$25,950**	**$27,622**	**$27,384**	**$26,807**	**$26,473**	**$25,921**	**$24,988**	**$20,361**	**$14,709**	**$13,003**	**$11,955**	**$8,017**	**$2,404**	**$-**	**$329,558**

17. RECOVERY METHODS

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, RDi Minerals, or based on the QP's experience and industry norms. The design criteria and flowsheet development are discussed in this section.

17.1 Process Design Criteria

The Mt Todd feasibility study has been completed for the treatment rate of 50,000 tpd.

A detailed Design Criteria has been developed for both development scenarios. The nominal headline design criteria are listed in Table 17-1.

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	1,211
Gold Production (average)	oz/a	430,050

The testwork results collated from the 2011 and 2012 testing campaigns and additional metallurgical and process test work conducted in 2016, 2017, 2018, and 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.

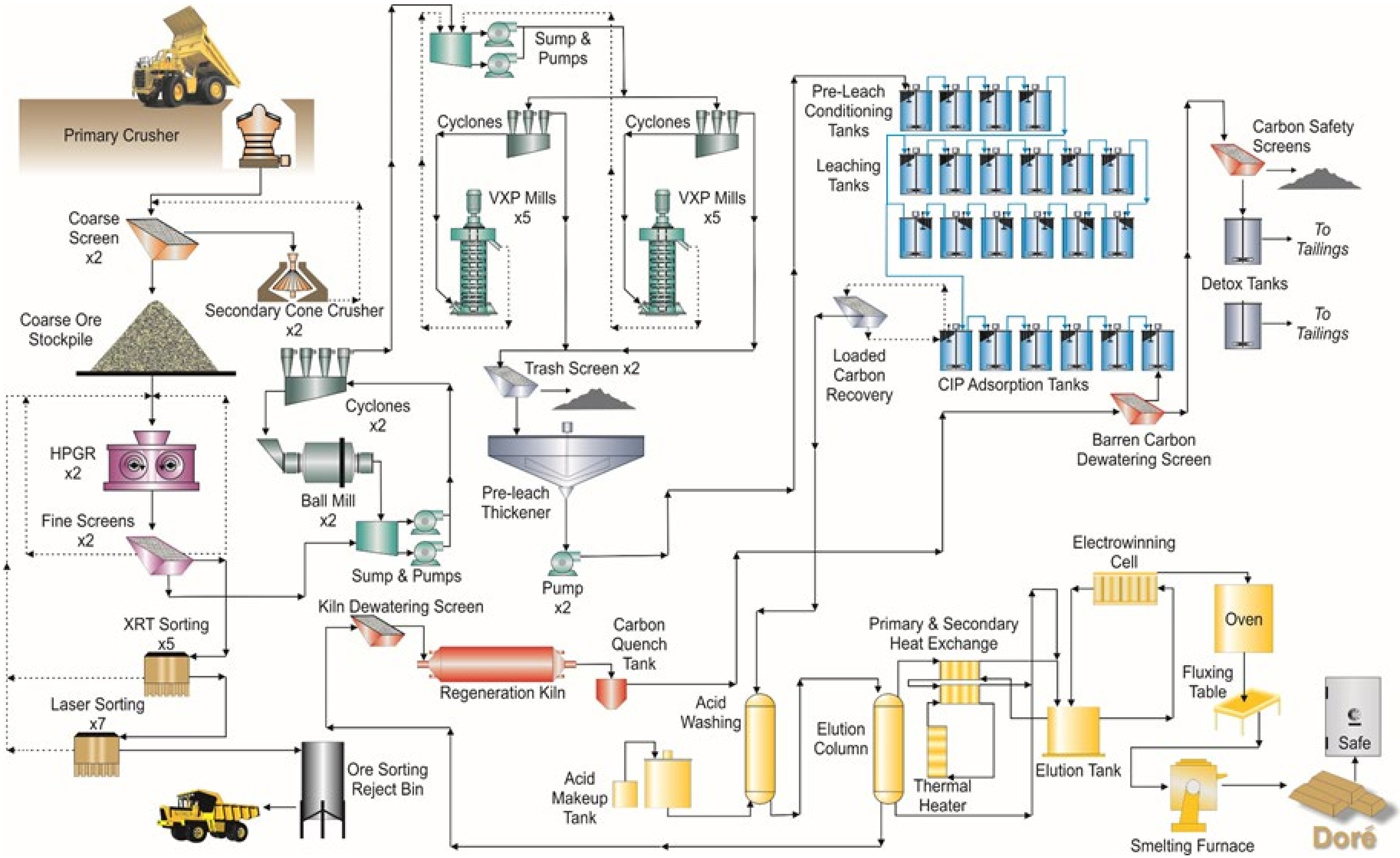


Figure 17-1: Simplified Process Flow Diagram

17.2.1 Crushing Modeling

Impact CWi tests were performed on eighty individual samples from the 2011 drill cores. The CWi values ranged from 3.2 kWh/t to 26.5 kWh/t. For design purposes, a CWi of 20 kWh/t was selected, which is 75% of the maximum value.

Unconfined compressive strength (UCS) was measured on 16 samples. The values ranged from 13.5 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 has a maximum particle size of 1000 mm and a F_{80} size 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, which is required as fresh 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 a product size P_{80} of 31.5 mm.

17.2.2 Primary Crusher

The primary crusher power was calculated utilizing 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 of these 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 modeled 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 48 mm, for a throughput of nominally 50,000 tpd.

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 2 times the fresh feed rate. HPGR testwork supported vendor recommendations as follows:

Nominal throughput of 50,000 tpd: two HPGR Polycom PM8-24/17, each equipped with 2 x 2,650 kW drives.

17.2.5 Grinding Modeling

A variety of internal models were utilized to provide the initial baseline ball mill power requirements, and vendors were approached for proposals. An evaluation was conducted that considered both price and technical acceptability. A submission was then selected for further interaction with the vendor calculations being compared against internal calculations. The circuit comprises two dual pinion drive ball mills to reduce P_{80} from 3.25 mm to 250 microns.

17.2.6 Thickener/Leach/CIP Design

THICKENER

Based on thickener sizing parameters received from RDI Minerals, that were in turn based on additional 2019 test work undertaken by Pocock Industrial for a final grind size of 40µm, a 67 m Pre-Leach thickener for a nominal throughput of 50,000 tpd was selected.

The test work also reported an underflow solids content of approximately 45% solids was achievable for the above thickener size.

LEACH AND ADSORPTION

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

The leach and adsorption circuits were modelled. A six-stage adsorption circuit is required to minimize solution losses. This configuration is also typical for other gold plants and minimizes potential shortcutting of pregnant liquor to tails. Target 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 1250 g/t, and carbon movement requirements to the gold recovery circuit will be on the order of 22 tpd for a nominal throughput of 50,000 tpd.

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. Taking into consideration downtime periods when the crushing system was not required, the average availability for the remaining duration was approximately 59%.

Additionally, TTP has access to 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 these 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 RESPEC that they would allow for the costs of an extra loader and for the build of an emergency stockpile on the ROM pad and to remove the downtime attributable to mining lack of supply in its entirety.

This resulted in an availability of 75.8%, or 6637 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 of live capacity. The primary and the secondary crushers discharge onto a common conveyor that is the conveyor feeding 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% has been used for the HPGRs and subsequent downstream processes, that is 7838 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 8760 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 HPGR's, with approximately 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 a nominal throughput of 50,000 tpd and will be 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 screen mid and oversize materials. 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 and comprises two stages, XRT and laser sorting. The two stages together reject 210 t/h representing approximately 10% of plant feed. This reject reduces subsequent grinding energy being unnecessarily spent on this very low-grade fraction of ore.

The above reject performance and nominal gold loss was derived from Outotec (Tomra) bulk ore sorting test work. Gold lost to ore sorting reject is minor at a nominal 0.07-1.13%, averaging 1% of gold entering plant.

17.3.3 Area 3300 – Grinding and Classification

Two ball mills will be used for the primary grinding circuit and will comprise two parallel closed-loop circuits 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 primary grinding circuit cyclone. The cyclone underflow will then gravitate to the ball mill feed. The overflow will gravitate to the secondary grind feed hopper. The secondary grinding cyclone overflow will be pumped to the pre-leach thickener, and the underflow will be sent to the VXP mills for further size reduction.

An automated ball charging system will be provided to deliver approximately 19 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 the leach tanks. 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 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 test work. Leach and adsorption will consist of eighteen mechanically agitated tanks in total, comprising twelve leach tanks and six adsorption tanks.

In order to maximize gold adsorption kinetics, lead nitrate will be dosed into the pre-leach conditioner underflow and oxygen will be dosed 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. Each adsorption tank will be equipped with interstage carbon screens to prevent carbon slurry from being transported downstream. These 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 (AARL) configuration. Eluant will be pumped through the column, heated to 120 °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 at least 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, quenched, and returned to the adsorption circuit.

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-Sulphite (SMBS) will be delivered to site as a 95% pure solid powder in sea containers. It will then be tipped or pneumatically conveyed using solids handling equipment to 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 2 days of nominal usage.

The Sodium Cyanide for Leaching and Elution will be delivered as briquettes in ISO tanks. The sodium cyanide will be consumed at a rate of approximately 43 tpd. Solids will be dissolved in the tanker and cyanide solution will be transferred into one of three storage tanks allowing three days nominal capacity. There will be a secured and covered facility to store cyanide briquettes in bulk bags in sea containers, on site as emergency storage. A mixing tank and bag breaker is included to allow for mixing of emergency stock.

The Hydrochloric Acid (HCl) for the acid wash column will be delivered as a 33% HCl solution and will have storage for 7 days of nominal usage.

Lime will be delivered as 92% activity quick lime powder in road tankers. The lime will be pneumatically transferred to storage silos with an approximately 4000 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 pellets 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 7-day combined mixing and storage capacity was included in 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. 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 lead nitrate solution will have storage for 2 days of nominal usage.

All reagents will have additional as-delivered storage of 15 days on-site as requested by Vista. The 15 days allows for a nominal 10 day emergency stock and 5 day operating stock.

17.3.8 Area 3800 – Process Plant Services

Approximately 750 Nm^3/h of medium pressure process air will be used to service the air requirements for leach and adsorption. Detoxification will be serviced by medium pressure air blowers at a consumption rate of approximately 5,200 Nm^3/h. High pressure compressors will be used to provide plant and instrument air.

Raw water will be supplied via the Raw Water Dam and will service process water make-up, fire water and gland seal water requirements. Raw water will also service the water treatment plant for generation of potable water required at the mining facilities, process plant and camp. Raw water consumption as informed by the site wide water balance work by Tetra Tech will be approximately 742 m^3/h.

Process plant water will be predominantly made-up of tailings decant return water and raw water, supplemented by water treated by the site water treatment plant (By Tetra Tech). Process water will be used for dilution and density control in the grinding circuits.

17.3.8.1 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 standby 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.

17.3.8.2 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/standby); and
- Leach and CIP (duty/standby).

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/standby 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/standby arrangement will be located in the cyanide detoxification area of the plant for process air in the cyanide detoxification tanks.

17.4 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. PROJECT INFRASTRUCTURE

The following section provides details on project infrastructure. Figure 18-1 shows the infrastructure layout, including the proposed pit, waste rock storage facility, tailings storage facilities, and process plant.

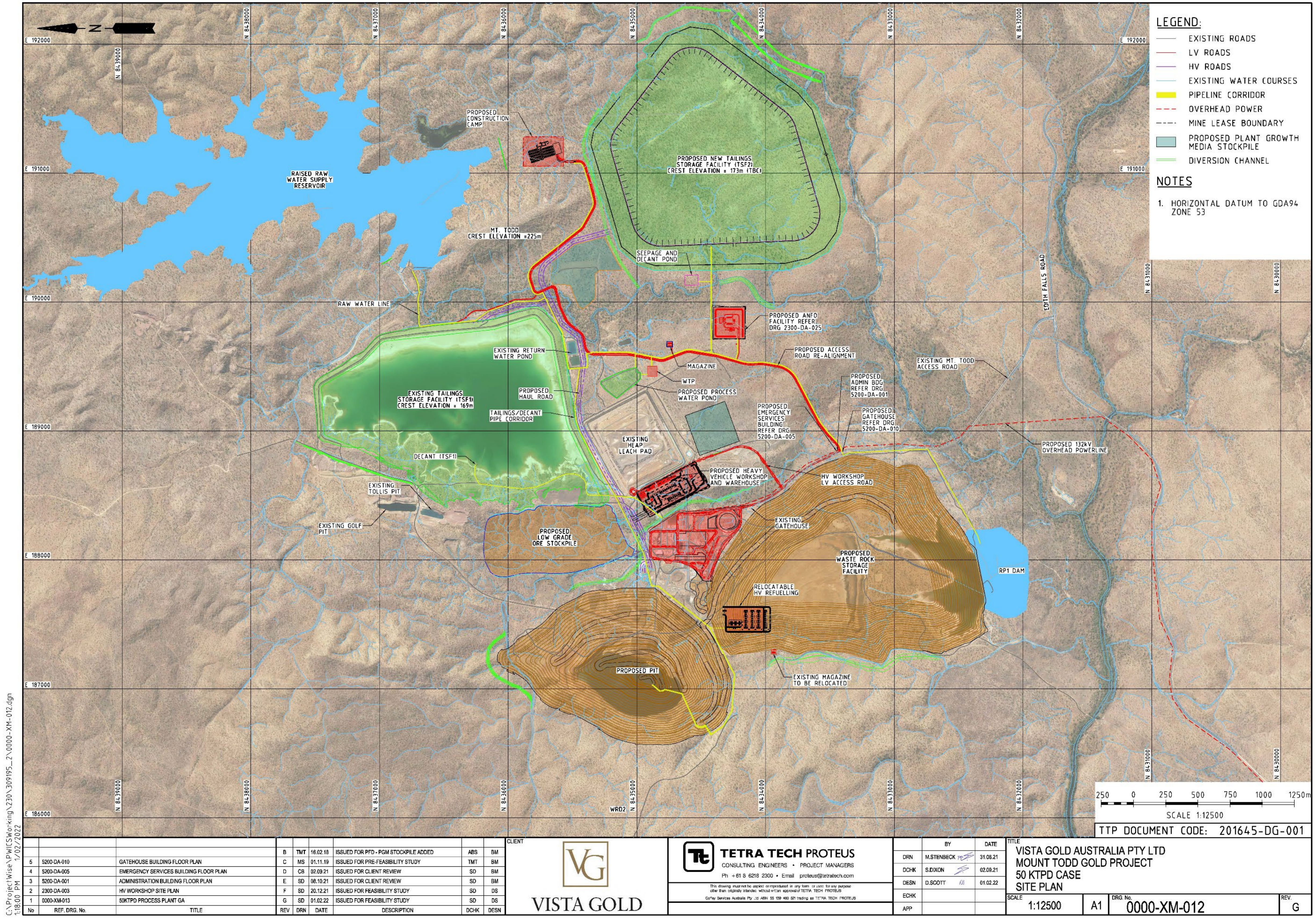


Figure 18-1: Site Plan

18.1 Facility 2000 – Mine

The following section provides a description of TTP's scope for 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 new process plant and existing Heap Leach Pad.

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. One service bay will be provided for Caterpillar D11 (or similar tracked vehicle). This bay will have cast-in steel rails or bars to reduce concrete surface wear and tear.

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 20 years.

A mobile crane will be used for the lifting and removal of vehicle parts.

18.1.1.2 Sub-Area 2310 Support Facilities – Bulk Fuel Storage

The bulk fuel storage is sized for 15 days diesel fuel storage and will consist of six 200kL storage tanks complete with one LV and one HV bowser located in the workshop area and four additional bowsers for dispensing into the HV fleet located at the waste rock dump. The storage capacity of the bulk fuel storage will be increased after year 4 of operation to include another five 200 kL storage tanks to bring the total storage to 2.2ML ML which will represent 15 days of operational usage expected after year 5.

Fuel will be pumped via a diesel transfer pump to a re-locatable HV refueling facility located on the waste rock dump. Refer to Section 18.1.1.3 for further details on this facility.

18.1.1.3 Sub-Area 2312 – Support Facilities – Relocatable Refuel Facility

The relocatable HV vehicle parking and refueling will be initially located on the northern side of the waste rock dump. This facility will be maintained and subsequently relocated by a mining contractor to suit mining operations and traffic requirements.

The facility will comprise a flat fully bunded area and HV refueling island with HV bowsers and 110kL local receiving tank (double skin - bullet type on a skid). A parking row with front wheels ditch will be provided to ensure safe parking of HV vehicles.

The footprint of the relocatable refueling facility will be increased in year 4 to accommodate the increased mining HV fleet in year 5 and thereafter.

The relocatable refueling facility will be (initially) located in the proximity of proposed pit. A Pit Blasting Management Plan must be prepared, reviewed, and approved prior any blasting operation to address potential adverse effects that blasting may have on the facility. This management plan shall be prepared following an approved Australian pit blasting template.

18.1.1.4 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.

18.1.1.5 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 initial size will be approximately 19.8 m by 14.4 m. This footprint will be increased by two bays to 26.4m by 14.4m in year 4 to accommodate the increased mining manning from year five 5) onwards.

18.1.1.6 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 20 years.

18.1.1.7 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.

18.1.1.8 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 tank is 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.

18.1.1.9 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. This 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 approximately 25 people.

18.1.1.10 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 covered area will be 21.7m wide. In addition to the covered area, a fully fenced open/uncovered racking area 25.6m wide will be provided as well.

The sea containers will be equipped with racking for additional storage.

The core storage facility will be complete with power and lighting and located in the northwestern corner of the HV workshop facility.

The dome shelter will be constructed of steel frame and tensile fabric with a fabric life expectancy of 20 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.

Fire Water supply to the Construction Camps is via local potable water system, with fire-fighting via fire hose-reels.

18.1.2.4 Sub-Area 2440 – Support Services – Air

Compressed air will be provided at the HV workshop, HV tire change, and HV washdown 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 mine support facilities via a connection from the 11 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 line (OPGW) and will terminate into a server room within the Mine Offices. Cat 6 ethernet cables will be reticulated to all required building and services.

18.1.3 Facility 3000 – Process Plant

18.1.3.1 Area 3100 – Crushing and Screening

PRIMARY CRUSHING

The Primary Crusher location will be east of the mining exclusion boundary, located at 187,825 E and 8,434,460 N on the eastern end of the Run of Mine (ROM) pad.

The Primary Crushing Building will be largely constructed of reinforced concrete with additional steelwork platforms for ancillary equipment and access ways. The ROM pad level at the primary crusher tip point is 28 m above the nominal ground level at the base of the primary crushing facility. The facility will be designed to provide tipping access from two sides-oriented 90° apart. The Primary Crusher Dump Pocket ahead of the Primary Crusher will be sized to hold the contents of two 200 tonne capacity dump trucks. A Rock Breaker will be provided to dislodge and break up oversize material jammed in the Primary Crusher.

The Primary Crusher will be a single FLSmidth Fuller-Traylor® 60x89HD gyratory crusher. This will be a top service unit designed so all maintenance activities can be completed from the top of the crusher. Maintenance access to the top of the crusher will be by mobile crane positioned on the ROM Pad.

Product from the Primary Crusher will pass to a 400-tonne capacity surge bin located directly below the crusher. Product will be reclaimed by an Apron Feeder driven by a hydraulic variable speed drive arrangement. The feeder will be installed directly above and parallel to the Coarse Screen Feed Conveyor, which will run in a south easterly direction.

A tramp metal magnet will be provided over the conveyor at the Primary Crushing Area immediately following the discharge from the apron feeder, to capture any large tramp metal which may have entered the process from the mine. The magnet will discharge to a tramp metal bin which can be removed by wheel loader.

Ancillary equipment such as lubrication units and hydraulic power packs will be located on platforms positioned close to the major equipment items and electrical switchgear will be housed in air-conditioned switch rooms at ground level adjacent to the plant.

Dust control at the Primary Crusher Dump Pocket during truck tipping and on the apron feeder discharge will be by means of a water spray system. Dust suppression water will be provided from a water tank and pumps located on the ROM Pad level.

Ventilation fans and associated ducting will be provided in the concrete vault areas below the Primary Crusher to ensure adequate air movement and ventilation.

Where overhead access from a mobile crane will not be possible, monorails and hoists will be provided.

COARSE SCREENING

The Coarse Screening Building will receive primary crushed ore and recirculating secondary crushed ore from the Coarse Screen Feed Conveyor. The building will contain two General Kinematic double deck vibrating screens (Double Deck Screen Model STM-D 4285), each of which will be sized at 4.2 m x 8.5 m. The screens will be fed by vibrating feeders drawing from the Coarse Screen Feed Bin.

The bin will serve as a buffer to level out minor surges or fluctuations in throughput, and the vibrating feeders provide an optimum ore distribution over the screen deck, thus increasing screen efficiency and resulting in a more uniform wear of the screen panels. The vibrating feeders will be fitted with wheels and configured in such that they can be retracted from the building on rails and removed by mobile crane for maintenance.

Oversize material from the screen will report to the Secondary Crushers, and the undersize fraction will report to the Coarse Ore Stockpile.

The Coarse Ore Feed Bin will be lined with wear resistant material to protect bin walls and will be equipped with hydraulically driven, sliding isolation gates to isolate the bins while work is carried out on the feeders and screens.

A dust collection system incorporating a venturi type wet dust scrubber will be installed at the Coarse Screening Building. Ducting will be provided to the screen discharge chutes and conveyor transfers to provide a negative pressure and prevent the egress of dust. The resulting slurry from the wet scrubber will be pumped to the Cyclone Feed Hopper in the Grinding Area.

A tramp metal magnet will be provided at the Coarse Screening Building over the Secondary Crusher Feed Conveyor to capture any tramp metal which may be present in the ore stream prior to the Secondary Crushers. The magnet will discharge to a tramp metal bin which can be removed by wheel loader. A tramp metal detector on the conveyor following this tramp magnet will provide an additional level of protection against tramp metal being fed to the Secondary Crushers.

SECONDARY CRUSHING

The Secondary Crushing Building will contain two FLSmidth Raptor 1300 cone crushers that will operate in parallel in a closed-circuit configuration. Oversize ore from coarse screening will be conveyed to the Secondary Crusher Feed Bin. The ore will be withdrawn from the bin and fed to the Secondary Crusher through a tapered slot by a belt feeder. The secondary crushed ore will be discharged for re-screening onto the Coarse Screen Feed Conveyor, which transfers ore from the Primary Crusher to the Coarse Screens.

The Secondary Crusher Feed Bin will be lined with wear resistant material to protect bin walls. The belt feeders will be mounted on rails and configured to allow for them to be retracted to gain clear access for crusher maintenance.

The Secondary Crushers will be packaged by the vendor and will contain hydraulic and lubrication modules and oil coolers. The hydraulic pack will be mounted on the platform adjacent to the crusher whereas the lubrication units and oil coolers will be mounted on the ground floor below the feed bin.

Dust control will be by means of a water spray system on the crusher feed inlet, the dust suppression water will be fed from the system in the Primary Crushing Building.

GENERAL

The main components of the Crushing and Screening Area including the Primary Crushing, Secondary Crushing, Coarse Screening and Coarse Ore Stockpile will all be aligned linearly, oriented in a south easterly direction. Each of these areas will be provided with concrete slabs and associated sumps at ground level to facilitate spillage clean up. The sumps will have pumps installed and have ramps for bobcat access.

The conveyor components will be rationalized to minimize the number of different types of components used in the plant. All elevated conveyors will have dual walkways for maintenance access.

Mobile crane access will be provided around the Secondary Crushers and Coarse Screens from both the north and south sides. Crane access for the Primary Crusher Building will be either via the ROM Pad for work on the Primary Crusher, or from the plant ground level east of the wing walls. An area has been allocated to the north of the Primary Crusher Building on the ground floor to ensure clearance for crane access is maintained.

All electrical switchgear for the Crushing and Screening Area will be housed in the Crushing Switchroom located adjacent to the Secondary Crushing Building, at ground level.

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

STOCKPILE AND RECLAIM

The Coarse Ore Stockpile will be situated to the south east of the Primary Crushing Facility. The stockpile will be on a waste pad approximately 15 m in height, inside of which will be the reclaim tunnel and vault. This will ensure the floor of the reclaim tunnel and vault under the stockpile remains at ground level to mitigate flooding concerns.

Ore will discharge from the head of the Stockpile Feed Conveyor to form a conical stockpile. The stockpile will have a live capacity of approximately 45,000 tonnes of ore with gravity reclaim and a total capacity of approximately 193,000 tonnes which will be fully reclaimed using a bulldozer. The stockpile will not be covered, it will be an open stockpile and dust suppression will be by continuously operated water sprays on the head of the Stockpile Feed Conveyor and ground level sprays at the base of the stockpile which are actuated when required to suppress dust.

The stockpile will be reclaimed by two apron feeders that will be situated inside a concrete vault. The Coarse Ore Reclaim Conveyor will be aligned directly beneath the apron feeders and will discharge from the vault to the northwest via a corrugated steel tunnel large enough to allow bobcat access along one side of the conveyor and personnel access along the other side. This vault will have a smaller corrugated steel tunnel extending to the south east for personnel emergency access / egress.

Reclaimed ore will be transferred into the HPGR Feed Bin via the Coarse Ore Reclaim Conveyor. The two apron feeders will be sized to provide the full plant downstream tonnage requirements with only one feeder operating.

The apron feeder inlet openings will be equipped with hydraulically driven, sliding isolation gates to isolate the stockpile while work is carried out on the feeders. The isolation gates will be able to support the full height of the stockpile above the gate and be able to withdraw the gate with a full stockpile. The gates will only be able to be closed when the stockpile is empty. Monorail beams and hoists will be positioned to assist with maintenance of equipment in the stockpile vault.

Ventilation fans and associated ducting will be provided in the concrete vault to ensure adequate air movement and ventilation. Dust control will be by means of a water spray system on the apron feeder discharge chutes, the dust suppression water will be fed from the system in the Primary Crushing Building.

A tramp metal magnet will be provided on the Coarse Ore Reclaim Conveyor to capture any tramp metal which may be present in the ore stream prior to the HPGR's. The magnet will discharge to a tramp metal bin which can be removed by forklift. There will also be a tramp metal detector on the conveyor following the tramp magnet to provide an additional level of protection against tramp metal being fed to the HPGRs.

HPGR AND FINE SCREENING

The HPGR Area will be northwest of the stockpile, on an area with suitable geotechnical properties for heavy dynamic loads. Two Thyssenkrupp Polycom PM8-24/17M HPGR units will each be fed via belt feeder from the HPGR Feed Bin.

Each HPGR will be mounted on an elevated concrete slab with the HPGR hydraulic pack and lube unit situated below. Maintenance on the HPGRs and removal of the rolls will be affected by a dedicated semi-portal crane. Each HPGR belt feeder will be retractable, and the rolls will be removed through the top of the HPGR and laid at ground level or placed directly onto specialized vendor transport to be taken to vendor's operations for refurbishment.

Crushed ore from each of the HPGRs will discharge onto the HPGR Product Conveyor and be transferred to the Fines Screen Building. The ore will discharge into the Fines Screen Feed Bin.

The Fines Screen Building will contain two General Kinematic double deck vibrating wet screens (Double Deck Screen Model STM-D4885), each of which will be sized at 4.8 m x 8.5 m. The screens will be fed by vibrating feeders drawing from the Fines Screen Feed Bin. The vibrating feeders will be configured such that they can be retracted from the building on rails and removed by mobile crane.

Oversize material from the Fines Screen will pass to the Fines Screen Oversize Conveyor. The mid-size material will be recirculated back to the HPGRs via the Fines Screen Mids Conveyor. The undersize fraction will flow to the Cyclone Feed Hopper via the Fines Screen Underpan and Launder. The Fines Screen Building will be located immediately alongside the Grinding Building to allow the fines screen undersize to gravity flow directly to the grinding circuit.

The Fines Screen Mids Conveyor tail end will be extended to service the Ore Sorting Area, and will have a tramp metal magnet followed by a tramp metal detector to provide protection against tramp metal being fed to the HPGRs.

The HPGR Feed Bin and Fines Screen Feed Bin will be lined with wear resistant material to protect bin walls and will be equipped with hydraulically driven, sliding isolation gates to isolate the bins while work is carried out on the feeders and HPGRs.

Mobile crane access will be provided to the south, east and west of the HPGR Building. The Fines Screening Building will be accessible by mobile crane from north and west.

Both buildings will be provided with a concrete slab with associated sump pumps for spillage clean up. The sumps will include a ramp for bobcat access.

All electrical switchgear will be housed in air conditioned switchrooms. The switchroom for the HPGR Building and Stockpile Reclaim Area will be located to the east of the HPGR Building, at ground level. The switchgear for the Fines Screening Building will be housed in the Grinding Switchroom.

ORE SORTING

The Ore Sorting Area will be northwest of the stockpile. Oversized ore will discharge from the head of the Fines Screen Oversize Conveyor onto the XRT Sorting Tripper Conveyor via the XRT Sorting Pocket Conveyor. The Tripper will discharge ore evenly across the XRT Sorting Feed Bin inside the XRT Sorting Building.

The XRT Sorting Building will contain five Tomra COM XRT 2400 2.0 units, each fed directly from the XRT Sorting Feed Bin. Ore material accepted by the XRT Sorters will be recirculated back to the HPGRs via the Fines Screen Mids Conveyor. Rejected ore material will be directed to the XRT Sorting Reject Conveyor.

Ore will discharge from the head of the XRT Sorting Reject Conveyor into the Laser Sorting Pocket Conveyor, which will feed the Laser Sorting Tripper Conveyor. The Tripper will discharge ore evenly across the Laser Sorting Feed Bin inside the Laser Sorting Building.

The Laser Ore Sorting Building will contain seven Tomra PRO Secondary LASER Dual 1200mm units, each fed directly from the Laser Sorting Feed Bin. Ore material accepted by the XRT Sorters will be recirculated back to the HPGRs via the Fines Screen Mids Conveyor. Rejected ore material will be directed to the Reject Bin Feed Conveyor via the Laser Sorting Reject Conveyor.

The conveyor head will discharge direct to an approximately 375 tonne capacity Ore Sorting Reject Storage Bin. The Ore Sorting Reject Storage bin will be equipped with a hydraulically driven, sliding isolation gate and a clam shell gate. Rejected ore material will be released into a 200 t haul truck as the bin reaches capacity. Any Ore Sorting Reject Storage Bin overflow will be directed to the Ore Sorting Reject Overflow Stockpile.

The XRT Sorting Feed Bin and Laser Sorting Feed Bin will be lined with wear resistant material to protect bin walls and will be equipped with hydraulically driven, sliding isolation gates to isolate the bins while work is carried out on the Ore Sorting units.

Mobile crane access will be provided to the North, South and West of the Ore Sorting Buildings. Monorails will be used to service the conveyors inside the Ore sorting building. The Fines Screening Building will be accessible by mobile crane from the north and west.

All electrical switchgear for the Ore Sorting Area will be housed in the Grinding Switchroom located adjacent to Primary Grinding, at ground level.

18.1.3.3 Area 3300 – Grinding and Classification

PRIMARY GRINDING AND CLASSIFICATION

The Primary Grinding and Classification Building will be located immediately north of the HPGR Area and adjacent to the Fines Screening Building on ground geotechnically determined to be competent and suitable for heavy dynamic loads. The building will contain two ball mills operating in parallel.

The underflow from each Fine Screen will gravitate to the Ball Mill Cyclone Feed Hopper. Each mill module will consist of a Fine Screen, Ball Mill Cyclone Feed Hopper, Ball Mill, and Cyclone Cluster.

Each Cyclone Feed Hopper will be in closed circuit with a Ball Mill, with slurry from the Cyclone Feed Hopper pumped to the Cyclone Cluster, cyclone underflow returning to the Ball Mill and cyclone overflow reporting to the Secondary Grinding Cyclone Feed Hopper within the Secondary Grinding and Classification Area. Each Cyclone Feed Hopper will be serviced by variable speed duty / standby Warman slurry pumps.

The Grinding Building will contain two FLSmidth 25′-0″ Dia. x 40′-0″ 14.5 MW ball mills. Each mill will have an inside diameter of 7.62 m and effective length of 12.2 m. Each mill will have dual main drives, each drive train consisting of motor, reducer, and pinion. An inching drive is provided for rotation during maintenance.

A proprietary mill liner handler will be used for liner maintenance. An RME “Thunderbolt” proprietary mill liner bolting device will be mounted on permanent monorail system alongside of each mill. Each mill will have a lubrication module and an oil cooler unit, both skid mounted. The mills will be equipped with a mill jacking system.

Steel balls of nominal diameter 68 mm will be used as grinding media. The mills will be charged via rotary ball charging device that will feed a steepwall conveyor to lift the media up to above the mill feed box level where a distribution conveyor will distribute the media to each of the mills as required. Media will be stored in a nearby bunker which will contain the required emergency stock of 15 days of ball mill media. The mill balls will be fed into a steel hopper by a wheel loader which will have a capacity of 1-day worth of media consumption for both ball mills. The hopper will be fitted with an isolation gate on the hopper discharge which will feed the rotary ball charger.

The mill balls will discharge through a trommel screen, with scats directed into an oversize collection bunker for periodic disposal.

Each grinding circuit cyclone cluster will consist of a Weir Minerals, 6 x 800CVX CAVEX cluster fitted with six 800 mm cyclones, of which four or five cyclones per cluster will be in operation (depending on feed density) at any time with one to two on stand-by.

The Primary Grinding and Classification Area will be provided with a suitably sloped concrete slab with associated sump pumps for spillage clean up purposes with spillage returned to the Cyclone Feed Hoppers. A suspended concrete slab will be provided at the feed end of the mills at feed spout level to facilitate access for the Mill Relining Machine.

Mobile crane access will be provided along the north, south and east perimeter of the Primary Grinding Area. Jib cranes will be located above each Cyclone Cluster to assist with maintenance on the individual cyclones within each cluster.

All electrical switchgear for the Grinding and Classification Area will be housed in an air-conditioned switch room located to the east of the Grinding Building, at ground level.

SECONDARY GRINDING AND CLASSIFICATION

The Secondary Grinding and Classification Area is located immediately adjacent Primary Grinding and Classification Area. The Secondary Grinding circuit will consist of two trains consisting of two stages each.

Overflow from the Ball Mill Cyclone Clusters will gravitate to the Secondary Grinding Cyclone Feed Hopper. Each secondary grinding mill module will consist of a Cyclone Cluster, Secondary Grinding Feed Box, three secondary grinding mills, Secondary Grinding Discharge Hopper and Trash Screen.

The Secondary Grinding Cyclone Feed Hopper will be in open circuit with the secondary grinding trains, with slurry from the Secondary Grinding Cyclone Feed Hopper pumped to the Secondary Grinding Cyclone Cluster, cyclone underflow reporting to the Secondary Grinding Feed Box and cyclone overflow reporting to the Secondary Grinding Discharge Hopper. Each hopper in the Secondary Grinding Area will be serviced by variable speed duty / standby Warman slurry pumps.

The Secondary Grinding area will contain ten FLSmidth VXP10000 Vertical Grinding Packages. Each mill will have an inside diameter of 1.9 m and effective length of 5.07 m. Each grinding package comes complete with grinding media collection tank and media pump, disc maintenance table, disc maintenance hoist, media dewatering screen, access platforms, pipework, and instrumentation. Each VXP10000 package will have an installed power of 3.0 MW excluding ancillary equipment.

Zirconia toughened alumina (ceramic) grinding media of nominal diameter 5-6 mm will be used as grinding media. The mills will be charged via the grinding media peristaltic pump which will pump from the grinding media collection tank up to the media dewatering screen discharging into the grinding media chamber of the VXP10000 mill.

Maintenance is performed by opening the valve at the bottom of the mill to allow the media to drain via gravity into the media collection tank. Maintenance of the urethane coated discs is carried out via the bottom of the mill where the discs are withdrawn and manipulated via the disc handling hoist and disc maintenance table.

The Stage 1 Secondary Grinding classification will consist of 1-off FLSmidth GMAX20-3140 fitted with 12 cyclones which will receive the grinding product of all 3 stage-1 VXP Mills. Stage-2 classification will consist of 1-off FLSmidth GMAX10-3139 per mill or 2-off GMAX10-3139 hydrocyclones fitted with 15 cyclones per train.

The cyclone overflow from each cyclone cluster will report to a 40 m^2 Tenova Delkor belt linear Trash Screen. Trash Screen oversize material will be collected in a trash disposal bunker for periodic disposal while trash screen undersize will report to the Pre-Leach Thickener Feed Box and will gravitate to the Pre-Leach Thickener to the east of the Grinding Area.

The Secondary Grinding and Classification Area will be provided with a suitably sloped concrete slab with associated sump pumps for spillage clean up purposes with spillage returned to the Secondary Grinding Cyclone Feed Hopper.

A semi portal overhead travelling crane will be provided to service the VXP Mills and equipment in the Secondary Grinding Area. Jib cranes will be located above each Secondary Grinding Cyclone Cluster to assist with maintenance on the individual cyclones within each cluster.

All electrical switchgear for the Secondary Grinding and Classification Area will be housed in an air-conditioned switch room located to the northeast of the Grinding Building, at ground level.

18.1.3.4 Area 3400 – Pre-Leach Thickening, Leach and CIP

PRE-LEACH THICKENER AND CONDITIONING

The Pre-Leach Thickener will be an on-ground 67 m diameter high-rate unit fitted with an automated rake lifting mechanism and adjustable deflector plate to distribute the feed uniformly across the thickener area at a controlled velocity. The thickener will be located southeast of the grinding circuit on a stand-alone section of plant due to its size.

Pre-Leach Thickener Underflow Pumps will be located at the entrance of the Pre-Leach Thickener access tunnel for ease of maintenance. Thickener underflow will report to the Leach Conditioning Tanks.

There will be two tanks for leach conditioning, each sized 21.6 m by 22.3 m. Each tank will have an SPX Lightnin 784Q220 agitator.

Tanks will be connected by intertank launders, the launder configuration and reagent piping will be such that either tank can be by-passed and removed from the circuit for maintenance.

Thickener overflow will gravitate to the Process Water Tank for distribution around the Process Plant.

Crane access will be provided in all directions surrounding the Pre-Leach Thickener.

LEACH / ADSORPTION

The Leach / Adsorption circuit will be located to the west of the Pre-Leach Thickener, with the slurry flowing from south to north before reporting to the Cyanide Detoxification circuit situated immediately to the north-east of the Adsorption circuit.

There will be twelve tanks for leach in a 2x6 configuration, each sized 21.6 m by 22.3 m. Each tank will have an SPX Lightnin 784Q220 agitator with process air for the cyanide leaching process injected through the hollow shaft.

There will be six tanks for adsorption, each sized 17.1 m by 17.8 m. Each tank will have an SPX Lightnin 783Q125 agitator.

All tanks will be connected by intertank launders, the launder configuration and reagent piping will be such that any tank can be by-passed and removed from the circuit for maintenance.

Fresh and regenerated carbon will be fed into the last adsorption tank in the train. Carbon will be moved up through the CIP circuit counter current to the slurry flow via recessed impeller pumps. The intertank launders will be fitted with three Kemix MPS 1900(P) pumping intertank screens in order to maintain constant slurry levels between tanks.

Loaded carbon will be pumped from CIP Tank No.1 over a Loaded Carbon Recovery Screen, with screen undersize slurry returning to the tank and carbon oversize reporting to the Acid Wash Column.

The slurry from the last tank will flow through intertank screens to the two Carbon Safety Screens mounted independently above the detoxification feed hopper and pump-set. Any oversize carbon remaining in the slurry will be washed and collected in a skip bin, while the undersize slurry will report to the Detoxification circuit.

Two gantry cranes will run the length of the Leach and CIP Tanks to facilitate maintenance of the Intertank Screens and agitators.

The entire Leach / CIP circuit will be provided with a bunded concrete slab sized to contain 110% of the largest tank volume. Each area will have dedicated sump pumps which will handle all spillage around the area and will return the spillage to the applicable process step. Sump pumps have not been sized to accommodate the complete failure of a tank. Tank rupture will require specialist cleanup and will require emergency equipment to be supplied. Spillage from the Carbon Safety Screen Area will be collected and returned to the process by the Detoxification Area Sump Pump.

18.1.3.5 Area 3500 – Desorption, Goldroom and Carbon Regeneration

STRIPPING PLANT

A fully automated, PLC controlled, modular elution plant will be provided with integral nominal 22 tonne capacity acid wash and elution columns, eluate tank, catholyte tanks, direct eluate heating system and feed pumps. This will include three skid platforms containing heaters, pumps, piping, electro-pneumatic valves, and controls. The upper floor area will contain electrowinning and sludge handling equipment in a security mesh screened area.

The electrowinning cell models will be 3x 125EC33 (125 ft^3), constructed with an SS304 body, with 33 cathodes and 36 anodes. The cells will each have four compartments, with 8-9 cathodes per compartment and a 2000 amp, 0-9 VDC plating rectifier. The specifications provided above are nominal and subject to final vendor package plant design.

GOLD ROOM

The Gold Room will be a modular construction complete with 600-T natural gas fired barring furnace, sludge drying ovens, gold doré safe, gold scales, flux scales, and gold room tools. The Gold Room will include extraction fans for the furnace and all ovens with fume hoods and ducting.

The Gold Room will include sludge handling equipment with pneumatic diaphragm sludge pumps for each Electrowinning (EW) Cell compartment, cathode wash bay (to suit 9 cathodes) with high pressure cathode washer, sludge pump and sludge settling tank.

The Gold Room equipment will be installed in two 7 m x 2.2 m two level skids, designed to be integral with the secure Electrowinning Area.

The specifications provided above are nominal and subject to final vendor package plant design.

CARBON REGENERATION

Carbon Regeneration will include a 22 tonne/day horizontal rotary carbon regeneration kiln, with natural gas fired burners and barren carbon feed hopper. The kiln will include a main electric drive, and battery operated emergency drive. Carbon feed will be controlled by a stainless steel screw feeder.

This area will include a 3.4 m x 3.1 m carbon quench tank, 1.5 tonne Safe Workload (SWL) fresh carbon handling monorail and electric hoist, carbon feed chute and carbon sizing screen.

The specifications provided above are nominal and subject to final vendor package plant design.

18.1.3.6 Area 3600 – Detoxification and Tailings

The Detoxification system will be located adjacent to the CIP circuit, on the northern end. The process slurry will enter the Detoxification Tanks from the Detox Feed Hopper under the Carbon Safety Screens. Slurry will be pumped via Warman Slurry Pumps from the Detox feed hopper to the Detoxification Tanks.

There will be two Detoxification Tanks with dimensions of 14 m diameter x 11.4 m high. Each tank will have an SPX Lightnin S783Q250 agitator with process air injected through the hollow shaft.

The tanks will be connected in series by overflow launders; however, will have the facility to bypass the first Detoxification Tank if required. An in-line crosscut sampler for metallurgical control and accounting purposes will sample the feed to the Detoxification system, with pressure pipe samplers supplied on the system discharge slurry.

Discharge from the Detoxification Tanks will report to the Tailings Hopper before being pumped to the Tailings Storage Facility.

Decant water from the Tailings Storage Facility will be pumped back to the Process Water Tank for use in the process. TSF1 will include two decant sources, one from the surface of the TSF and the other from TSF1's decant pond.

The Detoxification Area will be provided with a bunded concrete slab sized to hold 110% of the largest tank volume. A dedicated sump pump in the area will handle all spillage and return it to the appropriate point in the process.

Mobile crane access will be provided to service this area.

18.1.3.7 Area 3700 – Reagents

LIME SLAKING AND STORAGE

The lime storage and handling facility will be located in the south-eastern part of the Process Plant, in the reagents area adjacent to the cyanide preparation and storage.

The lime will be received as a powder delivered to site via pressurized road delivery tanker. Lime will be transferred to one 2055 tonne storage silo (15 day storage as requested by Vista) by pneumatic hose connected to the tanker. Lime will then be transferred by rotary valve, screw feeder and vibrating feeder to be processed in a 1.9 m diameter x 3.8 m long (EGL) lime slaking mill in closed circuit operation with a Weir hydrocyclone to ensure correctly sized slaked lime.

Lime slurry will be transferred to the 933 m^3 lime slurry storage tank for distribution to the process plant. The storage tank will include an SPX Lightnin 76Q20 agitator to maintain solids in suspension.

The lime area will be provided with concrete slabs and an associated sump pump to facilitate collection of spillages and clean up. Wet and dry areas of lime storage and slaking will be separated by exclusive bunds.

SODIUM CYANIDE HANDLING AND STORAGE

The sodium cyanide storage and handling facility will be located in the south-eastern part of the Process Plant, adjacent to the lime area. This area will contain a 228 m^3 mixing tank and three cyanide storage tanks. The cyanide storage will be in horizontal bullet tanks each with capacity of approximately 150 m^3.

This dissolution plant will have provision to dissolve Sodium Cyanide briquettes within up to 2 Isotainers, or alternatively by bulk bags using forklift and bag splitter. If the latter is used, the sodium cyanide will be unloaded into the mixing tank which will include an SPX Lightnin 74Q7.5 agitator for mixing. The cyanide transfer pump will transfer the solution to the cyanide storage tanks. Normal operation will be for Sodium Cyanide to be prepared using Isotainers.

The cyanide area will be provided with a bunded slab at ground level with an associated sump pump to facilitate collection of spillage and clean up.

SODIUM HYDROXIDE HANDLING AND STORAGE

Sodium hydroxide will be supplied in 1 tonne bulk bags that will be stored in the reagent bag storage area. A mixing tank with an SPX Lightnin CBQ0.75 agitator and bag breaker will be provided on the west side of the enclosing shed. Raw water will be added to make up the solution to the required solution strength of 50%.

The solution will be directly dosed using the sodium hydroxide metering pump to the elution column. The sodium hydroxide mixing tank level will be measured using an ultrasonic level transmitter and regulated automatically to setpoint by the flow control valve on the sodium hydroxide solution discharge line.

The sodium hydroxide metering pumps will operate in duty and stand-by mode, and will pump sodium hydroxide solution to the points of addition on a continuous basis. The duty to stand-by pump changeover for the sodium hydroxide metering pumps will be a manual operation. The sodium hydroxide metering pumps will be positive displacement pumps.

The sodium hydroxide area will have a dedicated sump pump which will pump the spillage from the bunded area to the tailings hopper.

FLOCCULANT HANDLING AND STORAGE

The flocculant mixing area will be sited in the western area of the reagent preparation and storage area.

A vendor packaged flocculant plant will be provided which will receive powdered flocculant in bulk bags. The vendor package will include bag breaker hopper and blower, 20 m^3 mixing tank with agitator and transfer pump.

Two 1000 m^3 flocculant storage tanks will be provided together with two positive displacement flocculant metering pumps in a duty/standby configuration. Each pump will be able to pump from either storage tank.

The flocculant area will also be provided with a bunded slab at ground level with an associated sump pump to facilitate collection of spillage and clean up.

SODIUM METABISULPHITE HANDLING AND STORAGE

The sodium metabisulfite (SMBS) storage and handling facility will be located centrally, in the northern area of the reagents storage and preparation area.

The SMBS will be received as a powder delivered to site in sea containers. SMBS will be unloaded by container tipper into a mixing tank for direct dissolving of SMBS into solution. The solution is then transferred into the 357 m^3 SMBS storage tank via SMBS transfer pump for distribution to the cyanide destruction plant.

All SMBS handling and storage plant will be fabricated from mild steel and coated according to vendor standard finishes.

The SMBS area will be provided with concrete bunded slabs and an associated sump pump to facilitate collection of spillage and clean up.

Hydrochloric Acid Handling and Storage

The hydrochloric acid storage and handling facility will be located centrally in the northern area of the reagents storage and preparation area.

Hydrochloric acid will be unloaded directly into a 35 m^3 storage tank by placing the IBC on top of the tank to facilitate draining.

The area will have a dedicated sump pump which will pump the spillage from the bunded area.

Lead Nitrate Handling and Storage

Lead Nitrate will be supplied in 1 tonne bulk bags that will be stored in the reagent storage area. A 36 m^3 mixing tank with agitator and bag breaker will be included indoors with sufficient ventilation.

A 60 m^3 storage tank will be provided together with transfer and metering pumps. The area will have a dedicated sump pump which will pump the spillage from the bunded area.

18.1.3.8 Area 3800 – Process Plant Services

Water

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

- Raw water supply
- Potable water supply
- Fire water supply
- Gland/seal water supply
- Process water supply.

Raw water will be delivered from the raw water pond to the 3700 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.

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

Raw water will also be used as feed to the Water Treatment Plant to provide potable water for general use and the safety showers.

The process water system will include a 12300 m^3 storage tank. Process water will be supplied to the plant via centrifugal pumps, one duty and one stand-by pump. 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 make-up all report to the process water tank.

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)
- HPGR's (duty only)
- Ore Sorting (duty/duty/duty/duty)
- Grinding and Classification (duty/stand-by)
- Leach and CIP (duty/stand-by).

Rotary screw compressors at each location will supply plant air and instrument air to the areas in which they are located. The air discharging from each compressor will be fed to a plant air receiver and distributed throughout the building. A take-off 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. Ore Sorting will utilize additional duty compressors to supply air to the XRT and Laser Sorting Buildings simultaneously.

A dedicated low pressure compressed air system in a duty/stand-by arrangement will be located in the leach area of the plant for process air dosing to the Leach 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.

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 600 m^3/hr.

18.2.1.2 Sub-Area 4120 – Raw Water

The raw water requirement will be approximately 740 m^3/hr, fluctuating due to current operations and weather. The existing line from the Raw Water Dam will be supplemented with an additional 450 mm HDPE 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 1.5 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. There will be nominally 100 m^3 of potable water consumed per day, with the Potable Water Treatment Plant nominally capable of producing 120 m^3/day.

18.2.2 Area 4200 – Power Supply

A summary of the nominal predicted power requirements and distribution is summarized in Table 18-1.

Table 18-1: Predicted Power Requirements

Area	Description	50,000 tpd Average Power (MW)	50,000 tpd Maximum Demand (MW)
1000	Geology	0	0
2000	Mine including; – ANFO – HV Workshop/Washdown – Mining Offices – Mines Support Services – Coreshed	0.28	0.34
3000	Process Plant including; – Crushing & Screening – HPGR & Ore Sorting – Classification & Grinding – Desorption & Gold Room – Detoxification & Tailings – Reagents	85.13	91.7
4000	Project Services including; – Pit Dewatering – Wastewater Treatment – Tailings Return – Diesel – Heap Leach – Bores	2.7	3.2
5000	Project Infrastructure including; – Administration Offices – Plant Offices – Gatehouse – Laboratory	0.17	0.23
6000	Permanent Accommodation	0.62	0.73
	Total	**88.9**	**96.2**

18.2.2.1 Sub-Area 4210 – Power Generation

Power Generation for the project will be by natural gas reciprocating engines located in the Power Station around 8km to the Southwest of the process plant. The power generated will be transmitted to the process plant by dedicated 132kV overhead power lines. At the process plant, the 132kV is stepped down to 33kV to feed the Main 33kV Switchroom. The Main 33kV Switchroom then distributes 33kV to other switch rooms across the process plant areas. Electrical power from the third-party power supplier is supplied via 132kV overhead power lines, and step down to 33kV at the process plant.

The existing Power and Water Corporation's 22 kV grid will no longer supply the process plant, however existing services which will remain will continue to be fed from the 22kV grid such as the RP1 pump station and associated MCC.

The existing Substation No. 0 and 11kV network will be supplied from the Main 33kV Switchroom via a 33/11kV step down transformer.

18.2.2.2 Sub-Area 4230 – High Voltage Electrical Distribution

The Main 33kV Switchroom will be the main point of connection for incoming power from the Power Station, as well as fiber optic communications from Telstra. The Switchroom includes the main 33 kV switchboard consisting of a main incomer, 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 network.

18.2.2.3 Sub-Area 4231 – Power Distribution

Within the process plant areas, 33 kV power will be reticulated from the Main 33kV Switchroom to other process plant switchrooms via buried high voltage cables. This eliminates the likelihood of mobile plant such as cranes interacting overhead power lines. The 33kV cables will be buried in electrical and in some cases shared services trenches.

Each process plant switchroom has transformers which step down 33kV to 6.6kV and 400V to supply power to drives and electrical equipment in the process plant areas. Cables from each switchroom will be run in cable ladders to relevant parts of the process plant. Some cables will be run in underground conduits to equipment that are far away or difficult to reach.

18.2.2.4 Sub-Area 4232 – Overhead Power Lines

The existing 11kV overhead powerline network supplied from Substation No. 0 will be replaced with new overhead lines and poles. The 11kV will also be extended to supply areas outside of the process plant such as the HV Workshop, ANFO Facility, Decant Pumps, Gatehouse, and the Accommodation Village. The Main 33kV Switchroom will supply the 11kV network via a 33/11kV step down transformer. The 11kV poles will be installed alongside access roads and away from structures as much as practical.

The 11kV power line will distribute power to the following facilities:

- ANFO Facility
- Heap Leach Pad (existing)
- Construction Camp/Residual Accommodation Camp
- Water Treatment Plant (WTP)
- Heavy Vehicle Workshop
- Mine Services
- Site Radio Communication Tower (depending on final location)
- Gatehouse and
- Future Tailings Storage/Decant.

The total length of overhead power line required to reach the above locations from the Main 33kV Switchroom is approximately 8 km.

The overhead power line will incorporate a fiber optic cable into the earth conductor (OPGW). Refer to Section 18.2.3.1. Overhead power lines will be suitably rated for a high dust and lightning strike region.

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 a redundant backbone fiber optic network. Each process plant switchroom will have a comms cubicle with FOBOTs which the fiber optic cables terminate into to provide networks to the switchroom. The fiber optic cables will generally be installed in underground conduits, although sections of the cables will be on cable ladders within the plant. 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 fiber optic cable will be installed underground between the Telstra communications hut (by others) and the site 11kV overhead power line network at the closest pole. The Main 33kV Switchroom will take incoming Telstra fiber optic and switch the network throughout the process plant areas using the backbone fiber optic network.

The administration, process plant offices area, and control room will be part of the backbone fiber optic network. A communications cubicle in these areas will provide access to the Process Control System network, the site IT (internet) network, the site Voice over Internet Protocol (VoIP) phone network, the site Closed Circuit TV (CCTV) and security network, and fire detection network. This will enable the SCADA to function in the control room and enables the engineers to access plant programmable logic controllers (PLCs) and associated network devices.

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 stations and any other remote plant that do not have the 11kV overhead powerline network running nearby.

The system will incorporate a Master Telemetry Station, located in process plant switchrooms, and a number of Slave (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 frequency. 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 252.4 Mt of process tailings will be stored in two separate tailings storage facilities (TSFs) over a design operating life of 17 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 113 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 combination of downstream (starter dyke) and upstream construction techniques. A total of approximately 156.5 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 18-2: 50 ktpd TSF 1 and TSF 2 Parameters

TSF 1 Design Parameter	Value
TSF 1 EXPANSION	
Design Tailings Storage Capacity	113 million tonnes
Average Tailings Dry Density	1.5 t/m^3
Design Life	17 years
TSF 2	
Design Tailings Storage Capacity	156.5 Mt
Average Tailings Dry Density	1.5 t/m^3
Design Life	17 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 of the conventional slurry tailings. There is approximately 7% contingency storage in the proposed TSF design capacity. 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 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. 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 embankment construction sequence and tailings deposition schedule was developed based on the tailings production rate, tailings characteristics, and to permit tailings consolidation and drainage in support of upstream raise construction. Tailings facility embankment raise construction and tailings deposition will be split between TSF1 and TSF2 to maintain a low rate of rise while meeting the storage and construction requirements.

18.2.5 Area 4500 – Waste Disposal

Sewage waste disposal for the project will be treated in 3 areas. A Biomax Wastewater Treatment Plant (WWTP) with an associated spray field for the dispersal of treated effluent will be installed adjacent the HV workshop (Mine Services). The Mine Support and Process Plant buildings will be connected to the WWTP via the sewer pipework reticulation system.

An appropriately sized Biomax Wastewater Treatment Plant with an associated spray field will also be installed at the construction camp which will be connected to the camp’s sewer reticulation system. A smaller system will be installed near the gate house to service that along with the administration and emergency services buildings.

18.2.6 Area 4600 – Plant Mobile Equipment

The plant mobile equipment to be purchased for the process plant will be as follows:

Table 18-3: 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	4
25t Container Forklift	1
25t Reach Stacker	3
80t Crane	1
Subtotal	**14**

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 (Batman Creek) on the east side of the proposed process plant. Batman Creek will be channelized adjacent to the process plant, the new channel sized to fully accommodate a 1:100 year storm. Plant pad earthworks will be above the 1:100 year flood level within the new channel.

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 have been developed in conjunction with each other.

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 next to Gatehouse and Emergency Services Buildings. 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 approximately 30 people.

18.3.2.2 Sub-Area 5211 – Process Plant Offices

The Process Plant Offices will be complexed with multiple transportable buildings located opposite Workshop and Warehouse in the North-East part of the process plant. The buildings will include the necessary system furniture and provide cellular and open planned offices.

The Process Plant Offices will be sized to accommodate approximately 17 people per shift.

18.3.2.3 Sub-Area 5220 – Workshop/Warehouse

The Workshop / Warehouse will have a footprint of approximately 60m by 23.3m. Within this footprint, it will comprise mezzanine offices, tool store, LV workshop with engine repair area, four (4) vehicle service bays with an overhead crane above all service bays, external gas storage and an oily water separator with drive-in pit/sump.

One service bay will be drive-through and all entry bays will have apron slabs to divert spillages to the oily water separator. Stair access for overhead crane service and maintenance will be provided next to the external gas storage

The building will be complete with services including overhead travelling 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 approximately 25 people per shift.

18.3.2.4 Sub-Area 5230 – As-delivered Reagent Store

The as-delivered Reagent Store will consist of three Dome Shelters supported by steel frame and concrete footings, with a concrete floor. The reagent store will be sized approximately 22 m by 55 m and includes segregated areas within which 15 days of each reagent is stored in its as-delivered form. Forms of storage include sea-containers, IBCs and bulk bags (within sea-containers).

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 20 years.

The as-delivered Reagent Store will be secured and cover an area of approximately 1,210 m^2.

18.3.2.5 Sub-Area 5240 – Crib/Ablutions

The Crib / Ablutions facilities will be complexed with transportable buildings located adjacent the process plant offices (Area 5211). 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 and Gatehouse and will be sized 14.4 m by 9.9 m. This area will include an undercover area for an ambulance bay and a parking 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 windsock. 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 and 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 in the PFS was a single transportable building located at the Primary Crusher. This single person 3m by 3m control room is deleted in the FS and replaced by another workstation/seat in the main control room.

18.3.2.11 Sub-Area 5281 – Control Building – Main Control Building

The Main Control Room will be a single transportable building in process plant located adjacent Gold Room 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 opposite the main control room building. This control room will be subdivided into a Control Room and a Titration Room. The buildings 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 current design and minor road repairs will be carried out.

The existing corrugated steel culverts at the Batman Creek crossing on the east side of the proposed Process Plant is suffering from corrosion. These corrugated steel culverts will be replaced.

Two new light vehicle access roads will be constructed, one teeing off the existing plant access road approximately 900 m south of the new plant, heading north around the eastern side of the existing heap leach pad, then east to the proposed construction camp. The second road tees off the existing access road approximately 300 m south of the plant to access the new Heavy Vehicle workshop. This road will utilize a floodway to cross Batman Creek.

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 18-4: 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.

18.3.5.2 Sub-Area 5820 – Communications Link to Telstra

Fiber optic cable and connections, which will link Mt Todd Process plant and supporting infrastructure to the Telstra network, will be determined and defined during detailed design phase.

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 approximately 468 construction workers based on the concept construction manning histogram developed for the Project. The construction manning histogram assumes there are components of each construction contract (Civil, Concrete, SMP) that can be brought forward. In some instances, the 'Fly Camp' may need to be used for a small contingent of workers under these contracts to conduct this work (in addition to the requirements described above). Final alignment and optimization of construction manning against the EPCM schedule will be optimized during the next phase of the project as contract packages are progressed to tender.

The Construction Camp will be located east of the existing TSF1, north of the proposed TSF2 and due south of the raw water reservoir.

The Construction Camp will be hired for the nominal 24-month construction duration with the exception of 80 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:

- 468 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

- Putrescible Waste Dump
- Waste Water Treatment Plant
- LV Parking Area and Bus Drop Off/Pick Up and
- 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. Refer to Section 21.1.2.12 for details regarding the indirect costs and assumptions used in the FS.

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 estimated separately using a combination of bottom-up and standard-factor approach.

18.6.2 Area 8200 – External Consultants/Testing

This area is a Prime Cost (PC) Sum allowed 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. The areas 9600 to 9900 are sums of money associated with working capital, escalation and exchange rate fluctuation, contingency and management reserve. Refer to Section 21.1.2.12 for details regarding the indirect costs and assumptions used in the FS.

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 spares 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 typically non-consumables (although some critical spares may be consumables, they are kept on hand in the event of an unexpected or unscheduled failure). These are large items that are not expected to be used, however must be kept in spare for the project due to long lead times and process importance. These items include but are not limited to spare mill motors, HPGR motors, HPGR rolls, intertank screens and conveyor components.

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 TTP, 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 management 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 management 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 will be an allowance to cover the fluctuation of foreign currencies and inflation from the time of this study until purchase date. Costs for this have not been included in the estimate as provision for this will be made 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. Refer to Section 21.1.2.13 for a detailed description of contingency provision.

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. Refer to Section 21.1.2.13 for a detailed description of management reserve.

18.8 Electric Power

The mine's electrical power demands are estimated to be approximately 84 MW for the normal operating load and 104 MW for the peak demand based upon the load list developed as part of this FS. Vista has historically looked at building and operating the Electric Power Plant (EPP) on its own. As part of this FS, trade-off studies were completed examining purchasing power from the grid, third-party power suppliers, and owner/operator EPPs. The results of the trade-off studies were:

- Examination of the local electric grid quickly showed that the local grid can neither meet the project demands nor is reliable enough to pursue this direction.
- Two, highly reputable third-party electric power generating companies responded to the Request For Proposal for supplying the project with electric power. Both of these companies operate EPPs in Australia, as well as elsewhere in the world, and in particular supply electricity to other mining operations.
- There was no longer a need for Vista to pursue an owner/operator EPP as part of the project based on the third-party proposals due to the competitive pricing proposals.

Vista has decided to accept one of the third-party EPP provider's proposals and has incorporated their costs into the economic analysis of this FS. As part of Vista's permitting process, Vista is in possession of the permits necessary to construct the EPP and the third-party EPP supplier will use these permits for their EPP.

The third-party supplier will be responsible for the EPP and the high-tension power line needed to operate the project. The connection point will be the sub-station at the plant site.

19. 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 strong, with prices sustained in the range of $1,900–$2,050 per ounce. The gold price used in this FS is US$1,800/oz. Detailed information used for the determination of the minable reserves can be found in Section 15.1—Pit Optimization of this Feasibility Study.

19.2 Contracts

Currently, there are no contracts in place for development and operations. However, Vista has obtained budgetary quotes, as is common for FS-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. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

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 June 2021 the Mining Management Plan (MMP) was approved by the Northern Territory Government Department of Industry, Tourism and Trade (DITT). This was the last approval required before works can occur and resulted in the "Mining Authority" being issued.

The QPs for Section 20 are Amy L. Hudson, Ph.D., CPG, SME RM; April Hussey, P.E.; and Vicki J. Scharnhorst, P.E., LEED AP. Each of these QPs is of the opinion, for their respective portions of this Section as defined in Section 28 of this report, that the environmental studies, permitting and plans, negotiations, or agreements with local individuals or groups adequately address the conditions and hazards for use in this study.

20.1 Environmental Studies

A number of environmental studies have been conducted at the Project in support of development of the 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 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 rock drainage and metal laden (ARD/ML) 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 ARD/ML 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.

Receipt of the Aboriginal Areas Protection Authority (AAPA) Certificate was required to identify and protect sacred sites from damage by setting out the conditions for using or carrying out works on an area of land. It is a legal document issued under the Northern Territory Aboriginal Sacred Sites Act.

Following extensive review, the AAPA determined that the use of, or work on, certain areas can proceed without a risk of damage to, or interference with, the sacred sites identified at Mt Todd. The AAPA Authority Certificate for Mt Todd covers the mining licenses and 1,581 km² of exploration licenses contiguous with the mining leases.

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 forming (PAF) or non-acid forming (NAF) material before being hauled to the WRD. NAF material will be stockpiled for use in reclamation covers or placed in the WRD. Construction of the WRD is described in Section 16—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 eleven additional raises to the embankment and construction of three new saddle dams at the west end of the impoundment. A second tailings storage facility, TSF 2, is to be constructed to the south east of the existing 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 and 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 existing water RPs (excluding the pit and 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 RPs 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 historical 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 continuous in-pit treatment, seepage management, treatment of 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 micronized limestone and quicklime. Treatment has been undertaken to produce water to be discharged at rates protective of water quality in the Edith River. As of the date of this study, the pit is essentially de-watered, with less than 0.5 GL of water remaining. The treatment methodology included raising the pH of the water within the pit lake to greater than pH 8.0 using micronized 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 resulted 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.

20.2.4.2 Seepage Management

Analysis of the potential infiltration and seepage conditions of the WRD has been completed through numeric modeling and observations of the existing WRD behavior. A thorough assessment of the infiltration and seepage conditions of the 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, 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—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 treating 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 2 to 15 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 was estimated from the numeric model of the facility with the preferred closure cover design placed over the benches and top surface. The quantity of seepage from the TSFs following closure was estimated by simply multiplying the predicted infiltration of daily precipitation through the proposed TSF closure covers by the ultimate two-dimensional surface area of each facility. Stochastic precipitation developed in the water balance model from site and Katherine gage data statistics developed and assembled for the 2018 Preliminary Feasibility Study (Tetra Tech, 2018) were used and 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 the TSFs, 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 all of its approvals and makes a decision to proceed to gold production.

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 Plan Approval from NT Department of Primary Industry and Resources	Approval April 2021 based on a 50kt/day operation. An amendment will need to be submitted for the minor changes as a product of the transition from PFS to this FS.	Approved Jun. 2021	NA
Heritage Act permit to destroy or damage archeological sites and scatters/ Aboriginal Areas Protection Authority Clearances	Authority Certificate Number C2021/028 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 and exploration activities.	Aboriginal Areas Protection Authority dated Jun. 07 2021	NA
Aboriginal Areas Protection Authority Certificate	The use of, or work on, certain areas can proceed without a risk of damage to, or interference with, the sacred sites identified at Mt Todd. Covers the mining licenses and 1,581 km^2 of exploration licenses contiguous with the mining leases.	Jun. 7, 2021	NA

Approval/Permit/License	Current Status	Approval/ Permit License Date	Expiration Date
Surface Water Extraction License	Provides the right to annually harvest 3.48 gigaliters of surface run-off to use for mine operations.	Jun. 1, 2021	Jun. 1, 2031
Approval to reopen and operate the existing Mt Todd Gold Mine	Approved in accordance with Part 9 of the Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act) by the Australian Department of the Environment and Energy – EPBC Ref: 2011/5967	Jan. 19, 2018	NA
Permit to Interfere with a Waterway Diversions – Approval from Department of Environment, Parks and Water Security	Assessment done as part of the MMP assessment in 2021, including a site visit. Approval IWW:VDG-001 Diversions	Approved Feb. 03, 2022	N/A
Permit to Interfere with a Waterway RWD – Approval from Department of Environment, Parks and Water Security	Assessment done as part of the MMP assessment in 2021, including a site visit. Approval IWW:VDG-002 Dam	Approved Feb. 27, 2022	N/A
Dangerous Goods Act (1988) permit for blasting activities	On hold until FID	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
Water Extraction License Approval from Department of Environment, Parks and Water Security	Approved via License No: 8141014 issued for 3,480 ML/year to be harvested via the Raw Water Dam	Jun. 01 2021	Jun. 01 2031
Waste Discharge License (under Section 74 of the Water Act 1992) for management of water discharge from the site	WDL 178-8 licensing discharge of treated water into the Edith River from the Mt Todd mine site, granted with conditions	Nov. 30 2020	Revoked at our request in 2021 as not required until operational
Waste water treatment system permits under Public Health Act 1987 and Regulations	Required for the waste water treatment system for the construction and operations accommodation village. Permit application not yet in progress pending FID.	NA	NA
Approval to Disturb Site of Conservation Significance (SOCS)	Batman pit expansion will disturb SOCS as breeding/foraging habitat for the Gouldian finch. Plan has been approved via EPBC 2011/5967 An extension will be applied for late 2022.	Jan. 19, 2018	Jan. 2023

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 Association has been consulted as part of the planning process 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 NAF 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 NAF Material		X		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		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 21.1.4—Reclamation and Closure. 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 24.5—Regional Groundwater Model and Mine Dewatering), a terminal-sink pit lake is anticipated to 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 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—Mining Methods, the WRD will be constructed with an encapsulating NAF material outer shell on each lift. Concurrent installation of a low permeability geosynthetic liner (i.e., LLDPE) following attainment of final grades will serve to reduce infiltration of precipitation into the WRD core. This liner system will include nonwoven geotextile placed above and below the LLDPE 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 approximately 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 NAF 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 WTP for treatment prior to release until it is feasible to treat this and other ARD/ML on-site using passive treatment systems.

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 NAF 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 below this cover.

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 NAF waste rock or 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 PWP will be left in place, up-graded if necessary, and used to treat 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, anticipated five years following cessation of processing.

20.5.5 Heap Leach Pad and Pond

The HLP and Pond will be 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 1 or 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 1 or 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/ML generated during operations reports to the PWP, and therefore the WTP.

20.5.7 Mine Roads

Mine access roads will remain in place to provide post-closure access to the area. 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, the 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. 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 and TSF seepage collection systems meets applicable standards, flows to the collection systems cease, or alternative passive water treatment system is installed and functioning adequately.

The return water, polishing and overdrain ponds for the TSFs shall remain post-closure and be incorporated into passive water treatment systems. 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, PAF 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 ultimate designs and following the closure plans discussed above. Closure costs are accrued and contained in the financial model.

21. CAPITAL AND OPERATING COSTS

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	16
Closure Period	Years	4
Operating Days	Days/Year	355

Costs are presented in Q4 2021 US dollars and are based on an US$0.69:AUD1.00 exchange rate, unless otherwise noted.

Section 21—Capital and Operating Costs presents costs 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—Economic Analysis 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.

21.1 Capital Cost

LoM capital cost requirements are estimated at US$1,746 million as summarized in Table 21-2. Initial capital of US$1,030 million is estimated to be required to commence operations. At the end of operations, the Project will receive a US$43 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	6.2%	$94,127	$5,738	**$99,865**	$583,957	$36,152	**$620,109**	$678,084	$41,890	**$719,974**
3000	Process Plant	11.0%	$560,796	$61,691	**$622,487**	$33,498	$3,539	**$37,036**	$594,294	$65,229	**$659,523**
4000	Project Services	9.6%	$56,893	$6,625	**$63,518**	$86,468	$7,082	**$93,551**	$143,361	$13,707	**$157,069**
5000	Project Infrastructure	10.5%	$49,389	$5,203	**$54,592**	$7,502	$773	**$8,275**	$56,891	$5,976	**$62,867**
6000	Permanent Accommodation	10.0%	$422	$42	**$464**	$0	$0	**$0**	$422	$42	**$464**
7000	Site Establishment & Early Works	12.6%	$26,553	$3,334	**$29,886**	$0	$0	**$0**	$26,553	$3,334	**$29,886**
8000	Management, Engineering, EPCM Svcs	12.0%	$111,185	$13,384	**$124,569**	$0	$0	**$0**	$111,185	$13,384	**$124,569**
9000	Pre-Production Costs	10.0%	$31,071	$3,098	**$34,169**	$0	$0	**$0**	$31,071	$3,098	**$34,169**
10000	Asset Sale	0.0%	$0	$0	**$0**	($42,756)	$0	**($42,756)**	($42,756)	$0	**($42,756)**
	Capital Cost	**9.2%**	**$930,436**	**$99,114**	**$1,029,550**	**$668,670**	**$47,546**	**$716,216**	**$1,599,106**	**$146,660**	**$1,745,766**

21.1.1 Mining

Table 21-3 shows the estimated mine capital requirements for the by year. The initial mine capital is estimated to be US$154 million, with life-of-mine capital of US$525 million. This includes capitalized operating costs of US$49.7 million for tailings construction, US$4.9 million for pre-stripping, and US$99.6 million for reclamation.

Table 21-3: Mine Annual Capital Costs (US$000s)

	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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Total
Primary Mining Equipment																						
Atlas Copco PV235	$ 11,546	$ 23,091	$ 15,394	$-	$-	$ 3,849	$ 3,849	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-		$ 57,728
Atlas Copco ROC 65 (165mm Bit)	$ -	$ 1,661	$ 1,661	$ 1,661	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 4,983
29m3 Hyd. Shovel (PC 5000)	$ 10,682	$ 10,682	$ 10,682	$ 10,682	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-		$ 42,728
19m3 Front End Loader (994)	$-	$ 10,470	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-		$ 10,470
CAT 793F Haul Trucks (227-tonne)	$ 20,389	$ 40,778	$ 61,167	$ 10,194	$ 61,167	$ 20,389	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-		$ 214,083
Total Primary Equipment	**$ 42,616**	**$ 86,682**	**$ 88,904**	**$ 22,537**	**$ 61,167**	**$ 24,237**	**$ 3,849**	**$-**	**$-**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**		**$ 329,992**
Support Equipment																						
630 Kw Dozer (D11)	$ 2,536	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 2,536
300 Kw Dozer (D9)	$-	$ 1,177	$ 1,177	$-	$-	$-	$-	$-	$-	$ 588	$ 1,177	$ 588	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 4,707
7.3 m Motor Grader (24M)	$-	$ 3,069	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 3,069
4.9 m Motor Grader (16H)	$ 1,474	$-	$ 1,474	$-	$-	$-	$-	$-	$-	$ 737	$ 1,474	$ 737	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 5,897
Water Truck - 70,000 Liter	$ 2,024	$ 2,024	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 4,048
RTD Dozer (834H)	$ 1,867	$-	$ 1,867	$-	$-	$-	$-	$-	$ 934	$ 934	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 5,602
Rock Breaker - Impact Hammer (691 Kg m)	$-	$ 65	$-	$-	$-	$-	$ 32	$ 32	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 130
Backhoe/Loader (1.5 cu m-446D)	$ 453	$-	$-	$-	$-	$-	$ 226	$ 226	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 906
Pit Pumps (5299 lpm)	$ 49	$ 49	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 98
36 ton Crane	$ 867	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 867
2 cm excavator (Cat 392)	$-	$ 499	$-	$-	$-	$-	$ 250	$ 250	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 998
Low Boy	$-	$ 1,037	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 1,037
Flatbed	$ 88	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 88
Manlift	$-	$-	$-	$-	$-	$ 151	$-	$-	$-	$-	$-	$-	$-	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 151

	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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Total
Total Support Equipment	**$ 9,359**	**$ 7,920**	**$ 4,518**	**$-**	**$-**	**$ 151**	**$ 508**	**$ 508**	**$ 934**	**$ 2,259**	**$ 2,651**	**$ 1,325**	**$-**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**		**$ 30,135**
Blasting																						
Skid Loader	$ -	$ 112	$ -	$ -	$ -	$ 112	$ -	$ 56	$ 56	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 335
Mine Maintenance																						
Lube/Fuel Truck	$ -	$ 1,527	$ -	$ -	$ -	$ -	$ 382	$ 382	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 2,291
Mechanics Truck	$ -	$ 251	$ -	$ -	$ 251	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 501
Tire Truck	$ -	$ 342	$ -	$ -	$ 342	$ -	$171	$171	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 1,025
Total Mine Maintenance	**$ -**	**$ 2,119**	**$ -**	**$ -**	**$ 592**	**$ -**	**$ 553**	**$ 553**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**		**$ 3,817**
Other Mine Capital																						
Light Plant	$ -	$ 115	$ 38	$ -	$ -	$ 77	$ 19	$ 29	$ 19	$ 10	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 306
Mobile Radios	$ 15	$ 48	$ 22	$ 4	$ 36	$ 7	$ 11	$ 13	$ 4	$ 2	$ 2	$ 1	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 165
Shop Equipment	$ -	$ 491	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 491
Engineering & Office Equipment	$ -	$ 200	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 200
Water Storage (Dust Suppression)	$ -	$ 149	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 149
Base Radio & GPS Stations	$ -	$ 105	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 105
Unspecified Miscellaneous Equipment	$ -	$ 150	$ -	$ 2,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 2,150
Access Roads - Haul Roads - Site Prep	$ 175	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 175
Light Vehicles	$ -	$ 776	$ 50	$ -	$ 863	$ 50	$ 301	$ 406	$ 130	$ 25	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 2,603
Total Other Mine Capital	**$ 190**	**$ 2,034**	**$ 111**	**$ 2,004**	**$ 899**	**$ 134**	**$ 332**	**$ 448**	**$ 153**	**$ 37**	**$ 2**	**$ 1**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**	**$ -**		**$ 6,345**
Capitalized Mine Operating Costs																						
Pre-Stripping Mining Cost	$ 4,901	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 4,901
Tailings Construction Costs	$ 4,202	$ 1,760	$ 2,768	$ 4,537	$ 2,843	$ 1,537	$ 1,419	$ 1,533	$ 1,979	$ 2,403	$ 2,846	$ 3,261	$ 3,523	$ 3,364	$ 3,293	$ 4,026	$ 3,368	$ 1,094	$ -	$ -		$ 49,757
Reclamation (Occurs in Years 13 and 14)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 17,343	$ 33,975	$ 16,632	$ -	$ -	$ -	$ 15,842	$ 15,842	$ 99,633
Total Capitalized Mining Costs	**$ 9,104**	**$ 1,760**	**$ 2,768**	**$ 4,537**	**$ 2,843**	**$ 1,537**	**$ 1,419**	**$ 1,533**	**$ 1,979**	**$ 2,403**	**$ 2,846**	**$ 3,261**	**$ 3,523**	**$ 20,707**	**$ 37,268**	**$ 20,658**	**$ 3,368**	**$ 1,094**	**$ -**	**$ 15,842**	**$ 15,842**	**$ 154,292**
Capital Summary																						
Primary Mining Equipment	$ 42,616	$ 86,682	$ 88,904	$ 22,537	$ 61,167	$ 24,237	$ 3,849	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 329,992
Support Equipment	$ 9,359	$ 7,920	$ 4,518	$ -	$ -	$ 151	$ 508	$ 508	$ 934	$ 2,259	$ 2,651	$ 1,325	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 30,135
Blasting	$ -	$ 112	$ -	$ -	$ -	$ 112	$ -	$ 56	$ 56	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 335
Mine Maintenance	$ -	$ 2,119	$ -	$ -	$ 592	$ -	$ 553	$ 553	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 3,817
Other Mine Capital	$ 190	$ 2,034	$ 111	$ 2,004	$ 889	$ 134	$ 332	$ 448	$ 153	$ 37	$ 2	$ 1	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		$ 6,345
Capitalized Mine Operating Costs	$ 9,104	$ 1,760	$ 2,768	$ 4,537	$ 2,843	$ 1,537	$ 1,419	$ 1,533	$ 1,979	$ 2,403	$ 2,846	$ 3,261	$ 3,523	$ 20,707	$ 37,268	$ 20,658	$ 3,368	$ 1,094	$ -	$ 15,842	$ 15,842	$ 154,292
Total - All Mining Capital	**$ 61,269**	**$100,628**	**$ 96,301**	**$ 29,079**	**$ 65,501**	**$ 26,171**	**$ 6,660**	**$ 3,098**	**$ 3,122**	**$ 4,699**	**$ 5,499**	**$ 4,587**	**$ 3,523**	**$ 20,707**	**$ 37,268**	**$ 20,658**	**$ 3,368**	**$ 1,094**	**$ -**	**$ 15,842**	**$ 15,842**	**$ 524,915**

21.1.1.1 Major Mining Equipment

Capital for major mining equipment is shown in Table 21-3 and discussed in the following subsections.

DRILLING AND BLASTING

Primary drilling equipment capital is based on equipment quotations for a total of 15 Atlas Copco Pit Viper 235 blast-hole drills required through the life-of-mine. Seven of the drills will be purchased at the start of mining in Year -1, an additional two drills purchased in Year 1, then four additional drills will be purchased in Year 2 and finally two in Year 5 at a cost of US$3,848,535 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$1,660,836 each. One drill is purchased in Year -1, one more in Year 1, and a replacement drill has been planned for Year 3.

Quotes for explosives trucks, powder magazines, and bulk ANFO storage have 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$111,639 during Year -1 and then two additional units would be purchased in Year 4 and Year 7.

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$10,681,943, which includes freight and assembly.

The cost of the 18-cubic meter loaders is based on a quote for a Caterpillar 994 loader, with both being purchased in Year 1, at a cost of US$5,235,188 each.

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 4 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:

Table 21-4: Haulage Truck Costs

Year	Number of Trucks Added	Trucks in Use
-1	9	9
1	4	13
2	10	23
3	7	30
4	9	39
5	3	42
6	0	42
7	0	42
8	0	42
9	0	42
10	0	42
11	0	42

Year	Number of Trucks Added	Trucks in Use
12	0	23
13	0	19
14	0	17
15	0	9
16	0	2
17	0	0
18	0	2

Throughout the mine life, a total of 42 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$5,097,220, including freight and assembly.

21.1.1.2 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$2,535,570 each quoted by EMG LLC);
- Two Caterpillar D9 track dozers (US$1,176,755 each quoted by EMG LLC);
- One Caterpillar D8 track dozers (US$854,952 each quoted by EMG LLC);
- One Caterpillar 24M motor grader (US$3,069,140 quoted by EMG LLC);
- Two Caterpillar 16M motor graders (US$1,474,244 quoted by EMG LLC);
- Two Caterpillar 777 trucks with 70K liter water tanks (US$2,024,135 quoted by EMG LLC);
- Two Caterpillar 834H rubber tire dozers (US$1,867,455 quoted by EMG LLC);
- One Caterpillar 330 excavators (US$499,101 quoted by EMG LLC);
- One Caterpillar 446D Backhoe (US$452,805 quoted by EMG LLC);
- One low-boy trailer complete with a used 60t haul truck to tow it (US$1,037,300);
- One flatbed truck (US$88,410);
- Two pit pumps (US$49,100 each);
- One rock breaker to be attached to the 392DL excavator as needed (US$64,900); and
- 8 light plants (US$19,152 each quoted by EMG LLC).

Replacements are purchased for most units in Year 6.

21.1.1.3 Maintenance

Capital for mine maintenance equipment includes three fuel/lube trucks (US$763,578 each), one mechanic's truck (US$250,700 each), and three tire trucks (US$341,600 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 mechanic's truck is put into service.

An additional US$491,033 has been included for shop equipment / tooling. Shop facilities were estimated by TTP and included in facility capital.

21.1.1.4 Mine Facilities

Mine facility capital has been estimated by TTP and is included in facility capital.

21.1.1.5 Light Vehicles

Initial capital for light vehicles is estimated to be US$689,500 while sustaining light vehicle capital is US$1,913,800. Initial and sustaining light vehicle capital is shown in Table 21-5.

Table 21-5: Mine Light Vehicle Capital

		Initial Capital			Sustaining Capital		
	Type	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	3	$ 36,400	109,200	8	$ 36,400	$ 291,200
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	$ 544,400
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	$ 44,100	$ 88,200	4	$ 44,100	$ 176,400
Mechanics / Labor	4wd Pickup	2	$ 36,400	$ 72,800	4	$ 36,400	$ 145,600
Total		**19**		**$ 725,900**	**47**		**$ 1,877,400**

21.1.1.6 Other Capital

Other miscellaneous capital includes mobile radios for mobile equipment (US$1,000 per unit), engineering and office equipment (US$200,000), water storage for dust suppression (US$148,830), GPS stations and surveying equipment (US$105,000), and other unspecified miscellaneous equipment (US$150,000). At the end of Year 3, Mt Todd personnel will take over the maintenance of equipment. Accordingly, as unspecified equipment has been added in Year 3 for additional maintenance equipment.

21.1.2 CIP Process and Infrastructure

Please note that this section describes costs in Australian Dollars (AUD).

The Capital Cost Estimate (CCE) is based on a +/- 15% class 3 estimate as defined by AACE, which is typical for a Feasibility Study. 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.

The Project Capital Cost Estimate for both process plant throughput options is summarized in the table below (Table 21-6):

Table 21-6: Estimated Capital Cost Summary (US$000s)

Capital Cost	Initial Capital (US$000s)
Facility 1000 – Geology	$-
Facility 2000 – Mine Infrastructure	$94.1
Facility 3000 – Process Plant	$560.8
Facility 4000 – Project Services	$56.9
Facility 5000 – Project Infrastructure	$49.4
Facility 6000 – Permanent Accommodation	$.4
Facility 7000 – Site Establishment & Early Works	$26.6
Facility 8000 – Management, Engineering, EPCM Services	$111.2
Facility 9000 – Preproduction Costs	$31.0
Subtotal Direct Costs	**$930.4**
Contingency Provision (11%)	$99.1
TOTAL EXPECTED COST	**$1,029.6**

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). This initial project budget is also the budget that is *expected* to be spent and essentially covers expected and anticipated costs.

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.

The Process Plant and indirect capital costs are detailed in Sections 21.1.2.1 and 21.1.2.2 respectively.

21.1.2.1 Process Plant Capital Cost Summary

The Capital Cost Estimate for areas within the Process Plant, for both throughput options is summarized in Table 21-7 below:

Table 21-7: Process Plant Capital Cost Summary

Area/Sub Area	million US$
Area 3100 - Crushing & Screening	$71.4
Area 3200 - Coarse Ore Stockpile, Reclaim & HPGRs	$125.9
Area 3300 - Classification and Grinding	$156.1
Area 3400 - Pre-leach Thickening, Leach & CIP	$112.3
Area 3500 - Desorption & Goldroom	$11.8
Area 3600 - Detoxification & Tailings	$12.6
Area 3700 - Reagents	$22.4

Area/Sub Area	million US$
Area 3800 - Process Plant Services	$109.8
Total Facility 3000 – Process Plant (Expected)	**$622.5**

Note: Table includes contingency

A summary of the bulk commodity quantity requirements for the Process Plant construction is given in Table 21-8 below.

Table 21-8: Quantity of Bulk Commodities for the Process Plant

Bulk Commodity	Units of Measure	Quantity
Concrete	m^3	44,766
Structural Steel	Tonnes	6,550
Platework	Tonnes	2,113
Tankage	Tonnes	4,871

Notes:
(a) Quantities include contingency allowances, figures rounded up.
(b) Structural steel quantities include structural steel of weights < 25 kg/m, 25-75 kg/m, >75 – 120 kg/m and >120 kg/m
(c) Due to the method of the estimate the above does not include quantities allowed for the primary crusher reinforced wall, dust extraction ducting, Desorption & Goldroom, the Lime Storage Silo, package plants and roof sheeting within the reagents area.

21.1.2.2 Indirects Capital Cost Summary

A summary of the indirect cost for both throughput options is presented in Table 21-9.

Table 21-9: Indirects Capital Cost Summary

Area/Sub Area	million US$
Area 7300 - Construction Camp	$29.9
Area 8100 - EPCM Services	$71.9
Area 8200 - External Consultants/Testing	$.9
Area 8300 - Commissioning	$6.6
Area 8400 - Owners Engineering/Management	$38.1
Area 8800 - License, Fees and Legal Costs	$3.5
Area 8900 - Project Insurances	$3.5
Area 9100 - Pre-Production Labor	$1.8
Area 9200 - Commissioning Expenses	$3.1
Area 9300 - Capital Spares	$20.6
Area 9400 - Stores and Inventories	$2.4
Total Indirect Costs	**$182.3**

Note: Table includes contingency.

21.1.2.3 Currency Exchange Rates

The majority of the budget quotations for mechanical equipment were provided in Australian dollars. Exchange rates, provided by Vista, were used for major equipment sourced overseas and are listed below:

- 1 AUD = 0.690 USD
- 1 EUR = 1.125 USD
- 1 ZAR = 0.0657 USD
- 1 THB = 0.0310 USD
- 1 CNY = 0.1465 USD

21.1.2.4 Exclusions

The TTP SoW 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.

21.1.2.5 Project Chart of Accounts

The Work Breakdown Structure (WBS) of the estimate is a detailed structure under which the scope and cost items are assigned in the CCE. There are three tiers to the CoA including the Facility, Area and Sub-Area codes and descriptions.

The CoA also defines whether the costs are classified as Direct or Indirect to the Project. This structure has been developed in close consultation with the Vista and other stakeholders. The previous sections of this report have been organized to reflect this same structure.

21.1.2.6 Capital Cost Estimating Methodology

ESTIMATE APPROACH

An FS design has been developed for a 50,000 tpd plant and forms the basis for a capital cost (CAPEX) estimate. The approach adopted is consistent with AACE Class 3 requirements for a +-15% cost estimate and predominately uses bottom-up calculations, 3D modelling and subsequent material take-offs to develop the estimate.

The methods used to estimate capital costs are further detailed in the following sections.

PROCESS PLANT CAPITAL COST ESTIMATE

The CAPEX for the process plant features the methodology shown in Table 21-10.

Table 21-10: CCE Methodology for Facility 3000 – Process Plant

Item	Methodology
Mechanical Equipment	A detailed mechanical equipment list, with supply and installation pricing based on multiple budget quotations and internal body of knowledge
Concrete	MTO's based on 3D model and budget quoted unit rates
Structural Steel	MTO's based on 3D model and budget quoted unit rates
Platework	MTO's based on 3D model and budget quoted unit rates
Tankage	MTO's based on preliminary design calculations and quoted unit rates
Piping	MTO's based on preliminary Piping and Instrumentation diagrams, process and site layouts, and budget quoted unit rates.

Item	Methodology
Electrical	Electrical design and nominated equipment based on Mechanical Equipment requirements, preliminary electrical calculations and process plant and site layout, with supply pricing based on budget quotations and internal costing database for bulks such as cables, cable ladders, and luminaires.
Instrumentation and Control	Instrumentation and Control design and MTO based on preliminary Piping and Instrumentation Diagrams, with supply budget quotations and internal costing database for bulks such as cables and conduits.

Estimate factors were then back-calculated for each bulk commodity as a percentage of the mechanical equipment supply cost. The resultant estimate factors were critiqued against published data and industry experience.

21.1.2.7 Other Area Capital Cost Estimates

The estimation of CAPEX for all areas outside of the process plant adopted the methodology shown in Table 21-11.

Table 21-11: Methodology for Other Areas of the Capital Cost Estimate

Area / Sub Area	Methodology
Area 2300 – Mine Support Facilities	Drawings developed for the buildings and priced largely on budget quotations and building per square meter basis. Earthworks 12D models developed for MTOs with rates bases on budget quotations
Area 2400 – Mine Support Services	Drawings developed for the buildings and priced largely on budget quotations and building per square meter basis. Earthworks 12D model developed for MTOs with rates based on budget quotations
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 a combination of TTP for minor piping to NPI and TTNA for major piping and equipment. Costs were developed based on MTOs, site layout and budget quotations.
Area 4200 – Power Supply	Based on length of power distribution cables and trenching, and overhead power lines using rates obtained from budget quotations.
Area 4300 – Communications	Based on MTOs for the fiber optic cables, phones and telemetry using budget quotations and rates developed from previous projects.
Area 4400 – Tailings Dam	Estimated by Tetra Tech North America (TTNA)
Area 4500 – Waste Disposal	Based on budget quotations for sewage treatment facilities and MTO of sewage lines.
Area 4600 – Plant Mobile Equipment	Vendor pricing of the proposed fleet for plant operation
Area 5100 – Site Preparation	Based on MTOs from preliminary drawings and 12D model and rates based on budget quotations
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	12D models developed for MTOs and rates based on budget quotation
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 the total length of fiber optic cable and trenching to be installed. Quantities were estimated and budget pricing used.
Area 6100 – Personnel Transport	Based on unit rates for bus shelters with an allowance for the small amount of concrete required

Area / Sub Area	Methodology
Area 7300 – Construction Camp	Earthworks based on 12D models for MTO of access roads and site works. Rates based on budget quotation. Vendor quotes for the camp and operation
Area 8100 – EPCM Services	Estimated using a combination of bottom-up and top-down approaches, using the preliminary project schedule as a basis.
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.
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. Commissioning spares estimated based on preliminary spares list.
Area 9300 – Capital Spares	Based on 5% of the mine and process plant mechanical equipment costs. Process plant capital spares estimated based on preliminary spares list.
Area 9400 – Stores & 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.

21.1.2.8 Schedule of Estimate Requirements

TTP utilizes five classes of estimate, each being relevant to the stage of development of a project. This includes Class 1 – Scoping, Class 2 – Pre-Feasibility, Class 3 – Feasibility, Class 4 – Project Control and Class 5 - Definitive estimate classes. This FS has been prepared to the requirements of a Class 3 estimate. The TTP standard Schedule of Estimate Requirements tabulates for each estimate class the required inputs to achieve an estimate with the associated accuracy range.

21.1.2.9 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 for each major contractor.

The construction labor rates developed for the CCE include the following construction contractors:

- Concrete
- Structural, Mechanical and Piping (SMP)
- 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:

1) All direct payments including the site allowances and special project allowances for straight time and overtime worked for personnel
2) Overtime at penalty rates
3) Provision for holiday leave and loadings thereon
4) Provision for sick leave
5) Provision for cost of travel time to site and return travel on job completion
6) Provision for additional manpower turnover, bereavement leave and miscellaneous paid non-work days.
7) Payroll tax
8) Workers compensation insurance
9) Superannuation considerations
10) Industry redundancy payments

A Rest & Recreation 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:

1) Project Management personnel
2) Construction Supervision personnel
3) Site Quality Assurance and Control personnel
4) Site Health, Safety, Environmental personnel
5) Other indirect labor (stores officer, surveyor etc.)
6) Contractor vehicles for the Project Management team
7) Office accommodation
8) Workshop and stores facilities
9) Staff travel including airfares
10) Office overheads
11) 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
- 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 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 21-12 below.

Table 21-12: Construction Gang Rate Development

Contractor	Base Labor Rate ($AUD/hr)	Contractor Indirect Rate ($AUD/hr)	Construction Plant Rate ($AUD/hr)	Construction Gang Rate ($AUD/hr)
Concrete	$119.00	$17.92	$24.14	$160.00
SMP	$141.17	$51.67	$29.67	$220.00
E&I	$138.71	$62.15	$23.17	$225.00

21.1.2.10 Mechanical Equipment

INTRODUCTION

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 Mechanical Equipment List was used as the list for pricing individual mechanical equipment items. The basis for estimating the mechanical equipment supply costs was largely based on budgetary pricing from vendors. The vendors were provided with a formal request for quotation document with attached preliminary specifications and/or data sheets for these equipment items.

Typically, a minimum of three vendors were engaged for each mechanical equipment package. Although, where equipment was specialized, or the value of the package was low, fewer vendors were asked to tender. Upon receiving the budget quotations, a budget pricing evaluation was conducted. This included a comparison of place of manufacture, lead time, commissioning rates, technical data, technical compliance and cost. Considering the above, the key reasons were noted for selecting the preferred vendor for the FS design. The budget quotations received from vendors have an expected accuracy level equal to +/- 10%.

All other minor equipment items were priced from a TTP database of costs from recent 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/estimates provided by the manufacturer or supplier

- Estimates based on the weight and volume of the load
- Estimates based on published and in-house guides for similar installations
- 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/estimates provided by the manufacturer or supplier
- Estimates based on the weight of the equipment
- 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 thickener were reviewed in detail against historical records and published guidelines.

21.1.2.11 Quantity Development and Unit Rates

INTRODUCTION

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 (e.g., Rawlinsons) and comprise of allowances for supply of the raw material, fabrication, freight and erection. A summary of the bulk commodity quantities is presented in Table 21-8.

CIVIL

Preliminary bulk earthwork quantities were estimated using specialized civil 3D modelling software (12D). 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 obtained from a budget quotation and checked against TTP's 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 the development of unit rates.

CONCRETE

Concrete quantities for foundations and ground slabs for all equipment and structures in the process plant were calculated using 3D model material take-offs. 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.

Composite 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

Quantities of steel required for the process plant structures were quantified using the 3D models developed by the structural drafters and checked by structural engineers. 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.

Composite unit rates for the supply and installation of structural steel were calculated using budget quotations that were checked against TTP's internal supply and installation rates of similar jobs. Supply of steel was based on rates obtained from multiple budget quotations. 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 historical data in similar projects for 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. Stiffening steelwork was estimated using a nominal 50% additional allowance of the base chute box platework as additional stiffening mass. Allowances for Bisalloy or rubber lining where applicable were MTO'd, or estimated and included.

Unit rates for platework were provided by steelwork fabricators as is described for structural steel.

TANKAGE

Quantities of steel required for custom designed tankage were quantified by structural engineers using the 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 and base plates. Top rings and other attachments will be covered by the contingency attached to the platework estimate.

Unit rates for supply and installation of tankage were based on fabricated steel rates obtained 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

Quantities of piping and piping related items including but not limited to elbows, tees, flanges, bolts and valves, were estimated based on preliminary piping and instrumentation diagrams and the 3D modelling of piping. 3D piping modelling was only conducted on 80mm nominal bore piping and above with small bore piping manually estimated utilizing modelled pipe racks and equipment locations.

Supply unit rates for piping and fittings were obtained from multiple vendor budget quotations, for all quantified items. Installation hours were developed using a combination of engineering experience and TTP's in-house database. Installation hours were inclusive of lift hours for heavy weight piping and elevated piping within the pipe racks.

Non-process piping and overland piping was quantified using satellite imagery, ground topology, and 3D plant models. Utilizing site plans overlaid on satellite imagery allowed for piping to be estimated up to building perimeters, assisting with the creation of services piping networks. These methods were used in the determination of potable water (safety shower) and fire water piping quantities, forming part of the greater MTO. The locations of these showers and hydrants are in compliance with AS4775 and AS 2419 respectively. From these locations, piping routes and pipe diameters were then confirmed.

ELECTRICAL

The estimate for the supply and installation of electrical components for the process plant was based on a high-level electrical design based on the Mechanical Equipment List and other process plant electrical requirements such as lighting, small power and general reticulation of power. Electrical components required for NPI facilities were estimated based on typical power requirements for each installation, and power distribution quantities to these facilities were based on a combination of minimizing cable length and maximizing maintainability.

Electrical equipment sizing and calculations were developed to size major components such as transformers, high voltage (HV) switchboards, Variable Speed Drives, and switchrooms. Single Line Diagrams, Switchroom Layout Drawings, equipment datasheets, and equipment specifications were developed to assist in obtaining the most accurate budget pricing for electrical equipment. The following items were also developed in order to maximize accuracy achievable by budget pricing for supply and installation:

- Cable schedules covering all cables from instrumentation to 33kV high voltage cables.
- Cable ladder routes and underground trenching take offs.
- Lighting and small power take offs.
- Earthing take-offs.

Instrumentation and Control

The estimate for the supply and installation of Instrumentation was based on budget pricing from an instrumentation vendor based on the Process Control Philosophy, Piping and Instrumentation Diagrams and Input-Output (I/O) list developed for the project. Process Control System hardware and development for the plant was based on budget pricing from various Systems Integrators based on the input-output (I/O) list and a highly automated gold plant design with all field instruments marshalled to Remote Input / Output (I/O) cabinets.

21.1.2.12 Indirect Costs

Construction Camp

An estimate of the construction facilities was developed from previous project experience for the various scopes of work and budget quotations. 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 an indicative budget quotation from an appropriate contractor.

The construction camp cost estimates issued for the FS are summarized in Section 21.1.2.2.

EPCM Services

A TTP fee estimate for the Engineering, Procurement, Construction Management and Commissioning (EPCM) was prepared using a combined bottom-up and top-down approach.

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.

The EPCM Services cost estimates issued for the FS are summarized in Section 21.1.2.2.

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.

The External Consultant/Testing cost estimates issued for the FS are summarized in Section 21.1.2.2.

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
- Pre-production Labor

The following costs were calculated based on industry validated percentages of the mechanical equipment supply cost for the project:

- Commissioning Expenses (Excluding commissioning spares)
- Stores and Inventories

The following costs were based on a preliminary engineering spares list:

- Commissioning spares
- Capital spares

A summary of the other indirect costs is detailed Section 21.1.2.2.

21.1.2.13 Contingency Provision and Management Reserve

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. Another way to describe this contingency provision is a budget provision that is expected to be used for cost items that are known to be required but are currently not estimated due to level of definition inherent in a Feasibility Study i.e., “Known Unknowns”.

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 occurs during the course of the project, as knowledge becomes firmer.
- Design Omissions.

The contingency provision does not include an allowance for the issues addressed under management reserve.

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.

The selection of contingency provisions for the FS was based on a Contingency Matrix and engineer experience and recognizes the body of knowledge available from which this FS has been developed.

MANAGEMENT RESERVE

The “Management Reserve” is the percentage range above and below the expected cost, within which the actual cost is expected to lie.

The Management Reserve provision represents costs which may be incurred to complete the project and therefore funds must be accessible by the company however, they are not part of the total funds placed under the direct control of the project manager and are not *expected* to be used. Another way to interpret this sum of money is to cover events or items that are unexpected or unanticipated, i.e., “Unknown Unknowns”.

The assessment of this range accounts for the areas of uncertainty such as:

- The validity of geological data, site conditions and engineering concepts on which the estimate is based.
- Variation in materials of construction, equipment selection or project standards which may become necessary as engineering exploration and metallurgical testwork advance.
- Effects of unusual weather conditions or other unforeseeable events, which could be encountered, but are abnormally high or low compared to experience on other projects.
- Premium payments if accelerated construction programs are required to recover lost time.
- Unexpected changes in market conditions.
- Ex gratia payments to Contractors to settle disputes.
- Errors in the estimate.

A weighted average Management Reserve of 20% was allowed for in the CCE summary calculation. The selection of Management Reserve quantity will rest with Vista and will be determined by Vista's attitude to risk.

21.1.3 Mine Dewatering

Mine dewatering capital costs are based on direct vendor quotes or Tetra Tech in-house estimates and include 4.25% indirect costs.

Table 21-13: 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	$524	$0	**$524**
2502	Self-Priming Pump on Pontoon	$0	$0	**$0**
2503	Pump	$0	$1,990	**$1,990**
2504	Piping in Pit	$0	$237	**$237**
2505	Piping from Pit to PWP	$0	$937	**$937**
2506	Electrical	$0	$0	**$0**
2507	Indirects	$0	$160	**$160**
	2500 Mine Dewatering/Drainage	**$524**	**$3,324**	**$3,848**

21.1.4 Reclamation and Closure

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$154.9 million for LoM.

Table 21-14: 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	$1,127	**$1,127**
2902	Low Grade Ore Stockpile	$52	$835	**$887**
2903	TSF 1	$80	$32,869	**$32,949**
2904	TSF 2	$1,239	$41,100	**$42,339**
2905	WRD (Liner Cover)	$894	$41,135	**$42,029**
2906	Process Plant Area	$0	$11,837	**$11,837**
2907	Soil Stockpiles	$8	$319	**$327**

WBS No.	Description	Initial Capital (US$000s)	Sustaining Capital (US$000s)	Total Capital (US$000s)
2908	Mine Roads	$0	$465	**$465**
2909	Batman Pit	$0	$254	**$254**
2910	Passive Treatment Systems	$0	$1183	**$1,183**
2911	Indirect Costs	$352	$21,104	**$21,456**
	2900 Mine Closure	**$2,626**	**$152,228**	**$154,854**

21.1.5 Water Treatment Plant

Water treatment plant capital costs are based on direct vendor quotes or Tetra Tech in-house estimates, initial capital costs are estimated at US$34.4 million and no sustaining capital improvements are expected.

Table 21-15: 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	$0	$0	$0
4112	Concrete	$1,545	$0	$1,545
4113	Building	$3,404	$0	$3,404
4114	Equipment	$18,342	$0	$18,342
4115	Mechanical	$3,819	$0	$3,819
4116	Electrical and Instrumentation	$4,582	$0	$4,582
4117	Engineering Procurement	$2,332	$0	$2,332
4118	Construction Management	$403	$0	$403
	4110 Water Treatment Plant	**$34,428**	**$0**	**$34,428**

21.1.6 Raw Water Dam

Raw water dam capital costs are based on direct vendor quotes or Tetra Tech in-house estimates, initial capital costs are estimated at US$9.4 million and no sustaining capital improvements are expected.

Table 21-16: 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	General & Site Preparation	$1,706	$0	$1,706
4122	Main Dam Materials and Construction	$2,655	$0	$2,655
4123	Main Dam Outlet Works	$35	$0	$35
4124	Saddle Dam Materials and Construction	$85	$0	$85
4125	Pump Station	$260	$0	$260
4126	Transmission Main	$1,1,418	$0	$1,418
4127	Pump Operation	$0	$1,373	$1,373
4128	Engineering Design	$924	$0	$924
4129	General, Admin, Construction Observation	$924	$0	$924
	4120 Raw Water Dam	**$8,006**	**$1,373**	**$9,379**

21.1.7 Tailings Storage Facilities

Tailings storage facility capital costs are based on direct vendor quotes or Tetra Tech in-house estimates. Initial capital costs are estimated at US$10 million with sustaining capital of US$92.2 million.

Table 21-17: 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	$583	$2,371	$2,954
2	Embankment Construction	$330	$5,887	$6,217
3	Downstream Embankment Toe Drain	$393	0	$393
4	Tailings Delivery & Return Pipelines	$3,476	$319	$3,795
5	Return Water Ponds	$180	0	$180
6	Diversion Channels	0	0	0
7	Equipment Purchase	$3,429	0	$3,429
8	Mobilization	$381	$390	$771
9	EPCM	$1,259	$1,286	$2,545
10	Instrumentation	$0	$518	$518
	4410 TSF 1	**$10,031**	**$10,771**	**$20,802**
4420	**TSF 2**			
1	Site & Foundation Preparation	$0	$18,390	$18,390
2	Underdrain Construction	$0	$889	$889
3	Downstream Toe Drain	$0	$655	$655
4	Embankment Construction	$0	$6,840	$6,840
5	Impoundment Liner	$0	$22,564	$22,564
6	Overdrain & Reclaim Sump/Pond Construction	$0	$2,662	$2,662
7	Tailings Delivery & Return Pipelines	$0	$12,208	$12,208
8	Surface Water Management	$0	$1,447	$1,447
9	Equipment Purchase	$0	$1,750	$1,750
10	Mobilization	$0	$3,064	$3,064
11	EPCM	$0	$10,111	$10,111
12	Instrumentation	$0	$828	$828
	4420 TSF 2	**$0**	**$81,406**	**$81,406**
	4400 Tailings Dam	**$10,031**	**$92,177**	**$102,208**

21.2 Operating Costs

LoM operating costs requirements are estimated to be US$19.33/t -milled as summarized in Table 21-18.

Table 21-18: Estimated LoM Operating Costs (US$)

Description	US$/t-milled	US$/t-moved
OPEN PIT MINE		
Mine General Service	0.10	0.03
Mine Maintenance	0.16	0.05
Engineering	0.06	0.02
Geology	0.04	0.01
Drilling	1.12	0.33
Blasting	1.20	0.36
Loading	0.81	0.24
Hauling	3.69	1.10
Mine Support	0.50	0.15
Mine Dewatering	0.01	0.01
Open Pit Mine	**7.68**	**2.30**
CIP PROCESS PLANT		
Labor	0.90	-
3100-Crush/Screen/Stockpile	0.48	-
3200-Reclaim & HPGR	0.78	-
3300-Classification & Grinding	4.17	-
3400-Pre-Leach,Thick/Aeration/CIP	0.21	-
3500-Desorption, Gold Room	0.03	-
3600-Detox & Tailings Pumping	0.09	-
3700-Reagents	3.47	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.02	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**10.30**	-
Project Services	0.30	-
G&A	1.05	-
Operating Costs	**19.33**	-

21.2.1 Mining

Mining costs are shown in Table 21-19. The following subsections describe the operating cost estimate by functionality. The total average mining cost (open pit to primary crusher only) is estimated to be US$2.46/t mined (based on a net operating cost of US$ 2,309 million and 923 million tonnes). The mine operating costs discussed in this section include capitalized operating costs. 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$0.34/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$0.26/t mined. 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$1.22/t. 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$0.16/t mined. Maintenance costs assume the use of MARC costs provided by EMG LLC.

This cost includes wall reinforcement costs to bolt and mesh the ultimate pit high wall on the east side of the pit as recommended by Call & Nicholas.

21.2.1.6 Mine-Maintenance Costs

Most maintenance will be done under a MARC cost structure for 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, and 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%. RESPEC 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. Tire men 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$0.05/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 Services 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. The average life-of-mine costs for General Services are estimated to be US$0.03/t mined.

Engineering and geology services are provided to maintain surveying, mine planning, and ore control for the operations. The average life-of-mine Engineering costs are estimated to be US$0.02/t mined. The average life-of-mine Geology costs are estimated to be US$0.01/t mined.

Table 21-19. Annual Mine Operating Costs

	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	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Total
MINED TONNES																							
Ore to Mill	k tonnes	-	10,037	11,185	16,606	6,914	13,958	17,823	15,141	10,156	10,823	14,194	16,136	17,774	14,553	12,568	11,489	4,574	-	-	-	-	203,931
Ore to Stkpl	k tonnes	119	9,049	7,103	13,249	7,067	9,173	8,954	4,223	348	-	-	1,540	1,902	362	-	-	-	-	-	-	-	63,090
Total Ore Mined	k tonnes	119	19,087	18,287	29,854	13,982	23,132	26,777	19,364	10,504	10,823	14,194	17,676	19,677	14,915	12,568	11,489	4,574	-	-	-	-	267,021
Re-handle Ore	k tonnes	-	2,296	6,565	1,144	10,884	3,792	-	2,609	7,619	6,952	3,556	1,614	-	3,222	5,182	6,261	9,524	5,223	-	-	-	76,444
Re-handle Waste	k tonnes	-	-	-	668	-	-	-	-	-	-	-	-	585	1,234	1,377	1,798	1,780	710	-	1,078	1,078	10,308
Re-handle Sorter Rejects	k tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14,082	14,082	28,164
Waste to Dumps	k tonnes	2,876	21,087	39,130	43,054	73,310	62,720	72,908	75,901	74,284	65,388	54,791	45,286	27,087	9,036	3,127	1,139	206	-	-	-	-	671,331
Total Tonnes Mined	k tonnes	2,995	40,173	57,417	72,909	87,292	85,852	99,685	95,265	84,788	76,210	68,985	62,962	46,764	23,951	15,695	12,628	4,780	-	-	-	-	938,352
Total Tonnes Moved	k tonnes	2,995	42,470	63,982	74,722	98,176	89,644	99,685	97,874	92,406	83,162	72,541	64,576	47,349	28,407	22,254	20,687	16,084	5,933	-	15,160	15,160	1,053,267
Strip Ratio	w:o	24.25	1.10	2.14	1.44	5.24	2.71	2.72	3.92	7.07	6.04	3.86	2.56	1.38	0.61	0.25	0.10	0.05					2.51
Mined Waste to Construction	k tonnes	1,199	1,041	2,130	2,512	2,050	920	870	930	1,160	1,290	1,370	1,450	895	206	43	2	-	-	338	338	-	18,744
Mined Material for Pre-Production	k tonnes	1,796	2,288	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4,084
Mined Waste for Closure	k tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	338	1,416	1,078	2,832
Net Tonnage Mined	**k tonnes**	**-**	**36,844**	**55,287**	**71,065**	**85,242**	**84,932**	**98,815**	**94.335**	**83.628**	**74,920**	**67,615**	**61,512**	**46,453**	**24,979**	**17,029**	**14,424**	**6,560**	**710**	**(676)**	**(676)**	**-**	**922,999**
MINING COSTS																							
Mine General Service	K USD	$406	$1,624	$1,758	$1,760	$2,045	$2,135	$2,144	$2,135	$2,138	$2,138	$2,135	$1,946	$1,567	$1,360	$1,342	$1,342	$859	$213	$25	$110	$110	$29,294
Mine Maintenance	K USD	$459	$2,259	$2,271	$2,547	$3,125	$3,213	$3,226	$3,213	$3,218	$3,218	$3,213	$3,213	$3,218	$2,753	$2,281	$2,276	$2,279	$1,179	$38	$517	$517	$48,236
Engineering	K USD	$263	$930	$1,061	$1,061	$1,064	$1,061	$1,065	$1,061	$1,062	$1,062	$1,061	$1,061	$1,062	$1,062	$1,061	$1,061	$959	$440	$12	$335	$335	$18,140
Geology	K USD	$245	$747	$747	$747	$749	$747	$750	$747	$748	$748	$747	$747	$748	$748	$747	$747	$406	$33	$-	$115	$115	$12,135
Drilling	K USD	$860	$15,841	$20,621	$27,009	$26,511	$28,235	$32,794	$29,902	$25,075	$22,985	$22,067	$21,387	$17,386	$9,958	$7,150	$6,061	$2,345	$-	$-	$-	$-	$316,188
Blasting	K USD	$1,157	$15,717	$21,140	$27,491	$30,213	$30,841	$35,717	$33,412	$28,994	$26,259	$24,307	$22,768	$17,815	$9,939	$7,012	$5,918	$2,347	$13	$-	$13	$13	$341,084
Loading	K USD	$672	$9,489	$15,376	$17,063	$21,907	$20,091	$21,999	$21,889	$21,101	$19,270	$16,754	$14,668	$10,782	$6,781	$5,722	$5,657	$4,489	$1,621	$-	$4,015	$4,015	$243,361
Hauling	K USD	$1,814	$27,843	$46,016	$63,923	$88,169	$99,070	$105,921	$103,739	$101,251	$100,762	$100,468	$100,636	$78,124	$43,162	$30,886	$26,184	$13,520	$2,455	$-	$6,309	$6,309	$1,146,562
Mine Support	K USD	$2,401	$6,624	$7,621	$9,090	$9,426	$9,512	$12,722	$13,743	$13,299	$12,932	$11,877	$9,463	$6,309	$4,792	$4,741	$5,332	$5,414	$2,444	$-	$3,000	$3,000	$153,741
Total Mine Cost	**K USD**	**$8,276**	**$81,075**	**$116,611**	**$150,691**	**$183,209**	**$194,907**	**$216,339**	**$209,843**	**$196,886**	**$189,374**	**$182,630**	**$175,890**	**$137,012**	**$80,557**	**$60,942**	**$54,579**	**$32,619**	**$8,397**	**$74**	**$14,414**	**$14,414**	**$2,308,740**
MINE COST PER TONNE MINED																							
Mine General Service	$ /t	$ 0.14	$ 0.04	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.03	$ 0.06	$ 0.09	$ 0.11	$ 0.18	$ -	$ -	$ -	$ -	$ 0.03
Mine Maintenance	$ /t	$ 0.15	$ 0.06	$ 0.04	$ 0.03	$ 0.04	$ 0.04	$ 0.03	$ 0.03	$ 0.04	$ 0.04	$ 0.05	$ 0.05	$ 0.07	$ 0.11	$ 0.15	$ 0.18	$ 0.48	$ -	$ -	$ -	$ -	$ 0.05
Engineering	$ /t	$ 0.09	$ 0.02	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.07	$ 0.08	$ 0.20	$ -	$ -	$ -	$ -	$ 0.02
Geology	$ /t	$ 0.08	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.03	$ 0.05	$ 0.06	$ 0.09	$ -	$ -	$ -	$ -	$ 0.01
Drilling	$ /t	$ 0.29	$ 0.39	$ 0.36	$ 0.37	$ 0.30	$ 0.33	$ 0.33	$ 0.31	$ 0.30	$ 0.30	$ 0.32	$ 0.34	$ 0.37	$ 0.42	$ 0.46	$ 0.48	$ 0.49	$ -	$ -	$ -	$ -	$ 0.34
Blasting	$ /t	$ 0.39	$ 0.39	$ 0.37	$ 0.38	$ 0.35	$ 0.36	$ 0.36	$ 0.35	$ 0.34	$ 0.34	$ 0.35	$ 0.36	$ 0.38	$ 0.41	$ 0.45	$ 0.47	$ 0.49	$ -	$ -	$ -	$ -	$ 0.36
Loading	$ /t	$ 0.22	$ 0.24	$ 0.27	$ 0.23	$ 0.25	$ 0.23	$ 0.22	$ 0.23	$ 0.25	$ 0.25	$ 0.24	$ 0.23	$ 0.23	$ 0.28	$ 0.36	$ 0.45	$ 0.94	$ -	$ -	$ -	$ -	$ 0.26
Hauling	$ /t	$ 0.61	$ 0.69	$ 0.80	$ 0.88	$ 1.01	$ 1.15	$ 1.06	$ 1.09	$ 1.19	$ 1.32	$ 1.46	$ 1.60	$ 1.67	$ 1.80	$ 1.97	$ 2.07	$ 2.83	$ -	$ -	$ -	$ -	$ 1.22
Mine Support	$ /t	$ 0.80	$ 0.16	$ 0.13	$ 0.12	$ 0.11	$ 0.11	$ 0.13	$ 0.14	$ 0.16	$ 0.17	$ 0.17	$ 0.15	$ 0.13	$ 0.20	$ 0.30	$ 0.42	$ 1.13	$ -	$ -	$ -	$ -	$ 0.16
Total Mine Cost	**$ /t**	**$ 2.76**	**$ 2.02**	**$ 2.03**	**$ 2.07**	**$ 2.10**	**$ 2.27**	**$ 2.17**	**$ 2.20**	**$ 2.32**	**$ 2.48**	**$ 2.65**	**$ 2.79**	**$ 2.93**	**$ 3.36**	**$ 3.88**	**$ 4.32**	**$ 6.82**	**$ -**	**$ -**	**$ -**	**$ -**	**$ 2.46**

21.2.2 Mine Dewatering

Operating costs are related to fuel consumption and routine maintenance of diesel engines for 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 operating costs were estimated to total US$3.8 million, for an average of US$0.012/t-milled.

21.2.3 CIP Process and G&A

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 AUD12.24/t treated as shown in Table 21-20.

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 21-20: 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%**
FUEL				
Vehicles	420,000	0.02		
Plant Gas	710,000	0.04		
Total	**1,130,000**	**0.06**	**2.52**	**0.5%**

Cost Center		OPERATING COST AUD/ a	AUD/ t	AUD/oz	%
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 RESPEC'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 ongoing 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 ongoing 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 industry experience.
- Flocculant consumption was changed to 40 g/t for the Pre-Leach thickener 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) 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 ongoing 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 most economical 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 Water Treatment Plant

Water treatment plant operating costs averaging US$0.09/t-milled.

21.2.5 Tailings Storage Facilities

Tailings operating costs are estimated to average US$0.22/t-milled over the LoM. Tailings operating costs include shaping and compaction of the mine waste in the tailings embankments that will be hauled as a mining cost. Pumping and power costs for tailings facility operation are included in the Process Plant costing.

21.2.6 General & Administrative

G&A is estimated to be an average of US$1.11/t-milled over the LoM.

22. ECONOMIC ANALYSIS

Project economics for the 50,000 tpd operation are based on inputs developed by RESPEC, TTP, and Tetra Tech. Economic results presented in the report suggest the following conclusions, assuming a 100% equity project, a gold price of US$1,800/oz and a US$0.69:AUD1.00 exchange rate:

- Mine Life .... 16 years;
- Pre-Tax NPV5% .... US$2,149.4 million, IRR: 29.4%;
- After-tax NPV5% .... US$1,131.4 million, IRR: 20.4%;
- Payback (After-tax) .... 4.0 years;
- NT Royalty Paid .... US$765 million;
- Australian Income Taxes Paid .... US$927 million; and
- Cash costs (including Royalties) .... US$913/oz-Au.

Costs and economic results are presented in Q4 2021 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, which 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	16
Closure Period	Years	4
Operating Days	Days/Year	355
Gold Price	US$	$1,800
JAAC Royalty	%	2%
Wheaton Royalty	%	1%
Exchange Rate	AUD:US$	0.69:1
Diesel Fuel	AUD/L	$1.05
Natural Gas	AUD/GJ	$8.00
Electric Power – From Grid	AUD/kWh	$0.300
Electric Power – From 3rd Party	AUD/kWh	$0.128

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$1,800/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$3.68/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	671,331	-	-
Ore	267,021	0.79	6,747
Heap Leach	13,354	0.54	232
Total Production*	**280,375**	**0.77**	**6,979**

*Total production excludes waste tonnes.

The Project has been planned as an open-pit truck and shovel operation. Open pit ore totals 267 Mt grading 0.79 g/t and contains 6.7 Moz of gold. Open pit production will have a 2.5:1 strip ratio over the 17-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—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 92.61%, 92.74%, and 90.92%, respectively. The heap leach pad will have a recovery of 90.74%. An additional 1% for net solution loss is applied to all the deposits and heap leach which results in a LoM average recovery of 91.6%.

22.3 Capital Costs

LoM capital cost requirements are estimated at US$1,746 million as summarized in Table 22-4. Initial capital of US$1,030 million is estimated to be required to commence operations. Sustaining capital of US$716 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	6.2%	$94,127	$5,738	**$99,865**	$583,957	$36,152	**$620,109**	$678,084	$41,890	**$719,974**
3000	Process Plant	11.0%	$560,796	$61,691	**$622,487**	$33,498	$3,539	**$37,036**	$594,294	$65,229	**$659,523**
4000	Project Services	9.6%	$56,893	$6,625	**$63,518**	$86,468	$7,082	**$93,551**	$143,361	$13,707	**$157,069**
5000	Project Infrastructure	10.5%	$49,389	$5,203	**$54,592**	$7,502	$773	**$8,275**	$56,891	$5,976	**$62,867**
6000	Permanent Accommodation	10.0%	$422	$42	**$464**	$0	$0	**$0**	$422	$42	**$464**
7000	Site Establishment & Early Works	12.6%	$26,553	$3,334	**$29,886**	$0	$0	**$0**	$26,553	$3,334	**$29,886**
8000	Management, Engineering, EPCM Svcs	12.0%	$111,185	$13,384	**$124,569**	$0	$0	**$0**	$111,185	$13,384	**$124,569**
9000	Pre-Production Costs	10.0%	$31,071	$3,098	**$34,169**	$0	$0	**$0**	$31,071	$3,098	**$34,169**
10000	Asset Sale	0.0%	$0	$0	**$0**	($42,756)	$0	**($42,756)**	($42,756)	$0	**($42,756)**
	Capital Cost	**9.2%**	**$930,436**	**$99,114**	**$1,029,550**	**$668,670**	**$47,546**	**$716,216**	**$1,599,106**	**$146,660**	**$1,745,766**

22.3.1 2000 Mining

LoM capital cost requirements are estimated at US$720 million with an initial cost of US$100 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%	$8,191	$819	**$9,010**	$130,419	$13,042	**$143,461**	$138,610	$13,861	**$152,471**
2200	Mine Production Equipment	2.9%	$50,661	$1,457	**$52,118**	$309,413	$8,902	**$318,315**	$360,074	$10,359	**$370,433**
2300	Mine Support Facilities	9.4%	$30,935	$2,912	**$33,847**	$2,540	$239	**$2,779**	$33,475	$3,151	**$36,626**
2400	Mine Support Services	20.0%	$1,450	$290	**$1,740**	$0	$0	**$0**	$1,450	$290	**$1,740**
2500	Mine Dewatering/Drainage	4.1%	$504	$21	**$524**	$3,194	$130	**$3,324**	$3,698	$151	**$3,848**
2900	Mine Closure	10.0%	$2,387	$239	**$2,626**	$138,390	$13,839	**$152,229**	$140,777	$14,078	**$154,854**
	Mining	**6.2%**	**$94,127**	**$5,738**	**$99,865**	**$583,957**	**$36,152**	**$620,109**	**$678,084**	**$41,890**	**$719,974**

22.3.2 3000 Process Plant

Estimated CIP process plant capital costs are shown in Table 22-6. Initial capital totaling US$622 million is estimated to be required for the CIP process plant; a total capital of US$660 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	10.4%	$64,690	$6,698	**$71,387**	$5,932	$614	**$6,546**	$70,622	$7,312	**$77,934**
3200	Coarse Ore Stockpile, Reclaim, HPGR	11.3%	$113,130	$12,810	**$125,940**	$8,178	$926	**$9,104**	$121,308	$13,736	**$135,044**
3300	Classification & Grinding	9.8%	$142,131	$13,963	**$156,094**	$13,015	$1,279	**$14,294**	$155,146	$15,241	**$170,388**
3400	Pre-leach Thickening, Leach & CIP	11.6%	$100,672	$11,672	**$112,343**	$2,659	$308	**$2,967**	$103,330	$11,980	**$115,310**
3500	Desorption & Goldroom	12.6%	$10,525	$1,322	**$11,846**	$1,642	$206	**$1,849**	$12,167	$1,528	**$13,695**
3600	Detoxification & Tailings	13.3%	$11,160	$1,489	**$12,649**	$320	$43	**$363**	$11,480	$1,532	**$13,012**
3700	Reagents	9.0%	$20,527	$1,852	**$22,379**	$1,600	$144	**$1,744**	$22,127	$1,996	**$24,123**

Area	Description	Cont. (%)	INITIAL CAPITAL (US$000s)			SUSTAINING CAPITAL (US$000s)			TOTAL CAPITAL (US$000s)		
			Estimate	Contingency	Total	Estimate	Contingency	Total	Estimate	Contingency	Total
3800	Process Plant Services	12.1%	$97,962	$11,886	**$109,848**	$151	$18	**$170**	$98,113	$11,904	**$110,017**
	Process Plant	**11.0%**	**$560,796**	**$61,691**	**$622,487**	**$33,498**	**$3,539**	**$37,036**	**$594,294**	**$65,229**	**$659,523**

22.3.3 4000 Project Services

Project services are estimated to have a LoM capital cost US$157 million, with an initial capital cost of US$64 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	13.7%	$37,335	$5,098	**$42,434**	$1,208	$165	**$1,373**	$38,544	$5,263	**$43,807**
4200	Power Supply	10.0%	$3,580	$358	**$3,938**	$0	$0	**$0**	$3,580	$358	**$3,938**
4300	Communications	13.6%	$499	$68	**$566**	$0	$0	**$0**	$499	$68	**$566**
4400	Tailings Dams 1 & 2	8.1%	$9,278	$753	**$10,031**	$85,260	$6,917	**$92,178**	$94,538	$7,670	**$102,208**
4500	Waste Disposal	15.0%	$305	$46	**$350**	$0	$0	**$0**	$305	$46	**$350**
4600	Plant Mobile Equipment	5.0%	$5,821	$291	**$6,112**	$0	$0	**$0**	$5,821	$291	**$6,112**
4800	Fuel Storage & Distribution (Plant)	15.0%	$76	$11	**$87**	$0	$0	**$0**	$76	$11	**$87**
4900	Project Services - Closure	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
	Project Services	**9.6%**	**$56,893**	**$6,625**	**$63,518**	**$86,468**	**$7,082**	**$93,551**	**$143,361**	**$13,707**	**$157,069**

22.3.4 5000 Project Infrastructure

The total project infrastructure is estimated to cost US$63 million, with initial costs of US$55 million. 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	10.3%	$30,855	$3,179	**$34,034**	$7,502	$773	**$8,275**	$38,357	$3,952	**$42,309**
5200	Support Buildings	10.1%	$8,047	$813	**$8,860**	$0	$0	**$0**	$8,047	$813	**$8,860**
5300	Access Roads, Parking & Laydown	10.0%	$7,231	$723	**$7,954**	$0	$0	**$0**	$7,231	$723	**$7,954**
5400	Heavy Lift Cranage	15.0%	$2,525	$379	**$2,903**	$0	$0	**$0**	$2,525	$379	**$2,903**
5500	TBA	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
5600	Bulk Transport	15.0%	$453	$68	**$521**	$0	$0	**$0**	$453	$68	**$521**
5700	Power Transmission	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
5800	Communications	15.0%	$278	$42	**$320**	$0	$0	**$0**	$278	$42	**$320**
	Project Infrastructure	**10.5%**	**$49,389**	**$5,203**	**$54,592**	**$7,502**	**$773**	**$8,275**	**$56,891**	**$5,976**	**$62,867**

22.3.5 6000 Permanent Accommodation

Total capital for Permanent Accommodations is estimated at US$464 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%	$422	$42	**$464**	$0	$0	**$0**	$422	$42	**$464**
	Permanent Accommodation	**10.0%**	**$422**	**$42**	**$464**	**$0**	**$0**	**$0**	**$422**	**$42**	**$464**

22.3.6 7000 Site Establishment & Early Works

Site Establishment and early works capital costs are estimated to total US$30 million 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	12.6%	$26,553	$3,334	**$29,886**	$0	$0	**$0**	$26,553	$3,334	**$29,886**
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	**12.6%**	**$26,553**	**$3,334**	**$29,886**	**$0**	**$0**	**$0**	**$26,553**	**$3,334**	**$29,886**

22.3.7 8000 Management, Engineering, EPCM Services

Management, engineering, and EPCM services are estimated to cost US$125 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	12.5%	$63,895	$7,987	**$71,882**	$0	$0	**$0**	$63,895	$7,987	**$71,882**
8200	External Consulting & Testing	12.5%	$828	$103	**$931**	$0	$0	**$0**	$828	$103	**$931**
8300	Commissioning	12.5%	$5,901	$738	**$6,638**	$0	$0	**$0**	$5,901	$738	**$6,638**
8400	Owner's Engineering & Management	11.0%	$34,311	$3,775	**$38,086**	$0	$0	**$0**	$34,311	$3,775	**$38,086**
8800	License, fees & Legal Services	12.5%	$3,125	$391	**$3,515**	$0	$0	**$0**	$3,125	$391	**$3,515**
8900	Project Insurance	12.5%	$3,125	$391	**$3,515**	$0	$0	**$0**	$3,125	$391	**$3,515**
	Management, Engineering, EPCM Svcs	**12.0%**	**$111,185**	**$13,384**	**$124,569**	**$0**	**$0**	**$0**	**$111,185**	**$13,384**	**$124,569**

22.3.8 9000 Pre-Production Costs

Pre-production capitalized cost is estimated at US$34 million as shown in Table 22-12. This cost will occur during 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	12.5%	$1,562	$195	**$1,758**	$0	$0	**$0**	$1,562	$195	**$1,758**
9200	Commissioning Expenses	12.5%	$2,789	$349	**$3,138**	$0	$0	**$0**	$2,789	$349	**$3,138**
9300	Capital Spares	12.5%	$18,294	$2,287	**$20,581**	$0	$0	**$0**	$18,294	$2,287	**$20,581**
9400	Stores & Inventory	12.5%	$2,137	$267	**$2,404**	$0	$0	**$0**	$2,137	$267	**$2,404**
9500	PPD Capitalized Operating	0.0%	$6,288	$0	**$6,288**	$0	$0	**$0**	$6,288	$0	**$6,288**
9600	Escalation & Foreign Currency Exchange	0.0%	$0	$0	**$0**	$0	$0	**$0**	$0	$0	**$0**
	Pre-Production Costs	**10.0%**	**$31,071**	**$3,098**	**$34,169**	**$0**	**$0**	**$0**	**$31,071**	**$3,098**	**$34,169**

22.3.9 10000 Asset Sale

Table 22-13 depicts a total asset sale value of US$43 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**	($24,494)	$0	**($24,494)**	($24,494)	$0	**($24,494)**
10200	Process Plant	0.0%	$0	$0	**$0**	($18,262)	$0	**($18,262)**	($18,262)	$0	**($18,262)**
	Asset Sale	**0.0%**	**$0**	**$0**	**$0**	**($42,756)**	**$0**	**($42,756)**	**($42,756)**	**$0**	**($42,756)**

22.4 Operating Costs

Estimated LoM operating costs are summarized in Table 22-14. The operating costs will average US$19.33/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	0.16	0.05
Engineering	0.06	0.02
Geology	0.04	0.01
Drilling	1.12	0.33
Blasting	1.20	0.36
Loading	0.81	0.24
Hauling	3.69	1.10
Mine Support	0.50	0.15
Mine Dewatering	0.01	0.01
Open Pit Mine	**7.68**	**2.30**
CIP PROCESS PLANT		
Labor	0.90	-
3100-Crush/Screen/Stockpile	0.48	-
3200-Reclaim & HPGR	0.78	-
3300-Classification & Grinding	4.17	-
3400-Pre-Leach,Thick/Aeration/CIP	0.21	-
3500-Desorption, Gold Room	0.03	-
3600-Detox & Tailings Pumping	0.09	-
3700-Reagents	3.47	-
3800-Plant Services	0.04	-
Mining, Infrastructure & Misc	0.06	-
General Consumables	0.01	-
Plant Mobile Equipment	0.02	-
Plant Gas Consumption	0.03	-
CIP Process Plant	**10.30**	**-**
Project Services	0.30	-
G&A	1.05	-
Operating Costs	**19.33**	**-**

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$2.30/t-mined (US$7.68/t-milled) over the LoM. Hauling is the highest cost item, US$1.10/t-mined (US$3.69/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	**$27,373**
Mine Maintenance	$0.05	$0.16	**$44,123**
Engineering	$0.02	$0.06	**$16,114**
Geology	$0.01	$0.04	**$11,365**
Drilling	$0.33	$1.12	**$313,015**
Blasting	$0.36	$1.20	**$336,806**
Loading	$0.24	$0.81	**$227,122**
Hauling	$1.10	$3.69	**$1,033,327**
Mine Support	$0.15	$0.50	**$139,679**
Subtotal	**$2.29**	**$7.66**	**$2,148,925**
Mine Dewatering	$0.005	$0.015	**$4,267**
Total Open Pit Mining	**$2.30**	**$7.68**	**$2,153,192**

22.4.2 CIP Process Plant

CIP process plant operating costs averaging US$10.30/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	$0.90	$251,205
3100 – Crush/Screen/Stockpile	$0.48	$135,113
3200 – Reclaim & HPGR	$0.78	$219,162
3300 – Classification & Grinding	$4.17	$1,170,235
3400 – Pre-Leach, Thick/Aeration/CIP	$0.21	$58,078
3500 – Desorption, Gold Room	$0.03	$7,557
3600 – Detox & Tailings Pumping	$0.09	$26,536
3700 – Reagents	$3.47	$973,956
3800 – Plant Services	$0.04	$11,006
Mining, Infrastructure, & Misc	$0.06	$18,062
Generable Consumables	$0.01	$3,368
Plant Mobile Equipment	$0.02	$5,428
Plant Gas consumption	$0.03	$9,459
Total CIP Process Plant	**$10.30**	**$2,889,166**

22.4.3 Water Treatment Plant

Water treatment plant operating costs averaging US$0.15/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	$0.00	$0
Chlorine	$0.00	$0
Polymer	$0.00	$9
Ferric Chloride	$0.03	$9,594
Ferrous Sulfate	$0.00	$0
Lime	$0.03	$8,362
Sodium Hydrosulfide	$0.01	$3,699
Sulfuric Acid	$0.00	$802
POWER		
Electricity	$0.03	$9,712
LABOR		
Operator	$0.01	$2,773
Maintenance	$0.02	$6,196
Total Water Treatment Plant	**$0.15**	**$41,148**

22.4.4 Tailings

Tailings will average US$0.17/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	$0.04	$10,866
Equipment	$0.13	$36,224
Total Tailings	**$0.17**	**$47,090**

22.4.5 General & Administrative

G&A will average US$1.05/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	$0.42	$116,868
Expenses	$0.18	$51,327
Transport & Accommodation	$0.09	$25,733
Fleet Vehicles	$0.02	$4,910
Corporate Overhead	$0.34	$94,375
Total G&A	**$1.05**	**$293,212**

22.4.6 Royalties

JAAC and Wheaton Royalty costs averaging US$1.16/t-milled are shown in Table 22-20.

Table 22-20: Estimated Royalty Costs (US$)

	US$/t-milled	Total (US$000s)
JAAC	$0.81	$227,286
Wheaton	$0.35	$96,753
Total Royalties	**$1.16**	**$324,038**

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	$0.02	$4,735
Golden Retention	$0.04	$11,364
Purchase Discount-Gold	$0.01	$3,157
Assay Fee	$0.00	$948
Environmental Fee	$0.01	$1,733
Freight & Insurance	$0.00	$1,268
Total Refinery Costs	**$0.08**	**$23,206**

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	$394,334	27.0%	$106,470	1	DP	1	$500,804
Mining Manager	$287,216	27.0%	$77,548	1	DP	1	$364,765
Processing Manager	$287,216	27.0%	$77,548	1	DP	1	$364,765
Admin Manager	$238,955	27.0%	$64,518	0	DP	0	$0
OHS Manager	$209,527	27.0%	$56,572	0	DP	0	$0
NPI Manager	$287,216	27.0%	$77,548	1	DP	1	$364,765
Subtotal				**4**		**4**	**$1,595,099**
HR Director	$208,049	27.0%	$56,173	1	DP	1	$264,222
Recruiting Officer	$121,646	27.0%	$32,844	2	DP	2	$308,981
Administration Secretary	$97,772	27.0%	$26,398	1	DP	1	$124,170
Administrative Assistant	$92,087	27.0%	$24,864	1	SW	3	$350,852
Receptionist	$75,034	27.0%	$20,259	1	DP	1	$95,293
Indigenous Liaison Officer	$109,140	27.0%	$29,468	1	DP	1	$138,608
Security Officer	$92,087	27.0%	$24,864	2	SW	6	$701,704
Community Liaison Officer	$109,140	27.0%	$29,468	1	DP	1	$138,608
Head of Security	$127,330	27.0%	$34,379	1	DP	1	$161,710
External Affairs Director	$202,364	27.0%	$54,638	1	DP	1	$257,003
Support Services Director	$202,364	27.0%	$54,638	1	DP	1	$257,003
Subtotal				**13**		**19**	**$2,798,154**
Financial Controller	$208,049	27.0%	$56,173	1	DP	1	$264,222
Senior Accountant	$155,752	27.0%	$42,053	1	DP	1	$197,806
Accountant	$127,330	27.0%	$34,379	1	DP	1	$161,710
Accounting Clerk	$86,403	27.0%	$23,329	1	DW	2	$219,463

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Payroll Clerk	$86,403	27.0%	$23,329	1	DP	1	$109,732
Subtotal				**5**		**6**	**$952,932**
IT Supervisor	$127,330	27.0%	$34,379	1	DP	1	$161,710
IT Technician	$103,456	27.0%	$27,933	1	DP	1	$131,389
Database Administrator	$103,456	27.0%	$27,933	1	DP	1	$131,389
Subtotal				**3**		**3**	**$424,488**
Metallurgical Superintendent	$216,759	27.0%	$58,525	1	DP	1	$275,283
Chief Metallurgist	$216,759	27.0%	$58,525	1	DP	1	$275,283
Plant / Production Metallurgist	$191,186	27.0%	$51,620	2	DP	2	$485,612
Process Control Engineer	$166,831	27.0%	$45,044	1	DP	1	$211,875
Metallurgical Clerk	$110,815	27.0%	$29,920	1	SW	3	$422,204
Gold Room Supervisor	$136,387	27.0%	$36,825	1	DP	1	$173,212
Refiner	$116,903	27.0%	$31,564	1	DW	2	$296,935
Gold Room Technician	$110,815	27.0%	$29,920	1	SW	3	$422,204
Subtotal				**9**		**14**	**$2,562,609**
Production Superintendent	$222,847	27.0%	$60,169	1	DP	1	$283,016
General Foreman	$185,097	27.0%	$49,976	1	DP	1	$235,073
Shift Foreman	$154,654	27.0%	$41,756	1	SW	3	$589,230
Plant Lead Operator	$136,387	27.0%	$36,825	1	SW	3	$519,636
Shift Operator - Crushing	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - HPGR	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - Mills	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - Leach	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - Elution	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - Detox / Tailings	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - Reagents	$124,210	27.0%	$33,537	2	SW	6	$946,480
Shift Operator - CCR	$124,210	27.0%	$33,537	1	SW	3	$473,240
Shift Operator - Tailings Dam	$110,815	27.0%	$29,920	2	SW	6	$844,408
Shift Operator - Day Gang	$110,815	27.0%	$29,920	6	DW	12	$1,688,817

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Subtotal				**21**		**53**	**$8,419,339**
Maintenance Superintendent	$195,198	27.0%	$52,703	1	DP	1	$247,901
Maintenance General Foreman	$166,686	27.0%	$45,005	1	DP	1	$211,691
Maintenance Planner	$172,169	27.0%	$46,486	1	DP	1	$218,655
Mechanical Fitter	$133,787	27.0%	$36,123	3	SW	9	$1,529,190
Crane Operator	$111,855	27.0%	$30,201	1	DW	2	$284,112
Boilermaker / Welder	$139,270	27.0%	$37,603	2	DW	4	$707,494
Pipe Fitters	$139,270	27.0%	$37,603	1	DW	2	$353,747
Greasers	$99,792	27.0%	$26,944	1	SW	3	$380,208
Trades Assistants	$94,309	27.0%	$25,463	1	SW	3	$359,318
Electrical General Foreman	$161,203	27.0%	$43,525	1	DP	1	$204,728
HV Electrical Supervisor	$133,787	27.0%	$36,123	1	DW	2	$339,820
Electrician	$133,787	27.0%	$36,123	3	SW	9	$1,529,190
Instrument Technician	$133,787	27.0%	$36,123	1	SW	3	$509,730
Apprentices	$66,894	27.0%	$18,061	2	SW	6	$509,730
Subtotal				**20**		**47**	**$7,385,512**
Laboratory Supervisor	$154,654	27.0%	$41,756	1	SW	3	$589,230
Chemist	$148,565	27.0%	$40,113	1	SW	3	$566,032
Lab Technician	$124,210	27.0%	$33,537	2	SW	6	$946,480
Sample Prep Technician	$92,549	27.0%	$24,988	3	SW	9	$1,057,830
Subtotal				**7**		**21**	**$3,159,572**
Engineering Superintendent	$214,863	27.0%	$58,013		DP		$0
Chief Mining Engineer	$190,206	27.0%	$51,356		DP	1	$241,562
Senior Mining Engineer	$166,724	27.0%	$45,015		DP		$0
Mining Engineer	$154,983	27.0%	$41,845		DW	3	$590,485
Senior Mine Planning Engineer	$154,983	27.0%	$41,845		DP		$0
Mine Clerk	$119,759	27.0%	$32,335		DP	1	$152,095
Subtotal				**0**		**5**	**$984,141**
Operations Superintendent	$196,093	27.0%	$52,945		DP		$0

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Mine General Foreman	$162,875	27.0%	$43,976		DP		$0
Drill and Blast Foreman	$136,086	27.0%	$36,743		DW	2	$345,659
Drill and Blast Technician	$97,510	27.0%	$26,328		DW		$0
Blasting Assistant	$92,153	27.0%	$24,881		DW	2	$234,068
Loading Operator	$125,371	27.0%	$33,850		SW	12	$1,910,648
Haul Truck Operator	$109,297	27.0%	$29,510		SW	88	$12,215,085
Drill Operators	$125,371	27.0%	$33,850		SW	27	$4,298,959
Mechanics	$130,728	27.0%	$35,297		SW		$0
Welders	$136,086	27.0%	$36,743		SW		$0
Servicemen	$86,795	27.0%	$23,435		SW		$0
Aux Equipment Operators	$114,655	27.0%	$30,957		SW	18	$2,621,018
Mine Shift Foreman	$136,086	27.0%	$36,743		SW	9	$1,555,464
Subtotal				**0**		**158**	**$23,180,901**
Maintenance Superintendent	$190,735	27.0%	$51,498		DP	1	$242,233
Maintenance General Foreman	$162,875	27.0%	$43,976		DP	2	$413,702
Light Vehicle Mechanic	$130,728	27.0%	$35,297		DW	2	$332,050
Tireman	$97,510	27.0%	$26,328		DW	2	$247,677
Shop Laborer	$92,153	27.0%	$24,881		SW	2	$234,068
Service, Fuel & Lube	$86,795	27.0%	$23,435		SW	5	$551,149
Maintenance Planner	$136,086	27.0%	$36,743		DP	2	$345,659
Subtotal				**0**		**16**	**$2,366,537**
Chief Surveyor	$178,465	27.0%	$48,186		DP		$0
Mine Surveyor	$137,371	27.0%	$37,090		DW	2	$348,923
Surveying Helper	$95,103	27.0%	$25,678		DW	2	$241,562
Subtotal				**0**		**4**	**$590,485**
Geology Superintendent	$222,825	27.0%	$60,163		DP	1	$282,988
Grade Control Geologist	$184,018	27.0%	$49,685		DW	2	$467,407
Exploration Geologist	$140,205	27.0%	$37,855		DP		$0
Resource Geologist	$196,537	27.0%	$53,065		DP		$0

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Pit Geology Technician	$113,916	27.0%	$30,757		DP		$0
Geology Field Technician	$101,398	27.0%	$27,377		DP		$0
Sampler	$101,398	27.0%	$27,377		DP	2	$257,551
Subtotal				**0**		**5**	**$1,007,946**
Purchasing Director	$170,956	27.0%	$46,158	1	DP	1	$217,114
Business Development Officer	$107,972	27.0%	$29,152	1	DP	1	$137,124
Logistics Officer	$148,461	27.0%	$40,085	1	DP	1	$188,546
Purchasing Officer	$107,972	27.0%	$29,152	1	DP	1	$137,124
Contracts Officer	$125,967	27.0%	$34,011	1	DP	1	$159,978
Store Person	$96,725	27.0%	$26,116	1	SW	3	$368,522
Subtotal				**6**		**8**	**$1,208,409**
OHS Superintendent	$197,170	27.0%	$53,236	1	DP	1	$250,406
Safety Officer	$129,601	27.0%	$34,992	1	SW	3	$493,779
Paramedic / Nurse	$129,601	27.0%	$34,992	1	SW	3	$493,779
Environmental Superintendent	$168,370	27.0%	$45,460	1	DP	1	$213,830
Environmental Officer - Monitoring	$118,524	27.0%	$32,001	1	DP	1	$150,525
Environmental Officer - Compliance	$118,524	27.0%	$32,001	1	DP	1	$150,525
Subtotal				**6**		**10**	**$1,752,843**
Training Coordinator	$168,370	27.0%	$45,460	1	DP	1	$213,830
Training Officer - Plant	$135,139	27.0%	$36,488	2	DW	4	$686,507
Training Officer - Mining	$135,139	27.0%	$36,488		DW	1	$171,627
Subtotal				**3**		**6**	**$1,071,964**
Camp Manager	$109,140	27.0%	$29,468	0	DP	0	$0
Camp Admin	$80,718	27.0%	$21,794	0	DW	0	$0
Cook Staff	$97,772	27.0%	$26,398	0	SW	0	$0
Cleaning Staff	$97,772	27.0%	$26,398	0	DW	0	$0
Camp Maintenance	$115,962	27.0%	$31,310	0	DP	0	$0
Bus Drivers	$92,087	27.0%	$24,864	4	SW	12	$1,403,409
Subtotal				**4**		**12**	**$1,403,409**

	Salary AUD	Salary On-Costs %	AUD	Number of Employees per Shift	Shift Codes	Total Employees Required	Annual Labor Costs Total AUD/Annum
Power Station Operator	$99,792	27.0%	$26,944	1	SW	3	$380,208
Electrician	$133,787	27.0%	$36,123	1	SW	3	$509,730
Mechanic	$133,787	27.0%	$36,123	1	SW	3	$509,730
Subtotal				**3**		**9**	**$1,399,668**
Power & Water Superintendent	$166,686	27.0%	$45,005	1	DP	1	$211,691
Water Plant Operator	$99,792	27.0%	$26,944	1	SW	3	$380,208
Water Plant Mechanic	$133,787	27.0%	$36,123	1	SW	3	$509,730
Subtotal				**3**		**7**	**$1,101,629**
Dozer Operator	$114,655	27.0%	$30,957	0	DW	0	$0
Loader Operator	$114,655	27.0%	$30,957	0	DP	0	$0
Haul Truck Operator	$109,297	27.0%	$29,510	0	DP	0	$0
Subtotal				**0**		**0**	**$0**
Dozer Operator	$114,655	27.0%	$30,957	0	DP	0	$0
Loader Operator	$114,655	27.0%	$30,957	0	DP	0	$0
Haul Truck Operator	$109,297	27.0%	$29,510	0	DP	0	$0
Crane Operator	$120,013	27.0%	$32,403	0	DP	0	$0
Subtotal				**0**		**0**	**$0**
Project Superintendent	$214,055	27.0%	$57,795	1	DP	1	$271,850
Project Engineer	$176,095	27.0%	$47,546	1	DW	2	$447,281
Civil Engineer	$139,189	27.0%	$37,581	0	DW	0	$0
Geotechnical Engineer	$160,278	27.0%	$43,275	0	DW	0	$0
CAD Draftsman	$93,847	27.0%	$25,339	0	DW	0	$0
Piping Engineer	$112,827	27.0%	$30,463	0	DW	0	$0
Document Controller	$80,139	27.0%	$21,638	0	DW	0	$0
Construction Supervisor	$139,189	27.0%	$37,581	0	DW	0	$0
Subtotal				**2**		**3**	**$719,131**
TOTAL ONSITE PERSONNEL						**410**	**$64,084,767**

*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
$394,334								Resident Manager
$287,216		Mining Manager			Processing Manager		NPI Manager	
$238,955								Admin Manager*
$214,055							Projects Supt.	
$208,049	Engineering Supt.	Operations Supt.			Production Supt			Financial Controller
								HR Director
$209,527	Geology Supt.		Maintenance Supt	Metallurgy Supt		Maintenance Supt.		OHS Supt.*
								External Affairs Director
								Support Services Director
$176,095							Project Engineer	
$190,206	Chief Mining Engineer							
$216,759	Resource Geologist			Chief Metallurgist		Maintenance Planner		
$178,465	Chief Surveyor	Mine General Foreman		Metallurgist	General Foreman	Mechanical General Foreman	Power & Water Supt	Environmental Supt
								Purchasing Director
								Training Coordinator
$184,018	Ore Control Geologist							
$166,686			Maintenance General Foreman			Electrical General Foreman		
$155,752								Sr. Accountant

Salary (AUD)	Mine Technical	Mine Operations	Mine Maintenance	Plant Technical	Plant Operations	Plant Maintenance	NPI	Administration
$154,983	Sr. Mine Planning Engineer							Logistics Officer
	Mining Engineer							
$154,654		Mine Shift Foreman	Maintenance Planner	Laboratory Supervisor	Plant Shift Foreman	Welder/Pipefitter		
$136,086			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		
$137,371	Mine Surveyor	Drill Operator						Safety Officer
		Shovel Operator						Paramedics
$140,205	Exploration Geologist				Plant Lead Operator			Contracts Officer
								Accountant
					Gold Room Supervisor			IT Supervisor
$114,655		Aux Equipment Operator						Env Officer - Monitoring
								Env. Officer - Compliance
$119,759	Mine Clerk	Haul Truck Operator			Crushing/ Sorting Operator	Crane Operator		Recruiting Officer
					Grinding/Leach Operator			Head of Security
$116,903					Refiner			Purchasing Officer
								Business Dev. Officer

Salary (AUD)	Mine Technical	Mine Operations	Mine Maintenance	Plant Technical	Plant Operations	Plant Maintenance	NPI	Administration
								Community Liaison Officer
								Indigenous Liaison Officer
$97,510	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			
$92,153		Blasting Assistant	Maintenance Shop Labor			Trades Assistant		Store Person
								Administrative Secretary
$95,103	Surveyor Helper		Fuel & Lube Technician					Administrative Assistant
	Geology Field Technician							Security Officer
								Bus Driver
$86,403				Sample Prep Technician				Accounting Clerk
								Payroll Clerk
$75,034								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	2800	g/t leach feed	$489	per tonne
Sodium Cyanide	876	g/t leach feed	$3,400	per tonne
Sodium Hydroxide	40	g/t ore	$1,519	per tonne
Flocculant	40	g/t leach feed	$3,770	per tonne
Sodium Metabisulphite (SMBS)	732	g/t leach feed	$617	per tonne
Hydrochloric Acid	81	g/t ore	$902	per tonne
Lead Nitrate	100	g/t ore	$3,489	per tonne
Activated Carbon	20	g/t ore	$3,153	per tonne
Borax	150	kg/t conc.	$2,641	per tonne
Silica	150	kg/t conc.	$1,575	per tonne
Soda Ash	100	kg/t conc.	$2,641	per tonne
Potassium Nitrate	30	kg/t conc.	$7,331	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	$364,199	per mantle
Primary Crusher concaves	272	days per set	$323,813	per set
Secondary Crushers Main frame Liners	481	days per unit	$72,532	per unit
Secondary Crushers Bowl Liners	61	days per unit	$276,755	per unit
Secondary Crusher Mantle	61	days per unit	$201,814	per unit
MILLING				
Mill Balls 65mm	0.06	kg/kWh	$1,603	per tonne
Mill Liners	1.0	sets per annum/mill	$2,309,187	per set
Secondary Grinding Media	0.35	kg/kWh	$4.72	per kg
HPGR				
Cheek plates	8,000	h/set	$62,425	per set
Tires	13,000	h/set	$2,281,366	per set

Consumables		Consumable Rate	Unit	Unit Cost (AUD)	Unit
LIME SLAKER					
Mill Balls	50 mm Lime Slaking Mill	0.5	kg/t lime	$1,603	per tonne

22.4.8.4 Diesel Consumption

The primary consumer of diesel is mining, which totals 787 million liters of diesel. The total project consumption of diesel is 696 million liters.

22.4.8.5 Plant Power Consumption

The primary consumer of power is the process facility, which totals 11,241,913 MWh power. The total project consumption of power is 11,301,903 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$1,800/oz and a US$0.69:AUD1.00 exchange rate:

- Mine Life .... 16 years;
- Pre-Tax NPV5% .... US$2,149.4 million, IRR: 29.4%;
- After-tax NPV5% .... US$1,131.4 million, IRR: 20.4%;
- Payback (After-tax) .... 4.0 years;
- NT Royalty Paid .... US$765 million;
- Australian Income Taxes Paid .... US$927 million; and
- Cash costs (including Royalties) .... US$913/oz-Au.

Table 22-26: Technical-Economic Results (US$000s)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
GOLD SALES			
Gold Produced (koz)	6,313	-	-
Gold Price (US$/oz)	1,800	-	-
Gold Sales	**11,364,288**	**40.53**	**1,800.00**
REFINING & ROYALTIES			
Refinery Costs	(23,206)	(0.08)	(3.68)
JAAC and Wheaton Royalties	(324,038)	(1.16)	(51.32)
Gross Income from Mining	**11,017,044**	**39.29**	**1,745.00**
OPERATING COSTS			
Open Pit Mine	(2,153,191)	(7.68)	(341.05)
CIP Process Plant	(2,889,166)	(10.30)	(457.62)
Project Services	(84,130)	(0.30)	(13.33)
G&A	(293,212)	(1.05)	(46.44)

Cash Flow Summary	LoM (US$000s)	Unit Cost US$/t-milled	US$/oz-Au
Operating Costs	**(5,419,700)**	**(19.33)**	**(858.43)**
Cash Cost of Goods Sold (COGS)	**(5,442,905)**	**(19.41)**	**(862.11)**
Operating Margin	**5,597,345**	**19.96**	**886.57**
CAPITAL COSTS			
Mining	719,974		
Process Plant	659,523		
Project Services	157,069		
Project Infrastructure	62,867		
Permanent Accommodation	464		
Site Establishment & Early Works	29,886		
Management, Engineering, EPCM Services	124,569		
Pre-Production Costs	34,169		
Asset Sale	(42,756)		
CAPITAL COSTS	**1,745,766**		
Pre-Tax Cash Flow	3,851,579		
$NPV_{5\%}$	2,149,401		
IRR (%)	29.4%		
After-tax Cash Flow	2,160,177		
$NPV_{5\%}$	1,131,432		
IRR (%)	20.4%		
After-tax Payback (years)	4.0		

Cash costs for the Project are presented in Table 22-27.

Table 22-27: Cash Costs and All-In Sustaining Costs (US$/oz)

Period	Cash Cost	Sustaining	AISC
First 7 years of Production	USD 845	USD 116	USD 961
LoM	USD 913	USD 120	USD 1,034

Cash costs as defined in guidance from the World Gold Council include non-cash remuneration for site personnel and AISC include corporate or regional general and administrative costs, including share-based remuneration. Project cashflows, cash costs/oz and AISC/oz are presented on a site-level basis and, therefore, do not include these elements.

If determined on a company-level basis, Vista estimates non-cash remuneration (inclusive of share-based compensation for site personnel would increase cash costs/oz by approximately $3/oz. AISC would increase by this $3/oz and an estimated additional $4/oz for corporate general and administrative costs.

Table 22-28: Annual Cash Flow

Cash Flow Summary	Units	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Payable Gold	Kozs	6,313	-	-	399	458	542	341	408	539	504	352	261	277	311	370	391	406	403	310	41	-	-	-	-
Gold Price	US$	$ 1,800	$ -	$ -	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ 1,800	$ -	$ -	$ -	$ -
Gold Sales	US$M	$ 11,364	$ -	$ -	$ 718	$ 824	$ 976	$ 614	$ 734	$ 970	$ 908	$ 634	$ 470	$ 499	$ 560	$ 665	$ 705	$ 730	$ 726	$ 557	$ 73	$ -	$ -	$ -	$ -
Operating Costs																									
Mining	US$M	$ (2,153)	$ -	$ -	$ (79)	$ (113)	$ (146)	$ (179)	$ (192)	$ (213)	$ (207)	$ (194)	$ (186)	$ (179)	$ (172)	$ (133)	$ (62)	$ (27)	$ (36)	$ (32)	$ (3)	$ (0)	$ (0)	$ (0)	$ -
Processing	US$M	$ (2,889)	$ -	$ -	$ (134)	$ (182)	$ (182)	$ (182)	$ (182)	$ (183)	$ (182)	$ (182)	$ (182)	$ (182)	$ (182)	$ (182)	$ (182)	$ (182)	$ (183)	$ (176)	$ (28)	$ -	$ -	$ -	$ -
G&A	US$M	$ (377)	$ -	$ -	$ (26)	$ (26)	$ (26)	$ (24)	$ (24)	$ (24)	$ (24)	$ (24)	$ (24)	$ (24)	$ (24)	$ (24)	$ (20)	$ (16)	$ (17)	$ (16)	$ (4)	$ (2)	$ (2)	$ (2)	$ (1)
Royalties	US$M	$ (324)	$ -	$ -	$ (22)	$ (25)	$ (29)	$ (18)	$ (22)	$ (29)	$ (27)	$ (19)	$ (13)	$ (13)	$ (15)	$ (18)	$ (19)	$ (19)	$ (19)	$ (15)	$ (2)	$ -	$ -	$ -	$ -
Refining	US$M	$ (23)	$ -	$ -	$ (1)	$ (2)	$ (2)	$ (1)	$ (1)	$ (2)	$ (2)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (0)	$ -	$ -	$ -	$ -
Sub-total: Operating Costs	**US$M**	**$ (5,767)**	**$ -**	**$ -**	**$ (262)**	**$ (348)**	**$ (385)**	**$ (405)**	**$ (421)**	**$ (451)**	**$ (442)**	**$ (420)**	**$ (406)**	**$ (400)**	**$ (394)**	**$ (359)**	**$ (285)**	**$ (247)**	**$ (256)**	**$ (240)**	**$ (38)**	**$ (2)**	**$ (2)**	**$ (2)**	**$ (1)**
Cash Operating Margin	US$M	$ 5,597	$ -	$ -	$ 457	$ 476	$ 591	$ 209	$ 313	$ 519	$ 466	$ 214	$ 65	$ 100	$ 165	$ 306	$ 420	$ 483	$ 470	$ 317	$ 35	$ (2)	$ (2)	$ (2)	$ (1)
Capital Costs																									
Initial Capex	US$M	$ (1,030)	$ (344)	$ (686)																					
Sustaining Capex	US$M	$ (508)	$ -	$ -	$ (105)	$ (133)	$ (65)	$ (76)	$ (34)	$ (17)	$ (7)	$ (9)	$ (8)	$ (12)	$ (10)	$ (8)	$ (7)	$ (6)	$ (7)	$ (4)	$ (0)	$ -	$ -	$ -	$ -
Closure	US$M	$ (251)	$ -	$ -	$ (5)	$ (1)	$ (1)	$ (4)	$ (5)	$ (4)	$ (4)	$ (3)	$ (3)	$ (4)	$ (2)	$ (1)	$ (21)	$ (40)	$ (17)	$ (1)	$ (2)	$ (15)	$ (72)	$ (43)	$ -
Salvage	US$M	$ 43	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8	$ -	$ -	$ -	$ -	$ 17	$ 18	$ -	$ -	$ -	$ -
Sub-total: Capital Costs	**US$M**	**$ (1,746)**	**$ (344)**	**$ (686)**	**$ (110)**	**$ (134)**	**$ (66)**	**$ (80)**	**$ (39)**	**$ (21)**	**$ (11)**	**$ (12)**	**$ (12)**	**$ (16)**	**$ (5)**	**$ (9)**	**$ (29)**	**$ (46)**	**$ (23)**	**$ 11**	**$ 15**	**$ (15)**	**$ (72)**	**$ (43)**	**$ -**
Working Capital Changes	US$M	$ 0	$ 15	$ 0	$ (8)	$ 3	$ (2)	$ 5	$ (2)	$ (0)	$ 1	$ 2	$ (0)	$ (1)	$ (0)	$ (4)	$ (0)	$ (2)	$ (0)	$ 0	$ (6)	$ 4	$ 4	$ (7)	$ (1)
Pre-Tax Cash Flow	US$M	$ 3,852	$ (328)	$ (686)	$ 339	$ 345	$ 523	$ 134	$ 272	$ 498	$ 456	$ 204	$ 53	$ 84	$ 160	$ 293	$ 391	$ 436	$ 446	$ 329	$ 45	$ (14)	$ (71)	$ (52)	$ (2)
Northern Territory Royalty	US$M	$ (765)	$ -	$ -	$ (20)	$ (30)	$ (40)	$ (22)	$ (25)	$ (77)	$ (70)	$ (22)	$ (18)	$ (19)	$ (20)	$ (68)	$ (87)	$ (97)	$ (99)	$ (74)	$ (8)	$ -	$ -	$ 33	$ -
Income Taxes	US$M	$ (927)	$ -	$ -	$ (38)	$ (67)	$ (101)	$ (9)	$ (46)	$ (98)	$ (90)	$ (33)	$ -	$ -	$ (25)	$ (57)	$ (87)	$ (104)	$ (101)	$ (71)	$ -	$ -	$ -	$ -	$ -
After-Tax Cash Flow	US$M	$ 2,160	$ (328)	$ (686)	$ 280	$ 248	$ 382	$ 103	$ 201	$ 323	$ 295	$ 148	$ 35	$ 65	$ 114	$ 169	$ 217	$ 235	$ 247	$ 184	$ 36	$ (14)	$ (71)	$ (20)	$ (2)
After-Tax Cumulative Cash Flow	US$M		$ (328)	$ (1,014)	$ (734)	$ (486)	$ (104)	$ (1)	$ 200	$ 523	$ 818	$ 966	$ 1,001	$ 1,066	$ 1,180	$ 1,349	$ 1,565	$ 1,800	$ 2,047	$ 2,231	$ 2,267	$ 2,253	$ 2,182	$ 2,162	$ 2,160
Pre-Tax NPV5%	US$M	$ 2,149.4																							
Pre-Tax IRR	%	29.4%																							
After-Tax NPV5%	US$M	$ 1,131.4																							
After-Tax IRR	%	20.4%																							

Production Summary	Units	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Ore	Ktonnes	267,021	-	119	19,087	18,287	29,854	13,982	23,132	26,777	19,364	10,504	10,823	14,194	17,676	19,677	14,915	12,568	11,489	4,574	-	-	-	-	-
Waste	Ktonnes	671,331	-	2,876	21,087	39,130	43,054	73,310	62,720	72,908	75,901	74,284	65,388	54,791	45,286	27,087	9,036	3,127	1,139	206	-	-	-	-	-
Total Material Mined	Ktonnes	938,352	-	2,995	40,173	57,417	72,909	87,292	85,852	99,685	95,265	84,788	76,210	68,985	62,962	46,764	23,951	15,695	12,628	4,780	-	-	-	-	-
Stripping Ratio (W:O)		2.51	-	24.25	1.10	2.14	1.44	5.24	2.71	2.72	3.92	7.07	6.04	3.86	2.56	1.38	0.61	0.25	0.10	0.05	-	-	-	-	-
Plant Feed																									
Mined Ore (RoM and Stockpiled)	Ktonnes	267,021	-	-	12,334	17,750	17,750	17,799	17,750	17,823	17,750	17,774	17,774	17,750	17,750	17,774	17,774	17,750	16,296	7,421	-	-	-	-	-
Heap Leach Material (HLM)	Ktonnes	13,354	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,454	9,289	2,612	-	-	-	-
Sub-total: Plant Feed	**Ktonnes**	**280,375**	**-**	**-**	**12,334**	**17,750**	**17,750**	**17,799**	**17,750**	**17,823**	**17,750**	**17,774**	**17,774**	**17,750**	**17,750**	**17,774**	**17,774**	**17,750**	**17,750**	**16,710**	**2,612**	**-**	**-**	**-**	**-**
Grade	g Au/tonne	0.77	-	-	1.10	0.88	1.04	0.66	0.79	1.03	0.97	0.69	0.52	0.55	0.61	0.72	0.76	0.79	0.78	0.64	0.54	-	-	-	-
Gold to Plant	Kozs	6,979	-	-	436	503	594	378	451	591	554	392	295	312	347	410	433	448	446	344	45	-	-	-	-
CIP Plant Feed (Post-Sorting)	Ktonnes	253,673	-	-	11,100	15,975	15,975	16,019	15,975	16,041	15,975	15,997	15,997	15,975	15,975	15,997	15,997	15,975	16,120	15,968	2,612	-	-	-	-
Grade	g Au/tonne	0.84	-	-	1.21	0.97	1.14	0.73	0.87	1.13	1.06	0.75	0.57	0.60	0.67	0.79	0.83	0.86	0.85	0.66	0.54	-	-	-	-
Gold to CIL Plant	Kozs	6,891	-	-	431	497	587	373	445	583	546	386	291	308	343	404	428	442	440	341	45	-	-	-	-
Recovery	%	91.6%	0.0%	0.0%	92.6%	92.1%	92.5%	91.3%	91.7%	92.4%	92.3%	91.2%	89.8%	90.1%	90.7%	91.4%	91.6%	91.7%	91.6%	90.7%	89.8%	0.0%	0.0%	0.0%	0.0%
Payable Gold	Kozs	6,313	-	-	399	458	542	341	408	539	504	352	261	277	311	370	391	406	403	310	41	-	-	-	-

Unit Cost Metrics	Units	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
Operating and Cash Costs																									
Mining	$/T Mined	$ (2.29)	$ -	$ -	$ (1.96)	$ (1.97)	$ (2.00)	$ (2.05)	$ (2.24)	$ (2.14)	$ (2.17)	$ (2.29)	$ (2.44)	$ (2.59)	$ (2.73)	$ (2.85)	$ (2.57)	$ (1.74)	$ (2.85)	$ (6.66)	$ -	$ -	$ -	$ -	$ -
Mining	$/T Processed	$ (7.68)	$ -	$ -	$ (6.40)	$ (6.38)	$ (8.20)	$ (10.07)	$ (10.82)	$ (11.97)	$ (11.66)	$ (10.91)	$ (10.46)	$ (10.08)	$ (9.68)	$ (7.49)	$ (3.47)	$ (1.54)	$ (2.03)	$ (1.91)	$ (1.31)	$ -	$ -	$ -	$ -
Processing	$/T Processed	$ (10.30)	$ -	$ -	$ (10.84)	$ (10.26)	$ (10.26)	$ (10.25)	$ (10.26)	$ (10.26)	$ (10.26)	$ (10.25)	$ (10.24)	$ (10.25)	$ (10.25)	$ (10.25)	$ (10.25)	$ (10.26)	$ (10.28)	$ (10.55)	$ (10.82)	$ -	$ -	$ -	$ -
G&A	$/T Processed	$ (1.35)	$ -	$ -	$ (2.12)	$ (1.49)	$ (1.49)	$ (1.33)	$ (1.33)	$ (1.32)	$ (1.34)	$ (1.36)	$ (1.35)	$ (1.37)	$ (1.38)	$ (1.37)	$ (1.15)	$ (0.91)	$ (0.96)	$ (0.95)	$ (1.67)	$ -	$ -	$ -	$ -
Sub-total: Operating Costs	**$/T Processed**	**$ (19.33)**	**$ -**	**$ -**	**$ (19.36)**	**$ (18.13)**	**$ (19.95)**	**$ (21.65)**	**$ (22.41)**	**$ (23.55)**	**$ (23.27)**	**$ (22.52)**	**$ (22.06)**	**$ (21.70)**	**$ (21.31)**	**$ (19.11)**	**$ (14.87)**	**$ (12.71)**	**$ (13.26)**	**$ (13.41)**	**$ (13.80)**	**$ -**	**$ -**	**$ -**	**$ -**
Royalties	$/T Processed	$ (1.16)	$ -	$ -	$ (1.75)	$ (1.39)	$ (1.65)	$ (1.03)	$ (1.24)	$ (1.63)	$ (1.53)	$ (1.05)	$ (0.71)	$ (0.75)	$ (0.84)	$ (1.00)	$ (1.06)	$ (1.10)	$ (1.09)	$ (0.89)	$ (0.75)	$ -	$ -	$ -	$ -
Refining	$/T Processed	$ (0.08)	$ -	$ -	$ (0.12)	$ (0.09)	$ (0.11)	$ (0.07)	$ (0.08)	$ (0.11)	$ (0.10)	$ (0.07)	$ (0.06)	$ (0.06)	$ (0.07)	$ (0.08)	$ (0.08)	$ (0.08)	$ (0.08)	$ (0.07)	$ (0.06)	$ -	$ -	$ -	$ -
Total: Cash Costs	**$/T Processed**	**$ (20.57)**	**$ -**	**$ -**	**$ (21.22)**	**$ (19.62)**	**$ (21.71)**	**$ (22.75)**	**$ (23.73)**	**$ (25.29)**	**$ (24.90)**	**$ (23.64)**	**$ (22.82)**	**$ (22.51)**	**$ (22.22)**	**$ (20.19)**	**$ (16.01)**	**$ (13.89)**	**$ (14.44)**	**$ (14.37)**	**$ (14.61)**	**$ -**	**$ -**	**$ -**	**$ -**
Full Production Years 1 - 7																									
Cash Costs	$/Oz	$ 845																							
AISC	$/Oz	$ 961																							
Life of Mine																									
Cash Costs	$/Oz	$ 913			$ 656	$ 761	$ 711	$ 1,187	$ 1,033	$ 836	$ 876	$ 1,192	$ 1,553	$ 1,440	$ 1,268	$ 971	$ 727	$ 608	$ 635	$ 775	$ 937				
AISC	$/Oz	$ 1,034			$ 931	$ 1,054	$ 833	$ 1,422	$ 1,129	$ 875	$ 898	$ 1,227	$ 1,597	$ 1,496	$ 1,309	$ 995	$ 801	$ 722	$ 693	$ 792	$ 1,007				

22.5.1 Taxes, Royalties

Taxes, royalties, and working capital were incorporated into the economic model by Vista.

22.5.1.1 Royalties

NORTHERN TERRITORY ROYALTY

Under the NT Mineral Royalty Act 1982 (as in force at May 21, 2021) (the "MRA), the holders of mining tenements that form part of a production unit are liable for the payment of royalty in respect of the production unit.

The royalty payable under the MRA is the greater of:

1) 20 percent of the net value from a production unit in a royalty year, less $10 000, and
2) the percentage of the gross production revenue, from the production unit in a royalty year, that applies to the royalty year as follows:
    a) 1% for the royalty payer's first royalty year that begins on or after July 1, 2019;
    b) 2% for the royalty year that follows the royalty year mentioned in subparagraph (a); and
    c) 2.5% for each royalty year that follows the royalty year mentioned in subparagraph (b).

The royalty payable under in a royalty year is nil if the gross production revenue from the production unit in the royalty year is AUD500,000 or less.

The MRA imposes a net value-based royalty, subject to an annual minimum royalty, on mine production (the "NT Royalty). The MRA codifies the basis for calculating the NT Royalty and grants a designated Secretary the authority to approve certain matters either specifically or by the promulgation of guidelines set out by the MRA. Such determinations by the Secretary or other project development concessions that may be granted by the Northern Territory Government may not fully be a matter of public record. Such agreements are understood to be generally confidential in nature and each is subject to a formal application process. The NT Royalty calculated for the Project cashflows is based on the MRA, together with Vista's assessment of approvals expected from the Secretary, where applicable, and estimates regarding the nature and amount of relief that appears to be available from the Northern Territory to new mines.

For calculating the rate of royalty under the MRA, the net value from a production unit in a royalty year is calculated in accordance with the following formula:

$$NV = GR - (OC + CRD + EEE + AD)$$

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.

The NT established the Minerals Development Taskforce (the "Taskforce") in November 2021 to investigate and identify opportunities to accelerate external investment in mining and downstream value-add projects in the NT. The Taskforce, as part of its findings, has recommended fundamental reform of the current NT royalty scheme by implementing an ad valorem scheme with rates that will be considered more competitive with other mining jurisdictions. Any changes require legislative approval, which has not been announced at this time. Should the current NT Royalty scheme be replaced with an ad valorem scheme, the estimated NT royalty payments to be paid by Mt Todd would be as follows:

Table 22-29: Estimated NT royalty payments to be paid, ad valorem scheme

Ad Valorem Rate	LoM (US$000s)
2.5%	$284,107
4.5%	$511,393
6.5%	$738,679

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. In addition to the aforementioned 1% royalty to the JAAC, Vista and the JAAC agreed to replace the 10% participating interest right previously granted to the JAAC with a sliding scale gross proceeds production royalty that varies between 0.125% and 2.000%, depending on the gold price and foreign exchange rate during each applicable production period.

Wheaton is entitled to receive 1%, subject to adjustment, of the gross revenue from Mt Todd.

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 the presently identified mineral resources or mineral reserves 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, capital deductions (i.e., depreciation), dividends, interest, royalties and rent. Assessable income may also include capital gains after offsetting capital losses. Normal business expenses are generally deductible.

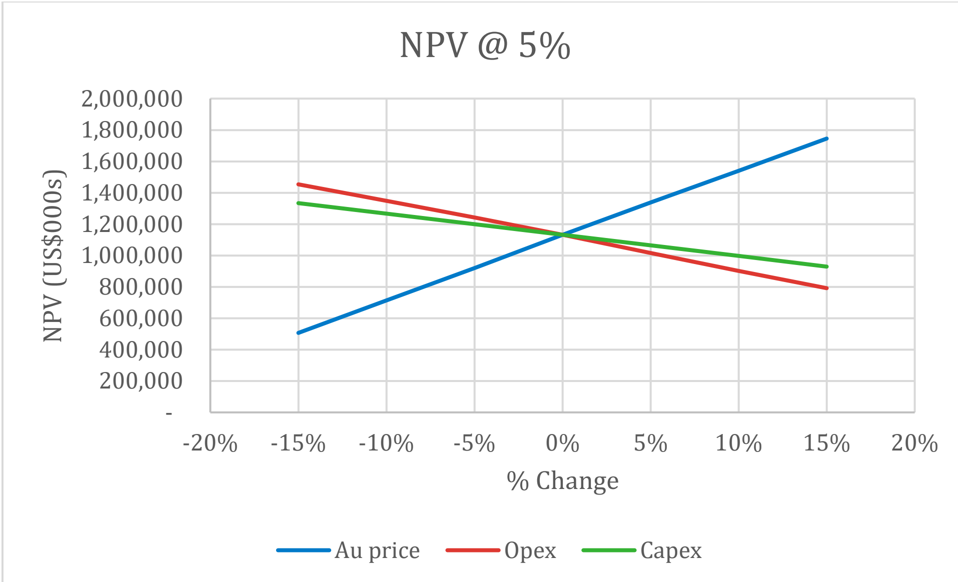
Tax losses may be carried forward indefinitely and utilized to offset future assessable income, providing a "continuity of ownership" (more than 50% of voting, dividend and capital rights) or a "same business" test is satisfied.

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 22-30: Project Sensitivity (NPV@5% discount rate, US$000s)

	NPV @ 5% US$000s						
Parameter	**85%**	**90%**	**95%**	**Base**	**105%**	**110%**	**115%**
Gold Price	506,657	713,854	919,230	1,131,432	1,337,580	1,540,376	1,745,224
Opex	1,453,576	1,347,794	1,241,627	1,131,432	1,015,992	902,553	791,712
Capex	1,333,738	1,266,303	1,198,867	1,131,432	1,063,997	996,562	929,127



Source: Tetra Tech, March 8, 2024

Figure 22-1: Project NPV (at 5% discount rate) Sensitivity

Table 22-31: Sensitivities of NPV (US$ M) to Gold Price versus NPV Discount Rate

Discount Rate (%)	GOLD PRICE (US$/oz-Au)								
	1,400	1,500	1,600	1,700	1,800	1,900	2,000	2,100	2,200
5	208	436	669	896	**1,131**	1,347	1,571	1,784	2,008
8	9	193	380	562	751	923	1,103	1,272	1,452
10	(90)	72	235	395	559	709	865	1,013	1,169

Table 22-32: Sensitivities of NPV @5% Discount Rate (US$ M) and IRR to Gold Price versus Foreign Exchange Rate (US$:AUD)

Foreign Exchange	GOLD PRICE (US$/oz-Au)																	
	1,400		1,500		1,600		1,700		1,800		1,900		2,000		2,100		2,200	
(US$/AUD)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)	IRR (%)	NPV(5)
0.63	11.4	407	14.7	632	17.8	858	20.9	1,095	23.7	1,322	26.4	1,534	29.0	1,761	31.5	1,971	34.0	2,194
0.66	9.8	309	13.0	535	16.1	763	19.1	994	22.0	1,228	24.6	1,440	27.2	1,667	29.6	1,878	32.1	2,101
0.69	8.1	208	11.4	436	14.6	669	17.5	896	**20.4**	**1,131**	22.9	1,347	25.5	1,571	27.9	1,784	30.3	2,008
0.72	6.6	105	9.9	337	13.1	572	15.9	800	18.8	1,031	21.3	1,253	23.9	1,478	26.2	1,688	28.6	1,914
0.75	5.0	2	8.4	238	11.6	474	14.5	707	17.2	934	19.8	1,155	22.3	1,385	24.6	1,594	27.0	1,820

23. ADJACENT PROPERTIES

There are no adjacent properties that are considered relevant to this Technical Report.

24. OTHER RELEVANT DATA AND INFORMATION

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:

1) Comprehensive Geotechnical Investigation for Mt Todd FS was prepared by Douglas Partners, Revision 1 issued on the 23 September 2021 (document 92101.00.R.001.Rev1.docx). This investigation may be supplemented by the previous geotechnical investigations and reports comprising:
2) Geotechnical Desktop Study Mt Todd Process Plant DFS undertaken by Coffey Geotechnics in December 2012.
3) 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.
4) Soil and Rock Engineering (SRE) geotechnical data from December 1992 and April 1993 for the original Mt Todd development.

Further geotechnical investigation is recommended during the detail design phase (execution phase) if the FS locations of heavy vibrating equipment comprising the primary and secondary crushers, screens, HPGRs and mills are modified. This additional investigation is not considered significant as it would require a limited number of test pits & boreholes for validation that the modified (new) locations of heavy vibration equipment are adequate.

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 12.1.1) to simulate 1 year of preproduction, 17 years of mine production (15 active mining years and 2 additional year processing stockpiles), and 5 years of closure at the Vista Project Site.

The SWWB was developed to simulate site conditions in order to:

- Validate adequacy of water treatment plant capacity;
- Validate adequacy of process water (PWP) pond sizing; and
- Quantify make up water requirements from the RWD and WTP for process make up water, dust control, and potable/elution needs.

24.2.1.1 Site-wide Water Balance Model

WATER BALANCE MODELING

The SWWB 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:

$$\text{Change in Storage} = \text{Inputs} - \text{Outputs}$$

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 (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 (TSF1); and
- Tailings Storage Facility 2 (TSF2).

GENERAL ASSUMPTIONS

Interaction between site features was modeled based on the following set of guidelines:

- RP1, LGRP, RP3, PRP HLP, and a 250 m^3/hr TSF bleed stream (dry season) report to the PWP which feeds the WTP during production;
- TSF1 is dewatered by pumping to the PWP during preproduction to allow for construction of embankment raises. TSF1 is dewatered for closure by pumping to the PWP during year 15 of production and then to the Batman Pit after mining is completed;
- 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 except for the PRP which is a sediment pond and allowed to overtop. The PWP was designed for a capacity of 185,000 m^3;
- A dry season TSF decant bleed stream of 250 m^3/hr is sent to the PWP to maintain proper chemistry of the process circuit;
- Process makeup water is prioritized to come from WTP effluent first and then the RWD to reduce discharges to the Edith River;
- Discharges to the Edith River are not allowed from any of the RPs except for the PRP which is a sediment pond that is allowed to overtop in accordance with the Waste Discharge Licence;
- WTP effluent is allowed to discharge to the Edith River at a dilution ratio of 19:1 (Edith River to WTP Discharge);
- The HLP and LGOS are run through process at the end of the Life of Mine (LoM);
- TSF2 is dewatered for closure by pumping to the Batman Pit after mining is complete;
- Seepage losses from the RP ponds are not modeled and are assumed to be zero; and
- TSF1 loses 20% of its floor seepage to the environment.

INITIAL CONDITIONS

- TSF1, LGRP, and the HLP were assigned initial water surface elevations based on the average water elevation for September 1st from 2015 to 2020;
- The PRP is assumed to be empty at the beginning of production; and
- RP3 is assumed to be at 50m AHD at year -1.

FLOW RATES

These rates represent the mean flows throughout the simulation unless stated otherwise:

- Process makeup water requirements throughout the LoM were 2,270 m3/hr. This value does not account for reagent or gland water;
- Gland water requirements were 190 m^3/hr and supplied by the RWD;
- Reagent water requirements were 87 m^3/hr and supplied by the WTP and RWD;
- TSF decant return flows were between 1,582 m^3/hr and 2,270 m^3/hr depending on the time of year and available water;
- A TSF decant bleed stream of 250 m^3/hr during the dry season is required to maintain proper water chemistry in the process/tailings circuit;
- RWD process makeup water flows ranged from 190 m^3/hr to 617 m^3/hr;
- WTP process makeup water flows ranged from 28 to 600 m^3/hr;
- Dust suppression requirements varied between 220 and 1,153 m^3/day; and
- WTP capacity is modeled as 600 m^3/hr.

CLIMATOLOGICAL INPUTS

The Vista Project SWWB 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:

- A 1000-year Synthetic precipitation dataset was developed using the Stochastic Climate Library (SCL) software. Inputs used to develop this synthetic precipitation dataset included site precipitation data for four rain gauges onsite, three gauges near the town of Katherine, and gridded SILO rainfall data for the Site. At that beginning of each modeled year (September 1), GoldSim randomly selected 1 full year of data from the 1000-year dataset to build a unique synthetic precipitation dataset for each of the 1000 model realizations. 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	317
February	243
March	164
April	50
May	8
June	0
July	0
August	0
September	3

Month	Precipitation (mm)
October	28
November	112
December	281

- Linking incidental rainfall and runoff within the Edith River and Horseshoe Creek using the Australian Water Balance Model (AWBM). Catchment parameters were calibrated to Edith River from 2010 to 2020 using an initial auto calibration process undertaken using the eWater Source software followed up by manual adjustments to optimize the calibration. These parameters were then input into the AWBM module within GoldSim to estimate flows in the Edith River and Horseshoe Creek which feeds into the RWD.
- The SWWB model used SILO average daily evaporation values based on which month the model was in. A 0.7 pan factor was used to convert from pan to lake evaporation.

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 1 year of preproduction, the 17-year LoM, and 5 years of closure were 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

Under the modeled conditions described previously the SWWB model results indicate that:

- The Batman Pit will see minor water storage during the wet season and later in the LoM. This is because of increased groundwater inflows and a large catchment area near the end of the LoM.
- The mean Process Plant Makeup water (including gland water) required from the RWD varied from 4,560 m^3/day to 14,340 m^3/day with the later occurring late in the dry season and early in the LoM. RWD requirements were found to be the most dependent upon the amount of WTP effluent available to provide makeup water to the Process Plant and on TSF decant volumes.
- RP1, LGRP, and the HLP show less than a 5% percent probability of having an overtopping event over the LoM[3].

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 600 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 178-08 (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.

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

24.2.2.2 Water Quality Standards for Waste Water Discharge

Discharges from the site are currently regulated by Waste Discharge Licence 178-08 (WDL), issued by the Northern Territory Government on November 30, 2020. 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 Mining Management Plan Section 6.14 (Vista Gold Australia, 2021) 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 Guidelines (ANZG) for Fresh & Marine Water Quality (ANZG 2018 Guidelines). This change will be reflected in a revision to WDL 178.

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 measured in the Edith River shall not exceed the TV during discharge events. For the constituents 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	SU	6-8	ANZG 2018 Guidelines
Dissolved Oxygen	% Saturation	85-120	ANZG 2018 Guidelines
Conductivity	µS/cm	20-250	ANZG 2018 Guidelines
Magnesium	mg/L	2.5	Van Dam, et al. 2010 Environ Toxicol Chem 29(2):410-421
Sulfate	mg/L	129	Elphick et al. 2011 Environ Toxicol Chem 30(1):247-253
Aluminum	µg/L	55	ANZG 2018 Guidelines
Cadmium	µg/L	0.2	ANZG 2018 Guidelines
Cobalt	µg/L	13	Canadian guideline adopted by ANZG 2018 Guidelines
Chromium (III)	µg/L	3.3	ANZG 2018 Guidelines
Chromium (VI)	µg/L	1.0	ANZG 2018 Guidelines
Copper	µg/L	1.4	ANZG 2018 Guidelines
Manganese	µg/L	1900	ANZG 2018 Guidelines
Nickel	µg/L	11	ANZG 2018 Guidelines
Lead	µg/L	3.4	ANZG 2018 Guidelines
Iron	µg/L	300	ANZG 2018 Guidelines
Mercury	µg/L	0.6	ANZG 2018 Guidelines
Zinc	µg/L	8.0	ANZG 2018 Guidelines

The TVs for magnesium and sulfate have been held over from previous work, and are not referenced in the ANZG 2018 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. First, an analysis was performed to evaluate the minimum dilution ratio required to meet the TV for sulfate at SW4 on the Edith River. Then upstream water quality values at sampling location SW2 on the Edith River and the minimum dilution ratio for sampling location SW4 downstream of the WTP discharge were used to calculate effluent limits at the WTP that maintain the site-specific trigger value at site SW4.

The equation used to determine the effluent limits is:

$$Q_{WTP}C_{WTP} + Q_{SW2}C_{SW2} = Q_{SW4}C_{SW4}$$

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

A minimum dilution ratio of 1:19 WTP flow rate to Edith River flow rate (Q_{WTP}:Q_{SW4}) is required to consistently achieve the sulfate TV at SW4. At a maximum WTP flow rate of 600 m^3/hr, the Edith River must have a minimum flow rate of 11,400 m^3/hr to discharge from the WTP to the Edith River. These conditions are typically met December through March and discharges to the environment will primarily occur between December and March. Table 24-3 presents Edith River flows at field monitoring location SW4.

Table 24-3: Edith River Flow at SW4 (m^3/h), February 2013 – June 2021

Month	Mean	Median	5th Percentile	95th Percentile	Maximum Day	Minimum Day
January	136,048	99,924	32,556	352,965	2,209,457	0
February	148,972	77,544	36,016	431,235	1,430,188	608
March	57,083	39,589	15,260	155,611	524,238	437
April	9,378	5,657	2,462	25,008	6,7106	0
May	2,602	1,516	0	8,666	16,801	0
June	706	87	0	3,388	6,557	0
July	577	0	0	2,893	4,720	0
August	2,365	0	0	12,280	110,348	0
September	1,690	0	0	8,779	79,007	0
October	2,176	0	0	11,313	101,972	0
November	5,733	3,050	119	17,102	180,450	0
December	24,824	9,352	1,006	93,294	764,616	0

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 one half 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 – March 2020

Analyte	Unit	No. of Samples	Minimum Value	Maximum Value	5th Percentile Value	95th Percentile Value	Average Value
Magnesium	mg/L	92	0.5	1	<0.5	1	0.68
Sulfate	mg/L	252	<1	19	<1	1	0.65
Aluminum	µg/L	252	30	3300	50	764	275
Cadmium	µg/L	251	<0.1	0.1	<0.1	<0.1	<0.1
Cobalt	µg/L	114	<1	1	<1	<1	<1

Analyte	Unit	No. of Samples	Minimum Value	Maximum Value	5th Percentile Value	95th Percentile Value	Average Value
Chromium	µg/L	105	<1	2	<1	<1	<1
Copper	µg/L	245	0.23	20	<1	2	0.88
Manganese	µg/L	105	7	51	8	24.8	14.6
Nickel	µg/L	105	0.5	2	0.5	0.9	0.56
Lead	µg/L	88	<1	<1	<1	<1	<1
Iron	µg/L	251	400	4440	450	1355	829
Mercury	µg/L	105	0.05	<0.05	<0.05	<0.05	<0.05
Zinc	µg/L	105	1	16	1	7.95	3.48

Using the TVs presented in Table 24-2, the minimum dilution ratio, and the background water quality in Table 24-4, 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	31	25
Sulfate	mg/L	1	129	2,561	N/A
Aluminum	µg/L	764	55	55	44
Cadmium	µg/L	0.1	0.2	2.1	1.7
Cobalt	µg/L	1	13	241	193
Chromium	µg/L	1	1.0	1.0	0.8
Copper	µg/L	2	1.4	1.4	1.1
Manganese	mg/L	0.025	1.9	37.5	30.0
Nickel	µg/L	0.9	11	203	162
Lead	µg/L	1	3.4	49	39
Iron	mg/L	1.4	0.3	0.3	0.24
Mercury	µg/L	0.05	0.6	11	8.8
Zinc	µg/L	7.95	8	8.9	8.0

The background water quality concentration at SW2 for aluminum, chromium, copper, and iron 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 operating years of the mine. The geochemistry model includes inputs from various sources on the mine site, and considers potential chemical reactions between the various inputs prior to entering the WTP. At the WTP, Vista is 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	212	25	88.3%
Sulfate	mg/L	2186	N/A [1/]	-
Aluminum	µg/L	33,747	55	99.9%
Cadmium	µg/L	93	1.7	98.2%
Cobalt	µg/L	1,059	193	81.8%
Chromium	µg/L	2.5	0.8	68.0%
Copper	µg/L	7,584	1.1	100%
Manganese	µg/L	5,076	30,000	0%
Nickel	µg/L	1,129	162	85.7%
Lead	µg/L	45	39	13.3%
Iron	µg/L	274	240	12.4%
Mercury	µg/L	- [2/]	8.8	-
Zinc	µg/L	21,780	8.0	100%

NOTE:
[1] The WTP will not remove sulphate. The sulphate TV at SW4 will be achieved by dilution in the Edith River.
[2] Water quality data for mercury is not available.

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 houses three self-priming centrifugal pumps in a duty/duty/standby configuration. The Feed Pump Station pumps the collected water to the WTP building for treatment. The WTP process will consist of two-stage high density lime treatment and chemical precipitation with 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 300 m^3/hr, with a maximum available treatment capacity at 600 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	$163,000
HDS Sludge Conditioning Tanks and Mixers	$195,000
HDS Reaction Tanks	$1,560,000
NaSH Reaction Tank & Clarifiers	$3,640,000
Pressure Filters	$2,048,000
Backwash Waste Clarifier	$328,000
Treated Water Holding Tank	$312,000
Ferric Chloride Feed System	$111,000
Lime Silo, Slaker, and Feed System	$1,300,000
Process Plant Return Pumps	$163,000
Polymer System	$133,000
Sodium Hydrosulfide Feed System	$104,000

Parameter	Cost (US$)
Sulfuric Acid Feed System	$79,000
Treated Water Pumps	$163,000
Dust Suppression Pumps	$7,200
Lime Sludge Pumps	$182,000
Backwash Waste Sludge Pumps	$39,000
Concrete	$1,174,000
Pre-engineered Building	$2,352,000
Electrical and Instrumentation	$3,166,000
Piping, Pipe Supports, and Valves	$2,638,000
Engineering, Procurement, Construction	$3,552,000
Contingency	$2,575,000
Cyanide Probes	$9,100
HCN Gas Alarms	$18,000
Total	**$26,011,000**

The opinion of probable operating costs consists of electricity, labor and chemical consumption. The estimated electrical use at the site is 2,827,000 kWh annually. The estimated labor use at the site includes one and a half (1.5) supervisor/ certified operators 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.

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, 42% (liquid)	tonne	61	55	58	51	46	31	27	27	30	33	58	61
Lime, 100% as CaO (solid)	tonne	169	153	162	142	127	85	75	75	83	91	157	169
Sodium Hydrosulfide, 35% (liquid)	tonne	15	13	14	12	11	7.4	6.5	6.5	7.2	7.9	14	15
Sulfuric Acid, 98% (liquid)	tonne	13	12	13	11	9.8	6.6	5.8	5.8	6.4	7.0	12	13

24.2.2.3 Raw Water Reservoir and Pipeline

The existing Raw Water Dam (RWD) is the sole source for potable and elution water, as it is the only freshwater source on site. The existing RWD will be enlarged for the planned operation. 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 m^3.

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 an onsite potable water supply. The reservoir is designed to fill 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 pipe is connected to the fresh water pipeline that extends to the process plant. The spillway is designed to safely convey the Probable Maximum Flood event; the spillway discharges in to Horseshoe Creek.

The existing line from the RWD will need to be extended with an additional 20 m of 375 mm pipe to extend past the RWD embankment raise and allow the dam to be drained in case of a dam safety emergency.

To provide the required water supply to the PP, a volume of 740 cubic meters per hour will be initially supplied using an open air pump station, once at full production most of the PP water requirements will be met by reclaimed water from the PWP. This reclaimed water will be augmented with approximately 10-15% makeup water from the RWD. This includes construction of a 450 mm HPDE pipeline that runs parallel to the existing fresh water pipeline.

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 present at the site, specifically:

- Greywacke;
- Shale; and
- Mixed greywacke/shale (interbedded).

Eighty-seven (87) waste rock samples were subjected to acid-base accounting (ABA), to assess the acid-producing and acid-neutralizing potential of overburden and waste rock prior to mining or other large-scale excavations. Nine samples, including three samples from each of the three distinct units were selected for kinetic testing using humidity cell tests. This test provides an estimation of chemical leaching over time of the samples under oxidizing conditions and is useful in determining the effect of natural weathering of said materials during and post-mining. Mineralogy was determined by quantitative x-ray diffraction (XRD) on the nine humidity cell test samples.

The greywacke waste rock sample average extractable (sulfide) sulfur content was 0.19 wt. % utilizing a nitric acid leach (HNO_3). This was comparatively low as the interbedded and shale samples were 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 ≤ 0.51 wt. % HNO_3 extractable sulfide sulfur; however, the potential for acid formation cannot be discounted due to the limited amount of neutralization potential (NP) in the rocks. On average, the samples showed an NP ≤ 11 kg $CaCO_3$/tonne rock. An acid base accounting (ABA) neutralization potential ratio (NPR) screening criteria of < 2 suggests that a majority of the waste rock samples are either potentially acid generating or highly likely to generate acid. Waste rock comprised of these samples may require isolation from surface and/or ground water to inhibit acid generation. It should be noted, however, that approximately 30% of the samples are highly unlikely to generate acid. These samples contained high insoluble sulfur (> 30 wt. %) which are tied up in sulfide species that are resistant to chemical weathering such as sphalerite (ZnS) and/or galena (PbS).

Site specific sulfur based characterization criteria were developed based on ABA and non-acid forming (NAF) pH results, to assist with waste rock management and closure planning. The specific sulfur based characterization criteria utilized to predict acid generating risk are:

- NAF waste rock is defined by a 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 PAF waste rock is greater than 0.4 wt. %; and
- Waste rock with greater than 1.5 wt. % sulfur was considered to be likely acid generating.

The sulfur based categories 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 (28 weeks for six samples and 158 weeks for three samples). Monthly leachate composites for dissolved constituent concentrations were also obtained over the testing period. Of the nine samples subjected to kinetic testing, two samples produced acidic leachate. The first humidity cell test with acidic leachate was a shale sample with 0.43 wt. % HNO_3 extractable sulfide sulfur and low NP = 3.7 kg $CaCO_3$/tonne rock. This material produced acidic leachate (pH less than 6) from the initiation of testing. The second humidity cell test with acidic leachate was an interbedded greywacke/shale sample characterized as having uncertain acid generation potential. The leachate from this test dropped below a pH of 6 after 151 weeks of testing. Elevated copper, lead, nickel, and zinc levels were observed in leachate from the acid generating cells. 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.

Two tailings samples underwent geochemical characterization including ABA, mineralogy, water leaching, and supernatant analysis. These 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. Humidity cell testing was conducted on one of the samples. 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. However, the tailings supernatant and water leach testing produced alkaline pH values. After 32 weeks, kinetic testing of one of the samples shows a neutral pH with low concentration of metals. Calculations indicate that abundant sulfide sulfur still remains, suggesting the sample has the potential to produce acidic leachate given ample time and continued chemical weathering.

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:

- RP1 – WRD Retention Pond;
- RP2 – Low Grade Ore Stockpile Retention Pond (LGRP);
- RP3 – Batman Pit;
- RP5 – Plant Site Runoff Settling Pond;
- HLP – Heap Leach Pad Pond; and
- RP7 – Tailings Storage Facility 1 (TSF1) Pond;

- RP8 – Tailings Storage Facility 2 (TSF2) Pond; and
- Precipitation.

Monthly 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 anticipation of re-commencing mining activities, the water in RP 3 has been lowered to a level below where mining is scheduled to occur. Treatment of RP3 water 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 178-08 during the wet season. Since 2012 approximately 10.5 gigalitres of treated pit lake water has been discharged from the Batman Pit, lowering the water level sufficiently to begin mining activities within the pit.

24.4 Surface Water Hydrology

The Project Site is drained by the perennial Edith River, located approximately 0.6 km south of the proposed RP 1 dam, 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 upstream 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. The water management system will be further improved by the construction of a new RP 1 dam, that is required due to the planned enlargement of the WRD. The planned RP 1 facility will have an additional 180,000 m^3 of storage, with a total of 1,400,000 m^3 of storage.

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 WRD Diversion channel, 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), the Process Plant Retention Pond (PRP), and the Process Water Pond (PWP). However, for a large part of the year (approximately May to October), no runoff from the mine area enters the Edith River.

The mining infrastructure (TSF's, WRD, Batman Pit, Low Grade Ore Stockpile (LGOS), the processing plant) is located near or encroach upon the existing streams. Diversion channels were designed to convey water around the landforms and other infrastructure. The diversions around the TSF's and WRD are designed to convey the 10-year annual return interval (ARI) event. The diversions around critical mine infrastructure (Batman Pit, the processing plant, and LGOS) were designed to convey the 100-year ARI event. The channels were designed with a minimum of 0.33 meters of freeboard to account for hydrologic uncertainty and debris in the channels.

24.5 Regional Groundwater Model and Mine Dewatering

The Project will enlarge and deepen 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 pertinent 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 Section 7—Geological Setting and Mineralization, Section 8—Deposit Types, Section 9—Exploration, and Section 10—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 (Northern Territory Geological Survey, Katherine (NT), Sheet SD 53-9, Second Edition, 1994) indicates that the formations in the vicinity of the Batman Pit 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 (Rockwater, 1994). 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 highly weathered and 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. On-site meteorological records indicate that the average rainfall at the Project site is 1,235 mm/year, and more than 80% of the total 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 streams 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 alluvial sediments and 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 toward the west and northwest.

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 calibration of the regional groundwater model has been completed, additional calibration would be beneficial and the model has not yet been finalized or verified by comparison to measured groundwater inflows to the pit and measured changes in groundwater levels. Thus, the estimates of groundwater inflow to the expanded Batman Pit and post-mining groundwater system recovery should be considered preliminary. The model can be verified and finalized once mining has begun, and measurements of pit inflows and groundwater level changes become available. At that time, the model can be finalized and used to generate updated estimates of dewatering flows and dewatering effects on the groundwater system.

For this Technical Report, Tetra Tech developed estimates of groundwater discharge into the pit based on model output coupled with historical observations as discussed below. Estimates from the groundwater modeling suggest that groundwater inflows should initially be approximately 3 m^3/hr, gradually increase to approximately 35 m^3/hr mid-way through the mining period, then decrease to approximately 7 m^3/hr through the latter part of the mining period. The overall average groundwater inflow was predicted to be approximately 11 m^3/hr. 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 450 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 on historical mine operations 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, comprising only an estimated 4.5% of the total volume of water predicted to enter the pit. However, the large amount of precipitation and storm-water runoff has historically been a cause for concern. Therefore, for dewatering conceptual design, timely removal of storm-water runoff is a primary consideration. While groundwater inflows are expected to be negligible in terms of dewatering system design, they will be 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 probabilistic estimates of daily precipitation that were derived from the site meteorological database. Precipitation and runoff volume estimates were calculated through the life of mine based on the expanding area of the Batman Pit. The probabilistic estimates of runoff volumes were combined with the predicted groundwater inflow volumes to generate estimates of the volumetric dewatering requirements for the pit for each month through the life of mine. Volumetric estimates of monthly dewatering requirements including storm water and groundwater inflows during representative years of mine operation are listed in Table 24-9.

Table 24-9: Seasonal Inflow Volumes and Dewatering Pump Operating Times for Mine Dewatering Design

Mining Year	Nov-Jan (Wet Season) Mean Monthly Inflow Volume (m^3)	Nov-Jan (Wet Season) Mean Monthly Dewatering Pump Operating Hours	Jun-Aug (Dry Season) Mean Monthly Inflow Volume (m^3)	Jun-Aug (Dry Season) Mean Monthly Dewatering Pump Operating Hours
1	65,700	88	5,900	8
5	126,000	168	20,200	27
10	169,100	225	10,300	14
15	207,200	276	9,800	13

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, and groundwater inflow and would discharge to the PWP. Table 24-9 shows monthly estimated dewatering pump operating hours during representative years of mine operation, at the dewatering system design pumping rate of 750 m^3/hr. The actual pumping rate is expected to vary depending on availability of water storage and treatment capacity, as the dewatering effluent may require treatment prior to discharge.

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—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. While groundwater-related mine inflow estimates can 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, precipitation from storm events is expected to be the primary driver for the dewatering system.

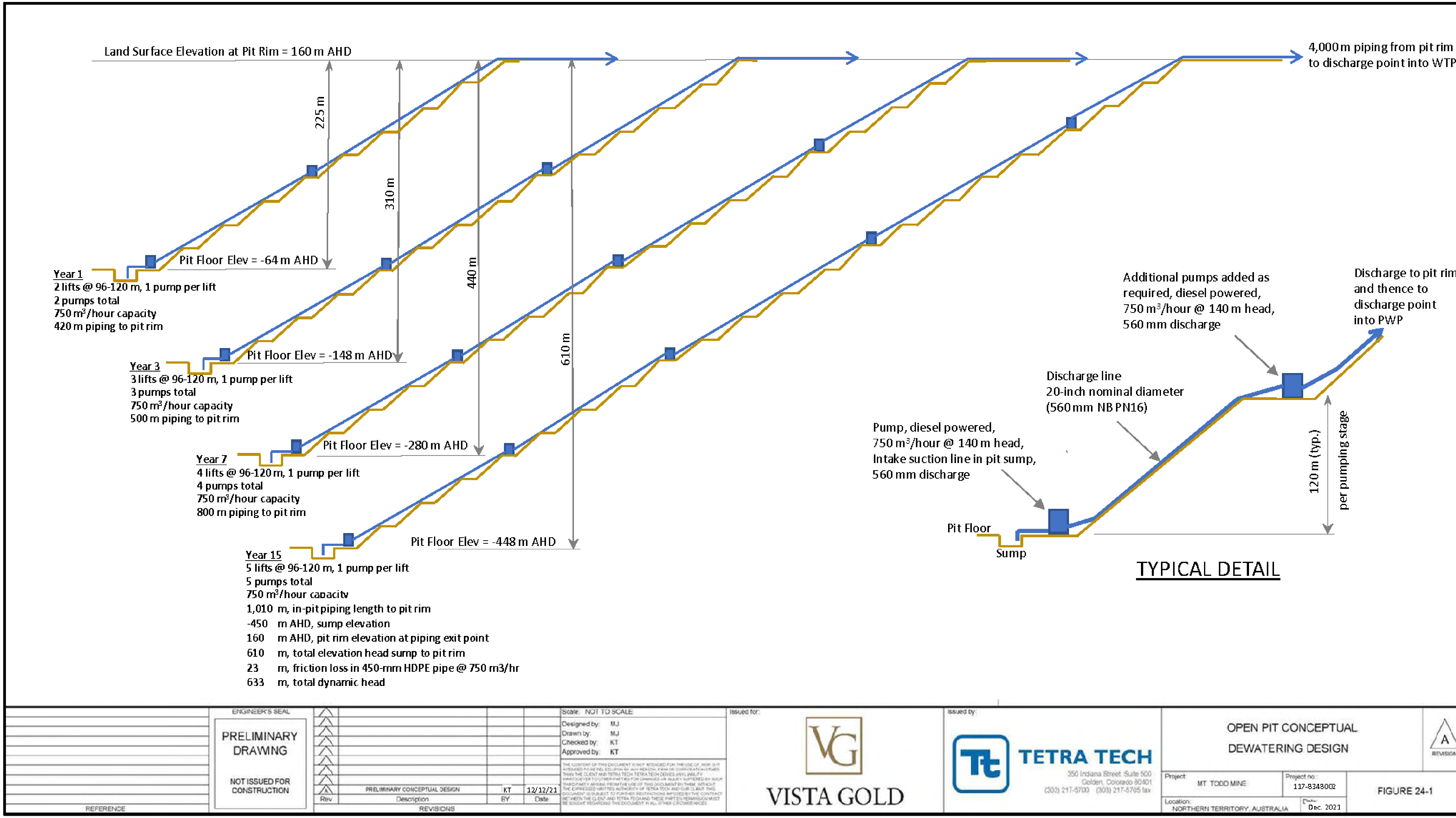


Figure 24-1: Open Pit Dewatering System Conceptual Design

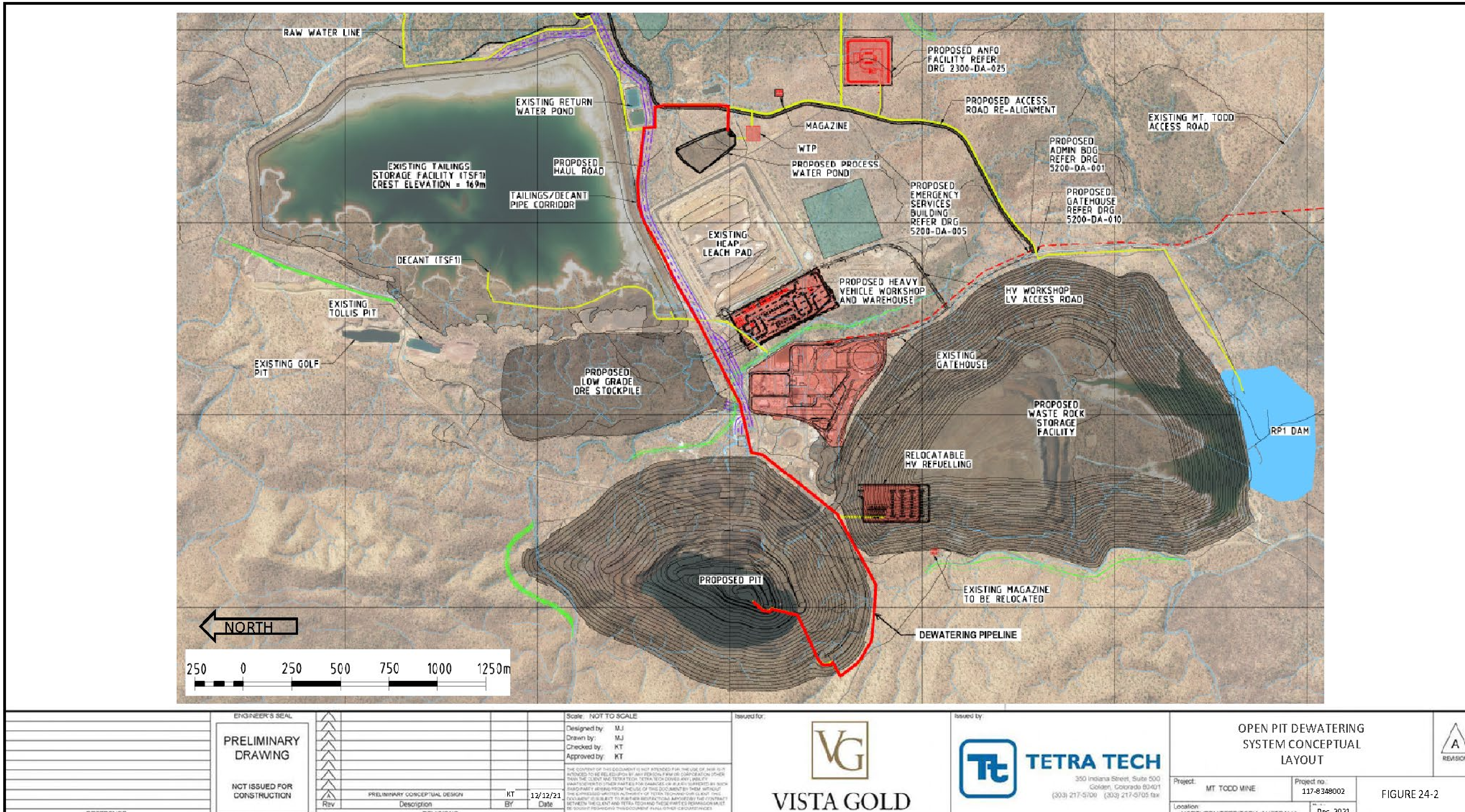


Figure 24-2: Conceptual Layout of Dewatering System

24.6 Project Implementation

24.6.1 Project Implementation Strategy

This section outlines a high-level Project Development Strategy, which will be further developed and confirmed during the next study phase of the Project.

The FS 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 with the prerequisite capability and experience to undertake the work.

To complement the EPCM approach, Vista may adopt Design and Construct (D&C) and Build Own and Operate (BOO) 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 EPCM 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.

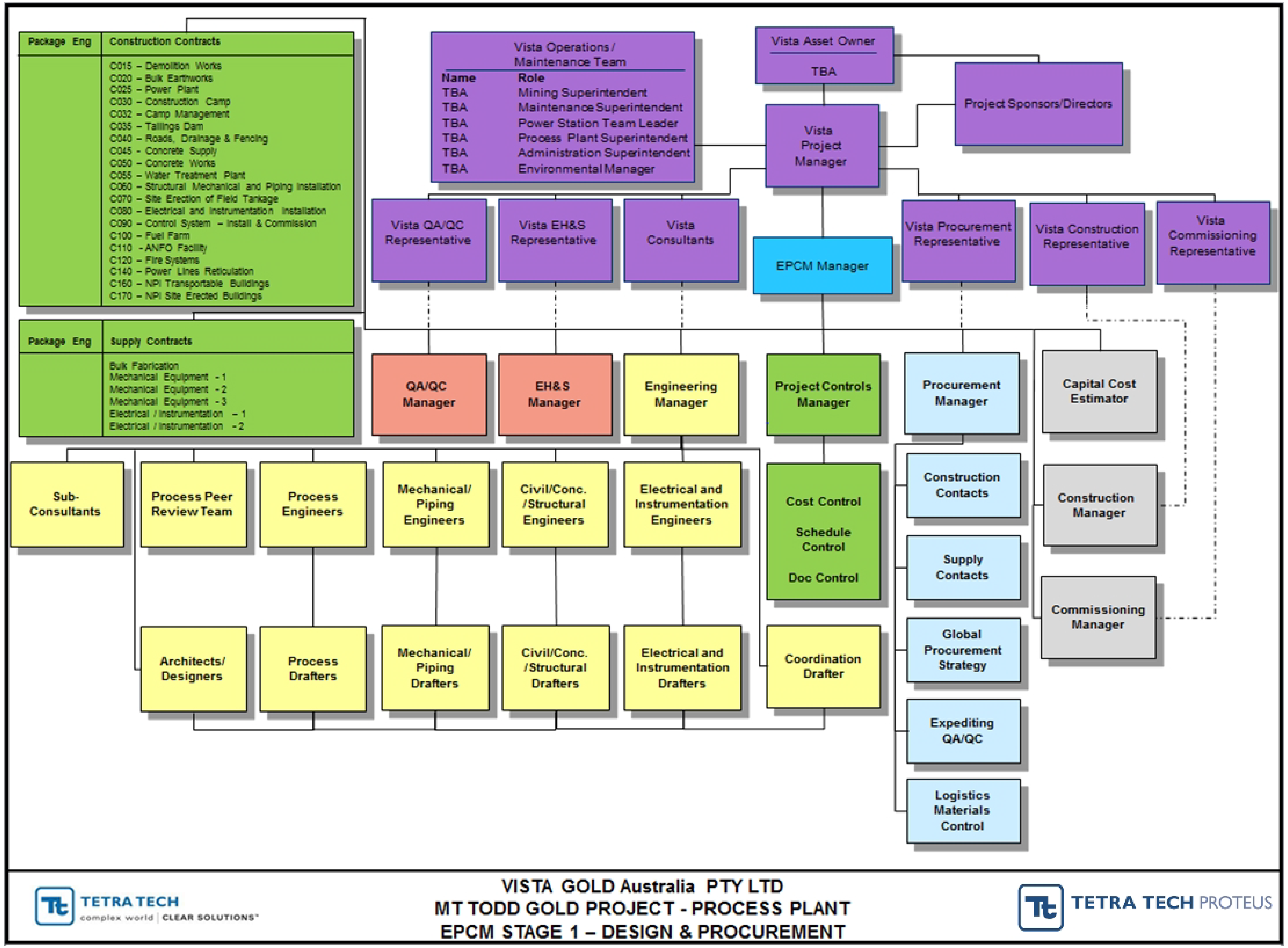


Figure 24-3: EPCM Stage 1 – Design & Procurement. Refer Diagram 1

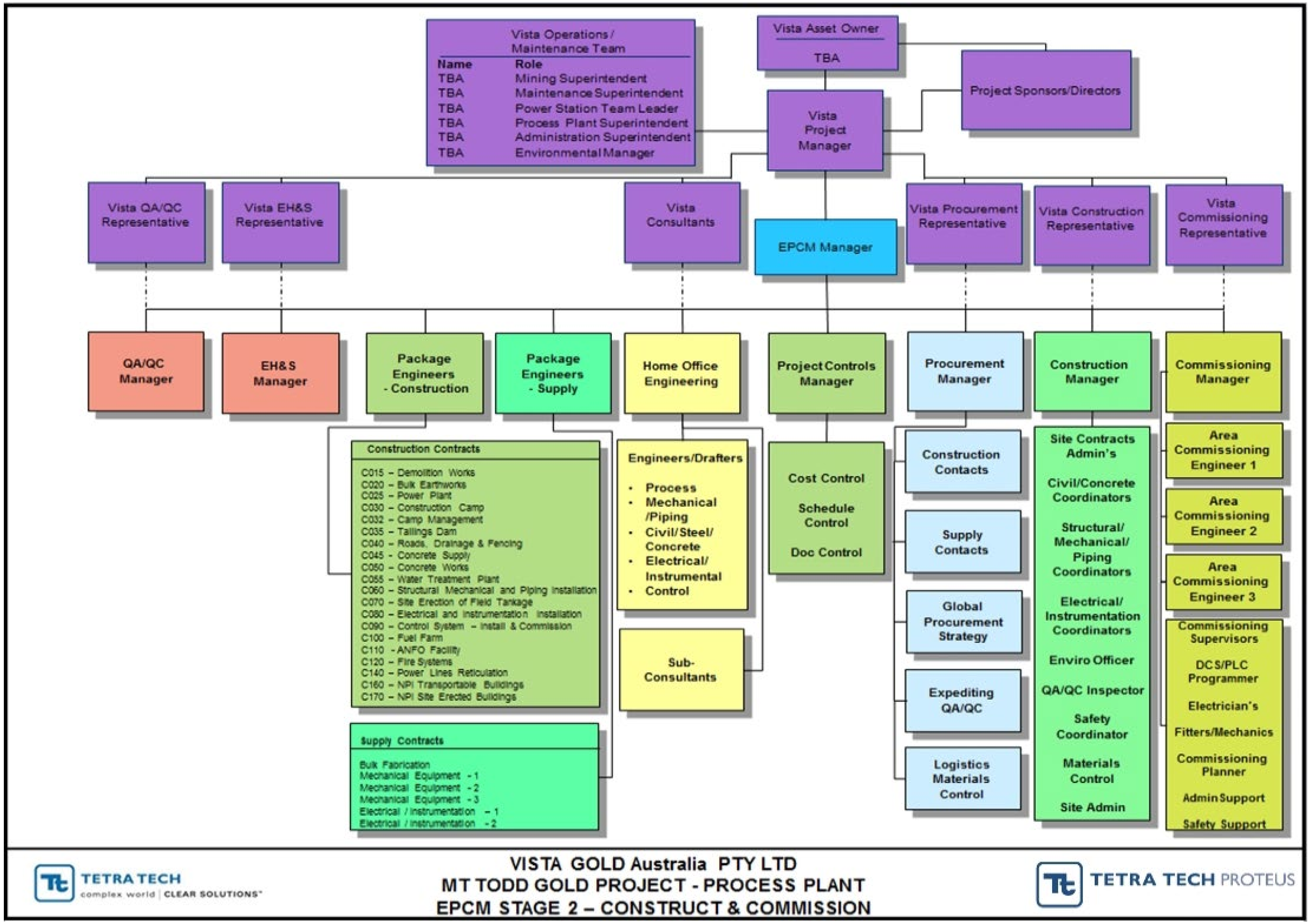


Figure 24-4: EPCM Stage 2 – Construct & Commission. Refer Diagram 2

24.6.2.2 D&C Contracts

D&C contracts that are proposed include:

1) Non-Process Infrastructure (NPI) – Transportable Buildings (Package C160).
2) NPI – Site Erected Buildings (Package C170).
3) Power Plant (Package C025 – by Power Engineers).
4) Gold Recovery Circuit package, Lime and Cyanide mixing plants – To be confirmed with vendor(s) in next phase of project.

24.6.2.3 EPCM Contract Scope of Services

Generally, the EPCM Contractor 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
- Collation and review of Operation and Maintenance manuals.

24.6.3 EPCM Management

The EPCM Contractor 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 Project Manager and other Managers associated with the project.

24.6.4 Engineering

The EPCM Contractor 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 as this detail is provided.

The EPCM 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 (PSR)
- 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 EPCM 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 exceed project environmental, sustainability, and community expectations.
- Meet or exceed the project schedule.
- Meet or better the project budget.
- Meet project quality objectives.

24.6.6.2 Procurement Overview

The EPCM Procurement Manager will report directly to the EPCM Project Manager but will also liaise directly with Vista's Commercial Manager.

The 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 (both 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 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 EPCM Procurement group. All purchase orders and contracts will be prepared by the EPCM Procurement group but issued through Vista's purchasing system.

Where goods and services are required from outside Australia, the EPCM 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 EPCM 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 Installation Works
C055	Water Treatment Plant
C060	Structural Mechanical and Piping Installation

Package No.	Package Description
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 EPCM Procurement Manager will be responsible for the development of an Equipment and Services Supply Contracting strategy. A preliminary strategy is documented in the Contracting and Procurement Plan.

The following supply packages are envisaged as a minimum:

Table 24-11: Supply Packages

Package No.	Package Description
P001	Ball 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
P019	SMBS Mixing Package
P020	Flocculant Mixing Package
P021	Lime Slaker
P023	Potable Water Plant
P024	Mill Relining Machine

Package No.	Package Description
P025	Overhead Travelling Cranes
P026	Air Compressors, Driers & Receivers
P027	Fuel Farm - Diesel
P028A	Conveyor Drives
P028B	Conveyor Pulleys
P028C	Conveyor Idlers
P028D	Conveyor Belts & Splicing
P028E	Conveyor Skirts
P028F	Conveyor Scrapers & Ploughs
P029	Ore Sorting
P030	Secondary Grinding Mills
P031	Wet Scrubber
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

Package No.	Package Description
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
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
P220	Fabricated Site Erected Tankage
P230	Fabricated Pipe Work

24.6.6.5 Indirect Packages

The EPCM 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 EPCM Contractor 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 EPCM Construction Manager will establish a core 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 EPCM 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 and 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 be comprised 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 EPCM Contractor will develop a Commissioning Management Plan, in collaboration with the Vista Commissioning Representative. The EPCM 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:

1) 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.;
2) 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
3) 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.

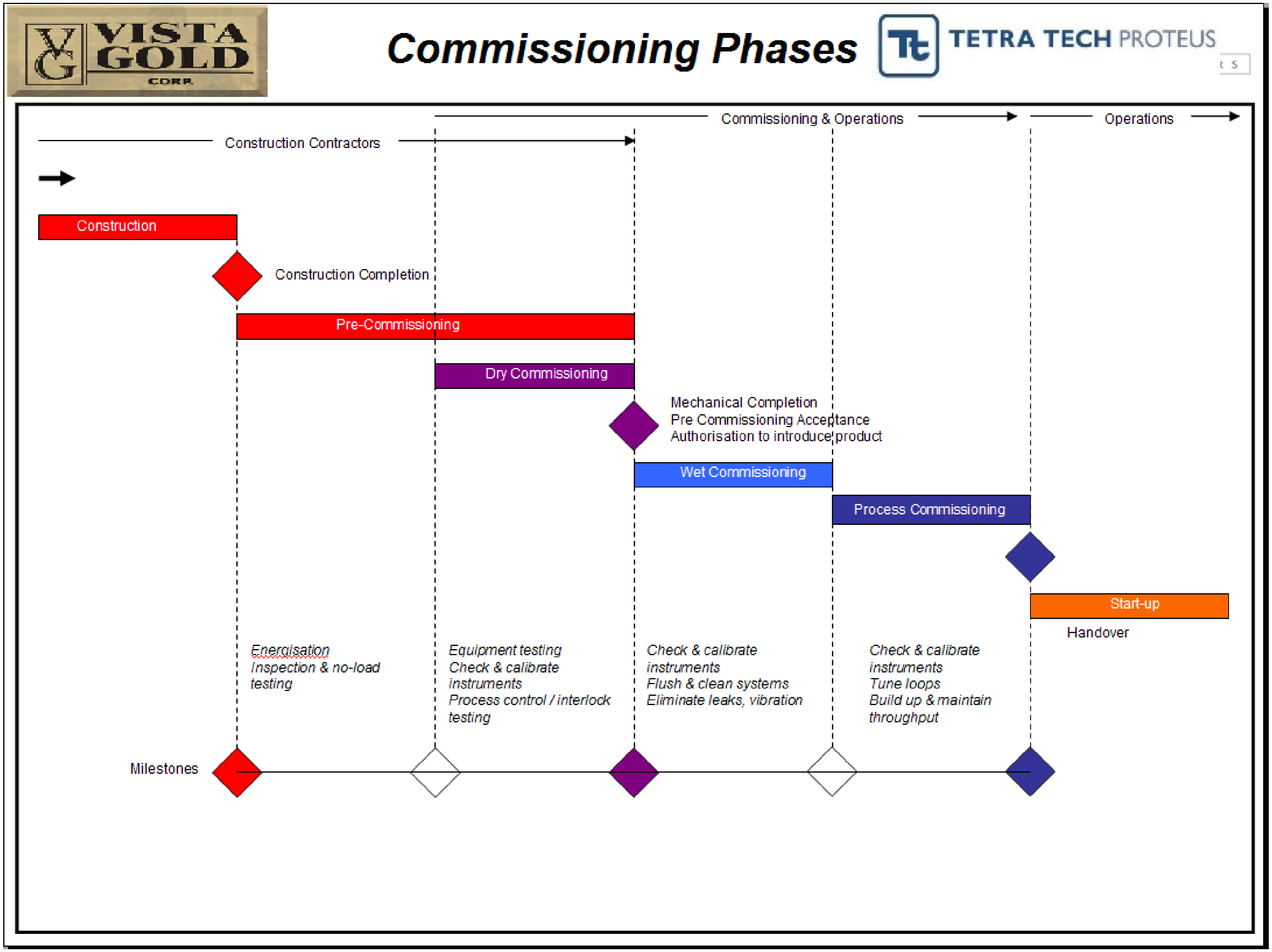


Figure 24-5: Commissioning Phases

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 (excluding accommodation) to an appropriate standard that complies with local EHS guidelines for offices and amenities and to the approval of the EPCM 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/NZS ISO 45001 Occupational Health and Safety Management System 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 EPCM Contractor 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 EPCM Contractor's Site Safety Officer and implementing the provisions of the relevant Occupational Health, Safety and Welfare Act.

The EPCM Contractor'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 EPCM 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 EPCM Contractor'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 FS 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 FS. 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 mid-2022. Startup, as defined by handover after completion of commissioning, is scheduled to late 2024.

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 ....8 weeks
- P001 Ball Mills Manufacture and Delivery...68 weeks
- P001 Ball Mills SMP Construction..36 weeks
- Area 3300 Verification and Commissioning ....4 weeks

The above critical activities determine a critical path of approximately 114 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
P003 – Secondary Crusher	55 weeks
P004 – HPGRs	60 weeks
P012 – Agitators	52 weeks
P024 – Mill Relining Machine	46 weeks
P029 – Ore Sorting	48 weeks
P030 – Secondary Grinding Mills	49 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 FS 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 EPCM 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

25. 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	Medium
Permitting & Regulatory Approvals	The Project has received EIS, EPBC, and MMP authorizations as described in Section 20.	Low	Medium
Property Holdings	Vista has secured the Mt Todd concession holdings as described in Section 4. 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 due to pass-through of costs from a third-party power supplier.	Low	Low-Medium
Reagents & Consumables	The process operating costs are sensitive to global changes in reagents and consumables pricing.	Medium	Medium
Fuel	The Project operating costs are sensitive to global changes in prices for diesel and natural gas.	Medium	Medium

Risk	Description	Probability	Severity
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	Low-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. Influent water quality is not completely understood and could impact treatability.	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 several mineralization events 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:

- 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 a metal price of approximately US$1,500 per ounce Au. The Mt Todd proven and probable reserves have been defined using economics based on a gold price of US$1,750 per ounce and an elevated cutoff grade of 0.35 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 was 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 the QP [Thomas L. Dyer, P.E.] 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 operation had 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. This flowsheet was adopted for this FS.

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 Process Risks

25.7.1 Equipment Performance

- Potential Issue: The test work showed considerable standard deviation in the CWi. This must be seen in conjunction with the site history of poor crushing throughput.
  - Mitigation Strategy: A conservative, yet pragmatic, upper quartile figure has been chosen for the CWi.
- Potential Issue: Normally ball mills have their feed tightly controlled in order to maximize throughput. As the screen product is not controlled in this case, feed to the mill and consequently the cyclone pressure control may fluctuate, reducing grinding and classification efficiency. The practice of automating the number of cyclones in operation is common in industry, as is manual operation of the cyclones.
  - Mitigation Strategy: Automating the number of operating cyclones, based on pressure would be a requirement, rather than an option for this plant.

- Potential Issue: If significantly extra fine screen oversize reports to the HPGR feed, then the combined HPGR feed moisture might become excessive and would lead to poor HPGR performance and excessive wear.
  - Mitigation Strategy: The screens can have weir bars added at a low cost to increase their capacity. The circuit is also designed with an adjustable water to solids ratio on the feed to the fine screens. These factors will mitigate this risk to an acceptable level.
- Potential Issue: Design of the ore sorting area was significantly impacted by the quantity of Ore Sorters nominated by the Vendors, which raises questions about actual throughputs for the plant.
  - Mitigation Strategy: The Vendor who had carried out testwork on their Ore Sorters was ultimately selected to mitigate risk. Ore sorting technology is being continually improved and could allow like-for-like replacement of units as is designed, potentially increasing throughput of the ore sorting plant with no significant change to bulk commodities.

25.7.2 Leach / Adsorption / Desorption Performance

- Potential Issue: A low copper leaching is expected, due to the mining plan controlling the blend to a very low oxide:sulphide ratio. There is no allowance for oxide "surges" which would significantly increase the copper leaching, potentially overloading the carbon adsorption and elution systems.
  - Mitigation Strategy: Vista plans to achieve this via blending of the ore to the ROM pad and so it is not required to be allowed for in the plant.
- Potential Issue: It is expected that the copper will precipitate out in detoxification and that the precipitate will settle in the tailings dam. If it is re-leached or remains in solution, then the copper would be brought back into the circuit with a high recirculating load and excessive detoxification reagent use.
  - Mitigation Strategy: The metallurgical testwork to date has shown that it would precipitate out in the dam. In addition, a bleed stream to the WTP will reduce the likelihood for excessive build-up of deleterious components in the aqueous phase within the process plant.

25.7.3 Operations

- Potential Issue: The reagent mixing schedule is conceptually defined but detailed vehicle traffic plans are not yet prepared/optimized.
  - Mitigation Strategy: This can be optimized in the next phase (FEED) as it would not make a material difference to any decision making from the FS.
- Potential Issue: The crushing system availability allowance assumes that downtime due to ROM ore supply is zero. If the mining fleet cannot achieve this, then the current crushing system could be undersized.
  - Mitigation Strategy: Vista plans to provide for an emergency dump on the ROM pad and a loader to feed from this storage into the mouth of the primary crusher.

25.7.4 Capital Cost and Operating Cost Risks

The following is a list of CAPEX and OPEX risks that existed or were identified during the FS and previous phases (if relevant).

- Vendors with competitive equipment offerings for Ore Sorting and Secondary Grinding were reviewed for merits, but weren't selected in favor of Vendors who had carried out testwork. Undertaking test work with other Vendors may give them the opportunity to prove their equipment/technology and could reduce CAPEX and OPEX in these areas.
- Containerized (bulk) supply of SMBS and cyanide, sparged supply of cyanide and bulk supply of grinding media has no vendor commitment as yet and therefore carries an element of risk. However, this risk has

been mitigated in the FS by selection of vendors that have local capability of bulk delivery at the scales required by the project.

- Due to the COVID pandemic, geopolitical factors and general volatile state of international trade, many vendors noted that prices for raw materials and shipping rates have been fluctuating significantly but best estimates were provided at the time of budget quotation. Due to these factors, an element of risk exists due to future fluctuation.

25.7.5 Schedule Risks

The following items are considered Long Lead Items and will need careful consideration in the procurement strategy to ensure they do not impact the overall schedule:

- Primary crushers
- Secondary Crushers
- Ball mills
- HPGRs
- Secondary Grinding Mills
- Ore Sorters
- Agitators

During the implementation phase, a number of further issues may become apparent and therefore issues such as these will need to be addressed in the following phases of the project:

- Securing Contracts with Reagent Suppliers in the product form included in the design
- Availability of skilled labor (If other projects are being run concurrently)
- Capacity to fabricate such a quantity of steelwork and pipework (especially if fabricated in transportable units from overseas)
- Accessibility to site during construction if a cyclone occurs further North.
- Disruptions to the global supply market due to COVID-19.

25.7.6 Hazard Identification Study

A Hazard Identification Study (HAZID) was held on TTP scope items. This reviewed the potential for a number of issues for each area of the plant, as well as for the overall site. The action items were addressed during the FS, with some exceptions expected to be addressed during the next phase of this project.

25.7.7 Health, Safety, Environment and Community

Whilst some broad Health, Safety, Environment and Community (HSEC) issues have been addressed in the design reviews to date, detailed HSEC analysis will not be required until later stages of the project.

25.8 Environmental and Social Conclusions

25.8.1 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.

25.8.2 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.

25.8.3 Social or Community Impacts

The Jawoyn Association has been consulted as part of the planning process for the future of the Project. Areas of aboriginal significance have been designated, and the Project is in receipt of the Aboriginal Areas Protection Authority (AAPA) Certificate. This was required as a legal means to identify and protect sacred sites from damage by setting out the conditions for using or carrying out works on an area of land. It is a legal document issued under the Northern Territory Aboriginal Sacred Sites Act.

Following extensive review, the AAPA determined that the use of, or work on, certain areas can proceed without a risk of damage to, or interference with, the sacred sites identified at Mt Todd. The AAPA Authority Certificate for Mt Todd covers the 1,501 km² of exploration licenses contiguous with the mining leases.

Community-based Staffing Discussion

Vista has worked closely with community and territory leaders in designing a community-based project, as opposed to the more traditional fly-in, fly-out (FIFO) operations commonly seen in Australia. Unlike many mining operations in the country, Mt Todd is easily accessible (approximately 250 km from Darwin) and conveniently located near well-established population centers. Mt Todd is approximately 30 minutes from Katherine and 45 minutes from Pine Creek. Katherine is a regional commerce center and home to approximately 14,000 people in the community and surrounding area.

The NT government strongly promotes job creation in the territory for territorians. A key focus is creating revenue that stays in the territory. The Katherine town council has expressed concerns about the influx of construction workers and requested that the construction camp be located north of the Katherine River. Vista has accommodated this request and selected a site at the project for the construction camp. They have also worked with the NT Department of Lands, Planning and Infrastructure to ensure that crown land will be made available for additional housing development in Katherine.

Vista is committed to hiring locally and will implement training programs, supported by both State and Federal Governments, to develop the skills needed to gain employment at the mine. They do not have a quota with regard to local or aboriginal workers, but expect these numbers to be an important part of their total employment. Vista is aware of a significant number of territorians who are employed at other mines in Australia on a FIFO basis. They believe a number of them will find the benefits of employment that allows them to be home every night to be very attractive.

Of the approximately 525 full-time employees at the peak, approximately 410 will be required in the early years of the project. Vista expects that ~40% will come from the local community and will participate in training programs to develop skills needed for employment. They anticipate that another ~20% will be experienced workers who reside in the territory, but presently work a FIFO roster elsewhere. Vista anticipates they will need to recruit and incentivize another 20-30% to move to the territory. They recognize that to fill certain key technical and management positions, they may have a small percentage of the workforce that works on some form of FIFO roster or resides in Darwin and lives in the scaled-down camp during the week.

Experience in the territory, specifically the construction of the IMPEX LNG facility, suggests that many professionals and tradespeople find the territory to be a wonderful place to live and don't want to leave after living there. Vista plans to provide a work environment that is very supportive of families and community living. They intend to continue to work closely with the Katherine town council to ensure that Katherine is able to meet the challenges of growth and provide services/opportunities for a thriving community.

25.9 Results of the Site-wide Water Balance Model

Under the modeled conditions, the SWWB model results indicate that:

- The WTP rate of 600 m^3/hr and process water pond (PWP) sizing of 185,000 m^3 of storage appear adequate for the 50,000 tpd production process water requirements.
- The greatest amounts of make-up water required from the raw water dam (RWD) was quantified as 15,000 m^3/day from year 2 to year 5 and 11,500 m^3/day from year 6 to 17. RWD requirements were found to be the most dependent upon TSF decant rates and water treatment plant (WTP) effluent make-up water availability.
- The Batman Pit will see minor water storage during the wet season especially late in the LoM when groundwater inflow and stormwater runoff volumes are highest.
- The Waste Rock Dump (WRD) retention pond (RP1), low grade ore stockpile retention pond (LGRP), and heap leach pad (HLP) were typically observed to overtop less than 5% of the time during production.[4]

25.10 Water Treatment Plant

In review of the SWWB, geochemical modeling and the Water Discharge Licence, conclusions reached for the Water Treatment Plant include the following:

- Two stage lime treatment at pH 6.5 and pH 10.0, followed by chemical precipitation and filtration is required to meet water quality goals based on the SWWB model results for treatment flow variations between wet season and dry season.
- The WTP water quality goals are based on a 1:19 flow dilution (WTP: Edith River) to maintain sulfate levels below the TV at SW4 in the Edith River.
- Influent water quality will not be known until mine operations commence and is expected to change over the life of the mine.

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

26. RECOMMENDATIONS

All required work is complete for this FS and no additional work is necessary for this phase. This FS presents a project that is ready for submission for financial and other support necessary to progress to the next phase.

The next phase of the project is detailed design, which follows a financial investment decision. The recommendations that follow are considered part of the detailed design phase and represent the normal progression of the project from an FS to construction.

26.1 Mining

- Continued review of WRD designs focusing on potentially reducing the overall footprint of the dumps may result in reduced closure costs.
- Concurrent reclamation of the WRD is planned, however most reclamation costs will be incurred following the completion of mining. Concurrent reclamation will allow operations people and equipment to be used for reclamation more efficiently.
- 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.
- Large stockpiles of low-grade ore are used for reasonably long periods of time. Opportunities to use these stockpiles as quickly as possible should be explored.

26.2 Environmental Studies

Along with the ongoing precipitation, stream flow, and wildlife data that is currently being—and will continue to be—monitored and collected, additional studies to further assess environmental baseline conditions and support detailed design, permitting, and closure planning for the Project, are recommended below.

- Erosion analyses.
- Waste and cover material (including WRD liner system) hydraulic properties characterization and analysis.
- WRD closure liner system hydraulic and geotechnical stability analyses, including interface strength analysis of liner system components and waste rock, slope stability analyses for static and pseudo-static conditions, deformation modeling, consolidation, and differential settlement evaluations.
- WRD liner longevity and liner breach evaluations.
- Seepage analyses for the WRD and TSFs reflecting additional site-specific data (as available), closure designs, and longer-term climactic conditions and potential variations.
- Pilot testing of the water treatment plant and passive water treatment wetlands.
- Pit lake modeling updates based on additional site-specific data (as available).
- Further investigation to identify a source of low-permeability material suitable for use in closure covers located closer to or within the project site boundaries.

26.3 Groundwater Hydrology

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

- 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.

26.4 Tailings Facility Design

The following studies and investigations are recommended to support future phases of TSF design for the project:

- A formal TSF risk assessment is recommended as part of the detailed design phase. The assessment should include a dam break and inundation study used to review and update the potential for impacts to business, social and environment, and the potential for loss of life associated with credible dam failure modes.
- The construction sequence will be a dynamic process during the life of the mine, as a range of influencing factors will change, such as tailings properties, deposition strategy, water management, and actual production rate. It is recommended that a detailed deposition plan be prepared, that deposition be divided in sectors along the embankment, and thin lift deposition practices be followed to allow effective tailings beach management.
- The reduction of the TSF2 embankment raise increments and TSF overall slope may be warranted if operation conditions or tailings properties change from current design basis.
- The condition of the existing toe drains, underdrains, and decant towers should be investigated to confirm their condition prior to re-commissioning of TSF1.
- As part of detailed design work, the waste rock strength and hydraulic conductivity of the waste rock used to construct the dam embankments should be validated to confirm adopted model parameters.
- TSF1 and TSF2 are to be constructed using upstream raise methods. It has become accepted practice that critical state soil mechanics should be used to evaluate liquefaction susceptibility, stability assessment, and triggering analyses in upstream facilities. Specifically, the material properties of the deposited tailings should be confirmed at several stages during operation as part of construction quality control and assurance. This would involve geotechnical drilling, including CPT investigation to evaluate the strength and drainage parameters.

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28. 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 QPs 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
Maurie Marks, P.Eng.	Senior Mining Engineer *Tetra Tech*	Sections 1.11, 1.13, 1.14, 19, 21, 21.1, 21.2, and 22
Rex Clair Bryan, PhD, SME RM	Principal Geostatistician *Tetra Tech*	Sections 1.4, 1.5, 6, 6.1, 6.2, 6.3, 6.4, 7, 8, 9, 10, 11, 12, 14, and 25.2
Thomas L. Dyer, P.E., SME RM	Mining Engineer *RESPEC LLC*	Sections 1.6, 1.7, 1.15, 15.1 to 15.5, 16, 21.1.1, 21.2.1, and 25.3
Amy L. Hudson, PhD, CPG, SME RM	Principal Hydrogeologist/ Geochemist *Tetra Tech*	Section 20.2 and 24.3
April Hussey, P.E.	Environmental Engineer *Tetra Tech*	Sections 20.5 and 21.1.4
Chris Johns, M.Sc., P.Eng	Geological Engineer *Tetra Tech*	Sections 18.2.4, 21.1.7, and 21.2.5
Max Johnson, P.E.	Civil Engineer *Tetra Tech*	Sections 24.2.1 and 25.9
Deepak Malhotra, PhD, SME RM	Principal Metallurgist *Forte Dynamics, Inc.* *(formerly with RDi)*	Sections 1.6.1, 1.8, 1.9, 6.5, 6.6, 13, 15.6, 17, and 25.4
Zvonimir Ponos, BE, MIEAust, CPeng, NER	Senior Principal Engineer *Coffey Services Australia Pty Ltd* *(trading as Tetra Tech Proteus)*	Sections 1.10, 18, 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, 18.8, 21.1.2, 21.2.3, 21,2.6, 24.1, 24.6, 25.5, 25.6, and 25.7
Vicki J. Scharnhorst, P.E., LEED AP	Principal *Tetra Tech*	Sections 1.1, 2, 3, 4, 5, 20.1, 20.3, 20.4, 23, 24.2, 24.4, 25.1, 25.8, 25.10, and 26
Keith Thompson, CPG, member AIPG	Professional Geologist *Tetra Tech*	Sections 21.1.3, 21.2.2, and 24.5

28.2 Table of Responsibility

QPs are responsible for all subsections listed beneath headings unless subsections are detailed below.

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24.2.1	Site-wide Water Balance	Johnson, Max
24.2.2	Wet Infrastructure	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
25	**INTERPRETATION AND CONCLUSIONS**	**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	Process Risks	Ponos, Zvonimir
25.8	Environmental and Social Conclusions	Scharnhorst, Vicki
25.9	Results of the Site-wide Water Balance Model	Johnson, Max
25.10	Water Treatment Plant	Scharnhorst, Vicki
26	**RECOMMENDATIONS**	**Scharnhorst, Vicki**
27	**REFERENCES**	**N/A**

Section No	Section Name	QP
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

Maurie Marks, P.Eng.
Senior Mining Engineer
Tetra Tech
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Maurie Marks, P.Eng.**, do hereby certify that:

1) I am a Senior Mining Engineer with Tetra Tech, with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, BC V6C 1N5.
2) I am a graduate of Montana Tech University (B.Sc Mining Engineering, 2013). My relevant experience includes 12 years of experience in the evaluation of mining projects, financial analysis, and mine planning and optimization. I have been involved in the technical studies of several base metals, gold, silver, and aggregate mining projects in Canada and abroad.
3) I am a member in good standing of Engineers and Geoscientists British Columbia, License number 45716.
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 not visited and inspected the property which is the subject of the Technical Report.
6) I am responsible for Sections 1.11, 1.13, 1.14, 19, 21, 21.1, 21.2, and 22 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 no prior involvement with the property that is the subject of the Technical Report.
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 16th day of April, 2024.

Signed, Sealed "Maurie Marks, P.Eng."
Signature of Qualified Person

Maurie Marks, P.Eng.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Rex Clair Bryan, Ph.D., SME RM
Principal Geostatistician
Tetra Tech
390 Union Blvd., Suite 400 | Lakewood, CO 80228
Telephone: (303) 217-5700
Email: Rex.Bryan@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Rex Clair Bryan, Ph.D.**, do hereby certify that:

1) I am a Senior Geostatistician with Tetra Tech, with a business address at 390 Union Blvd., Suite 400, Lakewood, Colorado 80228 USA.
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 more than 30 years since my graduation. My relevant experience is in the areas of resources and reserve reporting. I am a Competent/Qualified Person 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, 6, 6.1, 6.2, 6.3, 6.4, 7, 8, 9, 10, 11, 12, 14, and 25.2 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 qualified person for the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018; the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, amended and restated date July 7, 2014; and the Amended and Restated NI 43-101 Technical Report Resource Update Mt Todd Gold Project, Issue Date of Amendment and Restatement: April 11, 2012.
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 16th day of April, 2024.

Signed, Sealed "Rex Clair Bryan, Ph.D., SME RM"
Signature of Qualified Person

Rex Clair Bryan, Ph.D., SME RM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Thomas L. Dyer, P.E., SME RM
Mining Engineer
RESPEC LLC
3824 Jet Drive | Rapid City, SD 57703
Telephone: (605) 394-6400
Email: tom.dyer@respec.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Thomas L. Dyer, P.E.,** do hereby certify that:

1) I am a Principal Engineer with RESPEC LLC, with a business address at 3824 Jet Drive, Rapid City, South Dakota 57703 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 27 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 July of 2017 for 2 days.
5) I am responsible for Sections 1.6, 1.7, 1.15, 16, 15.1 to 15.5, 21.1.1, 21.2.1, and 25.3 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, amended and restated September 22, 2020; NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018; the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study amended and restated July 7, 2014; the 10.65 Mtpy Preliminary Feasibility Study NI 43-101 Technical Report Mt Todd Gold Project dated January 18, 2011; and the Preliminary Feasibility Study NI 43-101 Technical Report Mt Todd Gold Project dated October 1, 2010.
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 16th day of April, 2024.

Signed, Sealed "Thomas L. Dyer, P.E., SME RM"
Signature of Qualified Person

Thomas L. Dyer, P.E., SME RM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Amy L. Hudson, Ph.D., CPG, SME RM
Principal Hydrogeologist/ Geochemist
Tetra Tech
1750 Kraft Drive, Suite 1503 | Blacksburg, VA 24060
Telephone: (703) 885-5447
Email: Amy.Hudson@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Amy L. Hudson, Ph.D., CPG, REM**, do hereby certify that:

1) I am a Principal Hydrogeologist/Geochemist with Tetra Tech, 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 25 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). I am a Registered Member of SME (#4151963) 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 not visited and inspected the property which is the subject of the Technical Report.
5) I am responsible for Sections 20.2 and 24.3 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 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 16th day of April, 2024.

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

April Hussey, P.E.
Environmental Engineer
Tetra Tech
390 Union Blvd., Suite 400 | Lakewood, CO 80228
Telephone: (303) 217-5700
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **April Hussey, P.E.**, do hereby certify that:

1) I am an Environmental Engineer with Tetra Tech, with a business address at 390 Union Blvd., Suite 400, Lakewood, Colorado, 80228 USA.
2) I graduated with a Bachelor of Science degree in Chemical Engineering in 2001 from Montana State University, Bozeman, Montana. In addition, I graduated with a Master of Science degree in Civil Engineering in 2005 from the University of Colorado, Boulder, Colorado. I have worked as an Environmental Engineer for a total of 18 years since my graduation. My relevant experience is in the area of mine water management, environmental permitting, reclamation, and water treatment operations. I am a P.E. in Colorado (No. 43907).
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 20.5 and 21.1.4 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 and author of the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; and serving as a supporting project engineer primarily in areas of water management and treatment, environmental permitting, and reclamation and closure since 2011.
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 16th day of April, 2024.

Signed, Sealed "April Hussey, P.E."
Signature of Qualified Person

April Hussey, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Chris Johns, M.Sc., P.Eng.
Geological Engineer
Tetra Tech
1715 Dickson Avenue, Suite 150 | Kelowna, BC V1Y 9G6
Telephone: (250) 862-4832
Email: Chris.Johns@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Chris Johns, M.Sc., P.Eng.**, do hereby certify that:

1) I am a Senior Consultant with Tetra Tech, 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 over 25 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 personally visited and inspected the property which is the subject of the Technical Report on June 28th and June 29th, 2017.
5) I am responsible for Sections 18.2.4, 21.1.7, and 21.2.5 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 consisted of acting as a qualified person for the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; 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 16th day of April, 2024.

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

Max Johnson, P.E.
Civil Engineer
Tetra Tech
390 Union Blvd., Suite 400 | Lakewood, CO 80228
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Email: max.johnson@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Max Johnson, P.E.**, do hereby certify that:

1) I am a Civil Engineer with Tetra Tech, with a business address at 390 Union Blvd., Suite 400, Lakewood, Colorado, 80228 USA.
2) I graduated with a Bachelor of Science degree in Civil Engineering in 2012 from Colorado State University, Fort Collins, Colorado. I have worked as an Civil Engineer for a total of 9 years since my graduation. My relevant experience is in the area of mine water management, preparing mine site and facility specific water balances, collecting and evaluating meteorological information, hydrology, and reclamation,. I am a P.E. in Colorado (No. 0051790).
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 24.2.1 and 25.9 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 serving as a supporting project engineer primarily in areas of water management, water balance, and reclamation and closure since 2012.
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 16th day of April, 2024.

Signed, Sealed "Max Johnson, P.E."
Signature of Qualified Person

Max Johnson, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, Ph.D., SME RM
Principal Metallurgist
Forte Dynamics, Inc.
12600 W. Colfax Ave., Suite A-540 | Lakewood, CO 80215
Email: dmalhotra@fortedynamics.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Deepak Malhotra, Ph.D.**, do hereby certify that:

1) I am the Director of Metallurgy for Forte Dynamics, Inc., with a business address of 12600 W. Colfax Ave., Suite A-540, Lakewood, Colorado 80215 USA.
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. My relevant experience includes working as a metallurgist and mineral economist for over 50 years since my graduation with specific expertise in mineral processing, metallurgical testing, and recovery methods. I am a member of the Society of Mining Engineers.
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, 6.5, 6.6, 13, 15.6, 17, and 25.4 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 Process Development of the project. I was a qualified person for the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; the NI 43-101 Technical Report – Mt Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, issue date March 2, 2018; the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study amended and restated: July 7, 2014; the Amended and Restated NI 43-101 Technical Report Resource Update Mt Todd Gold Project, Issue Date of Amendment and Restatement: April 11, 2012; the 10.65 Mtpy Preliminary Feasibility Study NI 43-101 Technical Report Mt Todd Gold Project dated January 18, 2011; and the Preliminary Feasibility Study NI 43-101 Technical Report Mt Todd Gold Project dated October 1, 2010.
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 16th day of April, 2024.

Signed, Sealed "Deepak Malhotra, Ph.D., SME RM"
Signature of Qualified Person

Deepak Malhotra, Ph.D., SME RM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Zvonimir Ponos, BE, MIEAust, CPeng, NER
Senior Principal Engineer
Coffey Services Australia Pty Ltd (trading as Tetra Tech Proteus)
Level 1, 235 St Georges Terrace | Perth, Western Australia 6000
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This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Zvonimir Ponos, BE, MIEAust, CPeng, NER**, do hereby certify that:

1) I am a Senior Principal Engineer 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 38 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, 18, 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, 18.8, 21.1.2, 21.2.3, 21.2.6, 24.1, 24.6, 25.5, 25.6, and 25.7 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 a qualified person for the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; and 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 16th day of April, 2024.

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

Vicki Scharnhorst, P.E., LEED AP
Principal
Tetra Tech
1560 Broadway, Suite 1400 | Denver, CO 80202
Telephone: (303) 825-5999 | Facsimile: (303) 825-0642
Email: Vicki.Scharnhorst@tetratech.com

This certificate applies to the report entitled: "NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

I, **Vicki J. 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 41 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) 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 28–29, 2017 for two days.
5) I am responsible for Sections 1.1, 2, 3, 4, 5, 20.1, 20.3, 20.4, 23, 24, 24.2, 24.4, 25.1, 25.8, 25.10, and 26 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 for the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; and 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 16th day of April, 2024.

Signed, Sealed "Vicki J. Scharnhorst, P.E., LEED AP"
Signature of Qualified Person

Vicki J. Scharnhorst, P.E., LEED AP
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Keith Thompson, CPG, PG, member AIPG
Professional Geologist
Tetra Tech
1100 McCaslin Boulevard, Suite 150 | Superior, CO 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 Feasibility Study, Northern Territory, Australia" (Technical Report), effective date March 12, 2024 and issued on April 16, 2024.

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 44 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 21.1.3, 21.2.2, and 24.5 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 a qualified person for the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia, amended and restated September 22, 2020; 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 the NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Preliminary Feasibility Study amended and 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 16th day of April, 2024.

Signed, Sealed "Keith Thompson, CPG, PG"
Signature of Qualified Person

Keith Thompson, CPG, PG
Print Name of Qualified Person